LOS AZULES

CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT

PRELIMINARY ECONOMIC ASSESSMENT

LOS AZULES COPPER PROJECT

CREATED BY:

SAMUEL ENGINEERING
WE PROVIDE SOLUTIONS

McEWEN COPPER

Stantec

Knight Piésold CONSULTING

FOR:

McEWEN MINING

Effective Date: 05/09/2023
Date of Report: 05/31/2023

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1.0 EXECUTIVE SUMMARY

The Los Azules project is among the largest undeveloped copper deposits on the globe. Los Azules presents a multi-generational opportunity to design, build and operate a copper mine that is globally significant, technologically advanced, embraces regenerative design principles, and minimizes carbon footprint.

The future mine promises to be among the first major copper projects to be designed around regenerative principles from the ground up. Regenerative concepts are the ethos of having positive environmental, social, and cultural influences by carefully measuring impacts and prioritizing solutions that thoughtfully enhance the environment and community. By employing modern low-emission technologies and embracing renewable energy and non-traditional methods, the Los Azules project intends to produce copper with industry leading low greenhouse gas emissions.

This Technical Report is prepared for McEwen Mining Inc. (McEwen) trading under the symbol NYSE/TSX: MUX for the purposes of disclosing current updates and information related to its 51.9% owned subsidiary McEwen Copper Inc. (McEwen Copper), which controls the Los Azules copper property located in Argentina. Los Azules is an exploration and development project presently consisting of a large porphyry copper deposit located in the Andes Cordilleran region of San Juan Province, Argentina near the border with Chile (the "Project").

The Technical Report is prepared in accordance with the requirements set forth by Canadian National Instrument 43-101 ("NI 43-101") for the required disclosure of material information and is intended to meet the requirements of Form NI 43-101F1 as considered for a Preliminary Economic Assessment (PEA) level of study and disclosure as defined in the regulations and supporting reference documents (the "2023 PEA"). The effective date of this report is May 9, 2023, concurrent with the updated final resource estimates published herein. All currency shown in this report is expressed in May 2023 United States Dollars unless otherwise noted. Except for the purpose legislated under provincial securities law, any use of this report by any third party is at that party's sole risk.

The Project is at the exploration stage of investigation; consequently, this study is preliminary in nature and includes Inferred mineral resources in the conceptual mine plan and mine production schedules presented. Inferred mineral resources are considered too speculative geologically and in other technical aspects to enable them to be categorized as mineral reserves under the standards set forth in NI 43-101. There is no certainty that the estimates in this PEA will be realized.

Prior PEA reports were completed for the Los Azules property in 2009 and updated in 2010, 2013, and 2017. This 2023 PEA update supersedes the prior reports and reflects a revised development philosophy, processing flowsheet, updated resource model, metallurgical information, mine plans and economic parameters such as current capital and operating costs and associated new financial inputs and model. The effective date of this report is May 9, 2023, concurrent with the updated final resource estimates published herein.

The Los Azules deposit is a classic Andean-style porphyry copper deposit. The large hydrothermal alteration system is at least 5 km long and 4 km wide and is elongated in a north-northwest direction along a major structural corridor. The altered zone surrounds and includes the Los Azules deposit area, which is approximately 4 km long by 2.5 km wide. The limits of the mineralization along strike to the North and at depth have not been entirely constrained by drilling. Primary or hypogene copper mineralization extends to at least 1,000 m below the present surface. Near surface, leached primary sulfides (mainly pyrite and chalcopyrite) were redeposited below the water table in a sub-horizontal zone of supergene enrichment as secondary chalcocite and covellite. Hypogene bornite appears at deeper levels together with chalcopyrite. Gold, silver, and molybdenum are present in trace amounts, but copper is by far the most important economic constituent at Los Azules.

This 2023 PEA incorporates an updated development strategy with the following two phases: Phase 1 considers mining and processing resources associated with the oxide and supergene copper mineralization in the near surface portion of the deposit using heap leaching methods. Two production rates (175 kt and 125 kt per annum of copper) are considered for Phase 1. Phase 2 of the project considers the continued development of the deposit's primary copper mineralization found beneath the supergene copper layer. The focus of this PEA is the initial Phase 1 project with limited concepts presented for Phase 2. For clarity, the economic outcomes for the cases presented in this 2023 PEA include only Phase 1.

The Phase 1 implementation scheme for the Project is an open pit mine initially processing materials with crushing, bio-heap leaching and solvent extraction and electrowinning (SX/EW) facilities to produce LME Grade A copper cathodes for sale in Argentina or for export. Phase 1 mining extracts a total of 11.90 billion lbs. (5,404 kt) of contained copper from the resource, of which 8.68 billion lbs. (3,938 kt) is recoverable to copper cathodes. The total copper recovery expected is approximately 73% and considers scale-up efficiencies and production distribution over a two-year timeframe from placement of material on the leach pad.

The Phase 1 Base Case includes processing facilities to produce 175,000 tonnes per annum (tpa) of copper cathodes from higher grade, heap leachable copper content materials. The processing facility will function through to the completion of mining of Phase 1 in Year 21 with low grade stockpile reprocessing and residual leaching operations to Year 28 (the "Base Case"). Mining operations for the Base Case ramp up over the proposed mine life from approximately 70 million total tonnes per annum moving up to 175 million tonnes per annum through the life of the Project as copper grades decrease, and waste stripping increases.

To demonstrate the scalability of the project at lower initial capital requirements, an alternative case was also developed at a lower copper production rate of 125,000 tonnes per annum of copper cathodes. The processing facility will function through to the completion of mining of Phase 1 in Year 30 with low grade stockpile reprocessing and residual leaching operations to Year 33 (the "Alternative Case"). Mining operations for the Alternative Case ramp up over the proposed mine life from approximately 70 million total tonnes per annum to 110 million tonnes per annum through the life of the project as copper grades decrease, and waste stripping increases.

Phase 1 project development for either case is expected to take approximately 33 months to mechanically complete from notice to proceed and point of project financing. Construction development will prioritize the initial leach pad and ponds, crushing and stacking systems to facilitate the placement of leach materials on the pad during pre-stripping and prior to starting the rest of the facilities start. Ramp up to full leaching capacity is expected to take six to nine months from plant start-up and placement of mineralized material on the pad with commercial production of copper from the SX/EW plant is expected to be achieved in approximately 12 months from the start of leaching assuming the mining schedules shown. Finalization of the necessary permits to begin work is expected to be completed during the proposed feasibility study timeframe. Early works will commence, once project funding is available, with access road upgrades, site preparation, construction infrastructure and power line development.

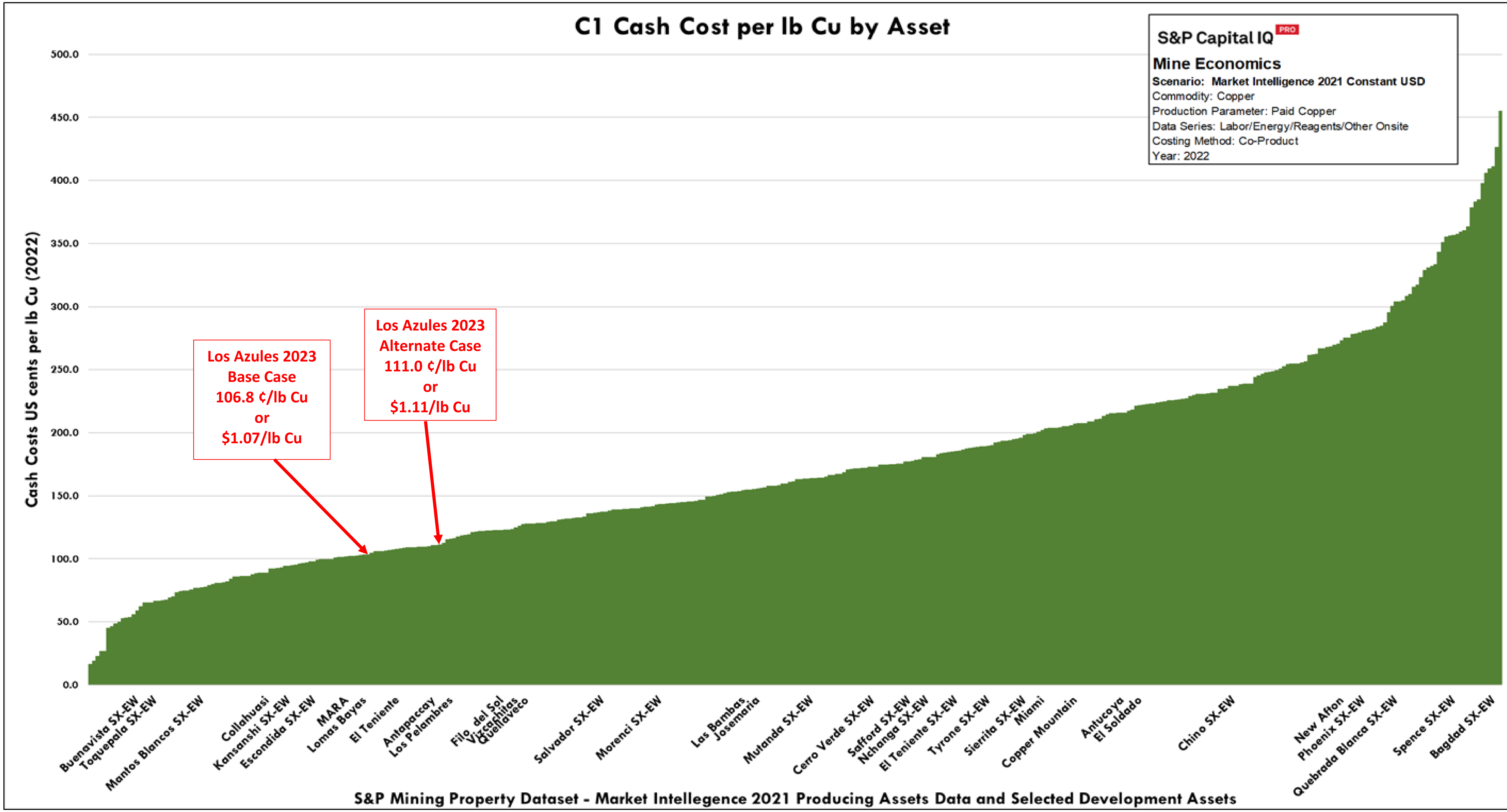
The Los Azules project area is in the semi-arid area of the Central Andes of Argentina, also known as the Dry Andes. This area of the Andes is characterized by significant daily temperature variations, low precipitation, and high solar radiation. Field geomorphology mapping, characterization, and distribution of all inventoried cryogenic geoforms including permafrost modelling has been conducted at Los Azules since 2010. To date, the cryogenic geoforms (landforms that may contain ice) in the area have been identified and characterized. Some of these geoforms have been excavated to check for ice distribution, and most are currently included in a topographic surveying program aimed at determining activity. Additional characterization work will be completed for specific geoforms in the coming field seasons. It is understood that no cryogenic geoforms will be impacted by Phase 1 operations.

Based on consensus estimates and independent analysis, long-term metal pricing used in this report (except for mineral resource estimation) and project economic analysis are Copper (Cu) - $3.75/pound; Gold (Au) - $1,700/ounce; and Silver (Ag) - $20.00/ounce. The 2023 updated financial outcomes for the Phase 1 initial project mine and facilities are shown in Table 1.1 below (expressed in Q1 2023 United States Dollars, after taxes).

Table 1.1: Project Phase 1 Life of Mine Economic Summary (After Taxes)

Project Metric	Units	Base Case 175k tpa Cu	Alt. Case 125k tpa Cu
Mine Life	Yr	27	32
Strip Ratio		1.16	1.16
Copper Production – cathode Cu	ktonnes	3,938	3,938
Initial Capital Cost	USD Millions	$2,462	$2,153
Sustaining Capital Cost	USD Millions	$2,243	$2,351
C1 Costs (Life of Mine)	USD/lb Cu	$1.07	$1.11
All-in Sustaining Costs (AISC)	USD/lb Cu	$1.64	$1.67
Internal Rate of Return (IRR)	%	21.2%	18.4%
Net Present Value (NPV) @ 8%	USD Millions	$2,659	$1,929
Pay Back Period	Yr	3.2	3.4

C1 cash costs are defined as the cash cost incurred at each processing stage, from mining through to recoverable copper delivered to the market, net of any by-product credits. C1 cash costs per pound of copper produced and all-in sustaining costs per pound of copper produced are non-GAAP ratios. If it were in production today, Los Azules' average C1 cash costs would be in the lowest cost quartile among copper producers. Figure 1.1 shows global cost data from S&P Capital IQ and SE showing how the Los Azules Base Case and Alternative Case average C1 cash costs compare to producing copper mines in 2022.



(Source: S&P Capital IQ Mine Economics Market Intelligence 2022 Data, SE Analysis)

Figure 1.1: C1 Cash Costs by Current Producer and Selected Development Projects

The opportunity to process primary sulfides directly through a heap leach rather than building a traditional copper concentrator in the future is the envisioned approach to the Los Azules development plan. The primary sulfides are presently not considered economically suitable for commercial heap leaching operation. Emerging technologies for improved leaching of sulfide copper ores are being developed, a proprietary catalytic bio-heap leaching technology that may provide an alternative approach to improving the leach performance of primary sulfide content in the processable materials considered in this report. This would have the potential to unlock the primary copper resources more economically with less environmental impact versus a mill/concentrator alternative and negate the need for a tailing storage facility.

Continuing the benefits of a hydrometallurgical approach is the project's preferred path for Phase 2 and ongoing development work efforts. Metallurgical work evaluating Nuton™ bio-leaching technology is being developed to potentially replace the need for a future milling operation in favor of continued leaching and copper cathode production for the life of the mining operations. Potential scenarios for the future operations employing the Nuton™ bio-leaching technology are presented and discussed in Section 25.2.1 of this report.

Although Nuton has completed larger scale testing at several global project sites and has developed proprietary modeling techniques to predict results, there are no commercial applications of the Nuton™ technology operating at the time of this report. Based on preliminary small-scale testing by Nuton and economic modeling inputs, these options provide the opportunity to extend the mine life to more than 50 years in some instances and increasing copper produced by more than 30% while adding significant additional value at lower LOM operating costs.

A significant testing program will be required to validate these preliminary estimates; therefore, these results are not considered suitable for inclusion at this time in the initial project phase cases presented and are only included as a demonstration of the potential future opportunity.

A conventional mill and flotation/concentrator option was considered to process primary copper mineralization to demonstrate economic viability employing conventional methods and support reserves estimation confidence. If a conventional mill and flotation/concentrator is constructed, the option leverages the smaller Alternative Case option to avoid oversizing the hydrometallurgical processing facilities. The milling facilities are brought online in Year 7 at a processing rate of 120,000 tonnes per day through completion of the project in Year 41. Crushing and stacking systems initially commissioned for use in the heap leaching process will be repurposed to provide ball mill feed and enable filtered tailings transport/storage options. Tailings storage management design would provide for a lined facility with filtered tailings (dry stacked) deposition for the applicable life of mine operations to minimize environmental impacts and freshwater usage. Details for this option can be found in Section 25.2.2.

The next steps for the Los Azules Project are continuing with infill resource drilling, variability and confirmatory metallurgical testing, environmental baseline studies, and commencing critical preliminary engineering such as hydrogeologic field investigations and geotechnical drilling at the heap leach pad site, tailings dam site and within the pit wall slopes to support a feasibility study.

A NEW VISION AND APPROACH

Copper is a key ingredient in the solutions to global climate change including initiatives in the automotive sector as it transitions to electric cars and to the energy sector as it moves to more renewable forms of energy. Los Azules aspires to be the world's first Regenerative Copper Mine, providing valuable materials for a renewably powered world.

Guiding regenerative principles were developed to reframe the approach to sustainable innovation within the mining industry and set forth high-reaching goals that are being explored for all facets of the mining processes considered for Los Azules. The project development seeks to significantly reduce the environmental footprint of mining operations and their associated greenhouse gas emissions by integrating the latest renewable and environmentally responsible technologies and processes. The Project aims to obtain 100% of its energy from renewable sources (wind, hydro, and solar) in a combination of offsite and onsite installations. Where possible, the project is also seeking to have long-term net positive impacts on the greater Andean ecosystem, the lives of miners, and the citizens of nearby communities, while contributing positively to the local and national economy of Argentina.

The project's core vision and approaches are summarized below and discussed in more detail in Section 2.1 of this report.

Rethinking Everything

It is time to rethink the copper mine – and to reinvent all the processes, technologies and equipment that goes into producing it. This is the question championed by McEwen Mining as it seeks to advance the Los Azules project. This is a remote and hard to access area – without infrastructure and existing activity – and provides an opportunity to rethink and renew the mining process in every way. Doing so requires changing conventional thinking and reinventing the industry that has done things a certain way for many decades.

The timeframe of the potential development of Los Azules is right on the cusp of this transition. Not only will it be harder to get approvals and goodwill without doing the right thing, but it is also morally untenable to pursue a major industrial activity without significant changes undertaken, given the science on climate change and being part of the future solution instead of continuing old ways. The world's greenest and most innovative mine will be a socially responsible copper mine that creates large economic benefits and results in a net positive impact to the environment and local communities.

The project concepts also allow for early adoption of emerging technologies under development and anticipated to be commercially viable over the mine life. Envisioning the ideal outcome, articulating that outcome into a goal, and then making decisions backwards through time that guarantee its attainment within the desired timeframe. Backcasting is a visionary process being employed that will produce the blueprints for the mine of the future by envisioning the ideal end game and the necessary steps to achieve those objectives as part of the initial project planning and development. By being 'future ready' the project will be poised to adopt newly emerging technologies and infrastructure opportunities rather than locking itself into old solutions.

Key project initiatives aimed at achieving these goals are described below.

Respecting the Lands We Use

The Los Azules project is committed to responsible stewardship of the land and minimizing disturbance of local glacial morphologies and wetlands ("vegas" in the local terminology) wherever possible. Careful consideration of how activities are conducted and where they are located is a key aspect to meeting these commitments, both in the short and long term. Minimizing land use and disturbance by consolidating uses to the extent possible is considered in the site layouts, individual site areas, facilities/buildings, and access to the mine site.

In 2010 Argentina passed the National Glacier Protection Law No. 26.639. It bans all activities (i.e., extractive such as mining or oil, tourism, and general infrastructure) on cryogenic geoforms, i.e., landforms that contain ice. The law appointed the national glacier institute or Instituto Nacional de Glaciología and Nivología (IANIGLA for its Spanish acronym) as the entity responsible for inventorying and classifying the glaciers in the country. Glaciers are morphologically classified as uncovered ("white") or covered ("rock") glaciers. There are no uncovered glaciers within the boundaries or in the immediate vicinity of the project area. This is important because there would be no impact on any glaciers or ice from dust or emissions from the activities at the site.

There are however several small cryogenic geoforms classified as rock glaciers in and near the Project area as determined by IANIGLA. The mine development and associated facilities at Los Azules have been considered such that none of the rock glaciers located within the property area are impacted Phase 1 operations, including buffer zones to ensure incidental impacts are not possible.

Although the vegas in the pit and leach pad areas will be impacted, minimizing the footprint of the site facilities, re-routing, and diversion of water courses to downstream connections are key design features for the mine and leaching areas water management plans to minimize these impacts. The leach pad design includes an underdrainage for non-contact water coming from upstream sources to flow through the same valley and to the Rio Salinas. Longer term, the water courses will be restored during reclamation of the mine site at the completion of activities to bring the area as close to its original state as possible.

Transforming Water Use and Quality

Climate change, population growth and the industrial and agricultural use of water are some of the factors that affect water availability. In addition, the expansion of urban infrastructure exerts pressure on the quantity and quality of natural water courses. Long-term water solutions must be flexible, adaptable, and environmentally sustainable and work within the ‘carrying capacity’ of its place and climate. Increasing the efficiency of water use is equivalent to increasing its productivity or, in other words, reducing the intensity of its use by maximizing the value of its uses and, in this way, improving its allocation among different competing uses.

The supergene copper mineral resources at Los Azules can be processed with the use of either concentration (milling and concentration by flotation) or hydrometallurgical (heap leaching and solvent extraction/electrowinning recovery to cathodes or “LX/SX/EW”) technologies currently in use. The Chilean Ministry of Mining, Comisión Chilena del Cobre (COCHILCO) published a report concerning water usage in the Chilean copper mining industry titled “Water Consumption in the Copper Mining Industry, 2017”. The report characterizes the relative water usage and usage intensity per tonne of material processed for copper mining projects in Chile based on its survey in 2017.

As seen in the COCHILCO data, selecting a hydrometallurgical process option for Los Azules could reduce effective water usage by 75% to 80% over a milling/concentrator alternative. Additionally at Los Azules, alternatives for improving precipitation/snow capture, site dust control, reuse/recycle and passive water treatment strategies are being developed.

Transforming the Energy/Carbon Nexus

An extensive review of power generation and supply options for the project was undertaken to consider the options for renewable energy. YPF Sociedad Anónima (“YPF S.A.” or “YPF”) owns and operates power generation facilities in Argentina based on wind, solar, geothermal, and hydroelectric sources through its subsidiary YPF-LUZ. YPF-LUZ has a rate structure based on 100% renewables sourced power generation that can be used as the project basis, eliminating hydrocarbon-based generation and associated emissions. The YPF-LUZ electric power supply option was selected for the Los Azules project at a small premium over other hydrocarbon-based power options.

In addition to energy supply, the reduction of energy consumption is also a key aspect to regenerative mining.

The Chilean Ministry of Mining, Comisión Chilena del Cobre (COCHILCO) published a report concerning energy usage in the Chilean copper mining industry titled “Energy Consumption in the Copper Mining industry, 2017”. According to the COCHILCO information, the specific energy demand per tonne of produced copper is about 13.6 gigajoules for the LX/SX/EW process (of which 10.8 gigajoules is electricity) and 9.3 gigajoules for a concentrator (of which 9.1 gigajoules is electricity). However, to be comparable in the case of plants equipped with concentrators, this value should also include for smelting and refining to get to a finished copper cathode product which increases the comparable energy value to 20.9 gigajoules for the concentrator/smelter/refinery processes (of which 14.2 gigajoules is electric power). In addition, transport to the smelter would also have to be accounted for.

Processing with the End Game in Mind

Given the context above, the most appropriate technology selection for Los Azules to minimize water usage is a hydrometallurgical approach, which is the basis for the Phase 1 project development. The hydrometallurgical option also provides lower overall project impacts from:

- Reduced energy usage by 35% over a concentration alternative to produce copper cathodes. The electric load reduction is about 25%.
- Lower transport requirements for product based on copper content of cathodes (99.99% Cu) versus concentrates (25%-35% Cu) and concentrate smelting options located outside of Argentina/South America.
- More efficient and minimized use of land for heap leach pad versus tailings storage facilities from concentration tailings discharge.
- On-site generation of sulfuric acid, using by-product sulfur supplied from local Argentinian sources, employing waste heat capture for on-site power generation and process heating – reduces grid based electric power requirements and eliminates hydrocarbon-based alternatives.
- Establishment of the infrastructure to be a rapid adopter of emerging heap leaching technologies for primary copper mineral resources when encountered – avoiding the future need for concentration methods as is the current industry practice.

Moving Rock and Decarbonizing Mining Operations

Maximizing electrification, coupled with renewable power supply is aimed at significantly reducing environmental impacts.

The initial mining concepts will use trolley-assisted diesel-electric mine haulage and support equipment initially to significantly reduce diesel emissions. However, the project will select equipment and methods to rapidly transition to fossil fuel-free alternatives as rapidly as the technology and manufacturing capacities allow. The transition would also include in-pit conveying alternatives to minimize fleet requirements. The ultimate vision is a fully electric mine and the elimination of emissions associated with fossil-fuels.

A Mining Camp for Maximum Livability – the healthiest, greenest mine camp in the world.

The remote nature of the Los Azules site necessitates onsite living. While the on-site work force will be minimized in favor of the local communities of Calingasta and San Juan and remote work concepts so that people can be closer to their families and communities, a necessary contingent will be needed at Los Azules itself. Creating living conditions that are exceptional and supportive of mental and physical health will reduce absenteeism and help McEwen maintain a productive, happy, and engaged workforce where jobs are coveted, and retention is high.

The Los Azules mine will house and support 1,000-2,000 workers at any time in various 'neighborhood' groupings organized in a linear fashion within the facility. The permanent camp is currently planned for construction and occupancy in year 5 of the project. Initially, the project will use the existing site all-weather modular camps and construction camp to begin operations.

The long-term permanent mine camp has been strategically located to optimize multiple variables. Worker safety, comfort, well-being, as well as the distance from the mine operations and access to the main road are major considerations. In addition, the specific layout and orientation have been selected to support passive heating and cooling strategies and solar energy generation, which are key considerations.

The Los Azules camp and mine will be forming a microgrid in a remote location, although Los Azules is closer to basic infrastructure than most other mines in the region. Even though the camp will be connected to offsite energy production, it is being sized for net-positive energy production, making it a candidate for International Living Future Institute's (ILFI) Net Zero Energy certification (living-future.org), the world's most rigorous green building standards. This self-sufficient energy system will serve the entire site's occupied footprint and will facilitate load stabilization through energy storage technology and carefully timed consumption.

The camp will also be designed to provide heating and climate control, acoustics, medical and support services including recreation and medical clinic, improved air quality using living plant systems and filtered air, and water management to capture rainwater and snowmelt, retain that water and treat it for reuse using natural systems. The camp will pursue ILFI certification based on the alignment with the Living Building Certification "Water Petal".

The camp will be designed to provide space for growing food in a self-sustaining environment. Finally, the camp will provide waste management systems to provide reuse of waste materials, either through direct reuse, recycling, composting and elimination of single-use plastics and packaging.

Minimizing the Carbon Footprint from Mine to Market

Copper mining emits an average 2.3-2.5 tonnes of carbon dioxide equivalent per tonne of copper metal produced (t CO2-e/t Cu), while smelting adds another 1.65 tonnes (Source: "Metals recycling to be a key plank for cutting emissions" by Pratima Desai, Reuters, July 14, 2021). By employing modern, low emission technologies, the Los Azules project intends to improve upon the standards set forth by "The Copper Mark" and set a new standard for CO2 emissions per unit of copper produced.

The Greenhouse Gas ("GHG") Protocol Corporate Standard classifies a company's GHG emissions into three 'scopes'. Scope 1 emissions are direct emissions from owned or controlled sources. Scope 2 emissions are indirect emissions from the generation of purchased energy. Scope 3 emissions are all indirect emissions (not included in scope 2) that occur in the value chain of the reporting company, including both upstream and downstream emissions.

Figure 1.2 Estimated Carbon Intensity versus Copper Equivalent Production Centiles 2022-2040 for mine site emission chart presents the relative estimated emissions for copper assets on an equivalent copper basis as obtained from the Emissions Benchmarking Tool – (Metals)™, a product of Wood Mackenzie Limited ("WoodMac"). The WoodMac database includes 394 individual global mining assets and covers Scope 1 and 2 emissions determined using the published methodology on their website. The highlighted assets represent comparable major Argentinian projects as included in the WoodMac modeled information. The Los Azules project metrics in the WoodMac data (red line highlighted) reflects the estimated emissions for the prior project concept. The WoodMac average Scope 1&2 emissions intensity for all 394 included assets the period between 2022 and 2040 is 1,980 kg CO2-e/t Cu Eq. (kilograms of Carbon Dioxide Equivalent per tonne of Copper Equivalent produced). Carbon Dioxide Equivalent means having the same global warming potential as any another greenhouse gas. For the 57 copper SX/EW assets included the average Scope 1&2 emissions intensity for the period between 2022 and 2040 is 1,723 kg CO2-e/t Cu Eq.

Based on the current project concepts considered for implementation at Los Azules, notably:

- Electrical energy sourced from 100% renewables (YPF Luz basis),
- Incorporation of site and mine electrification concepts (trolley assist for mine haulage, battery electric vehicles where possible),
- Regenerative design concepts for support infrastructure, and
- Hydrometallurgical extraction processes to produce copper cathodes.

The carbon intensity per unit of copper equivalent production (Cu Eq) was estimated by Whittle Consulting Pty Ltd ("WCPL") using the GHG Protocol Corporate Accounting and Reporting Revised Standard principles (published by the World Resources Institute (WRI), a U.S.-based environmental NGO, and the World Business Council for Sustainable Development (WBCSD), a Geneva-based coalition of 170 international companies) which provides requirements and guidance for companies and other organizations preparing a corporate-level GHG emissions inventory).

WCPL's estimations based on the preliminary information developed, the predicted carbon intensity for the Los Azules initial Base Case project is estimated to be to be 826 kg CO2-e/t Cu Eq for Scope 1 & 2 emissions. The Alternative Case project is estimated to be to be 670 kg CO2-e/t Cu Eq for Scope 1 & 2 emissions. The estimated Scope 1-3 emissions for the Base and Alternate cases are approximately 1066 and 902 kg CO2-e/t Cu Eq t respectively, assuming transport of copper cathodes to port facilities in either Chile or Argentina.

Figure 1.2 also shows the relative position of the Los Azules cases developed for this 2023 PEA and the prior Los Azules 2017 PEA project concept against the WoodMac average Scope 1&2 emissions intensity for all 394 included assets. Of significant importance is the improvement in the project compared to the prior concept and the project position in the lower 15% range of projects globally. Full electrification could drive emissions towards the lowest in the industry.

Continued implementation of newer and less impactful technologies, fully electric mine and equipment, EV use for materials and supplies transport to site and broader employment of regeneration applications throughout the mine site to further off-set carbon emissions is expected to deliver on McEwen's commitment to achieve net-zero carbon emissions from the Los Azules project by 2038, well ahead of its peers.

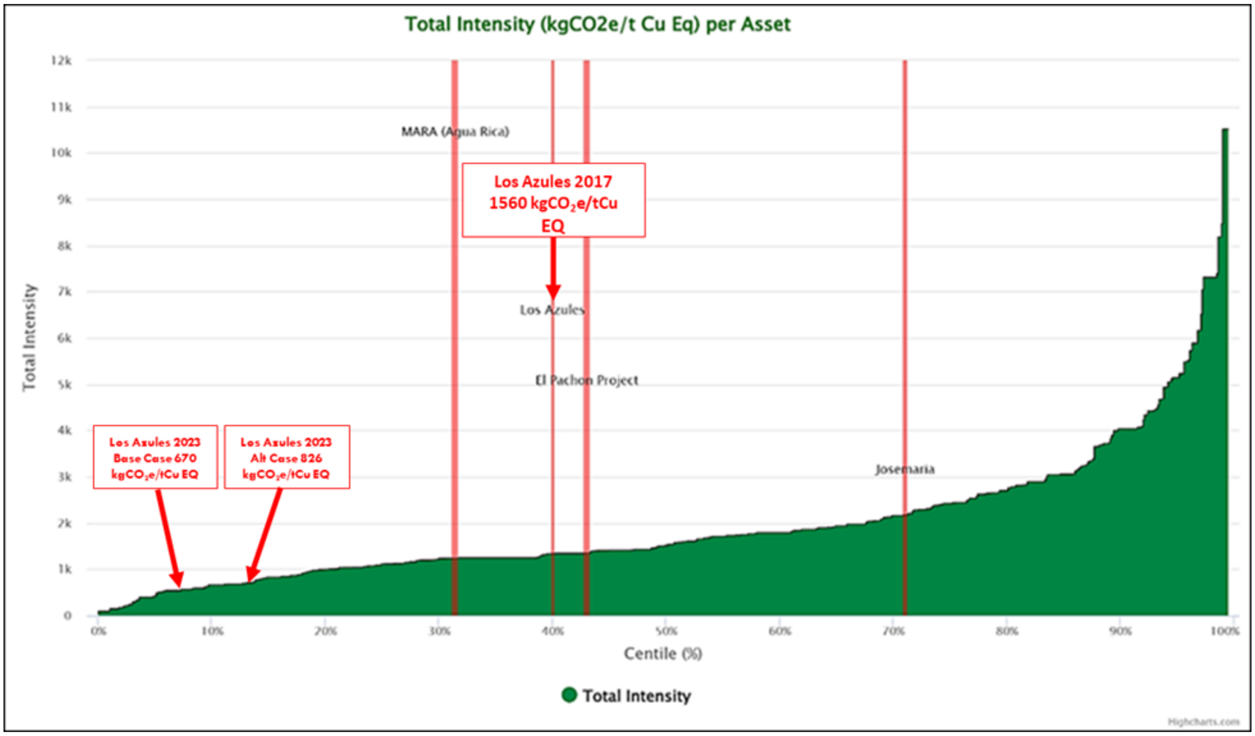


Figure 1.2: Estimated Carbon Intensity vs Copper Equivalent Production Centiles 2022-2040 (Scope 1 & 2 Emissions) - Wood Mackenzie 2022

NOTE: "The data and information provided by Wood Mackenzie should not be interpreted as advice and you should not rely on it for any purpose. You may not copy or use this data and information except as expressly permitted by Wood Mackenzie in writing. To the fullest extent permitted by law, Wood Mackenzie accepts no responsibility for your use of this data and information except as specified in a written agreement you have entered into with Wood Mackenzie for the provision of such data and information."

1.1 OWNERSHIP STRUCTURE

This subsection was prepared by J. Sorensen, FAusIMM, Samuel Engineering (Source Q1 2023 public filings).

McEwen was organized under the laws of the State of Colorado on July 24, 1979, and is listed on the New York Stock Exchange (NYSE) and on the Toronto Stock Exchange (TSX) under the symbol MUX. The Company's head office is Toronto, Canada. As of May 2023, the Company owns a fully diluted 51.9% interest in the Los Azules copper deposit in San Juan, Argentina through its subsidiary, McEwen Copper Inc. ("McEwen Copper") which owns a 100% interest in the Los Azules copper project in San Juan, Argentina, and the Elder Creek exploration project in Nevada, USA. The relevant ownership structure is shown in Figure 1.3 as provided by McEwen Mining.

FCA Argentina S.A., a subsidiary of Stellantis N.V. ("Stellantis"), invested ARS $30 billion in Argentina to acquire shares of McEwen Copper in a transaction that closed on February 24th, 2023.

Nuton LLC, a Rio Tinto Venture, invested $55 million to acquire shares of McEwen Copper in two transactions which closed on August 31, 2022, and March 15, 2023.

Both the Stellantis and Nuton investments included investor rights and product purchase rights discussed in detail in Section 2.4.

McEwen Copper has 28,885,000 common shares outstanding on a fully diluted basis, and its shareholders are: McEwen Mining Inc. 51.9%, Stellantis 14.2%, Nuton 14.2%, Robert R. McEwen 13.8%, Victor Smorgon Group 3.5%, and other shareholders 2.4%.

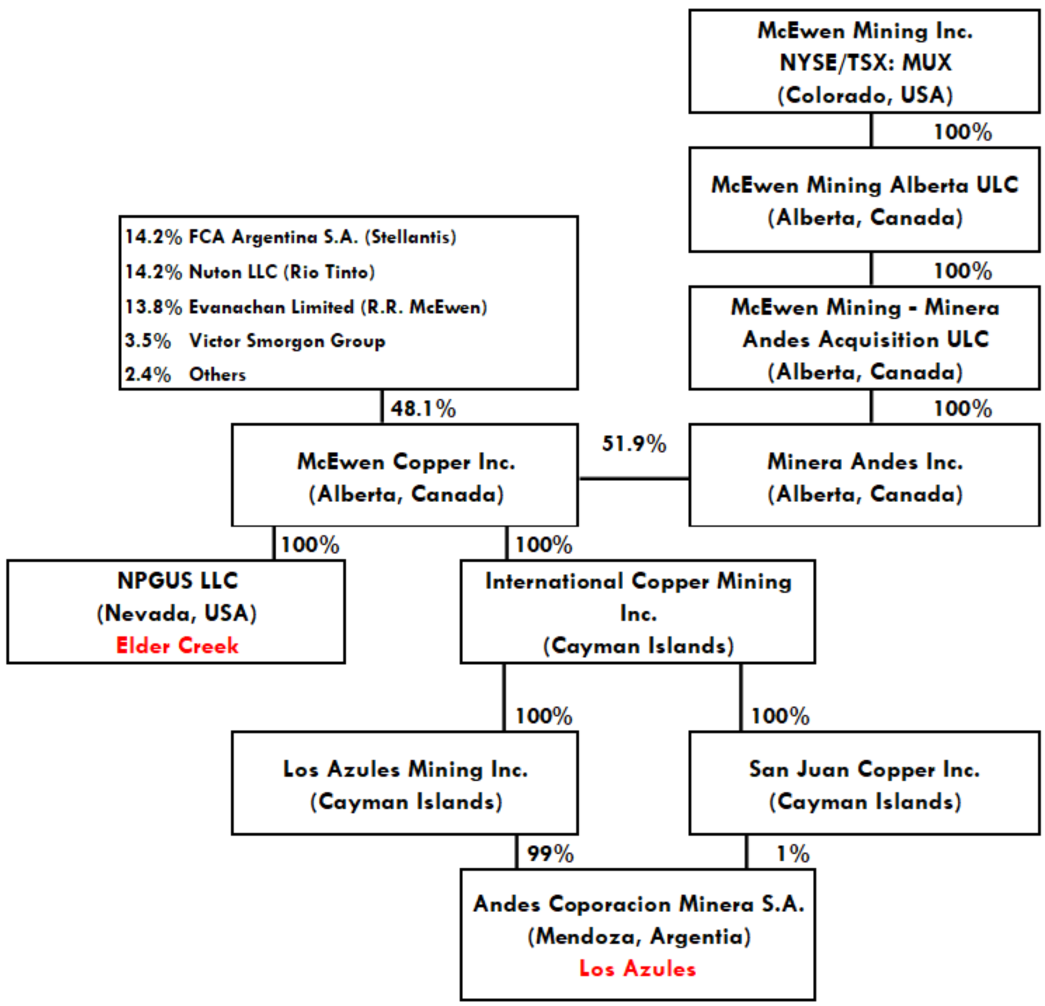


Figure 1.3: Los Azules Project Ownership Structure (McEwen, 2023)

1.2 LOCATION

The Los Azules Project is a porphyry copper development project located in the Andes Cordilleran region of San Juan Province, Argentina along the border with Chile. The project falls within the Calingasta Department of the San Juan Province. The Project is approximately 80 km west-northwest of the town of Calingasta, in the San Juan Province of Argentina at approximately 31° 06' 25" south latitude and 70° 13' 25" west longitude. The mine development is located approximately 6 km east of the border with Chile (Figure 1.4). Calingasta is located 173 km by road west of the city of San Juan along Route 12.

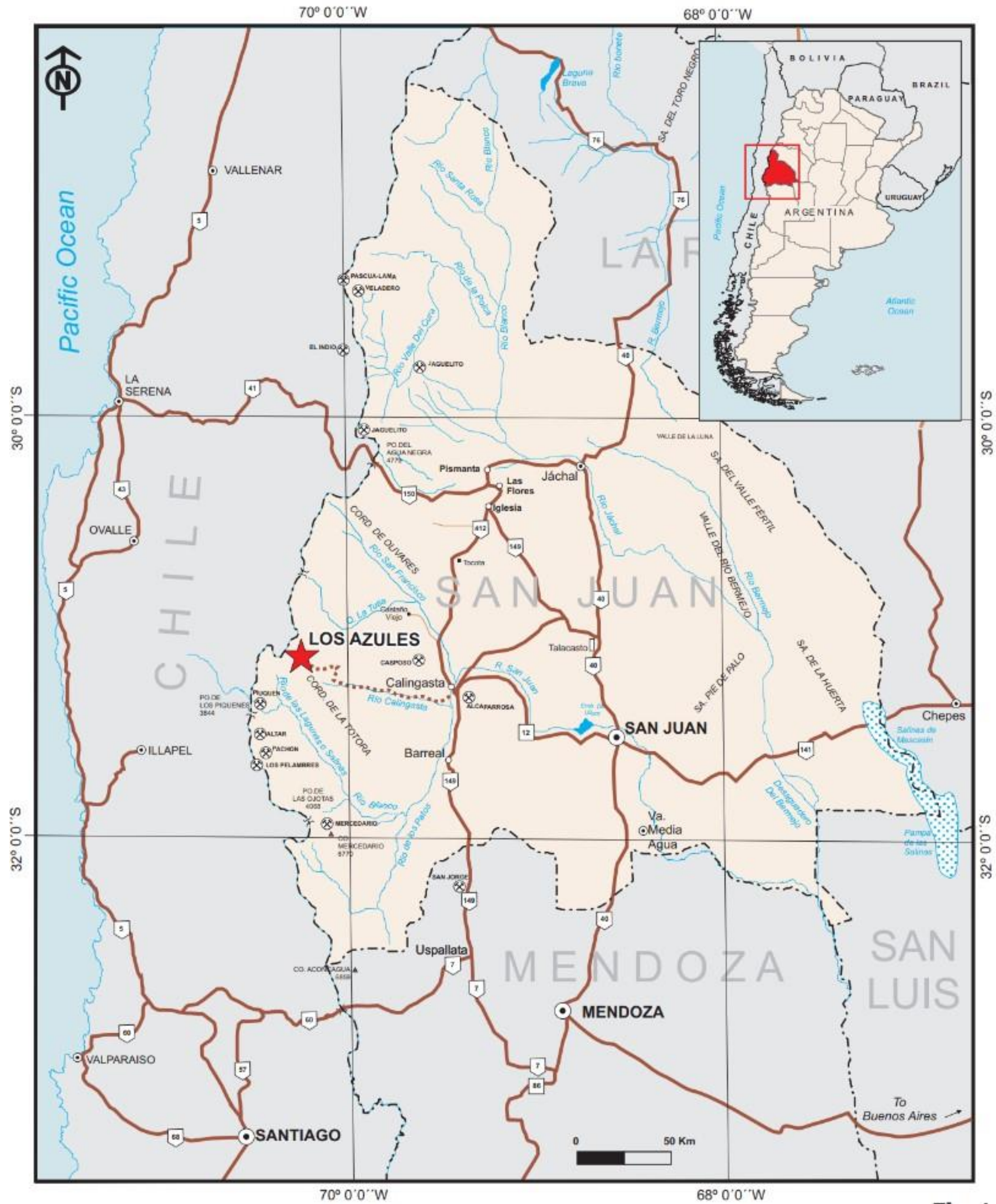


Figure 1.4: Location of Los Azules in the High Andes (Hatch, 2017)

The terrain elevation at the project site ranges between 3,200 meters above sea level (masl) at the proposed camp location and up to 4,500 masl on the high peaks in proximity to the Project. The proposed pit and facilities are located between 3,200 and 3,600 masl. The Project area is remote, and no infrastructure is present. There are no nearby towns, Indigenous residents, or settlements. Seasonal exploration work typically commences in October or November and terminates in April or May. Exploration operations are supported by means of two temporary camps within the Project site area.

The Mine is situated in a broad valley, with a central ridge called La Ballena ("the whale"). Vegetation is sparse and is virtually absent at higher elevations. Deposits of glacial debris (morainal materials) and scree mantle are present over much of the deposit and adjacent mountainsides. In the Project area, these materials locally exceed 60 m in thickness, but on La Ballena the cover is often 10 m or less.

The facilities and site arrangements contemplated for the Los Azules project are shown in Figure 1.5. Facilities are located to stay within the surface and mining rights currently held by McEwen.

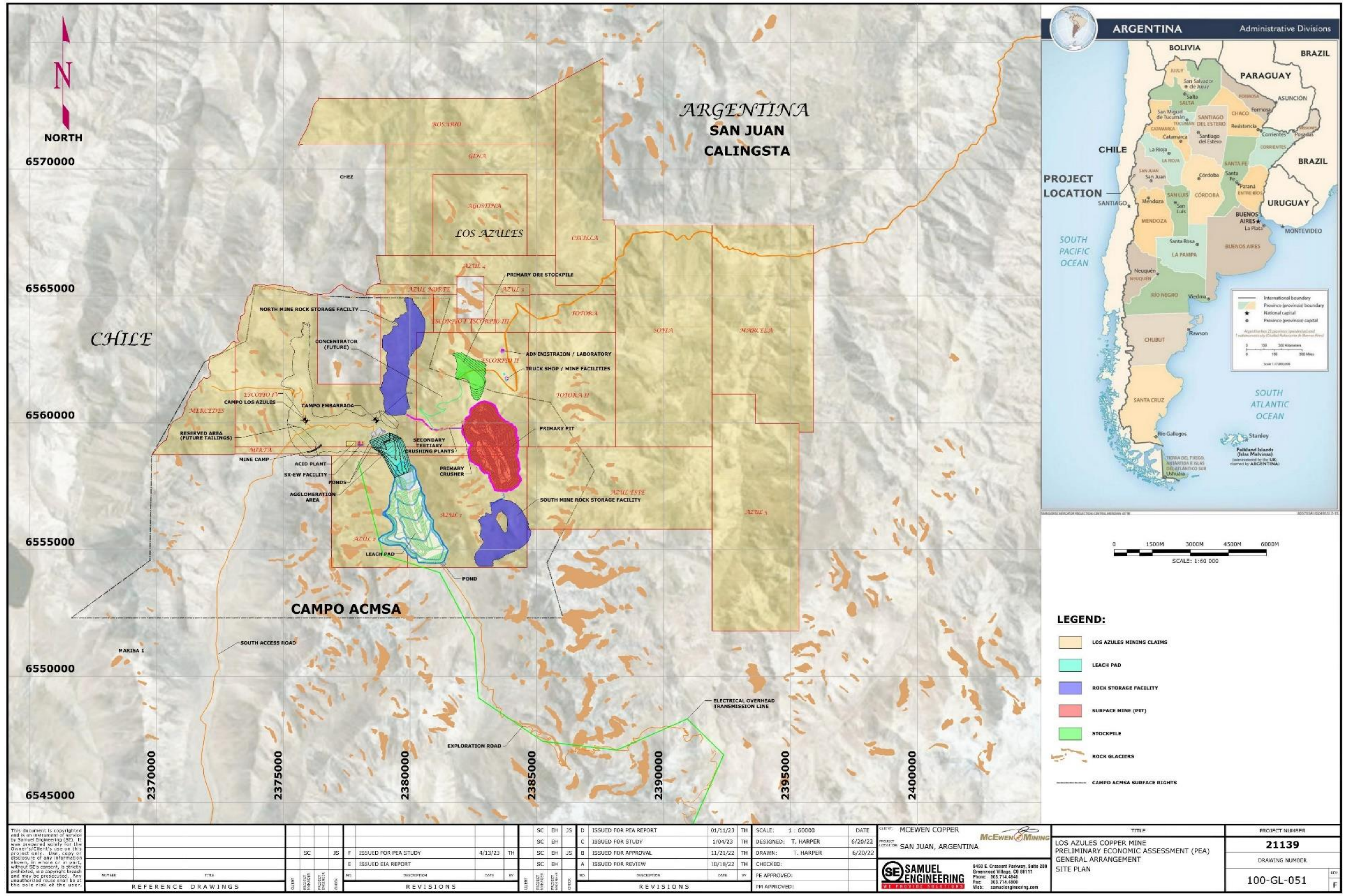


Figure 1.5: Overall Site Plan (Samuel, 2023)

There are no covered or uncovered "white glaciers" (classic ice glaciers) in the Project, although there are several small rock glaciers (or cryogenic geoforms) near the Project area that are not impacted by exploration or the proposed future development activities.

A preliminary seismic risk assessment of the Los Azules site was completed in April 2019 by the Instituto de Investigaciones Antisísmicas by Ing. Aldo Bruschi of the Universidad Nacional de San Juan (Institute of Anti-Seismic Research (IDIA) of the UNSJ). The Los Azules site corresponds to Argentinian Regulations (INPRES-CIRSOC 103 Zone 4). The INPRES-CIRSOC 103 regulation in force, is applicable to the scope of new constructions that are executed within the Argentine Territory, which was taken as a reference to define the seismicity of the area. The area where the Los Azules Project is located corresponds to Seismic Zone 4, considered very high. The highest seismic event recorded near the Project was magnitude 7.5 in 1977 affecting the entire province of San Juan.

Drill core storage and processing facilities are in the town of Calingasta. These facilities will be upgraded during the next phases of work to increase the storage capacity and provide for accommodations and staging of workers traveling to/from the project site.

Several existing mining operations and development projects are in the region surrounding the Los Azules project site as shown in Figure 1.6. The nearest to the Los Azules project site are the Altar copper-gold project site owned by Aldebaran Resources Inc. located approximately 40 km south and the El Pachon copper-gold project site owned by Glencore plc located approximately 90 km south of Los Azules. To the north, the distance to Filo del Sol and Josemaria is approximately 300 km.

Figure 1.6: Regional Mining Projects (McEwen, 2022)

1.3 ACCESS AND INFRASTRUCTURE

The Los Azules Project is currently accessed by 120 km of gravel road with eight river crossings and two mountain passes (both above 4,100 m elevation) described as the Exploration Road. This access is subject to snow accumulation and is passable only from October through to June. This 2023 PEA update describes in Section 18 “Project Infrastructure” a potential future northern access route within an existing easement that is less affected by snow. This option will be reevaluated once the project is established. Also described is an airstrip currently permitted for construction, but not for operation.

Additionally, an unimproved southern access road from the town of Barreal to the site, used by the El Pachon and Altar projects, exists, and was used in 2022 to support later season drilling. This route would be improved for the initial operations providing two points of access to the site.

The major Chilean population center of Santiago is approximately 270 kms south-southwest (400 kms by well-developed road) from Calingasta.

The map showing access routes into the site, including a future northern route, is provided in Figure 1.7. The main access will be by the Southern Road route to begin the project. This road will be upgraded for the expected year-round traffic travel to and from the site and extends approximately 192 km from the town of Barreal. The conceptual level designs and costs for the Southern Road upgrade were provided by Ruiz y Associados Consultoras S.R.L. (RyAC) and were estimated to be USD $138 million. These costs do not assume any contribution from other projects in the area.

The existing Exploration Road was upgraded in 2022/2023 to allow for access by larger vehicle traffic and safer transit. The road will continue to be maintained to provide seasonal secondary site access and support the incoming high voltage powerline routing.

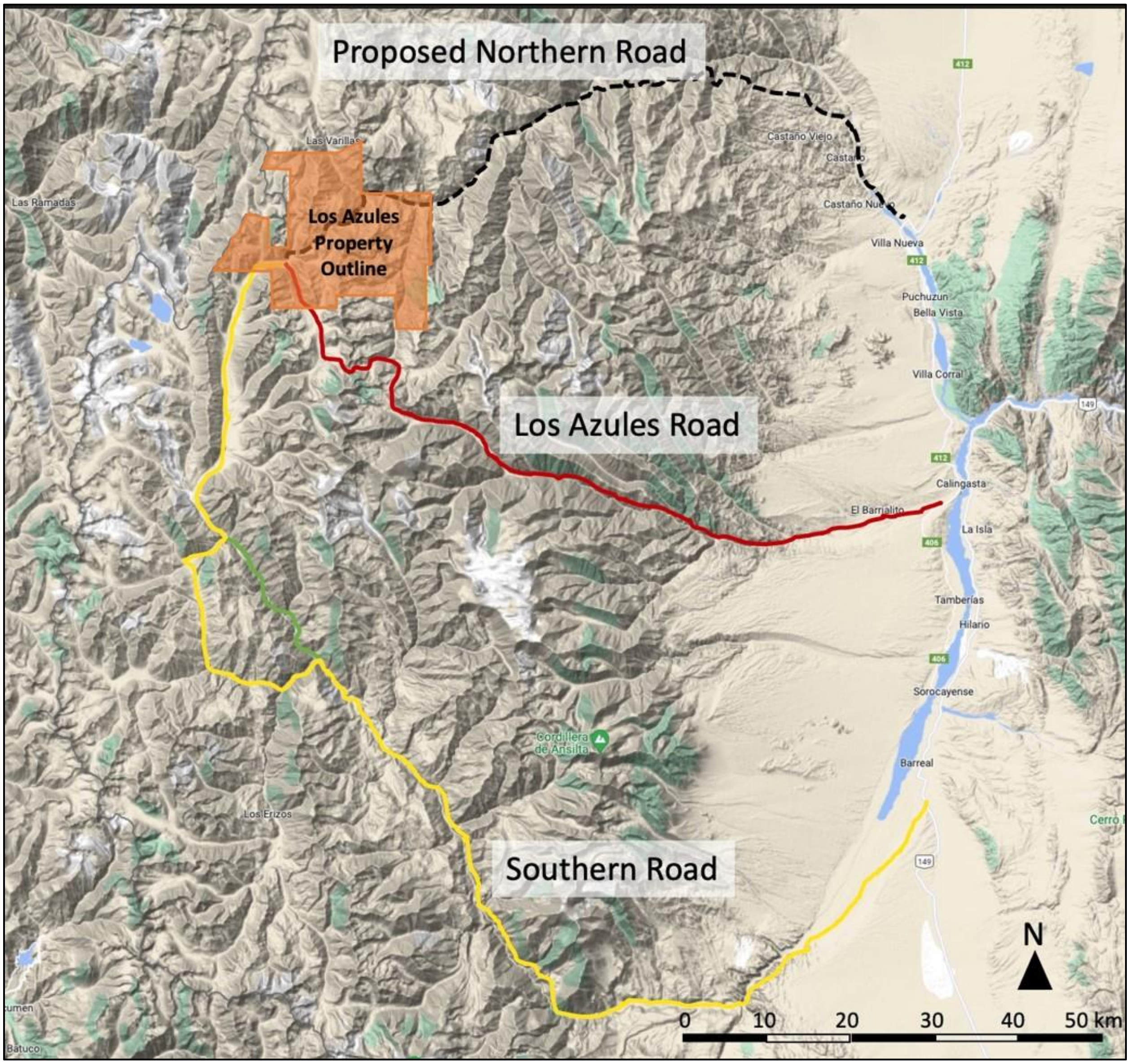


Figure 1.7: Existing Access & Infrastructure (ACMSA, 2022).

The city of Mendoza is located approximately 200 km south-southeast (275 road kms) from Calingasta. Mendoza is also the location of the nearest international airport (MDZ). Regional air service is also available from San Juan (UAQ).

Public research suggests that the import of copper into Argentina varies between 25,000 and 35,000 tonnes per annum, including fabricated copper components and raw materials. The project assumes export as the base case with an opportunity to sell some production into the Argentinian market. Port facilities for materials and product distribution include the inland port of Rosario in Argentina and Valparaiso, Ventanas, San Antonio and Coquimbo in Chile. The Rosario port facilities can be accessed via rail transport from the Cañada Honda rail depot located at the southern outskirts of San Juan.

The distances from the junction of the National Route (Ruta Nacional) RN 149 with the RN 153 (south of the town of Barreal) to the Chilean ports for copper cathode export are as follows:

- Ventanas: 380 km
- Valparaíso: 410 km
- San Antonio: 440 km

The entire distance is covered by paved roads, except for approximately 37 km of gravel in Argentina, on the RN 149, in Mendoza territory, between the junction with RN 153 (San Juan/Mendoza limit) and Uspallata (of the 55 km between RN 153 and Uspallata, 37 km are gravel and 18 km north of Uspallata are already paved). However, this stretch of gravel is passable all year round and it is highly likely that in the short term it will be paved prior to the project development.

There are no copper concentrate smelting/refining facilities in Argentina. Copper concentrates could be exported through the Port of Rosario in Argentina and Port of Coquimbo in Chile to global markets.

1.4 CLIMATE

The climate at the Los Azules site is categorized as semi-arid and classified as Köppen Climate Type ET-Tundra.

Using available site weather data from the January 2010 to May 2023 period, annual average rainfall recorded was 173 mm (when full years were recorded) and ranged between 120 mm in 2021 and 267 mm in 2015. The maximum snow depth recorded was 2.67 meters in June 2016. Most of the precipitation occurs during the winter months as moderate snowfalls during the winter. Temperatures varied seasonally and averaged 3.3°C with a low temperature of -24°C in July 2013 and a maximum temperature of 23.3°C in January 2017, indicating a large diurnal temperature range of approximately 20 degrees.

The area is subject to dry hot foehn wind (locally called Zonda). Average year-round wind speed is approximately 10 km/h with a maximum recorded wind speed at the man camp of 52 km/h. Stronger gusts are experienced in the higher mountain passes traversed by the existing exploration road and narrow canyons.

1.5 HISTORY

There are no formal records of exploration in the Project area prior to 1980. The only important active project in the area prior to 1980 was the El Pachón porphyry copper project, now owned by Glencore plc (Glencore), which is located approximately 90 km south of Los Azules. Evidence of prospecting (small trenches or pits) exists on some of the concessions.

In the mid-1980s through the mid-1990s, Battle Mountain Gold Corporation (BMG) explored the area and discovered a large hydrothermal alteration zone associated with dacite porphyry intrusions and stockwork zones. BMG drilled 24 reverse circulation holes during 1998 and 1999 looking for a high-grade gold deposit. Low-grade porphyry copper style mineralization was detected in the drilling, but BMG was focused on gold exploration. Concurrently during the mid-1990s, Minera Andes Inc., through its local subsidiary Andes Corporación Minera S.A. ("ACMSA"), acquired concessions in the area based on regional exploration and Landsat imaging. Minera Andes' claims adjoined the BMG claims to the south.

In December 2003, Minera Andes initiated an exploration program at Los Azules, including geologic mapping and sampling, ground magnetic and induced polarization (IP) geophysical surveys and core drilling. Minera Andes' initial core drilling intersected porphyry-style copper mineralization and in 2006 drilling intersected high-grade intervals up to 1.6% copper over 221 m and 1% copper over 173 m in separate holes.

After BMG merged with Newmont in 2000, part of the BMG properties was acquired by Solitario Resources (the "Solitario property"), a Canadian junior exploration company (subsequently called TNR Resources – "TNR") and part were acquired by an individual from San Juan named Hugo Bosque. MIM optioned the Solitario property in May 2004. Xstrata succeeded MIM and in April 2007 it exercised its option to acquire Solitario's concessions.

In 2007, Minera Andes (as operator) and Xstrata entered into an option agreement that consolidated Minera Andes' and Xstrata's properties. Pursuant to the Los Azules Option Agreement, dated November 2, 2007, entered into between MIM Argentina Exploraciones S.A., Xstrata Queensland Limited, Minera Andes S.A. and Minera Andes Inc. (the latter later acquired by US Gold Corporation and the business combination later renamed McEwen Mining Inc.), once a Feasibility Assessment is completed on "Los Azules" project, including properties named Mercedes and Mirta, a payment of USD 500,000 is due to Ms. Dina Myriam Elizondo de Bosque and Mr. Hugo Arturo Bosque.

In October 2009, Xstrata declined to continue to participate in the Project and as a result Xstrata assigned its properties to Minera Andes Inc. and the company owned 100% of the Project.

In January 2012, Minera Andes was acquired by US Gold Corporation, which was subsequently renamed McEwen Mining Inc.

Certain portions of the northern part of the Project that were formerly held by Xstrata and transferred to Minera Andes following the termination of the Los Azules Option Agreement were subject to an underlying option agreement between Xstrata and a subsidiary of TNR Gold Corp. This agreement was the subject of litigation in the Supreme Court of British Columbia, Canada.

Pursuant to terms of a settlement agreement in 2012, TNR retained a Back-in Right for up to 25% of the equity in the Solitario Properties. The Back-in Right was only exercisable after the completion of a feasibility study. To exercise, TNR must pay two times the expenses attributable to the back-in percentage (i.e., paying 2 × 25% of all the costs attributable to the Solitario Properties). Upon backing-in, TNR could have elected to continue to participate in the Project, or upon being diluted down to a 5% or less equity interest, have their interest converted to a 0.6% NSR on Solitario Properties.

The final Transfer Agreement between TNR Gold Corp (along with their wholly owned subsidiary, Compañia Minera Solitario Argentina S.A.), and McEwen Mining (including Los Azules Mining Inc. and ACMSA was executed on October 16, 2014, superseding the prior settlement agreement and transferred all rights on the Solitario Properties from TNR to McEwen Mining in exchange for an NSR royalty of 0.4% and TNR relinquishing the back-in right. The NSR terms are subject to a Net Smelter Returns Royalty Agreement executed on October 29, 2014, between Compañia Minera Solitario Argentina S.A. and ACMSA.

By the end of the 2012-2013 field season, 185 diamond drill holes totaling 59,518 m have been drilled at Los Azules. In addition, 52 reverse circulation holes had been drilled by BMG, Mount Isa Mines (MIM) and Minera Andes/McEwen Mining totaling 10,146 m.

In 2018, ACMSA filed a request to group all the Mining Permits together, file #1124.553-A-2018, in such way that, once all surveys are approved, all the Mining Permits be considered as one larger Mining Permit. This will allow investments to be distributed across the larger permit group and eliminate the need to spend on each individual Mina. It is expected that this request by ACMSA could be favorably resolved during 2023.

In July 2021, a private placement financing was initiated for McEwen Copper seeking to raise USD $80 million at a price of USD $10.00 per common share. Prior to the private placement McEwen Copper had 17,500,000 fully diluted shares outstanding. This financing subsequently closed in three tranches on August 23rd, 2021, June 21st, 2022, and August 31st, 2022, respectively.

As of December 31st, 2022, McEwen Copper had 25,685,000 common shares issued and outstanding on a fully diluted basis, of which 17,500,000 common shares were owned (68.1%) by its parent company McEwen Mining Inc., and 8,185,000 common shares were owned by third parties and affiliates.

During 2022, a 1.25% net smelter return (NSR) royalty was created encumbering the Los Azules project, which is held by McEwen Mining Inc.

The legal challenge by ACMSA for the peripheral properties under threat of forfeiture, Gina, Sofia, Torora II, and Marcela, was resolved in the company's favor and the forfeited rights were returned by the Mining Council on December 29th, 2022.

FCA Argentina S.A., a subsidiary of Stellantis N.V. ("Stellantis"), invested ARS $30 billion to acquire shares of McEwen Copper in a transaction that closed on February 24th, 2023.

Nuton LLC, a Rio Tinto Venture, invested USD $55 million to acquire shares of McEwen Copper in two transactions which closed on August 31st, 2022, and March 15th, 2023.

Both the Stellantis and Nuton investments included investor rights and product purchase rights discussed in detail in Section 2.4.

As of 2023, ACSMA/McEwen Copper continues to hold 100% of the Los Azules development and associated land holdings and mineral concessions and easements and continues to perform seasonal infill drilling and studies with a view to eventual project development.

1.6 PROPERTY

The information in the section relies upon a legal review and opinion report Re: "Incorporation and good standing status of Andes Corporación Minera S.A. (ACMSA) and of its mining rights" dated January 11th, 2023, by Abogado (lawyer) Jose Vargas CEI of Vargas & Galindez (V&G), a Mendoza based legal firm.

The Los Azules Project is comprised of properties (the "Properties") owned by Andes Corporación Minera S.A. (ACSMA), an Argentine subsidiary of McEwen Mining through its ownership in McEwen Copper. ACMSA is duly registered before the Dirección de Personas Jurídicas of the province of Mendoza, by Resolution #2025 dated November 2nd, 2005.

There are two types of tenures under Argentine mining regulations: Cateos (Exploration Permits) and Minas (Mining Permits). Exploration Permits are licenses which allow the property holder to explore the property for a period following a grant that is proportional to the size of the property. Mining Permits are licenses which allow the holder to exploit the property subject to regulatory environmental approval. To convert an exploration permit (Cateo) to a mining concession (Mina), some or all the area of a cateo must be declared as MD (Manfestación de Descubrimiento) and then converted to a Mina. Minas are mining concessions which permit mining on a commercial basis.

McEwen Mining controls approximately 31,746 ha of mining rights (Minas) around the Los Azules deposit. In addition, McEwen Mining owns sufficient surface rights for the Project pursuant to an agreement with CCM

S.A., on March 3rd, 2010, whereby ACMSA acquired the surface rights set out in Figure 1.8 (18,000 hectares in green outline).

The international border with Chile forms the limits of the owned property on the west side (shown as a black dashed line Figure 1.8). The surface rights limits of the property are represented by the green line in Figure 1.8 Based on the V&G review and opinion, ACMSA has good and valid, legal, and beneficial title to the mining rights shown in Figure 1.8.

The Escorpio IV property area (outlined in red in Figure 1.8) includes an area in the north central part of the property where survey information is still pending resolution and agreement with the Mining Council. Because of the location of the purchased property (on the border with Chile), the purchase required approval from the Dirección de Asuntos Técnicos de Frontera, Ministerio del Interior. Such approval was granted on August 31st, 2010, pursuant to Resolution 907 of the Dirección de Asuntos Técnicos de Frontera, Ministerio del Interior.

The Soberanía mine (File No. 259.299-C-84), located south of Azul 4, between the Azul Norte and Azul 3 properties. This property was declared expired under Argentine law and returned to the domain of the State. The State published this expiration giving the possibility for third parties to request the area. Andes exercised this right, but this is in dispute as on the same date and time, three other individuals (Hugo Bosque, Eduardo Caputo, and Deolinda Estela Rodríguez) exercised the same right, something allowed by Argentine law.

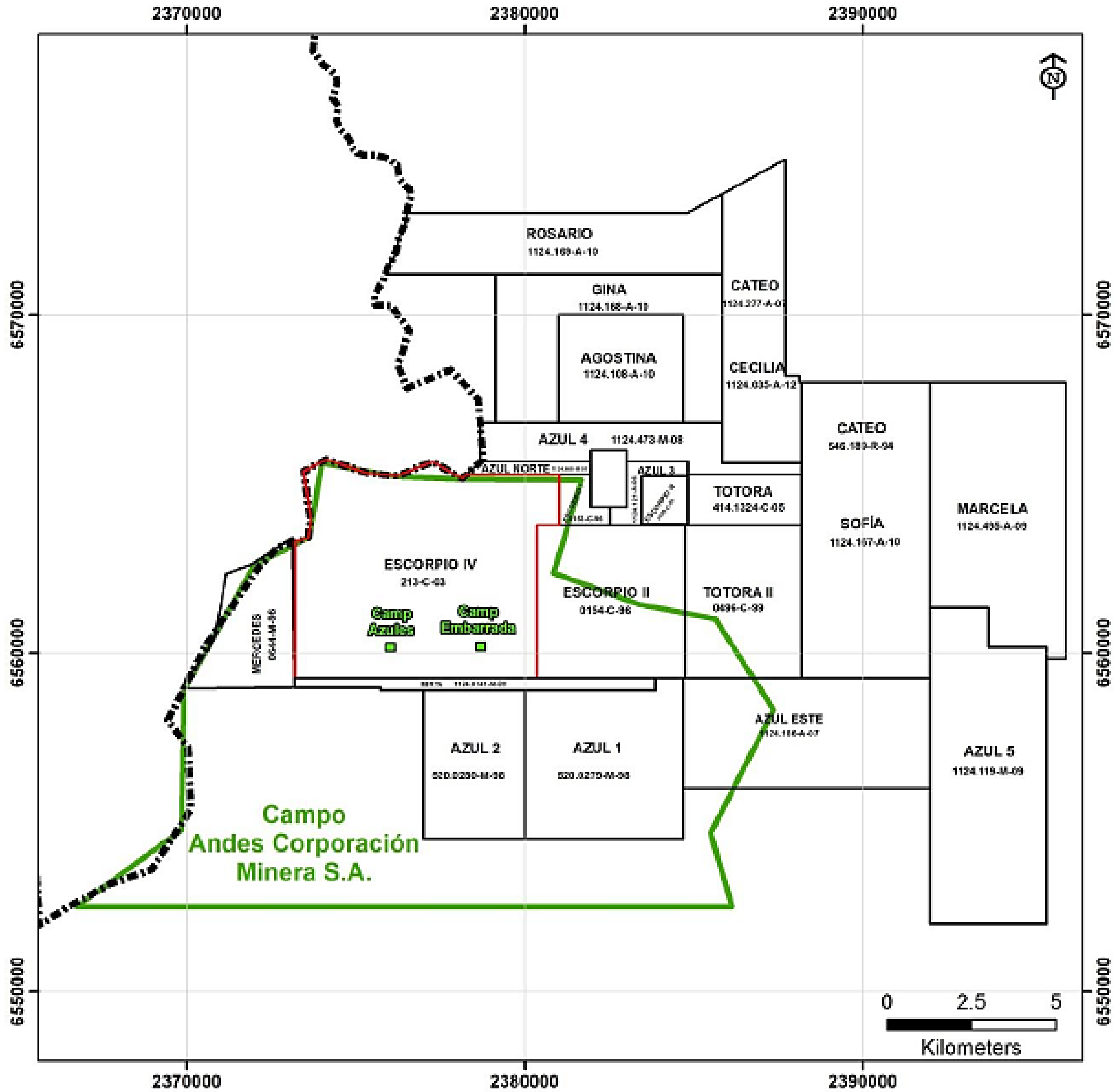


Figure 1.8: Los Azules Project Property Limits (V&G Report, 2023)

1.7 GEOLOGICAL SETTING

Los Azules is a porphyry copper deposit located in western San Juan Province in west-central Argentina. This region is characterized by a series of north-south elongated mountain ranges that rise in altitude from east to west to form the rugged Andean Cordillera along the border between Argentina and Chile.

Geology at Los Azules comprises Mesozoic volcanic rocks intruded by a Miocene diorite stock, itself intruded by a sub-parallel suite of diorite-dacite porphyry dikes along a major NNW-striking structural zone. Porphyry copper style mineralization and hydrothermal alteration are spatially, temporally, and genetically related to the diorite stock and the dikes.

1.8 MINERALIZATION

In many respects the Los Azules deposit is a classic Andean-style porphyry copper deposit. In the bedrock below a thick surface cover of scree and valley fill, a barren leached zone overlies a zone of secondary or supergene enrichment of variable copper grades and thickness. Primary or hypogene mineralization extends to at least 1,000 m below the present surface. Circulation of meteoric ground water near surface leached primary sulfides (mainly pyrite and chalcopyrite) from the host rocks over the past several million years and the leached copper was redeposited below the water table in a sub-horizontal zone of supergene enrichment as secondary chalcocite and covellite. Hypogene bornite appears at deeper levels together with chalcopyrite. Gold, silver, and molybdenum are present in trace amounts, but copper is by far the most important economic constituent at Los Azules.

The Los Azules hydrothermal alteration system is at least 5 km long and 4 km wide and is elongated in an NNW direction along a major structural corridor. The system disappears below volcanic cover to the north, so the ultimate extent is unknown. The altered zone surrounds the Los Azules deposit, which is approximately 4 km long by 2.5 km wide. The limits of the mineralization along strike and at depth have not been entirely constrained by drilling.

Recent geological studies have resulted in an updated geologic model (Mortimer, 2022) that shares many features with other well-known Andean porphyry copper deposits. These studies have defined the temporal sequence and spatial distribution of the following distinct alteration phases and mineralization zones.

- Intrusion of pre-mineral dioritic stock or pluton.
- Pervasive chlorite-magnetite alteration accompanied by chalcopyrite mineralization in the upper levels of the pluton grading into potassic alteration with chalcopyrite and bornite mineralization at depth.
- Intrusion of early mineralized porphyry dike phase and formation of magmatic- hydrothermal breccia bodies.
- Intrusion of later “inter-mineral” phase porphyry dikes and formation of magmatic- hydrothermal breccia bodies.
- Late sericite alteration accompanied by pyrite and chalcopyrite.
- Formation of erratic quartz veins containing base and precious metals.
- Supergene enrichment.

The construction methodology of the geological models is extremely robust. It breaks the deposit down into its component events and by understanding each of the controls related to that event, yields a greater understanding of the deposit and a more robust series of inter-related models. The modelling is carried out in sequence: structure – lithology – alteration – mineralization – zonation with iterative revision and reconstruction.

Overall, modelling shows that Los Azules is a large structurally controlled porphyry deposit, open towards the west, northwest and at depth. The extensive supergene enriched zone has developed down structures that transition into primary sulfide mineralization. Modelling shows multiple bornite centers within the primary zone highlighting exploration potential at depth and along the currently modelled structures.

1.9 EXPLORATION & DRILLING

Exploration at Los Azules commenced in the mid-1990's and has included various studies of geology, geophysics, and geochemistry, as well as drilling with both reverse circulation and diamond core drills, sampling and analysis of surface and drill core samples, and road construction. Exploration was conducted successively, and sometimes in cooperation, by Battle Mountain Gold, MIM-Xstrata, and Minera Andes/McEwen Mining and McEwen Copper, principally by the latter company.

Drilling programs have been undertaken at Los Azules between 1998 and 2023 by three different mineral exploration companies including BMG, MIM Argentina (now Glencore) and Minera Andes/McEwen Mining and McEwen Copper. Drilling included reverse circulation programs mostly for gold exploration and diamond drilling focusing on supergene and hypogene porphyry-style copper mineralization. Descriptions of these programs are detailed in the following sections. Table 1.2 provides a summary of the drilling information.

Table 1.2: Exploration Drilling by Year and by Company

Year	Company	No. of holes	Meters
1998 –1999	Battle Mountain Gold	24	5,681
2004	Glencore Xstrata (MIM)	4	864
2003 – 2011	Minera Andes	127	34,270
2011 – 2023	McEwen Mining	284	75,849
Total		**439(1)**	**116,664**

1. This table includes all drilling that has occurred on the property. Some holes were redrilled due to drilling difficulties and are not included in the database. Holes that were started in one season and completed the following season are counted in the year they were started, but the meters drilled in each season are shown for the respective seasons. The drilling reflects all holes to the effective date of May 9th, 2023.

1.10 SAMPLING AND ANALYSIS

The drilling programs that have occurred on the Los Azules property since 1998 have used both reverse circulation (RC) and diamond core (core) equipment. All holes drilled by BMG in 1998 and 1999 were RC type. MIM, now Glencore, drilled four RC holes in 2004. Since 2004, Minera Andes/McEwen Mining and McEwen Copper have mainly used core-drilling techniques.

McEwen sampling staff use hydraulic guillotine core splitters to split the more intact core fragments lengthwise as instructed by the logging geologist on the sampling sheet. Core is divided using a splitter to minimize the loss of sooty chalcocite, which could be lost by washing during cutting by a diamond saw. Depending on fracture density, RQD and general condition of the core, any core that is not whole or is significantly broken is divided with a trowel to obtain a reasonable sample. One half of the core is kept behind in the core box and later stored on shelving units in the warehouse for posterity and later reference.

Sampled core fragments are immediately placed in thick plastic bags labelled with unique ID codes for each sample and sealed with nylon zip ties. Usually, three to seven individually bagged samples are then placed into larger poly-woven rice bags, labelled accordingly, and secured with a uniquely numbered, tamper-proof security seal.

During the sample bagging and bundling process in Calingasta, McEwen staff inserted standards and blank samples in pre-determined regular intervals as well as duplicate samples. In a sequence of 40 samples there will be two blanks, two standards and one duplicate requested. The density of QC samples was more than adequate for the current program.

Once the samples are bagged in Calingasta, no McEwen employee is involved with any subsequent sample preparation. Samples are picked up regularly by staff from Alex Stewart International labs (ASI) and delivered directly to their labs in Mendoza. The sacks are not opened until they reach the laboratory where the inventory of sample numbers and security seals are checked and referenced to the existing Chain of Custody paperwork protocols followed to this point.

Since 2013, all sample preparation (crushing and pulverizing) and assaying was completed at Alex Stewart Labs in Mendoza. Historical assaying and sample preparation has taken place at other accredited labs in Argentina and Chile.

Alex Stewart International provides geochemical, metallurgical, and analytical services to the mining and mineral exploration industry in Africa, Europe, South America, and Asia. ASI laboratories are accredited to ISO 9001, 14001 and 17025 standards and participate in inter-laboratory tests and international round robins.

Laboratories utilized by McEwen have internal QC samples used in each batch of sampled material provided by McEwen. Each assay certificate lists the drill sample results, plus the laboratory's internal sample control results that consist of its own duplicates, blank and reference standard pulp with each batch assayed for its internal quality control on precision, instrument drift and accuracy to determine if there are any sampling issues for that run. Anomalously high values within batches are verified by re-assay as a matter of routine.

Sequential copper determinations for acid soluble copper (CuAS, copper capable of being dissolved in weak sulfuric acid solution) and cyanide soluble copper (CuCN, copper capable of being dissolved in weak sodium cyanide solution after first determination with sulfuric acid) are provided by a standard sequential copper assay methodology by the Alex Stewart laboratory. The methodology is comprised of a dilute sulfuric acid-controlled leach and assaying of the resulting dissolved copper followed by a dilute sodium cyanide-controlled leach and assaying of the resulting dissolved copper. Residual copper in the final tails

(CuRES) is processed in a four-acid digestion and assayed to determine the total copper content (CuT). Acid soluble and cyanide soluble assays are combined to determine the approximate soluble copper content (CuSOL= CuAS + CuCN)) of each sample based on the methodology. The sequential total copper assay determination is only used for comparative purposes and the total copper assayed in the drill program procedure is used in the drilling database.

Results from the control sample analysis indicate that the copper and gold assay processes are under sufficient control to produce reliable sample assay data for resource estimation and release of drill hole assay results. Inadequate standards from early field seasons were eliminated. The use of only one lab to produce assay results improves the consistency of results. Material that was assumed to be blank but contained low copper values was replaced. Later types of blank material improved the monitoring of potential contamination of the samples.

McEwen Copper decided to undertake an extensive re-assaying program of existing core to augment the database prior to the updated estimate was implemented. This program initiated in 2022 and is continuing. The re-assaying program on the historic core is aimed at obtaining missing data, benefiting from improved detection limits for deleterious elements (in particular arsenic), and to obtain sequential assay determinations where only total copper information was available.

All past deficiencies in the QC program have been addressed. The Los Azules sampling and assaying program is producing sample information that meets industry standards for copper and gold accuracy and reliability. The assay results are sufficiently accurate and precise for use in resource estimation and the release of drill hole results on a hole-by-hole basis.

1.11 MINERAL RESOURCE ESTIMATES

The mineral resources have been classified according to guidelines and logic summarized within the Canadian Institute of Mining, Metallurgy and Petroleum (CIM 2019) Definitions referred to in National Instrument 43-101. Resources were classified as Indicated or Inferred by considering geology, sampling, and grade estimation aspects of the model. For geology, consideration was given to the confidence in the interpretation of the lithologic domain boundaries and geometry. For sampling, consideration was given to the number and spacing of composites, the orientation of drilling and the reliability of sampling. For the estimation results, consideration was given to the confidence with which grades were estimated as Measured by the quality of the match between the grades of the data and the model.

The current database is adequate for the preparation of a long-range model that will serve as a basis for modeling associated with completing the PEA. The extent of mineralization along strike exceeds 4 km and the distance across strike is approximately 2.2 km. The deposit is open at depth. Over the approximately 2.5 km strike length where mineralization is strongest, the average drill spacing is approximately 150 meters to 200 meters but there are localized areas where drilling is on 100 meter spacing. The assay database considers 162 drillholes and 56,528 meters of assay interval data. Resource estimation work was performed using Datamine Studio modeling software.

As of the date of publication 47 holes and approximately 18,318 meters of drilling (mostly infill) have been completed but were not included in the database for resource estimation, data for which was cut off on date December 31st, 2022.

Overall, modelling shows that Los Azules is a large structurally controlled porphyry deposit, open towards the west, northwest and at depth. The extensive supergene enriched zone has developed down structures that transition into primary sulfide mineralization. Modelling shows multiple bornite centers within the primary zone highlighting exploration potential at depth and along the currently modelled structures.

The Indicated and Inferred resources for the enriched and primary zones are presented in Table 1.3. Mineral resources are determined using an NSR cut-off value to cover the processing cost for each recovery methodology. For supergene and primary material going to the leach pile the cutoff was \$2.74/t. For supergene going to the mill the cutoff was \$5.46/t and primary material going to the mill was \$5.43/t. The resource is further constrained by a pit shell that demonstrates the reasonable prospects of eventual economic extraction (RPEEE) of this material.

Generalized technical and economic parameters include a long-term copper price of \$4.00/lb, and a variable resource pit slope between 20° and 42° depending on depth. Other parameters used in the resource pit development are detailed in Section 14.13. Some of the deeper mineralization below the current Resource pit shell may not be economic due to the increased waste stripping requirements but future mining viability would be determined by copper price and other economic factors.

Resources are reported in two categories related to processing amenability: 1) materials that are suited for processing in a commercially proven conventional, ambient conditions, copper bio-leaching scheme (Leach); and 2) materials that are better suited to processing either in a more advanced bio-leaching scheme such as Nuton™ technology or traditional milling/concentrator approach (Leach+ or Mill).

Table 1.3: Mineral Resource Summary

			Million tonnes (MT)	Average Grade				Contained Metal		
				Cu% - tot.	Cu% - sol.	Au (g/t)	Ag (g/t)	Cu (Blbs.)	Au (Moz.)	Ag (Moz.)
Indicated	Supergene	Leach	944.2	0.46	0.30	-	-	9.54	-	-
		Mill or Leach+	73.0	0.13	-	0.09	1.10	0.21	0.20	2.58
	Primary	Mill or Leach+	218.1	0.25	-	0.036	1.06	1.19	0.25	7.43
	Total	Mill or Leach+	291.1	0.22	-	0.049	1.07	1.40	0.46	10.01
Total Indicated		Leach, Mill or Leach+	1,235.3	0.40				10.94	0.46	10.01
Inferred	Supergene	Leach	695.7	0.32	0.19	-	-	4.91	-	-
		Mill or Leach+	525.6	0.30	-	0.05	1.44	3.45	0.87	24.40
	Primary	Mill or Leach+	3,288.0	0.25	-	0.03	1.18	18.35	3.37	124.67
	Total	Mill or Leach+	3,813.6	0.26	-	0.035	1.22	21.79	4.24	149.07
Total Inferred		Leach, Mill or Leach+	4,509.3	0.31				26.70	4.24	149.07

Note: Mineral Resources do have demonstrated economic viability. No values are presented where the process recovery method does not consider this aspect of the materials. See Section 14.13 for additional qualifications.

1.12 MINING

The Los Azules deposit grades, geometry, and depth make it suitable for conventional, large-scale truck-shovel open pit mining methods. This includes the use of equipment such as blasthole drills, diesel hydraulic excavators, electric shovels, large off-highway haul trucks, and associated operations support equipment.

Two Cu cathode production rate cases were assessed during the mine engineering and planning process. A 175k tpa Cu cathode production scenario which is the 'base case' and a 125k tpa Cu cathode production scenario which is the 'alternate case'.

The detailed mine planning for this Technical Report included conventional optimization processes, phased and ultimate pit designs, surface layout planning and life of mine production scheduling.

The ultimate pit shell limit and intermediate pit shells (or phases) were developed with the use of Geovia Whittle™ pit optimization software. Using Net Smelter Return (NSR), surface restrictions / constraints, pit slope geotechnical parameters, mining parameters, and production rates resulted in a series of economic pit optimizations that were evaluated to define pushbacks and the ultimate pit. The Los Azules deposit also contains mineralized material that may be economic for producing a milled copper concentrate product, but this material was not considered for base case pit optimization or mine schedule.

Open pit mining would take place in phases from an initial starter pit, allowing for a shorter pre-strip and earlier access to mineralized material for processing. For the 175k tpa Cu cathode production base case, the material mined over the two-year pre-production period is 117.8M tonnes of which 16.8M tonnes is mineralized material that is either stockpiled or crushed and placed on the heap leach pad. There is a ramp-up in annual production to year 8, when peak material movement is reached at 175M tonnes. Material movement tonnage stays at approximately 175M tonnes through to year 12, 95.7M tonnes of lower grade mineralized material is stockpiled during these periods. The material moved decreases to approximately 75M tonnes and stays at this rate until the end of mining in year 21.

Total material stockpiled during the mine life is approximately 285.8M tonnes. Stockpiled material is re-handled, crushed, and placed on the heap leach pad at the end of the mine life in years 21 to 27.

The 125k tpa Cu cathode production alternative case follows a similar initial production profile but has a lower peak material movement of approximately 110M tonnes and an active mine life of 30 years followed by approximately 3 years of stockpile re-handle material that is crushed and placed on the heap leach pad for processing.

Figure 1.5 shows the Project site layout and specifically the location of the open pit, associated mined rock storage facilities, and process stockpile area. All mine infrastructure has a minimum offset of 50m from identified cryogenic geoforms and located to minimize haulage distances as pit mining advances.

To minimize emissions from mining operations, electrification of the mining fleet was considered during the equipment selection process. Trolley assist has been incorporated into the mine plan to allow conventional 363 tonne diesel haul trucks to travel uphill under electric power. Future studies will consider technology advancements and implement carbon reduction strategies where possible.

1.13 METALLURGICAL TEST WORK AND RECOVERY METHODS

Historical testing for McEwen Copper was conducted on samples from the resource in several phases. C. H. Plenge Laboratory (Plenge) in Lima, Peru, performed several scoping level investigations from 2008 to 2012. A mineral liberation analysis (MLA) was completed at Thompson Creek Metals Company in Challis, Idaho; in 2012 on rougher flotation samples from the Plenge lock-cycle testing. Additional samples from the resource were tested at the SGS Research Limited (SGS) to support a Preliminary Economic Assessment (PEA) by Hatch in 2017.

The historical work completed at both Plenge and SGS concentrated on evaluating sulfide resource processing options including flotation, pressure oxidation (POX) of flotation concentrate, and heap leaching. The evaluation of the historical data in the PEA in 2009 and 2010 resulted in the selection of a flotation process to produce a copper concentrate. In 2013, a change in the PEA resulted in a flotation concentrate being treated by a pressure oxidation (POX) leach circuit and heap leaching of low-grade material with solvent extraction and electrowinning (SX/EW) to produce copper metal cathodes.

Copper mineralization is complex and varied at Los Azules, as to be expected for deposits of this type. Metallurgical characterization testing has been completed as part of this study in the form of sequential assay (sulfuric acid and cyanide steps) for the resources considered, column testing and bottle roll testing. The sequential assay method used at Los Azules for both the resource assay and metallurgical programs provides an indication of the copper mineralization present in the form of acid soluble copper (CuAS) and cyanide soluble copper (CuCN), both assays combined provide an approximation for leachable/soluble copper (CuSOL) component of the total copper assay (CuT). There are several sequential assay methods in use in the industry, the methodology selected for Los Azules maintains consistency with historic assay methods within the resource data set.

The current metallurgical program consists of three concurrent phases of worked aimed at supporting a feasibility study level of investigation. In the current Phase 1 program work, existing drill core was selected for testing by lithology and material type to reflect economically processable material in the resource for this study. Phases 2 and 3 will utilize new metallurgical core obtained from the ongoing drilling program to investigate the potential metallurgical variability of the deposit and focus on the initial 3-5 years of materials to be mined.

For the Phase 1 program, assay data and bottle roll testing were completed for this study from the 21 column test samples (from nine (9) composites and two (2) drill holes) and are currently undergoing column testing using conventional acidic and bio-leach conditions. This reflects the metallurgical testing program currently being completed at SGS in Santiago, Chile (SGS).

Additionally, samples were shipped and delivered to Nuton in Melbourne, Australia to undertake scoping testing with enhanced bio-leaching techniques and over 4.5 tonnes of PQ/HQ core material from the 2022 drilling season were shipped to Hazen Research in Golden, Colorado, USA in September 2022. Nuton has set up a laboratory facility directly at Hazen, mirroring the technology and practices from their Bundoora facility in Melbourne, Australia.

As part of the Phase 1 program, samples were shipped to SEPRO laboratory in Canada to undergo microwave assisted comminution, to evaluate thermal stresses creating fractures (macro and micro).

As of the effective date of this report, the initial columns for the current Phase 1 program are still on-going pending final analyses. As such, results are considered preliminary until tail assays can be obtained to provide a calculated head assay and related recovery. The CuSOL content of a mineralized block is considered approximates the most readily leachable copper component based on the methodology used. The remaining residual copper content (portion of the assayed total copper value not included in the CuSOL determination) is partially leachable based on the metallurgical testing results to date.

A total of 9 drill holes over 4,217 m of PQ core (41,689 kg) was drilled during the 2022 campaign to support the Phase 2 Metallurgical test program at SGS; these samples were delivered in mid-October 2022. Additionally, samples from the 2022 drilling season of PQ core were sent to Hazen Laboratory in Golden, Colorado to undertake alternative leaching techniques, starting in Q1 of 2023.

Heap leach copper extraction is derived by two (2) different methods, one for leachable/soluble copper (CuSOL) and residual copper (CuRES) as derived from sequential copper assay methodology. The projected extraction for CuSOL is 100% for leachable/soluble recovery and 15% for CuRES. Residual copper assay is the difference between total assayed copper (CuT) and CuSOL. Copper recovered to cathodes considers a heap efficiency and inventory factor of 90% of the extractable copper based on general experience. Soluble copper recovery exceeding 100% implies partial leaching of material which was not categorized as "soluble" based on the sequential assaying method and data available.

The potential resource processing sources include Oxide/LIX, Supergene (sulfide enriched material) and Primary (sulfide enriched material containing primarily copper mineralization). LIX mineralization is the Spanish acronym for leached or leach cap. A summary of the resources considered for processing in the initial conventional bio-leaching project phase by source is provided in Table 1.4. Based on the resource assay data and column results, the apparent soluble copper recovery to cathodes is approximately 107%, with total copper recovery at 73%.

Table 1.4: Potential Leach Process Materials Distribution

Material Type	Potential Process Material (Mt)	Grade (CuT) (%)	Grade (CuSOL) (%)	Gold Grade (g/t)	Silver Grade (g/t)
Heap Leach Total	1,182	0.457	0.311	0.05	1.25
Oxide/LIX Material	22.2	0.079	0.051	0.04	0.80
Supergene Material	1,015	0.472	0.341	0.05	1.19
Primary Material	145	0.409	0.143	0.06	1.71

This 2023 PEA incorporates an updated development strategy for a multi-phased implementation approach. For the initial project phase, the Supergene and Oxide/LIX material is believed to be suitable for treatment in a conventional lined leach pad suitable for ambient bio-leaching sulfuric acid technology with a conventional solvent extraction (SX) and electrowinning (EW) process facility to produce copper cathodes at LME Grade A quality standards or ASTM B115-10 – Cathode Grade 1. Figure 1.9 describes the mining and processing methodology envisioned.

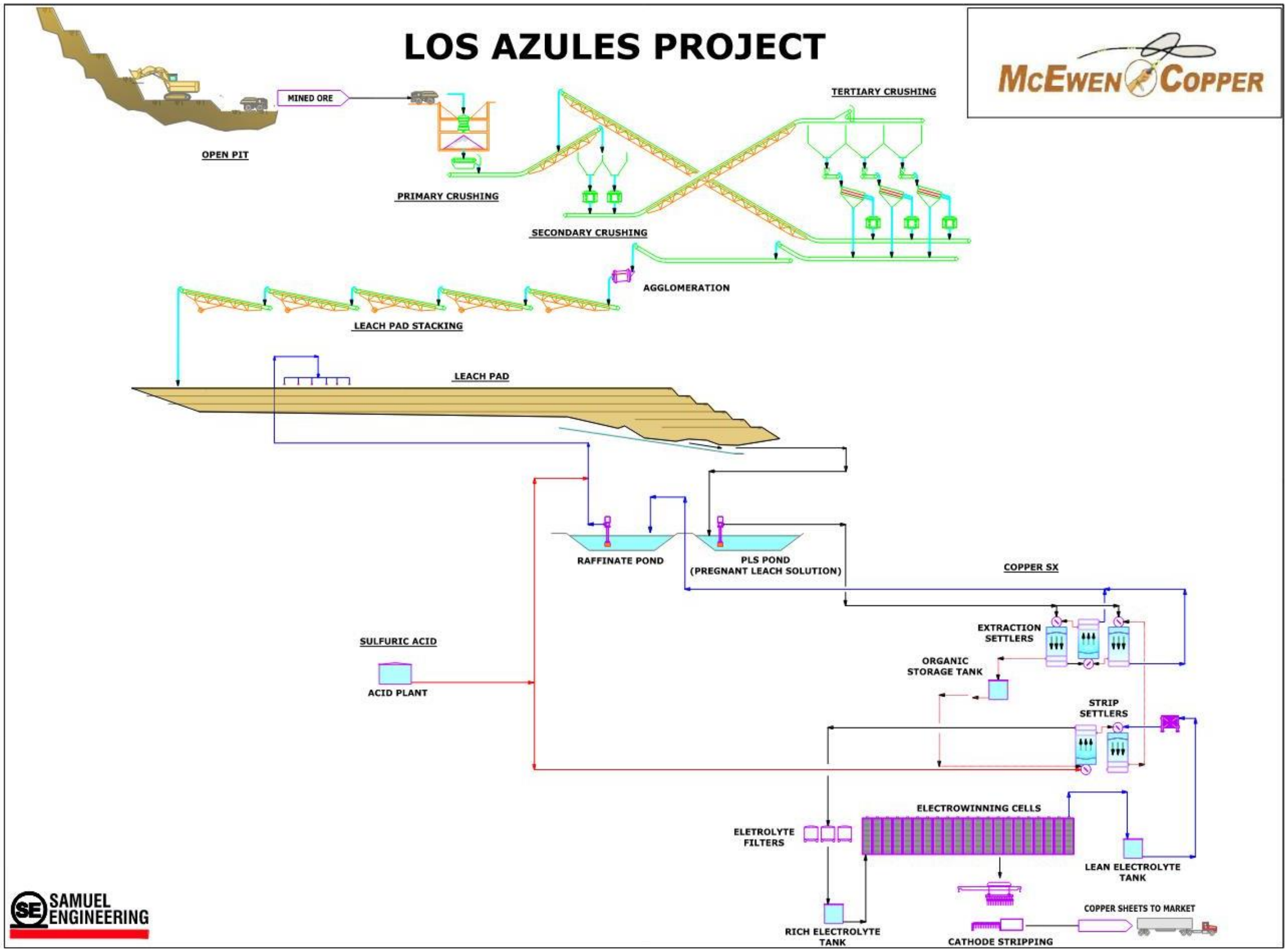


Figure 1.9: Simplified Process Flowsheet (Samuel, 2023)

The Phase 1 project Base Case option considers a processing facility to nominally produce 175,000 tonnes per annum (tpa) of copper cathodes from higher grade, highly leachable (soluble) copper content materials. An expansion of the mining rates and materials handling facilities is required by Year 7 to maintain copper production as the copper grade drops. This initial processing facility will function through to the completion of mining for the initial project phase in Year 21 with lower grade stockpile reprocessing and residual leaching operations to Year 28.

Base Case mining operations ramp up over the proposed mine life from approximately 70 million total tonnes per annum moved up to 175 million tonnes per annum moved through the life of the project as copper grades diminish, and waste stripping requirements increase.

Initial capital costs for the Base Case project includes camps and site infrastructure, mining equipment to support the initial mining rate of 80 million tonnes per annum, crushing and heap leach stacking systems to support the initial 25 million tonnes per annum throughput rate, SX/EW facilities to support 175,000 tonnes per annum nominal copper production with turn up capacity of 12%, sulfuric acid production at 200,000 tonnes per annum and heap leach facility to support placement of 79.5 million tonnes. A 220kV substation and power line will be provided by the regional utility YPF Luz to include renewable power supply. The costs are reflected in the cost of power.

Sustaining capital for the Base Case includes mine equipment additions, heap leach pad expansions approximately every 2-3 years, mine equipment additions as the pit matures, crushing plant expansions in years 4 and crushing/stacking expansion in 7, an additional SX train for increased flows in year 7, and acid plant expansions in years 4, 6 and 7 as processing rates and mining rates increase.

To demonstrate the scalability of the project at lower initial capital requirements, an Alternative Case was developed at a lower nominal copper production rate of 125,000 tonnes per annum of copper cathodes. This initial processing facility will function through to the completion of mining for the initial project phase in Year 30 with lower grade stockpile reprocessing and residual leaching operations to Year 33.

Mining operations ramp up over the proposed mine life from approximately 70 million total tonnes per annum moved up to 110 million tonnes per annum moved through the life of the project as copper grades diminish, and waste stripping requirements increase.

Initial capital costs for the Alternative Case project includes camps and site infrastructure, mining equipment to support the initial mining rate of 70 million tonnes per annum, crushing and heap leach stacking systems to support the initial 18 million tonnes per annum throughput rate, SX/EW facilities to support 125,000 tonnes per annum nominal copper production with turn up capacity of 12%, sulfuric acid production at 150,000 tonnes per annum and heap leach facility to support placement of 42.4 million tonnes. A 220kV substation and power line will be provided by the regional utility to include renewable power.

Sustaining capital for the Alternative Case includes mine equipment additions, heap leach pad expansions approximately every 2-3 years, mine equipment additions as the pit matures, crushing plant expansions in years 5, and 10, and crushing/stacking expansion in 13, an additional SX train for increased flows in year 10, and acid plant expansions in years 6, 10 and 13 as processing rates and mining rates increase.

All Primary copper mineralized material mined during the initial project phases will be stockpiled for future processing routes that may include a mill/concentrator or alternative bio-leaching technologies.

1.14 LOCAL RESOURCES AND INFRASTRUCTURE

The nearest settlement is the town of Calingasta, which is located approximately 80 km east of the Project. The road from Calingasta to the Project is 120 km over mostly gravel roads. Approximately 30 km of newly constructed road and upgrades to approximately 60 km of existing road will be required for the Project.

Calingasta is a historic mining town that was based on exploitation of alum (aluminum sulphate) and gold deposits. The principal current economic activity of the area is agriculture with fruit trees (apple and walnut). According to a 2015 estimate, the population of the department of Calingasta is 9,151 inhabitants and according to the 2010 census, the town of Calingasta had 2,700 inhabitants.

Water

Surface water is available on the property in adequate amounts for McEwen Mining's exploration activities. Preliminary hydrological and meteorological evaluations have indicated sufficient water exists for the proposed Los Azules mining and processing facilities and to provide the necessary fresh water needed to house employees at the mine site.

Table 1.5 provides the current estimates for potential water sources on the Los Azules property. Dewatering of the open pit is required to maintain pit wall stability and flooding as the mine is in a valley watershed.

Table 1.5: Los Azules Water Sources

Source	Rate	Comments
Average Surface Water flows	438 L/s	Based on 2001 to 2021 meteorology data
Surface Water flows – drought conditions	275 L/s	Based on 2016 to 2021 meteorology data
Estimated pit dewatering	525 L/s	Mar 2023 PEA pit; Ausenco 2011 hydro parameters
Average Total Water Flows	963 L/s	Combined Surface & Pit Dewatering Flows
Drought Conditions Total Water Flows	800 L/s	Combined Surface Drought & Pit Dewatering Flows

For the Base Case Phase 1 project the estimated average consumption is approximately 113 liters per second (L/s) in the initial years and 163 L/s in the later years as mining progress and processing throughputs increase. For the Alternate Case, the estimated average consumption is approximately 95 liters per second (L/s) in the initial years and 152 L/s in the later years as mining progress and processing throughputs increase. A more detailed evaluation of available water resources will need to be undertaken for a feasibility study.

Power

Initially, the Base Case project will require approximately 89 MW of power for all site activities. The ultimate average power requirement for the initial phase of the project increases to approximately 131 MW. The Alternate Case project will require approximately 74 MW of power for all site activities. The ultimate average power requirement for the initial phase of the project increases to approximately 117 MW.

Power will be supplied from the Argentinian grid via an initial 220 kV overhead transmission line approximately 120 km long and tie into the existing utility substation at Calingasta. The transmission line route will be parallel to the existing Exploration Road to Los Azules site and construction of 220kV/175MVA substation at the site. The expansion of the existing Rodeo and Calingasta 500kV substations connected to the national grid will also be necessary. The work will be carried out under the initiative and responsibility of National power provider YPF Luz. The estimated YPF investment is USD $155 million for their installation of the powerline and substation facilities, including the site substation.

YPF Luz has provided a referential rate sheet for a Power Purchase Agreements (PPA) contract to obtain power sourced exclusively from renewable resources. The power would be sourced from solar and hydroelectrical generation. The rate would depend on the contract term and would be a take or pay contract with a minimum power contract under payment. A rate of $0.065/kWh based on a minimum 10-year term is considered in this PEA.

Additionally, the proposed on-site sulfuric acid plant will be outfitted with a turbogenerator for power generation from the off gases to provide approximately 20% of the site requirements during operation. On-site solar power supplemental systems at the camp and other facilities are considered and will further minimize the grid power usage.

1.15 PROJECT ECONOMICS

The project initial capital costs are based on budgetary cost quotations and regional contractor estimates for major equipment and facilities obtained in Q4 2022 and Q1 2023. The capital costs for the project are summarized below and should be viewed with an expected level of accuracy for a preliminary analysis at +40%/-20% consistent with AACE International Recommended Practice No. 47R-11 Estimate Class 5. Owner's Costs include the initial mine fleet, preproduction stripping costs and preoperational costs for early crushing and material placement on the leach pad.

Table 1.6: Initial Capital Costs by Case

Capital Cost Level 1 Summary	**Base Case 175k tpa Cu**	**Alternative Case 125k tpa Cu**
WBS Area	Total (USD)	Total (USD)
100 - Mining	$65,600,000	$65,600,000
200 - Ore Storage & Handling	$234,500,000	$192,500,000
400 - Heap Leaching	$158,500,000	$142,100,000
500 – SX/EW Facilities	$250,400,000	$167,700,000
600 - Acid Plant	$94,900,000	$79,900,000
800 - Ancillary Facilities	$23,300,000	$23,300,000
900 - Site Development & Yard Utilities	$126,300,000	$112,200,000
2000 – Off-Sites	$167,400,000	$167,400,000
Total Direct Costs	**$1,120,900,000**	**$950,700,000**
Common Indirect Costs	**$379,200,000**	**$323,800,000**
Owners Costs	**$466,700,000**	**$455,900,000**
Subtotal	**$1,966,800,000**	**$1,730,400,000**
Contingency	$495,000,000	$423,100,000
Total Capital Cost	**$2,461,800,000**	**$2,153,500,000**

The project life of mine direct operating costs per tonne processed and per pound of copper produced for the two cases developed are summarized below. Costs vary with open pit development, feed head grades, acid requirements in leaching, power consumption increases over time and actual copper production.

Table 1.7: Life of Mine Operating Cost Summary				
OPEX SUMMARY	Life of Mine	Units	Base Case 175k tpa Cu	Alt. Case 125k tpa Cu
Mining OPEX	Per Lb Cu	$/lb Cu	$0.56	$0.57
	Per ton processed	$/t	$4.14	$4.27
Processing OPEX	Per Lb Cu	$/lb Cu	$0.37	$0.37
	Per ton processed	$/t	$2.73	$2.74
G&A	Per Lb Cu	$/lb Cu	$0.13	$0.15
	Per ton processed	$/t	$0.94	$1.11
Selling Costs	Per Lb Cu	$/lb Cu	$0.02	$0.02
	Per ton processed	$/t	$0.15	$0.15
TOTAL OPEX (C1 Costs)*	**Per Lb Cu**	**$/lb Cu**	$1.07	$1.11
	Per ton processed	**$/t**	$7.96	$8.27

*Note: Numbers may not add exactly due to rounding

Based on consensus estimates and independent analysis, long-term metal pricing used in this report (except for mineral resource estimation) and project economic analysis are Copper (Cu) - $3.75/pound; Gold (Au) - $1,700/ounce; and Silver (Ag) - $20.00/ounce. The 2023 updated financial outcomes for the Phase 1 initial project mine and facilities are shown in Table 1.8 below (expressed in Q1 2023 United States Dollars). **C1 cash costs are defined as the cash cost incurred at each processing stage, from mining through to recoverable copper delivered to the market, net of any by-product credits.** C1 cash costs per pound of copper produced and all-in sustaining costs per pound of copper produced are non-GAAP ratios.

Table 1.8: Project Economic Summary by Case			
Project Metric	**Units**	**Base Case 175k tpa Cu**	**Alt. Case 125k tpa Cu**
Mine Life	Years	27	32
Processing Life	Years	28	33
Tonnes Processed	Billion tonnes	1.182	1.182
Tonnes Waste Mined	Billion tonnes	1.366	1.366
Strip Ratio		1.16	1.16
Total Copper Grade	% Cu	0.457%	0.457%
Soluble Copper Grade (CuSOL)	% CuSOL	0.311%	0.311%
Copper Recovery (Total Copper)	%	72.8%	72.8%
Soluble Copper Recovery[1]	%	107%	107%
Copper Production (LOM avg.)[2]	tonnes/yr	145,850	123,060
Copper Production (Yr 1-5)	tonnes/yr	182,100	136,100
Copper Production – cathode Cu	ktonnes	3,938	3,938
Initial Capital Cost	USD Millions	$2,462	$2,153
Sustaining Capital Cost	USD Millions	$2,243	$2,351
Closure Costs	USD Millions	$180	$180
C1 Costs (Life of Mine)	USD/lb Cu	$1.07	$1.11
All-in Sustaining Costs (AISC)	USD/lb Cu	$1.64	$1.67
Initial Capex/tpa (LOM avg.)	USD/tpa Cu	$16,880	$17,500
LOM Capex/LOM tonnes Cu	USD/tonne Cu	$1,195	$1,143
Before Taxes			
Net Cumulative Cashflow	USD Millions	$15,819	$15,679
Internal Rate of Return (IRR)	%	26.5%	22.9%
Net Present Value (NPV) @ 8%	USD Millions	$4,436	$3,278

Table 1.8: Project Economic Summary by Case			
Project Metric	**Units**	**Base Case 175k tpa Cu**	**Alt. Case 125k tpa Cu**
After Taxes			
Net Cumulative Cashflow	USD Millions	$10,240	$10,159
Internal Rate of Return (IRR)	%	21.2%	18.4%
Net Present Value (NPV) @ 8%	USD Millions	$2,659	$1,929
Pay Back Period	Years	3.2	3.4

Notes:

1. Soluble copper recovery exceeding 100% implies partial leaching of material which was not categorized as "soluble" based on the sequential assaying method and data available.
2. Life of Mine production averages includes low grade stockpile rehandling and leaching production at the end of mining material from the open pit for each case.

A summary of the 8 Year and LOM Cash Cost can be seen in Table 1.9.

Table 1.9: Average First 8 Years and LOM Cash Costs*					
Category	**UoM**	**175k tpa Cu Production Case**		**125k tpa Cu Production Case**	
		First 8 years	**LOM**	**First 8 years**	**LOM**
Gross Revenue	**US$/lb Cu**	**3.75**	**3.75**	**3.75**	**3.75**
Selling Expenses	US$/lb Cu	(0.02)	(0.02)	(0.02)	(0.02)
Mining Cost	US$/lb Cu	(0.53)	(0.56)	(0.57)	(0.57)
Processing Cost	US$/lb Cu	(0.23)	(0.37)	(0.22)	(0.37)
Local G&A	US$/lb Cu	(0.10)	(0.13)	(0.14)	(0.15)
C1 Costs	**US$/lb Cu**	**(0.88)**	**(1.07)**	**(0.95)**	**(1.11)**
Unrecovered VAT	US$/lb Cu	(0.04)	(0.01)	(0.04)	(0.01)
Royalty	US$/lb Cu	(0.27)	(0.30)	(0.25)	(0.30)
C3 Costs	**US$/lb Cu**	**(1.19)**	**(1.38)**	**(1.24)**	**(1.42)**
Sustaining Capex	US$/lb Cu	(0.52)	(0.26)	(0.42)	(0.25)
All-in Sustaining Costs	**US$/lb Cu**	**(1.71)**	**(1.64)**	**(1.66)**	**(1.67)**
AISC Margin	%	54%	56%	56%	55%

*Note: Numbers may not add exactly in every case due to rounding

1.16 KEY PROJECT RISKS & OPPORTUNITIES

This subsection was prepared by J. L. Sorensen, FAusIMM, QP, Samuel Engineering and reviewed by the respective QP's for each area.

The Technical Report is prepared in accordance with the requirements set forth by Canadian National Instrument 43-101 ("NI 43-101") for the required disclosure of material information and is intended to meet the requirements considered for a Preliminary Economic Assessment (PEA) level of study and disclosure as defined in the regulations and supporting reference documents.

Based on the results of this preliminary assessment, contributing authors have identified important risks and opportunities related to the Los Azules project development. Below are what is believed to be the most significant "key" risks and opportunities. A complete list and description of the interpretations, conclusions, and recommendations to advance the Project is provided in Section 25.

Risks

- The Project is at the exploration stage of investigation; consequently, this study is at the scoping level of accuracy, preliminary in nature, and includes Inferred mineral resources in the conceptual mine plan and the mine production schedule. Inferred mineral resources are considered too speculative geologically and in other technical aspects to have the economic considerations applied to them that would enable them to be categorized as mineral reserves under the standards set forth in NI 43-101.
- Significant additional investigation and work is required to improve the confidence level of the analysis to support a project development decision. There is no certainty that the results, project development plans or estimates in this PEA will be realized.
- Maintaining the necessary engagement, project development and investment plans in the Los Azules project is necessary to maintain the mining rights.
- Potential new national laws under consideration by the Federal Government concerning the disturbance of wetlands ("vegas" in the local terminology) in Argentina is a significant risk if enacted prior to permitting completion. The established permitting processes consider impacts and mitigations on a case-by-case basis within each Province, whereas a national law could restrict case by case and Provincial laws and processes.
- The requirement to avoid impacting localized rock glaciers poses a risk to longer term mining opportunities, including some of those in the Phase 2 options considered in this report. Site investigations to confirm the characterization of the known geomorphologic structures should be completed in continued field programs to appropriately evaluate them and determine if avoidance impact constraints should apply.
- Geologic modeling of the deposit rock types, lithologies and other aspects have been improved in the work described in this report. However, a more robust geologic modeling effort is required to support an adequate understanding of the deposit and next stage of study.
- Limited information is available on the geotechnical characteristics and hydrology/hydrogeologic conditions affecting the open pit design and pit slopes, leach pad foundation design, water resources and management, and other site facilities. These areas pose both a risk to the facilities considered in this document and areas for potential opportunity.
- The exact location of the project development surface facilities is yet to be finalized and requires condemnation, hydrogeologic and geotechnical site investigations to support detailed design work to be performed.
- The preliminary nature of the metallurgical and geo-metallurgical aspects of the deposit poses a risk to the metallurgical performance expectations considered. Significant additional work is required to improve the confidence level of the analysis to support a project development decision.
- Pricing and delivery estimates for equipment and materials reflect current conditions and impacts related to inflation, geopolitical factors, and supply chain disruptions. While some of these impacts are easing, future impacts to market conditions are not considered in this analysis.
- Metal price assumptions were considered based on current market conditions at the time of the report and pose both a risk and opportunity to future economic expectations.

<u>*Opportunities*</u>

- The resource is presently limited by the drilling and associated information developed to date. Resources with limited drilling information due to access in the areas under the vegas is an opportunity to increase the near surface Indicated resource base within the current deposit. Additionally, opportunities for expansion of the resource base peripherally and at depth are apparent from the work completed. These should be investigated during the feasibility study drilling program.
- The open pit is presently constrained by the requirement to avoid impacting localized cryogenic geoforms currently identified as rock glaciers. Site investigations to confirm the character of these geomorphologic structures should be completed during field programs. A longer-term opportunity may exist to reclassify areas where no evidence of glacial activity is found.
- Within the glacier constraints, limited information is available on the geotechnical characteristics and hydrogeologic conditions affecting the open pit design and pit slopes. Generalized technical parameters include a variable pit slope between 30° and 42° depending on depth. Additional work to better understand these key areas represents an opportunity to reconsider the mine design parameters, potentially reducing stripping requirements and allow access to more of the deposit resources by extending and deepening the open pit. An initial analysis indicates that 10% to 15% additional resource is possible with a significant decrease in stripping required as the mine extends past the current base case.
- Materials inaccessible due to the glacier constraints should be considered in the context of alternative mining methods to the open pit.
- While not an option that will be pursued in favor of less impactful technologies, primary copper mineralization resources can be processed by conventional mill/concentrator methods to produce copper concentrates for sale. This option is accretive to the Phase 1 project based on the preliminary analysis completed.
- Incorporation of developing leaching technologies has the potential to improve copper recovery, reduce leaching times and minimize acid consumption requirements. Nuton™ technology is currently being evaluated in this capacity. This would also have the potential to unlock the primary copper resources more economically versus a mill/concentrator alternative and negate the need for a tailings storage facility.
- Pricing and delivery estimates for equipment and materials reflect current conditions and impacts, if supplier delivery times return to more typical durations at the time orders are placed, up to 6-12 months may be improved on the current execution timeframe.
- Direct inclusion of sulfur with the leaching material going to the leach pad to support acid consumption requirements, and limit acid plant expansions is being investigated.
- Strategies for recovery of contained gold may add value. Preliminary leaching tests indicate that a significant amount gold is recoverable with conventional technology. Recovery techniques should consider separate processing of higher-grade gold bearing materials considering pre- and post-copper leaching extraction methods.

1.17 QUALIFIED PERSONS RECOMMENDATIONS AND CONCLUSIONS

This subsection was prepared by J. L. Sorensen, FAusIMM, QP, Samuel Engineering and reviewed by the respective QP's for each area.

Based on the results of this preliminary assessment, contributing authors recommend that McEwen Copper complete additional work to further de-risk the Project, including more advanced stages of drilling to complete the work necessary for a Feasibility Study based on the findings of this Technical Report. Key issues and items are included below. A complete list of these tasks and, summary of the interpretations, conclusions, and recommendations to advance the Project are provided in Section 3 and Section 26.

Adequate work has been completed through the prior studies to define the project options going forward to select an option for Feasibility Study delineation, however metallurgical, geotechnical, geological, hydrological, and other aspects are not developed beyond a PEA level of study at this time.

Given the resources developed to date, project technical options considered and permitting basis to date, a Preliminary Feasibility Study (PFS) is considered an optional step and a Feasibility Study (FS) level of project definition is recommended to expedite the project development timeline and to also comply with the requirements of the property ownership agreements with the Provincial Mining Ministry.

As of the effective date of this report, the initial Feasibility Study resource drilling, geotechnical, hydrogeologic and metallurgical test work programs were started and in progress. The recommended technical program to complete the work deemed necessary to support the completion of a Feasibility Study is as follows:

- Complete an in-fill resource definition drilling program targeting Measured resource classification for the initial five years of the project and areas within the initial project supergene resource to Indicated classification as considered in this PEA. The program delineated for execution includes an additional 32,000 meters of diamond drilling with the objective of converting the resource classification for the initial 5 years of mining to predominantly Measured from the Indicated and Inferred resources defined in this report.
- Complete the site geotechnical, seismic, glacier, hydrology and hydrogeologic investigations to a feasibility study level of definition. The program delineated included 16,000 meters of geotechnical drilling, 9,250 meters of hydrogeologic drilling, 9,700 meters of condemnation and other miscellaneous drilling, reestablishment of local surface water monitoring and field surveys.
- Complete confirmatory metallurgical test work and geometallurgical definition for the initial project process. The program delineated includes 6,000 meters of additional metallurgical PQ core (and/or equivalent HQ core) drilling and sampling to obtain approximately 90 tonnes of material, additional column leaching metallurgical testing for both conventional and augmented bio-leaching technologies. The metallurgical work includes site testing of the leach concepts with materials from the bulk sampling campaign. Additional testing on primary mineralization materials for potential milling options is also considered.
- Update resource/geologic models and estimations, mine plans and schedules based on the additional data collected.
- Update leach pad, processing and site/off-site infrastructure facilities designs to feasibility level development and support ongoing permitting requirements. Finalize concepts for power supply, site access and logistics.
- Confirm critical consumables availability and pricing, including sulfur and sulfuric acid, fuels and

water.

- Update execution plans, costs and financial estimates and assumptions based on the updated project definition.
- Expand the inclusion of Regenerative design considerations to further improve the carbon footprint and social handprint features of the project.

A Feasibility Study level of definition is estimated to take 18-20 months to complete from the effective date of this report and assuming continuance of the work areas in progress.

Based on current information from work in progress, the estimated cost is approximately $232 million including estimates for McEwen Copper/ACMSA costs. The study cost areas are broken out in the below Table 1.10 for the recommended program to complete the Feasibility Study and other expenditures planned during the same timeframe except where noted. As of the date of publication, approximately $43.4 million has already been incurred in 2023, which should be deducted from the 2023 total shown for a forward-looking cost estimate.

Table 1.10: Expected Costs for Feasibility Study Development

Cost Category (USD Millions)	2023	2024	TOTAL
Drilling*	$48.3	$32.8	**$81.1**
McEwen Copper/ACMSA/McEwen Mining	$21.7	$16.5	**$38.2**
Camps/Site Services/Roads*	$34.6	$13.7	**$48.4**
Feasibility Study/Engineering	$9.8	$13.5	**$23.3**
Calingasta Development	$1.5	$0.1	**$1.6**
Contingency	$2.0	$8.0	**$10.0**
Cost	**$117.9**	**$84.7**	**$202.6**
Estimated VAT*	$18.8	$10.9	**$29.8**
Total	**$136.8**	**$95.6**	**$232.4**

Note: * Items account for costs only attributable to the Feasibility Study and do not extend through December 2024.

2.0 INTRODUCTION

2.1 2023 PEA UPDATE OVERVIEW

This Technical Report is prepared for McEwen Mining Inc. (McEwen) trading for the purposes of disclosing current updates and information related to its Los Azules copper property located in Argentina. The Technical Report is prepared in accordance with the requirements set forth by Canadian National Instrument 43-101 ("NI 43-101") for the required disclosure of material information and is intended to meet the requirements considered for a Preliminary Economic Assessment (PEA) level of study and disclosure as defined in the regulations and supporting reference documents.

The Project is at the exploration stage of investigation; consequently, this study is at the scoping level of accuracy, preliminary in nature, and includes Inferred mineral resources in the conceptual mine plan and the mine production schedule. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves under the standards set forth in Canadian National Instrument 43-101. There is no certainty that the estimates in this PEA will be realized.

Prior PEA reports were completed for the Los Azules property in 2009 and updated in 2010, 2013, 2017 as noted below.

- CANADIAN NATIONAL INSTRUMENT 43-101 Technical Report in Support of the Preliminary Assessment on the Development of the Los Azules Project, San Juan Province Argentina prepared by Samuel Engineering, Inc. Greenwood Village, Colorado USA effective March 19, 2009.
- CANADIAN NATIONAL INSTRUMENT 43-101 Technical Report Updated Preliminary Assessment Los Azules Project, San Juan Province, Argentina prepared by Samuel Engineering, Inc. Greenwood Village, Colorado USA effective December 16, 2010.
- CANADIAN NATIONAL INSTRUMENT 43-101 Technical Report Los Azules Porphyry Copper Project, San Juan Province, Argentina prepared by Samuel Engineering, Inc. Greenwood Village, Colorado USA effective August 1, 2013.
- CANADIAN NATIONAL INSTRUMENT 43-101 Technical Report Los Azules Porphyry Copper Project, San Juan Province, Argentina prepared by Hatch effective September 1, 2017.

This Technical Report provides information related to an updated resource estimation, revised processing flowsheet concepts and updated project economics since the last previously disclosed NI 43-101 Technical Report in 2017. This report has been prepared in collaboration with McEwen to assess the current potential economic viability of the Los Azules property.

This 2023 PEA update supersedes the prior reports and reflects a revised development philosophy, processing flowsheet, updated resource model and estimations, metallurgical information, mine plans and economic parameters such as current capital and operating costs and associated new financial inputs and model. The main differences between this 2023 updated study and the most recent previously disclosed 2017 Technical Report are:

- Updated post-pandemic costs and financial metrics.
- A multi-phased implementation approach to the Los Azules development with a focus on sustainable and regenerative design approaches to the project execution and operation.
- 100% renewable energy power sourced from an Argentinian provider.
- Broader consideration of heap leachable copper resources than previously studied.

- Revised site general arrangement for initial heap leach operations and future mill operations with filtered dry deposition of tailings for the life of operations.
- On-site generation of sulfuric acid supplied with sulfur supplied from local Argentinian sources and waste heat capture for on-site power generation of a portion of the site requirements and make-up water heating.
- Revision to production of copper cathodes initially and future copper concentrate production.
- Supplying the Argentinian refined copper consumption needs, off-take options with investor groups (Stellantis & Nuton) and exporting of copper cathodes and/or concentrates directly through ports in Argentina or Chile as the preferred logistics solution.

2.2 A NEW VISION AND APPROACH TO SUSTAINABLE COPPER PRODUCTION

One of the most important areas of the industrial economy is the mining sector, responsible for the extraction of the essential metals and minerals that make the technological world work. This sector is critical to the functioning of a modern society and the core building blocks of so much of what we use in everyday life, now and in the future. Copper will be a key ingredient of technological solutions to give the world a chance to turn things around with respect to climate impact and change in as short as possible a time frame by supporting a supply chain for the automotive sector as it transitions to electric cars and to the energy sector as it moves to more renewable forms of energy.

The Los Azules copper deposit is one of the largest known undeveloped copper resources in the world. A responsible, carbon positive mine at Los Azules could become a beacon of hope for a new form of industry that does more good than harm while enabling upstream supply chain transformation. Los Azules aspires to be the world's first Regenerative Copper Mine, providing valuable materials for a renewably powered world.

Guiding principles were developed in conjunction with McLennan Design, a regenerative architecture, planning, design and product design practice focused on deep green sustainability, community and education, to reframe the approach to sustainable innovation within the mining industry and set forth high-reaching goals that are being explored for all facets of the mining processes considered for Los Azules.

Mining and metals are a fundamental aspect of human existence, and this activity has had a significant impact on the planet and people's health over the last few centuries – often contributing to significant negative impacts to air, water, soil and the life of people and many other species. Copper mining is no different and copper is one of society's most important and versatile metals, used in nearly every sector from transportation to construction and with growing importance for a sustainable future based on electrification, renewable energy, and battery powered transportation. The International Copper Association (ICA) analysis based on 2018 data, copper production represents approximately 0.2% of global greenhouse gas (GHG) emissions, and while this contribution to global GHG emissions is currently low, copper demand is expected to double by 2050, driven in part by the need for copper for the clean energy transition.

As the world begins to decarbonize and move to a renewable future, copper will play a central role in moving beyond our industrial, fossil fuel-based legacies. It is not a question of whether we need copper in the decades ahead, but rather a question of the nature and magnitude of impacts associated with its mining, processing, and reuse and whether it contributes to a climate friendly future.

As a result, there is a huge transformation beginning, as society starts to take appropriate actions such as significantly reducing greenhouse gas emissions and the decarbonization and elimination of all fossil fuels projects like Los Azules can be a model of hope and inspiration. With positive economic and environmental models showing that rapid transformation is possible and economically successful it can only embolden larger steps by corporations, institutions, and entire countries to do better.

Rethinking Everything

It is therefore time to rethink the copper mine – and to reinvent all the processes, technologies and equipment that goes into producing it. The Los Azules site is a remote and hard to access area – without infrastructure and existing activity – is an opportunity to rethink and renew the mining process in every way. Doing so requires changing conventional thinking and reinventing an industry that has done things a certain way for many decades.

In this context, the mining industry faces strategic long-term risks for continued production; to reduce water and energy consumption and emissions in line with social, environmental, and economic pressures; and, as part of a sustainable development strategy, to understand the links between water, energy, and emissions so that an improvement in one area does not result in a greater adverse effect in another.

The timeframe of the development of Los Azules is right in the cusp of this transition. Not only will it be harder to get approvals and goodwill without doing the right thing, but it is also morally untenable to pursue a major industrial activity without significant changes undertaken, given the science on climate change and being part of the future solution instead of continuing old ways. The world's greenest and most innovative mine will be a socially responsible copper mine that creates immense economic value and results in a net positive impact to the environment and local communities.

The project concepts also allow for early adoption of emerging technologies under development and anticipated to be commercially viable over the mine life. Envisioning the ideal outcome, articulating that outcome into a goal, and then making decisions backwards through time that guarantee its attainment within the desired timeframe. Backcasting is a visionary process that will produce the blueprints for the mine of the future by envisioning the ideal end game and steps to achieve those objectives. By being 'future ready' we are poised to adopt newly emerging technologies and infrastructure rather than locking ourselves into old solutions.

In keeping with the Regenerative project approach and use of modern, less impactful technologies, the change in the processing approach from conventional milling/concentration to heap leaching and hydrometallurgical recovery of copper as a final product is a significant step towards reducing impacts while delivering a profitable copper producer into the world demand. The Nuton™ leaching technology presents one such developing technology opportunity to continue the initial copper leaching operations and significantly reduce energy, land and water requirements over conventional milling/concentration as the project moves into processing the primary copper mineralization materials.

Continued implementation of newer and less impactful technologies, fully electric mine equipment and broader employment of regeneration applications throughout the mine site to further off-set carbon emissions is expected to deliver on McEwen's commitment to achieve net-zero carbon emissions from the Los Azules project by 2038, well ahead of its peers.

Key project initiatives included aimed at achieving these goals are described below.

Respecting the Lands We Use

The Los Azules project is committed to responsible stewardship of the land and minimizing disturbance of local geo-morphologies and wetlands ("vegas" in the local terminology). The environmental baseline data on surface and groundwater volumes and quality as well as the flora and fauna data collection and additional studies on the vegas (including a compensation proposal) have been conducted since 2013 by the Insituto de Investigaciones Hidraulicas, a research center of the National University of San Juan, through their senior biologists Juan C. Acosta and Hector J. Villavicencio.

Careful consideration of how activities are conducted and where they are located is a key aspect to meeting these commitments, both in the short and long term. Minimizing land use and disturbance by consolidating uses to the extent possible is considered in the site layouts, individual site areas, facilities/buildings, and access to the mine site. In the case of the larger buildings, locations, and roof lines to capture solar energy and water from precipitation extends the use of the disturbances.

In 2010 Argentina passed the National Glacier Protection Law No. 26.639. It bans all activities (i.e., extractive such as mining or oil, tourism, and general infrastructure) on cryogenic geoforms, i.e., landforms that contain ice. The intent of the law is to protect potential drinking water supply sources. Technically speaking the laws create significant uncertainty because it bans activities in glacial and periglacial environments under the supposition that landforms in these environments contain ice and regulate hydrologic flows. The law appointed the National Glacier Institute or Instituto Nacional de Glaciologia and Nivologia (IANIGLA for its Spanish acronym) as the entity responsible for inventorying and classifying the glaciers in the country. The law provided IANIGLA with a six-month period to carry out the studies. IANIGLA organized the studies in three phases. It took IANIGLA eight years to complete the first of three phases and there is no chronogram proposed for the completion of the two additional phases.

Glaciers are morphologically classified as uncovered (“white”), covered, and rock glaciers. There are no uncovered or white glaciers within the boundaries or in the immediate vicinity of the project area. This is important because there would be no impact on any glaciers or ice from dust or emissions from the activities at the site. There are several small rock glaciers in and near the Project area as determined by IANIGLA. The mine development and associated facilities at Los Azules have been considered such that none of the rock glaciers located within the property area are impacted by the operations over the life of the mining activity, including buffer zones to ensure incidental impacts are not possible. Baseline monitoring of the glaciers continues and will be conducted throughout the mining timeframe.

Although vegas will be impacted, re-routing and diversion of water courses to downstream connections is a key design feature for the mine and leaching areas to minimize these impacts. The leach pad design includes an underdrainage for non-contact water coming from upstream sources to flow through the same valley and to the Rio Salinas. Longer term, the water courses will be restored during reclamation of the mine site at the completion of activities to bring the area as close to its original state as possible.

Transforming Water Use and Quality

Climate change, population growth and the industrial and agricultural use of water are some of the factors that affect water availability. In addition, the expansion of urban infrastructure exerts pressure on the quantity and quality of natural water courses. Long-term water solutions must be flexible, adaptable, and environmentally sustainable and work within the ‘carrying capacity’ of its place and climate. Increasing the efficiency of water use is equivalent to increasing its productivity or, in other words, reducing the intensity of its use by maximizing the value of its uses and, in this way, improving its allocation among different competing uses.

The supergene copper mineral resources at Los Azules can be processed with the use of either concentration (grinding and concentrate flotation) or hydrometallurgical (heap leaching and solvent extraction/electrowinning recovery to cathodes or “LX/SX/EW”) technologies currently in use. The Chilean Ministry of Mining, Comisión Chilena del Cobre (COCHILCO) published a report concerning water usage in the Chilean copper mining industry titled “Water Consumption in the Copper Mining Industry, 2017”. The report characterizes the relative water usage and usage intensity per tonne of material processed for copper mining projects in Chile based on its survey in 2017.

Relative intensity data is summarized in their report in Table 2.1 for both large scale (those that process at least 8,000 tpd and annual output of 50,000 tons of fine copper) and medium sized operations. The Chilean coper projects are the most comparable to Los Azules in terms of climate and geography.

Table 2.1: Water Usage Unit Coefficients in Copper Mining by Size of Operation, 2012-2017 (COCHILCO 2018)

CONCENTRATION	UNIT	2012	2013	2014	2015	2016	2017	Average
LARGE SCALE MINING	m^3/ton mined	0.59	0.57	0.53	0.50	0.49	0.44	0.52
MEDIUM SCALE MINING	m^3/ton mined	0.88	0.85	0.59	0.89	0.73	0.65	0.77
HYDROMETALLURGY								
LARGE SCALE MINING	m^3/ton mined	0.10	0.10	0.08	0.07	0.09	0.1	0.09
MEDIUM SCALE MINING	m^3/ton mined	0.10	0.06	0.15	0.25	0.27	0.25	0.18

As can be seen in the COCHILCO data presented, selecting a hydrometallurgical process option for Los Azules would reduce effective water usage by 75% to 80% over a concentration alternative. Given this context, the most appropriate technology selection for Los Azules to minimize water usage is a hydrometallurgical approach, which is the basis for the project development in this Technical Report. Additionally at Los Azules, alternatives for improving precipitation/snow capture, site dust control, reuse/recycle and passive water treatment strategies are being developed and included in the project design.

Transforming the Energy/Carbon Nexus

An extensive review of power generation and supply options for the project was undertaken to consider the options for renewable energy. YPF Sociedad Anónima ("YPF S.A." or "YPF") owns and operates power generation facilities in Argentina based on wind, solar, geothermal, and hydroelectric sources through its subsidiary YPF-LUZ. YPF-LUZ has a rate structure based on 100% renewables sourced power generation that can be used as the project basis, eliminating hydrocarbon-based generation and associated emissions. The YPF-LUZ electric power supply option was selected for the Los Azules project as a small premium over other hydrocarbon-based power options.

In addition to responsible energy supply, the reduction of energy consumption is also a key aspect to responsible regenerative mining.

The Chilean Ministry of Mining, Comisión Chilena del Cobre (COCHILCO) published a report concerning energy usage in the Chilean copper mining industry titled "Energy Consumption in the Copper Mining industry, 2017". The report characterizes the relative energy usage, as both electric power and combustibles, and usage intensity per tonne of material processed and per tonne of refined copper produced for copper mining projects in Chile based on its survey in 2017. Table 2.2 shows the reported energy demand for mining and concentrator and SX/EW processes.

As depicted in Table 2.2, the specific energy demand per tonne of produced copper is currently about 13.6 gigajoules for the SX/EW process (of which 10.8 gigajoules is electricity) and 9.3 gigajoules for a mill/concentrator (of which 9.1 gigajoules is electricity). However, to be comparable in the case of plants equipped with mill/concentrators, this value should also include for smelting and refining to get to a finished copper cathode product which increases the comparable energy value to 20.9 gigajoules for the mill/concentrator/smelter/refinery processes (of which 14.2 gigajoules is electric power).

Table 2.2: Unit Coefficients of Energy Consumption in Megajoules (MJ) per Metric Tonne of Fine Copper (TMF) Contained in Each Process (COCHILCO 2018)			
Process	**Type of Energy**	**Units**	**Average 2018/2001**
Open Pit Mine	Combustibles	MJ/ TMF Cu	6,369
	Electricity	MJ/ TMF Cu	618
	Sub Total	**MJ/ TMF Cu**	**6,987**
Underground Mine	Combustibles	MJ/ TMF Cu	1,405
	Electricity	MJ/ TMF Cu	1,872
	Sub Total	**MJ/ TMF Cu**	**3,278**
Mine	Combustibles	MJ/ TMF Cu	5,827
	Electricity	MJ/ TMF Cu	766
	Sub Total	**MJ/ TMF Cu**	**6,593**
Concentrator	Combustibles	MJ/ TMF Cu	228
	Electricity	MJ/ TMF Cu	9,088
	Sub Total	**MJ/ TMF Cu**	**9,316**
Smelter	Combustibles	MJ/ TMF Cu	4,899
	Electricity	MJ/ TMF Cu	3,848
	Sub Total	**MJ/ TMF Cu**	**8,747**
Refinery	Combustibles	MJ/ TMF Cu	1,530
	Electricity	MJ/ TMF Cu	1,285
	Sub Total	**MJ/ TMF Cu**	**2,815**
LX/SX/EW	Combustibles*	MJ/ TMF Cu	2,733
	Electricity	MJ/ TMF Cu	10,861
	Sub Total	**MJ/ TMF Cu**	**13,594**
Site Services	Combustibles	MJ/ TMF Cu	489
	Electricity	MJ/ TMF Cu	739
	Sub Total	**MJ/ TMF Cu**	**1,228**

* Note: Predominantly includes the solvent extractions process losses for the hydrocarbon-based copper extractant and diluent.

Selecting a hydrometallurgical process option for Los Azules would reduce effective energy usage by approximately 35% over a mill/concentration alternative to produce copper cathodes. The relative electric power requirement reduction is about 25%. Given this context, the most appropriate technology selection for Los Azules to minimize water usage is a hydrometallurgical approach, which is the basis for the project development in this Technical Report.

Processing with the End Game in Mind

The hydrometallurgical processing option also provides lower overall project impacts from:

- lower transport requirements for product based on copper content of cathodes (99.99% Cu) versus concentrates (25%-35% Cu) and concentrate smelting options located outside of Argentina/South America.
- more efficient and minimized use of land for heap leach pad versus tailings storage facilities from concentration tailings discharge.
- on-site generation of sulfuric acid supplied, using bi-product sulfur supplied from local Argentinian sources, employing waste heat capture for on-site power generation and process heating – reduces grid based electric power requirements and eliminates hydrocarbon-based alternatives.
- establishment of the infrastructure to be a rapid adopter of emerging heap leaching technologies

for primary copper mineral resources when encountered – avoiding the future need for concentration methods as is the current industry practice.

Moving Rock and Decarbonizing Mining Operations

The mining of copper resources is by its nature an impact to the local environment. However, how that impact is achieved must adopt less impactful means. Maximizing electrification, coupled with regenerative power supply is aimed at significantly reducing that impact.

The initial mining concepts will use trolley assisted diesel-electric mine haulage and support equipment initially to significantly reduce diesel emissions. However, the project will select equipment and methods to rapidly transition to completely battery electric power as rapidly as the technology and manufacturing capacities allow. The initial fleet would be converted to battery electric as part of this transition. The transition would also include in-pit conveying alternatives to minimize fleet requirements. The ultimate vision is a fully electric mine and the elimination of emissions associated with diesel energy.

The project will also adopt autonomous technology to minimize labor requirements and maximize equipment usage and efficiencies.

A Mining Camp for Maximum Livability – the healthiest, greenest mine camp in the world.

The remote nature of the Los Azules site necessitates the need for onsite worker accommodations. While the on-site work force will be minimized in favor of Calingasta and San Juan locations and remote work concepts so that people can be closer to their families and communities, a necessary contingent will be required at Los Azules itself. Therefore, creating camp conditions that are exceptional and supportive of mental and physical health will reduce absenteeism and help McEwen maintain a productive, happy, and engaged workforce where jobs are coveted and retention high – in fact coveted within the industry.

The Los Azules mine camp will house and support 1000-2000 workers at any time, with the flexibility to accommodate as needed. To accommodate alternate staff demands, the camp is designed for scalability and can be configured to house more employees if needed in various 'neighborhood' groupings organized in a linear fashion within the facility.

The mine camp has been strategically located to optimize multiple variables. Worker safety, comfort, well-being, as well as the distance from the mine operations and access to the main road are major considerations. In addition, the specific layout and orientation have been selected to support passive design and solar energy generation, which are key considerations.

The Los Azules camp and mine will be forming a microgrid in one of the most remote locations on the planet. Even though the camp will be connected to offsite energy production, it is being sized for net-positive energy production, making it a candidate for International Living Future Institute's (ILFI) Net Zero Energy certification. This self-sufficient energy system will serve the entire site's occupied footprint and will facilitate load stabilization through energy storage technology and carefully timed consumption.

The camp will also be designed to provide heating and climate control, acoustics, medical and support services including recreation and medical clinic, improved air quality using living plant systems, and water management to capture rainwater and snowmelt, retain that water and treat it for reuse using natural systems. The camp will pursue ILFI certification based on the alignment with the Living Building Certification "Water Petal".

The camp will be designed to provide space for growing food in a self-sustaining environment. Finally, the camp will provide waste management systems to provide reuse of waste materials, either through direct reuse, recycling, composting and elimination of single-use plastics and packaging.

Carbon

The Greenhouse Gas ("GHG") Protocol Corporate Standard classifies a company's GHG emissions into three 'scopes'. Scope 1 emissions are direct emissions from owned or controlled sources. Scope 2 emissions are indirect emissions from the generation of purchased energy. Scope 3 emissions are all indirect emissions (not included in scope 2) that occur in the value chain of the reporting company, including both upstream and downstream emissions. The GHG Protocol Corporate Accounting and Reporting Revised Standard (published by the World Resources Institute (WRI), a U.S.-based environmental NGO, and the World Business Council for Sustainable Development (WBCSD), a Geneva-based coalition of 170 international companies) provides requirements and guidance for companies and other organizations preparing a corporate-level GHG emissions inventory.

Figure 1.2 Estimated Carbon Intensity versus Copper Equivalent Production Centiles 2022-2040 for mine site emission chart presents the relative estimated emissions for copper assets on an equivalent copper basis as obtained from the Emissions Benchmarking Tool – (Metals)™, a product of Wood Mackenzie. The Wood Mackenzie database includes 394 individual global mining assets and covers Scope 1 and 2 emissions determined using the published methodology on their website. The highlighted assets represent comparable major Argentinian projects as included in the WoodMac modeled information. The Los Azules project metrics in the Wood Mackenzie data (red line highlighted) reflects the estimated emissions for the prior project concept. The Wood Mackenzie average Scope 1&2 emissions intensity for all 394 included assets the period between 2022 and 2040 is 1,980 kg CO_2-e/t Cu Eq. For the 57 copper SX/EW assets included the average Scope 1&2 emissions intensity for the period between 2022 and 2040 is 1,723 kg CO_2-e/t Cu Eq.

Based on the current project concepts considered for implementation at Los Azules, notably:

- Electrical energy sourced from 100% renewables (YPF Luz basis).
- Incorporation of site and mine electrification concepts (trolley assist for mine haulage, battery electric vehicles where possible).
- Regenerative design concepts for support infrastructure; and
- Hydrometallurgical extraction processes to produce copper cathodes.

The carbon intensity per unit of copper equivalent production (Cu Eq) was estimated by Whittle Consulting Limited ("Whittle") using the GHG Protocol Corporate Accounting and Reporting Revised Standard principles (published by the World Resources Institute (WRI), a U.S.-based environmental NGO, and the World Business Council for Sustainable Development (WBCSD), a Geneva-based coalition of 170 international companies) which provides requirements and guidance for companies and other organizations preparing a corporate-level GHG emissions inventory).

Based on Whittle's estimations, the predicted life of mine carbon intensity for the Los Azules initial Base Case project is estimated to be to be 670 kg CO2-e/t Cu Eq for Scope 1 & 2 emissions. The Alternative Case project is estimated to be to be 826 kg CO_2-e/t Cu Eq for Scope 1 & 2 emissions. The estimated Scope 1-3 life of mine emissions for the Base and Alternate cases are approximately 902 and 1066 kg CO_2-e/t Cu Eq respectively, assuming transport of copper cathodes to port facilities in either Chile or Argentina.

Figure 2.1 and Figure 2.2 show the Whittle estimated Scope 1 & 2 carbon intensity per tonne of copper recovered annually and distribution by source respectively for the Base Case. Figure 2.3 and Figure 2.4

shows the Whittle estimated Scope 1 & 2 carbon intensity per tonne of copper recovered annually and distribution by source respectively for the Alternate Case.

Scope 2 emissions, related to electric power usage and sources is negligible for the Los Azules site due to inclusion of 100% renewable energy provided by YPF Luz.

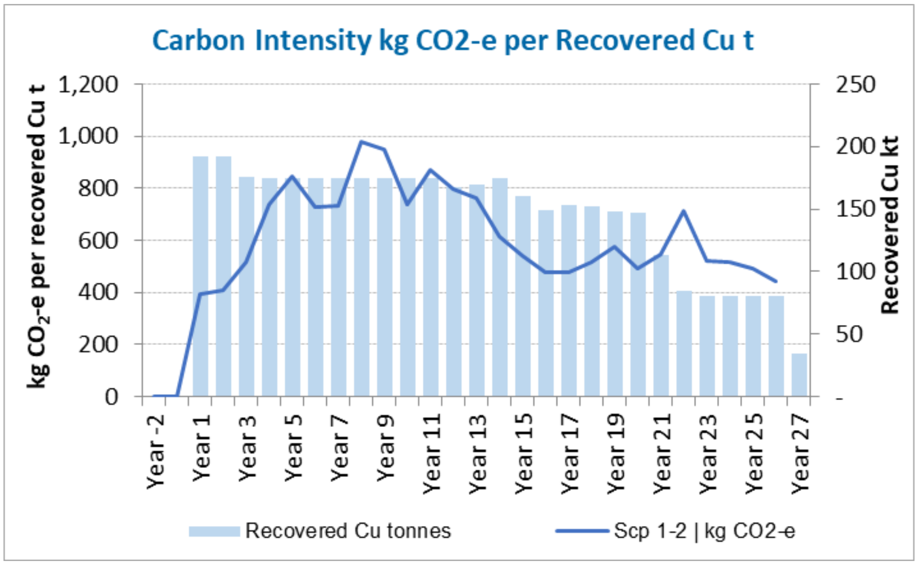


Figure 2.1: Annual Carbon Intensity kgCO$_2$/t Copper – 175k tpa Base Case (Whittle, 2023)

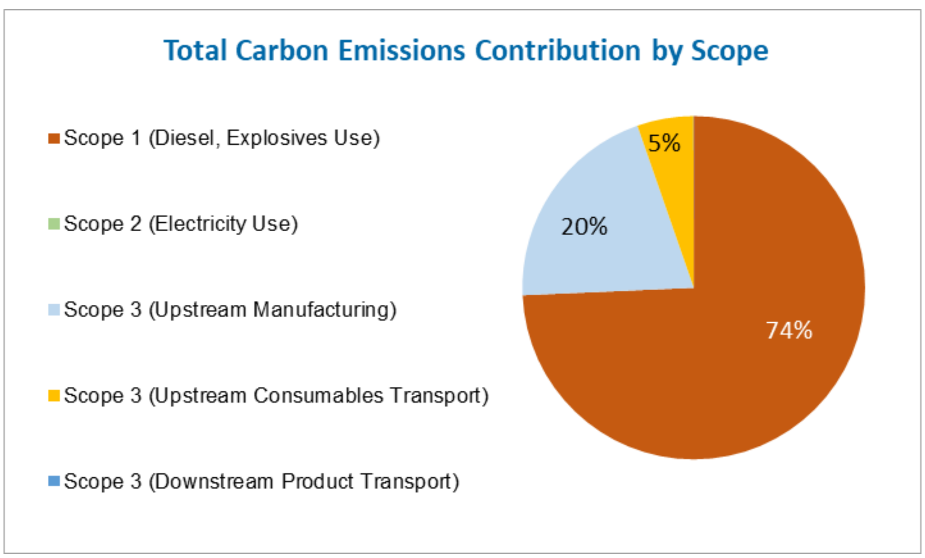


Figure 2.2: Emissions by Scope – 175k tpa Base Case (Whittle, 2023)

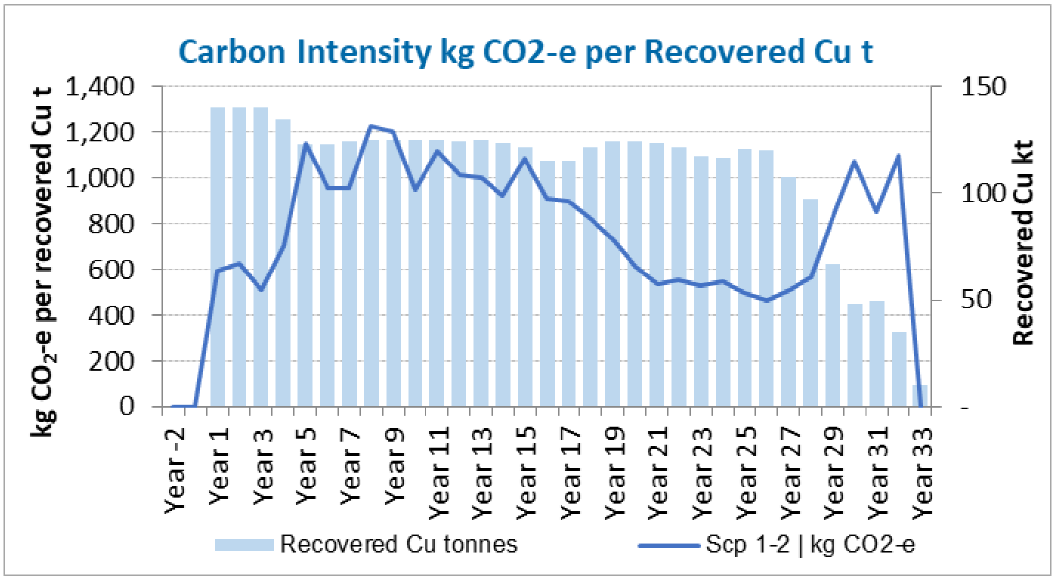


Figure 2.3: Annual Carbon Intensity kgCO_2/t Copper – 125k tpa Alt. Case (Whittle, 2023)

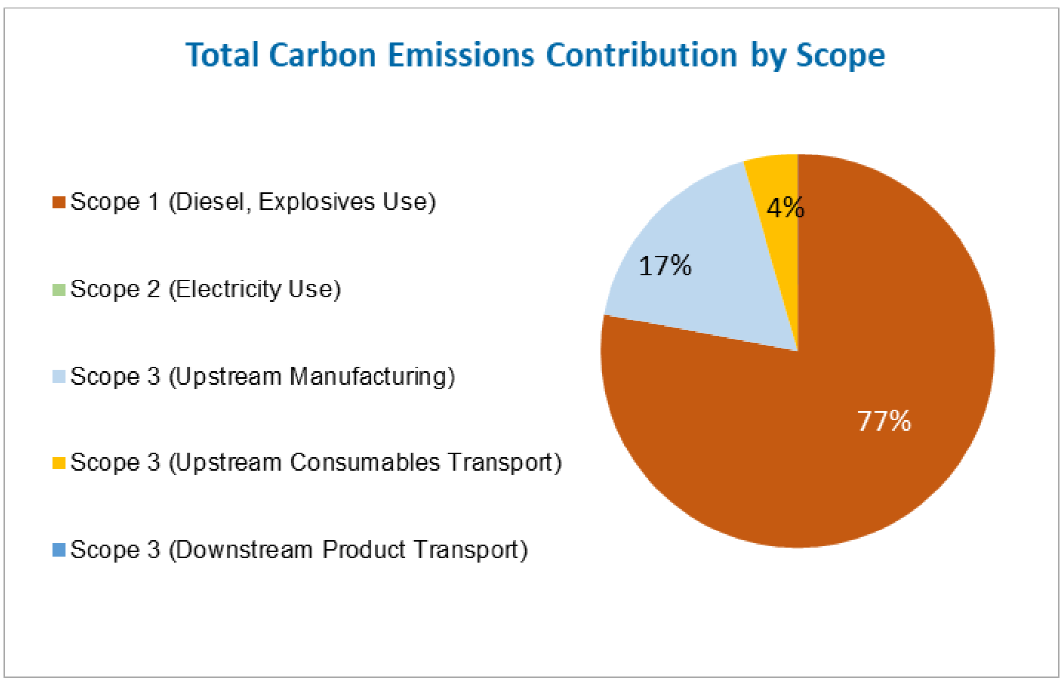


Figure 2.4: Emissions by Scope – 125ktpa Alt. Case (Whittle, 2023)

Continued implementation of newer technology, fully electric mine equipment and regeneration applications to further eliminate and off-set carbon emissions is expected to deliver on McEwen's commitment to achieve net-zero carbon emissions from the Los Azules project by 2038, well ahead of its peers. Key opportunities in order of impact are:

- Mine equipment upgraded to fully battery electric operation when available.
- Requiring transportation companies to use EV's for supplies, materials, and product transport.

- Broader application of deep green technologies in production areas (larger buildings)

2.3 QUALIFIED PERSONS

This PEA is triggered by McEwen's intention to publicly disclose the engineering and optimization studies completed by Samuel Engineering Inc. and Stantec Inc. in conjunction with McEwen during 2021 - 2023. The results from the updated PEA and the Los Azules property are material to McEwen.

The quality of information, conclusions and estimates contained herein are consistent with the level of effort involved in the authors' services based on: (i) information available at the time of preparation; (ii) data supplied by outside sources and (iii) the assumptions, conditions and qualifications set forth in this report. This report is intended to be read as a whole and sections should not be read or relied upon out of context.

This report was authored by the qualified persons (each a "QP" and collectively, the "QPs") listed in Table 2.3. Each QP only assumes responsibility for those sections or areas of the report that are referenced opposite their name in Table 2-1. None of such QPs, however, accept any responsibility or liability for the sections or areas of this report that were prepared by other QPs.

A summary of the Qualified Persons (QP's), as defined in NI 43-101, responsible for each section of the report and their respective company affiliation is provided in Table 2.3.

Table 2.3: Summary of Qualified Persons

Qualified Person (QP)	Company	Areas of Responsibility	Report Sections
Allan L. Schappert, CPG, SME-RM	Stantec Inc.	Mineral Resource Estimates, Geology, Sample Preparation, Exploration & Drilling	7, 8, 9, 10, 11, 12, 14, 16
Bruno Borntraeger, PE	Knight Piesold Ltd.	Heap Leaching Design, Environmental Studies & Permitting	4.10, 18.9, 20
W. David Tyler, SME-RM	McEwen Copper Inc.	Property, Ownership, Surface Rights, Project Infrastructure, Market Studies & Contracts	4, 5, 6, 18, 19
James L. Sorensen, FAusIMM	Samuel Engineering	Process, Mineral Processing, Metallurgical Testing & Recovery, Project Infrastructure, Project & Study Execution,	1, 2, 3, 13, 17, 18.1-3, 18.5-8, 21.1-2, 21.3.2-3, 25.1-3, 25.8.1-2, 26.1-2 and 27
Richard F. Reinke, P. Geo.	Stantec Inc.	Water Supply & Pit Dewatering	18.7
Robert J. Bowell PhD, C. Chem, C. Geo, P. Geo	SRK Consulting UK Limited	Geochemistry	20.2
Steven Guy Bundrock, PE	Stantec Inc.	Mine Rock Storage Facility	18.4
Satjeet Pandher, PE	Stantec Inc.	Mining	16, 21
Steven Alan Pozder, PE, MBA	Samuel Engineering	Economic Analysis	22
All	All	Information relating to areas of responsibility for Sections:	1, 2, 25, 26

2.4 PROPERTY DESCRIPTION AND OWNERSHIP OF THE LOS AZULES PROPERTY SUBJECT TO THIS STUDY

The Los Azules Project is comprised of properties (the "Properties") owned by Andes Corporación Minera S.A. (ACSMA), an Argentine subsidiary of McEwen Mining through its ownership in McEwen Copper.

ACMSA controls approximately 31,746 ha of mining rights (Minas) around the Los Azules deposit. In addition, ACMSA owns sufficient surface rights for the Project pursuant to an agreement with CCM S.A., whereby ACMSA acquired 18,000 ha in surface rights. In 2018, ACMSA filed a request to group all the Mining Permits together, file #1124.553-A-2018, in such way that, once all surveys are approved, all the Mining Permits be considered as one larger Mining Permit. This will allow investments to be distributed across the larger permit group and eliminate the need to spend on each individual Mina. It is expected that this request by ACMSA could be favorably resolved during 2023. These Properties are the subject of this Technical Report. Specific property details are discussed in Section 4 of this report.

As of May 2023, the Company owns a fully diluted 51.9% interest in the Los Azules copper deposit in San Juan, Argentina through its subsidiary, McEwen Copper Inc. ("McEwen Copper") which owns a 100% interest in the Los Azules copper project in San Juan, Argentina, and the Elder Creek exploration project in Nevada, USA. The relevant ownership structure is shown in Figure 1.3 of this document.

FCA Argentina S.A., a subsidiary of Stellantis N.V. ("Stellantis"), invested ARS $30 billion in Argentina to acquire shares of McEwen Copper in a transaction that closed on February 24th, 2023. In connection with the Transaction, McEwen Copper and certain of its affiliates entered into an Investor Rights Agreement with Stellantis (the "Stellantis IRA") and a Copper Cathodes and Concentrates Purchase Rights Agreement (the "CCCPRA"), which are described below.

The Stellantis IRA provides for the following principal terms:

- Stellantis will have the right to nominate one director to the Board of McEwen Copper,
- Stellantis will have the opportunity to provide local currency funding, in certain circumstances, for advancement of the Los Azules project,
- Comprehensive scientific, technical and strategic planning information rights,
- Pre-emptive right to maintain their ownership percentage in any follow-on equity offering,
- Agreement of McEwen Mining and Robert R. McEwen to not trigger Drag Along Rights in the event of a bid for McEwen Copper prior to the planned initial public offering (IPO), and
- McEwen Copper commits to achieve net-zero carbon emissions from the Los Azules project by 2038.

The CCCPRA provides an option to Stellantis and its affiliates that, if exercised to its maximum extent, would allow them to purchase a percentage of the copper cathodes or copper concentrates or both produced from the Los Azules project, in each case equal to their equity ownership percentage in McEwen Copper at the time of exercise.

Nuton LLC, a Rio Tinto Venture ("Nuton"), has invested a further USD $25 million to acquire shares of McEwen Copper in a transaction that closed on August 31st, 2022. In connection with the transaction, McEwen Copper entered into a collaboration agreement with Nuton (the "Nuton Collaboration Agreement"), to advance our understanding of the potential application of heap leach technology at Los Azules, including the testing of Nuton™ Technologies for compatibility with Los Azules copper mineralization. Leaching has many potential economic and environmental benefits over a conventional milling scenario, including lower water and energy consumption, no large tailings storage facility or dam, and typically lower capital and operating costs.

The principal terms of the Nuton Collaboration Agreement include:

- McEwen Copper and Nuton will jointly undertake copper leach testing using Nuton™ technologies with samples from Los Azules. McEwen Copper has agreed to grant exclusivity to Nuton for one year in the area of novel, patented or trade secret leaching technology, while it will continue its independent test work and studies using conventional leach technologies.
- Nuton will have the right to select one nominee who will be appointed as a director or observer to the Board of McEwen Copper. This right will continue for as long as Nuton holds greater than 7.5% of the issued and outstanding shares of McEwen Copper.
- McEwen Copper has agreed to limit related party transactions in certain situations until the earlier of the planned IPO (or alternative liquidity event) or Nuton ceasing to hold 7.5%.
- Customary standstill and lock-up agreement between the Investor and its affiliates and McEwen Copper and its affiliates.

Nuton has invested a further USD $30 million to acquire additional shares of McEwen Copper in a transaction that closed on March 15th, 2023.

In connection with the second Nuton transaction, McEwen Copper and certain of its affiliates entered into an Amended Collaboration Agreement (the "New Nuton Collaboration Agreement”) and a Copper Cathodes and Concentrates Purchase Rights Agreement (the “CCCPRA”), which are described below.

The New Nuton Collaboration Agreement provides for the following additional rights beyond those in the original Nuton Collaboration Agreement.

- Nuton will have the opportunity to provide local currency funding, in certain circumstances, for advancement of the Los Azules project,
- Comprehensive scientific, technical, and strategic planning information rights,
- Extension of exclusivity over investigating other novel, trade secret or patented copper heap leach technologies until August 10, 2024,
- Pre-emptive rights to maintain their ownership percentage in any follow-on equity offering,
- Agreement of McEwen Mining and Robert R. McEwen to not trigger Drag Along Rights in the event of a bid for McEwen Copper prior to the planned initial public offering (IPO).

The CCCPRA provides an option to Nuton that is equivalent to that of Stellantis described above.

After closing the second Nuton Transaction, McEwen Copper has 28,885,000 common shares outstanding on a fully diluted basis, and its shareholders are: McEwen Mining Inc. 51.9%, Stellantis 14.2%, Nuton 14.2%, Robert R. McEwen 13.8%, Victor Smorgon Group 3.5%, and other shareholders 2.4%.

2.5 PERSONAL INSPECTION OF LOS AZULES PROPERTY

The author considers the foregoing personal inspections to constitute a “current personal inspection” in accordance with NI 43-101 for the current level of study.

Mr. Allan Schappert (CPG, SME-RM) of Stantec Consulting, QP, visited the Los Azules property during the period from 24 April – 15 May 2022. The purpose of the visit was to observe, review, and comment on all aspects of data collection, recording, and analysis in preparation of the Mineral Resource Estimate. Activities and discussions included the following: visit and inspection of operating drill sites; care, custody, and control procedures of core boxes; core logging facilities at Los Azules camp; core storage and sampling procedures at the Calingasta warehouse; a tour of the independent assay lab in Mendoza; review of historical and current QA/QC protocols with a review of recent results.

Mine Technical Services (MTS) conducted two phases of database audits (2021 and 2022) including a site visit by Todd Wakefield and Francisco Ramos between April 18 – 27, 2022. Discussions of their findings and a review of their recommendations were made. Stantec's geological QP supported McEwen Copper's decision to undertake an extensive re-assaying program of existing core to augment the database prior to the updated estimate.

A site visit was performed 08 April 2022 to 13 April 2022 with a team physically on the site 09 April 2022 through 12 April 2022 and attended by the following Stantec individuals.

- Jason Reynolds – Geotechnical Leader
- Carrie Loar – Geology Senior Reviewer
- Julia Loffler – Lead Geologist
- Andrew Burgin – Geotechnical Designer

SRK Geochemists Rob Bowell and Brooke Clarkson visited the Los Azules core warehouse facility in Calingasta, November 7-9, 2022, and were accompanied by Hugo Bracamonte from McEwen Copper. The focus of the visit was to examine the drill core from the intervals selected for geochemical characterization. Field logging included a description of the lithology, alteration, mineralogy, and structure.

Bruno Borntraeger, PE of Knight Piesold (KP), QP for the leach pad design visited the site January 29-31, 2023. The visit focused on leach pad site locations and field-testing requirements for the geotechnical design of the leach pad.

David Tyler (SME-RM) is the McEwen Copper Project Director for the Los Azules project and has the responsibility for the study work. Mr. Tyler has visited the Los Azules site several times in 2022 and 2023.

3.0 RELIANCE ON OTHER EXPERTS

This report section has been prepared for McEwen by the respective QP's referred to in Table 2.3. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to the QP's at the time of preparation of this report, including the 2017 Preliminary Economic Assessment.
- Assumptions, conditions, and qualifications as set forth in this report.
- Data, reports, and other information supplied by McEwen.

For this report, the QP's have relied on property ownership information provided by McEwen through a legal review and opinion report titled "Incorporation and good standing status of Andes Corporación Minera S.A. (ACMSA) and of its mining rights" dated January 11, 2023, by Abogado Jose Vargas Gei of Vargas & Galindez (V&G), a Mendoza based legal firm. Samuel Engineering has not independently researched property title or mineral rights for the Los Azules property and expresses no independent opinion as to the ownership status of the property.

Metal pricing assumptions are derived from information provided by CIBC and the IMF.

McEwen Copper has provided the basis of the calculations for all associated royalties and taxes including Argentine Income, VAT and Credit and Debit Bank Tax.

A draft copy of the Report has been reviewed for factual errors by McEwen Mining. Any representations, statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false or misleading at the date of this Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Los Azules Project is a porphyry copper development project located in the Frontal Andes Cordilleran region of San Juan Province, Argentina along the border with Chile. The project falls within the Calingasta Department of the San Juan Province. The Project is approximately 80 km west-northwest of the small town of Calingasta, in the San Juan Province of Argentina at approximately 31° 06' 25" south latitude and 70° 13' 25" west longitude. Calingasta is located 173 km by road west of the Provincial Capital city of San Juan along Route 12.

The terrain elevation at the project site ranges between 3,200 meters above sea level (masl) at the proposed camp location and up to 4,500 masl on the high peaks in proximity to the Project. The Project area is remote, and no infrastructure is present. There are no nearby towns, Indigenous residents, or settlements. Seasonal exploration work typically commences in October or November and terminates in May or early-June. Exploration operations are supported by means of two temporary camps within the Project site area.

The mine development is located approximately 6 km east of the border with Chile (Figure 4.1).

McEwen Mining controls approximately 32,700 ha of mining rights and 18,000 ha of surface rights around the Los Azules project. Aerial photography and global positioning were utilized to locate the property in the field; the coordinates of the corners of the property are established in the government documents granting the mining rights.

The Los Azules Project is currently accessed by 120 km of unimproved road with eight river crossings and two mountain passes (both above 4,100 m elevation). This access is subject to snow accumulation and is passable only from November through to May. This 2023 update describes in Section 18 “Project Infrastructure” a potential future northern access route within McEwen owned lands that is less affected by snow. Also described is an airstrip currently permitted for construction.

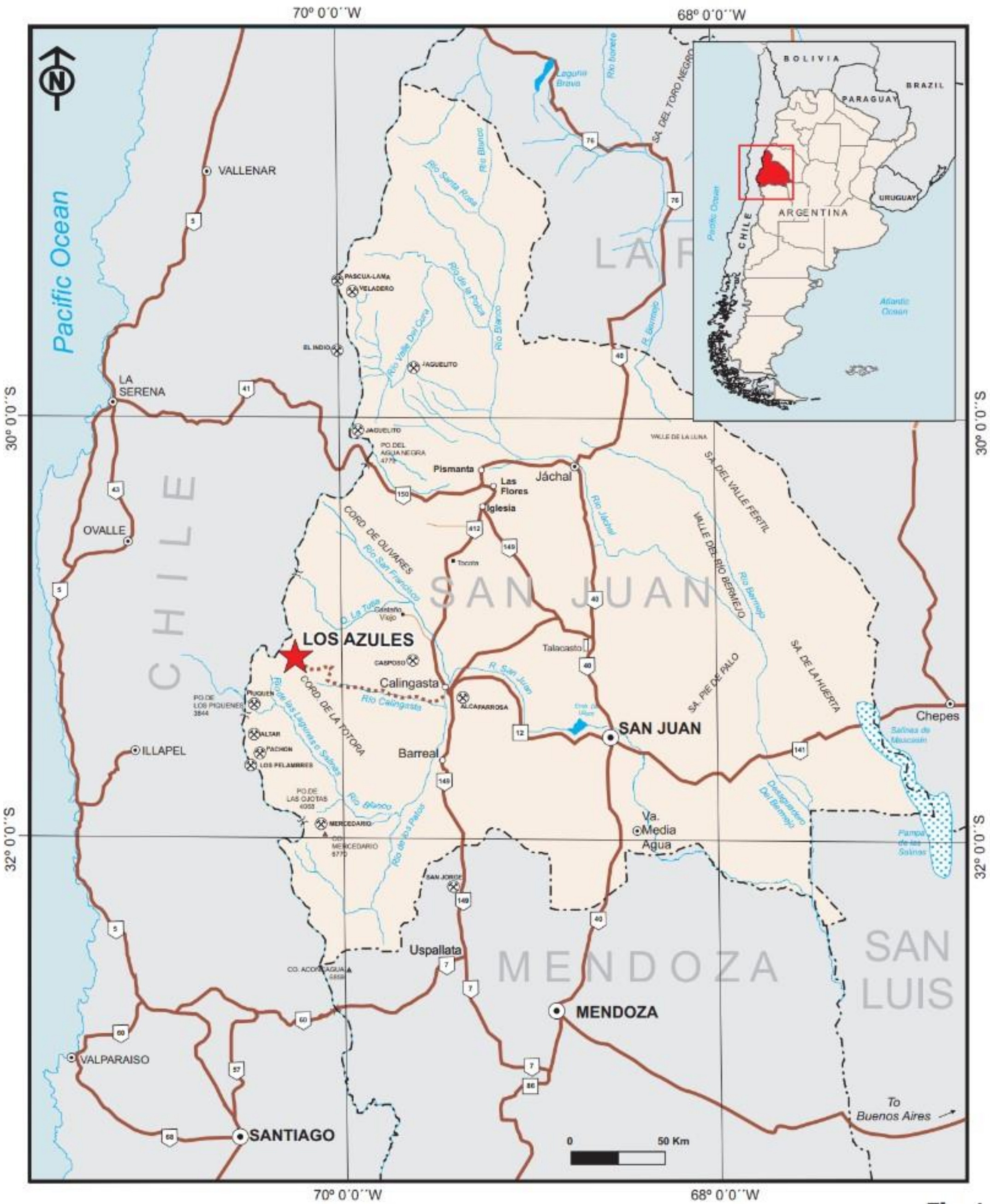


Figure 4.1: Project Location (Minera Andes 2009)

4.2 PROPERTY AND TITLE IN ARGENTINA

The laws, procedures, and terminology regarding mineral title in Argentina differ from those in the United States and in Canada. Mineral rights in Argentina are separate from surface ownership and are owned and administered by the provincial governments. The following summarizes some of the relevant provisions of the Argentine Mining Code and Argentinean mining law terminology to aid in understanding the McEwen Mining land holdings in Argentina.

The provinces are the owners of the natural resources located within their territories and each province retains the power to administer and regulate mineral rights according to the federal Mining Code and supplemental provincial laws and regulations.

Surface rights are separate from mineral rights, and they are treated separately under Argentine law. The Mining Code establishes that mining is in the public interest and therefore surface owners cannot prevent the granting of mining rights and properties or commencement and/or continuity of mining activities on their property, but surface owners have a right to collect an indemnity because of the use of the land by the miner and the damages derived from mining activities. Land over which a mining concession has been granted is legally subject to different types of easements (e.g., right of way, occupation of land, use of water, etc.), provided that an indemnity is paid to the owner of such land.

Mineral rights are considered forms of real property and can be sold, leased, or assigned to third parties on a commercial basis. “Cateos” (exploration permit) and “Minas” (mining concession) can be forfeited if minimum work requirements are not performed or if annual payments are not made. Generally, notice and an opportunity to remedy defaults are provided to the owner of such rights.

Grants of mining rights, including water rights, are subject to the rights of prior users. Further, the mining code contains environmental and safety provisions administered by the provinces. Prior to conducting operations, applicants must submit an environmental impact report (“Informe de Impacto Ambiental” or IIA in Spanish) to the provincial mining authority describing the proposed operation and the methods to be used to prevent undue environmental damage. When the provincial mining authority approves the IIA it issues a permit in the form of an official declaration (“Declaratorio de Impacto Ambiental” or DIA in Spanish). The IIA must be updated every two years, with a report on the results of the protection measures taken. If protection measures are deemed inadequate, additional environmental protection may be required. Mine operators are liable for environmental damage. Violations of environmental standards may cause exploration or mining operations to be shut down but without prejudice to mining title.

4.2.1 Cateo

A cateo is an exploration permit that does not allow commercial mining but gives the owner a preferential right to obtain a mining concession for the same area. Cateos are measured in 500 ha unit areas. A cateo cannot exceed 20 units (10,000 ha). No person may hold more than 400 units (200,000 ha) in a single province. The term of a cateo is based on its area: 150 days for the first unit (500 ha) and an additional 50 days for each unit thereafter. After a period of 300 days, 50% of the area over four units (2,000 ha) must be dropped. At 700 days, 50% of the area remaining must be dropped. At each stage, the land can be converted to one or more “Manifestaciones de Descubrimiento” (MD). Time extensions may be granted to allow for bad weather and difficult or seasonally restricted access. Cateos are identified by a file number or "expediente" number and are awarded by the following process:

1. Application for a cateo covering a designated area. The application describes a minimum work program for exploration.
2. Registration by the province and formal placement on the official map or Geographic Register.
3. Publication in the provincial official bulletin and notification to the surface owner.
4. A period following publication for third parties to oppose the claim.
5. Award of the cateo.

The length of this process varies depending on the province and commonly takes up to two years. Accordingly, cateo status is divided into those that are in the application process and those that have been awarded. If two companies apply for cateos on the same land, the first to apply has the superior right unless the area was released from a prior owner, at which point the property is awarded to one of the interested parties through a blind drawing. During the application period, the first applicant has rights to any mineral discoveries made by third parties in the cateo without applicant's prior consent.

Applicants for cateos may be allowed to explore on the land pending formal award of the cateo, with the approval of the surface owner of the land. The period after which the owner of a cateo must reduce the quantity of land held does not begin to run until 30 days after a cateo is formally awarded.

A fee (or canon in Spanish) of AR $400 per unit must be paid upon application for the cateo. This is paid only once. In addition, the 2012 tax act for the province of San Juan requires a fee to be paid upon application for a cateo. The actual value is AR $1,600 for each unit of 500 ha. This fee is only paid one time.

4.2.2 Mina

To convert an exploration permit (cateo) to a mining concession (mina), some or all the area of a cateo must be declared as MD (Manfestacíon de Descubrimiento) and then converted to a mina. Minas are mining concessions which permit mining on a commercial basis. The area of a mina is measured in pertenencias. Each mina may consist of one or more pertenencias (ownership pieces). Conventional pertenencias are 6 ha and pertenencias for disseminated deposits are 100 ha. Once granted, minas have an indefinite term assuming exploration development or mining is in progress and investment conditions according to the Mining Code are met. An annual canon fee of AR $19,000 per pertenencia is payable to the province.

Minas are obtained by the following process:

- Declaration of a MD, in which a point within a cateo is identified as a discovery point. The MD is used as a basis for location of pertenencias of the sizes described above. MDs do not have a definite area until pertenencias are proposed. Within a period following designation of a MD the claimant may do further exploration, if necessary, to determine the size and shape of the mineralized material.
- Survey (mensura) of the mina. Following a publication and opposition period and approval by the province, a formal survey of the pertenencias (together forming the mina) is completed before the granting of a mina. The status of a surveyed mina provides the highest degree of mineral land tenure and rights in Argentina.

4.2.3 Provincial Reserve Areas

Provinces are allowed to withdraw areas from the normal cateo/mina process. These lands may be held directly by the province or assigned to provincial companies for study or exploration and development.

4.3 OWNERSHIP OF THE LOS AZULES PROJECT

McEwen was organized under the laws of the State of Colorado on July 24, 1979, and is listed on the New York Stock Exchange (NYSE) and on the Toronto Stock Exchange (TSX) under the symbol MUX. The Company's head office is in Toronto, Canada. As of May 2023, the Company owns a 51.9% interest in McEwen Copper which owns a 100% interest in the Los Azules copper project in San Juan, Argentina, and the Elder Creek exploration project in Nevada, USA. The relevant ownership structure is shown in Figure 4.2, as provided by McEwen Mining.

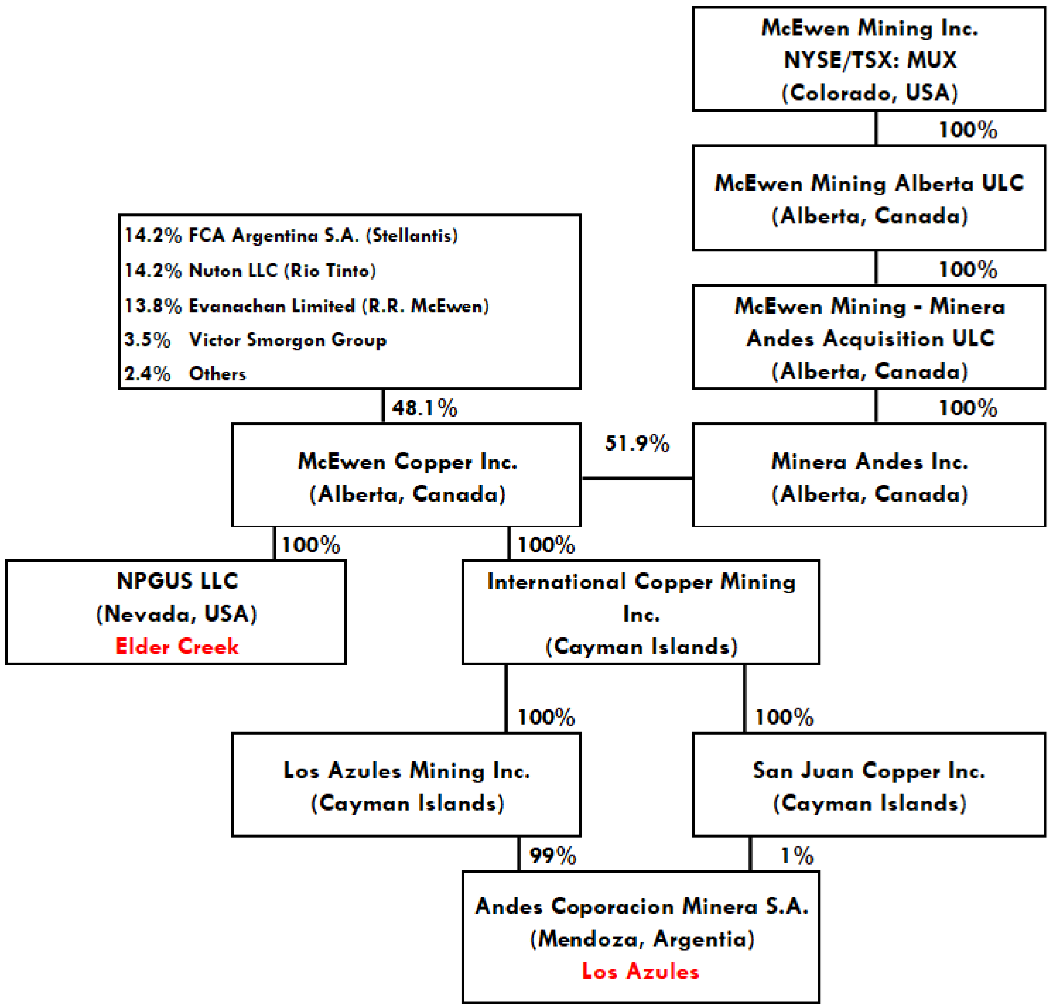


Figure 4.2: Los Azules Ownership Structure (McEwen, 2023)

4.3.1 Los Azules Mineral Rights

In 1994, Minera Andes S.A. (MASA), an Argentine subsidiary of Minera Andes, was granted the Cateo to explore the Cordon de Los Azules (file # 545.957-D-94). This cateo was divided and converted into two MDs on October 17, 1998, known as Azul 1 and Azul 2. These MDs cover part of the southern portion of the Project. In 2009 MASA transferred these two MDs to Andes Corporación Minera S.A. (ACSMA). The central portion of the Project is covered by MD Mirta and the northern portion by Escorpio II, all owned by ACSMA.

The Los Azules Project is comprised of properties (the "Properties") owned by ACSMA, an Argentine subsidiary of McEwen Mining through its ownership in McEwen Copper. The information in the section relies upon a legal review and opinion report Re: "Incorporation and good standing status of Andes Corporación Minera S.A. (ACMSA) and of its mining rights" dated January 11, 2023, by Abogado Jose Vargas Gei of Vargas & Galindez (V&G), a Mendoza-based legal firm.

Based on the V&G review and opinion, the following conclusions were included in their memorandum.

a. ACMSA has good and valid, legal, and beneficial title to the mining rights listed on Table 4.1. Mining rights coordinates are listed on APPENDIX X.
b. The mining rights listed on Exhibit A are all in good standing and comply with applicable regulations.
c. The annual canon for each mining right is paid up to the first semester of 2023 (see Table 4.1, for amount of canon paid for each mining right per year).
d. ACMSA has invested in Los Azules over 300 times the annual canon payment, reaching the minimum amount required by Article 217 of the MC, considering Los Azules Project as a whole unit.
e. Los Azules Project is subject to the payment mentioned in m) and n) below.
f. No inactivity has occurred on the mining rights listed on Exhibit A and Exhibit B for more than four (4) years.
g. "Labor Legal" [allowed exploration work] has been performed and the "Survey" has been performed on all mining rights. The Mining regulatory authority has observed these surveys and is discussing with the Company ways to improve them.
h. Pursuant to Resolution #3011, dated December 20, 2016, the Departamento de Hidráulica authorized ACMSA to use 316.8 m^3 (cubic meters) of water for the benefit of Los Azules project. This permit is renewable.
i. ACMSA has good and valid legal and beneficial title to the following easements (see Exhibit D):
   - File #520.0439-M-97 (exploration access road)
   - File #0680-F28-M-96 (southern access road)
   - File #1124.218-A-18 (northern access road)
   - File #1124.660-M-12 (Candadito camp)
j. ACMSA has requested the following easements, not yet granted (see Exhibit D):
   - File #1124.354-A-2018 (power line)
   - File #1124.544-A-2022 (surface occupation Illanes Mery property)
   - File #1124.231-A-2010 (surface occupation Estomonte property)
k. The 5th Environmental Impact Report update for exploration filed by ACMSA for Los Azules project has been approved by Resolution #317-MM-2021, dated June 3, 2021, and Resolution #352-MM-2021, dated June 17, 2021, both by the Ministerio de Minería of the province of San Juan.
l. Glacial and peri glacial studies have been carried out by the consulting firm Mountain Pass Consulting and have been incorporated into the Environmental Impact Report referred to above.
m. Pursuant to the Los Azules Option Agreement, dated November 2, 2007, entered into between MIM Argentina Exploraciones S.A., Xstrata Queensland Limited, Minera Andes S.A. and Minera Andes Inc. (the latter later acquired by US Gold Corporation and the business combination later renamed McEwen Mining Inc.), once a Feasibility Assessment is completed on "Los Azules" project, including properties named Mercedes and Mirta, a payment of USD $500,000 is due to Ms. Dina Myriam Elizondo de Bosque and Mr. Hugo Arturo Bosque.
n. Pursuant to a Transfer Agreement, dated October 16, 2014, between TNR Gold Corp., Compañía Minera Solitario Argentina S.A., Los Azules Mining Inc., ACMSA and McEwen Mining Inc., ACMSA agreed to pay Compañía Minera Solitario Argentina S.A. a 0.4% net smelter return royalty in respect of Los Azules Project.
o. As of the date of this opinion, there are no material claims against ACM.

The challenge by ACMSA for the peripheral properties under threat of forfeiture, Gina, Sofia, Torora II, and Marcela, was resolved in the company's favor and the forfeited rights were returned by the Mining Council on December 29, 2022.

As of January 17, 2023, the powerline easement was granted by the provincial government.

A list of those land holdings is detailed in Table 4.1 and are also shown on Figure 4.2. The corner coordinates are detailed in Table 28.3. The size of the property covered by those tenements, once actually granted, however, may differ from those set out below.

Table 4.1: Minera Andes S.A. Mineral Claim Descriptions						
Mining Right Name						
Exhibit A Properties						
Agostina	1124.108-A-10	Registered	$38,400	15/11/2010	1,183.59	In approval process
Azul 2	520.0280-M-98	Registered	$41,600	08/09/1999	1,319.51	In approval process
Azul 3	1124.121-A-06	Registered	$6,400	15/03/2013	166.71	In approval process
Azul 4	1124.473-M-08	Registered	$32,000	22/01/2014	902.99	In approval process
Azul 5	1124.119-A-09	Registered	$99,200	15/11/2010	3,000.47	In approval process
Azul Este	1124.186-A-07	Registered	$76,800	27/05/2008	2,371.75	In approval process
Azul Norte	1124.668-M-07	Registered	$6,400	15/11/2010	131.88	In approval process
Cecilia	1124.035-A-12	Registered	$57,600	08/11/2013	1,701.73	In approval process
Escorpio I	0153-C-96	Registered	$6,400	19/06/2008	168.75	In approval process
Escorpio III	0155-C-96	Registered	$6,400	01/11/2013	199.39	In approval process
Mercedes	0644-M-96	Registered	$28,800	04/06/1999	785.77	In approval process
Rosario	1124.169-A-10	Registered	$57,600	15/11/2010	1,767.50	In approval process
Totora	414.1324-C-05	Registered	$19,200	14/09/2010	504.70	In approval process
Gina	1124.168-A-10	Registered	$57,600	15/11/2010	1,762.59	In approval process
Sofia	1124.167-A-10	Registered	$108,800	17/02/2011	3,323.98	In approval process
Totora II	520.496-C-99	Registered	$51,200	26/09/2005	1,560.62	In approval process
Marcela						
Exhibit B Properties						
Azul 1	520.0279-M-98	Registered	$67,200	01/11/1999	2,097.40	Survey observed.
Escorpio II	0154-C-96	Registered	$64,000	05/05/2008	1,990.31	Survey observed
Escorpio IV*	425.213-C-03	Registered	$112,000	13/12/2005	3,500.00	Survey observed (reduced area by 911.71)

Table 4.1: Minera Andes S.A. Mineral Claim Descriptions						
Mining Right Name						
Mirta	1124.0141-M-09	Registered	$12,800	20/10/2010	354.79	Survey observed
				Total Hectares	**31,746.42**	

*__NOTE__: Escorpio IV was originally requested, by the previous owner, as a 4,411.71 hectares mining permit. However, mining law sets a limit to the size of the mining permits held by companies of 3,500 hectares. Consequently, Escorpio IV, when acquired by ACMSA, was 911.71 hectares above the legal limit. This led ACMSA to release the central area of Escorpio IV, as suggested by its geologists, as it being an area that does not affect ACMSA plans for Los Azules Project. The mineral claim locations are shown Figure 4.3.

By law, the released area is granted to Instituto de Exploraciones y Explotaciones Mineras (IPEEM, a provincial mining company), that, following certain procedures, can grant this area to private companies to be explored and exploited.

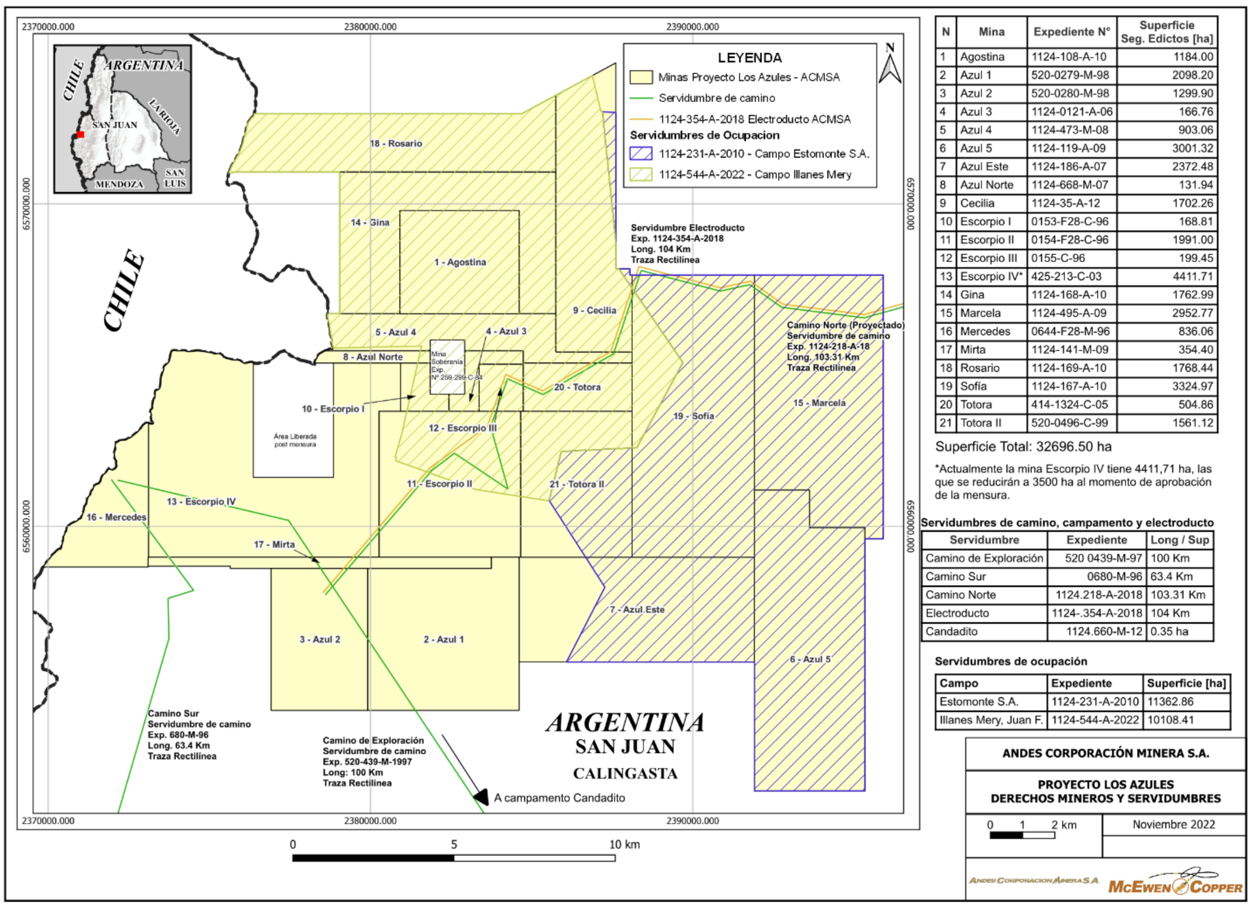


Figure 4.3: Map of Mineral Claims (Minas), Easements (Servidumbres) and Surface (Superficie) Ownership (Vargas & Galindez/McEwen 2022)

As for the mining right at the project center, labeled Soberania (File # 259,299-C-84), ACMSA and three other persons have claimed the right to this mine simultaneously. To date, the award of the mining right has not been resolved, but it is expected that it will be resolved favorably in favor of ACMSA, since the request and technical arguments of ACMSA are more relevant than those of the third parties.

The 21 mining rights have been grouped into a Mining Group, which is in process under file No. 1124.553-A-2018. According to Argentine laws, it is necessary to have approved measurements of each mining right to constitute a mining group, so these measurements are in process and this approval is expected during the year 2023.

An exploitation plan was filed in January 2023 and subsequently an Environmental Impact Statement for the exploitation was filed in April 2023 with the government of San Juan. This plan and EIS were in support of the maintenance of the ACMSA mining rights. The exploitation plan committed ACMSA to begin the development of the mine within 5 years, or before January 2028.

It should be noted that no facilities are foreseen in either the Soberania or the released area of the Escorpio IV mining rights. Figure 5.5 shows the layout of site facilities relative to the surface and mining rights.

4.3.2 Los Azules Surface Rights

In January 2010, Andes Corp. purchased 18,000 ha of surface rights in the Los Azules area. The purchase of this property, located near the Argentina/Chile border region, was subject to governmental approval. The approval was granted on August 31, 2010, by Resolution #907 of the Ministerio del Interior. Figure 4.4 shows the purchased surface rights. The surface rights currently held by ACMSA cover the area currently being explored by McEwen Mining. The area represented by the surface rights are also considered to be more than adequate for potential development of the mine, associated processing facilities and infrastructure considered in this technical report.

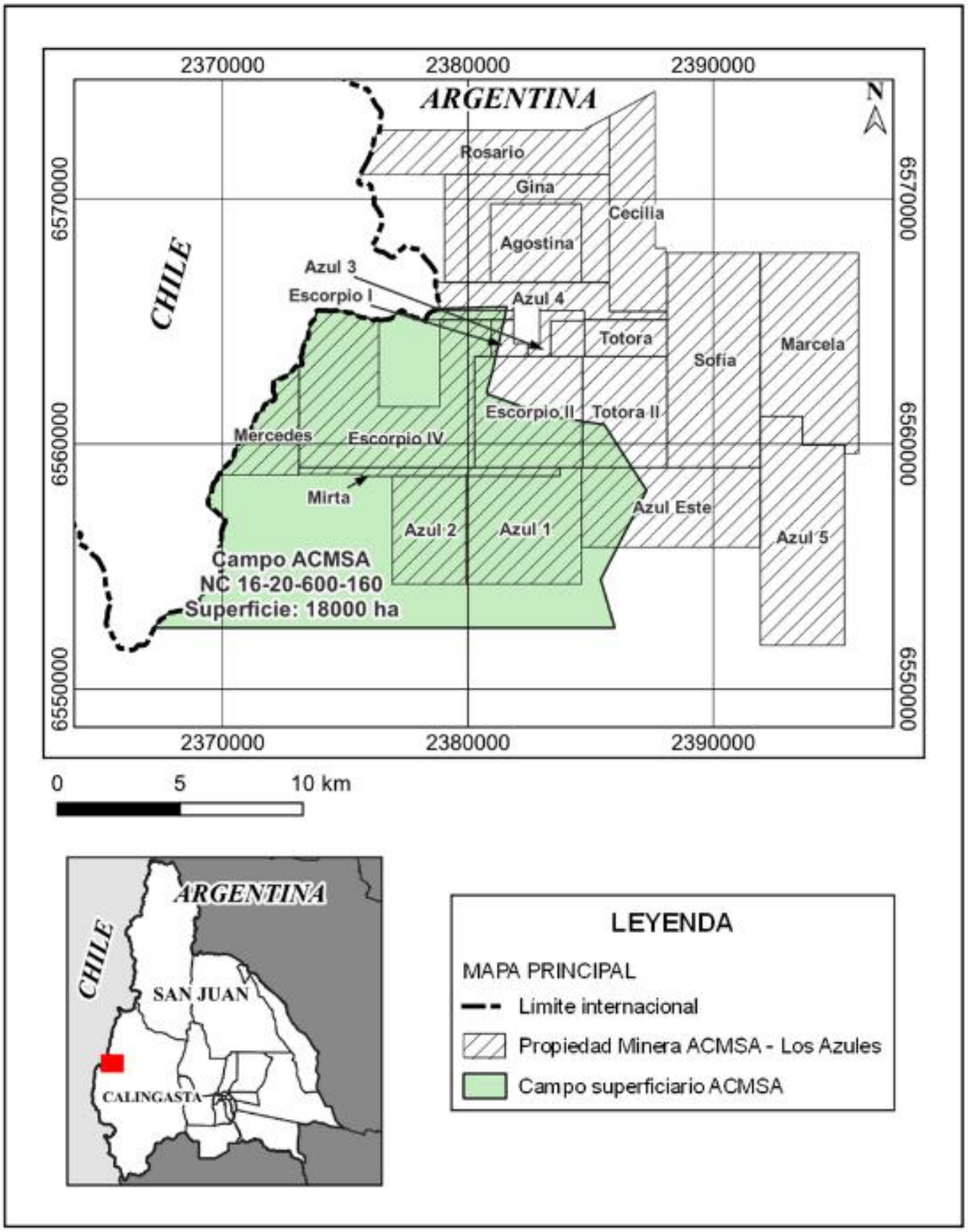


Figure 4.4: ACMSA Owned Propiedad Minera (Mining Rights) and Campo Superficiario (Surface Rights) (McEwen, 2022)

The green area on Figure 4.4 indicates the limits of McEwen surface rights (land holdings) relative to the mining rights. The western boundary of the property is the border with Chile. Below, Figure 4.5 shows the surface right owners within and adjacent to the Los Azules project, with the land held by Illanes Mery colored in orange, Campo Cortez Monroy to the south in yellow, Cortez Angel to the north in green, and the Estomonte property to the east in peach color.

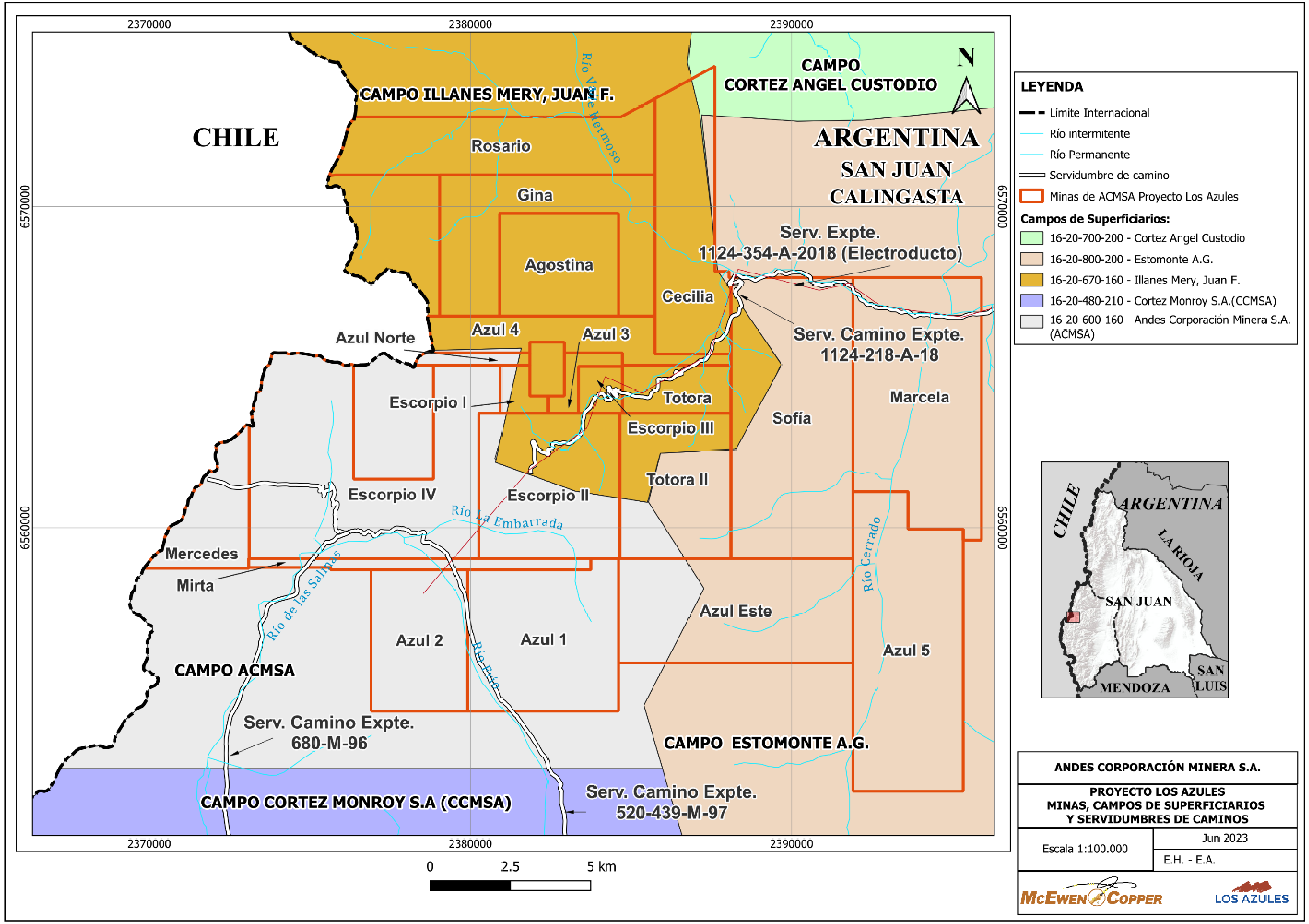


Figure 4.5: Map of mineral concessions and surface rights (campos) within or adjacent to project area (Vargas & Galindez/McEwen, 2022)

None of the mineral concessions that are not owned by McEwen but that fall within the McEwen owned surface rights will have any impact upon the Los Azules Project development. All the Los Azules facilities are located on lands where McEwen has both the surface and mining rights.

4.4 ROYALTIES AND RETENTIONS

There are no outstanding royalties, payments, or other agreements or encumbrances to which the property is subject other than a one-time USD $500,000 payment to be made to Hugo Bosque upon delivery of a feasibility study.

San Juan Province charges a 3% royalty based on the “mine head value”. The 3% is charged on the sale price less some costs (excluding depreciation of fixed assets and extraction costs). In other words, deductible expenses include: i) transport and freight costs; ii) crusher, milling and process (beneficiation) costs; iii) commercialization cost; iv) administrative cost (not related extraction cost) and v) smelting and refining costs. However, since July 2011, the Province of San Juan, through an agreement with the mining companies in operation, modified the calculation of the "mine head value" by a taxable base on gross sales, without any deductions. This change in the methodology has not been reflected in the legislation, and is implemented through agreements according to a series of conditions (e.g., metal prices, tax burden, etc.)

In addition, the Province of San Juan has an unlegislated practice of negotiating a voluntary contribution to a trust (“fideicomiso” in Spanish), usually 1.2% or 1.5% and on the same calculation basis as mining royalties (on gross sales, without deductions). These contributions are intended to finance infrastructure projects in the province, especially in the local area impacted by the mining operation.

TNR Gold Corp has a 0.4% NSR across the project, and McEwen has a 1.25% NSR.

4.5 BACK-IN RIGHTS

There are no back in rights.

4.6 ENVIRONMENTAL LIABILITIES

At the present time, there are no known environmental liabilities at the Project site, since it is an exploration project. Reclamation activities are comprised of re-grading the drill pad sites, access roads at site and some portions of the main access road to the Project site.

There are two principal activities that have environmental impacts in the Project area. One is the overgrazing of pasture lands and the second is access roads and drill platforms on the property.

Seasonal grazing by “veranadas” from Chile takes place on sparse foraging resources and wetlands in the Project area. The “veranadas” with large animal herds (primarily goats) have affected:

- Vegetation coverage on the grazing land.
- Erosion of the borders of streams.
- The surface drainage capacity due to compaction.

There are numerous previously existing excavation areas for exploration roads in the Project and surrounding areas, including drilling platforms.

4.7 PERMITTING REQUIREMENTS

Argentine laws and regulations differentiate between prospecting, exploration, and exploitation activities. It is understood that exploration activities include mapping, sampling (including bulk samples), geophysics, trenching and drilling, whereas prospecting activities include only mapping and sampling.

There are different sectorial permits that are required to conduct mining activities, but the most relevant ones are the ones associated with environmental permits. The provisions related to environmental protection applicable to mining activity were established in 1995 by the General Environmental Law and have been incorporated in Title Thirteen of the Mining Code.

The federal government is empowered to issue Minimum Environmental Protection Standard Laws (MEPSL), applicable in the whole country by the respective local authorities. The provinces are allowed to supplement and regulate the MEPSL with more stringent local or provincial environmental regulations.

4.7.1 Exploration and Prospecting Requirements

The main permit for the exploration phase at Los Azules is the Environmental Impact Declaration (Declaracion de Impacto Ambiental or DIA in Spanish), which must be updated at least every two years with the provincial mining authority. An EIA must be presented for each phase of the project development: prospecting, exploration, exploitation (including industrialization, storage, transportation, and marketing of minerals). The last DIA renewal was received on June 2, 2021, and the resolution was issued on June 17, 2021.

Ancillary permits for water usage (domestic, drilling and dust mitigation), archeological research and investigation, hazardous waste, sewage, and domestic waste facilities are renewed on an annual basis before the commencement of the exploration season. The permit renewals are expected to be approved on time as per prior exploration seasons. All necessary permits have been obtained for the work currently being carried out on the Project.

4.8 PERMITTING REGULATIONS

There are five main legal requirements that impact the Project during the different stages of development: environmental regulation, mining regulation, hazardous waste regulation, health and safety regulation and the Mining Investment Law.

4.8.1 Environmental Regulation

Environmental regulations applicable to Mining have four sources:

- Environmental specific regulations applicable to mining arising from the Mining Code,
- Environmental laws issued by Federal Congress as MEPSL applicable to all activities including mining,
- Local environmental regulations issued by the provinces the MEPSL and applicable to all activities including mining,
- Additional local/provincial environmental legislation if this does not contradict or is less stringent than a MEPSL,

Lack of compliance or other infringement of the environmental obligation may result in penalties ranging from fines to suspension of works or closure of the mine, but without effect upon title or ownership of the mining concession.

4.8.2 Mine Regulation

The acquisition, exploitation and use of minerals are regulated by the Mining Code (National Law 1919) and Provincial Law 688-M. In addition, the province of San Juan has adopted National Law 24585, environmental protection for mining activities.

4.8.3 Hazardous Waste Regulation

Other regulations affecting the Project are related to Hazardous Waste regulations set forth in National Law 24051, adopted by the province of San Juan. This law regulates the generation, handling, transportation, treatment, and disposal of hazardous waste materials.

4.8.4 Health and Safety Regulation

Health and safety regulations require that a mining company must hire an Occupational Hazard Insurer (ART, as per the acronym in Spanish) to identify and evaluate occupational hazards and to design preventive and emergency programs. For the mining sector, companies must give priority to riskier occupational activities and employee training.

4.8.5 Mining Investment Law

Mining Investment Law 240196 includes article 23, which relates to the preservation of the environment. To prevent and correct any impacts to the environment due to mining activities, companies may establish a special accounting provision for that purpose. The annual amount shall be left to the criterion of the company but shall be considered deductible for income tax purposes up to a sum equivalent to 5% of the operational costs of material extraction.

4.8.6 Archaeological Sites

Archeological sites are managed by the Ministry of Culture of the province of San Juan. Sites can be removed by applying to the Ministry for a permit that describes the tasks to be carried out, prepared by qualified professionals. A site plan of the proposed work area should also be submitted with the permit application. A permit is expected to take two months to obtain, and the work plan should be submitted one year in advance.

4.9 GLACIER PROTECTION LEGISLATION

In 2010 Argentina passed Federal Legislation to protect its water resources contained in glaciers prohibiting activities which could affect these resources.

- It mandates cataloguing all glaciers in the territory and their status of conservation or impact.
- The legislation created a conflict regarding federal versus provincial (state) ownership of the natural resources which sits now in front of the Supreme Court.

Following suit in July 2010, the Province of San Juan enacted provincial law 8144, “Glacier Protection Law”, in a compromise with Federal Law, which, among other things, restricts disturbance of glaciers by mining activities. In addition, the federal Congress issued a MESPL on the protection of glaciers and periglacial environment (Law 26639), which is different from the provincial law.

Since 2011, several independent studies have been conducted by the Company.

- No uncovered, or “white glaciers” or ice glaciers, have been identified on Los Azules property;

however, several small cryogenic geoforms identified as “rock glaciers” have been mapped onsite.

- The company believes it is in full compliance with the law, not having disturbed any glaciers that could be deemed a water resource.
- The provincial inventory has been completed with no rock glaciers having been found to be affected by exploration activities at Los Azules.
- None of these rock glaciers will be impacted by the company’s future exploration activities or the development of a mining project.
- The water storage and watershed contribution from any rock glaciers mapped will be evaluated as part of the Environmental Baseline Studies required for permitting.

The Los Azules exploration area was audited by a multi-agency environmental audit team in March 2013. There were no adverse findings and the audit results indicated that McEwen Mining is in full compliance in all areas protected by the provincial law.

In 2016 the Provincial government started to catalogue the glaciers present in the provincial territory to determine if any impacts have taken place or if any glaciers could impede mineral development in the province.

4.10 ENVIRONMENTAL BASELINE STUDIES

Between 2007 – 2012 Ausenco Vector has monitored and collected environmental baseline data on surface and groundwater volumes and quality, soils, flora and fauna, archeology, and weather. Several other consultants have been involved for all environmental aspects. Ausenco Vector has also studied the boggy wetlands, locally referred to as “vegas”.

Ausenco Vector implemented a plan to relocate or compensate the vegas where they may be impacted by the project. The plan did not produce satisfactory results. Andes Corporación Minera S.A. requested to the provincial environmental authority to propose an alternative compensation criterion however there has not been any response from the authority to date.

Dr. Andres Meglioli, of Mountain Pass LLC, has been monitoring cryogenic geoforms in the project area since 2011.

The environmental baseline data on surface and groundwater volumes and quality as well as the flora and fauna data collection and additional studies on the vegas (including a compensation proposal) have been conducted since 2013 by the Insituto de Investigaciones Hidraulicas, a research center of the National University of San Juan, through their senior biologists Juan C. Acosta and Hector J. Villavicencio. These are ongoing studies contracted by McEwen. After each drilling season, a report is prepared by the consultants and issued to McEwen that summarizes the work completed through the season.

In late 2017 and throughout 2018 McEwen in conjunction with consultants and specialists performed full year baseline studies for fauna, flora and hydrology that will require extended site access through all seasons and support using mules and using helicopter. Geotechnical studies such as water permeability tests may also be performed to enhance the existing data set.

In 2022 additional environmental baseline work was undertaken with the objective of completing an IIA (Informe Impacto Ambiental) equivalent to the English EIA for exploitation of the mine. The objective is to include all of this and prior baseline data into the IIA documentation that McEwen intends to be submitted to the authorities in the second quarter of 2023.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

Access to the site is through provincial routes to mining access easements granted under Argentine law. The province of San Juan is bisected by RN 40, which extends the length of Argentina. The map showing access routes into the site, including a future northern route, is provided in Figure 5.1: Access Roads and Camps.

The main access to the Los Azules Project from Villa Calingasta (Calingasta Department – Province of San Juan), is through Provincial Route No. 437 from the intersection with Provincial Route No. 406 to La Alumbrera, an approximate distance of 25 km. From there the mining road begins. This is the Exploration Road. The road includes the Candadito support camp that is covered by an easement granted under file No. 1124.660-A-2012. The route has a length of 87 km and is covered by a road easement recorded in administrative file No. 520-0439-M-97 on behalf of ACMSA.

The exploration road from La Alumbrera crosses the Calingasta River and continues in a westerly direction parallel to the Calingasta River and the Arroyo de la Totora, crossing from – east to west – three passes: Cuesta del Gringo, la Totora, Concontita and Cabeza de León. The road continues through the valley of the Frío River until the intersection with the Arroyo de Embarrada where the Embarrada camp is located, on the project site. Approximately 5 km to the east of Embarrada is the Los Azules camp.

ACMSA is responsible for the maintenance and signage of the route. According to Argentine law, the use of roads open to one or more mines shall be extended to all mines of the same mineral or seat, if they are paid for the costs of the work and maintenance expenses in proportion to the benefits they receive. The existing Exploration Road was upgraded in 2022/2023 to allow for access by larger vehicle traffic and safer transit. The road will continue to be maintained to provide seasonal secondary site access and support the incoming high voltage powerline routing.

Los Azules has alternative access to the project. This access is known as the Southern Access Road, which begins at the town of Barreal in the department of Calingasta and connects to Provincial Route No. 400 at La Junta and from there continues in a westward direction, and then North-Northwest, on a third-party easement issued to Glencore for the El Pachón project (file No. 156.424-C-72). The road continues up the La Pantanosa River to the Colorado River, partially following it north, until it reaches the Los Piuquenes River, Arroyo Verde and finally follows the Las Salinas River reaching the Los Azules camp. The distance from Barreal to the Los Azules Project is approximately 240 Km. Part of the Barreal – Los Azules route is covered by a separate road easement, (file No. 0680- M-96) for the benefit of Andes Corporación Minera SA.

The main access during operations will be by the Southern Road route to begin the project, this road will be upgraded for the expected year-round traffic travel to and from the site and extends from the town of Barreal. The routing and costs for the Southern Road upgrade (approximately 192 km) were provided by Ruiz y Associados Consultoras S.R.L. (RyAC) and were estimated to be USD $138 million.

Finally, Los Azules has a road project to access a project from the north which is to-date only partially constructed and known as the northern route. This route has an easement granted by file No. 1124-218-A-2018.

The following figure shows the accesses mentioned above, the support camp for the "Candadito" road and the "Los Azules" and "Embarrada" Camps. The latter constitute the project camps and are located at the project site.

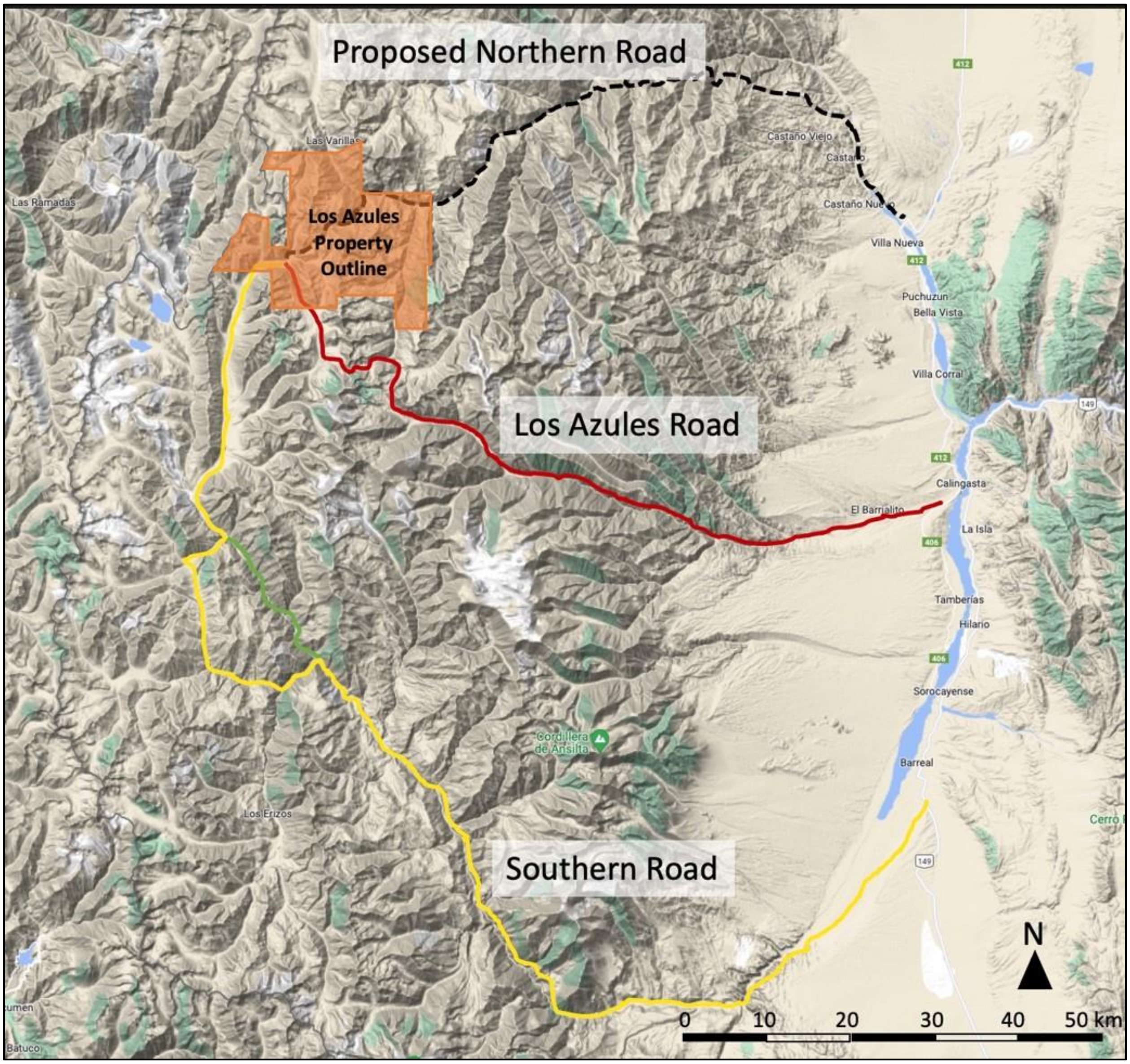


Figure 5.1: Access Roads and Camps

5.2 SURFACE RIGHTS

According to Argentine law, mineral rights supersede the overlying surface rights, and the holder of the latter is legally unable to impede access to the exploration or extraction of underlying mineralization. Fair compensation is provided to the surface rights holder for access and usage of the land in conjunction with exploration activities and mining operations. In January 2010 "Minera Andes", a company 100% owned by McEwen Mining Inc., purchased 18,000 ha of surface rights covering the Los Azules deposit and the associated surface facilities, as they are currently envisioned. The extent of surface rights and the proposed surface facilities are illustrated in the Los Azules General Arrangement in Figure 5.5.

5.3 CLIMATE AND LENGTH OF OPERATING SEASON

Typically, the field season at Los Azules starts in December and runs through to the end of May due to limited access. However, last year, access to the site was maintained through mid-June, and depending upon the winter snowpack conditions, it is possible in some years to access the site as early as October as was the case in the last two drill seasons.

A weather station was installed near the camp site in mid-2010 to obtain local climatic information. The station is powered by a solar panel and collects meteorological parameters at 30-minute intervals. The station was manufactured by Coastal Environmental and is built around the ZENO® 3200 datalogger. Data communication is via an Iridium satellite modem. Data is downloaded using a companion base station located in the United States. The weather station uses a stand-alone tower with sensors to obtain the following parameters:

- Wind direction (degrees)
- Wind speed (m/s)
- Wind gust (m/s)
- Standard deviation of wind direction (degrees)
- Air temperature (°C)
- Relative humidity (%)
- Barometric pressure (mPa)
- SW solar radiation (W/m^2)
- Rain intensity (mm/min)
- Accumulative precipitation (mm) in precipitation bucket.
- Contents of precipitation bucket (mm^3)
- Snow depth (mm) (installed Q2 2013)

Two new weather stations were purchased and installed this season to provide better coverage over the site.

Considering the types of recorded parameters, the Los Azules meteorological station meets the World Meteorological Organization (WMO) standards for a Principal Climatological Station.

Figure 5.2, Figure 5.3 and Figure 5.4, which were obtained from the site meteorological station, present monthly weather data for temperature, total precipitation, and wind speed. Snowfall accumulations are recorded by the station as snow water equivalent. Snowfall in the Project area is light, although heavy winter accumulations are common on the two high passes on the access road.

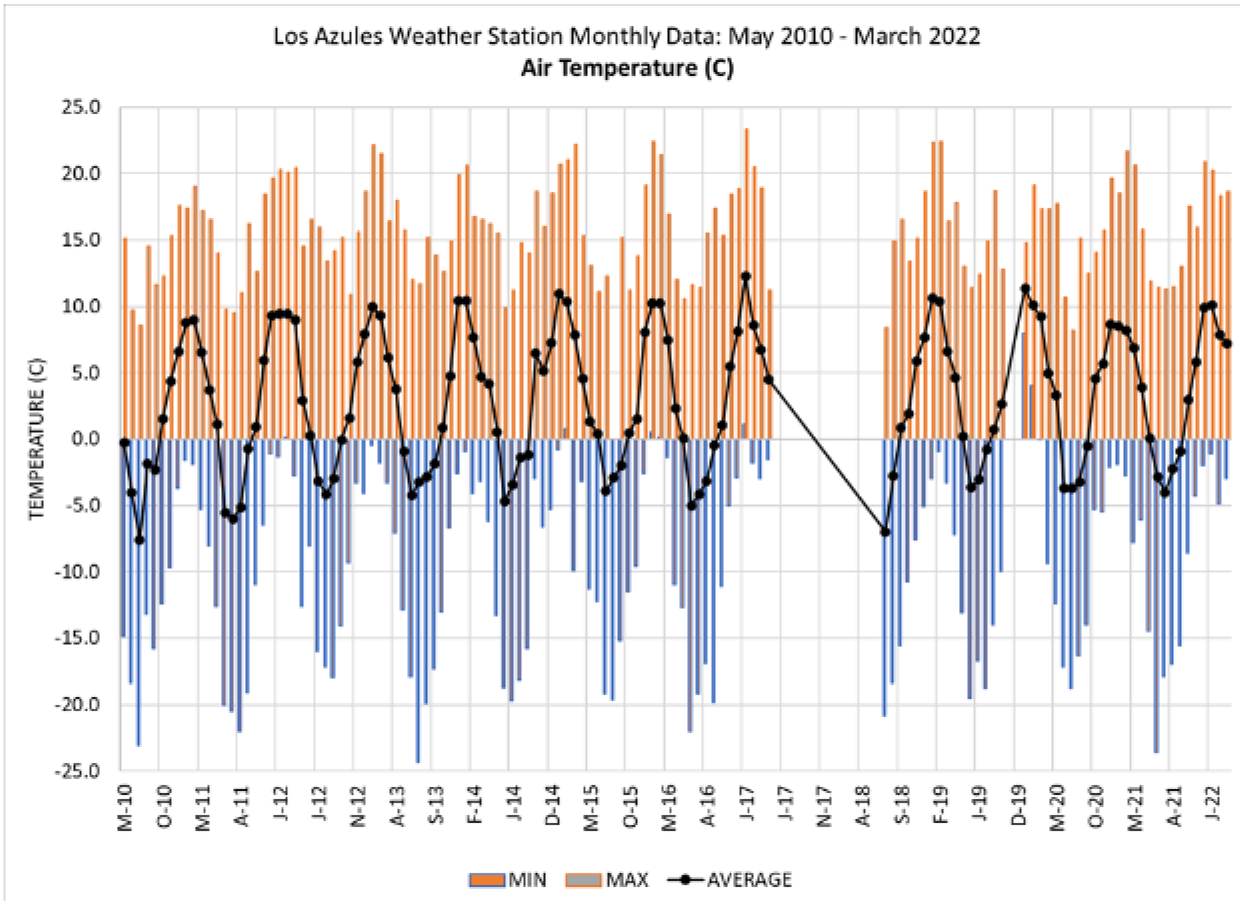


Figure 5.2: Monthly Temperature Data Apr-17-Jun-18, Nov-19 (McEwen 2022)

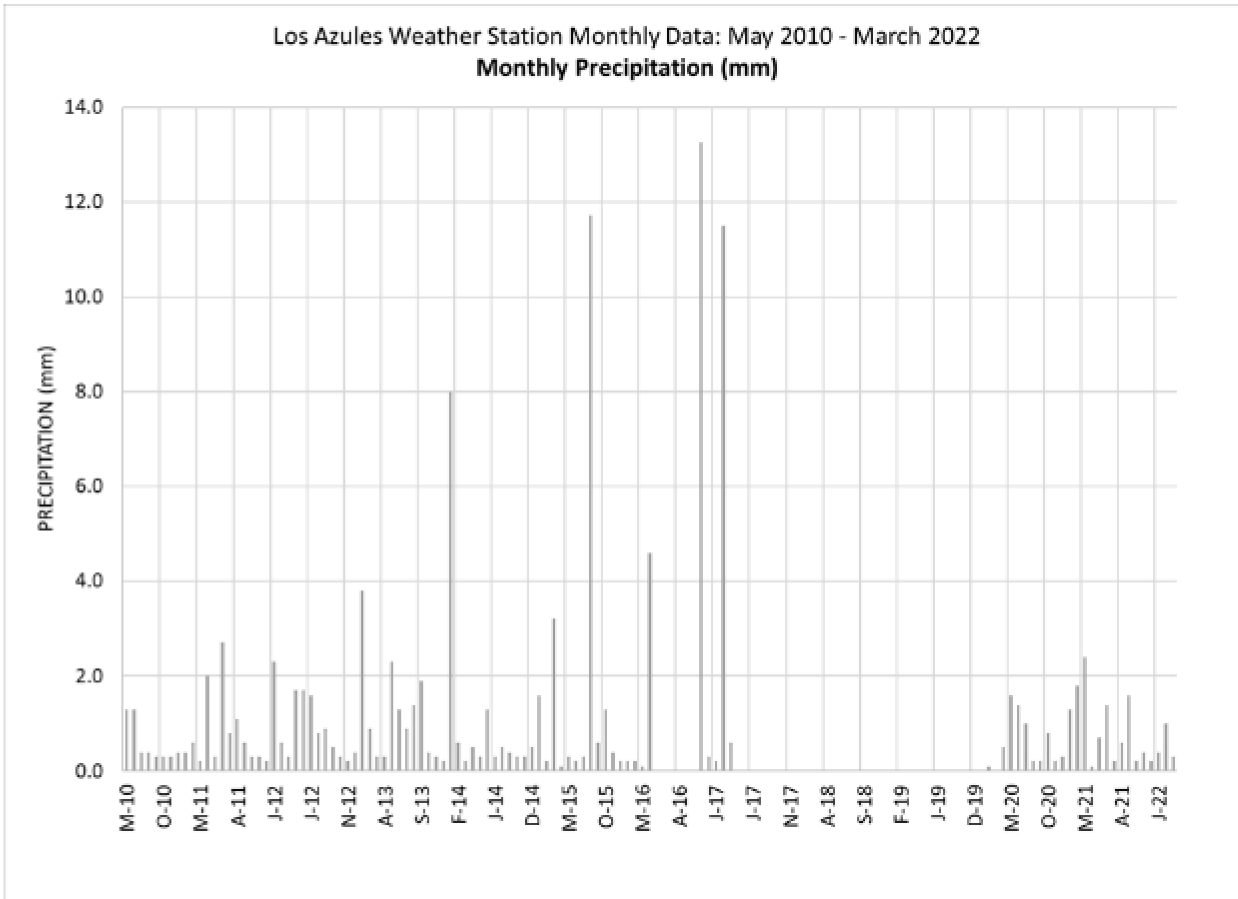


Figure 5.3: Monthly Total Precipitation Data – no data recorded Apr-17-Jun-18, Nov-19 (McEwen 2022)

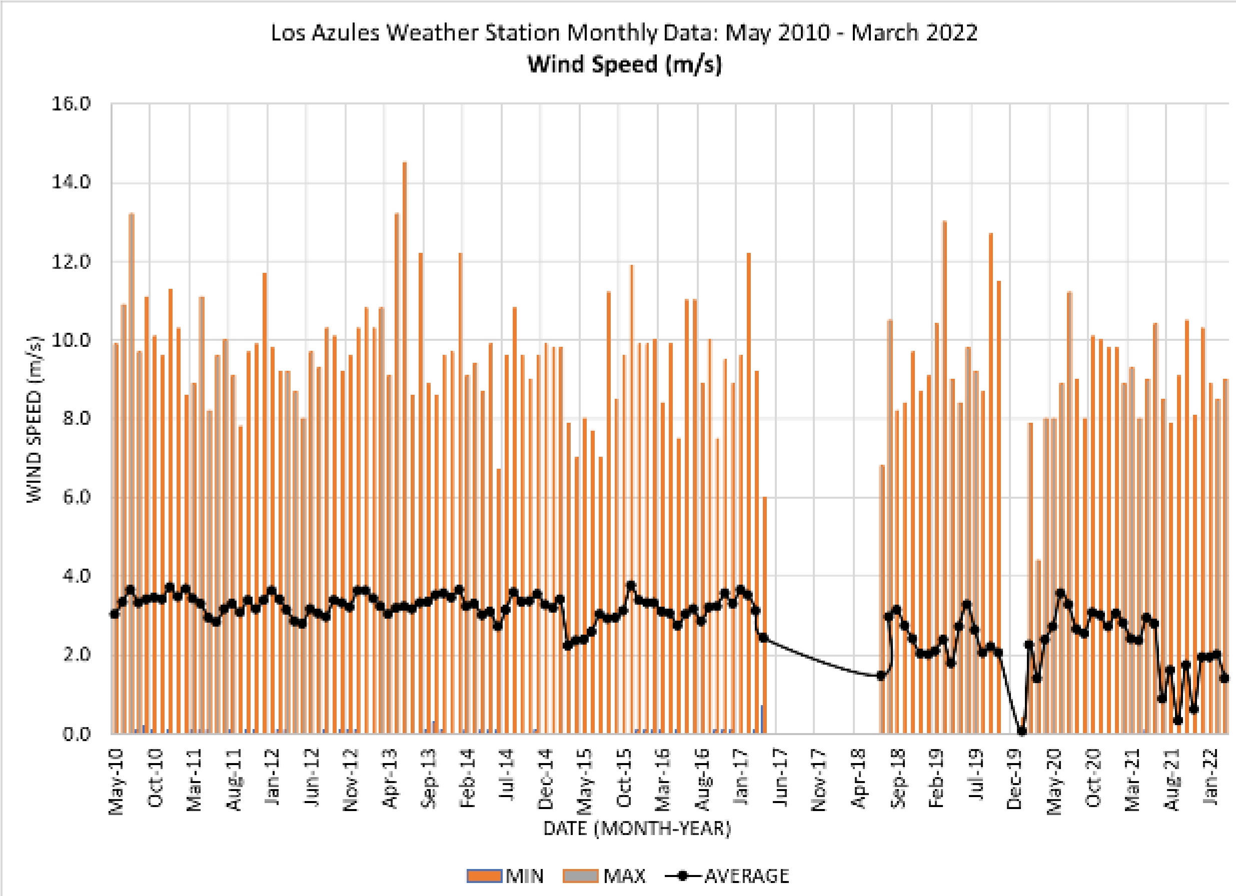


Figure 5.4: Monthly Wind Speed Data – no data recorded Apr-17-Jun-18, Nov-19 (McEwen 2022)

5.4 LOCAL RESOURCES AND INFRASTRUCTURE

The Project area is remote, and no infrastructure is present in the Project area. There are no nearby towns and/or settlements. Exploration operations are carried out by means of two-man camps within the Project development area.

5.4.1 Available Personnel

Historically, Villa Calingasta was a mining town whose economy was supported by the exploitation of alum deposits, which is used in water purification and gold mining at the Casposo mine. The United Nations Development Program (UNDP) and other national and international agencies have established programs to help remediate certain environmental liabilities associated with the alum mining activity.

The current principal economic activity of the area is agriculture with fruit trees (apple and walnut) as the principal activity, in addition to employment in the public sector. Lesser activities include the following:

- Timber and vegetables.
- Wood manufacturing activities.
- Cider manufacturing.
- Tourism (hotels, restaurants).

- Commercial activities (shops).
- Public service (health, safety, education).

According to the Argentine National Census Bureau (INDEC) 2010 census, the population of the Calingasta Department (subdepartment) was 8,453 people. In 2015, the population was estimated to be 9,151 and was projected to be 9,641 by 2022. The 2022 census was not available at the time of writing this report.

5.4.2 Power

The 2022 PEA considers the cost of connection at the existing Calingasta substation and routing a 220kV powerline parallel to the Exploration Road. A study by Energía Provincial Sociedad del Estado (Provincial Energy Society of the State, EPSE), a state-owned company, showed that the province was well suited to support the development of projects providing renewable energy to the province and projects.

For the initial phase of the project, an indicative price was received by YPF Luz that could provide 100% renewable power to the project for a cost of $67 per MWh for a 5-year contract.

5.4.3 Water

Surface water is available on the property in adequate amounts for McEwen Mining's exploration activities. Preliminary hydrological evaluations conducted by Ausenco Vector have indicated that there are sufficient sources of water to operate the Los Azules mining and processing facilities and to provide the necessary fresh water needed to house employees at the mine site.

5.5 TOPOGRAPHY, ELEVATION AND VEGETATION

The Project is in a broad valley, formed by faulting and glaciation and is bounded by steep ridges to the east and west. The deposit is centered on La Ballena ridge (English translation: the whale), a low NNW-SSE trending ridge. The Project area is rugged and ranges in elevation from 3,500 to nearly 4,500 masl. Vegetation is sparse and is absent at higher elevations.

Long, narrow vegetated areas ("vegas" in Spanish) occupy the valley floors on either side of La Ballena. These vegas are fed by ephemeral spring-water and snowmelt, but also reflect the groundwater regime as well, with standing water levels at approximately 3,600 m in elevation. Springs are noted at approximately 3,790 m in elevation upstream of the vegas along the west side of La Ballena ridge. Groundwater-fed springs and marshes are also noted around the range to the west of La Ballena between 3,800 and 3,900 m in elevation and along the eastern flank of the Cordillera de la Totora. These vegas feed the westerly flowing Rio La Embarrada, which joins the Rio Frio to the west before turning south into the Rio de las Salinas, a main tributary to the San Juan River.

Deposits of glacial debris (morainal materials) and scree account for much of the surface area covering the Los Azules deposit and adjacent mountainsides. In the area of the deposit, these materials locally exceed 60 m in thickness, but on La Ballena ridge, the cover is less, starting at 10 m thickness.

There are no covered or uncovered "white glaciers" (classic ice glaciers) within the Project area, although there are several small cryogenic geoforms identified as rock glaciers near the Project area, but these will not be impacted by McEwen's exploration or Project development activities.

5.6 AVAILABILITY OF AREA FOR MINE AND PROCESSING FACILITIES

The area around Los Azules provides limited options for siting of the Heap Leach Facility (HLF), Filtered Tailings Storage Facility (FTSF), mine rock storage facility (MRSF), low-grade process stockpile, the processing facilities and other infrastructure needed. All facilities and permanent infrastructure considered in this PEA are located within areas where ACMSA has both mining and surface rights or a current easement at this time.

The exact location of the project development surface facilities is yet to be finalized and requires hydrogeologic and geotechnical site investigations to support detailed design work to be performed.

The proposed Site General Arrangement of the various project facilities is presented below in Figure 5.5. This is further described in Sections 16, 17 and 18.

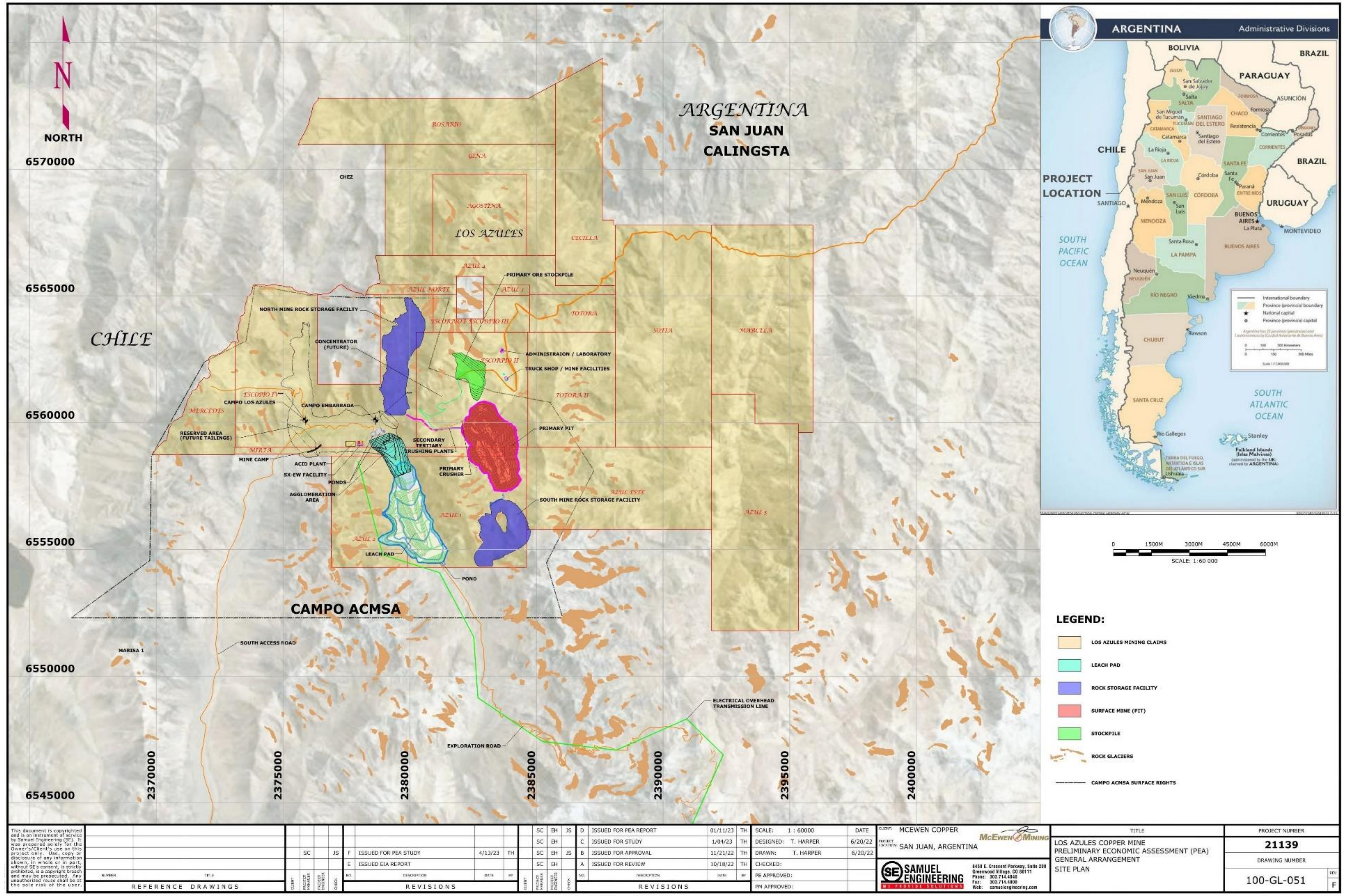


Figure 5.5: Los Azules Project Site General Arrangement

6.0 HISTORY

There are no formal records of exploration in the Project area prior to 1980. The only important active project in the area prior to 1980 was the El Pachón porphyry copper project, now owned by Glencore plc (Glencore), which is located approximately 90 km south of Los Azules. Evidence of prospecting (small trenches or pits) exists on some of the concessions.

6.1 PROPERTY HISTORY

In 1994, Minera Andes, through its subsidiary Minera Andes S.A. (MASA), acquired lands in the southern portion of the Los Azules area. Battle Mountain Gold Company (BMG) acquired lands immediately to the north through an option from Solitario Argentina S.A. (SASA). For the next couple of years, both companies independently explored for gold on their respective land holdings.

In 1998, a new access road was constructed by BMG while it conducted airborne geophysical surveys, mapping, trenching and drilled several reverse circulation (RC) holes. A large hydrothermal alteration zone associated with dacite porphyry intrusions and stockwork structural zones was recognized in the Project area and Minera Andes signed a Letter of Intent with BMG to form a joint venture to explore the combined land package.

In 1999, Minera Andes and BMG signed a definitive joint venture agreement. BMG subsequently drilled additional RC holes and porphyry copper mineralization was intersected close to the property boundary; however, no drilling was done on the Minera Andes properties.

In 2000, BMG merged with Newmont Mining Corporation (NMC). No further work was done by BMG/NMC, and the joint venture was allowed to dissolve without BMG earning any interest in the Minera Andes or Solitario lands. At that time, capitalizing on a surveying error, Mr. Hugo Bosque, an attorney from San Juan, acquired a small strip of land between the Minera Andes and Solitario lands.

In 2003, MIM Argentina S.A. (MIM) optioned the Bosque and Solitario lands and began exploration work. Independently Minera Andes began exploration on its own lands at Los Azules.

In 2005, a Letter of Intent was drafted between Minera Andes and Xstrata Copper (successor to MIM) for earn-in rights on the combined land package. More exploration occurred over the next couple of years.

International Copper Mining, Inc was incorporated in British Columbia, Canada on March 2, 2006, to hold ownership of exploration properties including the Los Azules property.

On November 2, 2007, Minera Andes Inc. entered into an Option Agreement with Xstrata whereby the exclusive right was granted to Minera Andes to explore and evaluate the area called “Los Azules” which included several properties owned by Xstrata as defined in the Option Agreement.

On May 15, 2009, the parties to the Option Agreement, together with Andes Corp. and Los Azules Mining, Inc. (LAMI), each wholly owned subsidiaries of Minera Andes, signed an Assignment and Amending Agreement whereby Minera Andes properties “Azul 1” and “Azul 2” were transferred to Andes Corp. together with the right to acquire from Xstrata 100% interest in and to the Los Azules properties (as defined in the Option Agreement). In addition, Minera Andes S.A. assigned and transferred to LAMI all of MASA’s right, title, benefit, and interest in, to and under the Option Agreement (as defined in the Assignment and Amending Agreement).

On May 29, 2009, Los Azules Mining Inc., exercised the option, by delivery of an Earn-in Notice (pursuant to the Option Agreement as amended by the Assignment and Amending Agreement) to acquire 100%

interest in Los Azules properties (as defined in the Option Agreement). Therefore, Xstrata subsequently transferred to Andes Corp. all its properties located in the Los Azules area.

On September 30, 2009, Xstrata elected not to exercise its option to acquire a 51% interest in the Project and has no remaining interests in Los Azules.

In January 2012, Minera Andes Inc., was acquired by US Gold Corporation, which was subsequently renamed McEwen Mining.

Certain portions of the northern part of the Project that were formerly held by Xstrata and transferred to Minera Andes following the termination of the Option Agreement were subject to an underlying option agreement between Xstrata and a subsidiary of TNR Gold Corp. This agreement was the subject of litigation in the Supreme Court of British Columbia, Canada.

The final Transfer Agreement between TNR Gold Corp (along with Compañia Minera Solitario Argentina S.A.), and McEwen Mining (including Los Azules Mining Inc. and Andes Corporacion Minera S.A.) was executed on October 16, 2014, superseding the prior settlement agreement and transferred all rights on the Solitario Properties from TNR to McEwen Mining in exchange for an NSR royalty of 0.4% and TNR relinquishing the back-in right. The NSR terms are subject to a Net Smelter Returns Royalty Agreement executed on October 29, 2014, between Compañia Minera Solitario Argentina S.A. and Andes Corporacion Minera S.A.

International Copper was continued from the Province of British Columbia to the Province of Alberta on December 31, 2012, as International Copper ULC. The Corporation was converted to a limited liability corporation and changed its name to McEwen Copper Inc. by way of articles of amendment dated August 20, 2021. At the creation of McEwen Copper Inc., it held a 100% interest in the Los Azules Copper project. McEwen Mining Inc. also transferred a 100% interest in the Elder Creek Exploration project to McEwen Copper to create a copper investment vehicle.

In 2018, ACMSA filed a request to group all the Mining Permits together, file #1124.553-A-2018, in such way that, once all surveys are approved, all the Mining Permits be considered as one larger Mining Permit. This will allow investments to be distributed across the larger permit group and eliminate the need to spend on each individual Mina. It is expected that this request by ACMSA could be favorably resolved during 2023.

In July 2021, a private placement financing was initiated for McEwen Copper seeking to raise USD $80 million at a price of USD $10.00 per common share. This financing subsequently closed in three tranches on August 23rd, 2021, June 21st, 2022, and August 31st, 2022, respectively. On August 23rd, 2021, a company controlled by Robert R. McEwen (Chairman and Chief Executive Officer of McEwen Mining) purchased 4,000,000 shares for USD $40,000,000. On June 21st, 2022, the Victor Smorgon Group purchased 1,000,000 shares for $10,000,000 and other investors purchased 500,000 shares for an additional $5,000,000. On August 30th, 2022, Nuton LLC (a Rio Tinto Venture) purchased 2,500,000 shares for USD $25,000,000 and other investors purchased 185,000 shares for an additional USD $1,850,000. In total, 8,185,000 shares were sold in the private placement for gross proceeds to McEwen Copper of USD $81,850,000.

During 2022, a 1.25% net smelter return (NSR) royalty was created encumbering the Los Azules project, which is held by McEwen Mining Inc.

As of December 31, 2022, McEwen Copper had 25,685,000 common shares issued and outstanding on a fully diluted basis, of which 17,500,000 common shares were owned (68.1%) by its parent company McEwen Mining Inc., and 8,185,000 common shares were owned by third parties and some affiliates.

The challenge by ACMSA for the peripheral properties under threat of forfeiture, Gina, Sofia, Torora II, and Marcela, was resolved in the company's favor and the forfeited rights were returned by the Mining Council on December 29, 2022.

FCA Argentina S.A., a subsidiary of Stellantis N.V. ("Stellantis"), invested ARS $30 billion in Argentina to acquire shares of McEwen Copper in a two-part transaction that closed on February 24th, 2023.

Nuton LLC, a Rio Tinto Venture, invested $55 million to acquire shares of McEwen Copper in two transactions which closed on August 31, 2022, and March 15th, 2023.

Both the Stellantis and Nuton investments included investor rights and product purchase rights, discussed in detail in Section 1.1.

As of 2023 ACSMA/McEwen Copper continues to hold 100% of the Los Azules development and associated land holdings and mineral concessions and easements and continues to perform seasonal infill drilling and studies with a view to eventual project development.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

This section relies heavily on geological studies conducted by Richard Sillitoe (2014) and Vázquez (2015) as well as other references cited in the section. A more fulsome description of the techniques used in the geological modelling used as a base for the mineral resource estimate is provided later in Section 14.3 (after Mortimer, 2022).

7.1 REGIONAL GEOLOGY

Los Azules is a porphyry copper deposit located in western San Juan Province in west- central Argentina. This region is characterized by a series of north-south elongated mountain ranges that rise in altitude from east to west to form the rugged Andean Cordillera along the border between Argentina and Chile. Los Azules lies within the highest altitude Cordillera Principal at an elevation of about 3,700 masl (Figure 7.1).

The Cordillera Principal is composed of strongly folded, faulted, and elevated Paleozoic-Mesozoic sedimentary and volcanic lithologies (Gondwanide orogeny) overlain by extensive Upper Miocene ignimbrites (Andean orogeny) as shown in Figure 7.2. Eocene to early Miocene volcaniclastic strata in the region accumulated in an extensional basin followed by plutonic intrusion and contractional deformation from 19 Mya to 16 Mya. These units were overlain and intruded by 16 Mya to 7 Mya volcanic flows and pyroclastic units with comagmatic 12 Mya to 8 Mya plutons and porphyry systems. This was followed by a compressional event at 8 Mya to 5 Mya with important crustal shortening, thickening, and regional uplift (Sillitoe and Perello, 2005). Figure 7.2 also shows the relative locations of other major mining projects in the area.

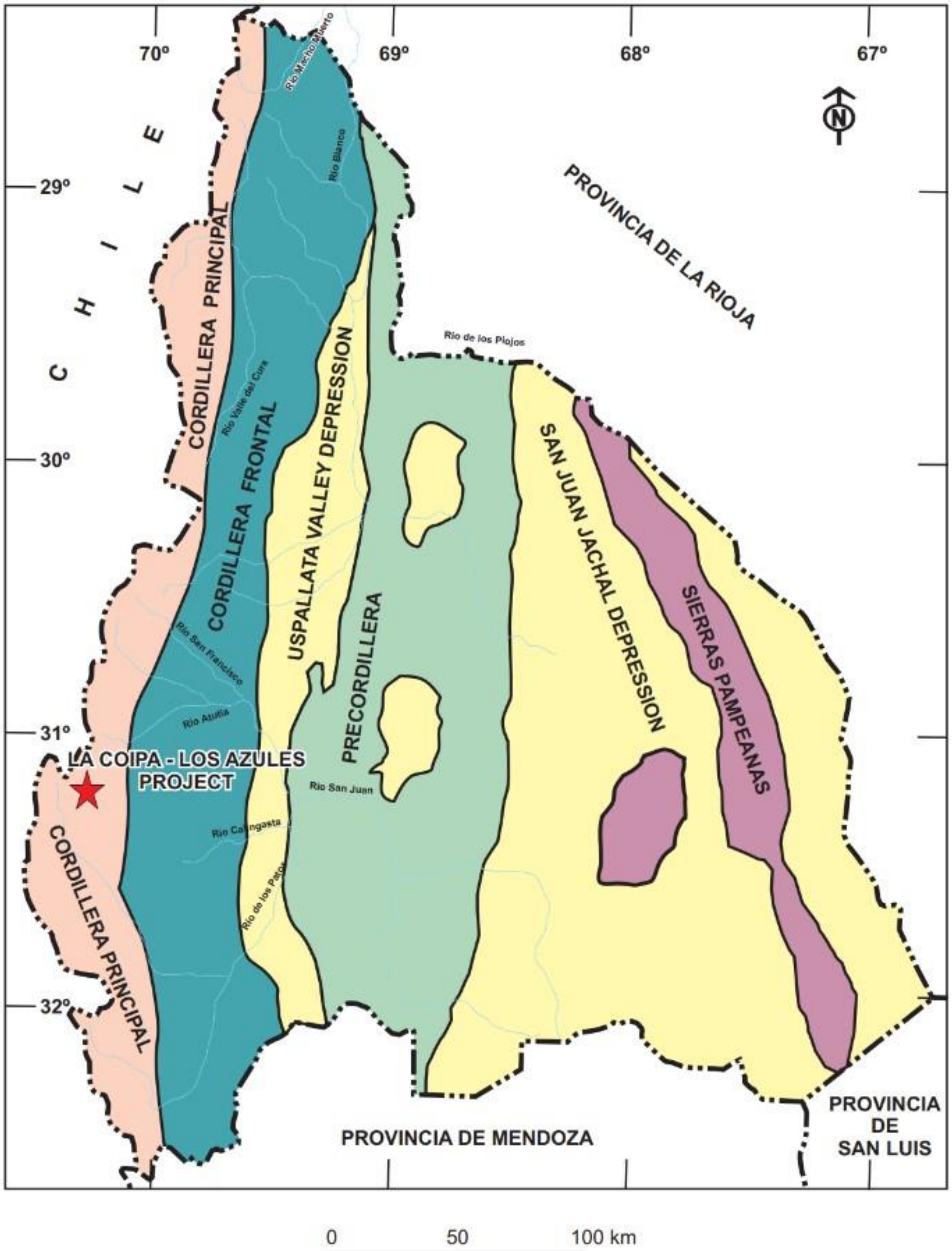


Figure 7.1: Physiographic features of San Juan Province, Argentina (Rojas 2010)

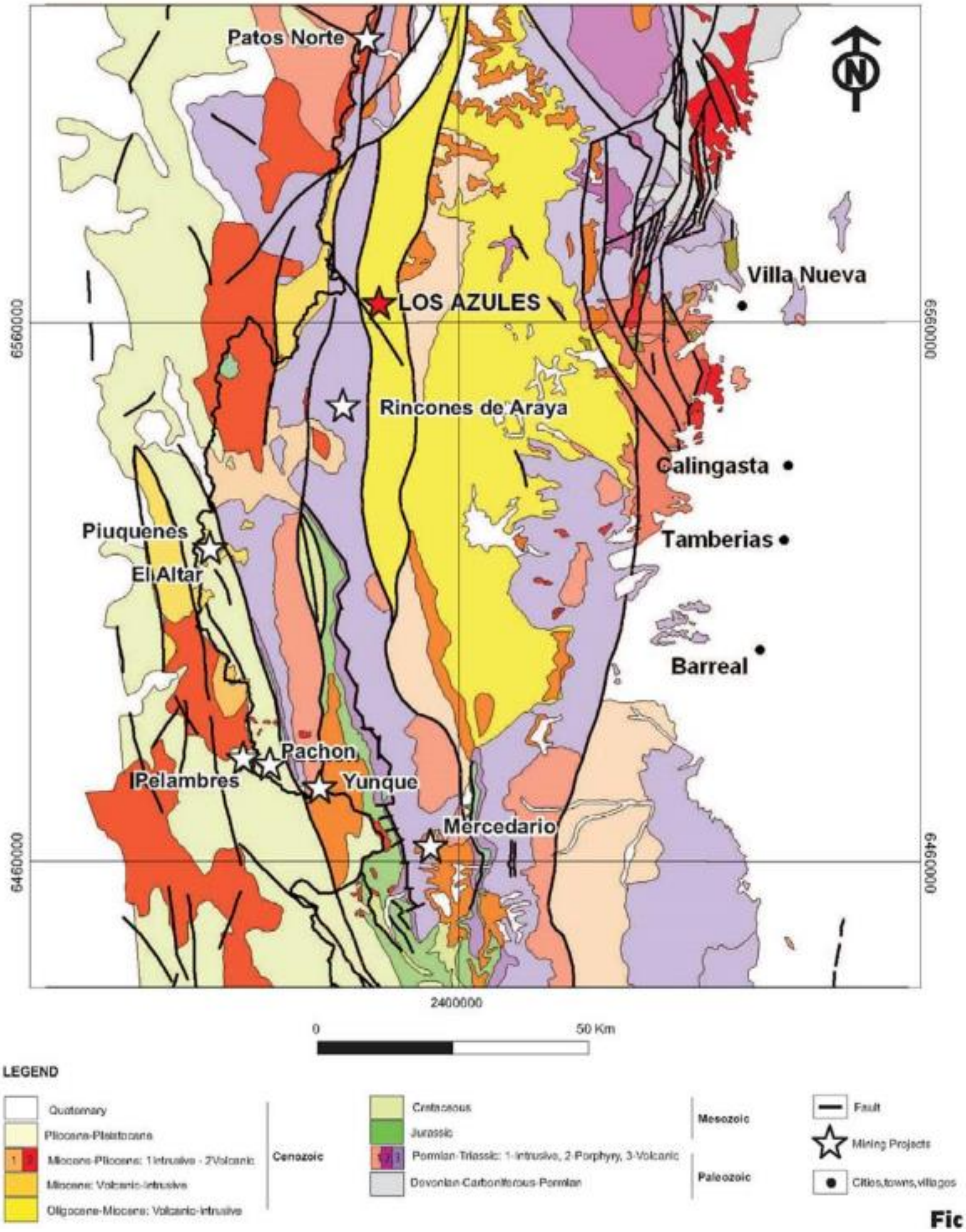


Figure 7.2: Regional geology of the Andean Cordillera of Argentina and Chile (Rojas 2010)

7.2 PROPERTY GEOLOGY

Los Azules has been geologically mapped on at least four separate occasions (Rojas, 2007; Zurcher, 2009; Almandoz, 2010; Pratt, 2010). The resulting geological maps and interpretations are in general agreement but differ in detail, and Jemielita (2010) reconciled most of the differences. The entire area comprising the Los Azules deposit is covered by thick scree or valley fill, so none of the rocks or structures are exposed in outcrop, although some near-surface exposures have been exposed in shallow trenching at the crest of the La Ballena ridge. Consequently, the interpretation of the structures and intrusive bodies is based almost entirely upon drill hole data.

In many respects the Los Azules deposit is a classic Andean-style porphyry copper deposit. In the bedrock below the surface cover a barren leached zone overlies a zone of secondary supergene enrichment of variable copper grades and thickness, and primary hypogene mineralization extends to at least 1,000 m below the present surface. The Los Azules hydrothermal alteration system is at least 5 km long and 4 km wide and is elongated in an NNW direction along a major structural corridor. The system disappears below volcanic cover to the north, so the ultimate extent is unknown. The altered zone surrounds the Los Azules deposit, which is approximately 4 km long by 2.5 km wide. The limits of the mineralization along strike and at depth have not been entirely constrained by drilling. In fact, many of the holes in the core resource area have been terminated in mineralization that exceeds the resource cut-off grade.

Hypogene minerals include chalcopyrite, lesser bornite, chalcocite-digenite, idaite and trace molybdenite, magnetite and lesser hematite, usually deposited on igneous mafic minerals. Chalcopyrite is the most important hypogene copper mineral in the upper levels of the deposit, and hypogene bornite appears at deeper levels together with chalcopyrite. Copper sulfides rarely exceed 2% to 3% of rock volume. Intervals of 0.1% to 0.35% copper are common in hypogene mineralization. Silver (approximately 1 gram/tonne), anomalous gold (up to approximately 150 parts per billion) and molybdenum (up to approximately 600 parts per million) are reported in some intersections.

Circulation of meteoric ground water leached primary sulfides (mainly pyrite and chalcopyrite) from the host rocks over the past several million years, and the leached copper was redeposited below the water table in a sub-horizontal zone, or blanket, of supergene enrichment as secondary chalcocite and covellite. The intensity of secondary enrichment diminishes with depth, except along major structures where it may extend to great depth.

Starting at the boundary between the barren leached zone and the supergene mineralization, secondary enrichment mineralization gradually transitions to predominately hypogene mineralization at depth.

Sillitoe (2014) examined about 9,000 m (approximately 25% at the time) of the available drill core and proposed a revised geologic interpretation for Los Azules, which is shown in Figure 7.3.

Sillitoe's interpretive model has features in common with the giant Río Blanco-Los Bronces and Los Pelambres deposits in Chile; both are part of the same Miocene-Pliocene porphyry copper belt as Los Azules. Vázquez (2015) subsequently relogged 44,000 m from 98 drill holes representing essentially all the drill core available at that time. Vázquez confirmed Sillitoe's interpretation, and he also refined the temporal sequence and spatial distribution of distinct phases of alteration and mineralization.

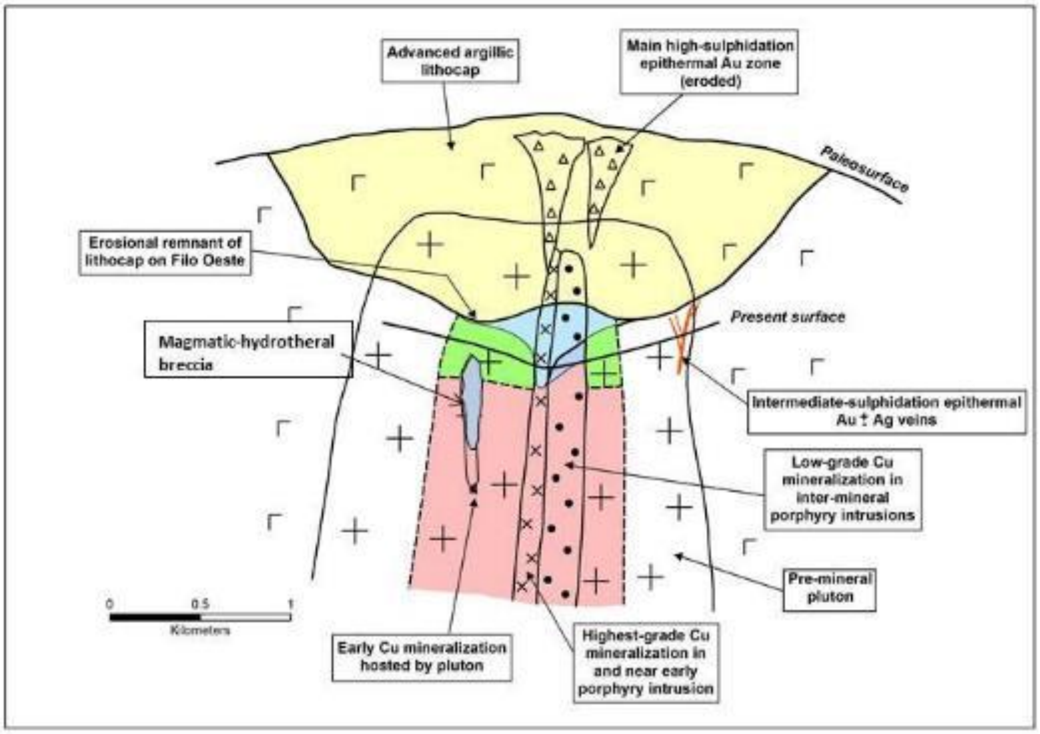


Figure 7.3: Model for Los Azules (pink: potassic alteration, green: chloritic alteration, blue: sericitic alteration, yellow: advanced argillic lithocap), (Sillitoe, 2014)

Sillitoe recognized the presence and importance of an early mineralized porphyry dike phase of igneous intrusion. Much of the hypogene mineralization as well as the supergene mineralization is associated with this phase; later dikes are not as well mineralized. Sillitoe referred to the later dikes as “inter-mineral” stage dikes.

Vásquez established the following chronological sequence of igneous and hydrothermal events at Los Azules, and these will be described in detail in the following sections.

1. Intrusion of dioritic stock or pre-mineral pluton (DIO / PMP).
2. Pervasive chlorite-magnetite alteration accompanied by chalcopyrite mineralization in the upper levels of the pluton grading into potassic alteration with chalcopyrite and bornite mineralization at depth.
3. Intrusion of the early mineralized porphyry dike phase (EMP).
4. Intrusion of the later “inter-mineral” phase porphyry dikes (IMP) and formation of magmatic-hydrothermal breccia bodies.
5. Late sericite alteration accompanied by pyrite and chalcopyrite.
6. Formation of erratic quartz veins containing base and precious metals.
7. Supergene enrichment.

7.2.1 Volcanic Country Rocks

Country rocks at Los Azules comprise volcanic lithologies of supposed Triassic age (Choyoi group or equivalents) including rhyolite intrusions and crudely bedded pyroclastics (flow-domes) ranging from fine grained tuffs to coarse breccias (Rojas, 2008; Pratt, 2010) as shown in Figure 7.4. The legend for Figure 7-4 is provided in Figure 7.5.

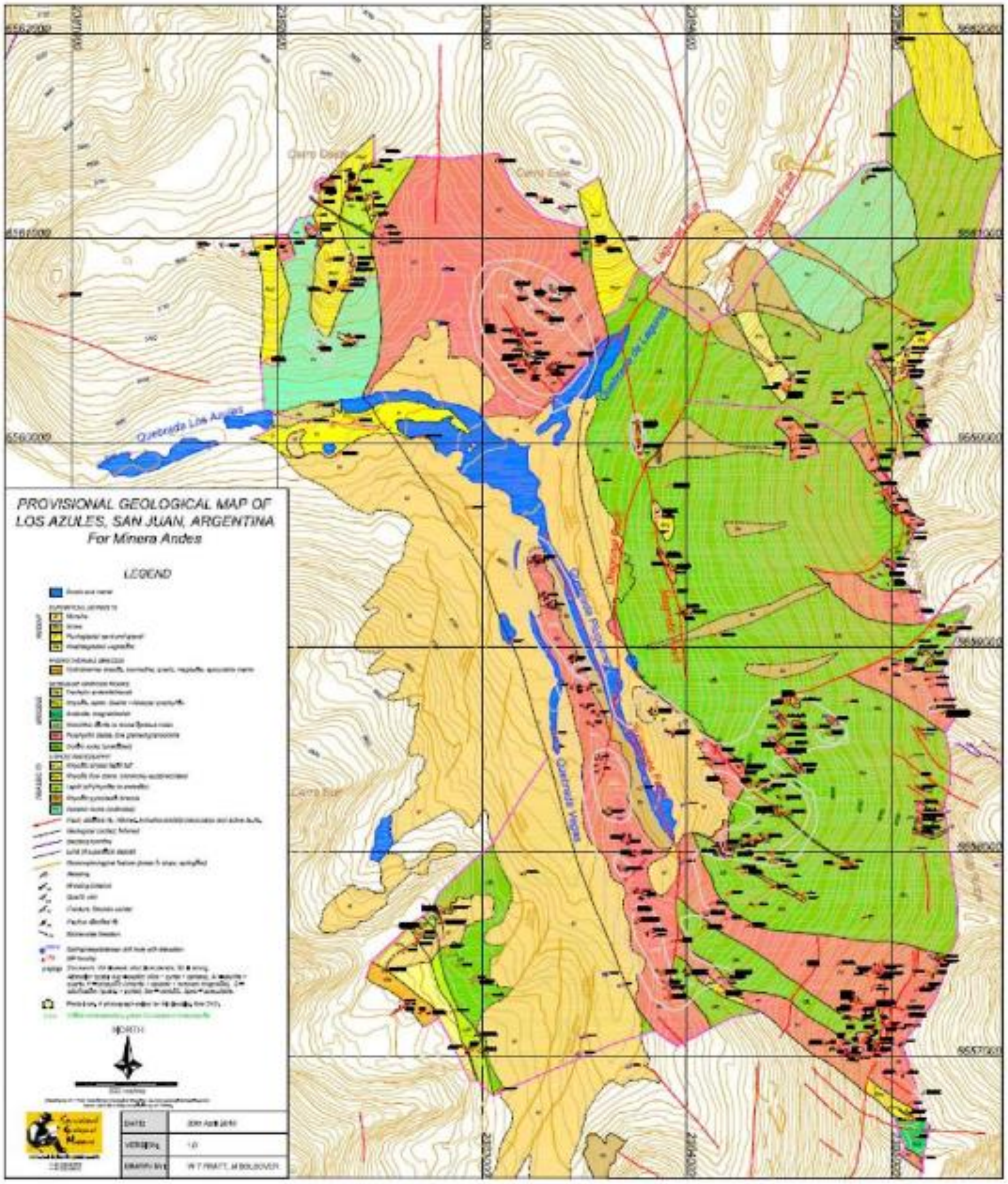


Figure 7.4: Geologic map of Los Azules (Pratt and Bolsover 2010)

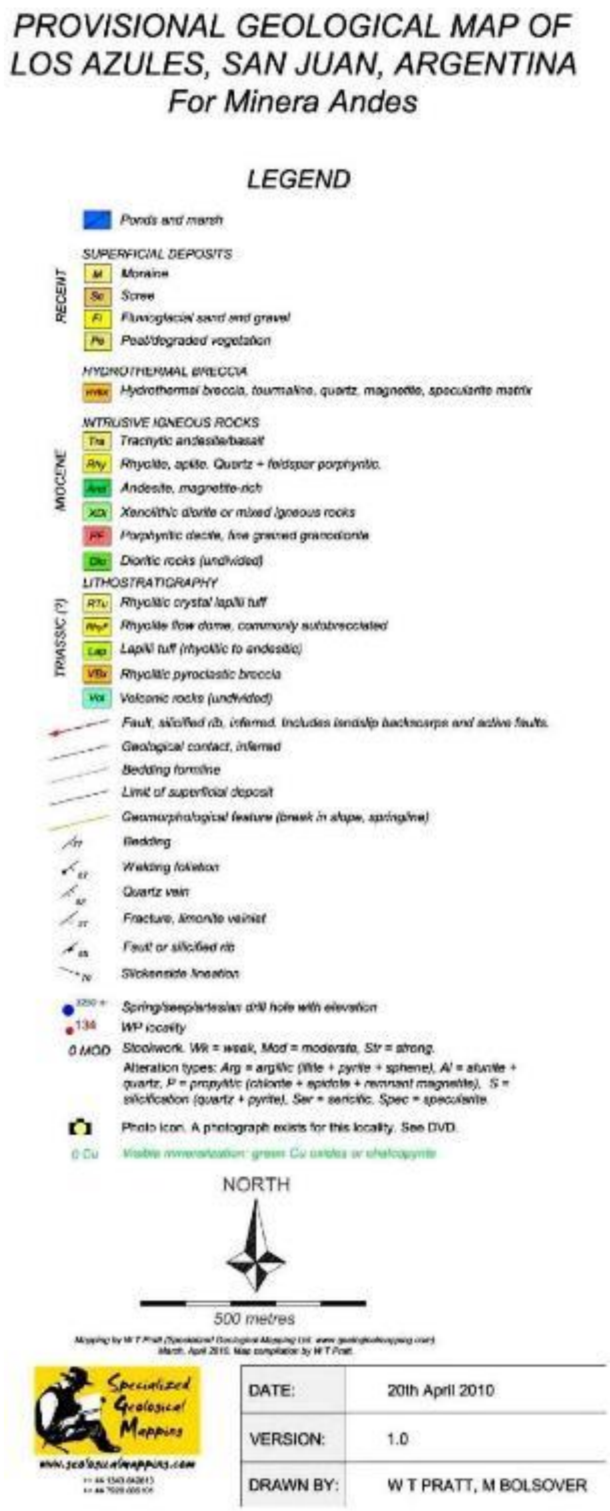


Figure 7.5: Legend for Figure 7.4 (Pratt and Bolsover 2010)

7.2.2 Pre-mineral Pluton

Triassic volcanic country rocks (VOLCS) at Los Azules are intruded by a pre-mineral multi-phase, calc-alkaline quartz diorite pluton dated at 10.6 Mya to 10.7 Mya (Zurcher, 2008b). The pluton is elongated in an NNW direction and is at least 7 km in north-south extent and at least 2.5 km wide (Figure 7.6). The pluton comprises numerous compositionally and texturally distinct phases, of which equigranularity, fine- to medium-grained diorite, monzodiorite and quartz diorite appear to be the most abundant within the mineralized zone. However, there are also phases such as felsic as quartz monzonite, and some of the quartz diorites are porphyritic.

Medium-grained granodiorite has been identified in drill core from the southern and southwestern sectors of the area and a finer-grained quartz diorite or tonalite phase is widespread in the east, northeast and northern sectors of the pluton. The pluton corresponds to the dioritic lithology on the geologic map shown in Figure 7.4.

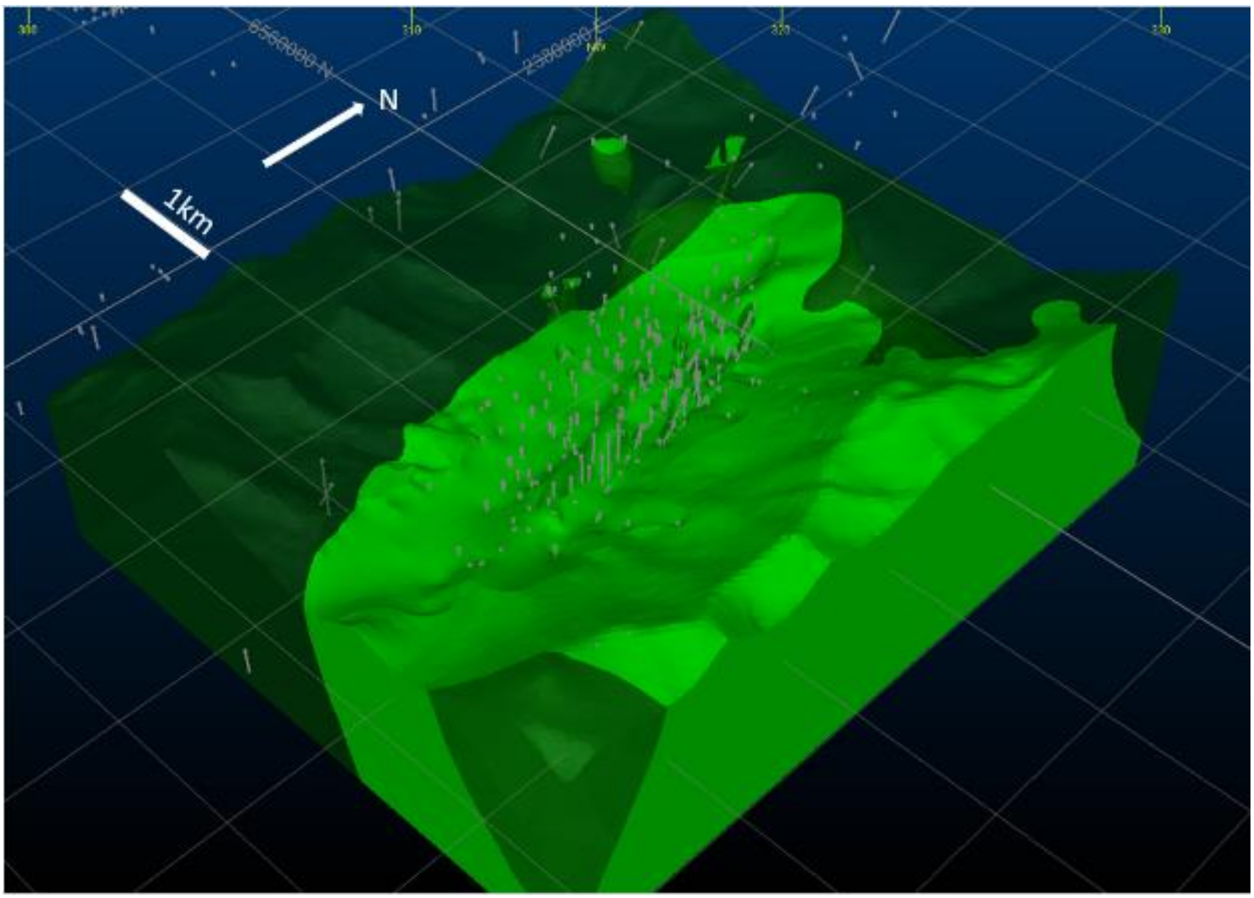


Figure 7.6: Triassic volcanic rocks (dark green) and the Pre-mineral diorite pluton (PMP) in lighter green. Most of the area is covered by gravel valley fill (not shown). Drillhole traces are shown in grey. (McEwen Copper, 2022).

7.2.3 Chlorite/Potassic Alteration

Shortly after emplacement, the pluton was affected by pervasive chlorite-magnetite alteration at higher levels in the system (up to depths of 300 m – 400 m below the present surface) contemporaneous with potassic alteration at deeper levels (Figure 7.7, Figure 7.8, Figure 7.9). No age dating is available for this event. The chloritic zone contains scattered specks of epidote, implying that it could be considered as propylitic. The upward transition of this early potassic to chlorite-magnetite alteration at Los Azules does not figure in conventional porphyry copper models, but it has been recognized by Sillitoe at Río Blanco-Los Bronces in Chile.

Vázquez estimated that approximately 50% of the hypogene copper at Los Azules was deposited during this hydrothermal event as veinlets and disseminations, but the grade corresponding to this hypogene mineralization is relatively low (e.g., 0.05-0.35% Cu). Copper mineralization in the pre-mineral pluton is homogeneous in the different lithologic facies of the pluton. However, pyrite and subordinate chalcopyrite characterize the chloritic zone, with chalcopyrite-only veinlets appearing beneath the chloritic-potassic alteration contact. At depths of 400 m – 500 m, bornite appears as an accompaniment to the chalcopyrite, and it increases at deeper levels to attain a maximum chalcopyrite/bornite ratio on the order of one to one.

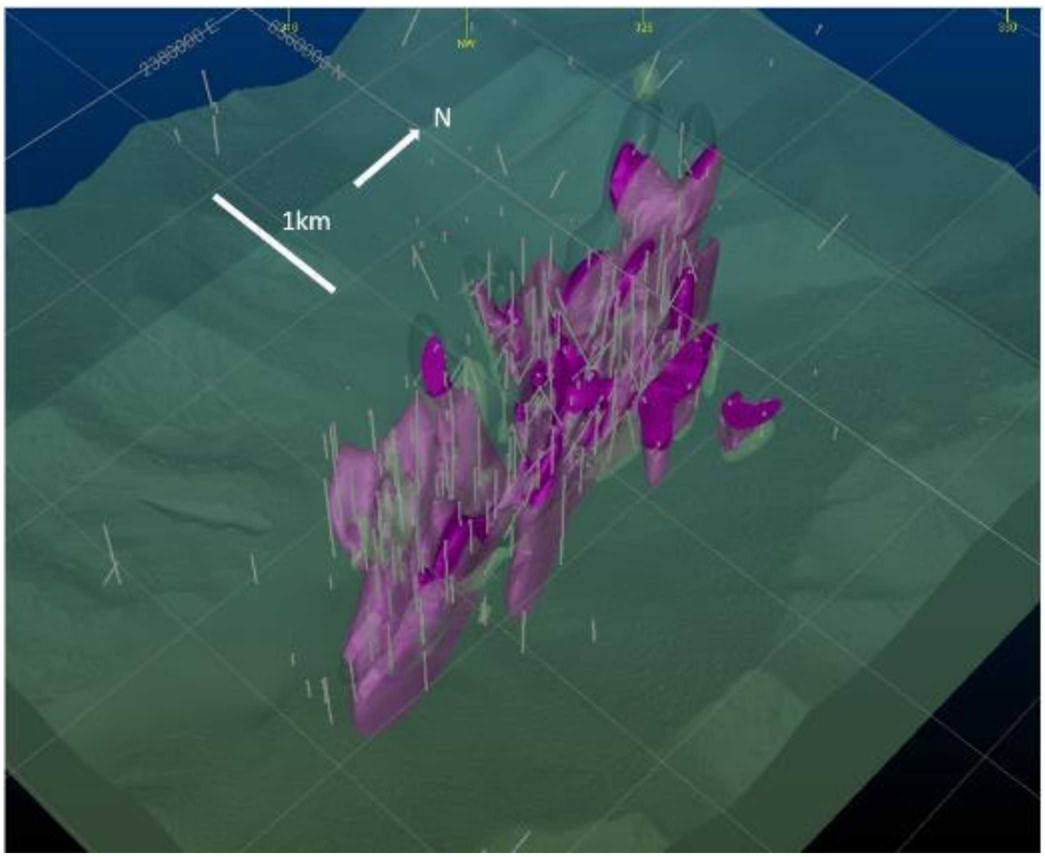


Figure 7.7: Pervasive early chlorite alteration in green with the deeper potassic alteration zone in pink. (McEwen Copper 2022).

Figure 7.8: Equigranular diorite (pre-mineral pluton) with chlorite alteration cut by a hairline type- D veinlet containing quartz and chalcopyrite with a sericite alteration halo (Hole AZ1284, 173 m).

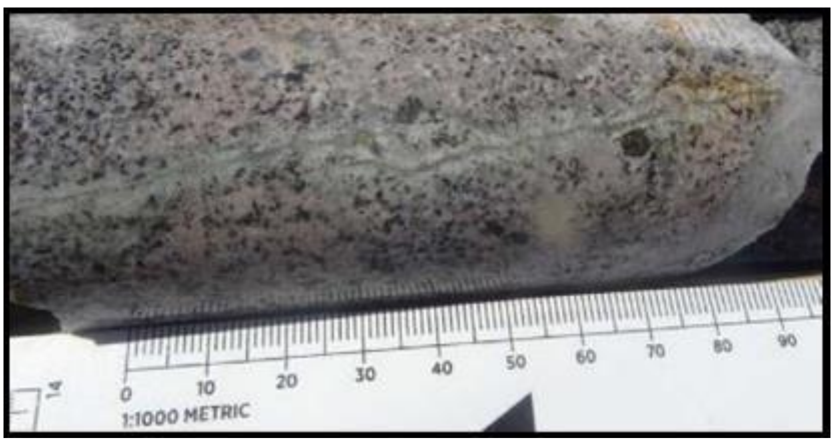


Figure 7.9: Diorite (pre-mineral pluton) with potassic alteration cut by a quartz-chalcopyrite type-A microveinlet (Vázquez, 2015).

Potassium feldspar alteration is characterized by pink orthoclase in veins/veinlets/stockworks and vein envelopes and pervasive replacements of plagioclase phenocrysts and/or matrix in diorite porphyry host rocks. Biotite alteration occurs as veins and pervasive- to selective- replacement of igneous biotite and hornblende in diorite porphyry host rocks in a zone peripheral to, and contemporaneous with, the main potassium feldspar alteration.

Advanced argillic litho-caps developed above, and contemporaneous with, the potassic alteration zones with their upper parts characterized by widespread quartz-alunite alteration and vuggy residual quartz bodies, show the potential to host high sulfidation epithermal gold ± silver± copper (enargite) mineralization (Figure 7.3). Only the root zones of these litho-caps, dominated by quartz-pyrophyllite and quartz-kaolinite assemblages, are now preserved at the highest elevations (Almandoz, 2010).

7.2.4 Early Mineralized Porphyry Dike

Zurcher (2008a) observed that the pre-mineral pluton is cut by a variety of porphyry dikes which he interpreted as trending generally NNW and inclined sharply to the east. Sillitoe made a distinction between "early mineralized porphyry dikes" and later "inter-mineral" dikes, and Vázquez established that there is one main NNW-trending rhyodacite porphyry dike, which is referred to as the "Early Mineralized Porphyry Dike", or "EMP" for short. The dike is about 1.8 km long, 50 m to 200 m wide, and it dips steeply to the east (Figure 7.10). This dike is responsible for the location and morphology of the prominent La Ballena ridge (Figure 7.11) because it is relatively resistant to erosion compared to other nearby rocks due to the silica content associated with its abundant type-A veins.

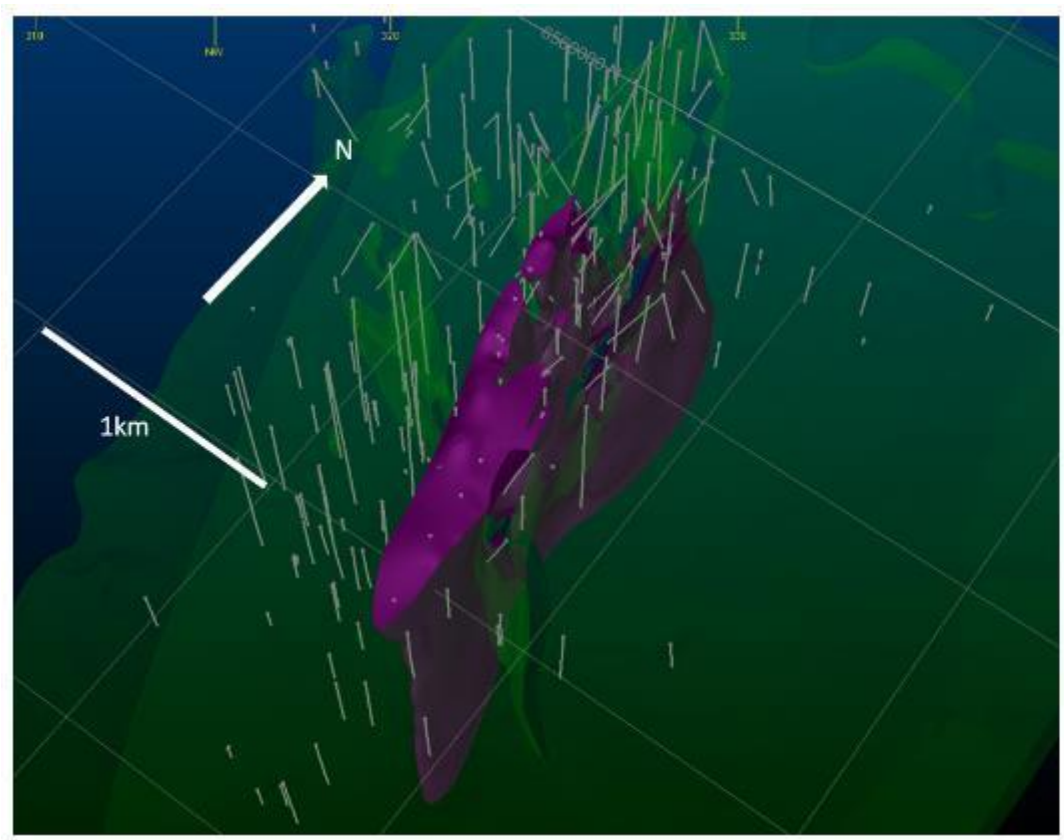


Figure 7.10: Early Mineralized Porphyry Dike (magenta), emplaced within the pre-mineral dioritic pluton in green. The entire dike is affected by potassic alteration. The dike is not yet constrained at depth by drilling (McEwen Copper, 2022).

Figure 7.11: The La Ballena ridge is characterized by the relatively resistant Early Mineralized Porphyry Dike.

The early mineralized porphyry dike(s) are rhyodacitic, and the texture is typically phaneritic-aphanitic with hornblende, biotite, and abundant, typically broken, feldspar phenocrysts and uncommon resorbed and cracked quartz eyes (1% - 5%). Phenocrysts are typically closely packed and grain-supported or set in an aplitic groundmass. This texture is locally referred to as "crowded" texture.

According to Vázquez, the emplacement of the dike took place during the dying stages of the magmatic phase. The dike was affected everywhere by potassic alteration immediately after its emplacement as evidenced by secondary biotite replacing magmatic amphiboles and biotite with associated type-A quartz veinlets with pyrite and chalcopyrite up to 2 cm in thickness. Despite locally intense alteration, the EMP can be readily distinguished from later "inter-mineral" phase dikes (described below) by the presence of relatively abundant "type-A" veinlets (Figure 7.12), which are commonly composed of pinkish, translucent, granular quartz and are typically 0.5 cm – 1 cm wide. In contrast, later-stage dikes contain few, if any, type-A veinlets, although "type-D" veinlets may be widespread in the later dikes. The classification of vein types follows the system of Gustafson and Hunt (1975). Vázquez established that the EMP corresponds to material that Zurcher (2008b) dated at 9.2 Mya.

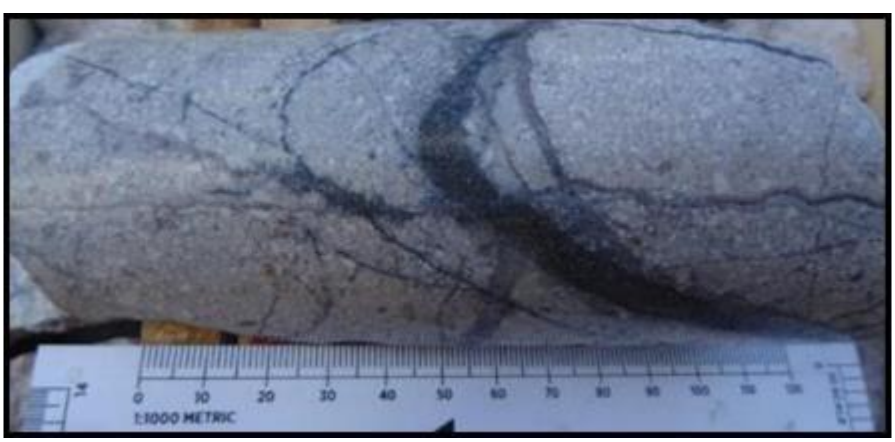
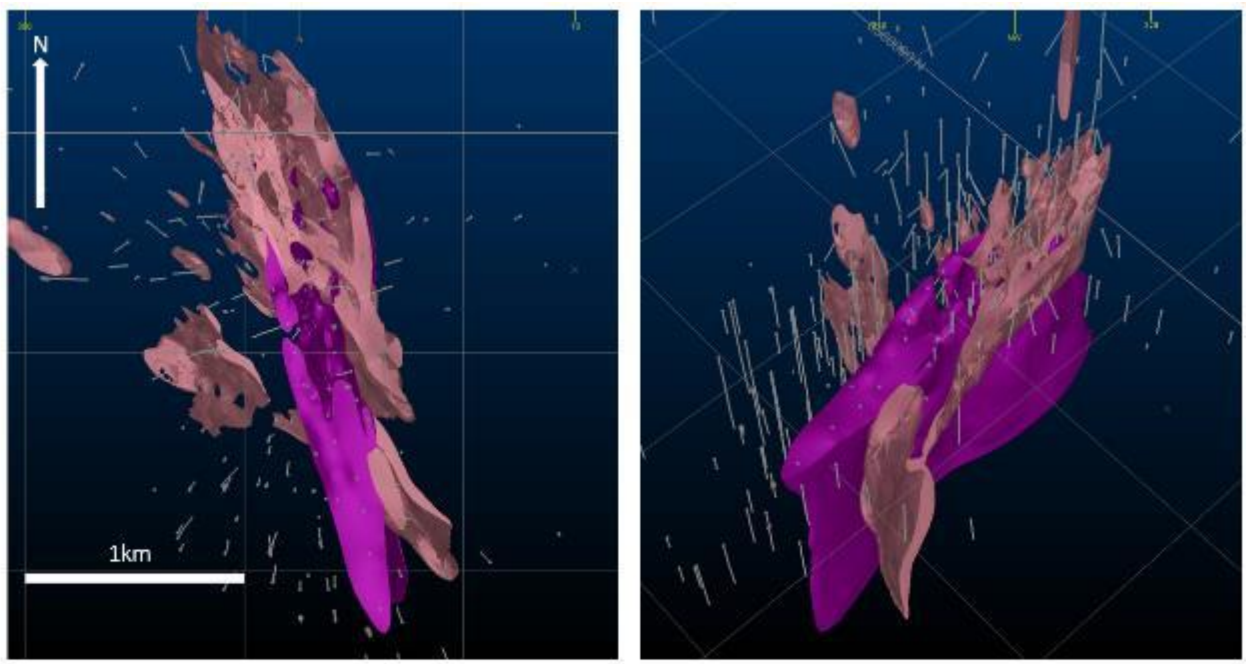


Figure 7.12: Early mineral porphyry with type-A quartz veinlets cut by type-D veinlets of pyrite replaced by supergene chalcocite. Pervasive sericite alteration. (Vásquez, 2015)

The EMP was responsible for the best grade of hypogene mineralization, and it typically contains 0.25% to 0.35% hypogene copper corresponding to about 15% of the hypogene copper in the system according to Vázquez.

7.2.5 Inter-mineral Porphyry Dikes

After the intrusion of the EMP, a series of NNW-trending dacitic porphyry dikes that dip steeply to the east were intruded into the diorite pluton. According to Vázquez, these dikes correspond to material that Zurcher dated at 8.2 Mya. The most prominent inter-mineral dike (“IMP”) is located along the east side of the EMP and is about 1.7 km long with widths ranging from 50 m to 150 m (Figure 7.13). There are also some minor inter-mineral phase intrusive bodies located west of the EMP. The inter-mineral dikes are composed of dacitic porphyry with crowded textures, consisting of plagioclase and quartz phenocrysts cut by only a few type-A veinlets. The inter-mineral dikes are intruded into the pre-mineral pluton, although to date no examples have been observed where it cuts the EMP.



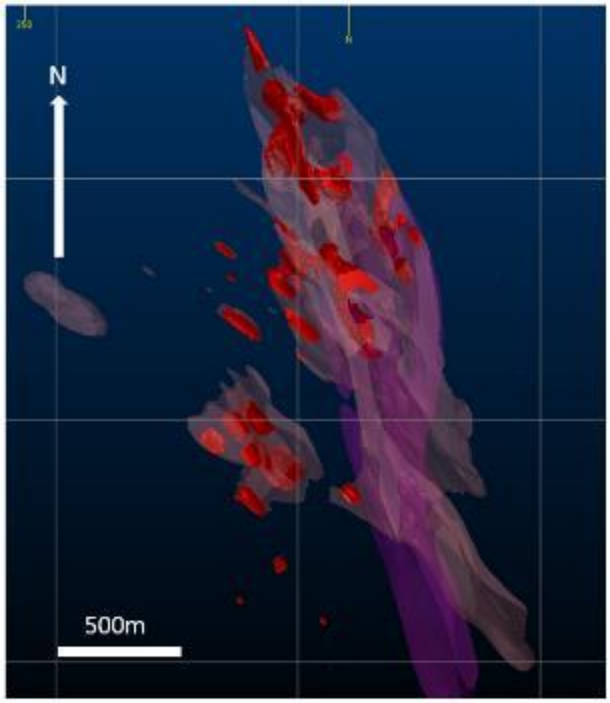


Figure 7.13: (Top): Two views of the Inter-mineral Dikes in pink and EMP in magenta. The most prominent IMP is located on the East side of the EMP. (Bottom): Plan view of hydrothermal breccias in red, coeval with the IMPs. Chloritic and potassic alteration zones removed for clarity. (McEwen Copper, 2022)

The hypogene grade of the inter-mineralized porphyry dikes is less than that of the EMP due to low content of type-A veinlets. The inter-mineral dikes typically contain 0.1% to 0.2% of hypogene copper and tend to be only weakly enriched. Vásquez estimated that the inter-mineral dikes correspond to about 5% of the hypogene copper in the system.

7.2.6 Hydrothermal-Magmatic Breccias

There are a series of small hydrothermal-magmatic type breccia bodies (MAG-HBX) generated by the release of over pressured magmatic fluid. Vázquez interpreted these breccias to be approximately coeval with the inter-mineral dikes because breccia clasts of porphyry were observed with type-A early veinlets of quartz and early hypogene mineralization. The breccia bodies are controlled by high angle pre-mineral faults as they generally trend NNW and dip steeply to the east, but they are discontinuous and relatively small compared to the total mineralized volume of Los Azules. There are two main sectors where these breccia bodies occur. The first is located west of the early mineralized porphyry dike between coordinates 6,558,600 N and 6,559,900 N as bodies of no more than 20 m to 30 m wide and up to 500 m long. The second domain occurs in the northern sector of the system between coordinates 6,559,800 N and 6,560,900 N (Figure 7.13) where the breccia bodies appear to have the form of pipes.

In general, breccias in both zones correspond to the crackle breccia type, characterized by a cement that consists of quartz, tourmaline, green sericite, specularite, chalcopyrite and bornite (sericite alteration association) that in some places is partly sericitized with disseminated chalcopyrite associated with this alteration. The sericite alteration transitions to potassic alteration at depth (Figure 7.14).

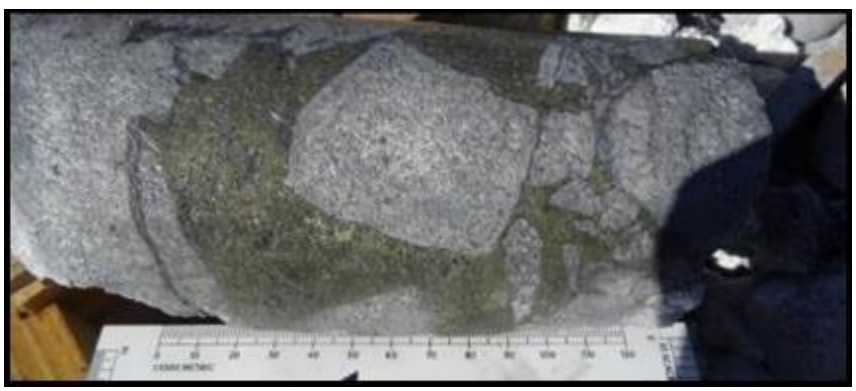


Figure 7.14: Magmatic-hydrothermal breccia with chalcopyrite and tourmaline in the breccia matrix. Clasts are partially sericitized (Hole AZ1297, 477 m) (Vázquez, 2015).

Some of the breccias were intersected in drilling over appreciable (>50 m) vertical intervals, but they are steeply inclined bodies, commonly less than 10 m wide. However, some may be wider because they appear to occur immediately above cupolas of porphyry intrusions in association with pegmatoidal K-feldspar-quartz pods, aplogranite veins and UST (unidirectional solidification texture) quartz layers (Figure 7.15).

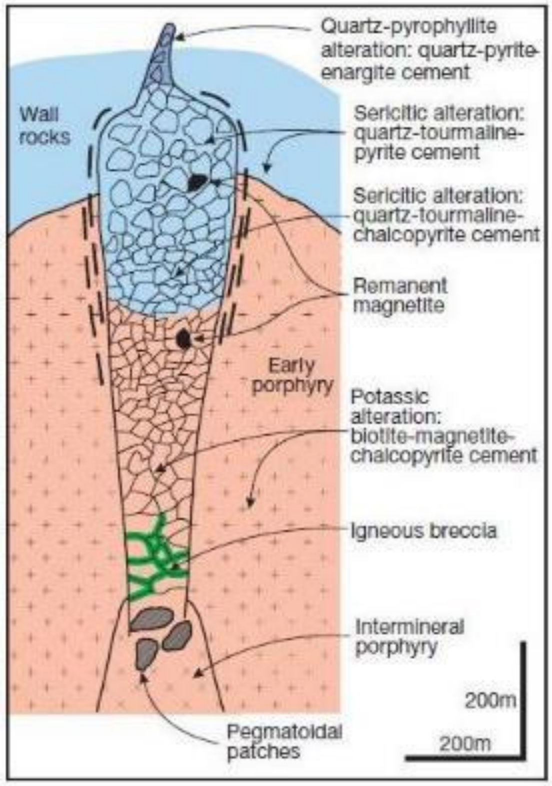


Figure 7.15: Schematic representation of a large magmatic-hydrothermal breccia body genetically linked to the apex of an inter-mineral porphyry intrusion (Sillitoe, 2010).

Many of these breccias contain particularly high-grade hypogene and supergene copper mineralization. Locally, the breccias contain elevated precious-metal values, including the highest recorded to date (8.4 g Au/t, 159 g Ag/t in Hole AZ12106). Vázquez estimated that the hydrothermal-magmatic breccias contain about 5% of the total hypogene copper.

7.2.7 Sericite Alteration

From coordinate 6,559,300 N toward the south, sericite alteration was superimposed over the potassic alteration after the emplacement of the inter-mineral dikes (Figure 7.12). This event supplied pyrite and chalcopyrite to the system and was responsible for the greatest contribution of hypogene copper in the system. This alteration is centered on the EMP and inter-mineral dikes and surrounding rocks, and Vázquez believed that this alteration was controlled by high-angle, steep-dipping, NNW-trending structures (Figure 7.16). This alteration transitions to chlorite-sericite at depth, and Vázquez interpreted that this alteration took place shortly after potassic alteration, which occurred during the cooling of the system.

There are no age dates to establish the age of this event. Hypogene copper grades were as much as 0.35% in and near the EMP, and this event accounted for about 25% of the hypogene copper according to Vázquez.

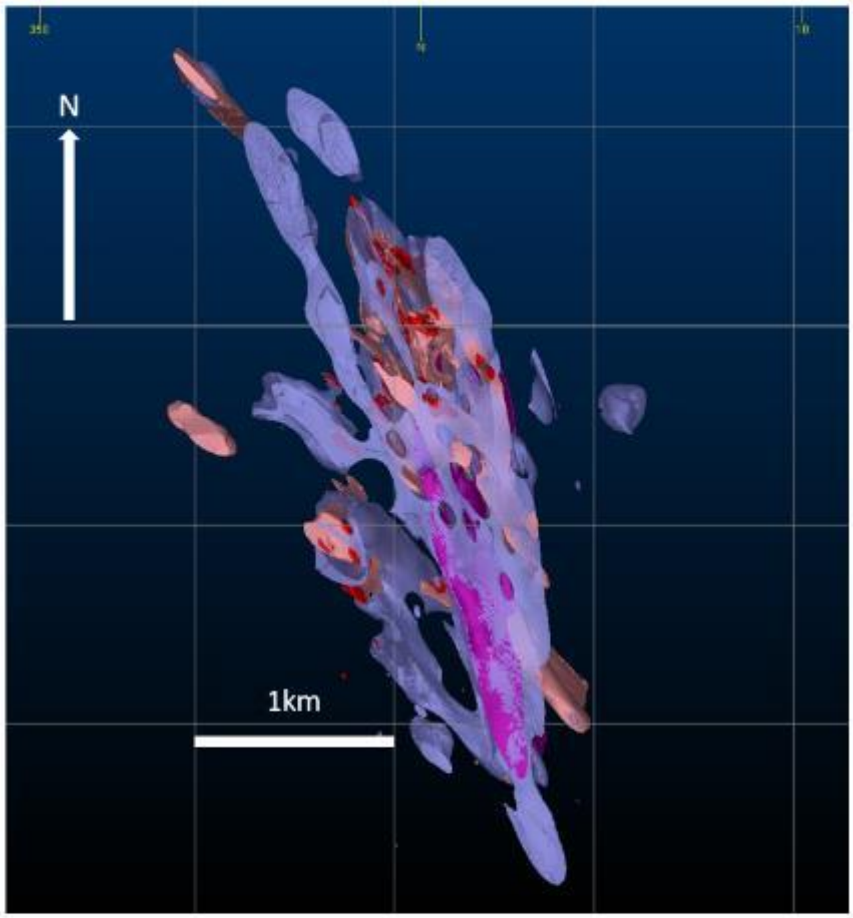


Figure 7.16: Sericite alteration zone shown in lilac shade. The sericite alteration affected the EMP, inter-mineral dikes and surrounding quartz diorite rock (not shown). (McEwen Copper, 2022)

7.2.8 Late Quartz-Sulfide Veins

Several late quartz-sulfide veins with base and precious metals were emplaced after the sericite alteration event to the west of the EMP, where they are common, and Vázquez believed that these veins are associated with NNW trending, steeply dipping structures. The veins are typically 0.1 m to 1.0 m in thickness (up to a maximum of 3 m) and typically consist of quartz and pyrite with variable quantities of chalcopyrite, sphalerite, and galena. Precious metal values are commonly in the range of 10 to 50 g Ag/t, and Vázquez suggested that most of the precious metal in the deposit may be related to these veins. However, they do not represent exploration targets because of their discontinuous nature.

7.2.9 Supergene Enrichment

Supergene mineralization at Los Azules comprises a sub-horizontal chalcocite-covellite supergene blanket ("enriched" zone) that grades downwards through a partially enriched zone of incomplete replacement (mixed hypogene-supergene sulfides) into underlying hypogene sulfide mineralization. A sterile oxidized leached cap overlays the supergene blanket. Sillitoe considered that the enriched zone is immature because of the relative youthfulness of the supergene processes, which was probably active only since about 4 Mya –5 Mya.

The leached cap ranges from 0 m to 180 m thick and consists of oxidized and argillic-altered rock. Limonitic boxworks and disseminated spots of jarosite, goethite, and hematite are common. Hematite is more abundant in the southern structural block; jarosite is best developed over the central block, while goethite appears more widespread in the northern block (Zurcher, 2008a). Primary magnetite is altered to hematite, and ferrimolybdite also occurs (after molybdenite), but copper minerals and sulfides are mostly absent (Rojas, 2008). Copper oxides are reported from the margins of the leached zone and include brochantite, minor cuprite, copper pitch and copper wad. Copper grades in the leached cap range between 0.01% and 0.10%.

Beneath the leached cap, a thin mixed sulfide-oxide zone gives way to a supergene sulfide zone where hypogene sulfides are replaced by chalcocite and minor covellite. The supergene copper blanket is best developed in the central and central-northern structural sectors and is characterized by a more jarositic oxide cap in the pyritic phyllic-altered zone located directly above the potassic alteration zone. Supergene (earthy) chalcocite and minor covellite partially (or rarely) completely replace hypogene sulfides, but pyrite usually survives. Traces of native copper and gypsum after anhydrite occur in the underlying potassic alteration zone.

The thickness of the supergene chalcocite blanket typically varies between 60 m and 250 m but can penetrate to more than 400 m down structures. The intensity of supergene mineralization gradually decreases with depth from the top of the zone and there is typically no distinct lower limit or boundary to the zone of enriched mineralization (Sim and Davis, 2015). Also see Section 14.5.4 for a geostatistical review and explanation.

Copper values in the supergene enriched zone vary between 0.4% Cu to greater than 1.0% in the north-central part of the system and decrease to the south and the peripheries to 0.2% to 0.4% Cu. Supergene mineralization is the most important mineralization of economic interest at Los Azules.

Cyanide soluble copper data is used to interpret the distribution of supergene enrichment mineralization. Mineralization with a ratio of cyanide soluble copper content to total copper content >50% is "enriched". See Section 14.3.6 for further details on the Copper Mineral Zonation Model. The limits may be modified slightly in places to match mineralization observed in the drill core. Figure 7.17 shows the enriched mineralization and the early mineralized dike. Lithology, alteration zones and structures have been removed from the illustration for clarity, except the early mineralized porphyry dike. Supergene mineralization penetrates all lithologies and alteration types, and supergene mineralization appears to extend to greater depths along fault structures.

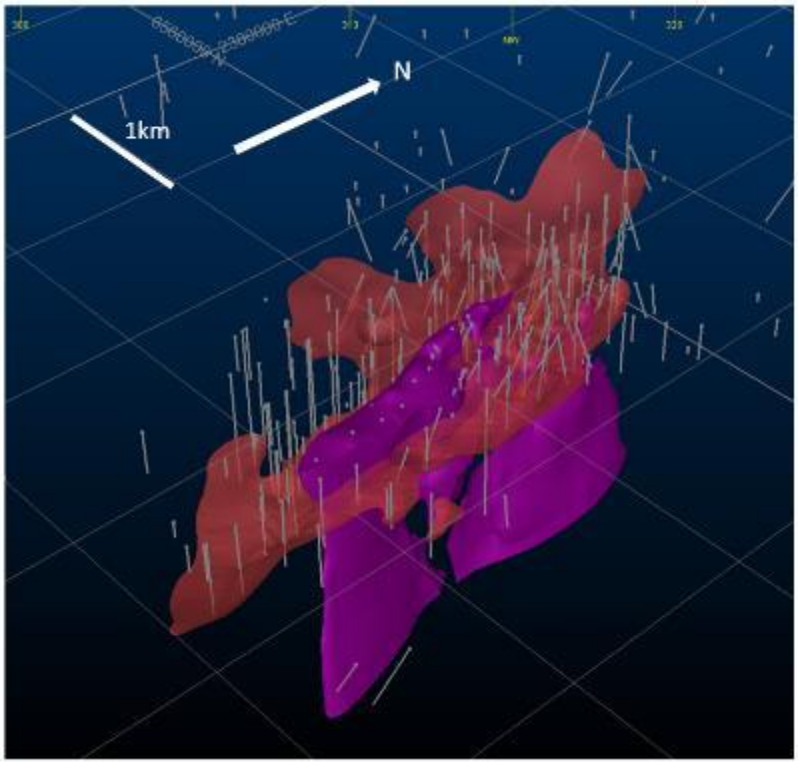


Figure 7.17: Early Mineralized Porphyry (magenta) with supergene enrichment zone (red) defined as the Soluble Cu ratio >50%. (McEwen Copper, 2022)

7.2.10 Sulfate Front

The Los Azules deposit is marked by a prominent and abrupt supergene sulfate front, ranging in depth from about 300 m to 700 m below surface, above which all anhydrite and gypsum have been removed by the descent of meteoric ground water. The front is marked by the downward appearance of gypsum, which is the supergene hydration product of anhydrite, lining fractures and filling cavities, especially in the breccias. Anhydrite is commonly seen as part of the matrix of hydrothermal magmatic breccias mainly where these breccias are affected by sericitic alteration. Anhydrite remnants become progressively more abundant downwards in the sulfate zone, irrespective of the alteration type in which the mineral is a hypogene constituent. Rock strength beneath the sulfate front increases abruptly, as shown by increased RQD indices.

The widespread occurrence of broken drill core from Los Azules reflects the strongly fractured nature of the rock as shown below in Figure 7.18. This could be a result of fault zone fracturing except that fractures are randomly oriented. Many planar fractures are coated with gypsum, after anhydrite. Jemielita (2010) suggested that there may have been a pervasive anhydrite stockwork throughout the porphyry system that has subsequently been hydrated, dissolved, and removed.

Figure 7.18: Typical Drill Core from Los Azules indicating the strongly fractured nature of the rock (Jemielita, 2010)

7.2.11 Structural Geology

Triassic volcanic country rocks at Los Azules are deformed into an anticline or monocline with the steep limb in west and the flat limb in the east (Pratt, 2010). The anticlinal axis strikes north and probably coincides with the NNW-striking structural corridor that controlled the locations of volcanic-intrusive centers in the region during the upper Miocene (Rojas, 2008). Near Los Azules this structural corridor appears to control the locations of porphyry dikes, hydrothermal alteration, and mineralization zones along a seven-kilometer strike length including the Los Azules porphyry system (Rojas, 2008).

The porphyritic dikes at Los Azules were emplaced along numerous, strong, pre- and syn- mineralization, north-northwest and northwest striking faults with important strike-slip components (Zurcher, 2008a). Based on the few surface exposures, Zürcher proposed a steep easterly dip for most of the north-northwest striking faults. Sillitoe and Vásquez both noted that evidence from diamond drill core indicates that these structures were active as faults during as well as after the deposition of the mineralization because post-mineral movement is evidenced by slickensides in areas with supergene mineralization. Numerous intervals of black fault gouge surrounded by broken zones are observed in the drill core. All the gouge zones are steep, ranging from 75° to vertical, and characterized by sub-horizontal slickensides; however, the amount of transcurrent displacement and strike of the faults is indeterminate.

In the northern sector, following the course of the Quebrada Las Lagunas, a major fault or fault zone occurs with a northeast orientation. However, there is no significant net displacement associated with this fault because mineralization, presence of hydrothermal magmatic breccias bodies and alteration are similar both north and south of the same faults.

Pratt (2010) interpreted a kinematic structural model of the Los Azules porphyry copper deposit. The Piuquenes Fault is part of the north-northwest striking "Vegas" fault system described by Rojas (2008). The northwest-striking faults were named Azules by Rojas (2008). Porphyry-related quartz veins (blue) and deeper level and older (than epithermal) alunite and vuggy quartz silicified ribs (red) are shown in Figure 7.19.

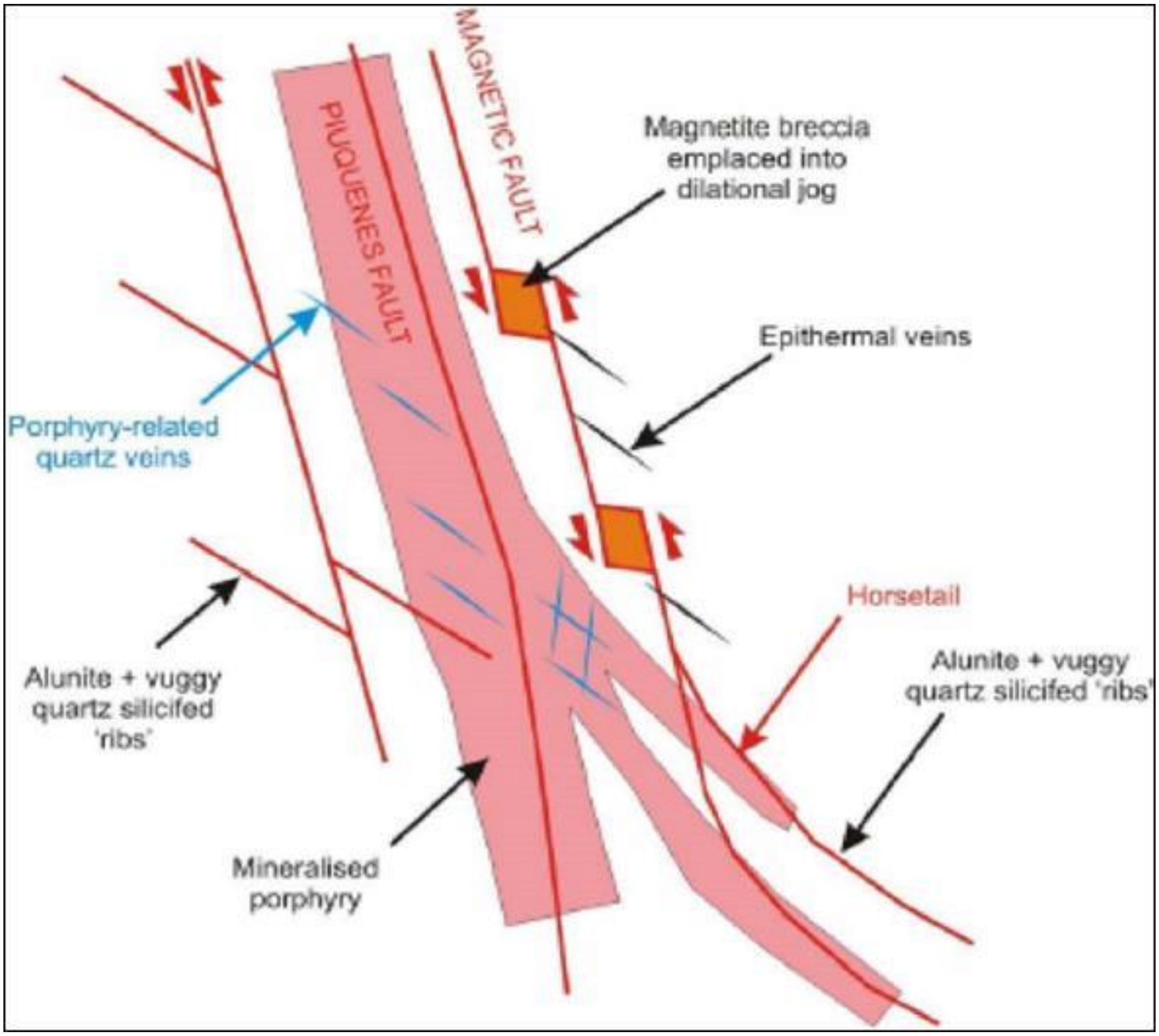


Figure 7.19: Kinematic structural interpretation of Los Azules porphyry copper deposit (Pratt 2010)

7.3 OTHER MINERALIZATION

Battle Mountain Gold explored Los Azules during 1998-1999 for gold and drilled three holes in altered pyroclastic volcanic rocks in a strongly pyrite-mineralized zone at La Hoya in the extreme northwest of the area, without significant success. The company may have been attracted by hydrothermal breccias with associated kaolinite-illite-dickite-quartz-alunite alteration that are reported in volcanic lithologies intruded by small intrusions and dikes of feldspar porphyry in the Cerros Centrales (Cerro Oeste) area.

Indications of potential gold-silver mineralization around the Los Azules porphyry copper system include late-stage, intermediate-sulfidation epithermal quartz veins described by Pratt (2010). These veins are mainly quartz (with minor sphalerite and galena). A variety of precious metals deposits commonly occur peripheral to many porphyry copper systems, but the district around Los Azules has not been systematically explored for such mineralization.

The existence of a thick leached cap and supergene chalcocite blanket at Los Azules indicates that oxidation, dissolution, vertical transportation and redeposition of copper occurred in the system. Copper may also have been transported laterally away from the deposit and redeposited to form so-called “exotic” copper mineralization (Sillitoe, 2010). No exploration for this style of mineralization has yet been undertaken in the vicinity of Los Azules.

8.0 DEPOSIT TYPES

8.1 INTRODUCTION

Los Azules is located within the Central Chile segment (400 km-long) of the Miocene-early Pliocene porphyry copper belt (6,000 km-long) of the north and Central Andes as shown in Figure 8.1. The figure also shows locations of the major porphyry copper and related epithermal deposits, limits of the porphyry copper belt and permissive northwest-trending structural corridors that influence the location of mineralization along the porphyry belt.

Porphyry copper deposits in this sub-belt are 12 to 4 Mya in age and include the world-class Los Pelambres (Cu-Mo), Rio Blanco-Los Bronces (Cu-Mo) and El Teniente (Cu-Mo) porphyry deposits, the Maracunga belt porphyries (Cu-Au) in Chile and El Pachón (Cu) and Bajo de la Alumbrera (Cu-Au) in Argentina, as well as numerous other porphyry and related deposits (Sillitoe and Perello, 2005).

Mineralization at Los Azules is Andean-Cordilleran, late Miocene, (quartz-) diorite-hosted, oxidized porphyry copper style with a well-developed leached cap and supergene chalcocite- covellite blanket. Los Azules displays numerous features in common with other porphyry deposits as described below.

Panteleyev (1995) describes the common features of porphyry deposits as large zones of hydrothermally altered rock containing quartz veins and stockworks, sulfide-bearing veinlets, fractures, and lesser disseminations in areas up to 10 km^2 in size. These are commonly associated with hydrothermal and/or intrusion breccias and/or dike swarms.

Deposit boundaries are determined by economic factors that define mineralized zones located within larger areas of low-grade, often concentrically zoned mineralization. Important geological controls on porphyry mineralization include igneous contacts, cupolas and the uppermost, bifurcating, parts of stocks and dike swarms. Intrusive and hydrothermal breccias and zones of intensely developed fracturing, respectively due to intersecting or parallel multiple mineralized fracture sets, commonly coincide with the greatest metal concentrations.

Surface oxidation commonly modifies porphyry deposits in weathered environments. Low pH meteoric waters leach copper from the oxide zone which is then transported and redeposited as secondary chalcocite and covellite usually immediately below the water table to form sub- horizontal, tabular zones of supergene copper enrichment. This process forms a copper-poor leached cap above a relatively thin but often high-grade zone of supergene copper enrichment that itself caps a thicker zone of often moderate grade hypogene copper mineralization at depth.

Alternatively, or additionally, porphyry systems can exhibit hypogene enrichment related to the introduction of late hydrothermal, copper-enriched fluids along structurally prepared pathways, or the leaching and redeposition of hypogene copper, or a combination of the two. Hypogene copper mineralogy in this instance comprises covellite and chalcocite, often with elevated hypogene copper grades.

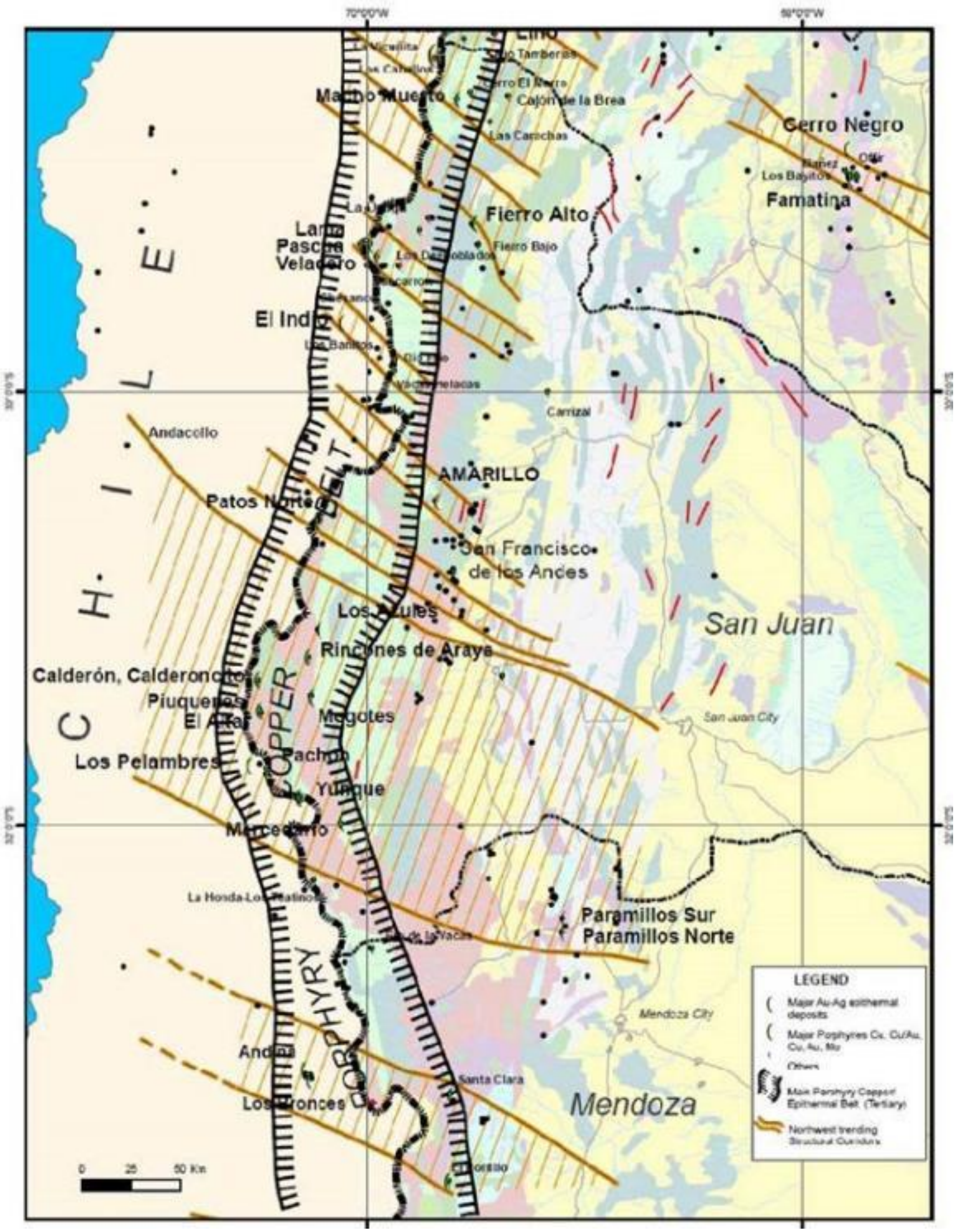


Figure 8.1: Part of the Central Chile Segment of the Miocene-early Pliocene Porphyry Copper Belt (Rojas 2008)

Other deposit styles often spatially, temporally, and genetically associated with porphyry deposits include:

- Exotic copper deposits, formed by the lateral migration of copper-bearing fluids away from the main body of porphyry mineralization.
- Mineralized breccia pipes, skarns, sedimentary replacements (mantos) and precious metals-bearing mesothermal-epithermal vein deposits located peripheral to and progressively distant (laterally and vertically) from the porphyry copper center as shown in Figure 8.2.

The figure shows the spatial relationships between a porphyry copper system and its surrounding environment including host rocks and peripheral styles of mineralization such as skarns, carbonate replacement (chimney-manto), sediment-hosted disseminated sulfides, mesothermal polymetallic veins and higher-level high/intermediate/low sulfidation epithermal gold-silver veins and disseminated deposits.

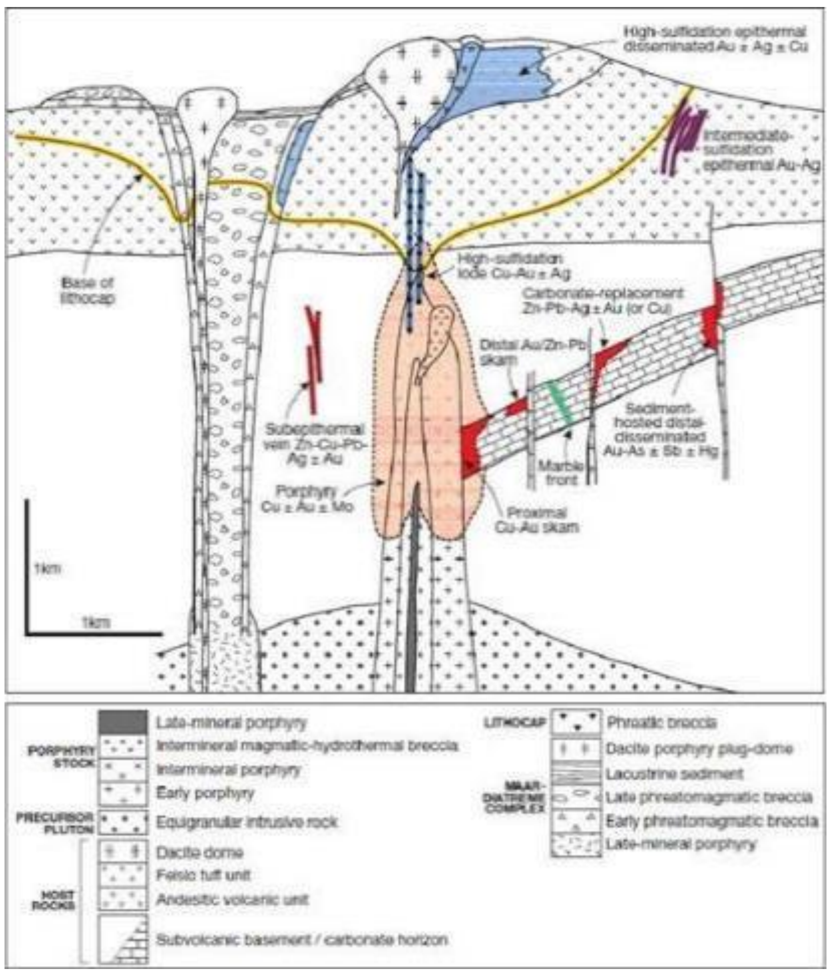


Figure 8.2: Diagram Showing Spatial Relationships between a Porphyry Copper System and the Surrounding Environment (Sillitoe 2010)

9.0 EXPLORATION

9.1 EXPLORATION HISTORY

Exploration at Los Azules commenced in the mid-1990s and has included various studies of geology, geophysics, and geochemistry, as well as drilling with both reverse circulation and diamond core drills, sampling and analysis of surface and drill core samples, and road construction. Exploration was conducted successively, and sometimes in cooperation, by Battle Mountain Gold, MIM-Xstrata, and Minera Andes/McEwen Mining and McEwen Copper principally by the latter company.

9.2 GEOLOGICAL MAPPING AND STUDIES

The most comprehensive and up-to-date geological map of Los Azules was produced by Pratt and Bolsover in 2010, as described in Section 7.2. An earlier detailed geological map, with cross sections, was compiled by Rojas (2007); Almandoz (2010b) produced a geological map at 1:5000 scale, and Zürcher (2008a) made a detailed map of the central portion of the north-northwest-trending La Ballena ridge that focused on hydrothermal alteration and mineralization. The latter map shows no lithological boundaries, reflecting the difficulty of separating igneous lithologies in the mineralized zone, a problem also reported by Pratt (2010).

Surface and drill core samples have been analyzed since 2004 as part of a mineralogical study using a portable infrared spectrometer (PIMA; Lasry, 2005). Petrographic studies were made in Argentina after the 2006 exploration campaign (Sumay and Meissi, 2006).

Petrographic studies of polished sections collected by Zurcher from drill cores, and surface samples were initially studied by DePangher (2008) in Oregon, and then by GEOMAQ in Santiago de Chile (Rojas, 2010). Zurcher (2008b) reported a series of U-Pb age dates for the igneous intrusions.

In 2014 Sillitoe examined about 9,000 m (approximately 25% at the time) of the diamond drill core and proposed a revised geologic interpretation for Los Azules which is described in Section 7.2. Sillitoe recognized the presence and importance of an early mineralized porphyry dike phase of igneous intrusion. Much of the hypogene mineralization as well as the supergene mineralization is associated with this phase; later dikes are not as well mineralized.

In 2015 Vázquez relogged 44,000 m from 98 drill holes representing essentially all the drill core at the time. Vázquez confirmed Sillitoe's interpretation, and he also refined the temporal sequence and spatial distribution of distinct alteration phases and mineralization zones as described in Section 7.2.

9.3 GEOPHYSICS

Various geophysical studies were conducted at Los Azules by Battle Mountain Gold and by MIM-Xstrata respectively in 1998-1999 and 2004 and by Minera Andes (Quantec) in early 2010 and McEwen Mining (Quantec) in 2012. Work done and results for these surveys are described in the following section.

9.3.1 Battle Mountain Gold (1998-99)

GEODATOS, a Chilean geophysical company, conducted an airborne geophysical survey in early 1998. The survey covered a 20 km by 10 km area elongated east-west including the Los Azules and Paso de la Coipa areas. Lines were flown north-south at 200 m intervals and control lines were flown east-west at 1,000 m intervals. Instrument altitude was maintained at 20 m during flights.

Results suggested the existence of a structural corridor striking northwest and structures striking east-northeast associated with strong to moderate magnetic low signatures in the Los Azules mineralized body. A total field magnetic plot identified a magnetic high anomaly surrounding a central magnetic low that extended 6 km north-northwest and 3 km northeast as shown in Figure 9.1. Battle Mountain Gold interpreted the magnetic low as altered rocks associated with the mineralized body.

Four lines of induced polarization (IP) were oriented east-west averaging two kilometers long and spaced at 600 m to 900 m apart. The lines were positioned to cross the locations of mineralized drill holes LA04-98, LA-06-98, and LA-08-98. One of the lines extended north to lithocap outcrops with anomalous copper (advanced argillic alteration possibly associated with gold mineralization and underlying porphyry copper mineralization). IP results indicated high chargeability and low resistivity corresponding with the location of the Los Azules porphyry copper deposit.

Two ground magnetic surveys totaling 103 km were conducted in the Los Azules mineralized porphyry and the nearby Sector Mantos, which is 1 km west of Cerro Oeste.

Lines were oriented east-west at 100 m spacing and 10 m stations. Results confirmed the existence of north-northwest- and north-northeast-striking structures as indicated by aeromagnetics. Results also confirmed the presence of a magnetic low anomaly in the vicinity of drill holes LA-98-04, LA-98-06 and LA-98-08 and suggested the presence of a magnetic low along the alteration system of La Ballena ridge as shown on Figure 9.1.

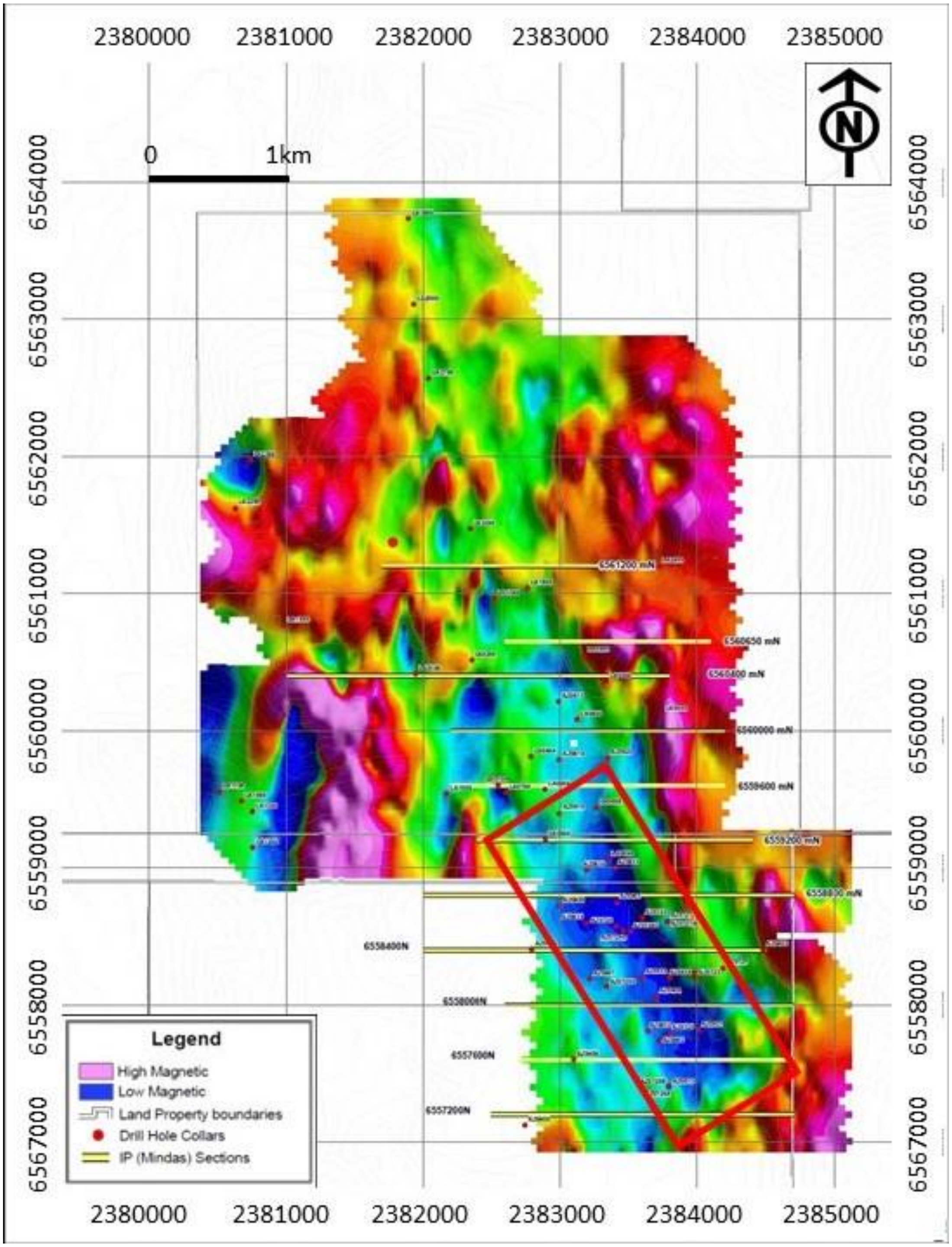


Figure 9.1: Magnetic Map of Los Azules (Reduced to Pole) and IP lines. (Rojas, 2008 after Xstrata, 2003). Note: Red box indicates the mag low across the Ballena Ridge.

9.3.2 MIM Xstrata (2003-2004)

During 2003-2004 MIM-Xstrata carried out a magnetic survey of approximately 70-line km at Los Azules. Lines were oriented east-west across the area controlled by the company at that time. In addition, MIM-Xstrata ran six lines of MIMDas (MIM-Xstrata proprietary IP system) east-west totaling 11.8 km. At the request of Minera Andes, MIM-Xstrata extended their geophysical lines south into Minera Andes ground completing five additional lines for a total 11.3 km in 2004. Total surveying by MIMDas was 23.1 km.

Magnetometry indicated a magnetic low beneath the Los Azules porphyry copper system and suggested that it extended north-northwest towards the La Hoya zone (Cerros Oeste and Este). The total field plot identified a magnetic high anomaly surrounding the magnetic low. The magnetic low extends 7 km to 8 km north-northwest and up to 2 km east-northeast confirming the interpretations made by Battle Mountain Gold.

MIMDas IP surveying (2003-2004) indicated high resistivity in the north-northwest zones at Los Azules with much lower resistivity within the porphyry copper system. Chargeability is relatively low to the north but becomes much lower at the porphyry although it increases significantly at depth. These results reflect the occurrence of more superficial sulfides in the Lagunas area of the system (north of the porphyry deposit) and a thicker leached cap in the more altered part of the system.

9.3.3 Minera Andes TITAN 24 Survey (2010)

Titan-24 DCIP-MT data were acquired at Los Azules during April and May 2010 by Quantec Geoscience Ltd., on behalf of Minera Andes Inc. The Titan-24 system acquires three types of geophysical data– magnetotelluric resistivity (MT), direct current resistivity (DC), and induced polarization (IP). The survey consisted of twelve parallel lines (L58400 N to L62450 N). From L58400 N to L62000 N the lines were 400m apart, L62550N was 550 m north of L62000 N and L63450 N was 900 m further north. Each line comprised one single spread of 3.6 km, except for L63450 N that was 3.3 km long. Full MT tensor data was acquired in all the lines and DCIP was collected in all but L59200 N and L59600 N. In total ten spreads of DC and IP data were acquired covering 35.7 km and twelve spreads of MT covering 42.9 km. Grid azimuth was 90° and the station interval was 150 m. These coordinate references are in Campo Inchauspe.

Over 130 IP anomalies were identified. Of these, 20 were classed as priority 1, 20 as priority 2, and 12 as priority 3. The priority 1 anomalies are larger targets, at least 200 m across, and described by Quantec as being consistent with the porphyry and near- porphyry mineralization model.

Two large deep resistivity anomalies, one high to the east, under the Los Azules mineralization, and one low to the west are well defined by the MT survey. The anomalies occur at depths-to-center ranging from 800 m to 1.5 km. Depth-to-top is rarely less than 500 m. The width of the anomalies is 800 m to 1 km for the resistivity low and 500 m to 800 m for the resistivity high. Quantec postulated that the deep anomalies are most likely related to conductive sulfides perhaps in a disseminated pyrite/sulfide shell surrounding a concealed porphyry intrusion. These anomalies, which are referred to as the "Southwest Target", are the targets that were tested in Hole T-01B in 2011 and Hole 1279 in 2012 (Figure 9.2).

Hole T-01B is located 200 m north of section 58,400N, and Hole 1279 is located 100 m south of the drill section. The section shows the limit of mineralization prior to the 2010 and 2011 drilling campaigns.

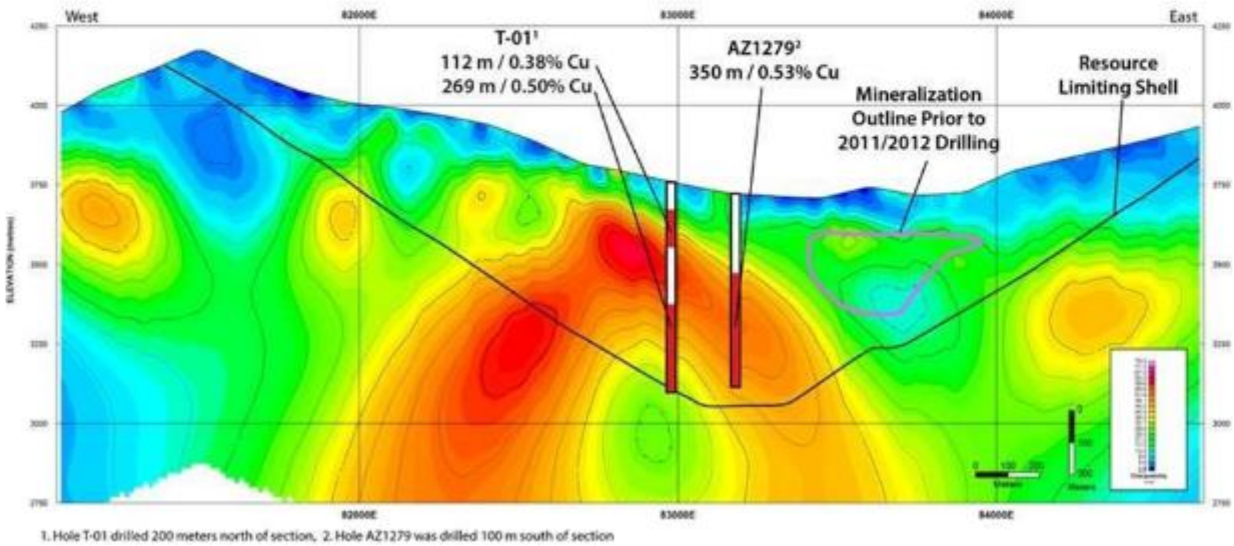


Figure 9.2: Section 58,400N Showing 2D IP Inversion Anomaly (Southwest Target) (McEwen 2012).

9.3.4 McEwen Mining: Ground Magnetic Survey (2012)

During January 2012, Quantec Geoscience Argentina S.A. performed a ground magnetic survey on the southwest portion of the Project. The survey consisted of 37 lines ranging from 1.1 km to 2.5 km, for a total of 57.2 line-km. The objective of the survey was to identify anomalous magnetic signatures that might be related to copper porphyries. The survey was acquired on a “stop-and-go” configuration, collecting data at 10 m intervals. The data was presented as maps of the Total Magnetic Field, Reduction to the Pole transform, Analytic Signal, Tilt Derivative and First Vertical Derivative.

Figure 9.3 is the Total Magnetic Field map for the 2012 survey overlain on the image shown in Figure 9.1. The 2012 magnetic data shows a discontinuous north-northwest trending magnetic low southwest of and roughly parallel to the prominent magnetic low that corresponds to the location of the main Los Azules deposit.

Areas of high magnetic response indicate the presence of elevated levels of magnetic minerals such as magnetite, pyrrhotite and hematite, whereas areas of low magnetic response may be caused by alteration processes such as magnetite destruction or may simply indicate rock types that never had magnetic minerals. This anomaly was tested with one drill hole during the 2012 season and intersected only traces of copper mineralization.

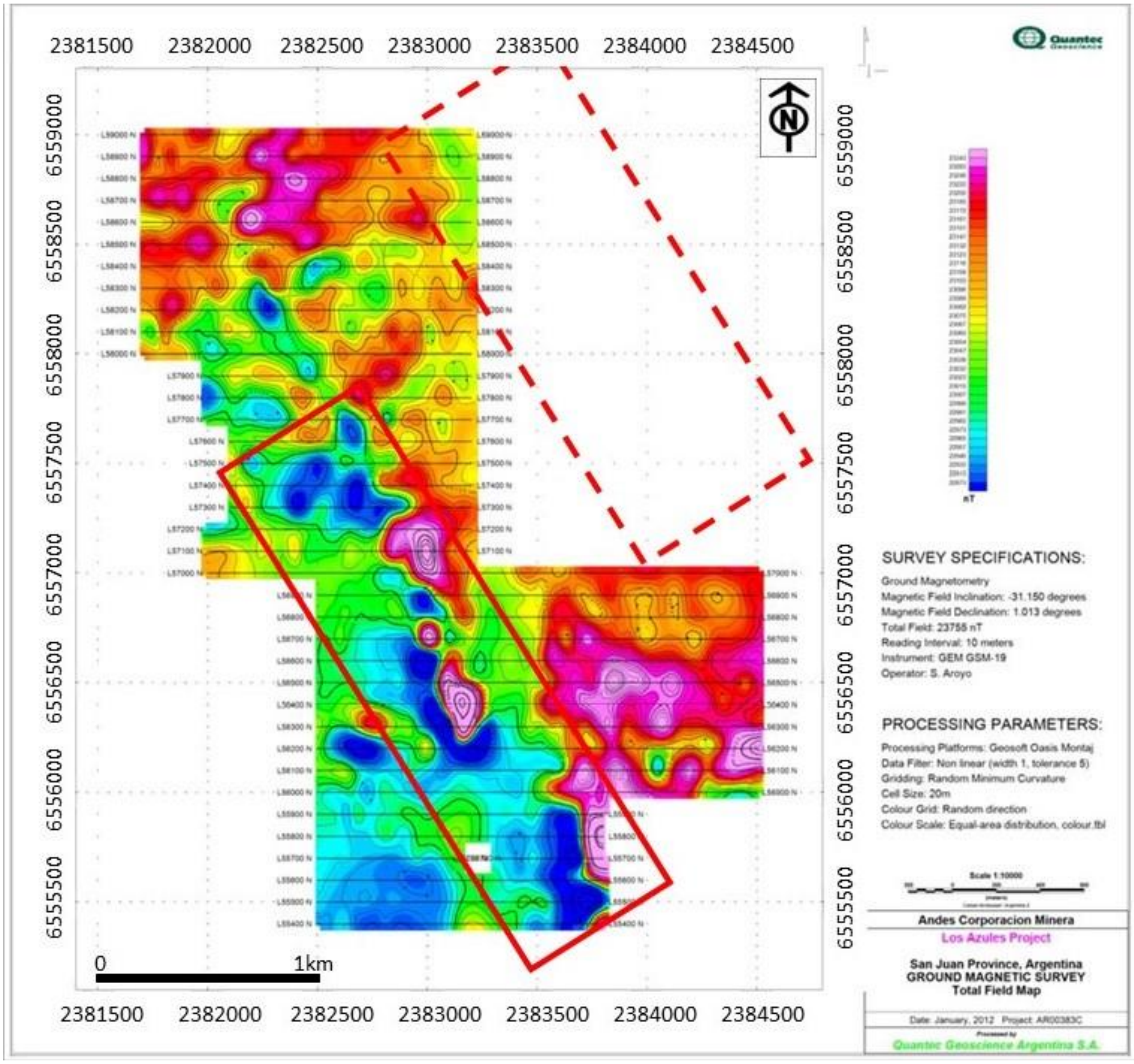


Figure 9.3: Total Magnetic Field Map of Los Azules. (Quantec, 2012). Note: Dashed red box indicates the mag low across the Ballena Ridge seen above in Fig 9.1 – the solid red box indicates the discontinuous mag low to the southwest.

9.4 SURVEYS AND INVESTIGATIONS

Mineral exploration at Los Azules has been carried out successively by Battle Mountain Gold, MIM-Xstrata and Minera Andes-McEwen Mining, McEwen Copper and/or professional consultants or contractors employed by these companies.

Jemielita (2010) reviewed the exploration program and data and reported that "Mineral exploration at Los Azules appears to have been carried out in a competent manner and to accepted industry standards.", although he noted that he did not conduct a rigorous confirmation of the quality of exploration work.

In 2017, McEwen Mining engaged consultant Rodrigo Diaz to conduct an evaluation of remote spectral geology (RSG) over a 17 km by 20 km area at Los Azules and later extended to include an area 38 km by 42 km. After numerous tests, spectral data of Landsat 8 (30-15 m pixel and 16-bit radiometric resolution), and for completing the analysis and interpretation, spectral data of Aster (30-15 m pixel and 8-bit radiometric resolution) were selected and used; additionally, spectral data of the Sentinel 2 (20-10 m pixel and 16-bit radiometric resolution) and Sentinel 1-Radar (10 m pixel) was also used.

In 2022, McEwen Copper undertook a program of continuous hyperspectral scanning and high-resolution core photography on the entire available archive of drill core completed in 2022 and previous programs stored at Calingasta. By July 2022, this represented some 64,000 m of scanned material available to augment completion of an updated geological model and support the design of the ongoing metallurgical program. This work is expected to be a protocol for all future drill programs.

Also in 2022, McEwen Copper engaged the services of Murphy Geological Services to complete a structural interpretation of Sentinel-2 and high-resolution imagery of the Los Azules property and immediate surrounding area. Sentinel-2 is a new earth observation sensor with 13 spectral bands having resolutions up to 10 m which was launched by the European Space Agency in June 2015 and is a significant improvement on the 15m resolution pan-sharpened Landsat-7 and ASTER data and allows more detailed structural analysis. An interpretation of a 45 km (E-W) by 35 km (N-S) Sentinel-2 extract centered on Los Azules was undertaken at up to 1:10,000 scale and a more detailed structural analysis of high-resolution satellite imagery for Los Azules project area at up to 1:2,000 scale.

Completion of the Diaz work in 2017 and Murphy work in 2022 is foundational to designing and re-establishing more regional reconnaissance exploration at Los Azules.

9.5 FUTURE EXPLORATION

The goals of future exploration at Los Azules include the establishment of upside potential on the property, ongoing geological model refinements, deposit growth, resource category upgrades, and identification/discovery of new porphyry mineralization as extensions of the Los Azules deposit, as well as new porphyry systems.

Future exploration work programs should carry out reconnaissance study, field mapping, geophysical surveying, and core drilling to achieve these goals. More specifically, these activities should include:

- An updated regional scale Spectral study for alteration definition, characterization of known mineralization and generation of new targets.
- Satellite-based litho-structural mapping to complement updated spectral study.
- Reconnaissance geological mapping and geochemistry to increase geological and structural understanding of the known mineralization, to ground truth interpretations of the satellite mapping study and identification or refinement of potential exploration target areas.
- Continue core relogging and validation versus hyperspectral scanning results to ensure a unified geological model of the deposit supported by all datasets from current and historic programs.
- Reprocessing of the raw 2010 Quantec Titan Survey data.
- Strategic core drilling of interpreted Los Azules deposit extensions and over selective high-quality exploration targets to be generated on the property.
- Continued infill core drilling to upgrade the priority portion of the resource based on goals of the planned Feasibility Study.

10.0 DRILLING

Drilling programs have been undertaken at Los Azules between 1998 and 2023 by three different mineral exploration companies including BMG, MIM Argentina (now Glencore) and Minera Andes/McEwen Mining and McEwen Copper. Drilling included reverse circulation programs mostly for gold exploration and diamond drilling focusing on supergene and hypogene porphyry-style copper mineralization. Descriptions of these programs are detailed in the following sections. Table 10.1 provides a summary of the drilling information.

Table 10.1: Exploration Drilling by Year and by Company

Year	Company	No. of holes	Meters
1998	Battle Mountain Gold	16	3,614
1999	Battle Mountain Gold	8	2,067
2004	Glencore Xstrata (MIM)	4	864
2003 - 2004	Minera Andes	9	2,064
2005 - 2006	Minera Andes	11	2,602
2006 - 2007	Minera Andes	17	3,501
2007 - 2008	Minera Andes	18	5,469
2009 - 2010	Minera Andes	28	10,229
2010 - 2011	Minera Andes	44	10,405
2011 - 2012	McEwen Mining	8	2,830
2012 – 2013	McEwen Mining	22	15,873
2017	McEwen Mining	17	6,469
2018	McEwen Mining	79	4,274
2022	McEwen Copper	65	23,811
2023	McEwen Copper	93	22,592
Total		**439(1)**	**116,664**

1. This table includes all drilling that has occurred on the property. Some holes were redrilled due to drilling difficulties and are not included in the database. Holes that were started in one season and completed the following season are counted in the year they were started, but the meters drilled in each season are shown for the respective seasons. The drilling reflects all holes to the effective date of May 9th, 2023.

The drill plan showing collar locations by the year drilled is shown in Figure 10.1.

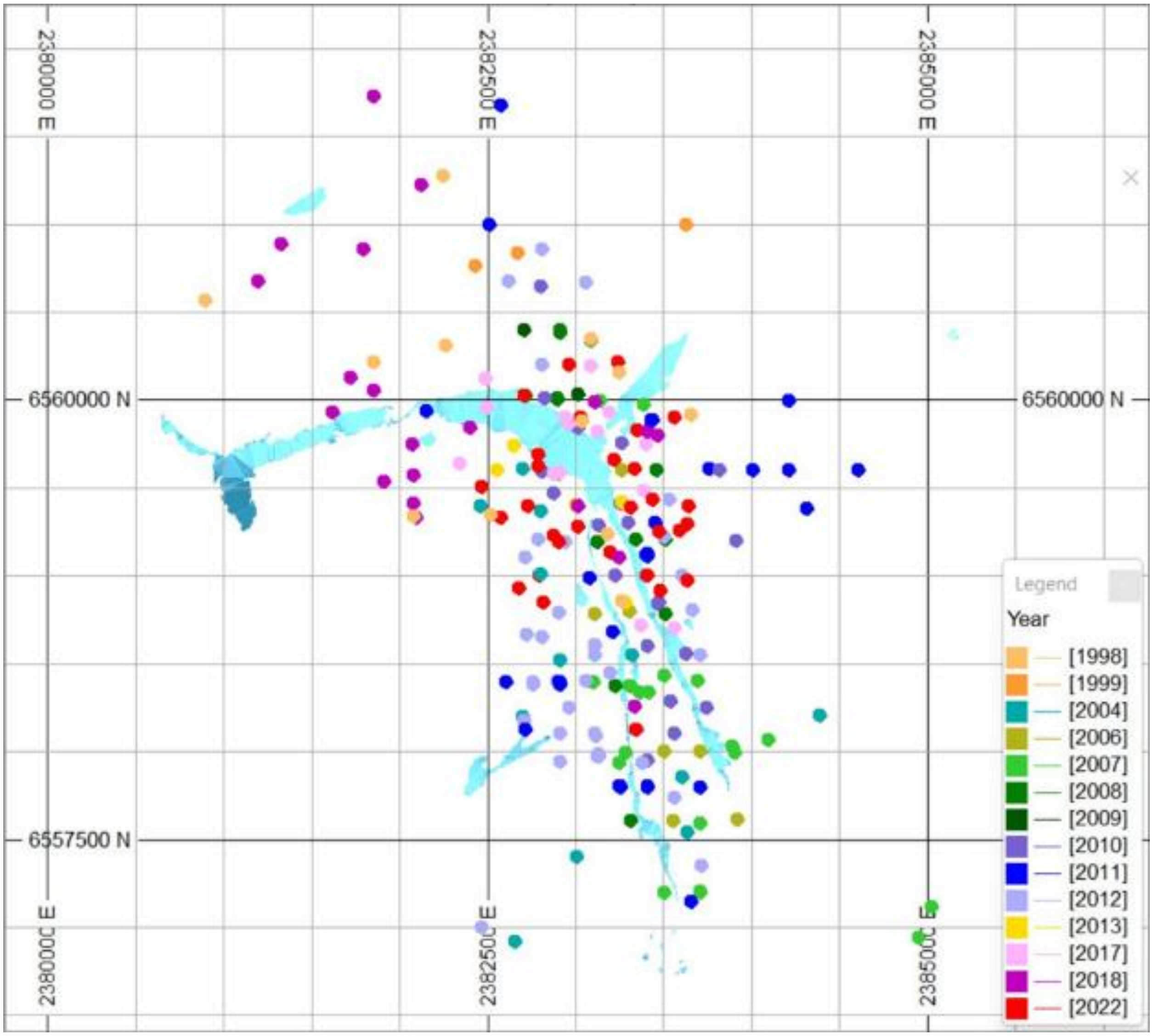


Figure 10.1: Plan Showing Locations of drill holes at Los Azules (CRM 2022)

10.1 DRILLING PROCEDURES AND CONDITIONS

Drilling by McEwen Mining Inc. was contracted to various drilling companies including Connors Drilling, Patagonia Drill Mining Services, Adviser Drilling, Boland Minera, Major Drilling, Foraco Argentina, HG Perforaciónes, Conosur, and Boart Longyear. Drilling conditions have been particularly difficult especially in faulted intersections or in areas of unconsolidated surface scree/talus.

10.2 BATTLE MOUNTAIN GOLD (1998-99)

In 1998 and 1999 BMG drilled 24 reverse circulation (RC) holes for a total of 5,681 m during a gold exploration program. Chalcopyrite, chalcocite and covellite mineralization was encountered in at least three drill holes (Rojas, 2010).

10.3 MIM-XSTRATA (2004)

In 2004 MIM Argentina (now Glencore) drilled four RC holes (864 m) at Los Azules (Rojas,2010).

10.4 MINERA ANDES/MCEWEN MINING (2004-2017)

Minera Andes/McEwen Mining drilled 174 drill holes for a total 59,442 m in nine campaigns (2003-2004, 2005-2006, 2006-2007, 2007-2008, 2009-2010, 2010-2011, 2011-2012, 2012-2013 and 2017). Drilling concentrated on identifying a zone of secondary enrichment in a grid with holes spaced at 200 m along east-west lines spaced at 400 m. Infill diamond drill holes were drilled during the 2009-10 campaign with a target depth of 400 m, achieved or exceeded in seventeen holes, four of which exceeded 600 m in depth. During the 2009-2010 campaign three RC holes for hydrologic and geotechnical testing were completed. Drilling during the 2010-2011 campaign included 16 infill or step-out diamond drill holes, six diamond drill holes for hydrology and geotechnical testing and 20 RC holes for condemnation and hydrology testing. Drilling during the 2011-2012 campaign comprised 10 infill and step- out diamond drill holes. During the 2012-2013 campaign all 22 diamond drill holes were for the purposes of expanding the resource either to depth or laterally. The 2017 program included fourteen delineation holes in the northern part of the deposit plus three holes drilled for geotechnical purposes.

10.5 MCEWEN MINING (2018)

A total of 79 holes and 4,274 m of drilling were completed in the 2018 program. This was made up of one new core hole of 450 m and 78 reverse circulation (RC) holes totaling 3,824 m. The activities performed were mainly technical site investigations and environmental base line monitoring work, to advance permitting efforts.

10.6 MCEWEN COPPER (2022-2023)

Over the period of two drilling seasons from January 2022 to May 2023, McEwen Copper completed 40,815 m of core drilling in 99 holes. The primary purpose of the drilling was for mineral resource upgrading of Inferred material to Indicated (66 holes for 28,221m) and metallurgical purposes (33 holes for 12,594m). In addition, there were a further 59 holes for 5,588m drilled by RC, core and sonic drilling methods used for geotechnical, hydrological and ground investigation work in the area. Figure 10.1 shows the location and distribution of Los Azules drill holes based on core and RC drilling methods.

10.7 LOGGING

Samples taken from drill holes at Los Azules are logged at the Project camp by geologists employed or contracted by McEwen Copper. Sampling procedures are described in Section 11.2. Emphasis is given to recording rock-types, alteration associations, types and distribution of mineralization and the presence of various types of veinlets and structures. These features are logged onsite (Figure 10.2) then transferred to a digital database.

Figure 10.2: Logging and inspection of drill core

Geotechnical observations and parameters are recorded including percentage of core recovery, RQD, Schmidt Hammer hardness determinations, point load testing, fracture density and angle relative to the length of the hole, as well as fracture fill material (Figure 10.3). This information is transferred to the digital database.

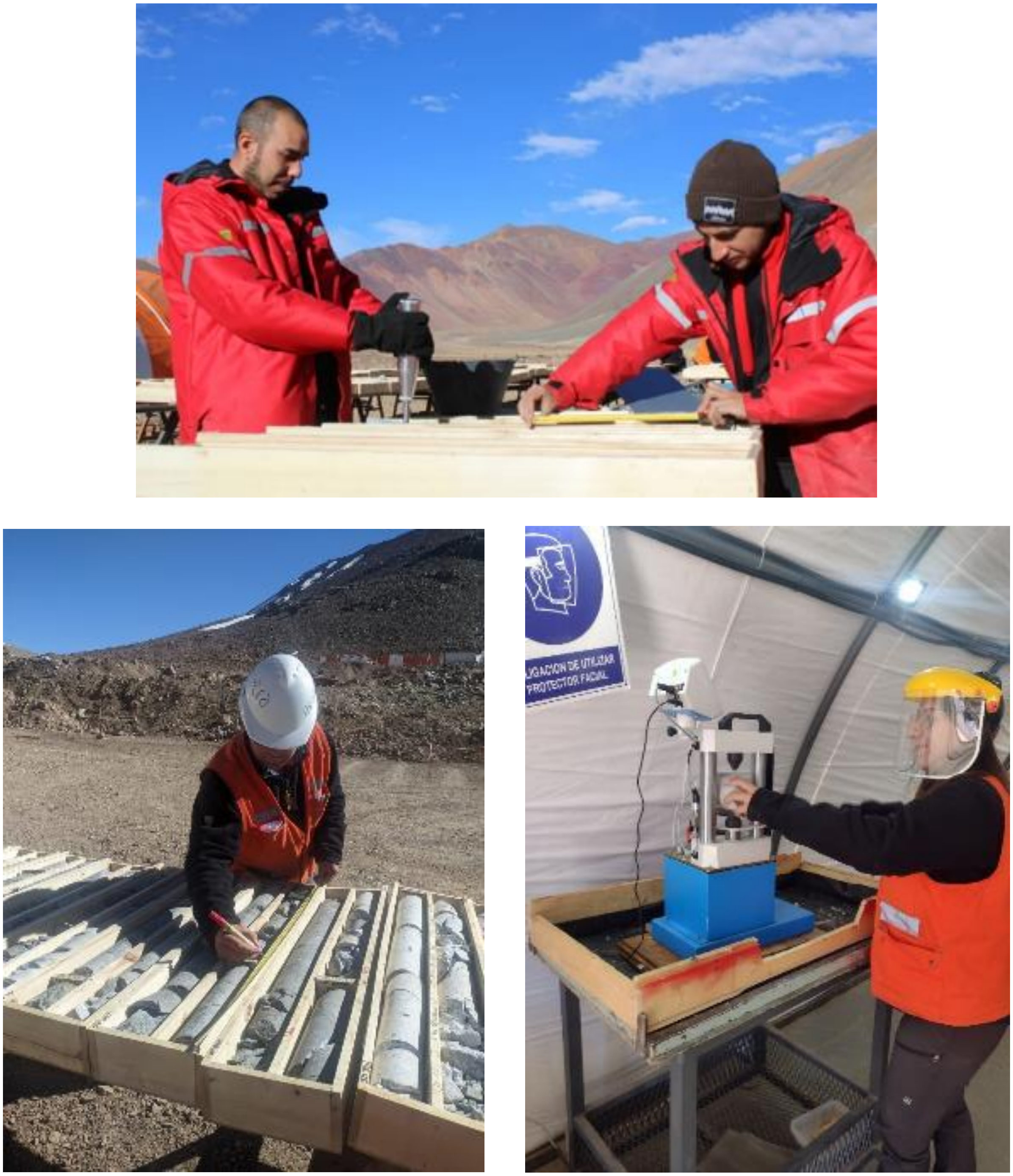


Figure 10.3: Geotechnical logging and data collection

Log sheets are coded, and details recorded for interval depth, interval width, lithology, alteration types, alteration intensities, alteration minerals, structure, percentage vein quartz, percentage total disseminated sulfides, mineralization minerals, mineral zone (hypogene or supergene), jarosite, goethite, hematite, covellite, chalcocite, pyrite, chalcopyrite, bornite and other observations.

10.8 SURVEYS

According to McEwen Copper staff, downhole surveying is done on drill holes by the drilling contractors using REFLEX and/or Sperry-Sun tools. Density determinations were made for 915 drill core samples prior to the 2022 drill program. During the 2022 campaign, a program of hyperspectral scanning of the entire available core archive of some 64,000 m was completed.

10.9 DRILL HOLE RESULTS

There are a total of 439 drill holes in the Los Azules database with a cumulative length of 116,664 m. A summary of the significant drilling results is found in Table 10.2 for campaigns prior to 2018 and Table 10.3 for the 2022 drilling campaign.

Drilling has confirmed the presence of a hypogene porphyry copper deposit in a continuous body, as well as the presence and continuity of an overlying supergene chalcocite enrichment blanket. The extent of the mineral resource measures approximately 4 km north-south by 1.5 km west-east. Many of the drill holes in the central and northern parts of the deposit have been terminated in mineralization that exceeds the 2017 PEA base case cut-off grade of 0.20% Cu. Drilling during the 2012-2013 campaign extended the depth of the mineralized system in the southwestern part of the deposit to at least 1,000 m.

Table 10.2: Examples of Significant Drilling Results Prior to 2022

Drill Hole ID	TD (m)	Intersection		Interval (m)	Total Copper (%)
		From (m)	To (m)		
AZ0401	195.0	130.0	195.0	65.0	0.62
Including		150.0	192.0	42.0	0.82
AZ0402	330.5	164.0	304.0	140.0	0.38
Including		164.0	190.0	26.0	0.47
Including		230.0	304.0	74.0	0.42
AZ0404	300.8	162.0	282.0	120.0	0.54
Including		162.0	202.0	40.0	0.59
Including		236.0	282.0	46.0	0.64
AZ0407	168.8	96.0	152.0	56.0	0.44
Including		126.0	152.0	26.0	0.58
AZ0610	261.4	174.0	261.4	87.4	0.83
AZ0611	270.7	112.0	270.7	158.7	0.51
AZ0614	224.6	132.0	180.0	48.0	1.13
Including		136.0	158.0	22.0	1.40
AZ0617	183.5	66.0	183.5	117.5	0.63
Including		66.0	124.0	58.0	0.84
AZ0619	299.4	78.3	299.4	221.2	1.62
Including		78.3	116.0	37.8	2.22
Including		134.0	146.0	12.0	3.94
AZ0620	253.3	80.0	226.0	146.0	1.10
Including		80.0	106.0	26.0	1.54

Table 10.2: Examples of Significant Drilling Results Prior to 2022					
Drill Hole ID	TD (m)	Intersection		Interval (m)	Total Copper (%)
		From (m)	To (m)		
AZ0722	271.2	119.0	155.0	36.0	0.99
AZ0724D	278.2	124.0	160.0	36.0	0.79
AZ0729B	226.9	130.0	214.0	84.0	0.73
Including		172.0	204.0	32.0	0.94
AZ0730	342.6	123.0	323.8	200.8	0.89
Including		140.0	253.0	113.0	1.04
AZ0832	420.0	80.0	140.0	60.0	0.78
AZ0833	387.8	73.0	313.0	240.0	0.94
AZ0837A	541.0	326.0	516.0	190.0	0.82
AZ0841	400.2	241.0	285.0	44.0	1.83
AZ0843	176.0	67.0	131.0	64.0	0.69
AZ0946	469.4	110.0	469.4	359.4	0.63
Including		115.0	260.0	145.0	1.08
AZ1047	493.1	74.0	493.1	419.1	0.50
Including		102.0	182.0	80.0	0.92
AZ1048	466.1	105.0	466.1	361.1	0.77
Including		123.0	339.0	216.0	1.01
AZ1049	491.2	62.0	491.2	429.2	0.75
Including		62.0	298.0	236.0	1.05
AZ1050	408.5	94.0	408.5	314.5	0.30
Including		94.0	132.0	38.0	0.68
AZ1051	620.2	69.0	620.2	551.2	0.35
Including		363.5	426.0	62.5	1.12
AZ1052	425.0	103.0	425.0	322.0	0.42
AZ1053A	650.0	48.9	650.0	601.1	0.54
Including		122.0	230.0	108.0	1.03
AZ1055	408.5	116.0	408.5	292.5	0.55
AZ1056	295.3	70.0	295.3	225.3	0.47
Including		192.0	223.0	31.0	0.88
AZ1057	503.6	173.0	503.6	330.6	0.43
Including		173.0	225.0	52.0	0.84
Including		255.0	293.0	38.0	0.83
AZ1058	451.8	70.0	451.8	381.8	0.52
Including		96.0	181.0	85.0	0.99
AZ1059	656.4	88.0	656.4	568.4	0.47
Including		330.0	404.0	74.0	0.90

Table 10.2: Examples of Significant Drilling Results Prior to 2022					
Drill Hole ID	TD (m)	Intersection		Interval (m)	Total Copper (%)
		From (m)	To (m)		
AZ1060A	402.5	116.0	402.5	286.5	0.50
Including		130.0	170.0	40.0	0.69
AZ1061A	293.4	71.0	293.4	222.4	0.90
Including		71.0	250.0	179.0	1.04
AZ1062	280.0	130.0	280.0	150.0	0.64
Including		130.0	248.0	118.0	0.70
AZ1063	427.1	94.0	427.1	333.1	0.72
Including		94.0	232.0	138.0	0.81
AZ1064	170.1	136.0	170.1	34.1	0.47
AZ1064A	404.4	120.0	248.0	128.0	0.75
And		248.0	404.4	156.4	0.39
AZ 1168	569.3	148.0	569.3	421.3	0.66
AZ 1169	315.8	86.0	315.8	229.8	0.36
AZ 1170	349.3	112.0	349.3	237.3	0.63
AZ 1175	355.2	74.0	340.0	266.0	0.22
And		340.0	355.2	15.2	0.72
AZ 1176	393.4	162.0	292.0	130.0	0.63
T-01B	656.0	80.0	192.0	112.0	0.38
And		387.0	656.0	269.0	0.50
AZ 1279	622.7	272.0	456.0	184.0	0.38
And		456.0	622.7	166.7	0.71
AZ 1282	482.1	309.5	314.0	4.5	2.60
AZ 1289	367.0	220.0	367.0	147.0	0.44
AZ 1291	890.5	72.0	232.0	160.0	0.61
And		562.0	790.0	228.0	0.40
And		790.0	890.5	100.5	0.71
AZ 1294	861.9	62.2	74.0	11.8	0.53
And		252.0	861.9	609.9	0.47
AZ 1295	1044.5	422.0	1044.5	622.5	0.51
Including		580.0	618.0	38.0	1.07
Including		720.0	744.0	24.0	1.16
Including		970.0	1044.5	74.5	0.61
AZ 1296	523.2	156.0	244.0	88.0	0.92
AZ 1297	980.8	276.0	690.0	414.0	0.50
Including		436.0	490.0	54.0	1.07
AZ 1299	1074.6	78.0	94.0	16.0	0.55

Table 10.2: Examples of Significant Drilling Results Prior to 2022

Drill Hole ID	TD (m)	Intersection		Interval (m)	Total Copper (%)
		From (m)	To (m)		
And		546.0	1074.6	528.6	0.44
AZ 12101	237.0	168.0	237.0	69.0	0.87
AZ 12114	814.5	224.0	374.0	150.0	0.70

Source: Minera Andes press releases dated May 5, 2004, May 31, 2007, November 14, 2007, April 16, 2008, June 6, 2008, March 8, 2010, June 21, 2010, and June 27, 2011, and McEwen Mining press releases dated May 10, 2012, January 17, 2013 and March 28, 2013. TD = total depth

Table 10.3: Examples of Significant Copper, Gold and Silver Drilling Results From 2022 Campaign									
Hole-ID	Section	Predominant Mineral Zone	From (m)	To (m)	Length (m)	Cu%	Au (g/t)	Ag (g/t)	Comment
AZ22137A	36	Total	133.0	557.3	424.3	0.47	0.027	0.008	
incl		Enriched	133.0	342.0	209.0	0.49	0.028	0.016	
and		Primary	342.0	557.3	215.3	0.44	0.026	0.001	incl. 8m of 1.00% Cu in Primary
AZ22138	36	Total	138.0	660.1	522.1	0.42	0.064	1.883	
incl		Enriched	138.0	348.0	210.0	0.60	0.064	2.180	incl. 28m of 0.87% Cu in Enriched
and		Primary	348.0	660.1	312.1	0.30	0.065	1.683	
AZ22139	36	Total	114.5	282.6	168.2	0.08	0.060	1.447	
incl		Enriched	206.5	282.6	76.1	0.12	0.032	1.164	
AZ22140	36	Total	117.4	342.8	225.4	0.16	0.030	1.117	
incl		Enriched	117.4	314.0	196.6	0.16	0.032	1.173	
and		Primary	314.0	342.8	28.8	0.16	0.019	0.732	
AZ22141	40	Total	183.1	551.0	367.9	0.50	0.069	1.535	
incl		Enriched	183.1	360.0	176.9	0.50	0.044	1.437	
and		Primary	360.0	551.0	191.0	0.50	0.092	1.625	
AZ22142	36	Total	92.0	511.1	419.1	0.79	0.152	3.508	Incl. 32m of 1.11% Cu &
incl		Enriched	92.0	278.0	186.0	0.93	0.095	3.544	104m of 1.00% Cu in Enriched
and		Primary	278.0	511.1	233.1	0.67	0.198	3.479	46m of 1.59% Cu in Primary
AZ22143	36	Total	92.5	403.0	310.5	0.20	0.015	0.880	
incl		Enriched	92.5	266.0	173.5	0.22	0.016	0.985	
and		Primary	266.0	403.0	137.0	0.18	0.014	0.747	
AZ22144	36	Total	58.0	506.6	448.6	0.30	0.02	0.84	
incl		Enriched	58.0	204.0	146.0	0.31	0.01	0.52	
and		Primary	204.0	506.6	302.6	0.29	0.02	1.00	incl 104.6m of 0.48% Cu
AZ22145	40	Total	76.0	257.0	181.0	0.18	0.02	1.90	
incl		Enriched	76.0	194.0	118.0	0.16	0.03	2.25	
and		Primary	194.0	257.0	63.0	0.21	0.01	1.26	

Table 10.3: Examples of Significant Copper, Gold and Silver Drilling Results From 2022 Campaign									
Hole-ID	**Section**	**Predominant Mineral Zone**	**From (m)**	**To (m)**	**Length (m)**	**Cu%**	**Au (g/t)**	**Ag (g/t)**	**Comment**
AZ22146	40	Total	91.0	421.5	330.5	0.83	0.11	2.30	
incl		Enriched	91.0	394.0	303.0	0.86	0.11	2.26	incl. 103.4m of 1.31% Cu
and		Primary	394.0	421.5	27.5	0.50	0.10	2.76	
AZ22147	48	Total	60.0	240.8	180.8	0.03	0.02	0.50	
incl		Enriched	60.0	67.0	7.0	0.10	0.08	1.27	
AZ22148	40	Total	76.0	315.0	239.0	0.26	0.02	1.01	
incl		Enriched	76.0	212.0	136.0	0.33	0.02	0.85	
and		Primary	212.0	315.0	103.0	0.16	0.02	1.23	
AZ22149	48	Total	131.6	428.0	296.4	0.55	0.04	1.62	
incl		Enriched	131.6	278.0	146.4	0.34	0.02	0.32	
and		Primary	278.0	428.0	150.0	0.76	0.06	2.91	incl 54m of 1.38% Cu from 376m
AZ22150	44	Total	78.0	257.4	179.4	0.14	0.01	0.53	
incl		Enriched	78.0	126.0	48.0	0.04	0.01	0.25	
and		Primary	126.0	257.4	131.4	0.17	0.01	0.63	
AZ22158	30	Enriched	72.0	294.0	222.0	0.95	0.09	1.57	incl 44m of 1.38% Cu from 144m
and		Primary	294.0	300.0	6.0	0.34	0.05	0.43	
AZ22161	48	Enriched	116.0	354.0	238.0	0.58	0.07	1.19	
AZ22162	36	Enriched	102.0	450.0	348.0	0.28	0.40	1.13	
AZ22163	44	Enriched	92.0	286.0	194.0	0.56	0.04	0.68	
AZ22164	38	Leached	18.0	242.0	224.0	0.04	0.02	1.32	
AZ22165	48	Leached	24.5	200.0	175.5	0.04	0.04	1.27	
AZ22166	30	Enriched	82.7	199.6	116.9	0.13	0.02	0.81	incl 53.6m of 0.25% Cu from 146m
AZ22167	44	Enriched	72	152.4	80.4	0.21	0.02	0.78	incl 54.4m of 0.25% Cu from 98m

Source: McEwen Copper press release dated August 4, 2022

10.10 TRUE THICKNESS OF MINERALIZATION

Supergene mineralization forms a sub-horizontal zone measuring over 5 km north-south by 1.5 km west-east. It is underlain by hypogene mineralization that extends to depths greater than 1 km below surface. The sub-vertical geometry of key deposit lithologies and structural elements, coupled with predominantly vertically oriented drill holes prior to 2022, effectively represent the true thicknesses of mineral zones, but lacked effective constraints on temporal relationships impacting grade. The use of inclined drill holes for resource delineation drilling beginning with the 2022 campaign has served to improve the interpretation of the constraining sub-vertical geological elements.

10.11 ADEQUACY STATEMENT ON SECTION 10

The deposits at Los Azules are relatively broad zones, with variable orientations of breccias. This has required reorienting drill programs as the deposit is better understood to more accurately capture the volumes of higher-grade mineralization. Insufficient drilling that captures the fabric of the deposits risk over or under estimation of the volumes of the Mineral Resource. Lower grade zones are well behaved and are well represented. The deposits remain open at depth and along the primary axis of continuity.

The QP believes that the quantity and quality of the lithological, collar and downhole survey data collected during the exploration and infill drill programs completed at Los Azules are acceptable to support Mineral Resource estimation.

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 INTRODUCTION

Mr. Allan Schappert (CPG, SME-RM) of Stantec Consulting, QP, visited the Los Azules property during the period from 24 April to 15 May 2022. The purpose of the visit was to observe, review, and comment on all aspects of data collection, recording, and analysis in preparation of the Mineral Resource Estimate. Activities and discussions included the following: visit and inspection of operating drill sites; care, custody, and control procedures of core boxes; core logging facilities at Los Azules camp; core storage and sampling procedures at the Calingasta warehouse; a tour of the independent assay lab in Mendoza; review of historical and current QA/QC protocols with a review of recent results.

The results of the historical and current drilling program were discussed with the Project staff and select intervals from a series of drill holes were reviewed. A series of surface exposures were visited at the deposit site. Active drill sites were visited and a series of (completed) drill holes collars were observed.

Mr. Schappert reviewed the logging, sampling procedures and Quality Assurance/Quality Control (QA/QC) practices used during the drilling program. The sampling practices were found to adhere to accepted industry standards. Standard reference material (SRM) was prepared and certified by Alex Stewart laboratory in Mendoza, Argentina from local source rocks. Blank material was initially made from “barren” quartz with a small portion of leached material “to add some color” (i.e., to appear anonymous in the sample sequence). As discussed later in the section, this material is not completely sterile, and another source of blank material was obtained for QA/QC programs after 2008. “Coarse” duplicates taken at site in 2008 were core duplicates obtained from quarter core splits. Coarse reject duplicates were eventually submitted for 2008 and included in the 2009 and subsequent programs.

Assay results from blank material fell within acceptable limits in all programs after 2009 when silica sand was used instead of the previous blank material. In the 2021/22 drilling season, a coarse quartz material sourced from Alex Stewart Labs replaced the previous blank samples.

Since 2013, all sample preparation (crushing and pulverizing) and assaying is completed at Alex Stewart Labs in Mendoza. Historical assaying and sample preparation has taken place at other accredited labs in Argentina and Chile.

Laboratories utilized by McEwen have internal QC samples used in each batch of sampled material provided by McEwen. Each assay certificate lists the drill sample results, plus the laboratory’s internal sample control results that consist of its own duplicates, blank and reference standard pulp with each batch assayed for its internal quality control on precision, instrument drift and accuracy to determine if there are any sampling issues for that run. Anomalously high values within batches are verified by re-assay as a matter of routine.

11.2 SAMPLING METHODS

The drilling programs that have occurred on the Los Azules property since 1998 have used both reverse circulation (RC) and diamond core (core) equipment. All holes drilled by BMG in 1998 and 1999 were RC type. MIM, now Glencore, drilled four RC holes in 2004. Since 2004, Minera Andes/McEwen Mining and McEwen Copper have mainly used core-drilling techniques. The procedure for logging the core is described previously in Section 10.

Once geotechnical and geological logging has occurred (including the mark-up of the sample locations on the core), the boxes of core are transferred to one of the nearby camp tents where a dedicated photo booth setup enables the core boxes to be photographed in order and in a consistent fashion (Figure 11.1). Photos are taken both wet and dry and are labelled according to hole ID, box number and the from-to information (Figure 11.2). The photographs are stored in the digital database for later reference if needed.

Figure 11.1: Dedicated static photo booth for consistent photography of core

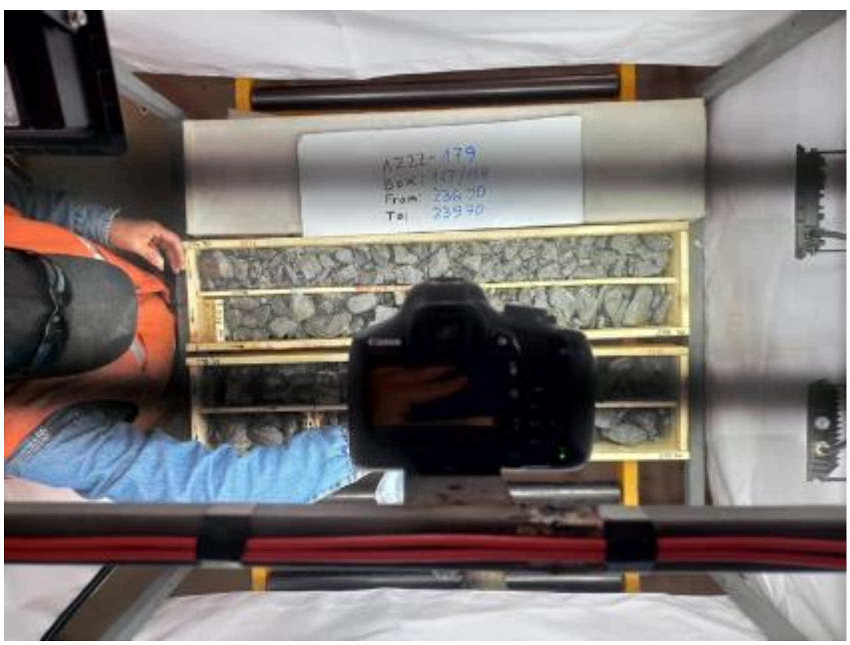


Figure 11.2: An example of the labelling of core boxes for photography

Once photography is complete, approximately 40-48 core boxes (depending on core size) are stacked on a pallet in the correct downhole order. Pallets of core are properly secured by plastic sheets and strapping (Figure 11.3) and are stored in a locked and security-sealed sea container awaiting shipment to the core warehouse in Calingasta. Twice weekly shipments are tracked by McEwen using Chain of Custody paperwork to ensure a secure delivery.

Figure 11.3: The securing and loading of the core boxes for shipment to Calingasta

Upon arrival at the Calingasta warehouse facility, a check is made on the integrity of the delivery according to the Chain of Custody documentation. Core boxes are unloaded into the warehouse where they are firstly processed using a GeoLOGr hyperspectral scanner unit (Figure 11.4). This unit uses continuous short-wave infrared (SWIR) point spectroscopy to provide objective drill core logs in a rapid and reliable manner. Currently two scanners are used to image all the core drilled on the project, with priority given to the freshly delivered core, but when time allows all historical core from previous campaigns will be scanned. In addition, high-resolution images of the surface of the core are taken simultaneously that can be used later when viewing the hyperspectral data core logs. After scanning, the core boxes are transferred immediately by roller tables to the sample prep area in the warehouse.

Figure 11.4: The hyperspectral scanning unit and the hydraulic core splitter

McEwen sampling staff use hydraulic guillotine core splitters (Figure 11.4) to split the more intact core fragments lengthwise as instructed by the logging geologist on the sampling sheet. Core is divided using a splitter to minimize the loss of sooty chalcocite, which could be lost by washing during cutting by a diamond saw. Depending on fracture density, RQD and general condition of the core, any core that is not whole or is significantly in the form of rubble is divided with a trowel to obtain a reasonable sample. One half of the core is kept behind in the core box and later stored on shelving units in the warehouse for posterity and later reference.

Sampled core fragments are immediately placed in thick plastic bags labelled with unique ID codes for each sample and sealed with nylon zip ties. Usually, three to seven individually bagged samples are then placed into larger poly-woven rice bags, labelled accordingly, and secured with a uniquely numbered tamper proof security seal (Figure 11.5).

Secured sacks are palletized and kept in a padlocked and security sealed storage area (Figure 11.5) until samples are dispatched weekly to the laboratory for analysis. CCTV and security lighting have been installed in the outdoor Calingasta warehouse facility that monitors the storage area. An inventory of samples and associated security seals is maintained and is used in the Chain of Custody paperwork protocols when samples are dispatched to the lab and checked when received by the lab.

Figure 11.5: Showing the sequence of bagging, tagging, sealing, and securing the samples for dispatch

The receiving, preparation, storage, and dispatch procedures at a well-organized and secure core facility produces samples from Los Azules that are appropriate for subsequent use in resource estimation.

11.3 SAMPLE PREPARATION AND ANALYSES

Once the samples are bagged in Calingasta, no McEwen employee is involved with any subsequent sample preparation. Samples are picked up regularly by staff from Alex Stewart International labs and delivered directly to their labs in Mendoza. The sacks are not opened until they reach the laboratory where the inventory of sample numbers and security seals are checked and referenced to the existing Chain of Custody paperwork protocols followed to this point.

Alex Stewart International (ASI) provides geochemical, metallurgical, and analytical services to the mining and mineral exploration industry in Africa, Europe, South America, and Asia. ASI laboratories are accredited to ISO 9001, 14001 and 17025 standards and participate in inter-laboratory tests and international round robins.

Sample preparation protocols consist of the following:

- Samples are dried until the desired moisture content is achieved. The entire sample is crushed to 80% passing 2mm (10 mesh).
- A sample splitter obtains a 600g fraction which is then pulverized to 95% passing 105 microns (140 mesh).
- Crushers and pulverizers are cleaned with high pressure air after every sample; granulometry tests are performed every fifteen (15) samples and reported in the final certificates.

Assays using the following methods are performed at the ASI labs in Mendoza:

- **Au4-30**: Gold analysis by fire assay and determination by AAS using a 30g sample.
- **ICP-AR 39**: multi-element suite analysis in aqua regia; determination by ICP-OES Radial.
- **ICP-ORE**: 19-element overlimit analysis for the above method; ICP-OES Radial.
- **LMC-140**: Sequential Copper Analysis to determine Acid Soluble Copper (using sulfuric acid), Cyanide Soluble Copper, Residual Copper, and Total Copper by AAS determination.

Sequential copper determinations for acid soluble copper (CuAS) and cyanide soluble copper (CuCN) are provided by a standard sequential copper assay methodology by the Alex Stewart laboratory. The methodology is comprised of a dilute sulfuric acid-controlled leach and assaying of the resulting dissolved copper followed by a dilute sodium cyanide-controlled leach and assaying of the resulting dissolved copper. Residual copper in the final tails is processed in a four-acid digestion and assayed to determine the total copper content. Acid soluble and cyanide soluble assays are combined to determine the approximate soluble copper content (CuSOL) of each sample based on the methodology. The sequential total copper assay determination is only used for comparative purposes and the total copper assayed in the drill program procedure is used in the drilling database.

Complete and final assay certificates are transmitted electronically to the Database Manager in Excel and pdf formats. If the assays pass the McEwen QA/QC protocols (see below) they are then entered into the Fusion database for later use in interpretation, modelling, and resource estimation.

Historically, sample preparation and assaying has variously taken place at Alex Stewart Labs, ALS Chemex (Chile), ACME labs (Mendoza and Chile). ALS Chemex and ACME are also ISO 9001:2000 certified labs. The sample prep at the different labs is very similar with minor variations in crushing, pulverizing and sample size but all to generally accepted industry standards.

McEwen Copper decided to undertake an extensive re-assaying program to augment the database prior to the updated estimate. The main objectives for the re-assaying program were aimed at obtaining missing data, benefiting from improved detection limits for deleterious elements (particularly for arsenic), benefiting from a consistent assay methodology using just one laboratory and to obtain sequential assay determinations where only total copper information was available. This program was initiated in 2022 and completed in 2023 - a total of 159 holes for 24,704 samples were re-assayed under this program using either remaining core or historical sample pulps and rejects (located at the Calingasta warehouse) and sent for re-assay at Alex Stewart Labs, Mendoza.

Analysis of the re-assayed samples focused on soluble copper with the objective of providing values for previously missing assay intervals when sequential assays were not performed, primarily in the early years of the drilling programs at Los Azules. In comparing the grades of re-assay and original assay pairs, a strong re-assay, high-bias was detected for acid soluble copper which is considered due to sample oxidation occurring in the stored samples. Based on this analysis, the resource estimate does not consider acid soluble re-assay data.

For cyanide soluble copper, strong correlation between the assay and re-assay values was observed for enriched zone samples although a possible small high-bias (<4%) was observed. For primary zone samples, poor correlation between the original and cyanide soluble re-assay values was observed. For the enriched zone, samples originally lacking cyanide soluble copper were re-assayed. For these samples, the re-assayed total and cyanide soluble grades were used in the resource estimate. For the Enriched zone, overall, the scatter is reasonable however there are small groups of data that are different, with the re-assay finding higher grades.

For total copper there appears to be a bias high in the original assays when compared to the re-assays and is most obvious in the early years of the drilling programs. For some years, notably 2006, 2007 and 2008, there are some large differences between the two results (poor precision) and there also groups of samples that are different from each other. In the later years (2017 and 2018) scatter reduces significantly and no systematic differences occur.

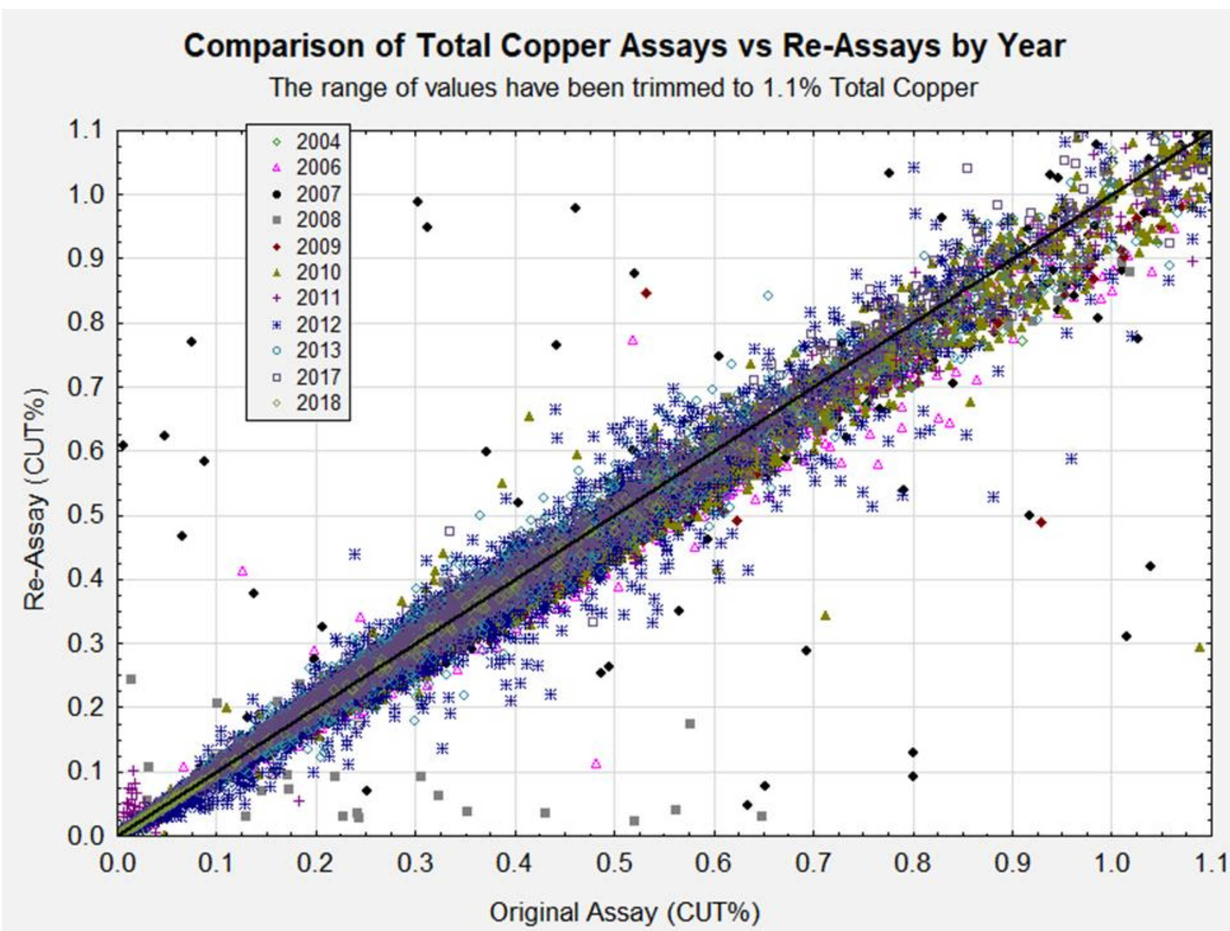


Figure 11.6: Total Copper Assays vs Re-Assays

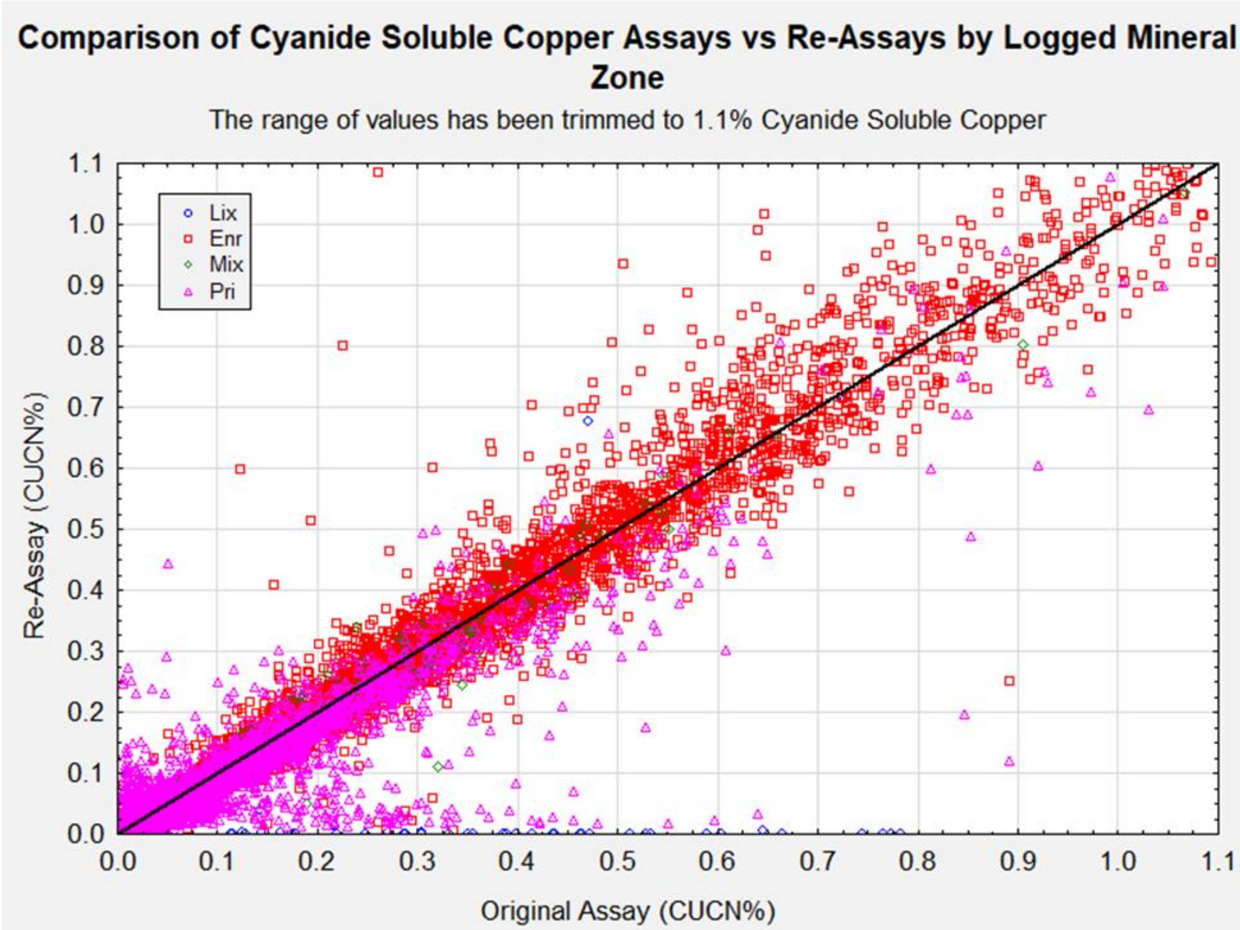


Figure 11.7: Cyanide Soluble Copper Assays vs Re-Assays

11.3.1 QC sample insertion

During the sample bagging and bundling process in Calingasta, McEwen staff insert standards and blank samples in pre-determined regular intervals as well as duplicate samples. In a sequence of forty samples there will be two blanks, two standards and one duplicate requested. The density of QC samples is more than adequate for the current program.

In addition to the McEwen QC samples, ASI perform their own internal QC checks that include both commercial and internal standards, blanks, and duplicates. Results of their internal QC are reported in the final certificates.

11.3.2 Chain of Custody

The chain of custody has been outlined in the previous paragraphs in this section. It appears that any tampering with individual bags or the ties would be immediately evident when the samples arrived at the lab. Any tampering with the larger bags would also be apparent on arrival at the lab. Documentation was provided such that it would be difficult for a mix up in the samples to occur either during shipment or at the lab.

All procedures were being carefully attended to and met or exceeded industry standards for collection, handling, and transport of drill core samples.

The Fusion database where the final assay results are stored has a strict read / write permissions policy and protocol to ensure no unauthorized access or editing of the database is possible. Back-ups of the database occur on a regular basis.

11.4 CONTROL SAMPLES

Control samples consist of blanks, duplicates, and standard reference samples. Independent third-party labs are also often used for check samples to assure assaying accuracy beyond standard QC protocols. Blank samples test for contamination; duplicates test for contamination, precision, and intra-sample grade variation; and reference standards test for assay precision and accuracy.

11.4.1 Standard reference materials (Standards)

Control standards and blanks used during the 2007 and 2008 field seasons were prepared using composites of coarse rejects from the 2006 field season. Color was added to the blanks by adding small amounts of coarse reject from the leached horizon of the deposit. Six standards ("SRM") were prepared with distinct copper and gold contents as shown in Table 11.1.

Table 11.1: Sample Control Standards (2007-2008)

Sample	Total Cu%	Au (ppm)
STD B	0.0047	0.0500
STD 01	0.1096	0.0470
STD 03	0.3135	0.0330
STD 06	0.5300	0.0260
STD 08	0.8830	0.0680
STD 20	1.9540	0.0670

Note: Values were obtained from statistical analysis received from Alex Stewart

For the programs from 2009 - 2010, Alex Stewart prepared and certified additional standard material with the same certified values for copper. It should be noted that the lack of precision in the gold assays precluded their use as gold SRM. This was due to the low gold values and assay detection limit effects. It was not a failing of either sample preparation or assaying.

For the 2011 program and since, Acme Laboratories prepared, and certified standard material as shown in Table 11.2. The gold values were, again, affected by detection limit effects and not used as SRM.

Table 11.2: Sample Control Standards (2011-2022)

Sample	Total Cu%	Au (ppm)
STD 01	0.101	0.014
STD 03	0.278	0.039
STD 10	1.030	0.059

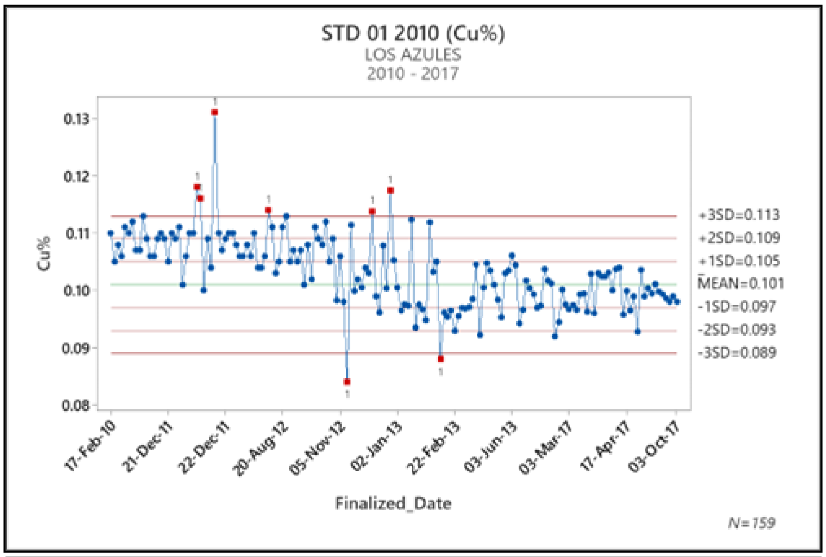
Note: Values were obtained from statistical analysis received from Acme.

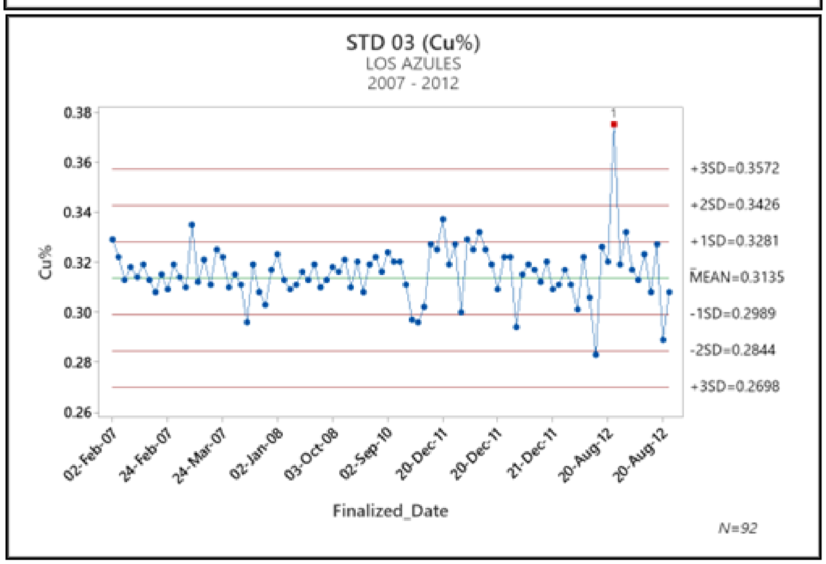
11.4.2 Control Sample Performance

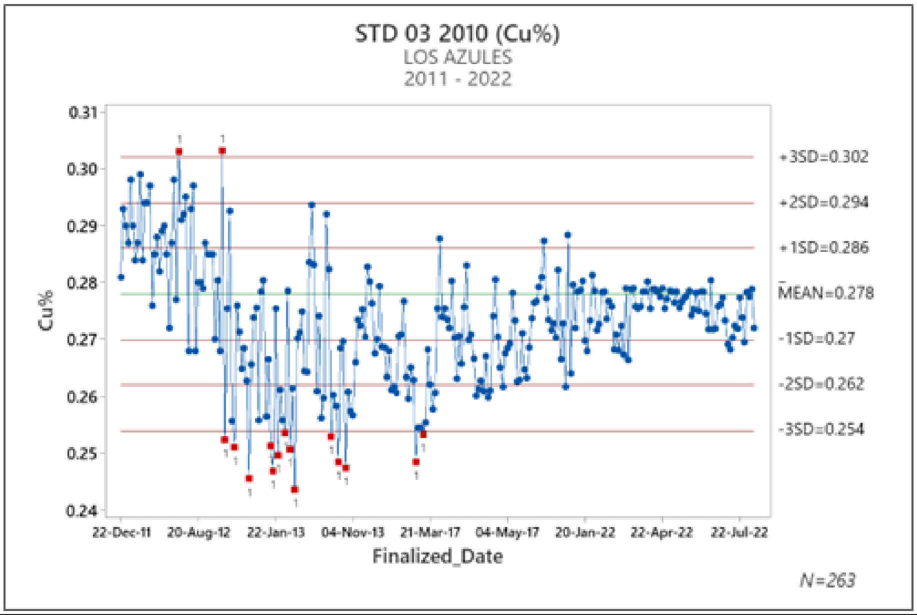
The performance of reference standards is evaluated using diagnostic charts whereby outliers can be identified if they lie outside of +/-3sd of the certified or expected value of the sample. Control charts are similar in that they monitor the consistency of performance at the lab when 90% of the results must fall within ±10% of the mean value of the assays for the process to be "in control".

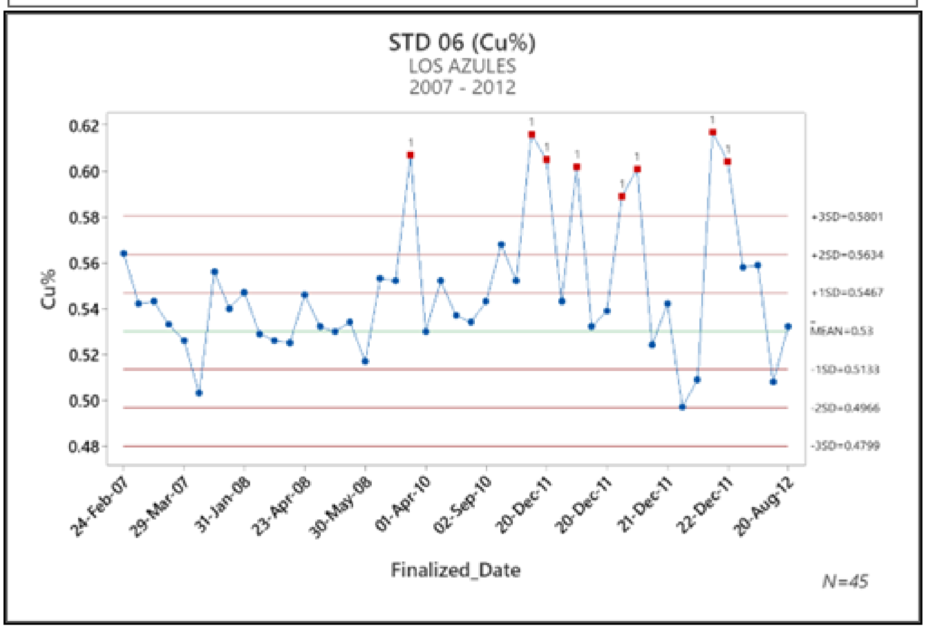
McEwen consistently and routinely monitors the QC results as they are received from the lab and not at the end of a drilling campaign, to ensure that outliers are caught and fixed as soon as possible. Obvious outliers or a collection of outliers in a batch or distinct period will be flagged for re-assaying whereby a bracket of 10 samples either side of the standard will be resubmitted for repeat assays until the QC results are satisfactory.

Examples of diagnostic charts are provided in Figure 11.8. In the charts, the expected assay value appears as a green horizontal line (middle line). Red lines either side of the expected value lines indicate +/-1,2 or 3 standard deviations (SD) away from the expected value with outliers being considered outside of these ranges.






STD 03 2010 (Cu%)
LOS AZULES
2011 - 2022
Cu%
Finalized_Date
+3SD=0.302
+2SD=0.294
+1SD=0.286
MEAN=0.278
-1SD=0.27
-2SD=0.262
-3SD=0.254
22-Dec-11
20-Aug-12
22-Jan-13
04-Nov-13
21-Mar-17
04-May-17
20-Jan-22
22-Apr-22
22-Jul-22
N=263


STD 06 (Cu%)
LOS AZULES
2007 - 2012
Cu%
Finalized_Date
+3SD=0.5801
+2SD=0.5634
+1SD=0.5467
MEAN=0.53
-1SD=0.5133
-2SD=0.4966
-3SD=0.4799
24-Feb-07
29-Mar-07
31-Jan-08
23-Apr-08
30-May-08
01-Apr-10
02-Sep-10
20-Dec-11
20-Dec-11
21-Dec-11
22-Dec-11
20-Aug-12
N=45

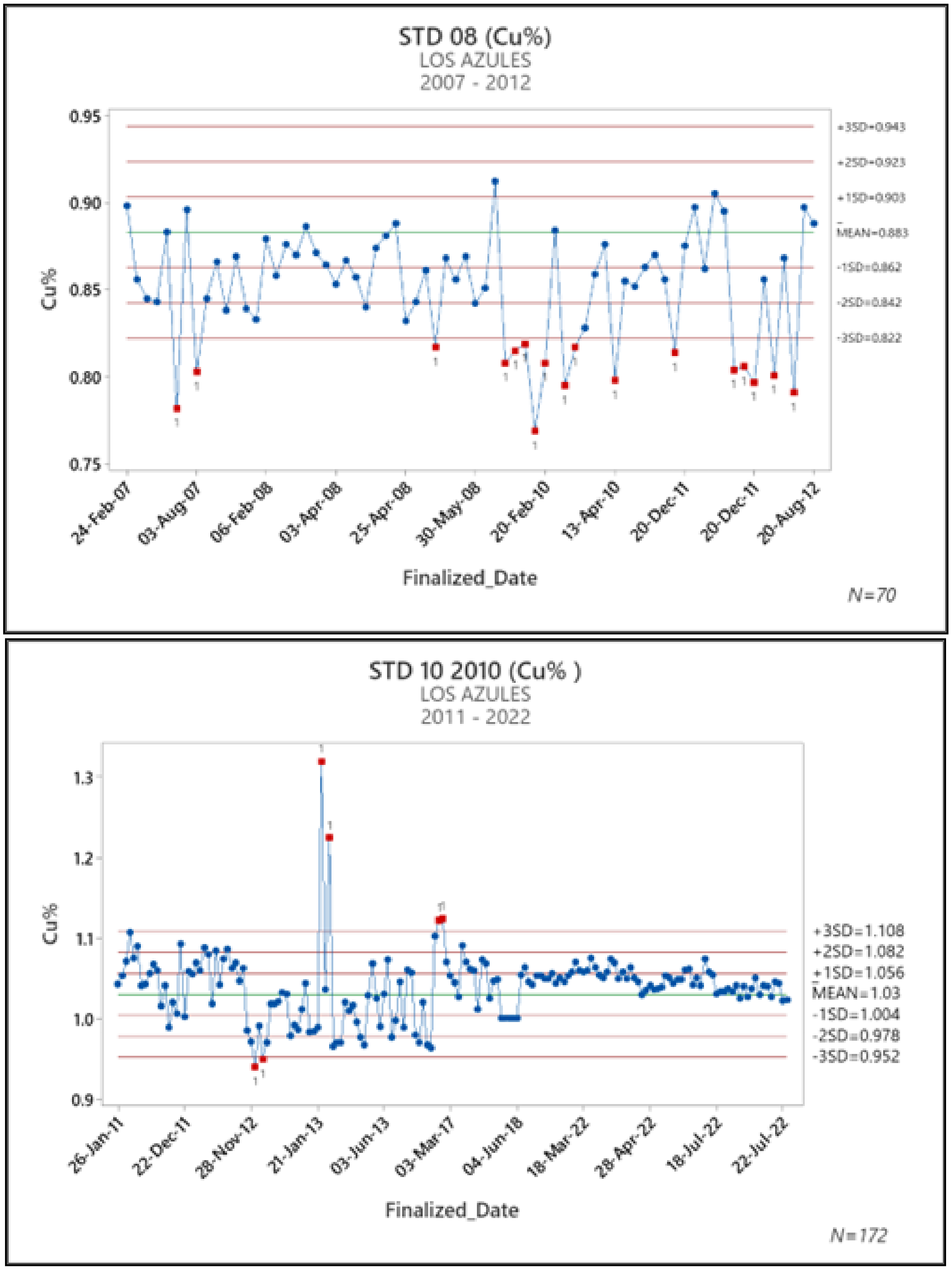


Figure 11.8: Diagnostic charts for standards used at Los Azules 2007-2022

11.4.2.1 Copper

The performance of copper standards was similar across the drilling from 2007 through the 2017 field seasons with exceptions as outlined. Minor variations in the different lab preparation and assaying methodology likely played a part in the erratic nature of the earlier campaigns, however results that "bounce" around the expected values is normal and expected. All outlier values were sent for re-assay.

In the 2009-2010 field season, hole 1049 produced significant QC errors which were addressed by remedial assaying for that hole. In the 2010-2011 field season standards indicated copper values were consistently higher than expected. The errors were addressed by a program of re-assaying in 2012. Original values in the database were replaced by the 2012 re-assay results which were validated by control values. The 2011-2012, 2012-2013 and 2017 field seasons assaying produced no significant QC errors. The 2018 and recent 2022 campaign show a good trace of stable standard results for STD03 and STD10 with no errors seen.

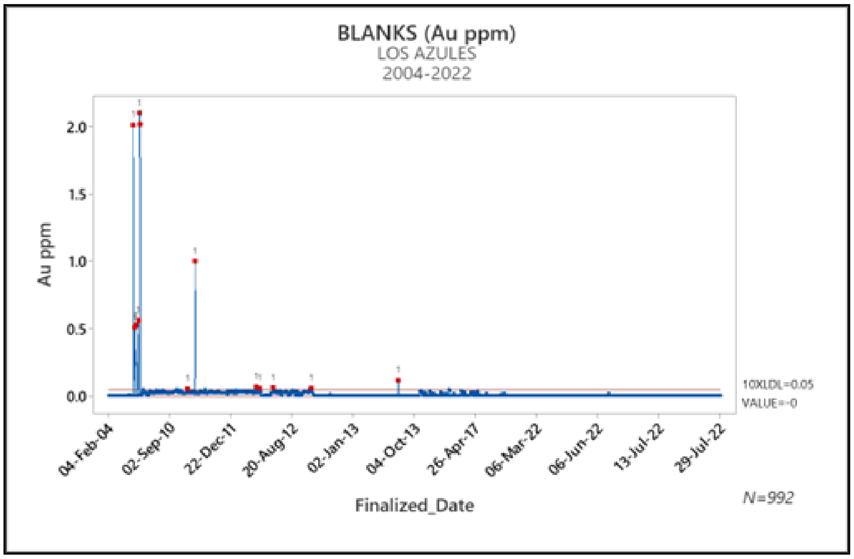
11.4.2.2 Gold

Due to the low values of gold, control using standards of comparable values is not possible due to the lack of precision in the assay process; however, duplicates show no indication of systematic assay problems in gold.

11.4.3 Blank Sample Performance

In the field seasons prior to 2009, the blank material was discovered to be mineralized. This generated a significant number of false positive results. All out of control results during this period were subjected to remedial procedures. No evidence of contamination from sample to sample was detected by the remedial work.

In the 2009-2010 field season, the blank was replaced with a commercial silica sand and continued to be used until 2018. There were a handful of small blank failures in the assaying, later mitigated by re-assaying of the batch. In the 2021/22 drilling season, a commercially sourced coarse quartz material replaced the previous blanks. There have been no blank failures for copper or gold since 2013 (Figure 11.9).



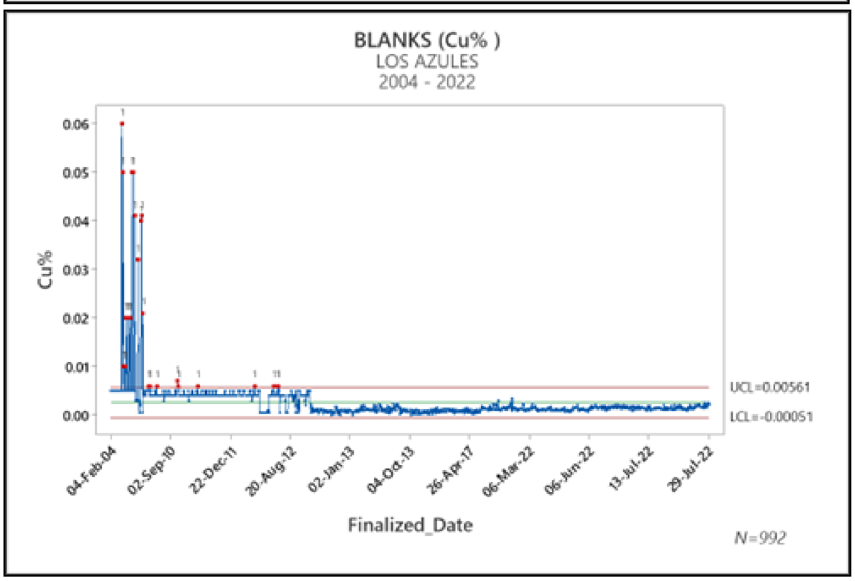
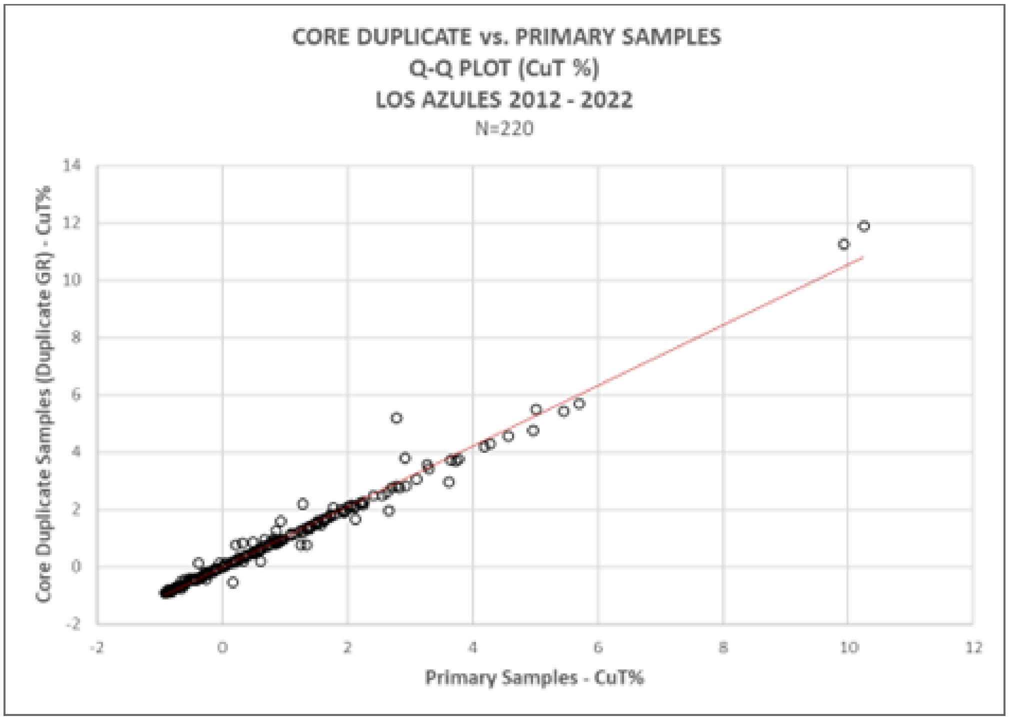


Figure 11.9: Control charts for the blanks at Los Azules 2004-2022

11.4.4 Duplicate Sample Performance

Duplicate samples of coarse reject material are assayed to check the sample preparation protocol. If the protocol is adequate, 90% of the duplicate pairs of assays should fall within ±30% of each other. During all field seasons, coarse reject copper duplicates fell within control limits (Figure 11.10).



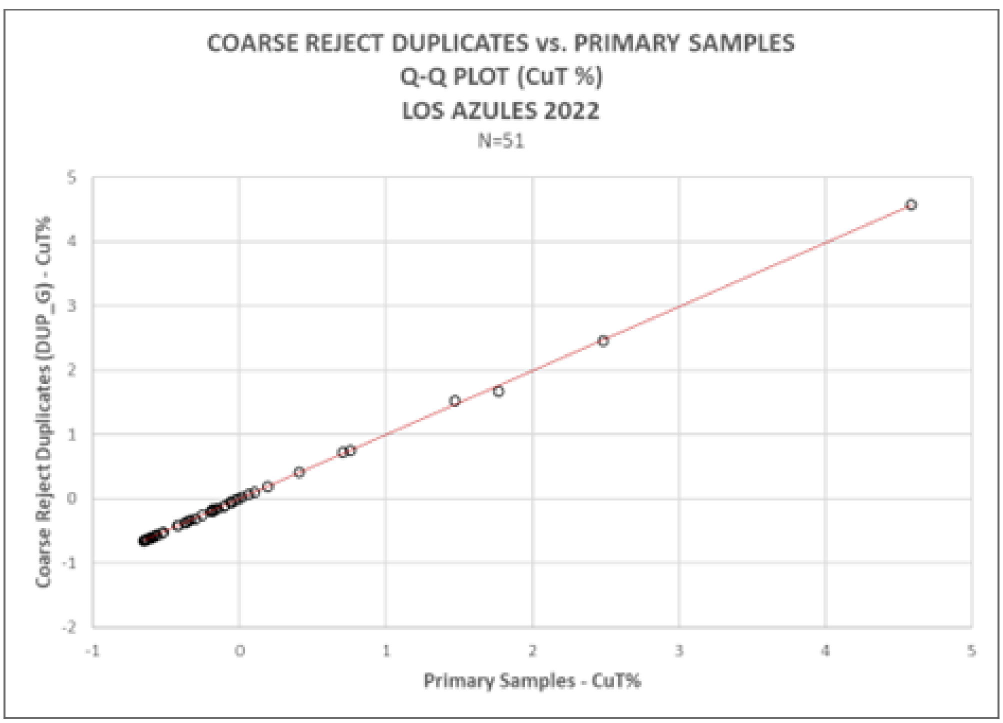


Figure 11.10: QQ plots for core duplicates 2012-2022

11.4.5 Pulp Duplicate Sample Performance

Duplicate samples of pulp material are assayed as another check on assay accuracy and precision. For all seasons where duplicates were taken, copper duplicates from pulp material fell within control limits above the prescribed rate of 90% within ±10%. Differences with gold duplicates in 2009 – 2010 have been addressed. There are no other outstanding issues with pulp duplicate performance (Figure 11.11).

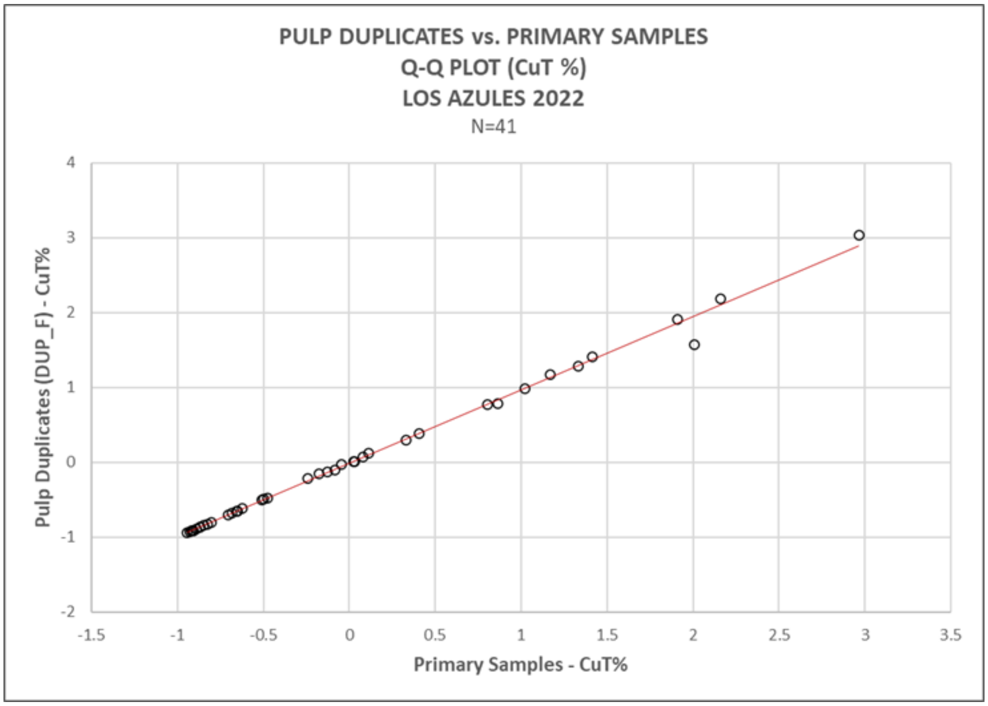


Figure 11.11: QQ plot for the pulp duplicates in the 2022 campaign

11.5 CONCLUSIONS

Results from the control sample analysis indicate that the copper and gold assay processes are under sufficient control to produce reliable sample assay data for resource estimation and release of drill hole assay results. Inadequate standards from early field seasons were eliminated. The use of only one lab to produce assay results improves the consistency of results. Material that was assumed to be blank but contained low copper values was replaced. Later types of blank material improved the monitoring of potential contamination of the samples.

All past deficiencies in the QC program have been addressed. The Los Azules sampling and assaying program is producing sample information that meets industry standards for copper and gold accuracy and reliability. The assay results are sufficiently accurate and precise for use in resource estimation and the release of drill hole results on a hole-by-hole basis.

12.0 DATA VERIFICATION

During April 25 – May 05, 2022, the QP conducted a site visit in Argentina and met with the resource estimation team in San Francisco. The purpose of the visits was to observe, review and comment on all aspects of data collection, recording and analysis in preparation of the MRE. Several project related areas were visited and are summarized below.

12.1 DRILL SITE INSPECTION, LOS AZULES PROJECT SITE

The Los Azules deposit is located at an elevation of 3,700 m in the Andes mountains.

The QP made several trips to the project drilling area to view activities at various drill sites. Rigs at both Major Drilling and Foraco platforms were observed. At each rig, at least one complete cycle was observed with core extracted from the core barrel, core barrel put back in the hole, advance the hole about three meters, pull the core barrel out again and empty core onto tube for transfer to a core box.

Each site has a McEwen Copper employee acting as a Drill Hole Coordinator. He or she is responsible for monitoring the drilling, measuring the full core barrel for core recovery upon extraction, and placing the core in the core box. They label and number each core box with hole number and drill intervals and prepare the box for transport to the logging area at camp.

Much of the core coming out of the core barrel is highly fractured and loose. Care was taken to keep the core as intact as possible with little mixing of the pieces as they were moved. Figure 12.1 is a photo of the Drill Hole Coordinator carefully transferring the highly fractured core to core box. The box is then labelled and sealed and transported to the logging area.

Figure 12.1: Transferring Core to Core Box

Considerable care is taken to ensure drill collar locations and drill site preparations are completed on a timely basis. A series of three surveyed and flagged stakes were set up in groups to indicate the azimuth of the proposed hole. The actual collar location is marked with a flagged and labelled stake that shows the hole number, azimuth, and dip. Figure 12.2 shows a drill pad prepared for a drill rig to move onto soon.

Figure 12.2: Drill Pad Preparation

While on site the QP checked several drill collars with a hand-held GPS. The coordinates were compared to data in the project's drilling database. All holes checked showed agreement within the margin of error of the instrument being used.

12.2 CORE LOGGING COMPOUND, LOS AZULES PROJECT SITE

Drill core is delivered directly from the drill site and sorted and stacked by drill hole. Once the complete hole is delivered, it is laid-out on tables for logging. A team of well-experienced senior geologists log core for lithology, mineralogy and structure and select sample intervals for assaying and bulk density determinations. Concurrently, geological technicians log the hole for rock quality designations (RQD) and measure fracture angles. Core is photographed at site before sampling and splitting.

A technician also conducts Schmidt Hammer determinations at one-meter intervals and point load testing (PLT) at regular intervals. Core is scribed with a line along the long axis where appropriate. Technicians then either cut larger pieces in half with a hydraulic splitter or select half of the fractured core within the defined interval and place the rock and sample tag in prelabelled bags. Up to ten of these plastic bags are placed in a large mesh bag, labeled inside and out, sealed, and placed in a locked secure container awaiting transport to the assay lab.

12.3 CORE WAREHOUSE, CALINGASTA

A visit was made to the Calingasta core storage facility for a day to observe and review activities at that location. The facility is in the foothills of the Andes mountains at an elevation of 1500 m. Core is delivered to the facility after it has been logged, split, and sampled by geologists and technicians at the Los Azules project site. Core is immediately sorted, coded, and labelled and stored indoors in well-made storage racks. Pulps and rejects are stored in dedicated buildings at the warehouse compound. The use of hyperspectral scanners and high-resolution photography was reviewed. Figure 12.3 is a photo of the core storage racks at the Calingasta compound.

Figure 12.3: Core storage racks at Calingasta

Hole AZ22142 was selected for review during which the core logging protocol, sample selection, sample labelling for assaying and bulk density testing, common mineralogical and lithologic controls were discussed.

12.4 ALEX STEWART ASSAY LAB, MENDOZA

A visit to Alex Stewart Inc (ASI) labs took place to follow the routing of incoming samples through drying, sample prep, analysis, reporting and included discussions of internal QA/QC practices, gold assaying procedures and detection limits used by ASI, the protocol for sequential copper assays, bulk density protocols and reporting limits, and the use of a new high temperature atomic absorption (AA) unit. ASI is an International Organization for Standardization (ISO) certified lab (17025). The secure outside storage area (walled, fenced, and gated) for incoming samples was visited and sample security and chain of custody procedures were reviewed.

12.5 GLOBAL DATABASE MANAGER, DATABASE CURATOR & EXPLORATION MANAGER, SAN JUAN

Discussions with the Global Database Manager, Database Curator and Exploration Manager included: the multiple historical interpretations seen in the logging; the presentation of drilling results, the effect of Dr. Sillitoe's 2014 site visit and adjustments made to interpretive work thereafter. Additionally, discussions on the historical use of multiple labs for assay analysis, historical and current QA/QC protocols including a review of recent results, the preliminary results of an initial database audit by MTS were reviewed. The ongoing topographic survey with discussion of various national grid systems used were described.

12.6 CRM (RESOURCE ESTIMATION), SAN FRANCISCO

CRM is the consultant modelling team based in Santiago, Chile, and San Francisco USA chosen by McEwen Copper to complete the Mineral Resource Estimate (MRE). Primary discussions involved the modelling work completed to-date and included data validation of new data provided for the modelling process, interpretive steps and methods used to construct grade estimation boundaries, re-evaluation of mineral zone contacts using sequential assay results, and the use of satellite mapping for fault traces. There was discussion of the use of implicit modelling for interpretive work, discussion on density measurements and gold assay detection limits and methods used to cap high grade outliers in the drilling database and associated composite file.

12.7 MINE TECHNICAL SERVICES (MTS) DATABASE AUDITS

MTS conducted two phases of audits (2021 and 2022) including a site visit by Todd Wakefield and Francisco Ramos between April 18 – 27, 2022. Discussions of their findings and a review of their recommendations were made. Stantec's geological QP supported McEwen Copper's decision to undertake an extensive re-assaying program of existing core to augment the database prior to the updated estimate.

12.8 GEOLOGICAL MODELLING

The drill hole database utilized in the geological modelling and subsequent mineral resource estimation consists of historical information and the drill hole data collected by the Issuer.

The historical drill data were presented as a series of .csv data tables which were imported into an MS Access database for review and verification. The drill hole data was then loaded into Leapfrog Geo software which provides standard checks for drill data integrity, considering a sequential 'from' and 'to' for the interval tables, the end of hole detailed in the collar table matching those in the interval tables, duplication of data, and checks for erroneous readings in survey deviation. There were no errors found in the historical drill hole data tables.

The drill hole data from the current drilling campaign was captured on site using Excel data sheets which were then validated upon import to a Fusion X SQL data management system ensuring the integrity of data captured during the logging process. The assay data certificates were loaded directly into the database, importing the information contained therein in its entirety without any manual editing. The database manages the assay data, using a series of rules and profiles designed to export the assay data from the optimum analytical method and converting any below detection limit values to numeric data. The data tables, Collar, Survey, Lithology, Assay, Alteration, Mineralization, and Structure were extracted as a series of .csv files with their table structure set up for direct import in the Leapfrog Geo modelling platform. These data were combined with the historical data, checked for drill hole data integrity, and promoted as error free data used in the geological modelling.

The results of the data verification indicate that the database is sound and reliable for the purposes of resource estimation.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Copper mineralization is complex and varied at Los Azules, as to be expected for deposits of this type. The potential resource processing sources include Oxide/LIX, Supergene and Primary mineralized Zones. LIX mineralization is the Spanish acronym for "leached" or leach cap. Primary or hypogene copper mineralization extends to at least 1,000 m below the present surface at Los Azules. Near surface, leached primary sulfides (mainly pyrite and chalcopyrite) were redeposited below the water table in a sub-horizontal zone of supergene enrichment as secondary chalcocite and covellite. Very little oxide copper mineralization is present in the deposit and found in the Oxide/LIX and Supergene enrichment Zones. Hypogene (Primary Zone) bornite appears at deeper levels together with chalcopyrite. Gold, silver, and molybdenum are present in trace amounts, but copper is by far the most important economic constituent at Los Azules.

This report update leverages the prior metallurgical work completed for the Los Azules project and reported in prior Technical Reports. Historical testing for McEwen Copper (McEwen) was conducted on samples from the resource in several phases. C. H. Plenge Laboratory (Plenge) in Lima, Peru, performed several scoping level investigations from 2008 to 2012 to support a Preliminary Economic Assessment (PEA) by Samuel Engineering in 2009, 2010, and 2013. Additional samples from the resource were tested at the SGS Research Limited (SGS) to support a Preliminary Economic Assessment (PEA) by Hatch in 2017. A mineral liberation analysis (MLA) was completed at Thompson Creek Metals Company in Challis, Idaho; in 2012 on rougher flotation samples from the Plenge lock-cycle testing.

The historical work completed at both Plenge and SGS concentrated on evaluating sulfide resource processing options including flotation, pressure oxidation (POX) of flotation concentrate, and column leaching. The evaluation of the historical data in the PEA in 2009 and 2010 resulted in the selection of a flotation process to produce a copper concentrate. In 2013, a change in the PEA concepts resulted in a flotation concentrate being treated by a POX leach circuit and solvent extraction / electrowinning (SX/EW) to produce copper metal cathodes.

Metallurgical characterization testing has been completed as part of this study in the form of sequential assay (sulfuric acid and cyanide component steps) for the resources considered, column testing and bottle roll testing. The sequential assay method used at Los Azules for both the resource assay and metallurgical programs provides an indication of the copper mineralization present in the form of acid soluble copper (CuAS) and cyanide soluble copper (CuCN), both assays combined provide an approximation for leachable/soluble copper (CuSOL) component of the total copper assay (CuT).

There are several sequential assay methods in use in the industry, the methodology selected for Los Azules maintains consistency with historic assay methods included with the resource assay data set. Re-assaying of older pulps and core samples was completed; however, results did not provide sufficient confidence in the updated sequential assaying information obtained for use in the resource models.

The current metallurgical program is developed in three (3) phases based on sample availability to support this PEA and continuing in parallel to support future study and objectives. The Phase 1 program is complete and pending some final analysis at the time of this report and considered for the PEA analysis along with the historical information. The Phase 2 and 3 programs are started, and metallurgical sampling and sample preparation is in progress.

Existing drill core was selected by lithology and material type to reflect economically processable material in the resource for this study. Assay data and bottle roll testing was completed for this study from the 21 column test samples (from nine (9) composites and two (2) drill holes) are currently under acidic and bio-leach conditions. This reflects the Phase 1 metallurgical test program currently being completed at SGS.

Additionally, 90 kg samples from Composite 6, Composite 7, Composite 8, and Composite 9 (for a total of 360 kg) from this program were shipped and delivered to Rio Tinto's facility in Bundoora in early November 2022 to undertake alternative leaching techniques.

Additionally, over 4.5 tonnes of PQ/HQ core material from the 2022 drilling season were shipped to Hazen Research in Golden, Colorado in September 2022. Nuton has set up a laboratory facility at Hazen, mirroring the technology and practices from the Bundoora facility near Melbourne.

As part of the Phase 1 program, samples were shipped to SEPRO laboratory in Canada to undergo microwave assisted comminution; to evaluate thermal stresses creating fractures (macro and micro).

A total of 9 drill holes over 4,217 m of PQ core (41,689 kg) were drilled during the 2022 campaign to support the Phase 2 Metallurgical test program at SGS; these samples were delivered in mid-October 2022. Additionally, samples from the 2022 drilling season of PQ core were sent to Hazen Laboratory in Golden, Colorado to undertake alternative leaching techniques, starting in Q1 of 2023.

Sampling for a Phase 3 metallurgical test program is currently under way, sampling PQ core to develop a large bulk sample to support 305 mm diameter by 9 m height columns for support of a feasibility.

Based on typical recovery estimates for CuAS and CuCN as provided by a standard sequential copper assay methodology developed at the Plenge Laboratory facility in Lima, Peru, projected copper recovery estimates have been derived based on leachable/soluble (CuSOL) copper content that will be validated in the ongoing column testing program at SGS, Santiago. Information in this section of the report has been updated and received as of 05/01/2023.

The metallurgical work completed to date and ongoing at the effective date of this report provides an adequate understanding of the expected performance characteristics for a PEA level of analysis. For Los Azules the anticipated copper extraction of the CuSOL fraction of the assayed copper in each block is 100%. Additionally, approximately 15% of the residual copper component can also be extracted based on the metallurgical results to date. Copper recovered to cathodes also considers a heap efficiency and inventory factor of 90% of the extractable copper based on general experience.

Based on the resource assay data and column results, the apparent soluble copper recovery to cathodes is approximately 107%, with the overall LOM total copper recovery expected for the initial project is approximately 73% as calculated for the components. Soluble copper recovery exceeding 100% implies partial leaching of material which was not categorized as "soluble" based on the sequential assaying method and data available. Given the implied recovery results, some adjustment to the sequential assay methodology could be considered to approximate the actual column and eventual heap leaching results better directly.

Copper recovery is also distributed over a two-year timeframe (60% of expected production in year 1 and 40% in year 2) to achieve at least two (2), 180 day leaching cycles in the multi-layer stacked pad design from the time of initial placement on the pad.

The current Phase 1 project engineering design uses a sulfuric acid copper bio-heap leach process to produce a Base Case 175,000 tonne/annum (tpa) copper cathodes by SX/EW or in an Alternative Case 125,000 tpa considering the LIX and Supergene material types. This process design is described in Section 17 and was used to estimate capital and operating costs for the economic assessments of the Project.

13.2 REVIEW OF HISTORICAL METALLURGICAL TEST WORK

13.2.1 Plenge 2008 Historical

Three composites and 16 drill core samples were sent to Plenge for the metallurgical test work (Los Azules Copper Project Metallurgical Investigation No. 6976-6991 / 7026-7027 and 7028) indicated the flotation response on the samples tested are typical of material seen in porphyry deposits for the recovery of copper, gold, and silver. Composite 1 is a Supergene sulfide composite with 17% of the copper in chalcopyrite, while Composite 2 is a Primary sulfide composite with 49% of the copper in chalcopyrite. Composite 3 included a small amount of high-grade sample of Primary sulfide with 75% of the copper in chalcopyrite was also sent to Plenge for metallurgical test work and it was confirmed that the high-grade sample responded well to copper recovery by flotation. Composites 1 and 2 included work on bond work index (BWi), flotation lock-cycle testing, reagent screening, and sulfuric acid bottle rolls. The 16 drill hole samples were utilized for variability flotation over a variety of Primary grind sizes. Composite 3 included work on flotation lock-cycle testing, reagent screening, and cleaner pyrite suppression.

13.2.1.1 Bottle Roll Results

The tests were performed at 10 and 50 g/L sulfuric acid (H_2SO_4) concentration and the acid consumption during the tests indicated a reasonable sensitivity to the acid concentration.

Table 13.1: 2008 Bottle Roll Results

Composite	H_2SO_4	H_2SO_4	Head (%)		Residue (%)		Extraction (%)	
	g/L	kg/t	Copper	Iron	Copper	Iron	Copper	Iron
1	10	18.2	0.843	1.9	0.843	1.8	31.3	5.3
1	50	29.6	0.85	1.9	0.850	1.8	36.7	8.8
2	10	21.6	0.568	2	0.496	1.9	14.3	7.5
2	50	45.9	0.557	1.9	0.468	1.7	18.0	12.7

13.2.1.2 Grinding

Bond Work Index (BWi) were determined for Composites 1 and 2 by Plenge to be 12.5 kW-hr/t and 13.7 kW-hr/t. The values for this work index suggest a mineralized material of medium hardness.

13.2.1.3 Flotation – Baseline Variability

Kinetic reagent selection flotation tests were first performed on composites No. 1 and 2 with eight different collectors to settle on Primary collector A-3477 and secondary collector Z-14 for further testing with varying grind size and pH. Kinetic reagent selection flotation tests were also performed on composite No. 3. The collectors found to be the most effective and used for further testing were C-4920 as a Primary collector and Z-14 as a secondary.

A set of four kinetic reagent flotation tests were performed on Composites 1 and 2 to determine a Primary grind size. Results indicate that the copper and gold recoveries decrease with an increase in grind size; therefore, 80% passing product size (P80) of 125 microns ("µm") was chosen. Grind size optimization testing was done on Composite 3, and it was also found that recovery decreases with an increase of grind size. Additional tests were performed to determine the optimal rougher pH. Composite 1 responded well at a pH of 9, while Composite 2 yielded improved recoveries at a pH of 10. Composite 3 received the best results with a pH of 11. Since Composite 2 is the dominant mineralized material type, a pH of 10 will be used in the rougher.

Through process optimization testing it was discovered that the composites require a regrind between 37 µm and 25 µm; therefore, a nominal regrind P_{80} of 30 µm was chosen.

Various depressants were tested for pyrite depression, and it was noted that cyanide and sodium bisulfite reagents had no effect on controlling pyrite.

The variability tests were based against results for rougher flotation.

Lock-cycle tests were performed on each of the two composites and Sample 3. The results of the locked-cycle tests are shown in Table 13.2.

Table 13.2: Lock-cycle Test work Results						
	Concentrate Assay			Metal Recoveries (%)		
Composite	Copper (%)	Silver (g/t)	Gold (g/t)	Copper	Silver	Gold
1	34.87	101.4	2.69	94.1	70.5	56.0
2	30.98	80.3	3.86	94.7	61.9	66.5
3	34.91	85.6	2.05	95.2	82.9	73.3
3[1]	33.88	84.0	2.37	95.2	83.5	74.2

1 - Pyrite Depression

13.2.2 Plenge 2010 Historical

In 2010, three separate composites compared to the 2008 work were sent to Plenge for the metallurgical test work (Los Azules Copper Project Metallurgical Investigation No. 6976-6991 / 7026-7027 and 7028). For additional information on variable sulfuric acid bottle roll testing, 15 additional tests were completed in 2010 (Los Azules Copper Project Metallurgical Investigation 7652-54), the results can be found in Table 13.3. The copper recovery from the porphyry sample ranged from 40% to 50% of the total copper depending on the testing conditions. The copper recovery from two breccia samples was lower, i.e., between 18% and 40%. The results were in the ranges expected based on the mineralogy and the sequential copper analyses.

Table 13.3: 2010 Bottle Roll Results								
Composite	Test	H_2SO_4	Fe	Head	Residue	Recovery	GAC[1]	NAC[2]
		kg/t	g/L	%	%	%	%	kg/t
1	1	10	0	0.22	39.2	39.2	32.7	16.3
1	2A	10	2	0.21	51.6	51.6	31.5	14.1
1	2B	10	2	0.21	48.3	48.3	31.5	13.7
1	3	50	0	0.21	44.3	44.3	119.8	35.5
1	4	50	2	0.22	50.1	50.1	117.9	30.8
2	1	10	0	0.23	17.9	17.9	29.8	12
2	2	10	2	0.22	23.1	23.1	28.6	9.8
2	3A	50	0	0.22	20.6	20.6	114	26

Table 13.3: 2010 Bottle Roll Results								
Composite	Test	H_2SO_4	Fe	Head	Residue	Recovery	GAC[1]	NAC[2]
		kg/t	g/L	%	%	%	%	kg/t
2	3B	50	0	0.24	19.5	19.5	115	25.1
2	4	50	2	0.23	22.9	22.9	111.9	17.3
3	1	10	0	0.14	30.4	30.4	29.4	12.4
3	2	10	2	0.16	37.6	37.6	29.3	11.2
3	3	50	0	0.14	36.2	36.2	118	25.3
3	4A	50	2	0.15	40.7	40.7	112	22.1
3	4B	50	2	0.15	40	40	112	20.2

1 – GAC = Gross Acid Consumption
2 – NAC – Net Acid Consumption

13.2.3 Plenge 2011-2012 Historical

Plenge received 1,745 kg of drill core material in 2011 for metallurgical test work (Los Azules Copper Project Metallurgical Investigation No. 9247-69.) This was used in additional test work conducted to further evaluate the flotation response of the sulfide material and sulfuric acid leaching of LIX low-grade sulfide materials. Additionally, flotation optimization and variability tests were carried out on the material, followed by a confirmation testing through lock-cycle. Samples were split into Supergene and Primary material types. Mineralogy was completed by Thompson Creek Metals Company on rougher flotation tailings, cleaner scavenger flotation tailings, cleaner flotation tailings, and a concentrates sample for both Supergene and Primary.

Low grade sulfide material from the Oxide/LIX material type had sulfuric acid bottle rolls and column leach tests for the potential recovery of copper by heap leach. Produced flotation concentrate from Primary and Supergene materials were sent to a POX circuit to be oxidized and copper extraction evaluated.

13.2.3.1 Bottle Roll and Column Leaching Tests

The most recent test work at Plenge was performed on two composites, one which was relatively high grade and the other which was relatively low grade. A total of 16 bottle roll tests and 11 column leach tests were run to determine copper recovery and acid consumption. The 16 samples were directly from the Oxide/LIX zone and considered low grade when compared to the total resource.

The average copper recovery was 53.4% for the eight Oxide/LIX high grade bottle roll tests (Table 13.4) and 40.9% for the eight Oxide/LIX low grade bottle roll tests (Table 13.5). There was a good correlation of copper extraction with acid and ferric iron (Fe^{3+}) concentrations in the leach solutions. Acid consumption was directly related to the acid concentration in the leach solution.

Table 13.4: Oxide/LIX High Grade Bottle Roll Results							
Test Number	H_2SO_4	Ferric	CuT Head	CuT Residue	CuT Extraction	H_2SO_4	GAC
	g/L	g/L	%	%	%	kg/ t	kg/ t
9207 - 1	10	0	0.16	0.09	45.0	11.6	10.5
9207 - 2	10	5	0.16	0.08	50.8	11.3	10.0

Table 13.4: Oxide/LIX High Grade Bottle Roll Results							
Test Number	H_2SO_4	Ferric	CuT Head	CuT Residue	CuT Extraction	H_2SO_4	GAC
	g/L	g/L	%	%	%	kg/ t	kg/ t
9207 - 3	30	0	0.16	0.08	52.9	24.7	23.4
9207 - 4	30	2	0.16	0.07	54.9	54.2	52.9
9207 - 5	30	2	0.16	0.08	54.5	54.8	53.5
9207 - 6	30	5	0.16	0.07	56.0	49.0	47.6
9207 - 7	50	2	0.16	0.07	56.4	100.5	99.0
9207 - 8	50	5	0.16	0.07	56.9	94.5	93.1

Table 13.5: Oxide/LIX Low Grade Bottle Roll Results							
Test Number	H_2SO_4	Ferric	CuT Head	CuT Residue	CuT Extraction	H_2SO_4	GAC
	g/L	g/L	%	%	%	kg/ t	kg/ t
9207 - 1	10	2	0.07	0.04	38.1	23.6	23.2
9207 - 2	10	5	0.07	0.04	38.7	20.2	19.8
9207 - 3	10	5	0.07	0.04	38.4	21.3	20.9
9207 - 4	30	0	0.07	0.04	40.1	75.8	75.4
9207 - 5	30	2	0.07	0.04	42.3	69.7	69.2
9207 - 6	30	5	0.07	0.04	42.2	66.7	66.2
9207 - 7	50	0	0.07	0.04	43.0	121.3	120.8
9207 - 8	50	5	0.07	0.04	44.0	117.5	117.1

The five high grade column leach tests showed an average total copper extraction of 81.9% with an average acid consumption of 36.5 kilograms per tonne. The average copper extraction for the six low grade column leach tests was 65% with an average acid consumption of 24.0 kilograms per tonne. For both sets of tests, the copper extraction was enhanced with higher acid and ferric concentrations in the leach solution. Acid consumption was again related to the acid concentration in the leach solution.

The heap leach tests were completed in two separate column sizes was 145 mm diameter by 1 m in height and 145 mm diameter by 3 m in height. Both column sizes were loaded with a crush size of 100% passing ½ in. Each column included a varying degree of free acid concentration or ferric iron (Fe^{3+}) addition.

The material loaded was not agglomerated before being placed under a leach solution. The columns ran in closed circuit with solvent extraction to remove copper for a total of 180 days. The irrigation rates were 25 L/hr-m^2 for the first 5 days and decreased to 10 L/hr-m^2 thereafter. Acid was dosed regularly, being added enough to have the average column pH approximately 1, throughout the 180 days of leaching.

Head assays for the column material are in Table 13.6. The high-grade column data is summarized in Table 13.7 with total copper extraction leach curves in Figure 13.1 and soluble copper extraction leach curves in Figure 13.2. The low-grade column data is summarized in Table 13.8 with total copper extraction leach curves in Figure 13.3 and soluble copper extraction leach curves in Figure 13.4.

Table 13.6: Oxide/LIX - High-Grade and Low-Grade Head Assays		
	Oxide/LIX High Grade Column	**Oxide/LIX Low Grade Column**
Total Copper (CuT)	0.165	0.069
Acid Soluble Copper (CuAS)	0.062	0.020
Cyanide Soluble Copper (CuCN)	0.059	0.014
Total Soluble Copper (CuSOL)	0.121	0.034
Cu Residual (CuRES)	0.040	0.032
Iron (Fe)	2.06	2.09
Sulfur (STOT)	0.44	0.16
Sulfur Sulfide (SS)	0.30	0.01

Table 13.7: Oxide/LIX High Grade Column Results											
Test	Loaded Wt. (kg)	Column Size (mm)	Acid Addition (g/L)	Fe^{3+} Addition (g/L)	Calculated Head (%)	Tail Grade (%)	CuT Extraction (%)	CuSOL Extraction (%)	Gross Acid Consumption (kg/t)	Gangue Acid Consumption (kg/t)	Specific Gangue (t/t Cu)
1	25	145 DIA X 1 m H	10	2	0.163	0.031	80.9	109.0	50.60	48.57	36.83
2	25	145 DIA X 1 m H	10	5	0.170	0.029	83.1	116.8	49.50	47.32	33.48
3	25	145 DIA X 1 m H	30	2	0.165	0.024	85.4	116.5	104.36	102.19	72.52
4	75	145 DIA X 3 m H	10	2	0.166	0.035	78.9	108.2	32.00	29.98	22.91
5	75	145 DIA X 3 m H	10	5	0.167	0.032	80.7	111.4	31.84	29.76	22.09

Table 13.8: Oxide/LIX Low Grade Column Results											
Test	Loaded Wt. (kg)	Column Size (mm)	H_2SO_4 Addition (g/L)	Fe^{3+} Addition (g/L)	Calculated Head (%)	Tail Grade (%)	CuT Extraction (%)	CuSOL Extraction (%)	Gross Acid Consumption (kg/t)	Gangue Acid Consumption (kg/t)	Specific Gangue (t/t Cu)
1	75	145 DIA X 3 m H	10	2	0.068	0.028	59.3	118.7	22.96	22.34	55.37
2	75	145 DIA X 3 m H	10	5	0.069	0.028	59.1	119.9	21.95	21.32	52.30
3	75	145 DIA X 3 m H	20	2	0.069	0.021	70.1	142.2	37.09	36.35	75.20
4	25	145 DIA X 1 m H	10	2	0.070	0.025	63.7	131.2	53.32	52.63	117.96
5	25	145 DIA X 1 m H	10	5	0.070	0.025	64.6	132.9	26.62	25.92	57.34
6	25	145 DIA X 1 m H	20	2	0.071	0.018	74.8	156.2	60.64	59.82	112.67

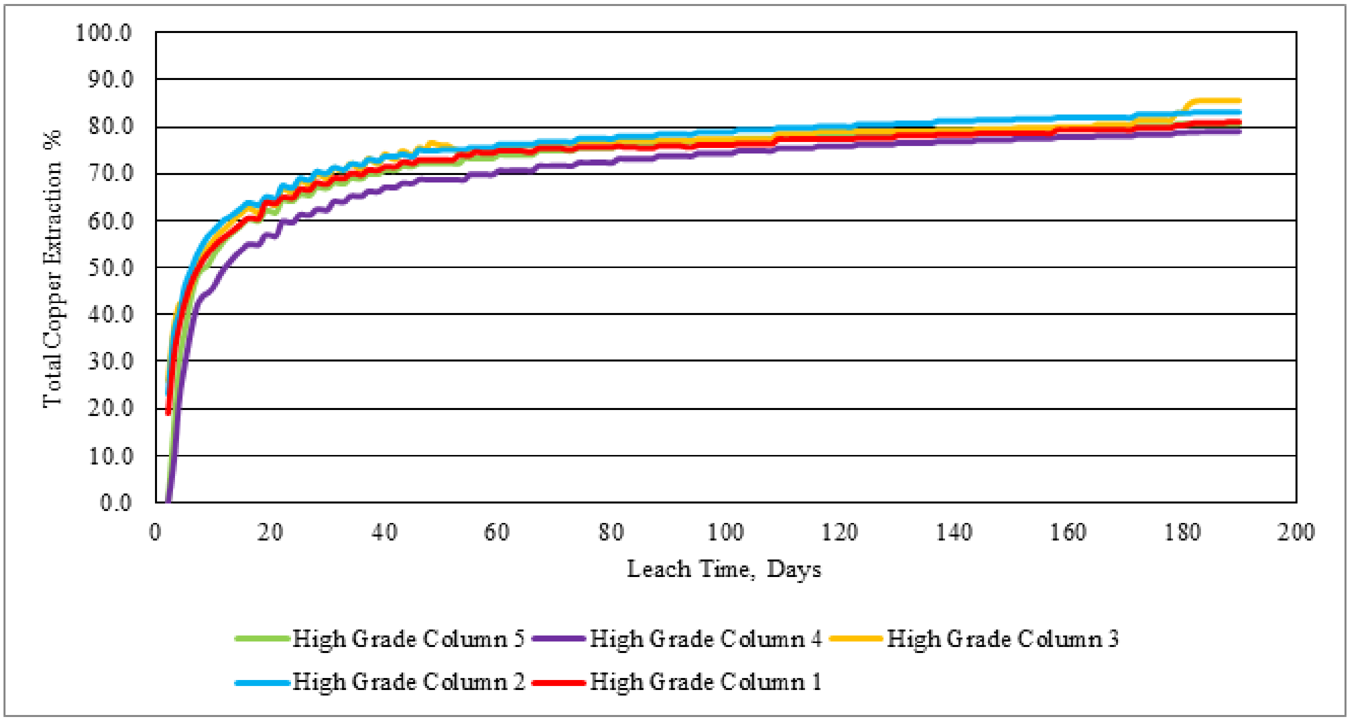


Figure 13.1: Oxide/LIX High-Grade Column Total Copper Extraction Curve

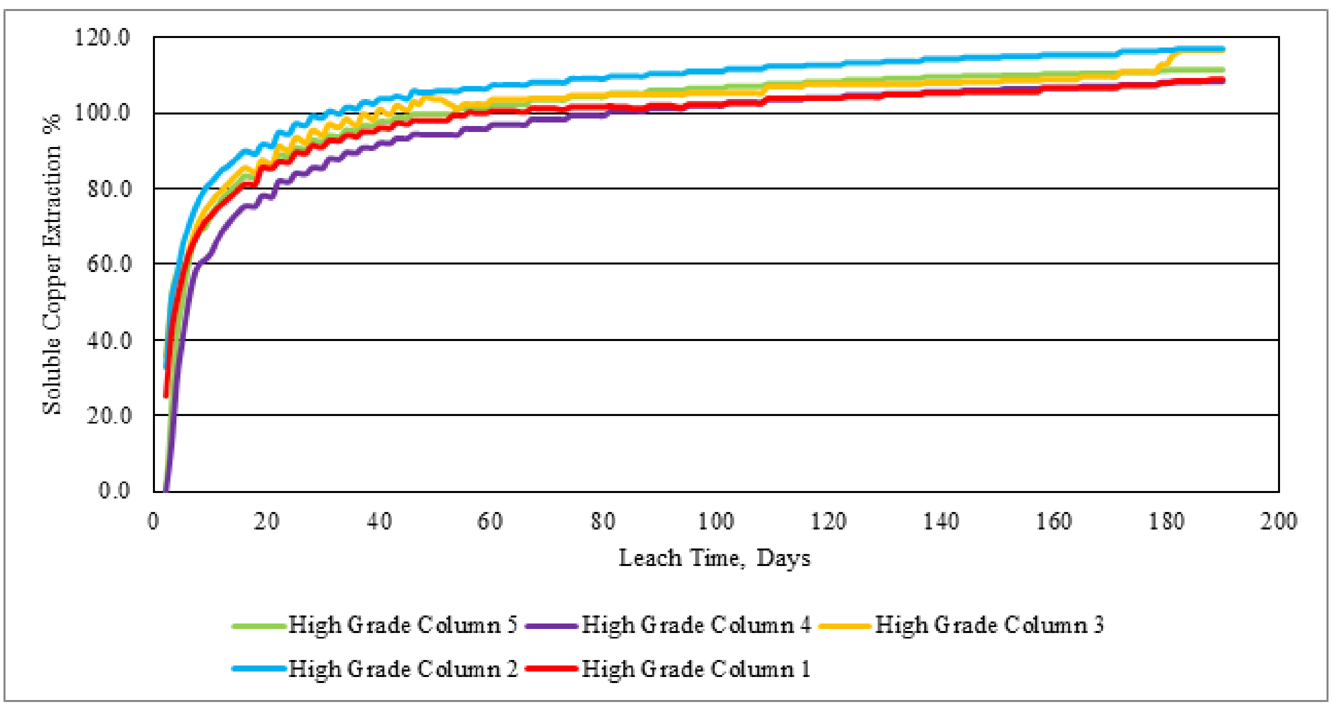


Figure 13.2: Oxide/LIX High-Grade Column Soluble Copper Extraction Curve

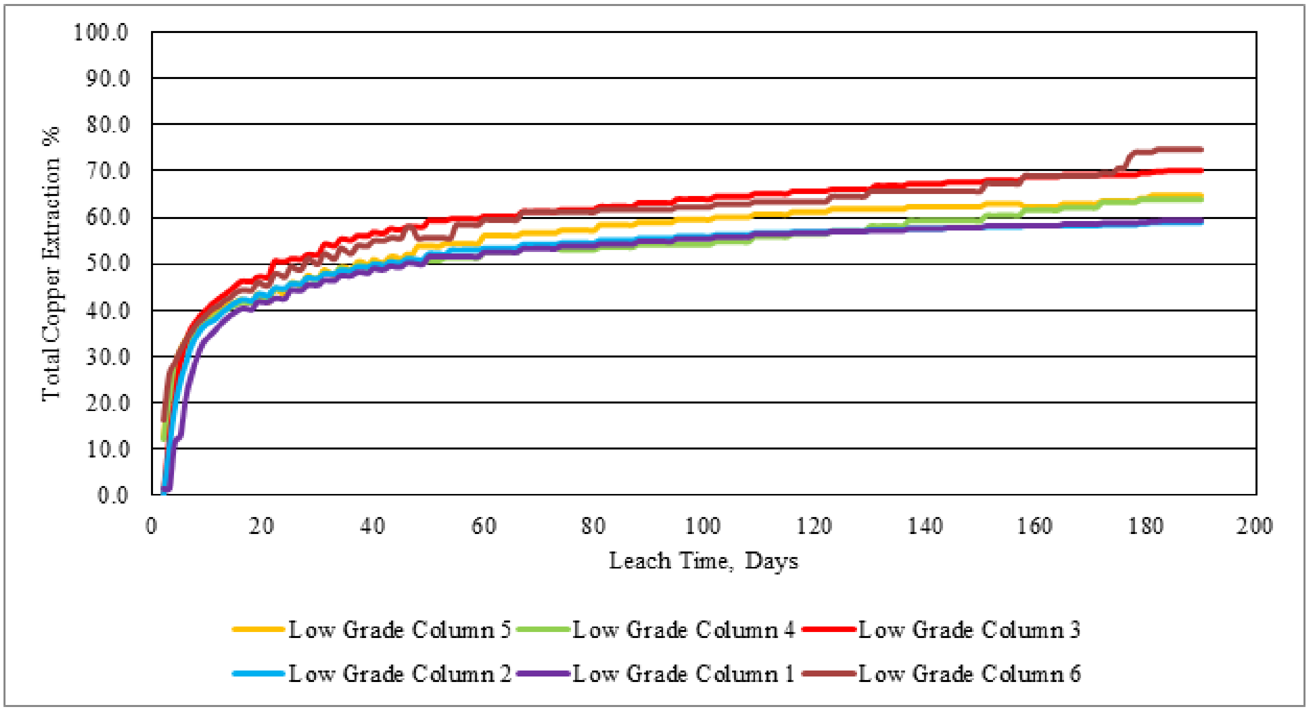


Figure 13.3: Oxide/LIX Low Grade Column Total Copper Extraction Curve

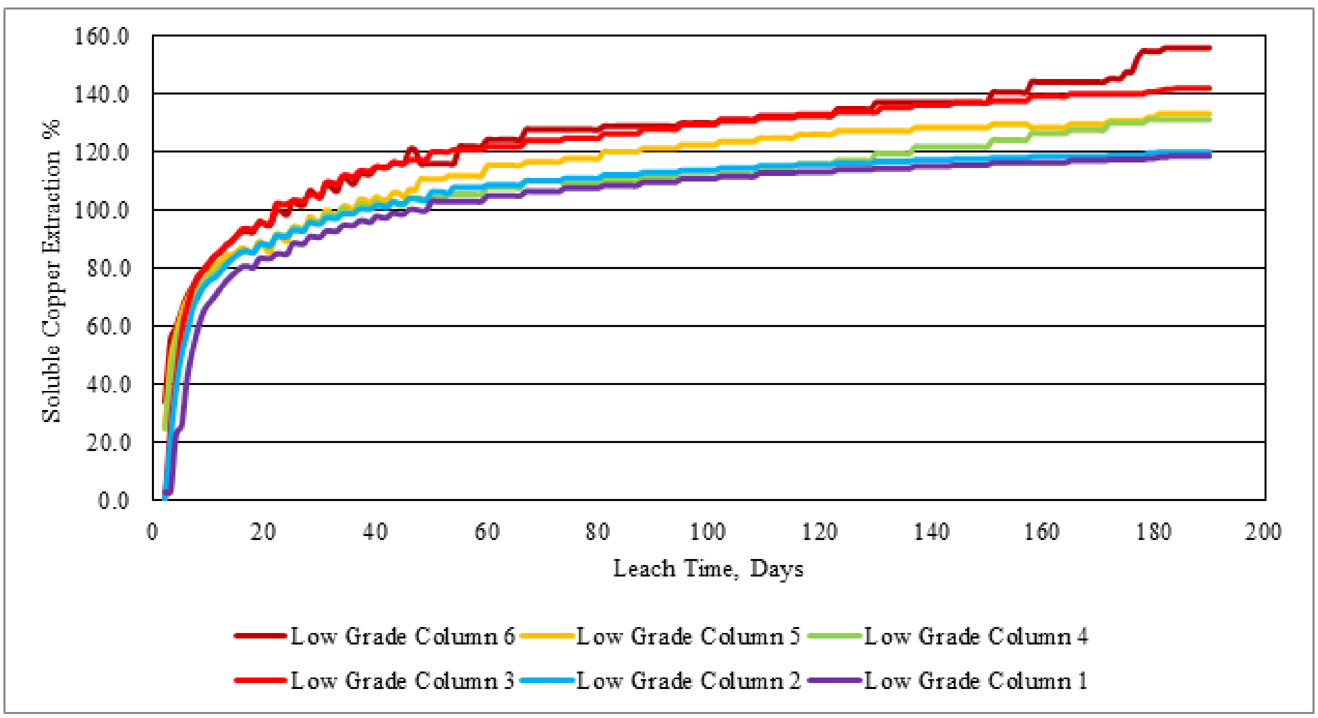


Figure 13.4: Oxide/LIX Low Grade Column Soluble Copper Extraction Curve

The Plenge data and information was only available as PDF files at the time of writing this report. While the original spreadsheets used to calculate total copper recovery were requested, Plenge no longer retained copies of these files. As such, the author has used the PDF to replicate the graphs and produce soluble copper recovery. Unlike Excel files, the PDF copy numbers are only as useful as the significant figures reported.

Higher than 100% recovery on soluble copper suggests the sequential assay method Plenge was employing at the time-of-service under reports the column recoveries achievable. Some copper reported in the residual assay component is apparently recoverable based on the testing results.

After the columns were completed, a tail screen analysis was not completed. Neither was there any humidity cell, nor TCLP 8 RCRA metals (As, Ba, Cd, Cr, Pb, Se, Ag, Hg) analysis of final leach residues.

13.2.3.2 Variability Flotation

Variability test work on the Primary and Supergene samples was carried out in 2011 by Plenge using the flowchart shown in Figure 13.5. The sample data for the variability test work is summarized for the Primary data in Table 13.9 and the Supergene in Table 13.10. The copper grade varied from 0.2% to 0.5% with a single sample at 0.9% copper in the Primary zone and from 0.4% to 1.9% copper in the Supergene zone.

The average rougher recovery of the Primary zone samples was 94.9%, with an average copper grade of 3.7%, while the Supergene rougher recovery was 92.5%, and the copper grade in the rougher concentrate was 5.6%. The enrichment ratios (ERs) for the Primary and the Supergene zones were 8.3 and 7.4 respectively.

When evaluating the flotation response (recovery and ER) of the two zones, the variability test work report showed that the flotation efficiency of the Primary zone sample was better than that of the Supergene zone sample in the rougher stage.

The average of the overall flotation recovery and concentrate copper grade for Primary zone samples was 84.5% and 27.1% Cu, respectively. The test work on the Supergene zone samples shows average overall flotation recovery and concentrate copper grade of 79.8% and 30.7% Cu, respectively.

To have a better understanding of the overall flotation efficiency, the overall ER and recovery for both samples were compared. In this regard, the test work results of the Primary zone samples show 84.5% recovery and a 60.8 ER, whereas for the Supergene material the average recovery is 79.8% and with an ER of 40.4. Therefore, the flotation efficiency of the Primary zone material is better than that of the Supergene material, which is a result of the combination of differences in mineralogy, flotation conditions, and particle size distribution.

Determination of the mineralogy and grade variation effects on the flotation performance is the main aim of a variability test work program. Based on the samples assay data available for the study, the variation of the valuable minerals is not completely defined as only the assays of iron, copper, gold, and silver were reported. It was observed, however, that the predominant copper minerals in the Primary zone are chalcopyrite and bornite with minor amounts of native copper and native iron/copper alloy and in the Supergene zone the same copper minerals plus chalcocite are found.

Flotation conditions were determined through the optimization test work and the optimum flotation conditions so derived were used in the variability tests.

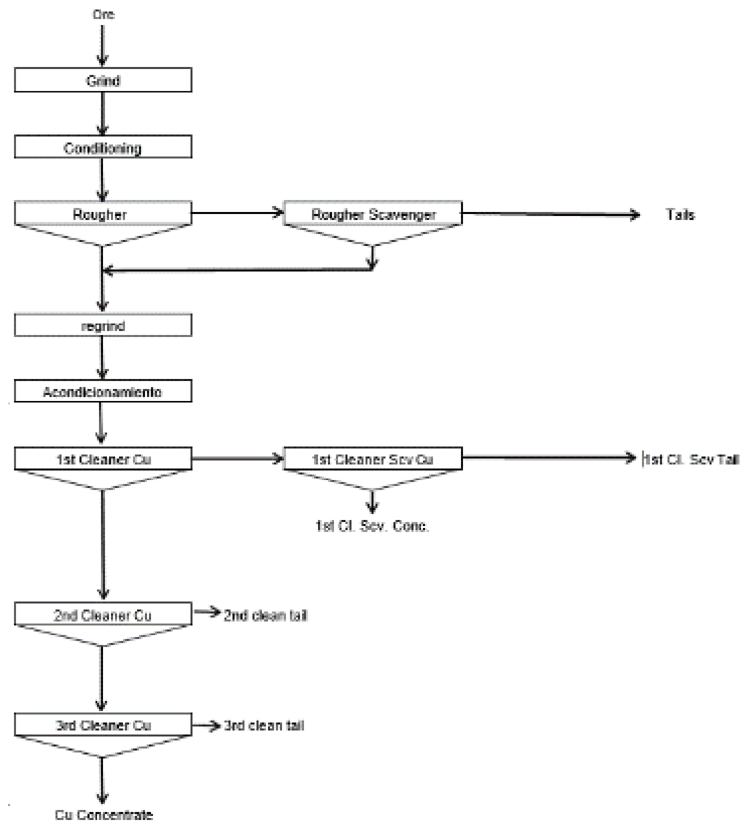


Figure 13.5: Variability Flow Scheme

Table 13.9: 3rd Cleaner Copper Concentrate Primary Variability Flotation Results										
Sample	P80	P80	Assays					Recovery %		
			Ag	Au	Cu	Fe	S	Ag	Au	Cu
	Grind µ	Regrind µ	g/t	g/t	%	%	%	%	%	%
AZ1047	164	38	120.00	0.86	18.80	23.60	32.80	69.20	34.20	80.70
AZ1053A	121	35	65.10	0.91	24.60	13.90	20.50	62.30	36.10	87.90
AZ1057	121	34	33.90	2.98	24.90	20.60	27.60	48.40	56.00	88.60
AZ1059	158	30	141.90	5.03	29.80	18.70	28.50	79.20	66.40	88.90
AZ1160A	122	24	73.80	4.69	37.20	24.20	31.00	70.20	52.60	88.80
AZ1161A	169	20	105.00	3.40	34.40	26.50	34.30	57.80	39.70	79.40
AZ1162	136	25	19.20	1.18	17.80	34.90	41.40	23.40	25.50	69.60
AZ1064A	94	30	57.90	1.32	12.10	36.10	42.20	63.00	50.40	80.00
AZ1168	138	25	159.60	5.66	39.30	20.60	29.30	79.70	53.10	91.40
AZ1170	183	26	49.20	9.90	32.50	22.50	28.30	44.20	59.80	89.70
Average	**141**	**29**	**82.60**	**3.59**	**27.10**	**24.10**	**31.60**	**59.70**	**47.40**	**84.50**

Table 13.10: 3rd Cleaner Copper Concentrate Supergene Variability Flotation Results											
Sample	P_{80}	P_{80}	R.C	Assays					Recovery %		
				Ag	Au	Cu	Fe	S	Ag	Au	Cu
	Grind µ	Regrind µ		g/t	g/t	%	%	%	%	%	%
AZ 1056	92	18	35.87	12.90	0.82	23.70	30.80	38.70	40.10	50.50	88.50
AZ 1173	109	19	99.00	17.40	1.30	25.80	30.20	39.50	25.80	38.30	64.50
AZ 1047	167	20	93.12	177.60	1 .21	43.60	17.70	26.60	72.30	38.20	80.30
AZ 1053 A	181	20	35.02	52.80	1.69	34.10	26.10	37.80	64.90	34.40	82.00
AZ 1054 A	137	20	44.41	38.70	0.17	20.70	33.30	45.20	60.90	27.20	89.00
AZ 1057	172	25	38.80	20.40	1.58	11.20	39.70	42.60	46.30	49.40	73.40
AZ 1059	292	30	100.63	143.70	8.41	39.50	18.30	21.20	65.80	53.90	72.00
AZ 1060 A	146	20	54.50	21.00	0.82	19.60	34.50	38.30	41.30	51.60	81.80
AZ 1061 A	114	20	49.54	32.40	1.40	51.60	12.20	25.90	55.20	45.30	88.70
AZ 1062	100	17	30.06	22.80	1.59	47.10	16.30	31.10	59.00	57.90	89.20
AZ 1064 A	130	20	48.36	6.60	0.39	11.50	38.00	47.60	21.20	17.20	61.40
AZ 1068	109	20	76.97	39.90	1.86	31.40	27.60	34.80	50.30	47.30	91.00
AZ 1170	227	20	80.78	54.60	1.08	38.20	24.20	32.70	48.20	12.40	75.70
AVERAGE	**152**	**21**	**60.54**	**49.30**	**1.72**	**30.60**	**26.80**	**35.50**	**50.10**	**40.30**	**79.80**

Increasing, or coarsening, the grind size shows a decrease in the overall flotation recovery of both the Primary and Supergene samples. However, increasing the grind size improves the enrichment ratio in both types of samples and the final concentrate contains a higher copper grade. Table 13.11 shows the correlations of recovery and enrichment ratio with grind size. Based on the recovery and enrichment ratio correlations with the grind size, a P_{80} of 175 µm is the optimum grind size for the Primary and Supergene materials. The concentrate regrind size in cleaner flotation did not show a significant effect on the recovery, but the enrichment ratio decreased by increasing the grind size in the Primary material samples. Therefore, a P_{80} of 25 µm was selected as the optimal grind size to maximize the flotation efficiency.

Table 13.11: Copper Distribution in Rougher Tails of Samples										
Primary						Supergene				
Grind Size P_{80} microns						Grind Size P_{80} microns				
Sample	150µm	75 µm	45 µm	-45 µm		Sample	150µm	75 µm	45 µm	-45 µm
9247	55.8	17	9.8	17.3		**9257**	16.6	6.9	5.5	71.1
9248	5.8	31.5	17.7	44.9		**9258**	11.3	15	16.2	57.6
9249	24.3	12.9	11	51.9		**9259**	42.6	26.8	6.9	23.7
9250	48.1	7	20.5	24.4		**9260**	21.3	14.6	16.1	48
9251	9.4	11.7	9	69.9		**9261**	30.5	18.4	7.3	43.8
9252	46.5	20.1	7.6	25.8		**9262**	7.1	4.9	8.7	79.3
9253	24.4	14.4	13.4	47.7		**9263**	62.8	6.6	7.1	23.5
9254	17.6	17.7	9.9	54.9		**9264**	27.6	12.5	5.7	54.2
9255	33	10.3	7	49.7		**9265**	17.2	14.3	8.1	60.4
9256	29.8	14.3	7	48.9		**9266**	34.4	12.3	7.8	45.4
Average	**29.5**	**15.7**	**11.3**	**43.5**		**9267**	5.5	2.8	13.9	77.7
						9268	14	9.9	8	68.2
						9269	24.7	2.5	14.5	58.3
						Average	**24.3**	**11.3**	**9.7**	**54.7**

13.2.3.3 Locked Cycle Flotation Testing

A single flotation flowsheet configuration was utilized in all the test work programs (depicted in Figure 13.6); therefore, the locked cycle test results are comparable. In the case of comparison of Supergene and Primary flotation efficiency, the locked cycle test results demonstrate that the copper recovery to concentrate for the

Supergene material is lower than for the Primary zone material. The summary of results of the locked- cycle tests are shown in Table 13.12.

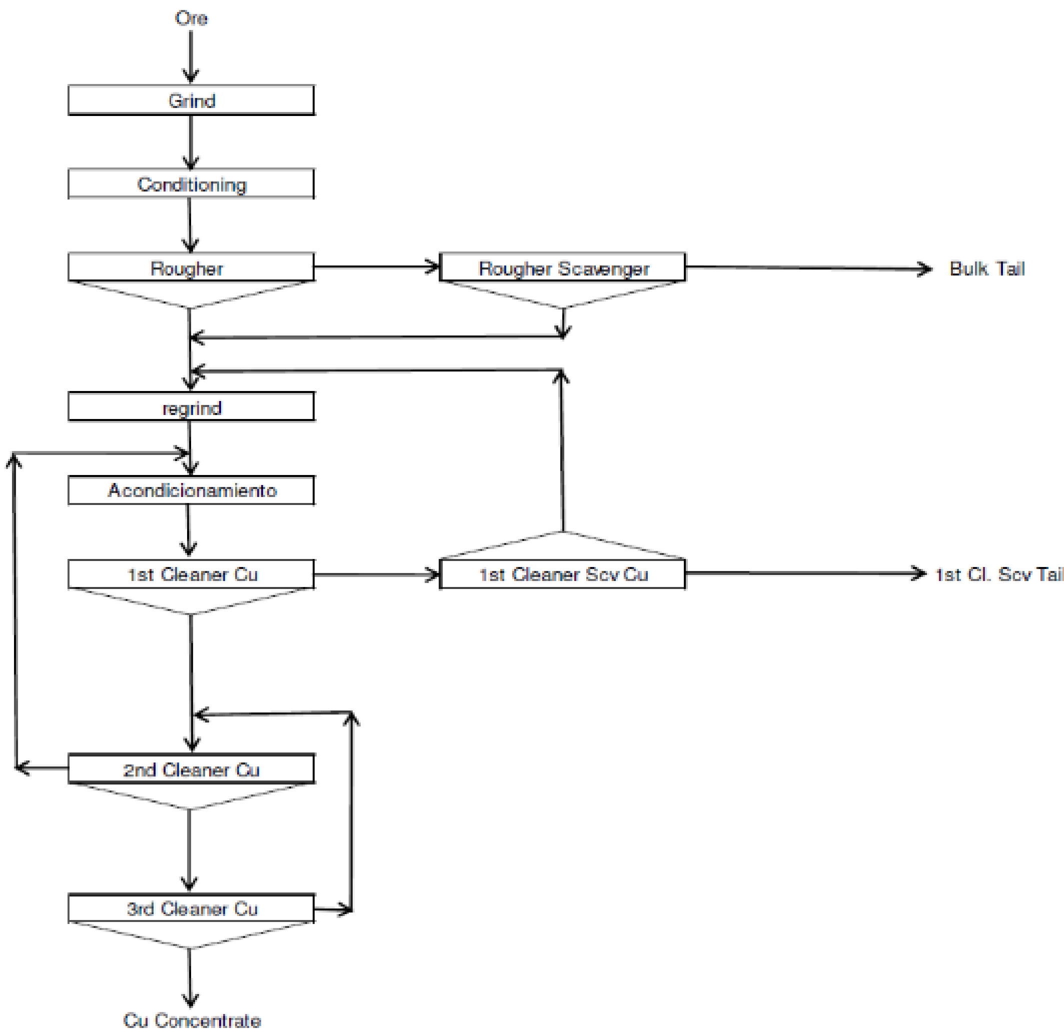


Figure 13.6: Lock-cycle Flow Scheme

Table 13.12: Optimized Lock-cycle Flotation Results							
Composite	**Concentrate Wt. % Of Feed**	**Copper Concentrate Grade**			**Recovery %**		
		Ag g/t	**Au g/t**	**Cu %**	**Ag**	**Au**	**Cu**
Primary	**1.45**	97	3.79	**31.96**	68.8	62.9	**93.2**
Supergene	**2.1**	28.6	3.58	**28.53**	54.0	65.6	**89.3**

With locked cycle testing completed, Table 13.12 provides the basis for recovery of copper, gold, and silver produced through a concentrate.

13.2.3.4 Flotation Concentrate Leach

The copper extractions by pressure oxidation from the Primary Table 13.13) and Supergene Table 13.14) concentrates were excellent at 99% and 98% respectively. The oxidation of sulfides ranges from 60% to 97%, and its effect on copper extraction was not measurable.

The average gold extraction by cyanidation of the autoclave residue is 72% and gold extraction correlates with the amount of sulfur oxidation. The residue weight loss correlates with the acid added to autoclave feed. The average autoclave conditions tested were 225°C, 75 psig oxygen overpressure, 9% solids, and 70-minute retention time.

Table 13.13: Primary Material POX Results

				Pressure (psig)			Residue				PLS					Extraction			
Run	Time (min)	Acid (g/L)	Fe^{3+} (g/L)	Over Pressure	Steam	Total	Wt. %	Cu (%)	Fe (%)	S^{+2} (%)	pH	ORP (mv)	FAT (g/L)	Cu (g/L)	Fe (g/L)	Cu (%)	Fe (%)	Oxidation (%)	O_2 (t/t)
1	70	15	5	100	322	422	35.5	1	51.9	4. 4	< 0.1	502	50.62	34.6	14.2	98.86	40.2	95.3	0 .54
2	50	15	20	100	322	422	83.8	0.5	46.3	11.4	< 0.1	602	69.59	34.3	6.8	98.63	14.2	70.8	0 .64
3	50	25	5	100	322	422	54.2	1	47.6	5.8	< 0.1	607	67.23	33.5	5.1	98.25	16.2	90.4	0 .69
4	60	20	12.5	100	322	422	50.1	1.3	50.2	4.6	< 0.1	532	66.99	34.5	13.3	97.94	32.4	93	0 .65
5	50	25	20	100	322	422	75.7	0.8	43.3	10.5	< 0.1	600	74.38	33.6	12.3	98.09	25.4	75.7	0 .60
6	50	15	5	100	322	422	37	1	52.3	1.8	< 0.1	500	54.98	33.5	12.6	98.8	37.5	97.9	0 .64
7	70	25	5	100	322	422	38.7	12.3	39.3	12.5	< 0.1	490	42.8	30.3	19.3	84.34	51.9	85.3	0 .49
8	70	25	20	100	322	422	95.5	2	38.5	12.3	< 0.1	582	66.44	34.4	8.7	93.97	16.9	64.2	0 .58
9	70	15	20	100	322	422	80.8	2.4	44.3	10.4	< 0.1	502	70.75	34.2	8.6	93.74	18.7	74.4	0 .72

Table 13.14: Supergene Material POX Results

				Pressure (psig)			Residue				PLS					Extraction			
Run	Temp (C)	Tim (min)	Acid (g/L)	Over Pressure	Steam	Total	Wt. %	Cu (%)	Fe (%)	S^{+2} (%)	pH	ORP (mv)	FAT (g/L)	Cu (g/L)	Fe (g/L)	Cu (%)	Fe (%)	Oxidation (%)	O_2 (t/t)
1	225	52 .5	45	75	355	430	16.2	0.78	37.9	40.8	< 0.1	494	51	26.8	40.4	99.5	85.1	92.9	0.52
2	220	60	30	50	322	372	10.4	1.96	25.9	16.7	< 0.1	491	38	28.4	44.3	99.1	93.3	95.5	0.99
3	220	45	30	100	322	422	96.9	0.14	33.9	14.3	< 0.1	607	77	27.7	8.1	99.4	18.4	64.2	0.75
4	230	45	30	50	391	441	8.3	4.55	30.1	28.8	< 0.1	438	46	27.3	43.3	98.3	93.5	93.5	0.82
5	230	60	60	50	391	441	25.3	0.68	43.2	46.4	< 0.1	469	63	29	35.1	99.3	73	69.6	0.88
6	230	45	60	100	391	491	7	0.65	25.8	23.6	< 0.1	510	69	28.7	46	99.8	95.5	95.7	0.84
7	220	45	60	50	322	372	13.2	1.17	38.2	36.6	< 0.1	486	63	28.7	40.8	99.3	87.5	87.5	0.91
8	220	60	60	100	322	422	9.3	0.29	34.9	34.7	< 0.1	470	66	28.6	43.9	99.9	92.1	91.6	0.53
9	230	60	30	100	391	491	92.5	0.23	33.6	17	< 0.1	645	73	28.3	9.5	99.1	22.2	59.3	0.89
10	220	45	30	50	322	372	8.3	1.99	26.6	16.6	< 0.1	462	38	29.2	45.3	99.3	94.6	96.4	0.81
11	225	52.5	45	75	355	430	9	1.63	34.2	33.5	< 0.1	473	50	28.8	43.7	99.4	92.4	92.1	0.87
12	220	60	30	100	322	422	13.2	2.18	42.9	7.9	< 0.1	500	62	28.2	40	98.8	85.9	97.3	0.93
13	230	45	30	100	391	491	74.4	0.21	32.6	16.3	< 0.1	654	69	28.4	16.4	99.3	39.2	68.5	0.87
14	220	45	60	100	322	422	45.4	4.78	43.2	47.6	< 0.1	465	47	26.5	25.5	90.7	52.1	44	0.73
15	230	60	30	50	391	441	95.4	0.14	35.1	15.7	< 0.1	602	71	27.4	8	99.4	17.9	61.2	1.02
16	230	60	60	100	391	491	9.3	0.33	32.2	34.3	< 0.1	465	65	29.6	44	99.9	92.6	91.7	0.92
17	220	60	60	50	322	372	69	17	34.7	38.8	< 0.1	390	45	15	17.9	99.6	93.2	92.2	0.71
18	230	45	60	50	391	441	77.2	20.1	33.8	38.9	< 0.1	370	46	9.9	15.5	99.5	43.9	73.8	0.82

13.2.3.5 *Mineralogy*

The MLA mineralogy was performed by Dr. Paul Miranda at Thompson Creek Mine.

The study revealed that the following species characterize the material:

Table 13.15: Primary Material Mineralogy, mesh

Mineral	Chemistry	+70	-70 / +100	-100 / +200	-200 / +325	-325
Albite	$NaAlSi_3O_8$	67.49	50.28	39.51	46.74	43.97
Quartz	SiO_2	14.94	20.26	35.77	23.43	22.65
K_Feldspar	$KAlSi_3O_8$	8.7	14.91	19.27	14.1	18.16
Biotite	$K(Mg, Fe)_3(Si_3Al)O_{10}(OH)_2$	5.3	7.08	2.25	13.28	12.91
Native Copper Iron	CuFe	1.53	1.37	1.74	0.93	0.59
Native Copper	Cu	0.96	0.98	0.58	0.58	0.57
Chalcopyrite	$CuFeS_2$	0.45	0.62	0.33	0.33	0.51
Pyrite	FeS_2	0.46	0.23	0.31	0.32	0.35
Bornite	Cu_5FeS_4	0.11	0.27	0.25	0.29	0.28

Table 13.16: Supergene Material Mineralogy, mesh

Mineral	Chemistry	+65	-65 / +100	-100 / +200	-200 / +325	-325
Albite	$NaAlSi_3O_8$	52.9	44.27	34.22	38.19	60.34
Quartz	SiO_2	26.46	23.41	33.28	30.24	20.45
K_Feldspar	$KAlSi_3O_8$	17.62	17.08	23.49	22.62	13.13
Biotite	$K(Mg, Fe)_3(Si_3Al)O_{10}(OH)_2$	1.31	10.35	3.02	3.26	3.05
Native Copper	Cu	0.47	1.36	1.24	0.66	0.36
Native Copper Iron	CuFe	0.24	1.34	0.73	0.61	0.18
Chalcocite	$CuFeS_2$	0.2	0.87	0.67	0.52	0.14
Ilmenite	FeS_2	0.15	0.51	0.67	0.15	0.05
Chalcopyrite	Cu_5FeS_4	0.13	0.36	0.57	0.02	0.04
Bornite	$NaAlSi_3O_8$	0.09	0.22	0.29	0.01	0.01

13.2.4 SGS 2016-2017 Historical

Samples were shipped to SGS in Santiago, Chile for SMC comminution testing. Additionally, an investigation into coarse material flotation was carried out by SGS Lakefield laboratories in 2016. The flotation conditions were selected based on the optimum flotation conditions determined from the previously reported tests on the Primary and Supergene samples.

13.2.4.1 *Grind*

Samples from ten different drill cores and depths of the Los Azules deposit, representing the first five years of operation, were sent to SGS Minerals Services in Santiago, Chile in early June 2017. SMC Tests were conducted on each of the ten samples to determine the JKSimMet and SMC Test comminution parameters.

The JK Drop Weight Tests are reported as A, b, and ta values. The A and b are parameters which describe the response of the material under test to increasing levels of input energy in single impact breakage. The Axb values for the tested Los Azules samples were compared by SGS to the JKTech database. The vertical orange lines shown in Figure 13.7 represent the values for Los Azules and the green line represents the frequency distribution from the JKTech database.

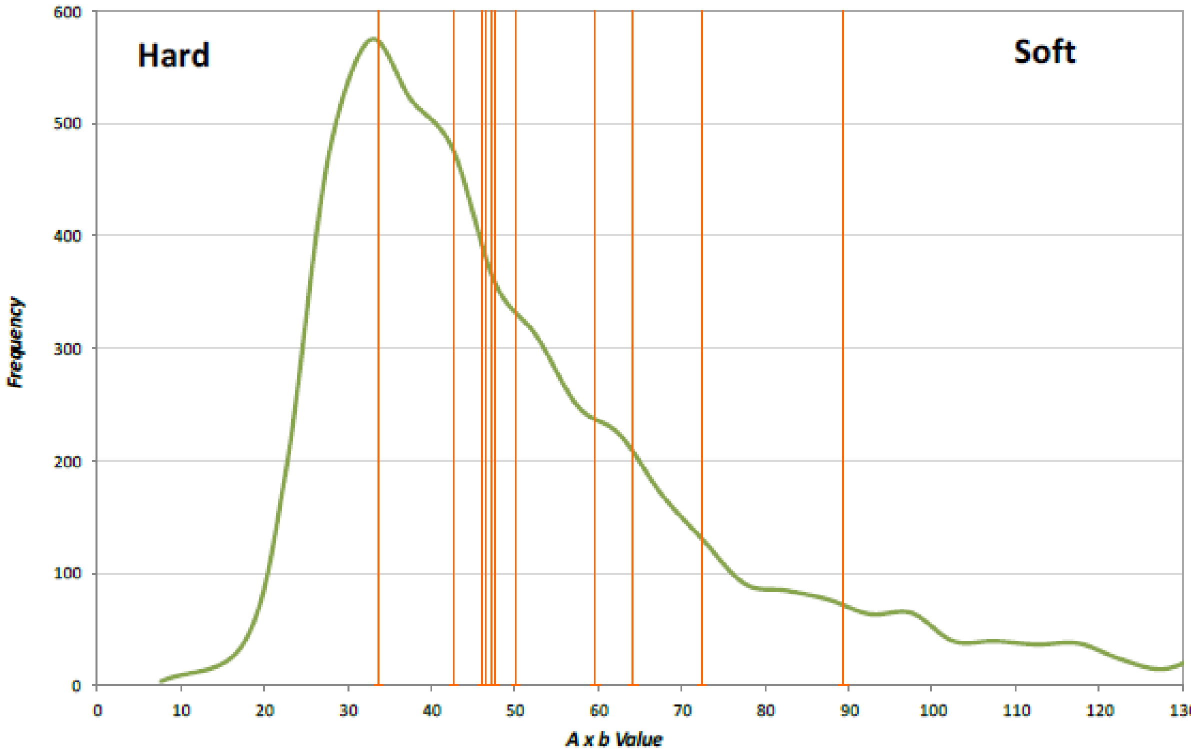


Figure 13.7: Frequency Distribution of Axb values in JKTech Database

13.2.4.2 Coarse Material Flotation Test work

An investigation into coarse material flotation was carried out by SGS Lakefield laboratories in the summer of 2016. Based on the overall life of mine plan tonnages, a blend of 45% from the Primary zone and 55% from the Supergene zone was tested. The flotation conditions were selected based on the optimum flotation conditions determined from the previously reported tests on the Primary and Supergene samples. In this regard, the following reagents were selected for the coarse flotation tests: lime as a modifier, A-3477, Flexon 715 and SIBX as collectors and F-150 as frother and the overall flotation retention time was 19 min for the kinetic tests. Table 13.17 shows neither a fine (92 µm) nor a coarse grind size (262 µm) provide the optimum metallurgical performance. Instead, the medium grind size (134 µm) demonstrates better metallurgical results.

On the other hand, Table 13.17 demonstrates that increasing the grind size from 125 µm to 206 µm decreases the rougher mass pull by 25% and increases the rougher concentrate grade from 4.49% to 4.76% after 10 minutes of flotation. Copper recovery decreases by 2.7%.

Table 13.17: A Summary of Mass Pull, Grade and Recovery of the Copper Concentrate

	Grind size (µm)						
	92	**125**	**134**	**150**	**175**	**206**	**262**
Mass pull %	16.2	14.2	13.7	13.1	12.3	11.4	9.4
Copper Conc. Grade %	3.51	3.97	4.10	4.24	4.47	4.76	5.27
Copper Recovery%	96.5	96.2	96.1	95.5	94.6	93.5	84.6

An optimum therefore exists based on capital and operational cost savings versus reduced recovery at coarser grind sizes. A preliminary trade-off study was conducted which recommended a 175 µm grind size as it allows for smaller equipment sizes without a drastic loss in overall recovery.

13.3 CURRENT METALLURGICAL TEST WORK PROGRAMS

The current metallurgical program consists of three (3) concurrent phases of worked aimed at supporting a feasibility study level of investigation. In the current Phase 1 program work, existing drill core was selected for testing by lithology and material type to reflect economically processable material in the resource for this study. Phases 2 and 3 will utilize new metallurgical core obtained from the ongoing drilling program to investigate the potential metallurgical variability of the deposit and focus on the initial 3-5 years of materials to be mined.

McEwen and SE developed and implemented the main metallurgical test program (Phase 1) at a well-established Mineral Processing research and development firm in Santiago, Chile (SGS Santiago). The metallurgical test program has been developed and supervised by Samuel Engineering.

The Phase 1 Metallurgical test program focused on the following:

- Head Characterization (Sequential Copper, Fire Assay, Sulfur Speciation, Carbon Speciation, ICP-MS (50 elements), Fluoride, Chloride, and Mercury)
- Comminution test work by lithology (SPI, SMC, LEIT, BWi, Ai, SG, and Bulk Density)
- Grind vs rougher flotation recovery curves by lithology
- Sulfuric Acid Bottle Rolls by lithology
- Composite Sulfuric Acid Column Leach by material type
- Solid-Liquid settling and filtration by lithology
- Head Sample Mineralization (XRD, Clay Analysis, XRF, and TIMA-X PMA)
- Acid Generation Prediction and Humidity Cell Testing

The Phase 1 metallurgical test program was designed to determine the metallurgical response variability of the resource to the selected operating parameters from historical parameters, using several resource samples representing the depth and breadth of the resource. The program was also designed to develop data to project metal recoveries, process reagent requirements, and to support process equipment sizing and selection.

A total of 78 separate samples were chosen by SE, representing the Los Azules resource covering the major lithologies (Diorite, Porphyritic Diorite, Hydrothermal Magmatic Breccia, Quartz Vein, Dacite Porphyry, and Rhyodacite Porphyry) and material types (Oxide/LIX, Supergene, and Primary). Cancha Geometallurgy Software was utilized to provide analysis and interpretation to the material body; allowing for geostatistical functions to ensure that samples are representative. Figure 13.8 and Figure 13.9 show the spatial representation of the 78 samples compared to the 5 Year Pit and Ultimate Pit.

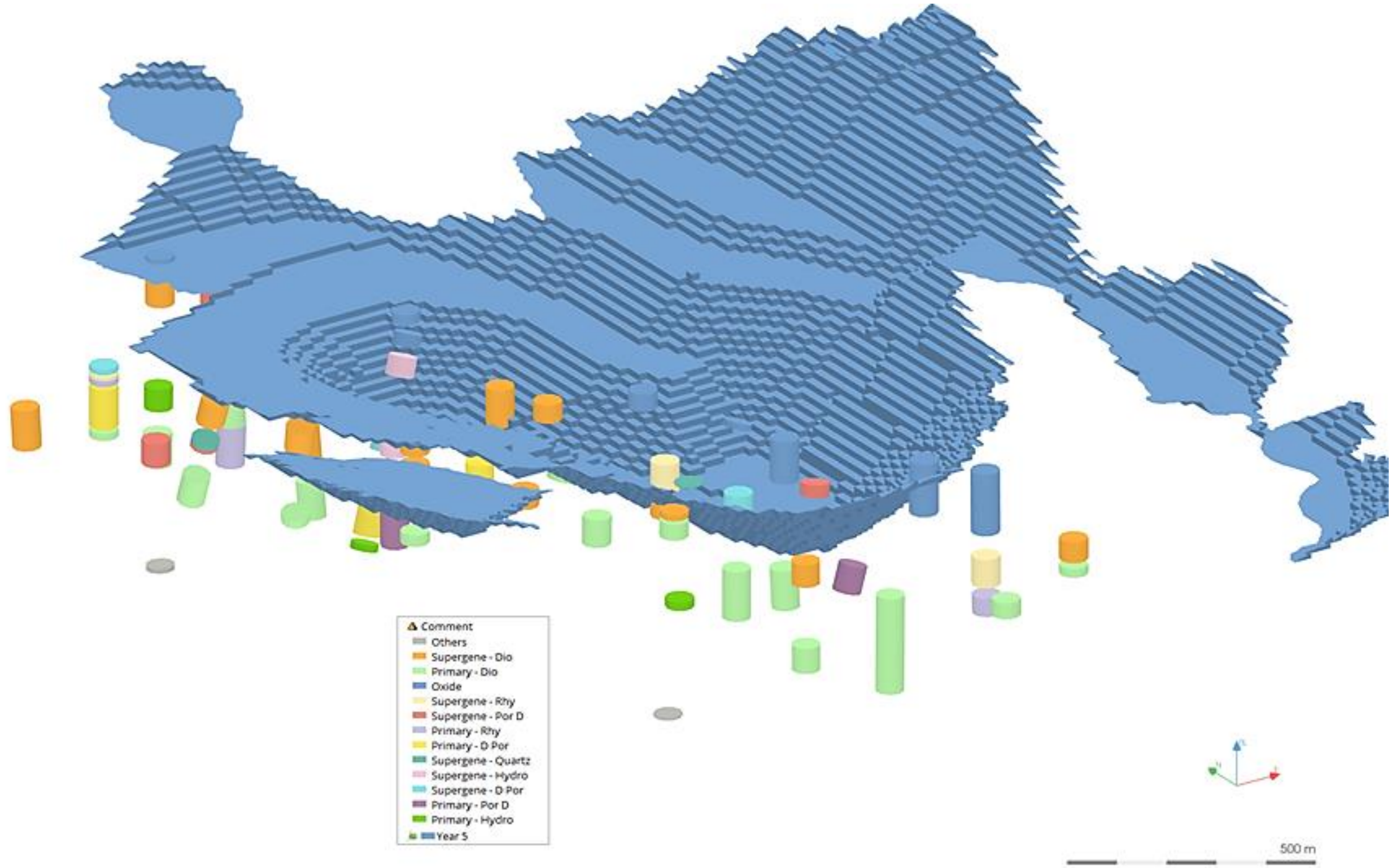


Figure 13.8: Spatial Representation of Phase 1 Metallurgical Samples in the 5 Year Pit (SE, 2023)

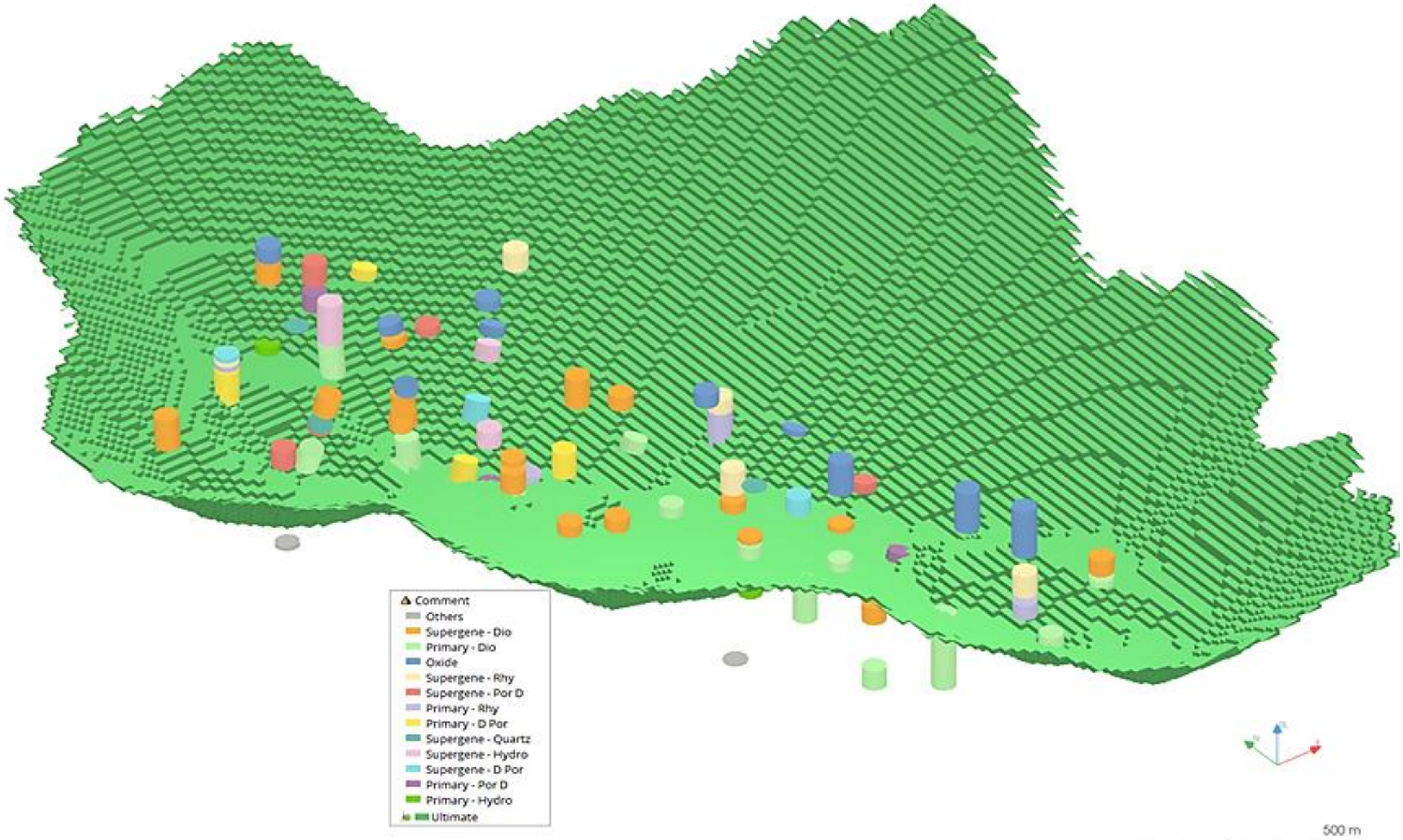


Figure 13.9: Spatial Representation of Phase 1 Metallurgical Samples in the Ultimate Pit (SE, 2023)

McEwen personnel identified and shipped existing core samples to SGS. In June of 2022 a total of 7,490 kg of existing drill core was delivered to SGS.

In Table 13.18, the event timings and their associated lithologies are shown for the two (2) major pits being utilized in the PEA document; one for the 5 Year Pit and the next for the Ultimate Pit. The percentages are a tabulation of number of blocks (standard 20x20x15 block) by lithology in the block model within each of those pits. Table 13.19 and Table 13.20 are from the information produced by CRM-SA, LLC.

Table 13.18: Event Timings and Associated Lithologies

Event Timing	Associated lithologies	Ultimate Pit	5 Year Pit
Hydrothermal Breccia	Hydrothermal Breccia	0.10%	0.31%
Early Mineral Porphyry	Rhyodacite Porphyry	9.06%	19.09%
Pre-Mineral Pluton	Diorite; Porphyritic Diorite; Monzodiorite; Quartz Diorite	72.97%	65.22%
Inter Mineral Porphyry	Dacite Porphyry	13.66%	13.89%
Overburden / Cover	Overburden	Not Calculated	Not Calculated
Magmatic Hydrothermal Breccia	Magmatic Hydrothermal Breccia	3.04%	1.17%
Volcanics	Andesite; Andesitic Tuff; Rhyolite	1.16%	
Late Quartz Veins	Quartz Vein	Not Modelled in Leapfrog	

Table 13.19: Event Timings and Material Types in the Ultimate Pit															
Mineral Zone	Hydrothermal Brx (101)		Early Mineral Porphyry (102)		Pre-Mineral Pluton (103)		Inter Mineral Porphyry (104)		Overburden (105)		Magmatic Hyd. Breccia (106)		Volcanics (107)		Late Qtz Veins (108)
	Number of Blocks	Average Total Cu (%)	Number of Blocks	Average Total Cu (%)	Number of Blocks	Average Total Cu (%)	Number of Blocks	Average Total Cu (%)	Number of Blocks	Average Total Cu (%)	Number of Blocks	Average Total Cu (%)	Number of Blocks	Average Average Total Cu (%)	Not Modeled in Leapfrog
2 - Oxide/LIX	18	0.11	2,835	0.015	15,647	0.026	3,179	0.021	10,924	0.028	94	0.116	90	0.002	0
3 - Supergene	55	1.45	2,478	0.750	2,420	0.803	673	0.611			211	1.341			0
4 - Mixed	13	0.75	1	0.120	99	0.647	18	0.233			21	0.201			0
5 – Bornite/Chalcopyrite															
6 – Bornite															
7 – Primary			3	0.530											
Total	86	1.06	5,317	0.358	18,166	0.133	3,870	0.125	10,924	0.028	326	0.914	90	0.002	0

Table 13.20: Event Timings and Material Types in 5 Year Pit															
Mineral Zone	Hydrothermal Brx (101)		Early Mineral Porphyry (102)		Pre-Mineral Pluton (103)		Inter Mineral Porphyry (104)		Overburden (105)		Magmatic Hyd. Breccia (106)		Volcanics (107)		Late Qtz Veins (108)
	Number Of Blocks	Average Total Cu (%)	Number Of Blocks	Average Total Cu (%)	Number Of Blocks	Average Total Cu (%)	Number Of Blocks	Average Total Cu (%)	Number Of Blocks	Average Total Cu (%)	Number Of Blocks	Average Total Cu (%)	Number Of Blocks	Average Total Cu (%)	Not Modeled in Leapfrog
2 - Oxide/LIX	18	0.11	2,835	0.015	15,647	0.026	3,179	0.021	10,924	0.028	94	0.116	90	0.002	0
3 - Supergene	55	1.45	2,478	0.750	2,420	0.803	673	0.611			211	1.341			0
4 - Mixed	13	0.75	1	0.120	99	0.647	18	0.233			21	0.201			0
5 – Bornite/Chalcopyrite															
7 - Primary			3	0.530											
Total	86	1.06	5,317	0.358	18,166	0.133	3,870	0.125	10,924	0.028	326	0.914	90	0.002	0

Sample preparation was completed by SGS using the preparation diagram as shown in Figure 13.10. Four (4) lots of each sample were grind calibrated for TESCAN Integrated Mineral Analyzer (TIMA-X) particle mineral analysis (PMA).

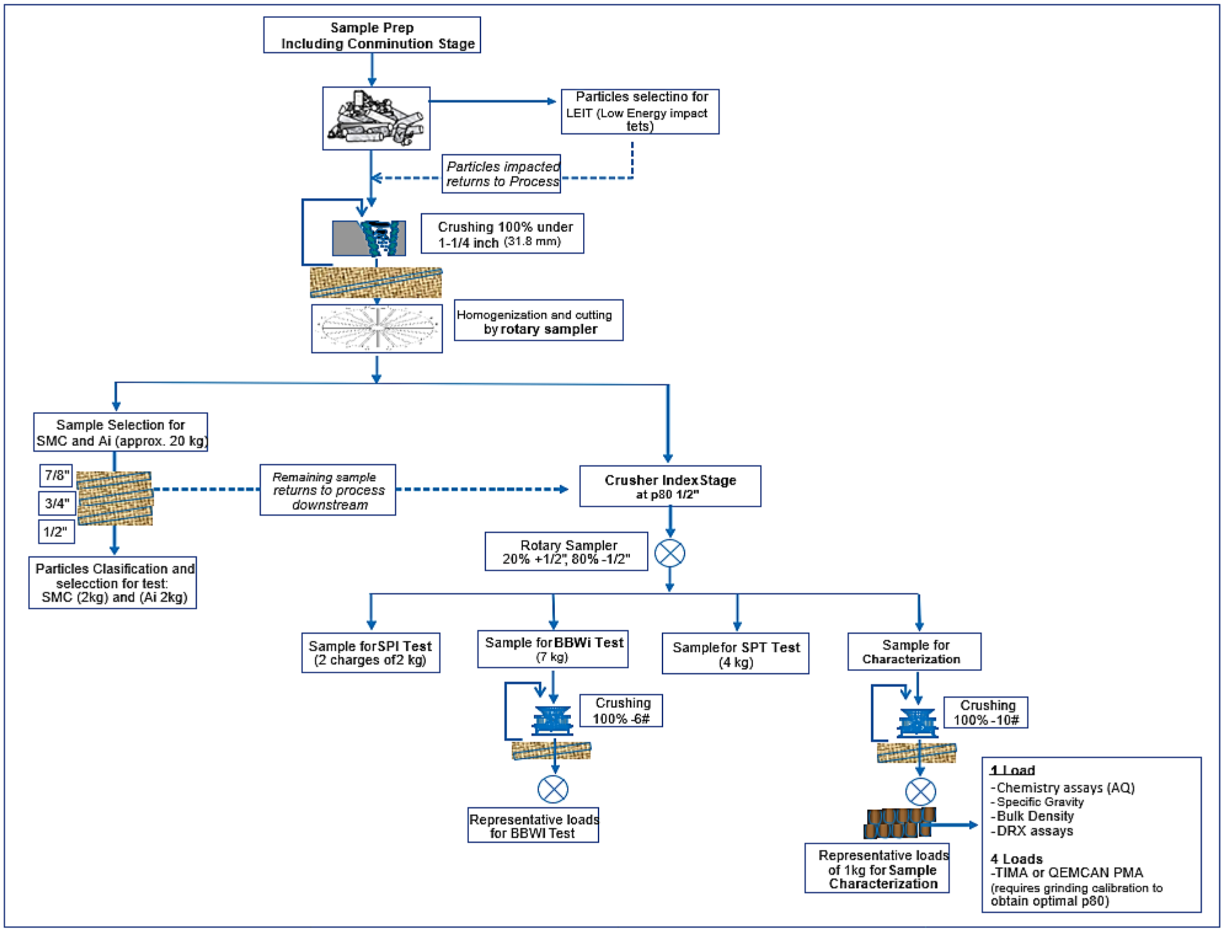


Figure 13.10: SGS Sample Preparation (SGS, 2022)

Of the 78 samples, six (6) were labelled as Dacitic Porphyry, 32 were labelled as Diorite, six (6) were labelled as Hydrothermal Magmatic Breccia, 13 were labelled as Oxide/LIX, 8 were labelled as Porphyry Diorite, five (5) were labelled as Quartz Vein, and eight (8) were labelled as Rhyodacite Porphyry.

Out of the 78 total samples sent to SGS, nine (9) composites were created based on the differing material types of, as shown in Table 13.21.

Table 13.21: SGS Composite Creation				
Composite Number	**Composite Name**	**Hole #**	**From**	**To**
Composite 1	Oxide/LIX Low	AZ1294	58	110
Composite 1	Oxide/LIX Low	AZ1047	72	102
Composite 1	Oxide/LIX Low	AZ1061A	54	68
Composite 1	Oxide/LIX Low	AZ1057	15	158

Table 13.21: SGS Composite Creation				
Composite Number	**Composite Name**	**Hole #**	**From**	**To**
Composite 1	Oxide/LIX Low	AZ1170	62	102
Composite 2	Oxide/LIX Mid	AZ1171	86	144
Composite 2	Oxide/LIX Mid	AZ1059	42	86
Composite 2	Oxide/LIX Mid	AZ1062	6	95
Composite 2	Oxide/LIX Mid	AZ17130	114	130
Composite 2	Oxide/LIX Mid	AZ1064A	14	112
Composite 3	Oxide/LIX High	AZ1053A	66	94
Composite 3	Oxide/LIX High	AZ17122	64	100
Composite 3	Oxide/LIX High	AZ1060A	68	96
Composite 4	Supergene Low	AZ0614	132	180
Composite 4	Supergene Low	AZ0618	90	108
Composite 4	Supergene Low	AZ1059	86	154
Composite 4	Supergene Low	AZ0835	276	293
Composite 4	Supergene Low	AZ1047	116	138
Composite 5	Supergene Mid	AZ12100	208	242
Composite 5	Supergene Mid	AZ1063	120	162
Composite 5	Supergene Mid	AZ0835	127	168
Composite 5	Supergene Mid	AZ17122	100	158
Composite 5	Supergene Mid	AZ17128	196	258
Composite 5	Supergene Mid	AZ0835	144	153
Composite 6	Supergene High	AZ0838	194	278
Composite 6	Supergene High	AZ17130[1]	217	286
Composite 6	Supergene High	LA9803	221	250
Composite 6	Supergene High	AZ1053A	264	322
Composite 6	Supergene High	AZ1055	128	188
Composite 6	Supergene High	AZ1047	276	372
Composite 6	Supergene High	AZ12114	130	186
Composite 7	Primary Low	AZ12116	320	414
Composite 7	Primary Low	AZ1284	196	310
Composite 7	Primary Low	AZ17130	470	492
Composite 7	Primary Low	AZ0618[1]	202	236
Composite 7	Primary Low	AZ0614[1]	194	214
Composite 7	Primary Low	AZ0618	290	316.4
Composite 7	Primary Low	AZ1058	212	290
Composite 7	Primary Low	AZ1065	322	384
Composite 8	Primary Mid	AZ1294	252	310
Composite 8	Primary Mid	AZ1059	451	526

Table 13.21: SGS Composite Creation				
Composite Number	**Composite Name**	**Hole #**	**From**	**To**
Composite 8	Primary Mid	AZ0946	372	464.9
Composite 8	Primary Mid	AZ1064A	312	404
Composite 8	Primary Mid	AZ1054	292	356
Composite 8	Primary Mid	AZ1053A	426	506
Composite 8	Primary Mid	AZ1173	120	157
Composite 8	Primary Mid	AZ1047	408	492
Composite 8	Primary Mid	AZ0946	252	370
Composite 8	Primary Mid	AZ1057	303	345
Composite 9	Primary High	AZ0838	278	340
Composite 9	Primary High	AZ1059	344	396
Composite 9	Primary High	AZ1055	336	408
Composite 9	Primary High	AZ1168	318	354
Composite 9	Primary High	AZ1053A	522	538
Composite 9	Primary High	AZ1067	342	362
Composite 9	Primary High	AZ1048[1]	429	466
Composite 9	Primary High	AZ1280	76	136
Composite 9	Primary High	AZ12113	92	126
Composite 9	Primary High	AZ1299	70	108
Composite 9	Primary High	AZ17120	168	217

1 – Denotes all material was used to create the composite, no head characterization was able to be completed.

13.3.1 Head Characterization

Detailed head assays and multi-element assays for each sample and composite used in the Phase 1 Metallurgical program were performed. Table 13.22 shows the head assays and multi-element assays for the various lithologies and rock types. The analyses are consistent with the sample analyses noted in the historical metallurgical data. The summary of data is represented in Table 13.22 for Oxide/LIX, Table 13.23 for Supergene, and Table 13.24 for Primary material types.

Table 13.22: Oxide/LIX Head Assays

BHID	From (m)	To (m)	Zone	CuT (%)	CuAS (%)	CuCN (%)	CuSOL (%)	CuSOL/CuT Ratio (%)	Fe (%)	Mo (%)	Ag (ppm)	Au (g/t)	S (%)	Sulfide (%)	Sulfate (g/t)	C (%)	CO3 (%)	Fluoride (ppm)	Hg (g/t)	Cl (kg/t)	As (ppm)
AZ1047	72	102	Oxide/LIX	0.358	0.053	0.162	0.215	60.06%	1.764	0.004	5.13	0.28	0.39	0.32	<0.01	0.03	0.16	487	<0.15	0.10	429
AZ1053A	66	94	Oxide/LIX	0.287	0.047	0.095	0.142	49.48%	1.384	0.004	2.26	0.07	0.14	0.13	<0.01	0.06	0.29	615	0.62	0.10	61
AZ1057	15	158	Oxide/LIX	0.018	0.002	0.009	0.0011	61.11%	1.225	0.003	0.78	0.11	0.26	0.05	<0.01	0.03	0.15	292	0.32	<0.1	22
AZ1059	42	86	Oxide/LIX	0.056	0.005	0.011	0.016	28.57%	3.021	0.004	1.27	<0.03	0.1	0.07	<0.01	0.03	0.16	515	0.39	0.10	92
AZ1060A	68	96	Oxide/LIX	0.063	0.013	0.012	0.025	39.68%	1.976	0.001	1.39	0.05	0.05	0.01	<0.01	0.02	0.11	428	<0.15	<0.1	28
AZ1061A	54	68	Oxide/LIX	0.032	0.003	0.008	0.011	34.38%	1.561	<0.001	1.19	0.14	0.14	0.09	0.15	0.04	0.19	436	0.81	<0.1	135
AZ1062	6	95	Oxide/LIX	0.015	<0.002	0.006	0.008	53.33%	1.635	0.002	0.47	0.06	0.27	0.07	0.2	0.02	0.12	595	0.88	<0.1	83
AZ1064A	14	112	Oxide/LIX	0.014	<0.002	0.005	0.007	50.00%	1.557	0.003	0.63	0.05	0.26	0.05	0.07	0.04	0.19	489	0.41	<0.1	56
AZ1170	62	102	Oxide/LIX	0.03	0.003	0.006	0.009	30.00%	1.418	<0.001	0.77	0.09	0.22	0.17	<0.01	0.02	0.1	627	<0.15	<0.1	30
AZ1171	86	144	Oxide/LIX	0.085	0.009	0.04	0.049	57.65%	1.241	<0.001	0.37	<0.03	0.8	0.79	<0.01	0.02	0.1	534	<0.15	<0.1	12
AZ1294	58	110	Oxide/LIX	0.271	0.016	0.098	0.114	42.07%	1.782	0.005	1.68	<0.03	0.43	0.38	<0.01	0.03	0.13	498	<0.15	0.10	30
AZ17122	64	158	Oxide/LIX	0.119	0.016	0.064	0.08	67.23%	2.065	<0.001	0.42	<0.03	1.11	1.08	<0.01	0.02	0.11	752	0.95	<0.1	95
AZ17130	114	130	Oxide/LIX	0.036	0.004	0.005	0.009	25.00%	1.57	0.003	1.6	0.04	0.07	0.04	<0.01	0.02	0.09	684	<0.15	<0.1	24

Table 13.23: Supergene Head Assays

BHID	From (m)	To (m)	Zone	Lithology	CuT (%)	CuAS (%)	CuCN (%)	CuSOL (%)	CuSOL/ CuT Ratio (%)	Fe (%)	Mo (%)	Ag (ppm)	Au (g/t)	S (%)	Sulfide (%)	Sulfate (g/t)	C (%)	CO3 (%)	Fluoride (ppm)	Hg (g/t)	Cl (kg/t)	As (ppm)
AZ0835	127	168	Supergene	Dacitic Porphyry	0.209	0.022	0.117	0.139	66.51%	1.732	0.003	0.69	<0.03	0.44	0.4	<0.01	0.04	0.18	452	0.42	0.10	8
AZ0835	174	269	Supergene	Dacitic Porphyry	0.222	0.012	0.078	0.09	40.54%	1.15	0.004	1.04	<0.03	0.68	0.65	0.07	0.02	0.11	551	0.7	0.14	14
AZ1053A	264	322	Supergene	Dacitic Porphyry	0.4	0.022	0.194	0.216	54.00%	1.166	0.002	1.49	0.04	0.6	0.51	<0.01	0.02	0.1	576	0.41	0.10	134
AZ0614	132	180	Supergene	Diorite	1.131	0.055	0.716	0.771	68.17%	2.226	0.003	1.79	0.16	1.17	1.12	0.07	0.04	0.22	783	<0.15	<0.1	21
AZ0618	90	108	Supergene	Diorite	0.26	0.024	0.172	0.196	75.38%	1.851	0.002	1.18	<0.03	1.27	0.71	<0.01	0.02	0.12	566	<0.15	<0.1	24
AZ0618	132	192	Supergene	Diorite	0.153	0.016	0.076	0.092	60.13%	1.638	0.002	0.6	<0.03	0.69	0.64	0.06	0.02	0.12	267	0.84	<0.1	19
AZ0835	276	293	Supergene	Diorite	0.189	0.006	0.024	0.03	15.87%	2.667	0.004	1.51	0.06	1.12	0.99	0.04	0.19	0.95	486	<0.15	<0.1	206
AZ1047	116	138	Supergene	Diorite	1.001	0.087	0.591	0.678	67.73%	1.485	0.005	4.53	0.12	0.94	0.94	0.01	0.05	0.25	513	1.75	<0.1	198
AZ1047	276	372	Supergene	Diorite	0.452	0.008	0.231	0.239	52.88%	1.855	0.005	1.4	0.04	0.33	0.3	<0.01	0.19	0.94	522	<0.15	0.15	24
AZ1055	128	188	Supergene	Diorite	0.7	0.045	0.008	0.053	7.57%	1.999	0.002	0.9	<0.03	1.12	1.09	<0.01	0.04	0.2	459	0.24	<0.1	67
AZ1059	86	154	Supergene	Diorite	0.347	0.019	0.005	0.024	6.92%	2.337	0.004	1.25	0.03	0.26	0.25	0.46	0.02	0.1	758	0.43	<0.1	28
AZ1063	120	162	Supergene	Diorite	0.927	0.035	0.566	0.601	64.83%	1.366	0.003	0.83	0.09	1	0.95	<0.01	0.02	0.09	892	0.36	<0.1	49
AZ1064A	206	224	Supergene	Diorite	0.643	0.023	0.418	0.441	68.58%	1.392	0.002	1.56	0.08	1.48	1.33	<0.01	0.11	0.53	375	<0.15	0.10	53
AZ12100	208	242	Supergene	Diorite	0.221	0.015	0.102	0.117	52.94%	1.917	<0.001	0.3	<0.03	0.64	0.61	<0.01	0.02	0.08	416	<0.15	0.20	8
AZ12106	118	206	Supergene	Diorite	0.994	0.04	0.602	0.642	64.59%	1.474	0.005	1.76	0.1	0.59	0.7	0.09	0.03	0.17	331	<0.15	0.13	14
AZ17130	217	286	Supergene	Diorite	0.35	0.016	0.163	0.179	51.14%	1.417	0.003	1.57	0.05	0.33	0.31	<0.01	0.02	0.11	462	0.21	0.13	27
AZ0946	128	250	Supergene	Hydrothermal Magmatic Breccia	1.026	0.024	0.471	0.495	48.25%	1.674	0.005	3.56	0.12	0.8	0.76	0.04	0.02	0.12	379	<0.15	0.11	79

Table 13.23: Supergene Head Assays

BHID	From (m)	To (m)	Zone	Lithology	CuT (%)	CuAS (%)	CuCN (%)	CuSOL (%)	CuSOL/ CuT Ratio (%)	Fe (%)	Mo (%)	Ag (ppm)	Au (g/t)	S (%)	Sulfide (%)	Sulfate (g/t)	C (%)	CO3 (%)	Fluoride (ppm)	Hg (g/t)	Cl (kg/t)	As (ppm)
AZ1053A	116	160	Supergene	Hydrothermal Magmatic Breccia	1.279	0.07	0.826	0.896	70.05%	1.144	0.005	2.02	0.06	0.7	0.66	0.01	0.02	0.1	525	<0.15	0.10	30
AZ12116	152	202	Supergene	Hydrothermal Magmatic Breccia	1.039	0.034	0.705	0.739	71.13%	1.429	<0.001	0.49	<0.03	1.44	1.12	<0.01	0.02	0.09	559	0.31	0.10	11
AZ1055	246	254	Supergene	Porphyry Diorite	0.39	0.019	0.176	0.195	50.00%	1.594	0.002	2.95	<0.03	0.71	0.67	<0.01	0.05	0.25	441	<0.15	0.10	16
AZ12114	130	186	Supergene	Porphyry Diorite	0.233	0.017	0.143	0.16	68.67%	2.752	<0.001	0.56	<0.03	3.04	2.98	<0.01	0.02	0.08	636	<0.15	<0.1	101
AZ17127	230	255	Supergene	Porphyry Diorite	0.226	0.026	0.127	0.153	67.70%	1.098	0.001	0.97	<0.03	0.26	0.25	<0.01	0.01	0.06	336	0.18	<0.1	10
AZ17128	288	342	Supergene	Porphyry Diorite	0.239	0.011	0.067	0.078	32.64%	1.831	0.002	1.14	0.05	0.76	0.24	1.39	0.04	0.2	788	<0.15	<0.1	96
AZ17128	196	258	Supergene	Porphyry Diorite	0.121	0.004	0.033	0.037	30.58%	2.409	0.002	0.91	<0.03	0.22	0.19	<0.01	0.02	0.11	348	0.42	<0.1	55
AZ17134	166	194	Supergene	Porphyry Diorite	0.589	0.025	0.268	0.293	49.75%	1.288	0.001	1.03	0.04	1.1	1.02	<0.01	0.08	0.39	420	<0.15	0.10	32
AZ0611	151.8	155	Supergene	Quartz	0.156	0.015	0.099	0.114	73.08%	0.497	<0.001	1.42	0.05	0.51	0.42	0.08	0.03	0.14	264	0.18	<0.1	68
AZ0843	127	132	Supergene	Quartz	0.405	0.025	0.25	0.275	67.90%	1.061	0.001	30	0.25	0.93	0.87	0.04	0.06	0.3	515	0.25	<0.1	49
AZ1055	232	244	Supergene	Quartz	0.479	0.023	0.276	0.299	62.42%	1.031	0.002	3.78	<0.03	1	0.81	0.16	0.07	0.34	466	<0.15	<0.1	23
AZ0835	144	153	Supergene	Rhyodacite Porphyry	0.216	0.014	0.085	0.099	45.83%	1.727	0.003	0.28	<0.03	0.4	0.37	0.04	0.03	0.17	336	<0.15	0.10	4
AZ0838	194	236	Supergene	Rhyodacite Porphyry	0.453	0.02	0.248	0.268	59.16%	1.254	0.001	1.04	0.04	1.07	0.99	<0.01	0.05	0.23	685	0.58	<0.1	10
AZ1057	213	279	Supergene	Rhyodacite Porphyry	0.625	0.029	0.449	0.478	76.48%	1.427	0.002	1.19	0.07	1.62	1.44	0.16	0.03	0.15	432	<0.15	<0.1	57
AZ12100	114	184	Supergene	Rhyodacite Porphyry	0.584	0.033	0.485	0.518	88.70%	1.391	0.002	1.66	0.04	1.18	1.14	0.07	0.02	0.1	577	0.34	<0.1	13

Table 13.24: Primary Head Assays

BHID	From (m)	To (m)	Zone	Lithology	CuT (%)	CuAS (%)	CuCN (%)	CuSOL (%)	CuSOL/CuT Ratio (%)	Fe (%)	Mo (%)	Ag (ppm)	Au (g/t)	S (%)	Sulfide (%)	Sulfate (g/t)	C (%)	CO3 (%)	Fluoride (ppm)	Hg (g/t)	Cl (kg/t)	As (ppm)
AZ1053A	426	506	Primary	Dacitic Porphyry	0.305	0.012	0.082	0.094	30.82%	1.312	0.002	1.67	<0.03	0.68	0.4	0.48	0.05	0.25	495	0.84	<0.1	12
AZ1058	212	290	Primary	Dacitic Porphyry	0.339	0.015	0.024	0.039	11.50%	1.359	0.002	1.01	0.05	1.32	0.77	1.13	0.05	0.23	482	0.15	<0.1	123
AZ17120	168	217	Primary	Dacitic Porphyry	0.965	0.039	0.618	0.657	68.08%	1.653	<0.001	1.01	0.04	1.6	1.52	<0.01	0.02	0.09	463	<0.15	<0.1	20
AZ0618	290	316.4	Primary	Diorite	0.241	0.003	0.016	0.019	7.88%	2.059	0.002	0.84	<0.03	1.79	1.73	0.06	0.02	0.12	838	<0.15	<0.1	14
AZ0946	372	464.9	Primary	Diorite	0.308	0.008	0.079	0.087	28.25%	1.393	0.003	1.24	0.05	0.81	0.28	1.43	0.06	0.3	557	0.4	0.16	81
AZ1047	408	492	Primary	Diorite	0.279	0.017	0.148	0.165	59.14%	1.388	0.002	1.16	0.05	0.65	0.27	1.05	0.08	0.41	672	0.67	0.17	58
AZ1054	292	356	Primary	Diorite	0.253	0.014	0.052	0.066	26.09%	1.764	0.001	0.41	<0.03	0.67	0.66	0.13	0.02	0.09	443	<0.15	<0.1	8
AZ1055	336	408	Primary	Diorite	0.415	0.029	0.175	0.204	49.16%	1.667	0.003	0.96	<0.03	0.51	0.47	0.03	0.03	0.17	435	0.75	0.11	12

Table 13.24: Primary Head Assays																						
BHID	From (m)	To (m)	Zone	Lithology	CuT (%)	CuAS (%)	CuCN (%)	CuSOL (%)	CuSOL/CuT Ratio (%)	Fe (%)	Mo (%)	Ag (ppm)	Au (g/t)	S (%)	Sulfide (%)	Sulfate (g/t)	C (%)	CO3 (%)	Fluoride (ppm)	Hg (g/t)	Cl (kg/t)	As (ppm)
AZ1059	451	526	Primary	Diorite	0.285	0.007	0.105	0.112	39.30%	1.914	0.001	1.76	0.05	2.23	0.31	6.36	0.12	0.61	405	0.2	0.17	28
AZ1064A	312	404	Primary	Diorite	0.314	0.009	0.056	0.065	20.70%	1.099	<0.001	0.8	0.05	2.06	0.5	3.46	0.04	0.19	499	<0.15	<0.1	12
AZ1168	318	354	Primary	Diorite	0.611	0.022	0.253	0.275	45.01%	1.047	0.004	1.92	0.07	0.56	0.52	<0.01	0.04	0.2	377	<0.15	0.20	10
AZ1173	120	157	Primary	Diorite	0.208	0.012	0.045	0.057	27.40%	2.552	0.002	0.71	<0.03	0.53	0.41	<0.01	0.03	0.15	496	<0.15	<0.1	15
AZ12113	92	126	Primary	Diorite	0.676	0.058	0.388	0.446	65.98%	2.534	0.02	0.7	<0.03	1.26	1.24	0.05	0.02	0.09	408	<0.15	<0.1	301
AZ1280	76	90	Primary	Diorite	0.428	0.032	0.192	0.224	52.34%	2.222	0.002	0.47	<0.03	1.09	1.04	0.05	0.03	0.14	481	<0.15	<0.1	55
AZ1280	98	136	Primary	Diorite	0.109	0.003	0.011	0.014	12.84%	2.874	0.002	0.25	<0.03	0.41	0.36	0.02	0.05	0.26	574	<0.15	<0.1	144
AZ1284	196	310	Primary	Diorite	0.186	<0.002	0.033	0.035	18.82%	1.99	0.011	0.62	<0.03	0.55	0.52	0.07	0.05	0.27	603	<0.15	0.10	15
AZ1285	195.1	416	Primary	Diorite	0.144	0.003	0.018	0.021	14.58%	2.361	0.003	0.55	<0.03	0.84	0.35	1.22	0.08	0.42	654	<0.15	<0.1	29
AZ1294	252	310	Primary	Diorite	1.013	0.014	0.097	0.111	10.96%	2.506	0.002	2.09	0.07	1.71	1.63	<0.01	0.05	0.23	540	<0.15	0.20	199
AZ1299	70	108	Primary	Diorite	0.315	0.059	0.204	0.263	83.49%	2.667	0.002	0.38	<0.03	0.52	0.47	0.03	0.01	0.05	984	0.25	<0.1	308
AZ1053A	522	538	Primary	Hydrothermal Magmatic Breccia	1.483	0.011	0.31	0.321	21.65%	3.314	0.004	5.46	0.19	2.51	1.35	3.09	0.09	0.43	564	<0.15	0.16	41
AZ1059	344	396	Primary	Hydrothermal Magmatic Breccia	0.772	0.027	0.332	0.359	46.50%	2.056	0.003	3.33	0.07	2.52	0.47	5.55	0.05	0.26	544	0.92	<0.1	120
AZ1067	342	362	Primary	Hydrothermal Magmatic Breccia	0.42	0.02	0.122	0.142	33.81%	1.026	0.02	2.5	<0.03	0.98	0.89	0.12	0.04	0.18	313	0.62	<0.1	105
AZ1065	322	384	Primary	Porphyry Diorite	0.27	0.003	0.034	0.037	13.70%	1.189	0.002	1.25	0.04	2.07	0.59	3.9	0.05	0.26	818	0.27	<0.1	19
AZ12116	320	414	Primary	Porphyry Diorite	0.231	<0.002	0.029	0.031	13.42%	1.19	0.003	1.05	<0.03	1.12	0.35	2.57	0.06	0.3	394	<0.15	<0.1	16
AZ12100	705	709	Primary	Quartz	0.268	<0.002	0.019	0.021	7.84%	1.611	0.005	1.76	0.03	5.11	1.16	11.3	0.43	2.15	574	0.92	<0.1	82
AZ12114	422	432	Primary	Quartz	0.544	<0.002	0.022	0.024	4.41%	3.181	0.005	23.3	0.38	5.53	3.91	3.99	0.14	0.7	797	2.41	<0.1	175
AZ0838	237.6	340	Primary	Rhyodacite Porphyry	0.278	0.013	0.118	0.131	47.12%	1.151	0.002	0.91	<0.03	0.56	0.56	<0.01	0.06	0.28	372	<0.15	0.10	23
AZ0946	252	370	Primary	Rhyodacite Porphyry	0.319	0.007	0.135	0.142	44.51%	1.34	0.004	1.43	0.05	0.64	0.22	1.12	0.05	0.26	573	0.58	0.15	79
AZ1057	303	345	Primary	Rhyodacite Porphyry	0.277	0.014	0.086	0.1	36.10%	1.868	0.001	0.62	0.05	1.75	1.7	0.05	0.03	0.14	507	<0.15	<0.1	8

13.3.2 Comminution Tests

Thirty of the variability samples were selected for comminution testing. Due to sample size and sample physical characteristics, a varying number of each type of comminution tests were completed, as follows:

- 78 Specific Gravity (SG)
- 74 Bulk Density
- 63 Low Energy Impact Tests (LEIT)
- 71SAG Mill Comminution (SMC)
- 71 SAG Power Index (SPI)
- 71 Bond Work Index (BWi)
- 71 Abrasion Index (Ai)
- 71 Static Pressure Test (SPT)
- SPI testing was selected for SAG mill sizing as this was the only type of small sample-size test that uses the whole sample particle-size spectrum to determine SAG mill power requirements. SPT can measure the energy required for High Pressure Grinding Rolls (HPGR). SMC is used to characterize material samples for grinding circuit design. The test data comparing the results from the three types of tests is summarized in Table 13.25 by material type and Table 13.26 by lithology.

Table 13.25: Comminution Data by Material Type

				LEIT	SMC		Ai	CWi	SPI	BWi	SPT	
		Average SG (g/cc)	Bulk Density (g/cc)	Average Impact Work Index (kWh/t)	Axb	t_a	Abrasion Index	Crusher Index	SPI (Minute)	BWI (kWh/mt)	SPT Energy	HPi
All Samples	Std Dev.	0.06	0.04	3.15	21.45	0.23	0.05	5.51	22.70	1.33	0.25	2.66
	Average	2.53	1.69	11.57	54.06	0.56	0.11	18.76	55.61	13.04	1.59	14.09
	Max	2.76	1.83	17.96	144.48	1.59	0.28	33.92	116.40	16.74	2.22	19.91
	Min	2.36	1.61	4.01	29.79	0.30	0.04	5.81	17.89	10.16	1.02	7.80
Oxide/LIX Material Type	Std Dev.	0.06	0.04	4.13	31.07	0.34	0.03	5.10	16.02	1.05	0.21	2.30
	Average	2.47	1.67	11.51	75.72	0.80	0.08	19.64	39.99	11.99	1.46	12.35
	Max	2.54	1.73	17.96	144.48	1.59	0.14	28.51	65.17	13.37	1.89	16.97
	Min	2.36	1.61	4.52	43.68	0.45	0.04	14.75	17.89	10.16	1.04	7.80
Supergene Material Type	Std Dev.	0.05	0.04	3.08	16.60	0.18	0.05	6.22	22.45	1.45	0.29	2.94
	Average	2.53	1.69	11.73	53.51	0.55	0.11	19.02	54.62	13.04	1.61	14.19
	Max	2.75	1.77	17.43	103.73	1.08	0.27	33.92	109.22	16.74	2.22	19.91
	Min	2.47	1.63	4.01	34.79	0.35	0.04	5.81	20.45	10.16	1.02	8.94
Primary Material Type	Std Dev.	0.05	0.05	2.69	12.20	0.12	0.05	4.96	22.12	1.04	0.21	2.19
	Average	2.57	1.69	11.44	44.60	0.45	0.14	18.08	63.93	13.54	1.64	14.79
	Max	2.76	1.83	15.69	74.36	0.75	0.28	30.61	116.40	15.58	2.02	19.21
	Min	2.48	1.63	4.87	29.79	0.30	0.06	11.00	27.35	11.01	1.13	9.77

Table 13.26: Comminution Data by Lithology Type												
				LEIT	SMC		Ai	CWi	SPI	BWi	SPT	
		Average SG (g/cc)	Bulk Density (g/cc)	Average Impact Work Index (kWh/t)	Axb	t_a	Abrasion Index	Crusher Index	SPI (Minute)	BWI (kWh/mt)	SPT Energy	HPi
Dacitic Porphyry	Std Dev.	0.05	0.01	2.53	9.77	0.11	0.05	4.89	13.34	1.23	0.26	2.59
	Average	2.54	1.67	12.42	43.67	0.45	0.13	17.42	55.48	13.10	1.77	15.60
	Max	2.61	1.70	15.06	57.34	0.60	0.21	22.64	69.70	14.54	2.22	19.91
	Min	2.48	1.65	9.24	30.02	0.30	0.06	10.07	39.39	11.90	1.50	13.30
Diorite	Std Dev.	0.04	0.04	2.44	17.01	0.18	0.05	5.02	22.65	1.07	0.23	2.40
	Average	2.54	1.69	10.94	48.08	0.49	0.11	18.08	63.30	13.36	1.65	14.80
	Max	2.62	1.77	14.79	103.73	1.08	0.28	30.61	116.40	15.58	2.02	19.21
	Min	2.48	1.63	4.01	29.79	0.30	0.04	11.00	20.45	11.01	1.08	9.77
Hydrothermal Magmatic Breccia	Std Dev.	0.06	0.04	1.73	20.45	0.22	0.06	7.96	21.06	1.75	0.16	1.92
	Average	2.53	1.69	13.83	55.18	0.57	0.13	20.68	53.88	13.01	1.58	13.94
	Max	2.62	1.74	15.69	90.52	0.95	0.21	33.92	83.90	15.29	1.77	16.06
	Min	2.47	1.64	11.96	36.92	0.37	0.05	13.82	26.74	10.16	1.30	10.58
Porphyry Diorite	Std Dev.	0.08	0.06	3.04	12.20	0.12	0.03	6.18	30.94	1.73	0.35	3.50
	Average	2.56	1.69	14.13	47.56	0.48	0.12	17.06	66.08	13.70	1.62	14.75
	Max	2.75	1.77	17.43	74.36	0.75	0.17	25.00	109.22	16.74	1.97	18.34
	Min	2.48	1.63	8.53	34.79	0.35	0.09	5.81	24.93	12.01	1.02	8.94
Quartz	Std Dev.	0.12	0.08	2.35	9.78	0.11	0.10	4.48	5.28	0.29	0.16	1.81
	Average	2.63	1.74	6.53	62.82	0.62	0.16	21.68	46.73	13.67	1.42	12.59
	Max	2.76	1.83	8.19	74.00	0.74	0.27	26.63	51.12	14.01	1.59	14.62
	Min	2.54	1.68	4.87	55.84	0.52	0.08	17.92	40.87	13.48	1.27	11.13
Rhyodacite Porphyry	Std Dev.	0.03	0.04	2.21	9.49	0.10	0.04	6.73	21.70	1.55	0.28	2.93
	Average	2.53	1.70	10.93	49.43	0.51	0.13	20.05	47.34	12.72	1.50	13.22
	Max	2.57	1.74	14.51	63.39	0.65	0.19	33.25	79.20	15.36	1.83	16.44
	Min	2.49	1.63	8.60	37.64	0.38	0.08	13.09	24.40	10.90	1.13	9.77

The design comminution parameters for the Los Azules comminution systems were determined using "S" curves that were generated using the comminution data set and the projected proportions of each lithology from the most current Los Azules resource model. The projected cumulative weight percentage of a given rock type was plotted as a function of the comminution parameter data.

Once the "S" curve plots were developed, the design comminution parameters were chosen as the point that would cover 80% of the resource. The "S" curve was generated for the Specific Gravity shown in Figure 13.11.

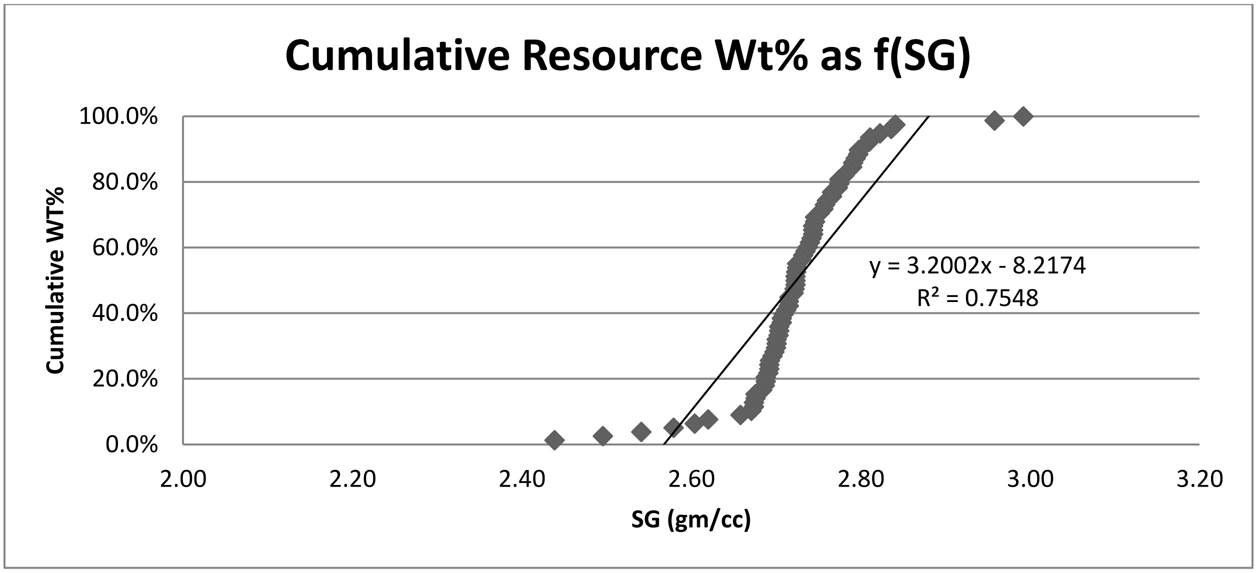


Figure 13.11: "S" Curve - Cumulative % as a Function of SG

In a similar manner an "S" curve was generated for the Bulk Density data shown in Figure 13.12.

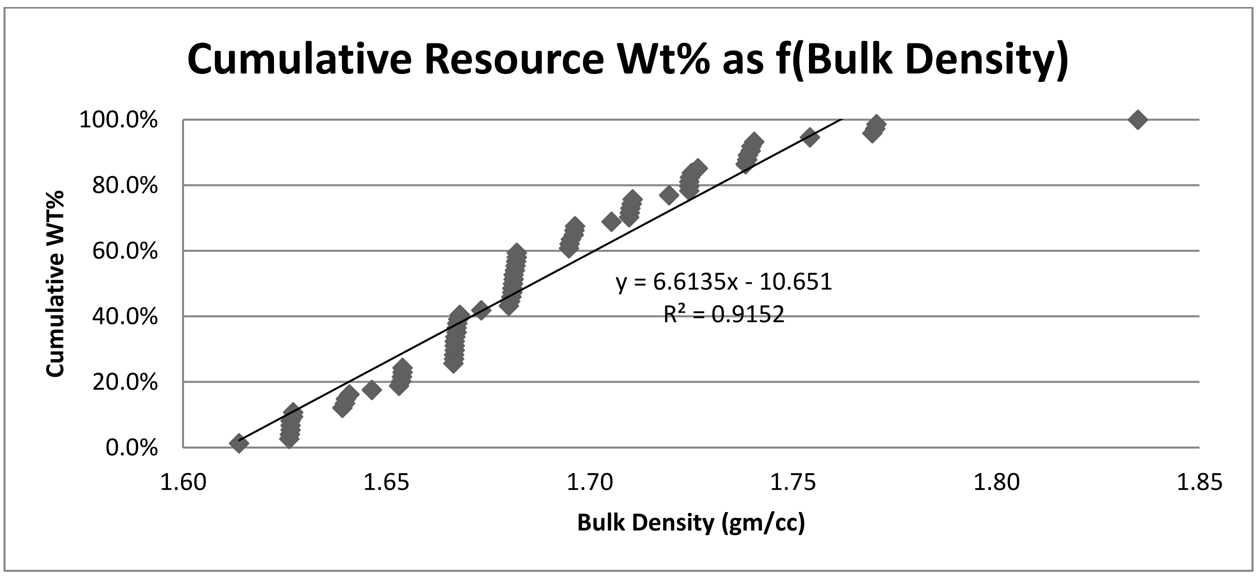


Figure 13.12: "S" Curve - Cumulative % as a Function of Bulk Density

In a similar manner an "S" curve was generated for the LEIT data shown in Figure 13.13.

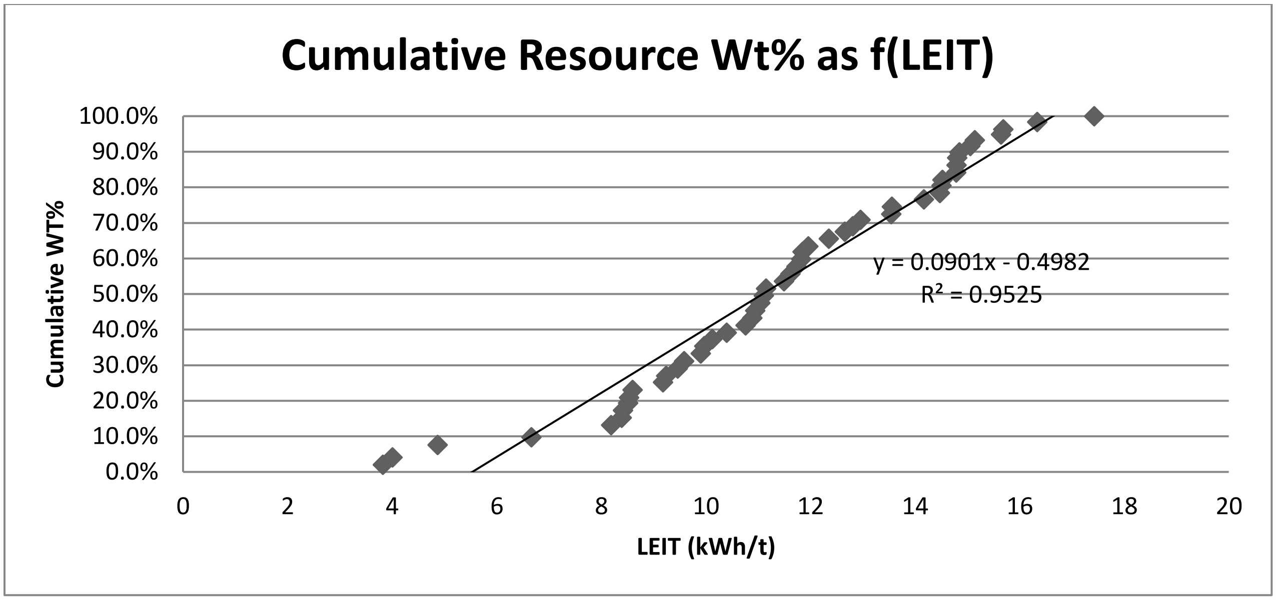


Figure 13.13: "S" Curve - Lithology Cumulative % as a Function of LEIT

In a similar manner an "S" curve was generated for the SAG Circuit Specific Energy (SCSE) data shown in Figure 13.14. SMC Tests were conducted on each of the 10 samples to determine the JKSimMet and SMC Test comminution parameters that could then be used to assess material behavior in the proposed process flowsheet.

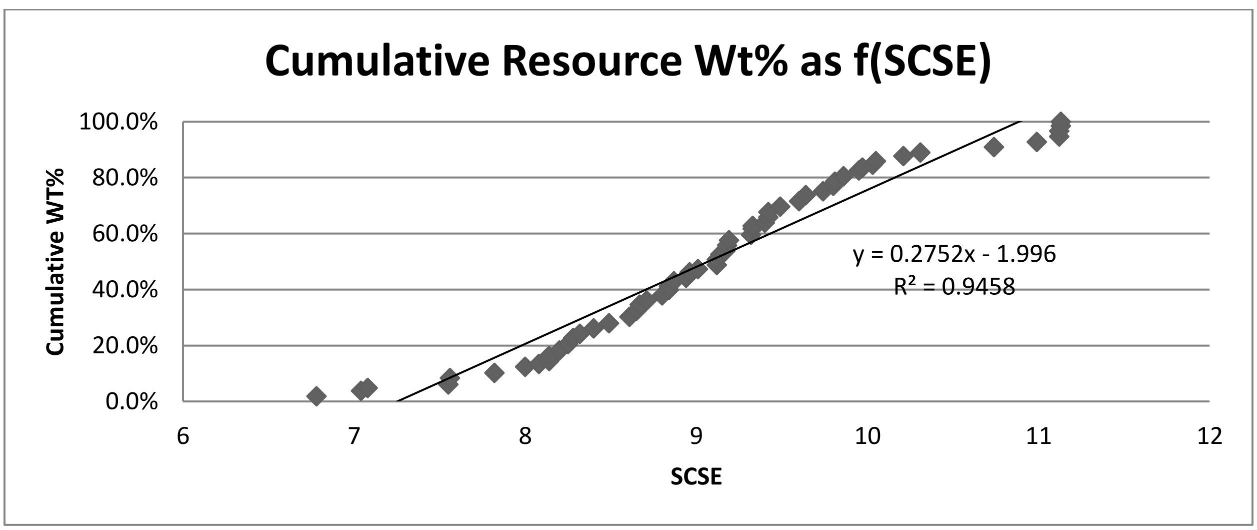


Figure 13.14: "S" Curve - Lithology Cumulative % as a Function of SMC (SCSE)

All SMC data was analyzed by JKTech against their existing database. The vertical blue lines shown in Figure 13.15 and Figure 13.16 represent the values for Los Azules and the purple lines represent the frequency distribution from the JKTech database.

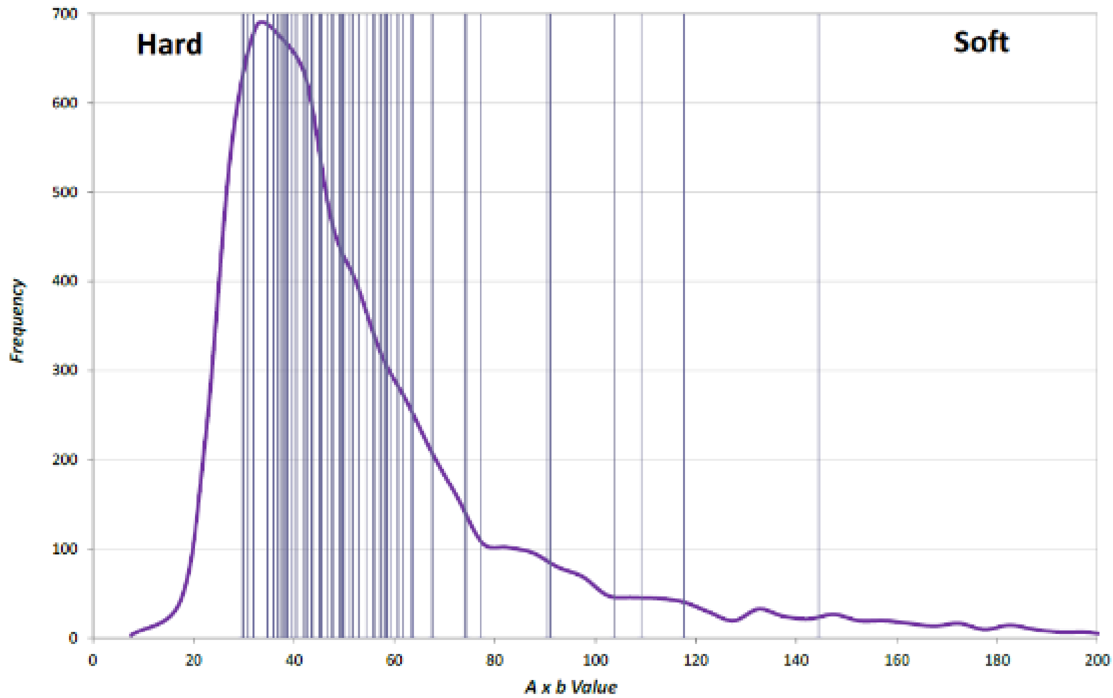


Figure 13.15: Frequency Distribution of Axb values in JKTech Database

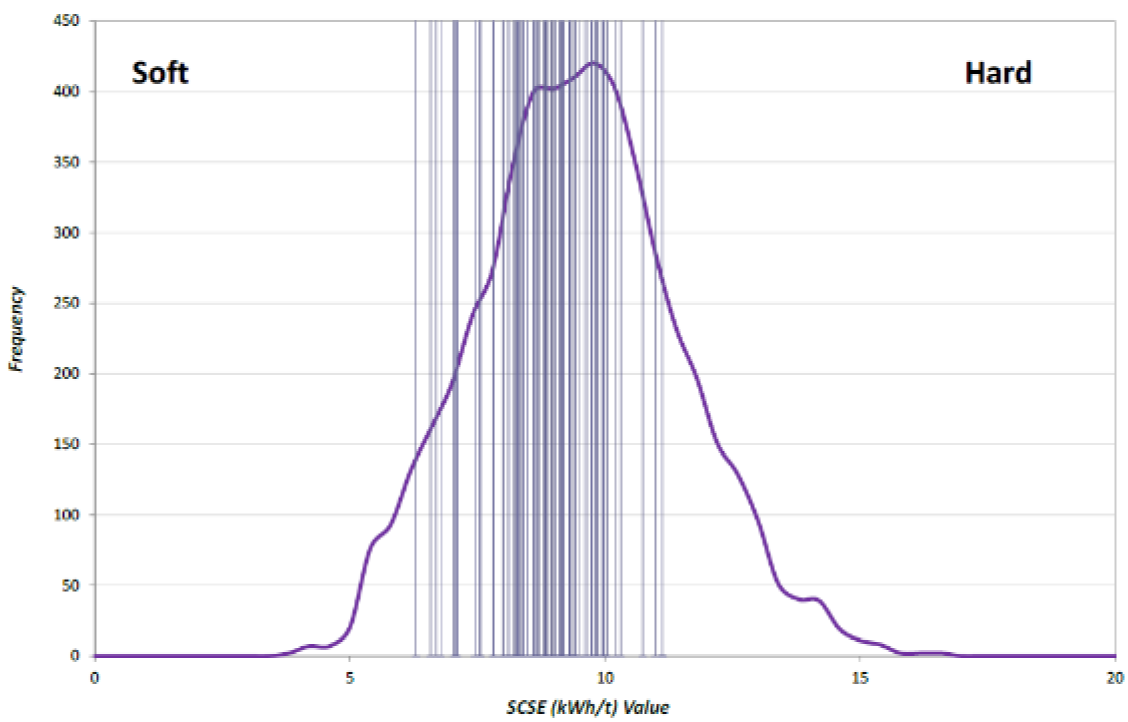


Figure 13.16: Frequency Distribution of SCSE values in JKTech Database

In a similar manner an "S" curve was generated for the SPI data shown in Figure 13.17.

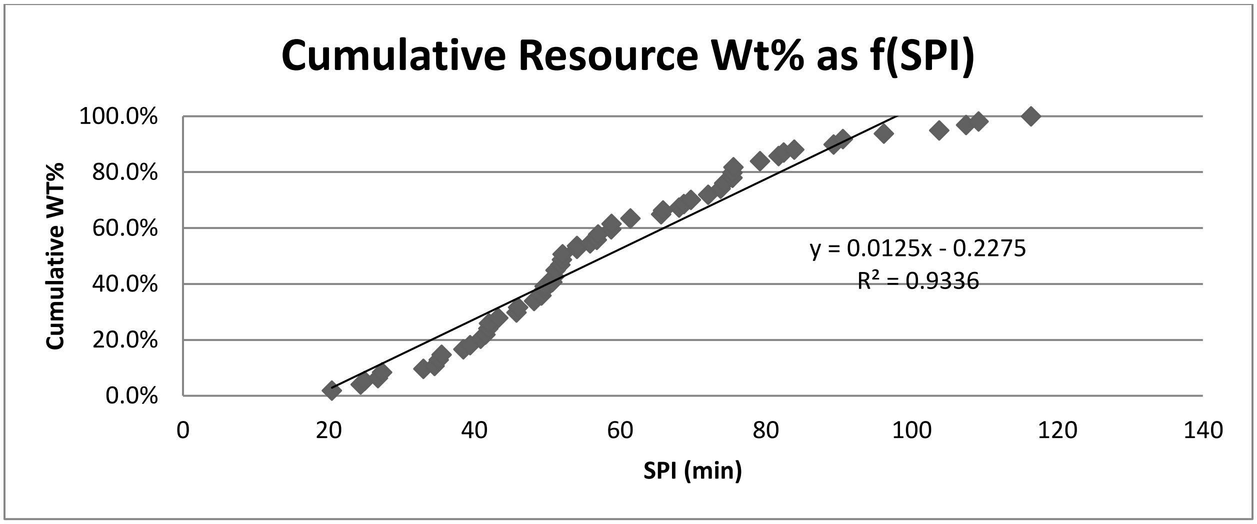


Figure 13.17: "S" Curve - Lithology Cumulative % as a Function of SPI

In a similar manner an "S" curve was generated for the BWi data shown in Figure 13.18.

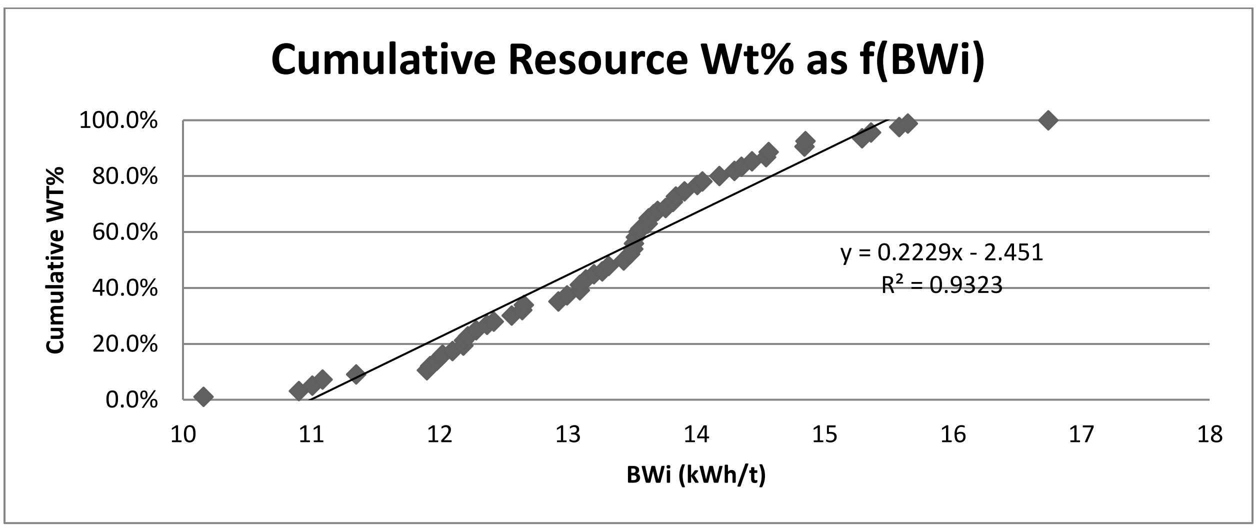


Figure 13.18: "S" Curve - Lithology Cumulative % as a Function of BWi

In a similar manner an "S" curve was generated for the CWi data shown in Figure 13.19.

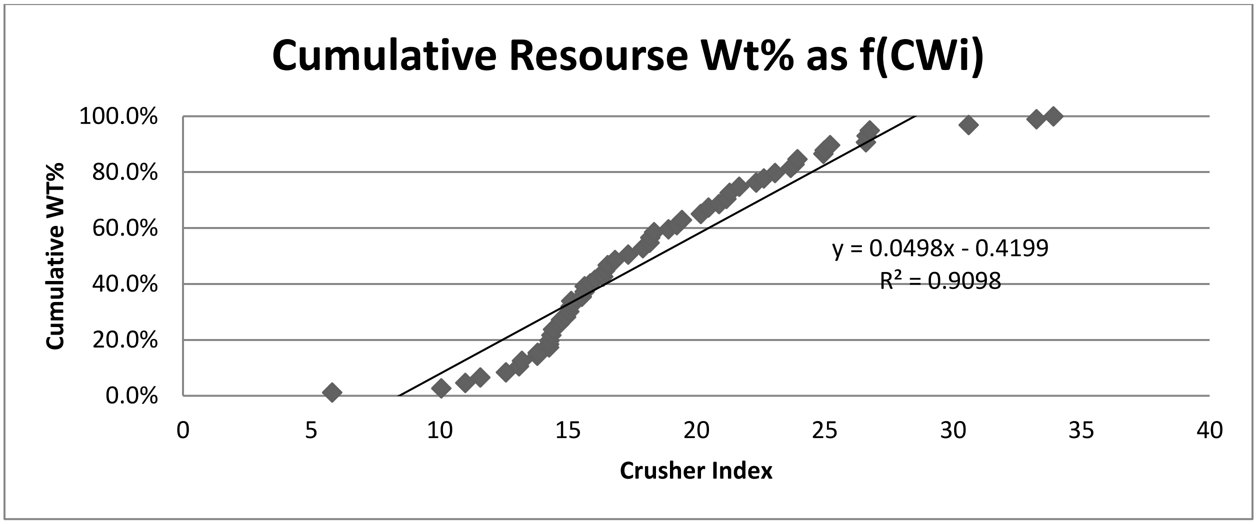


Figure 13.19: "S" Curve - Lithology Cumulative % as a Function of CWi

In a similar manner an "S" curve was generated for the Ai data shown in Figure 13.20.

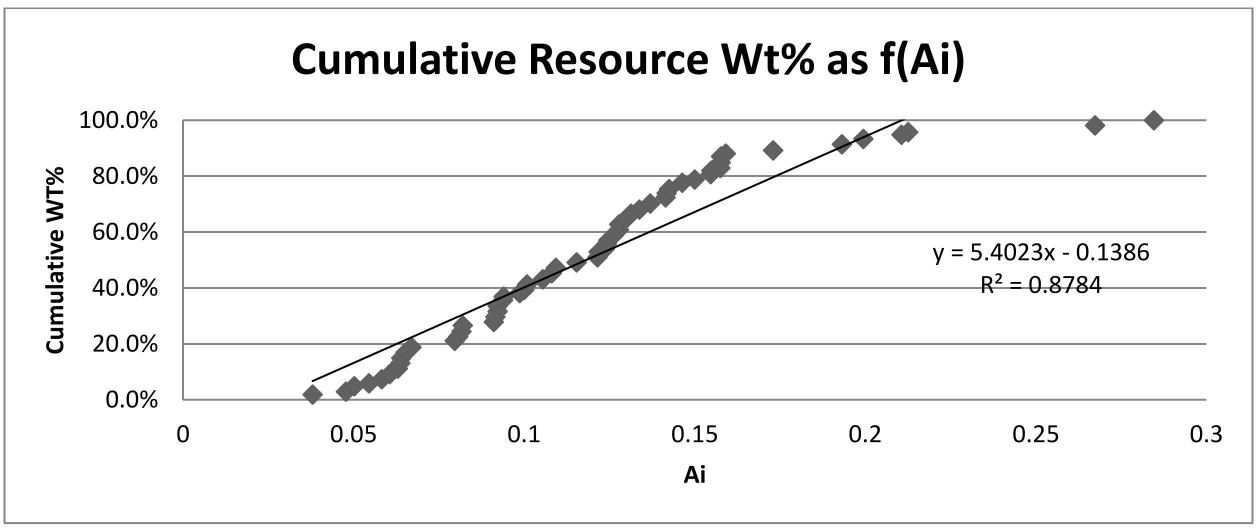


Figure 13.20: "S" Curve - Lithology Cumulative % as a Function of Ai

In a similar manner an "S" curve was generated for the SPT data shown in Figure 13.21.

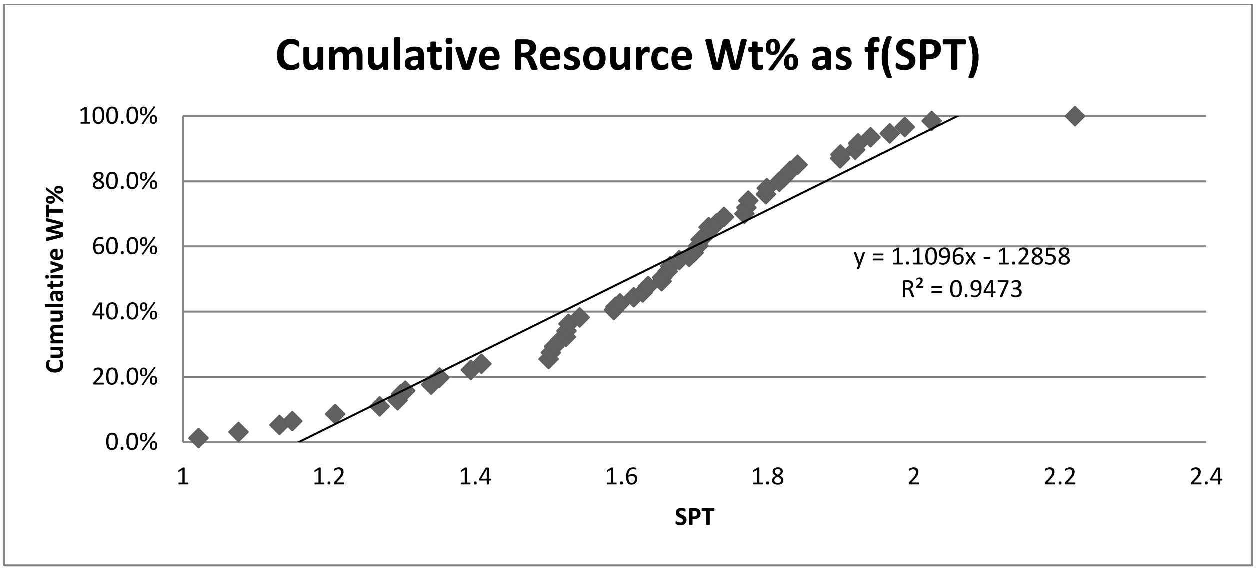


Figure 13.21: "S" Curve - Lithology Cumulative % as a Function of SPT

Based on the test work results from Table 13.27 and the "S" curves, the following design parameters were determined for the PEA:

Table 13.27: Comminution Design Data for PEA

			LEIT	SMC		Ai	CWi	SPI	BWi	SPT	
	SG (g/cc)	Bulk Density (g/cc)	Impact Work Index (kWh/t)	A x b	ta	Abrasion Index	Crusher Index	SPI (Minute)	BWI (kWh/mt)	SPT Energy	HPi
Design Number	**2.82**	**1.73**	**14.41**	**63.4**	**0.60**	**0.17**	**24.50**	**82.20**	**14.59**	**1.88**	**16.20**
Oxide/LIX	2.70	1.67	15.16	75.7	0.80	0.11	24.39	54.22	12.92	1.66	12.35
Dacitic Porphyry	2.68	1.67	14.06	43.7	0.45	0.17	20.90	65.22	14.00	1.96	15.60
Diorite	2.76	1.69	14.05	48.1	0.49	0.16	23.15	85.78	14.41	1.88	14.80
Hydrothermal Magmatic Breccia	2.69	1.69	14.78	55.2	0.57	0.17	26.59	68.67	14.28	1.70	13.95
Porphyry Diorite	2.72	1.69	16.46	47.6	0.48	0.15	22.10	90.53	15.10	1.91	14.75
Quartz Vein	2.79	1.73	6.86	62.8	0.62	0.20	23.52	48.87	13.80	1.48	12.59
Rhyodacite Porphyry	2.72	1.70	12.27	49.4	0.51	0.16	25.48	64.18	13.90	1.71	13.21

Design parameters were based on the 80th percentile of test work results, all other information reported utilizes an average.

13.3.3 Rougher Flotation - Grind vs Recovery

A total of 38 samples, comprising 29 drill core samples and the nine (9) composites were selected for various grind sizes (125, 175, 250, and 350 microns) and rougher flotation to complete a total of 152 flotation tests on. Of the 29 drill core samples, 16 were identified from the Primary material type, and 13 from the

Supergene material type. This allowed for a grind versus recovery curve on the material types and various lithologies and their metallurgical response. A total of 36 flotation tests, corresponding to the nine (9) composites have not yet been completed and are not available at the effective date of this report.

The conditions previously set at the Plenge programs and at SGS in 2017 provided the reagent suite for Primary and Supergene material types. The following rougher flotation conditions were carried out on all 152 samples:

- Flotation Charge – 1,000 grams,
- Percent Solids (%) – 30% using Santiago potable water,
- Air Flow – 10 L/min,
- Agitation – 1,300 rpm,
- Flotation pH – 9, lime used for pH adjustment,
- Flotation Time – 7 minutes,
- Reagents – AERO 2477, AERO 3894, Dowfroth-250, and MBS.

The average rougher recovery of the Primary material type 92.87% at 125 microns, 91.43% at 175 microns, 86.79% at 250 micron and 79.72% at 350 microns. This was at an average copper concentrate grade of 3.40%, 4.14%, 5.38% and 6.32% respectively. Average data for all 64 Primary rougher flotation samples is summarized in Table 13.28 and Figure 13.22. Data from the Plenge 2011 lock-cycle and variability flotation work is visually summarized in Figure 13.23 alongside the Primary lithology. The weighted average is calculated based on the information provided in Table 13.18 within the Ultimate Pit.

Table 13.28: Primary Material Rougher Flotation Results

			Concentrate Grades					Recovery to Concentrate				
	Primary Grind (micron)	Head Grade (Total Cu %)	Cu (%)	Fe (%)	Mo (%)	Ag (%)	Au (%)	Cu (%)	Fe (%)	Mo (%)	Ag (%)	Au (%)
Weighted Average of all Primary drill core	125	0.299	3.40	5.63	0.02	10.49	0.33	**92.87**	25.09	62.36	18.61	46.81
	175	0.299	4.14	6.68	0.03	13.58	0.36	**91.43**	23.43	57.80	18.30	41.97
	250	0.299	5.38	7.95	0.11	17.00	0.44	**86.79**	19.61	55.73	16.55	38.97
	350	0.299	6.32	8.84	0.05	16.73	0.48	**79.72**	19.22	45.49	12.35	33.83
Average of all Primary Dacitic Porphyry	125	0.333	2.96	0.70	0.01	9.00	0.30	**96.71**	9.00	62.55	21.16	54.40
	175	0.333	3.66	0.88	0.01	11.00	0.40	**94.79**	8.66	55.57	19.72	54.36
	250	0.333	4.45	1.09	0.02	13.00	0.60	**87.58**	7.57	53.07	17.77	57.09
	350	0.333	5.95	1.28	0.02	17.00	0.60	**80.49**	6.66	17.80	17.30	49.61
Average of all Primary Diorite	125	0.267	3.26	6.70	0.03	9.33	0.30	**91.71**	26.68	61.04	15.93	43.40
	175	0.267	3.94	7.99	0.03	12.33	0.31	**90.35**	24.77	56.20	15.86	37.40
	250	0.267	5.22	9.73	0.14	15.00	0.37	**86.23**	22.56	54.61	13.88	33.48
	350	0.267	5.93	10.34	0.05	14.33	0.38	**79.12**	20.00	48.21	9.42	28.23
Average of all Primary Hydrothermal Magmatic Breccia	125	1.128	10.61	6.08	0.02	40.50	1.25	**97.21**	21.76	72.47	52.99	79.09
	175	1.128	12.69	6.88	0.03	48.00	1.30	**95.65**	21.51	72.47	52.08	76.47
	250	1.128	14.60	11.00	0.03	69.50	1.35	**89.05**	29.26	61.84	55.50	67.99
	350	1.128	17.65	12.93	0.04	44.00	2.15	**82.25**	25.74	50.59	34.86	72.36
Average of all Primary Porphyry Diorite	125	0.251	3.29	5.97	0.02	13.00	0.25	**95.80**	35.22	50.46	21.17	40.15
	175	0.251	4.12	7.22	0.02	16.00	0.30	**93.88**	33.08	54.99	20.29	38.69
	250	0.251	4.96	8.73	0.02	20.50	0.45	**86.97**	31.23	51.71	20.00	41.81
	350	0.251	5.90	9.80	0.03	22.50	0.50	**80.34**	27.93	41.65	17.53	37.82
Average of all Primary Rhyodacite Porphyry	125	0.236	2.82	4.24	0.02	12.00	0.30	**94.96**	37.65	69.32	24.80	52.08
	175	0.236	3.67	4.87	0.03	16.00	0.35	**93.60**	35.57	69.14	24.49	48.55
	250	0.236	5.04	2.98	0.03	21.50	0.50	**89.39**	10.79	66.71	23.19	46.15
	350	0.236	6.20	6.78	0.04	26.50	0.60	**82.52**	29.66	63.67	20.89	42.22

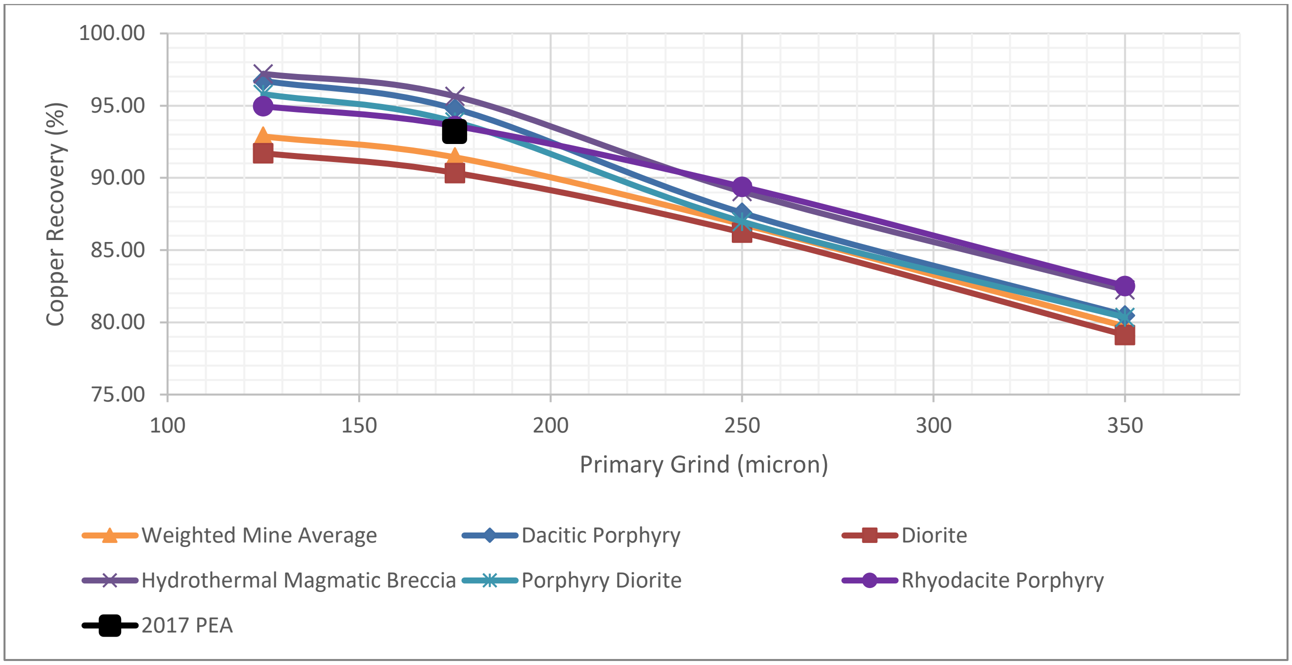


Figure 13.22: Graphic Representation of Averages of Primary Lithology Rougher Flotation Results

Figure 13.23: Graphic Representation of Primary Lithology and 2011 Rougher Flotation Results

The average rougher recovery of the Supergene material type 92.58% at 125 microns, 88.81% at 175 microns, 86.19% at 250 micron and 78.56% at 350 microns. This was at an average copper concentrate grade of 4.43%, 5.13%, 5.89%, and 6.11% respectively. Average data for all 64 Supergene rougher flotation samples is summarized in Table 13.29 and Figure 13.24. Data from the Plenge 2011 lock-cycle and variability flotation work is visually summarized in Figure 13.25 alongside the Supergene lithology. The weighted average is calculated based on the information provided in Table 13.18 within the Ultimate Pit.

Table 13.29: Supergene Material Rougher Flotation Results

			Concentrate Grades					Recovery to Concentrate				
	Primary Grind (micron)	Head Grade (Total Cu %)	Cu (%)	Fe (%)	Mo (%)	Ag (%)	Au (%)	Cu (%)	Fe (%)	Mo (%)	Ag (%)	Au (%)
Average of all Supergene drill core	125	0.583	4.43	7.62	0.03	14.96	0.46	**92.58**	44.19	66.03	27.03	57.99
	175	0.583	5.13	8.44	0.03	13.94	0.49	**88.81**	42.48	62.84	23.02	55.34
	250	0.583	5.89	9.65	0.03	16.40	0.56	**86.19**	40.21	59.36	21.78	53.05
	350	0.583	6.11	10.64	0.03	18.70	0.53	**78.56**	36.77	55.84	20.98	45.33
Average of all Supergene Dacitic Porphyry	125	1.026	2.20	6.02	0.04	12.00	0.20	**92.50**	52.98	80.62	24.24	41.55
	175	1.026	2.58	6.90	0.04	14.00	0.20	**90.94**	52.27	76.80	23.85	37.36
	250	1.026	2.98	8.63	0.04	15.00	0.30	**85.45**	51.75	74.79	20.56	40.83
	350	1.026	3.19	9.39	0.05	16.00	0.20	**77.94**	47.60	73.05	18.75	27.77
Average of all Supergene Diorite	125	0.483	4.74	8.08	0.03	16.00	0.53	**92.29**	38.58	64.26	27.40	60.93
	175	0.483	5.47	8.80	0.03	14.00	0.57	**87.60**	36.51	61.12	22.15	58.78
	250	0.483	6.32	10.01	0.03	16.67	0.63	**85.53**	33.95	57.20	21.38	54.57
	350	0.483	6.48	11.10	0.03	19.67	0.60	**77.47**	30.77	53.43	20.72	46.76
Average of all Supergene Hydrothermal Magmatic Breccia	125	0.710	8.47	5.99	0.03	17.00	0.57	**94.16**	50.51	69.74	31.48	65.02
	175	0.710	9.77	6.49	0.03	20.00	0.63	**93.18**	47.75	61.75	31.65	64.33
	250	0.710	10.61	6.82	0.03	26.50	0.60	**89.78**	45.04	62.63	25.22	61.26
	350	0.710	11.81	7.55	0.03	21.67	0.67	**84.50**	42.63	56.51	28.09	61.21
Average of all Supergene Porphyry Diorite	125	0.369	2.70	4.16	0.01	9.67	0.27	**94.81**	36.08	59.82	23.84	52.73
	175	0.369	3.56	4.60	0.02	13.00	0.37	**93.24**	31.88	57.35	22.61	52.09
	250	0.369	4.50	5.38	0.02	16.67	0.47	**88.25**	28.48	53.74	20.50	46.83
	350	0.369	5.01	6.06	0.03	17.33	0.50	**81.63**	27.16	50.94	18.93	44.61
Average of all Supergene Rhyodacite Porphyry	125	0.677	3.93	6.90	0.01	10.33	0.27	**94.53**	74.00	56.98	26.80	56.75
	175	0.677	4.74	8.49	0.01	11.33	0.27	**93.90**	74.05	55.97	25.94	51.69
	250	0.677	5.21	9.27	0.01	13.00	0.37	**91.37**	71.62	52.35	25.68	56.51
	350	0.677	5.67	9.93	0.01	14.00	0.40	**86.23**	66.86	49.09	24.09	54.94

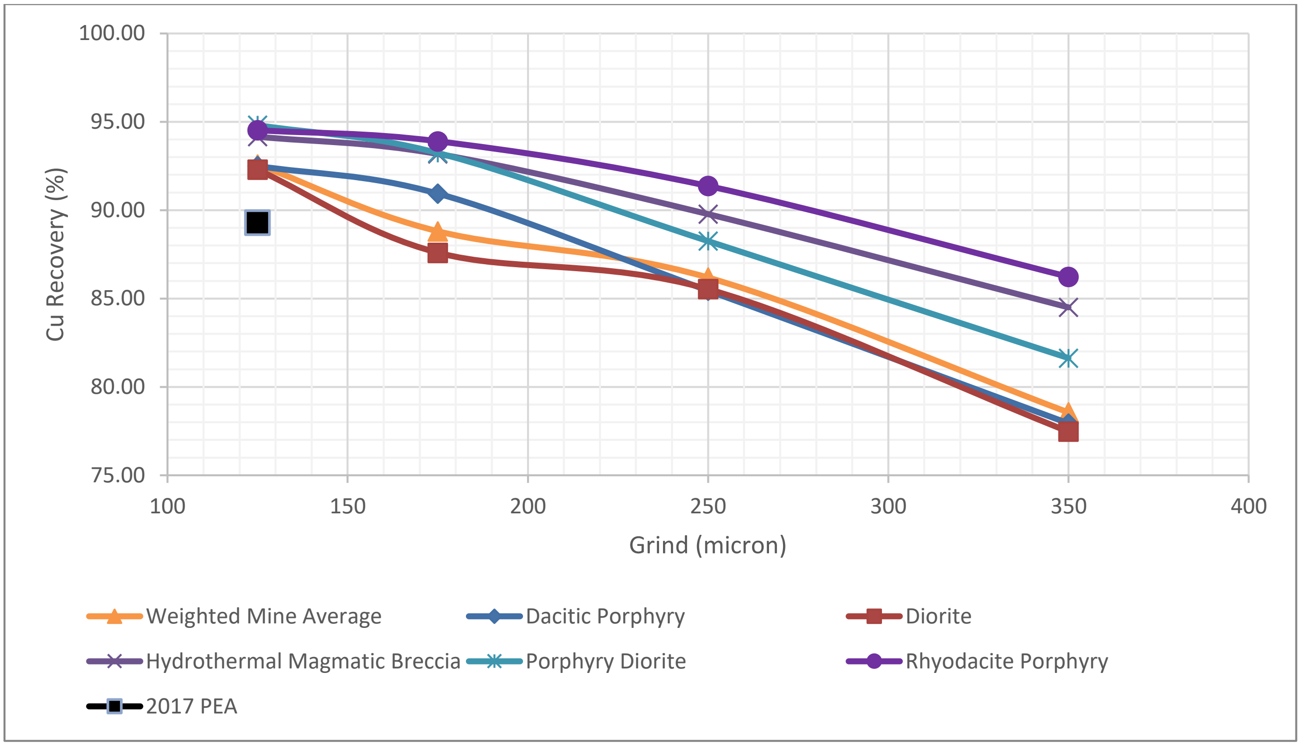


Figure 13.24: Graphic Representation of Averages of Supergene Lithology Rougher Flotation Results

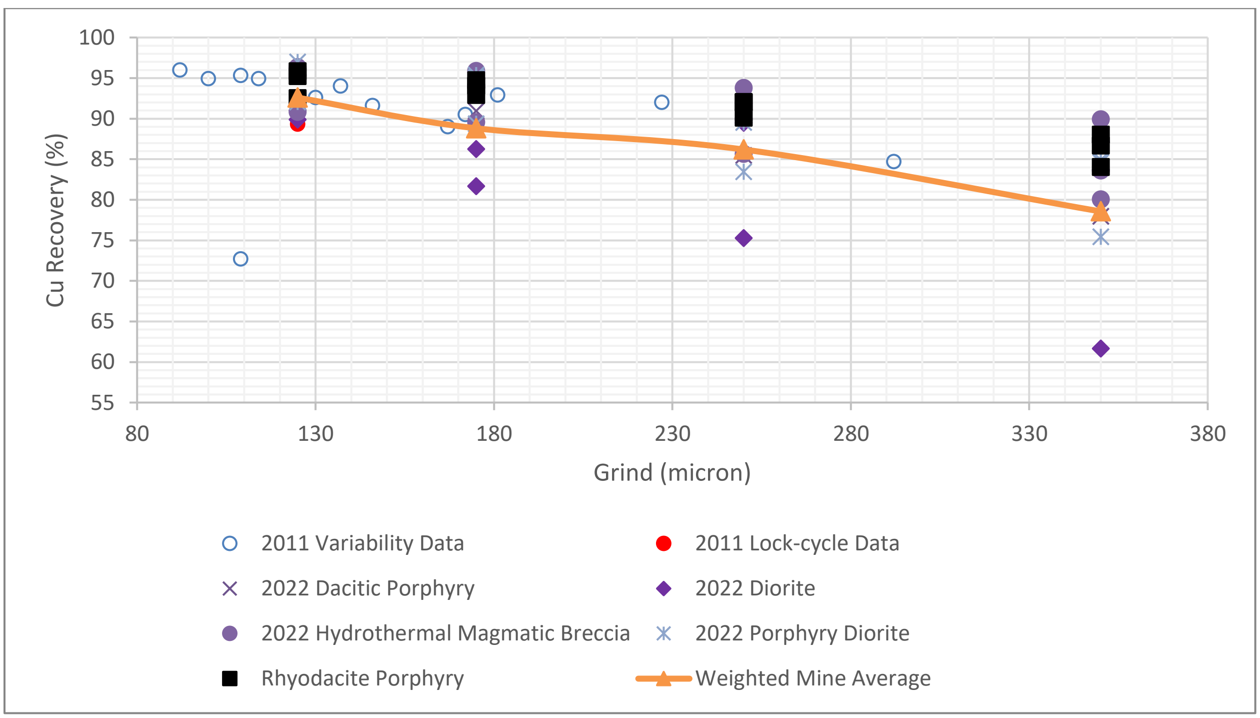


Figure 13.25: Graphic Representation of Supergene Lithology and 2011 Rougher Flotation Results

All concentrate samples were sent to SGS Lakefield in Lakefield, ON for analysis of deleterious minerals (Hg, As, Bi, Cd, Pb, Sb, Se, Te, and Ti). The results are presented in Table 13.30 for Primary and Table 13.31 for Supergene. Using the enrichment ratio and expected copper grades for Primary and Supergene materials from Table 13.12 an estimated upgrade ratio was calculated to provide an estimated grade for the deleterious minerals in the final concentrate. None of the estimated 3rd Cleaner (final) concentrates exceeded 1% in Arsenic, which if exceeded could draw penalties at a smelter. Some of the concentrates are being analyzed at SGS Lakefield for minerology to understand the relationship in the concentrate to copper mineralization. At the writing of this report, the analysis is not available.

Table 13.30: Primary Deleterious Flotation Results

		Rougher Concentrate Deleterious Mineral Grades									Estimated 3rd Cleaner Deleterious Mineral Grades								
	Primary Grind (micron)	Hg (ppm)	As (%)	Bi (ppm)	Cd (ppm)	Pb (ppm)	Sb (ppm)	Se (ppm)	Te (ppm)	Ti (ppm)	Hg (ppm)	As (%)	Bi (ppm)	Cd (ppm)	Pb (ppm)	Sb (ppm)	Se (ppm)	Te (ppm)	Ti (ppm)
Average of all Primary drill core	125	0.53	0.04	8.14	3.79	364.39	161.01	20.88	4.38	1.66	1.28	0.27	43.70	23.99	2456.10	1186.28	102.69	148.77	2.62
	175	0.61	0.05	9.66	4.48	425.66	193.28	25.06	4.44	1.65	1.04	0.20	42.84	22.90	1890.48	1200.38	99.09	92.31	1.67
	250	0.79	0.06	13.58	5.25	517.66	262.01	31.13	4.75	1.70	1.05	0.18	79.28	22.55	1699.11	1215.95	99.73	58.12	1.11
	350	0.88	0.07	16.26	6.11	564.57	312.92	38.73	5.47	1.75	0.89	0.17	47.01	21.50	1358.56	1065.17	103.58	45.37	0.92
Average of all Primary Dacitic Porphyry	125	0.40	0.05	1.70	4.00	596.00	36.00	13.00	4.00	1.40	1.29	0.23	15.25	18.86	5803.34	286.55	102.57	242.78	2.31
	175	0.50	0.06	2.10	4.70	646.00	47.70	17.00	4.00	1.40	1.32	0.20	15.20	17.00	4452.63	328.55	114.55	158.55	1.51
	250	0.60	0.06	2.20	4.50	770.00	64.30	22.00	4.00	1.40	1.29	0.17	11.29	10.55	4280.98	404.02	129.83	107.30	1.02
	350	0.70	0.07	2.70	5.60	885.00	82.20	27.00	4.00	1.50	0.98	0.12	9.53	9.15	3168.95	369.99	109.58	60.12	0.66
Average of all Primary Diorite	125	0.59	0.04	4.47	3.90	300.02	186.02	15.22	4.33	1.98	1.26	0.30	36.72	27.21	2529.46	1312.78	87.88	167.62	3.39
	175	0.68	0.04	5.44	4.69	349.62	214.32	17.78	4.33	1.94	1.02	0.18	36.58	25.67	1770.39	1368.82	78.21	100.16	1.97
	250	0.92	0.06	8.63	5.64	448.50	308.63	22.89	4.67	2.06	1.11	0.16	79.60	26.22	1534.28	1488.75	78.72	60.63	1.34
	350	0.99	0.06	9.33	6.66	435.95	369.99	27.38	4.88	2.16	0.83	0.15	38.15	26.18	1073.01	1247.20	83.12	47.50	1.22
Average of all Primary Hydrothermal Magmatic Breccia	125	0.35	0.07	34.25	3.75	408.50	194.55	62.00	5.50	0.85	0.12	0.19	90.05	7.55	759.74	786.92	152.53	19.91	0.23
	175	0.40	0.08	38.55	4.10	466.00	240.05	73.00	6.00	0.85	0.10	0.18	80.91	6.46	724.00	783.84	149.78	16.24	0.15
	250	0.50	0.10	46.65	4.50	513.00	290.45	85.00	7.00	0.80	0.09	0.18	90.90	6.05	705.81	808.04	154.22	17.05	0.09
	350	0.55	0.12	62.20	5.60	607.50	339.00	107.00	11.50	0.95	0.07	0.19	108.35	6.34	656.28	762.95	166.76	31.73	0.09
Average of all Primary Porphyry Diorite	125	0.75	0.01	4.05	3.90	762.00	68.50	15.00	4.00	1.35	3.17	0.08	32.31	14.17	4206.21	642.08	89.42	137.43	1.25
	175	0.85	0.02	5.05	4.55	926.00	87.65	19.00	4.00	1.35	2.44	0.08	34.41	13.08	3826.33	709.80	92.22	86.59	0.84
	250	1.00	0.02	10.75	5.15	1076.00	110.00	24.00	4.00	1.35	2.37	0.09	126.30	12.71	3521.21	788.35	102.65	59.75	0.59
	350	1.25	0.02	6.80	5.55	1265.50	121.75	29.00	4.00	1.35	2.49	0.08	34.27	11.59	3356.07	715.45	107.09	43.07	0.45
Average of all Primary Rhyodacite Porphyry	125	0.30	0.05	5.85	3.15	96.55	169.90	15.00	4.00	1.45	0.61	0.43	54.35	38.37	398.57	2010.49	132.79	157.17	3.08
	175	0.35	0.06	8.10	3.70	117.00	230.20	20.00	4.00	1.55	0.52	0.41	55.20	39.63	380.43	1785.47	141.53	105.68	2.73
	250	0.40	0.08	11.30	4.70	149.00	274.65	26.00	4.00	1.50	0.34	0.34	53.19	38.36	321.15	1229.81	121.80	61.68	1.67
	350	0.50	0.10	14.30	5.20	175.00	365.10	31.50	4.00	1.45	0.28	0.33	52.56	34.02	300.36	1336.55	115.70	45.42	1.16

Table 13.31: Supergene Deleterious Flotation Results

		Rougher Concentrate Deleterious Mineral Grades									Estimated 3rd Cleaner Deleterious Mineral Grades								
	Primary Grind (micron)	Hg (ppm)	As (%)	Bi (ppm)	Cd (ppm)	Pb (ppm)	Sb (ppm)	Se (ppm)	Te (ppm)	Ti (ppm)	Hg (ppm)	As (%)	Bi (ppm)	Cd (ppm)	Pb (ppm)	Sb (ppm)	Se (ppm)	Te (ppm)	Ti (ppm)
Average of all Supergene drill core	125	0.48	0.04	3.22	7.01	217.97	95.57	19.62	4.00	2.28	1.43	0.28	16.91	50.38	808.68	887.70	94.60	132.72	3.47
	175	0.52	0.04	3.79	7.80	241.95	115.40	22.54	4.08	2.33	1.38	0.25	17.37	50.49	786.15	800.14	90.90	91.86	3.06
	250	0.62	0.05	4.18	8.72	256.78	151.92	25.23	4.00	2.27	1.94	0.27	17.02	48.17	651.55	975.61	86.26	66.09	1.96
	350	0.63	0.06	4.64	9.91	275.35	170.54	28.00	4.00	2.32	1.54	0.29	18.43	54.63	629.93	1132.24	90.22	52.17	1.73
Average of all Supergene Dacitic Porphyry	125	0.40	0.01	1.40	8.80	180.00	90.70	15.00	4.00	2.00	0.30	0.13	12.24	109.16	788.05	1211.78	147.49	161.35	4.07
	175	0.40	0.01	1.70	10.60	203.00	88.00	15.00	4.00	2.20	0.22	0.11	13.09	114.96	727.48	827.92	107.05	117.11	3.57
	250	0.50	0.02	2.00	12.50	214.00	146.00	19.00	4.00	2.10	0.25	0.12	13.59	119.91	606.39	1709.34	128.82	87.84	2.44
	350	0.60	0.02	2.20	13.70	231.00	152.00	22.00	4.00	2.00	0.32	0.12	14.39	126.01	618.12	1620.81	151.10	76.84	1.94
Average of all Supergene Diorite	125	0.93	0.07	2.37	10.20	295.00	91.87	17.67	4.00	2.67	4.48	0.63	18.78	82.17	1507.35	1045.98	84.66	92.65	3.92
	175	1.07	0.07	2.70	10.93	339.33	104.17	20.33	4.00	2.63	4.70	0.59	20.12	81.32	1661.62	1047.85	83.82	78.30	3.23
	250	1.40	0.09	3.20	13.67	351.00	134.60	24.00	4.00	2.63	7.32	0.70	20.73	80.84	1238.86	1331.00	76.24	52.20	2.06
	350	1.40	0.10	3.50	14.93	390.00	167.53	24.33	4.00	2.63	5.80	0.76	21.28	94.92	1348.08	1793.03	72.99	41.83	2.00
Average of all Supergene Hydrothermal Magmatic Breccia	125	0.33	0.04	6.77	9.53	204.67	118.30	37.00	4.00	2.00	0.28	0.12	17.63	29.77	260.70	468.55	114.05	63.43	1.41
	175	0.33	0.04	7.20	10.00	218.27	105.40	38.67	4.33	2.13	0.20	0.10	15.93	24.65	215.34	288.83	95.84	45.80	1.13
	250	0.37	0.04	7.07	10.17	237.40	125.73	40.67	4.00	2.07	0.22	0.09	15.46	24.46	212.12	429.89	96.38	33.06	0.91
	350	0.33	0.04	7.93	11.80	248.53	147.57	47.67	4.00	2.13	0.14	0.10	17.35	27.55	192.83	529.06	111.12	25.32	0.78
Average of all Supergene Porphyry Diorite	125	0.30	0.04	2.20	2.07	135.53	141.07	12.33	4.00	2.03	0.82	0.33	20.81	19.19	873.44	1750.71	86.75	246.51	4.14
	175	0.30	0.05	3.27	2.20	139.17	223.57	17.33	4.00	2.03	0.60	0.29	20.72	12.66	639.57	1702.88	94.87	150.15	4.40
	250	0.37	0.07	4.37	2.77	176.00	311.83	22.67	4.00	1.93	0.44	0.28	20.73	13.43	709.27	1747.16	93.44	99.31	2.11
	350	0.37	0.08	4.43	3.43	190.33	329.97	24.00	4.00	1.90	0.33	0.30	19.15	16.46	544.77	1878.37	85.07	73.74	1.44
Average of all Supergene Rhyodacite Porphyry	125	0.37	0.01	2.17	5.63	249.33	32.67	13.00	4.00	2.50	0.49	0.09	11.97	50.80	600.12	177.54	75.32	118.74	4.20
	175	0.40	0.02	2.70	7.13	284.00	37.60	16.33	4.00	2.57	0.38	0.08	14.12	61.84	647.64	151.75	83.67	84.77	3.30
	250	0.40	0.02	2.83	7.00	277.00	37.50	15.67	4.00	2.50	0.32	0.07	12.32	50.04	461.00	149.81	64.79	72.56	2.58
	350	0.43	0.02	3.50	8.20	287.33	43.27	18.00	4.00	2.73	0.31	0.07	17.29	55.82	437.98	165.64	71.43	59.55	2.62

13.3.4 Sulfuric Acid Bottle Rolls

Bottle roll testing was completed at the SGS laboratory in Santiago. The tests were completed with 1 kg charge of material at 100% passing -10 mesh. Material charges were placed in a 10L bottle with a leach solution at pH 1.5 and a slurry density of 1:2 (w/w solid liquid). A total of 114 bottle rolls were completed at SGS. This testing spanned 38 different samples (9 composites, 29 drill hole samples) with three (3) bottles at the same conditions except varying additions of Fe at 0, 2, and 5 g/L.

Bottle rolls pH was adjusted and maintained at 1.5 pH with control checks at 2, 4, 6, 12, 24, 48, 72, and 96 hours. After 96 hours, the tests were completed with the solution/solids filtered and analyzed.

Two main parameters were to be demonstrated from the work: maximum acid consumption; and maximum acid soluble copper recovery.

The summary of the Supergene results is presented in Table 13.32 and Primary results are presented in Table 13.33. A summary of the composite test acid bottle rolls is presented in Table 13.32 through Table 13.42 with AZ-1285 presented in Table 13.43 and AZ-0946 presented in Table 13.44.

Table 13.32: Supergene Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Average	0	0.586	0.590	0.52	0.492	18.37	20.7	19.1	37.9
	2	0.586	0.565	0.96	0.379	31.61	13.0	10.0	15.8
	5	0.586	0.497	1.38	0.226	41.32	8.4	4.2	11.1
Average Diorite	0	0.445	0.468	0.40	0.392	18.23	26.3	25.1	76.4
	2	0.445	0.442	0.62	0.321	26.84	11.9	9.8	33.6
	5	0.445	0.351	1.02	0.151	34.25	10.8	7.8	36.8
Average Dacitic Porphyry	0	0.222	0.235	0.20	0.198	17.74	27.1	26.5	67.3
	2	0.222	0.260	0.33	0.202	27.02	11.1	10.1	16.9
	5	0.222	0.233	0.35	0.160	31.35	4.7	3.6	5.2
Average Rhyodacite Porphyry	0	0.554	0.545	0.70	0.414	24.44	20.5	18.4	13.2
	2	0.554	0.543	1.17	0.318	41.35	12.9	9.4	4.1
	5	0.554	0.560	1.53	0.254	54.51	6.7	2.0	0.7
Average Hydrothermal Magmatic Breccia	0	1.115	1.100	0.86	0.944	15.10	19.7	17.0	11.6
	2	1.115	1.088	1.73	0.755	30.15	16.9	11.7	3.9
	5	1.115	0.875	2.69	0.357	46.43	10.9	2.8	0.5
Average Porphyry Diorite	0	0.351	0.365	0.239	0.32	15.93	14.4	13.5	40.9
	2	0.351	0.290	0.540	0.18	29.65	10.9	9.2	21.4
	5	0.351	0.292	0.640	0.17	33.42	6.4	4.5	8.2

Table 13.33: Primary Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Average Primary	0	0.369	0.369	0.15	0.345	9.50	27.2	26.7	278.7
	2	0.369	0.352	0.29	0.303	16.89	19.9	19.0	125.7
	5	0.369	0.146	0.33	0.054	19.28	7.4	6.4	50.1
Average Diorite	0	0.246	0.240	0.17	0.211	12.35	26.1	25.6	247.1
	2	0.246	0.236	0.28	0.185	20.94	22.1	21.3	122.5
	5	0.246	0.159	0.32	0.048	23.77	7.9	6.9	52.9
Average Dacitic Porphyry	0	0.333	0.340	0.03	0.339	1.56	26.3	26.2	503.5
	2	0.333	0.333	0.04	0.338	2.16	23.1	23.0	320.0
	5	0.333	0.007	0.04	0.000	2.07	4.4	4.3	62.8
Average Rhyodacite Porphyry	0	0.299	0.302	0.17	0.273	12.13	21.5	21.0	75.6
	2	0.299	0.299	0.36	0.233	24.50	19.6	18.5	32.2
	5	0.299	0.171	0.41	0.087	29.04	4.8	3.5	4.6
Average Hydrothermal Magmatic Breccia	0	1.128	1.145	0.26	1.114	5.58	35.7	34.9	67.3
	2	1.128	1.167	0.61	1.065	12.70	17.3	15.5	13.7
	5	1.128	1.109	0.68	0.960	14.35	11.2	9.1	7.5
Average Porphyry Diorite	0	0.251	0.256	0.03	0.256	1.92	29.6	29.6	723.5
	2	0.251	0.125	0.03	0.121	2.61	11.1	11.0	248.5
	5	0.251	0.138	0.04	0.131	2.88	5.5	5.4	119.0

Table 13.34: Composite 1 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Composite 1	0	0.047	0.050	0.086	0.033	36.08	4.7	4.4	25.9
	2	0.047	0.048	0.103	0.028	43.11	4.2	3.9	19.2
	5	0.047	0.049	0.118	0.028	45.27	4.0	3.6	17.1

Table 13.35: Composite 2 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Composite 2	0	0.019	0.021	0.024	0.016	25.01	11.6	11.6	243.3
	2	0.019	0.018	0.030	0.012	30.64	4.7	4.7	79.9
	5	0.019	0.020	0.029	0.015	27.17	4.5	4.4	85.7

Table 13.36: Composite 3 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Composite 3	0	0.123	0.124	0.304	0.068	46.14	16.8	16.0	28.1
	2	0.123	0.124	0.359	0.057	54.68	7.8	6.7	10.0
	5	0.123	0.124	0.370	0.051	59.70	7.7	6.5	8.9

Table 13.37: Composite 4 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Composite 4	0	0.45	0.481	0.616	0.374	25.28	12.6	10.9	9.6
	2	0.45	0.478	1.015	0.293	42.51	10.9	8.0	4.2
	5	0.45	0.470	1.073	0.273	45.16	11.6	8.5	4.2

Table 13.38: Composite 5 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Composite 5	0	0.405	0.409	0.531	0.306	25.68	17.1	15.5	14.9
	2	0.405	0.403	0.935	0.222	44.87	11.5	8.7	4.8
	5	0.405	0.394	1.043	0.203	49.18	8.3	5.2	2.6

Table 13.39: Composite 6 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Composite 6	0	0.475	0.465	0.361	0.397	15.36	15.6	14.4	19.8
	2	0.475	0.453	0.872	0.281	37.12	9.4	6.7	3.8
	5	0.475	0.461	1.081	0.249	45.43	7.6	4.3	2.0

Table 13.40: Composite 7 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Composite 7	0	0.238	0.247	0.027	0.246	2.06	15.9	15.8	322.9
	2	0.238	0.241	0.028	0.244	2.40	6.0	5.9	103.2
	5	0.238	0.249	0.031	0.248	2.66	6.0	5.9	93.4

Table 13.41: Composite 8 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Composite 8	0	0.375	0.353	0.142	0.332	7.26	18.0	17.6	64.7
	2	0.375	0.351	0.227	0.309	12.79	2.9	2.1	4.4
	5	0.375	0.350	0.281	0.290	16.88	7.2	6.2	9.9

Table 13.42: Composite 9 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
Composite 9	0	0.578	0.555	0.455	0.462	17.09	17.2	15.6	15.8
	2	0.578	0.551	0.824	0.383	31.30	7.2	4.4	2.4
	5	0.578	0.545	0.967	0.372	31.95	9.3	6.5	3.5

Table 13.43: AZ-1285 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
AZ-1285	0	0.186	0.197	0.014	0.197	1.48	31.8	31.7	1154.0
	2	0.186	0.180	0.016	0.18	1.71	16.2	16.1	506.2
	5	0.186	0.005	0.025	0	2.61	8.8	8.8	180.5

Table 13.44: AZ-0946 Bottle Roll Results									
Sample	Ferric Addition	Copper (%)				Recovery	H_2SO_4 Consumption		
	Fe^{+2} (g/L)	Assayed Head CuT (%)	Calculated Head CuT (%)	Cu in Solution (%)	Tail Assay CuT (%)	CuT Recovery (%)	Gross kg/t	Net kg/t	Specific (Gangue) kg H_2SO_4 / kg Cu
AZ-0946	0	0	1.026	1.010	0.633	12.27751	30.48	28.5	22.7
	2	2	1.026	0.986	1.089	19.51744	16.07	13.0	6.5
	5	5	1.026	0.311	1.563	30.3056	6.70	1.9	0.6

13.3.5 Sulfuric Acid Column Bio-leach

Compositing of material from Table 13.21 was completed at SGS in 2022 based on creating graded material types to compare to the Plenge data. Additionally, based on additional material present, AZ-1285 and AZ-0946 were selected for column testing.

The composite columns are currently in progress and results to date are considered preliminary. Material for the column tests were crushed to 100% passing 19 mm (3/4 inch) and 12.7 mm (1/2 inch) for size recovery. All columns with 19 mm material were conducted in 152.4 mm I.D. by 3 m tall columns containing approximately 75 kg of material. All columns with 12.7 mm material were conducted in 101.6 mm I.D. by 3 m tall columns containing approximately 35 kg of material. Column leach testing is being completed in a closed circuit with solvent extraction (SX). A synthetic raffinate was used to start the column leach, comprising of Santiago tap water, 5 g/L sulfuric acid, and 2 g/L ferric.

Column material standing moisture was approximately 2%, whereas SGS agglomerated the material to 5% moisture using the synthetic raffinate before being placed in each column. Columns were allowed to rest for two (2) days before being fed with the raffinate. The raffinate is being added at a rate of 6 $L/hr/m^2$ with biomass inoculated into the raffinate solution to the columns. The active biomass was produced from an already active SGS culture to simulate a mature bio-leaching system.

All 21 columns were started on October 1st, 2022, running at ambient temperature (average temperature of columns was 18.6 C when loaded) and without air addition. Solution breakthrough was completed after two (2) days for the 12.7 mm material and three (3) days after the 19 mm material. Acid is added to the columns when the pH arises above 2.2, however the columns are maintained at a pH between 1.8 and 2.2.

Additionally, 90 kg samples from Composite 6, Composite 7, Composite 8, and Composite 9 from this program were shipped and delivered to Melbourne, Australia in early November 2022 to undertake alternative leaching techniques.

The head assays for each of the columns prepared, is presented in Table 13.45.

Table 13.45: Column Head Assays											
Composite Number	**Column Number**	**Size Fraction**	**Material Type**	**CuT (%)**	**CuAS (%)**	**CuCN (%)**	**CuSOL (%)**	**CuSOL/CuT Ratio (%)**	**Fe (%)**	**Au (g/t)**	**S (%)**
Composite 1	Column 1	P100% - 19mm	Oxide/LIX	0.047	0.005	0.021	0.026	55.32	1.177	0.12	0.28
	Column 12	P100% - 12.7mm	Oxide/LIX								
Composite 2	Column 2	P100% - 19mm	Oxide/LIX	0.019	0.002	0.006	0.008	44.11	1.587	0.05	0.28
	Column 13	P100% - 12.7mm	Oxide/LIX								
Composite 3	Column 3	P100% - 19mm	Oxide/LIX	0.123	0.016	0.046	0.062	50.41	1.956	0.30	0.81
	Column 14	P100% - 12.7mm	Oxide/LIX								
Composite 4	Column 4	P100% - 19mm	Supergene	0.45	0.024	0.272	0.296	65.78	2.329	0.06	0.70
	Column 15	P100% - 12.7mm	Supergene								
Composite 5	Column 5	P100% - 19mm	Supergene	0.405	0.018	0.22	0.238	58.77	1.894	0.03	0.81
	Column 16	P100% - 12.7mm	Supergene								
Composite 6	Column 6	P100% - 19mm	Supergene	0.475	0.018	0.254	0.272	57.26	1.897	0.03	0.74
	Column 17	P100% - 12.7mm	Supergene								
Composite 7	Column 7	P100% - 19mm	Primary	0.238	0.003	0.018	0.021	8.82	2.591	0.04	1.45
	Column 18	P100% - 12.7mm	Primary								
Composite 8	Column 8	P100% - 19mm	Primary	0.375	0.008	0.075	0.083	22.13	1.828	0.05	1.17
	Column 19	P100% - 12.7mm	Primary								
Composite 9	Column 9	P100% - 19mm	Primary	0.578	0.021	0.232	0.253	43.77	2.076	0.27	1.02
	Column 20	P100% - 12.7mm	Primary								
AZ-1285	Column 10	P100% - 19mm	Supergene	0.144	0.003	0.018	0.021	14.58	2.361	<0.03	0.84
	Column 21	P100% - 12.7mm	Supergene								
AZ-0946	Column 11	P100% - 19mm	Primary	1.026	0.024	0.471	0.495	48.25	1.674	0.12	0.80

A screen analysis was completed for all the column composites with sequential assays by size fraction. The 19mm column head screen analysis is projected in a graphical representation is included for the soluble copper in Figure 13.26 and for total copper in Figure 13.27. The 12.7mm column head screen analysis is projected in a graphical representation is included for the soluble copper in Figure 13.28 and for total copper in Figure 13.29.

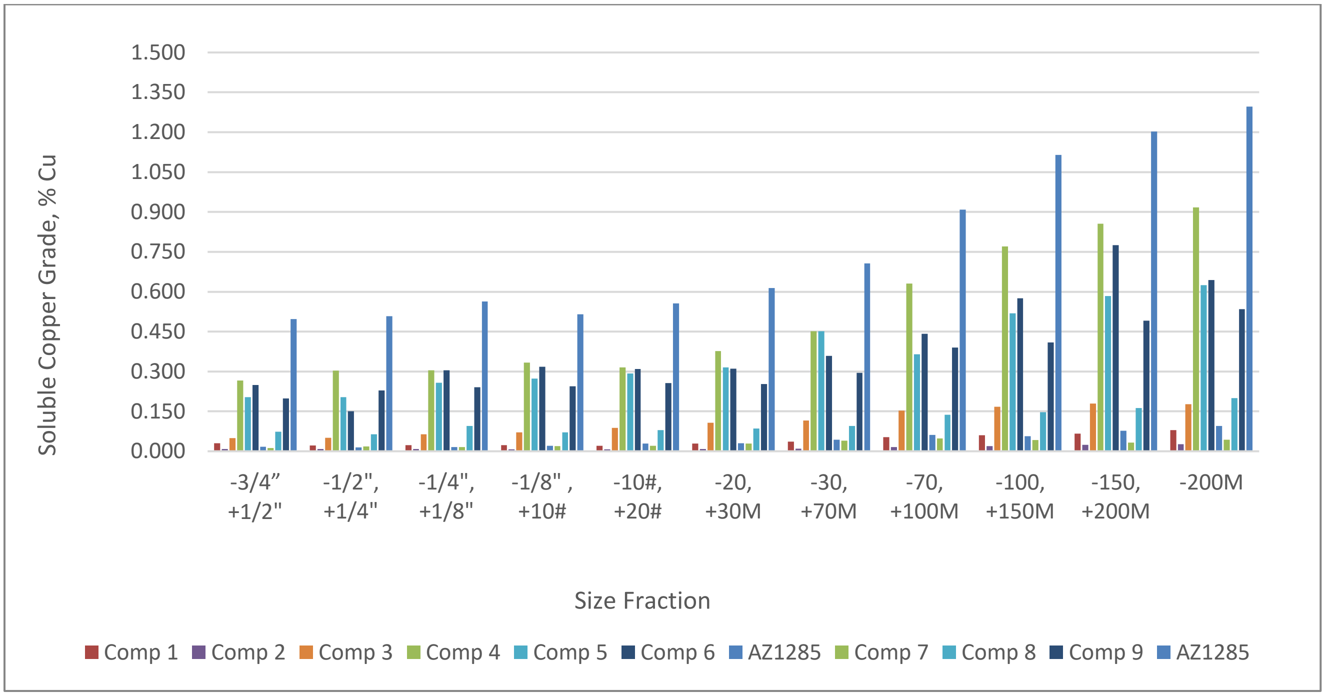


Figure 13.26: 19mm Head Screen Analysis of Soluble Copper

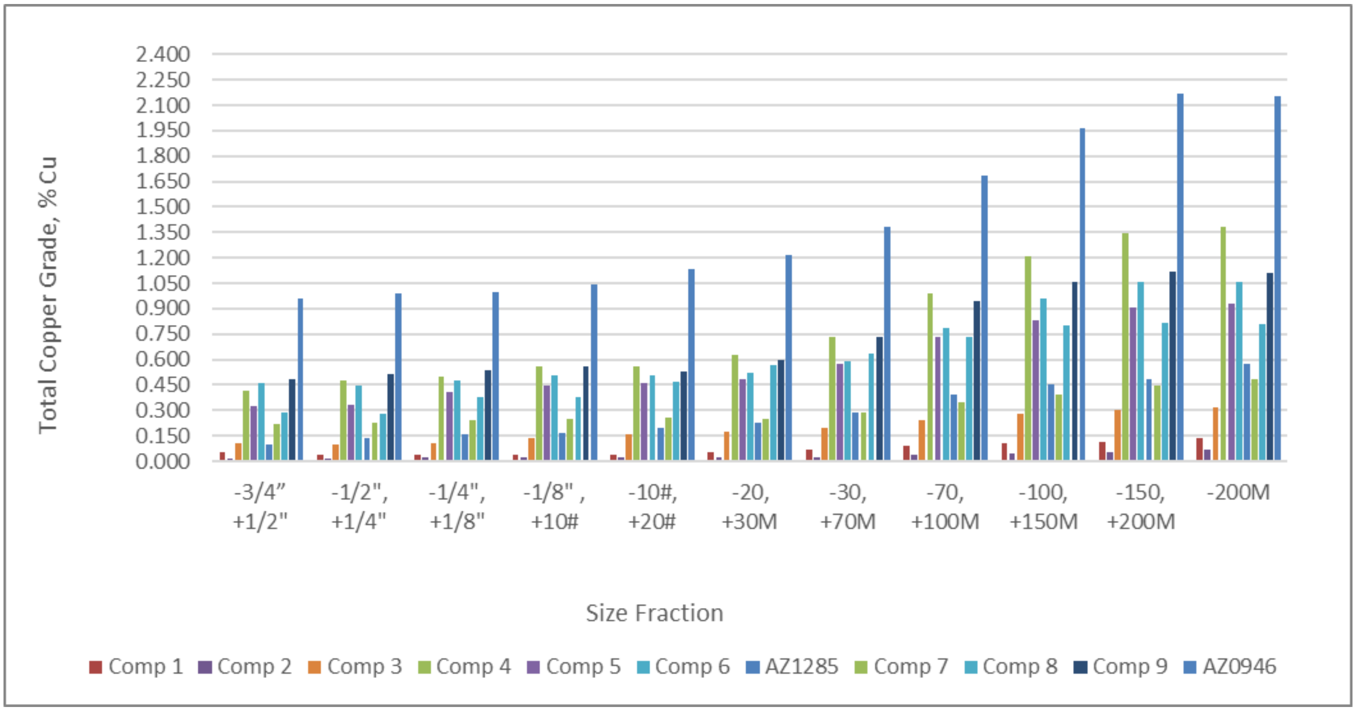


Figure 13.27: 19mm Head Screen Analysis of Total Copper

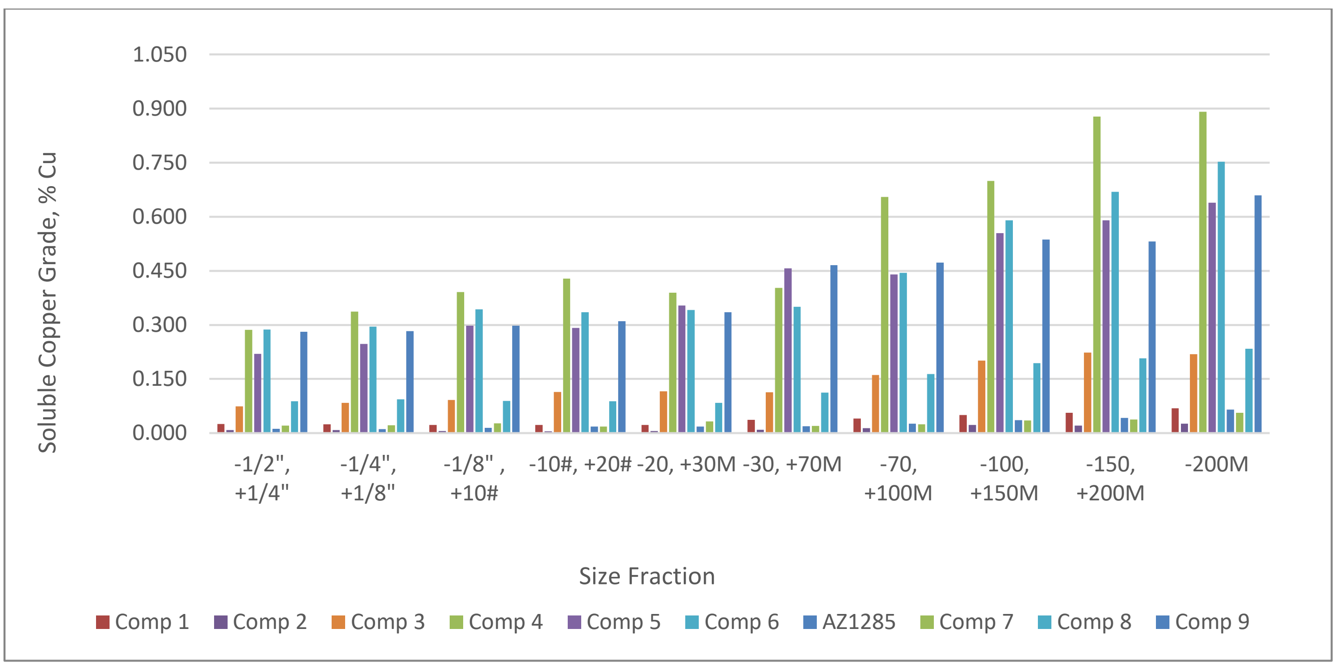


Figure 13.28: 12.7mm Head Screen Analysis of Soluble Copper

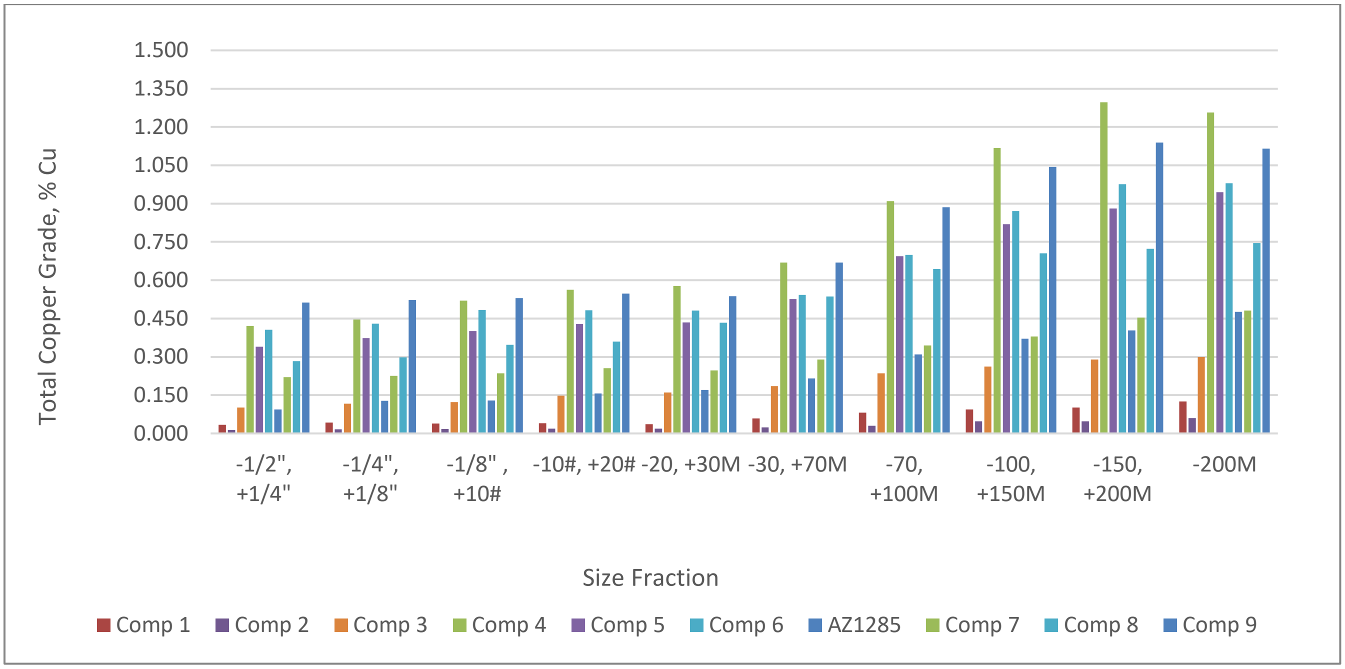


Figure 13.29: 12.7mm Head Screen Analysis of Total Copper

As of the effective date of this report, the Phase 1 columns were completed and results from leaching on a solution and head assay basis has been reported by SGS. Column residues were being prepared for analysis. Until tails assays are confirmed, copper recovery based on head assays are only indicative and significant variations may be present. Fitted logarithmic curves are included with each figure to represent the best approximation of copper recovery in the future.

A breakdown of copper recovery for soluble copper (CuAS + CuCN), and total copper (CuT) recovery by column is presented in Figure 13.30 through Figure 13.40. The copper extraction recovery calculations are based on solution assays received and the head assays completed.

Due to the very low-grade nature of Composites 1 and 2 were observed once head assays were completed, both samples were tested for a total of 30 days prior to being taken offline and washed. All four (4) of these columns will have the tails screened, Toxicity Characterization Leaching Procedure (TCLP) for eight (8) metals (As, Ba, Cd, Cr, Pb, Se, Ag, Hg) with analysis of final leach residues and humidity cell testing.

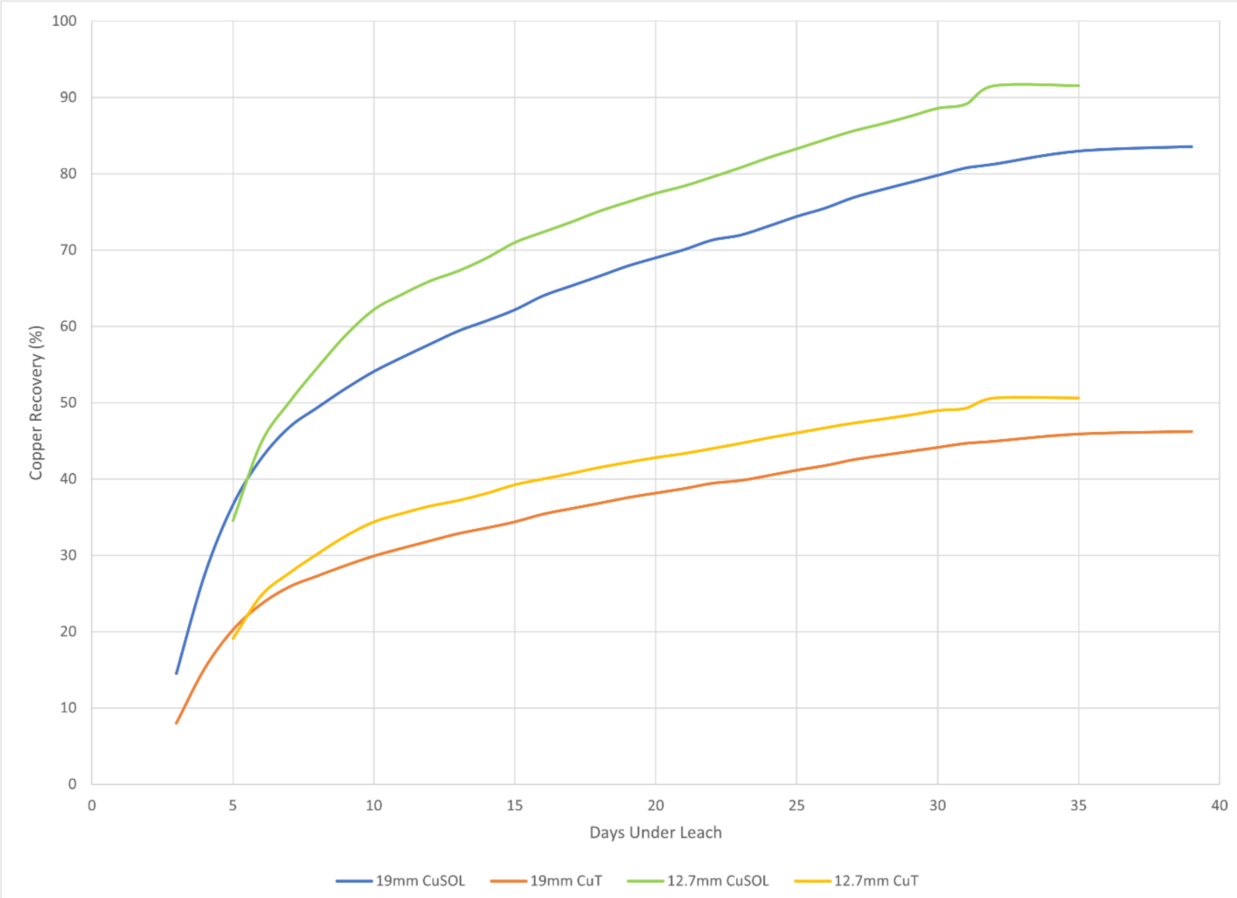


Figure 13.30: Composite 1 Copper Extractive Recovery (Soluble and Total)

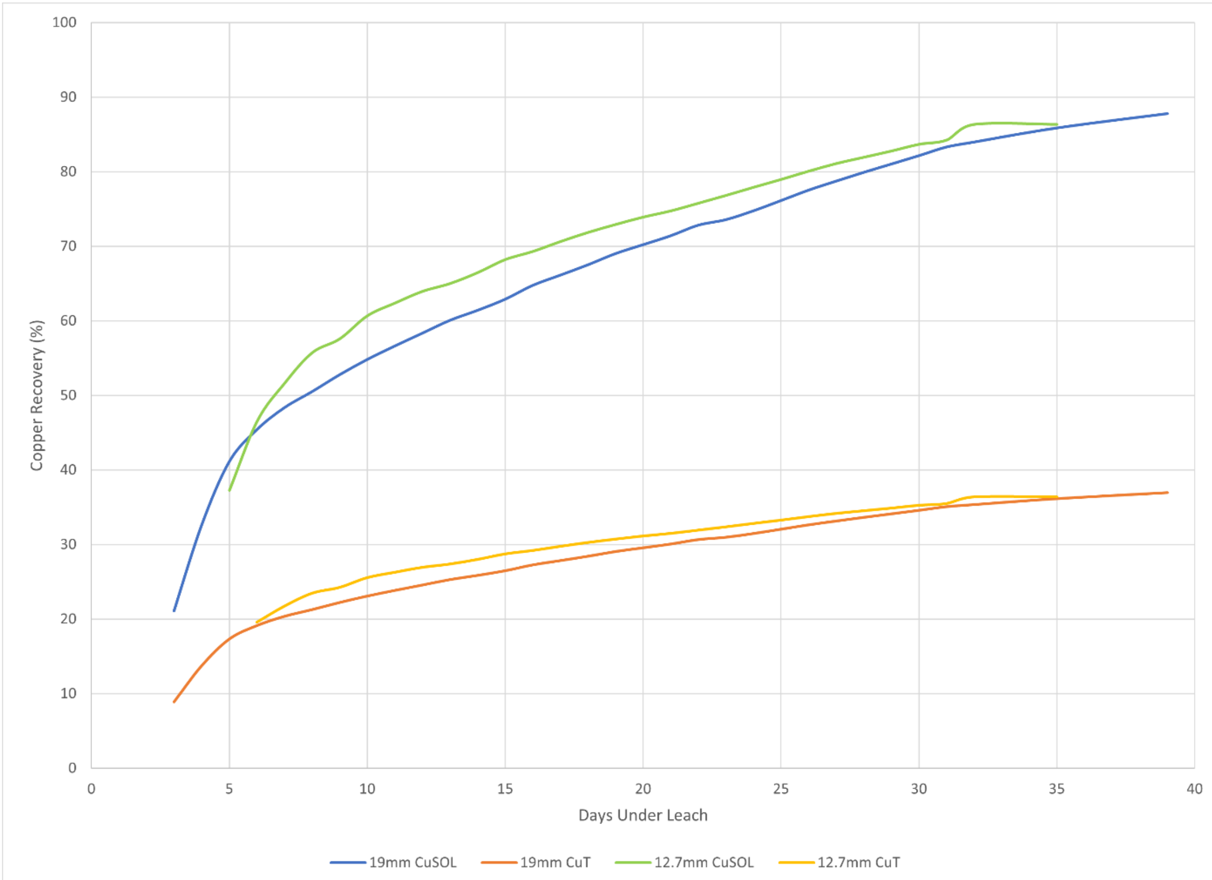


Figure 13.31: Composite 2 Copper Extractive Recovery (Soluble and Total)

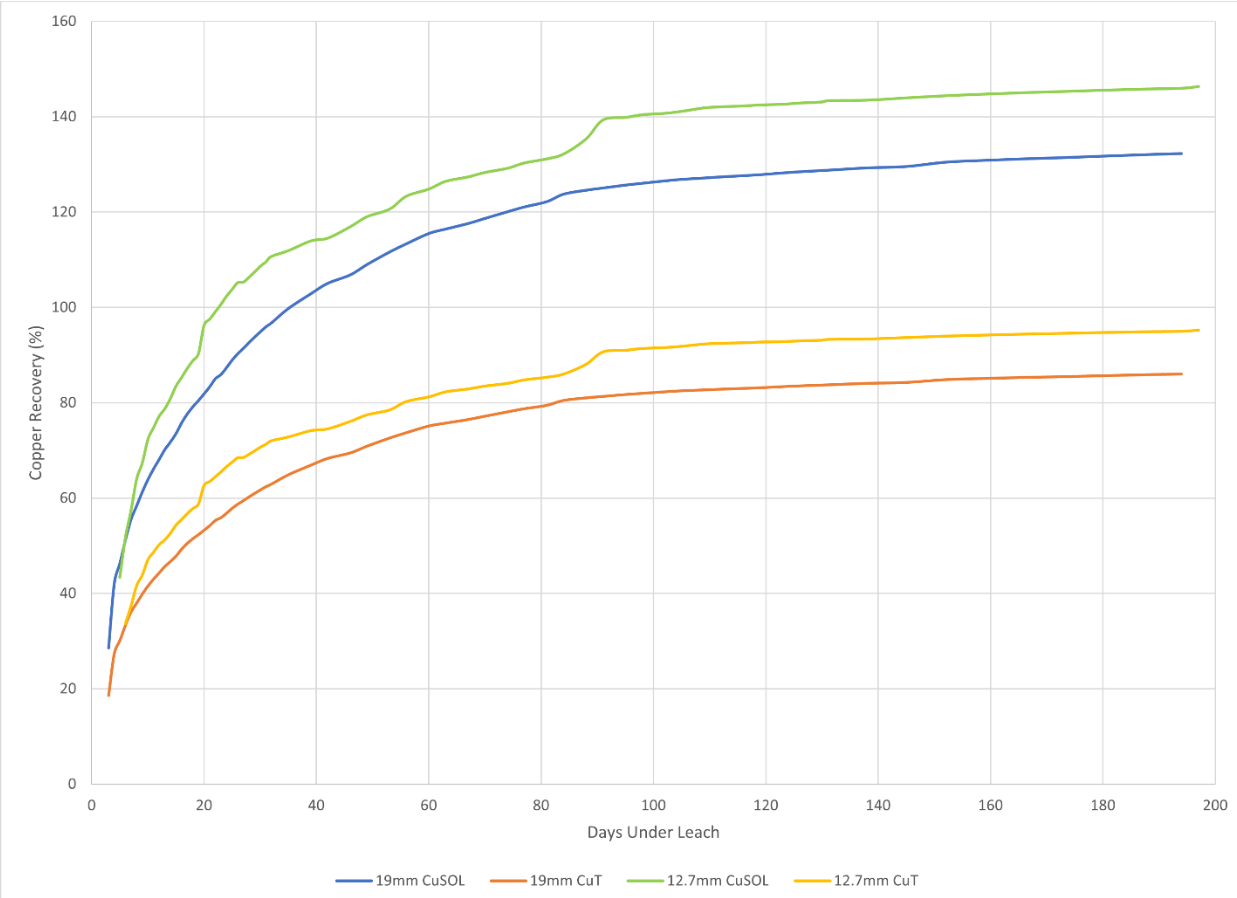


Figure 13.32: Composite 3 Copper Extractive Recovery (Soluble and Total)

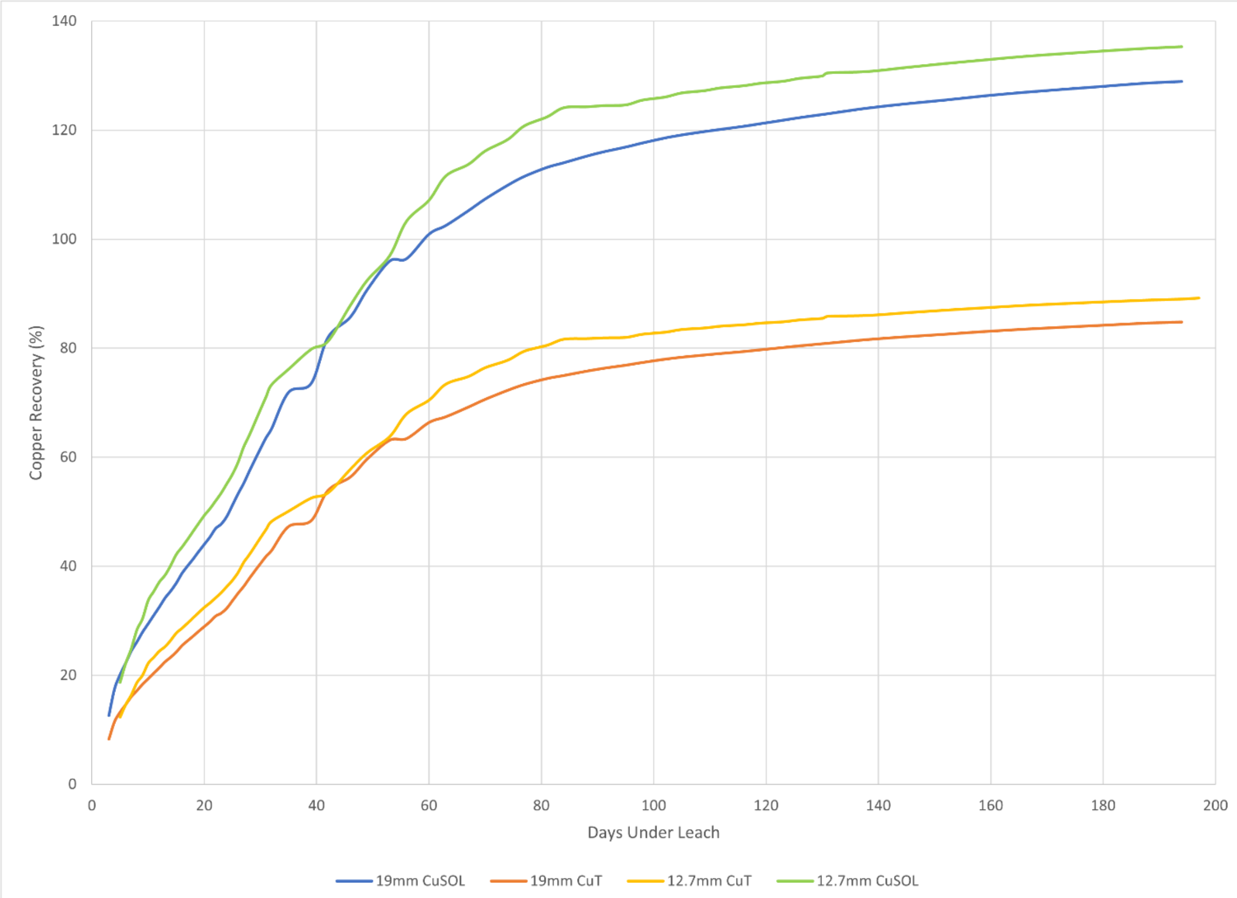


Figure 13.33: Composite 4 Copper Extractive Recovery (Soluble and Total)

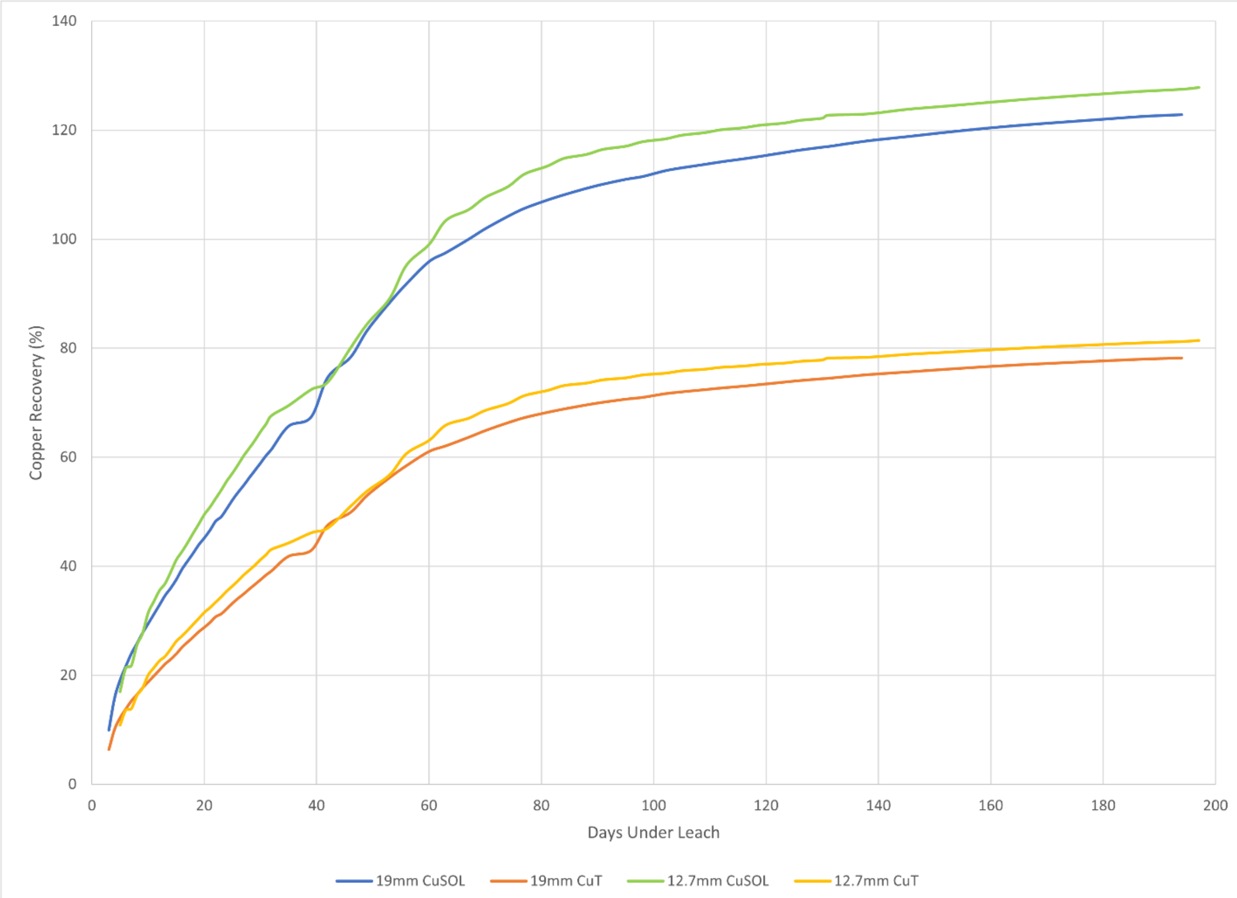


Figure 13.34: Composite 5 Copper Extractive Recovery (Soluble and Total)

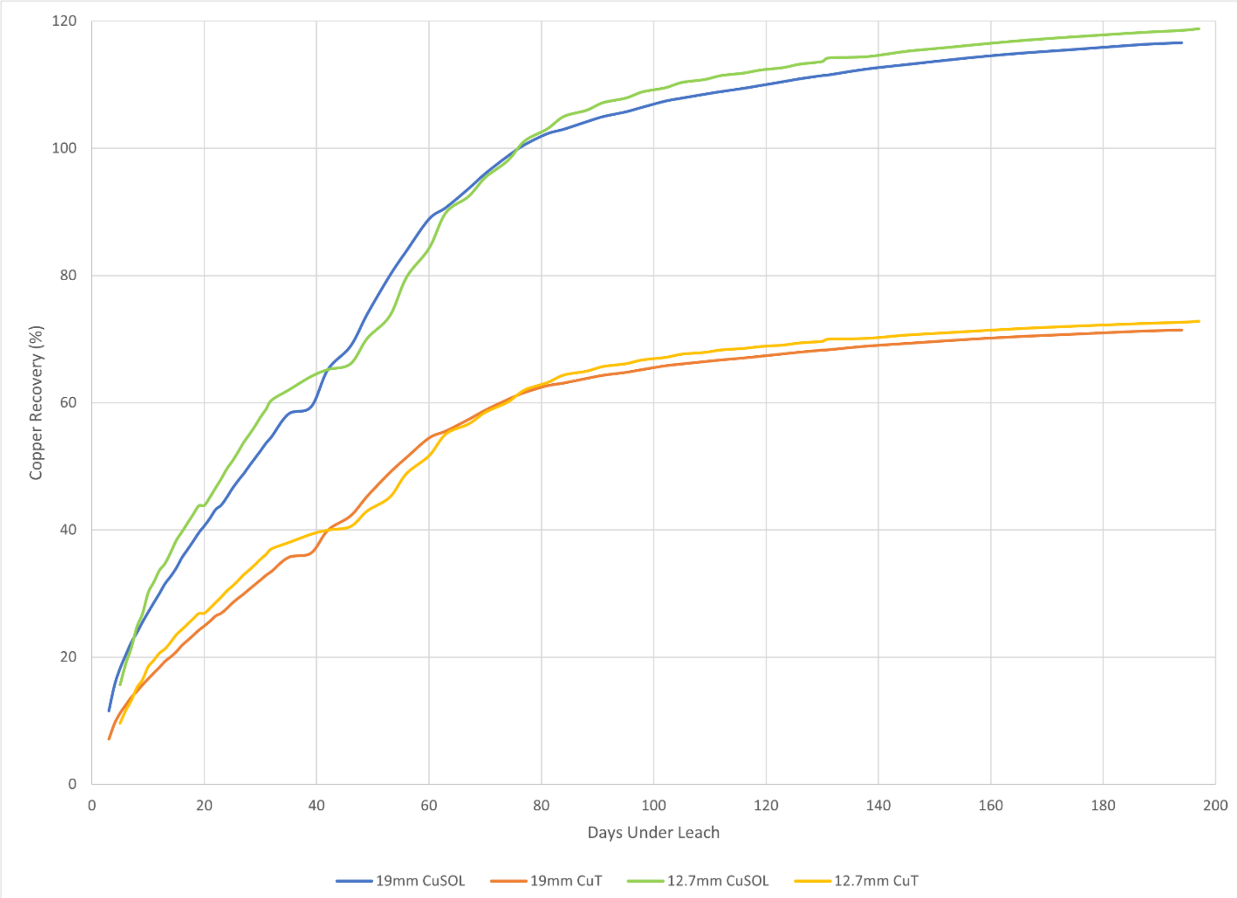


Figure 13.35: Composite 6 Copper Extractive Recovery (Soluble and Total)

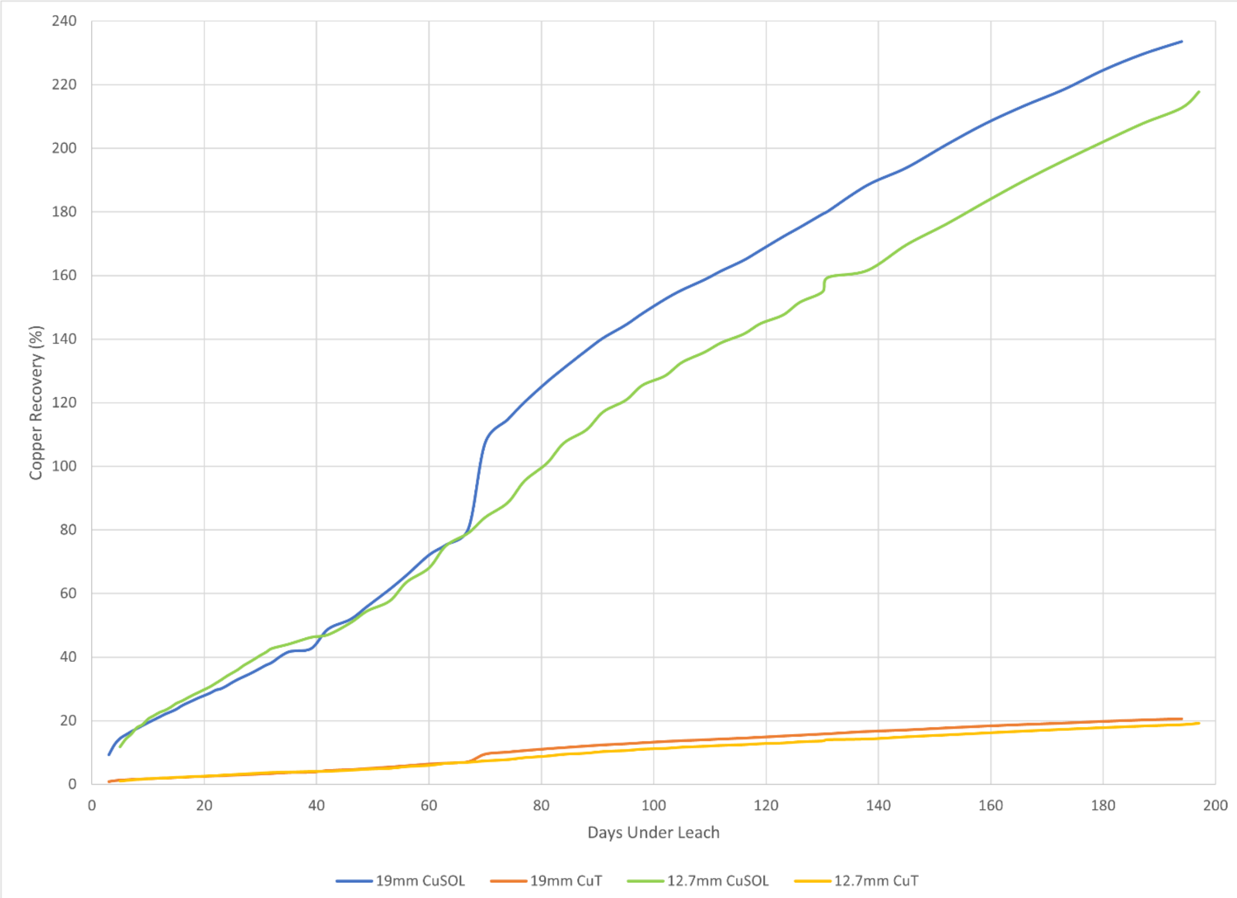


Figure 13.36: Composite 7 Copper Extractive Recovery (Soluble and Total)

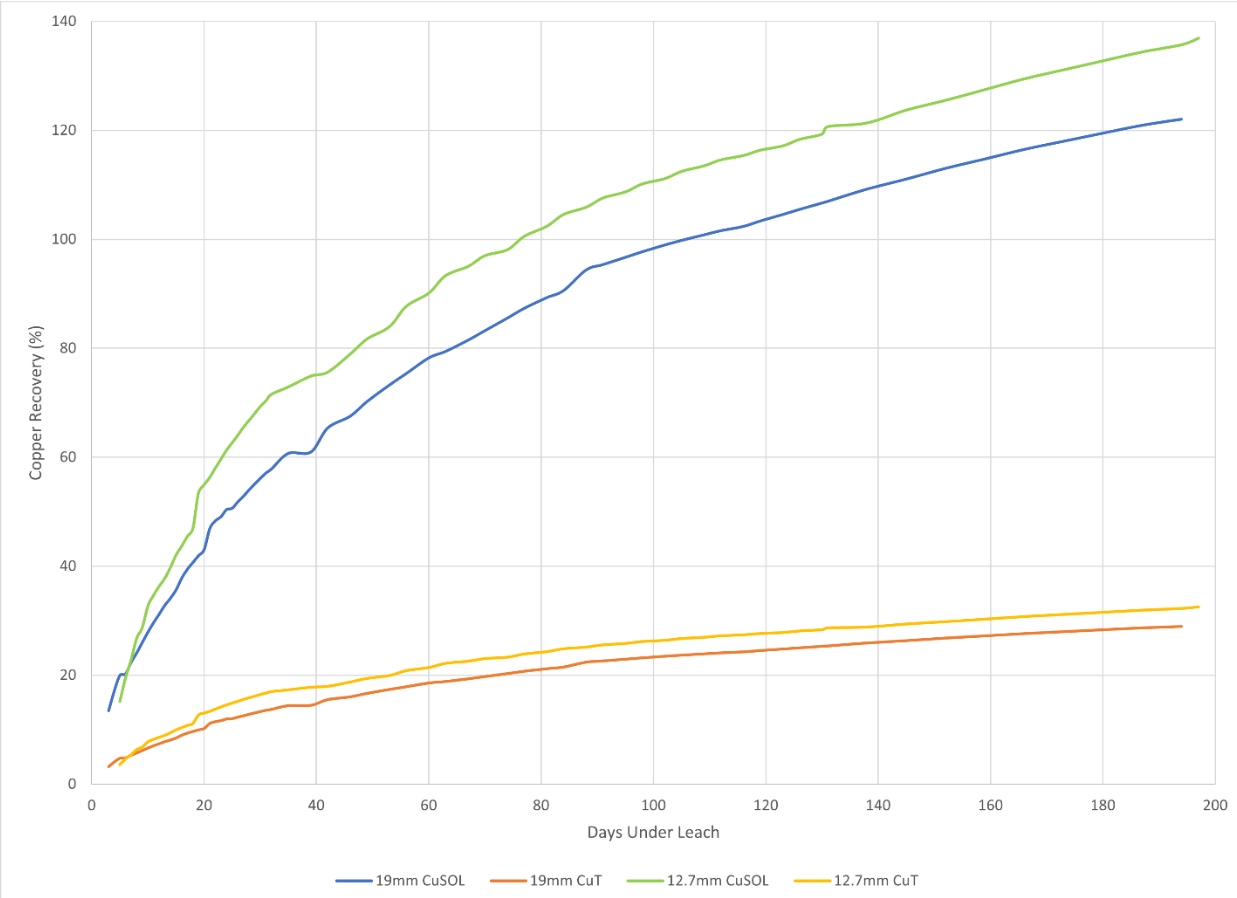


Figure 13.37: Composite 8 Copper Extractive Recovery (Soluble and Total)

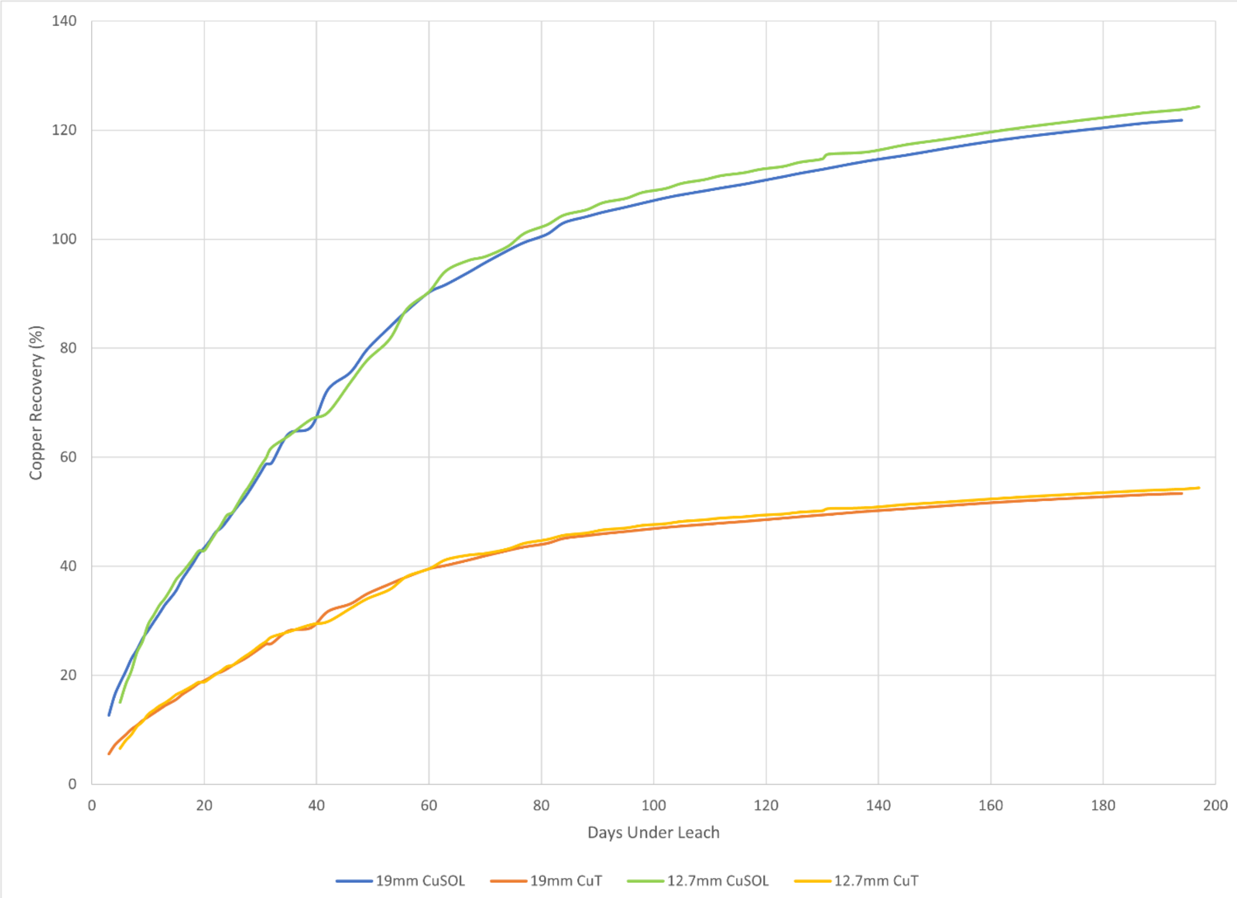


Figure 13.38: Composite 9 Copper Extractive Recovery (Soluble and Total)

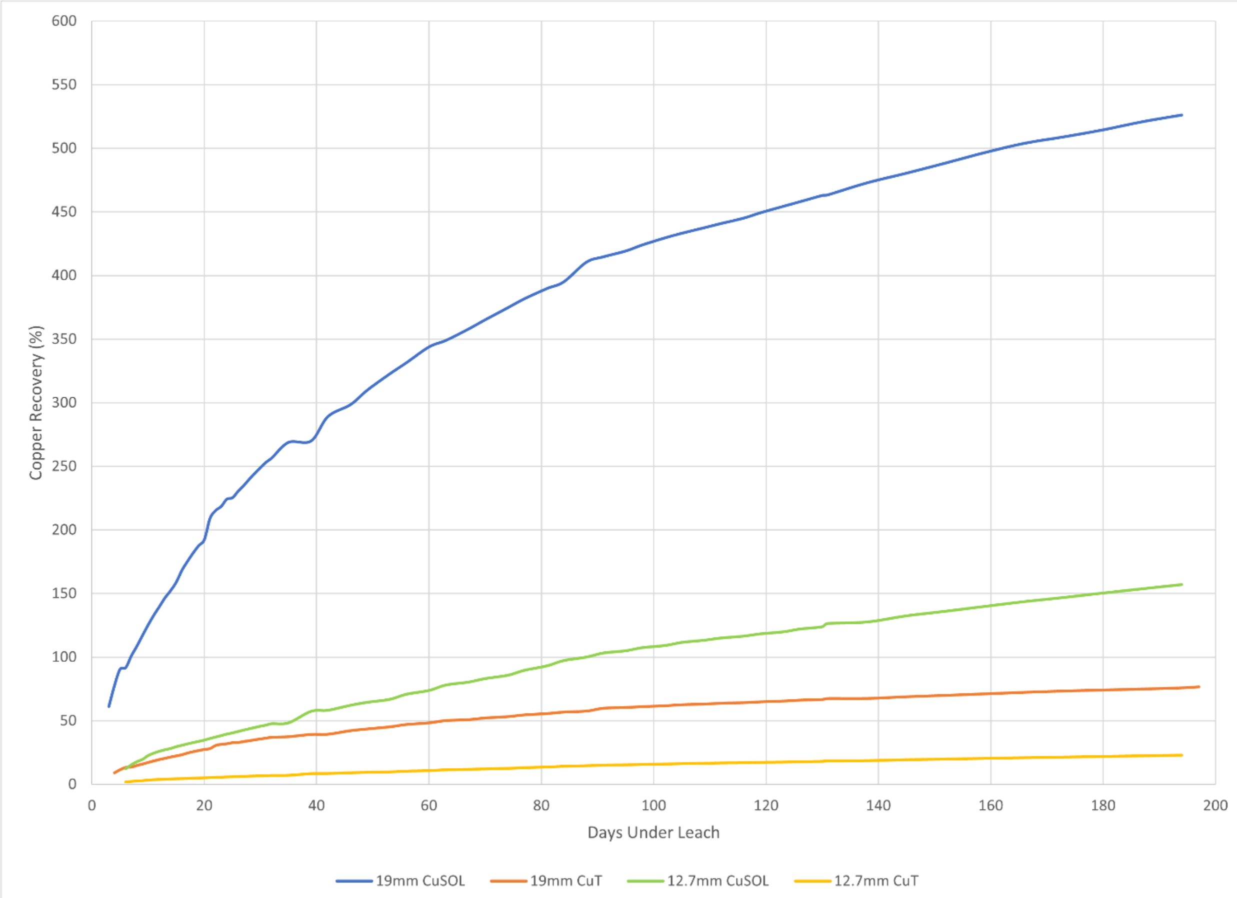


Figure 13.39: AZ-1285 Copper Extractive Recovery (Soluble and Total)

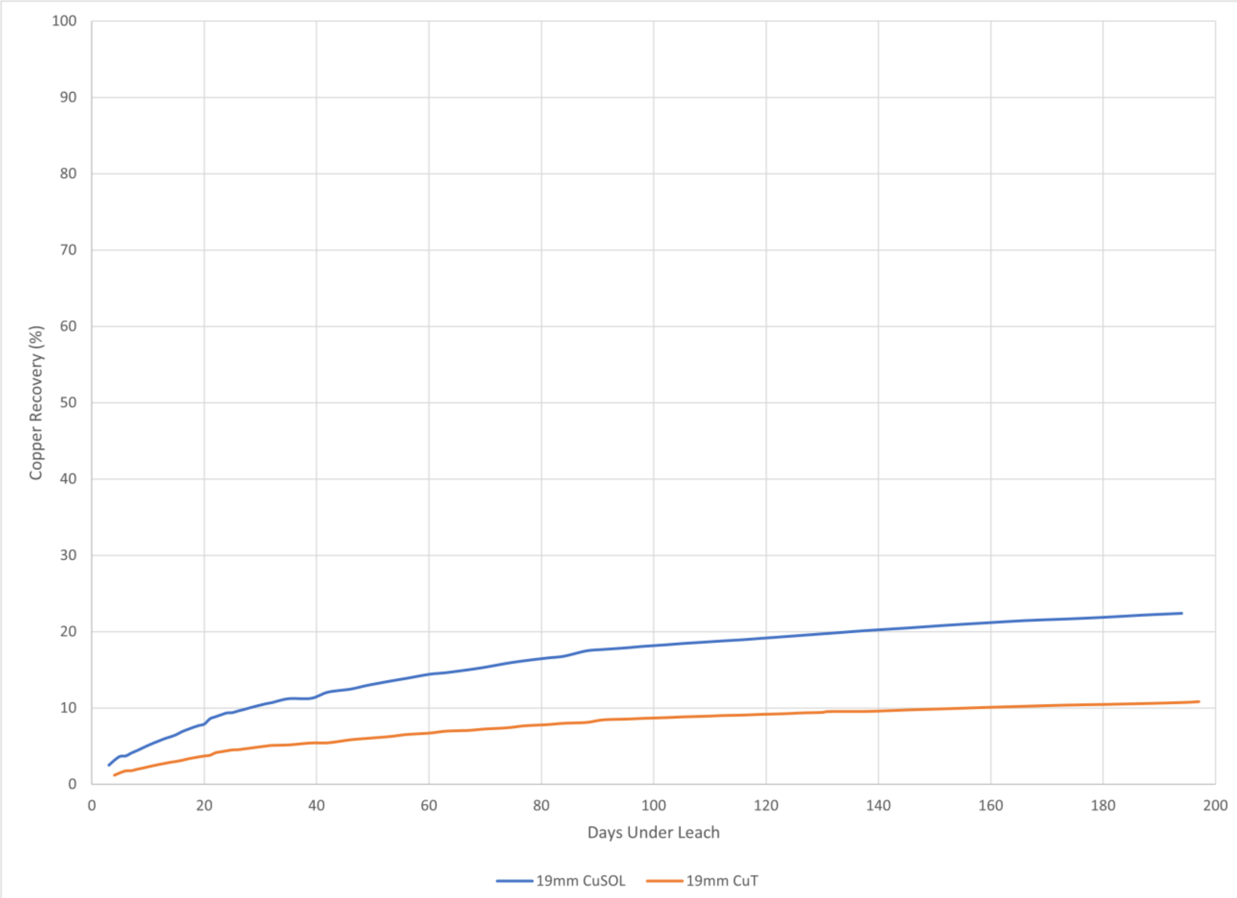


Figure 13.40: AZ-0946 Copper Extractive Recovery (Soluble and Total)

Oxidation-reduction potential (ORP) and redox potential (Eh) was measured to understand how the bacteria are functioning in the columns, the Eh results for the 19 mm columns is shown in Figure 13.41 and the 12.7 mm is shown in Figure 13.42. Eh measurements more than 600mV/ENH indicate strong iron and copper sulfide oxidation is occurring in the test columns.

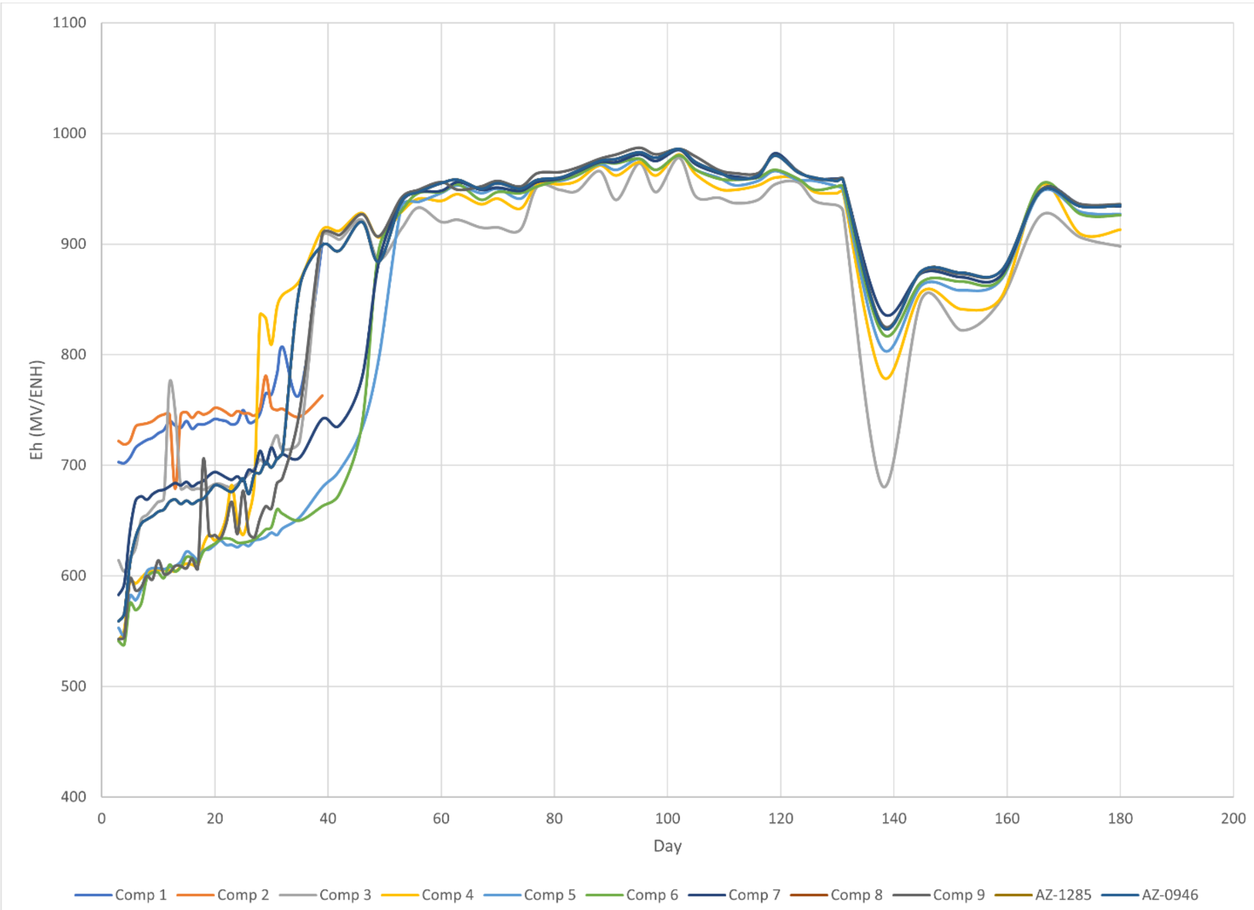


Figure 13.41: 19mm Column Eh Results

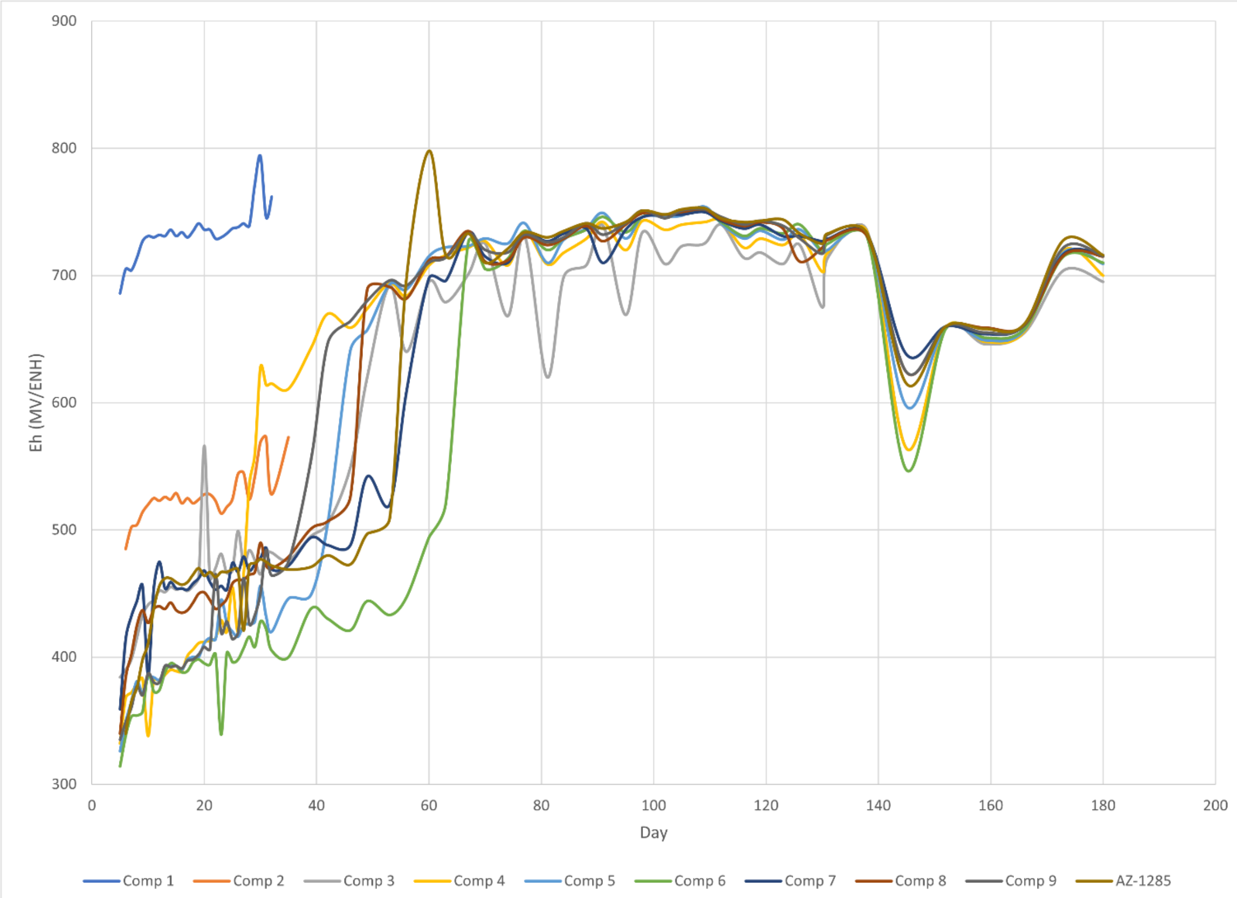


Figure 13.42: 12.7mm Column Eh Results

13.3.5.1 Column Test work and Recoveries

Based on the Plenge test work from 2010 and the start of the Phase 1 Metallurgical test work program, the following observations are provided.

- Based on the results to date, a fixed base copper recovery of 100% soluble copper (CuSOL = CuAS + CuCN) is recommended for the resource evaluation and economic assessment at this time. Additionally, with soluble copper recovery above 100%, a recovery of residual copper (after soluble copper) at 15% is recommended for the resource evaluation and economic assessment currently.
- Soluble Copper recovery continues to progress with columns in progress, currently the column performance has exceeded expectations. Plenge test work-maintained columns at approximately 1pH with a high dosage of raffinate at the start, explaining the fast kinetic leach at the start of the test.
- Column test work is in progress and results presented herein are indicative in nature only until columns are taken down and assayed for a calculated head grade. Only then, can the recoveries and grades be reconciled.
- Copper recovery is consistent with the materials tested so far, and copper extraction and acid consumption recommendation should be used for resource evaluation.
- Based on current bottle roll results and the current column consumption (Figure 13.47 and Figure 13.48), an average gross acid consumption is 18 kg/t of material.
- Figure 13.49 and Figure 13.50 present the pH of the operating columns. Currently the columns are overdosed with acid and a decrease in dosage of acid is taking place as of the writing of this report to keep the columns operating between 1.8 and 2.2 pH.
- Figure 13.43 and Figure 13.44 show present the current copper soluble recovery results of the Supergene columns program versus the Plenge soluble copper recovery and the 100% soluble copper recovery design parameter for this report. Figure 13.44 most matches the material as all data is 100% passing 12.7mm (1/2 inch). Logarithmic fitted curves have been added to present where the data may fall after 180 days of leaching.

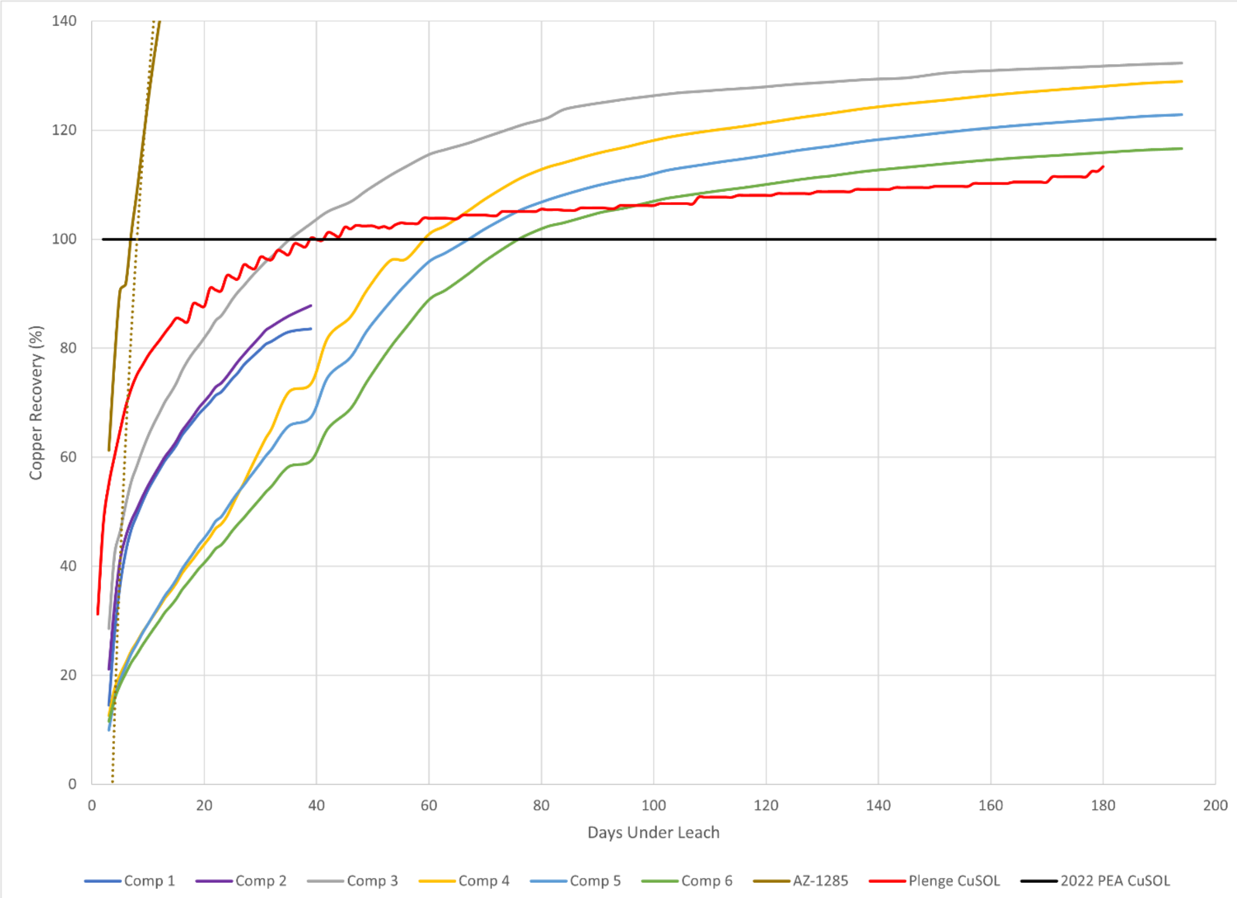


Figure 13.43: 19mm Supergene Column Soluble Copper Recovery

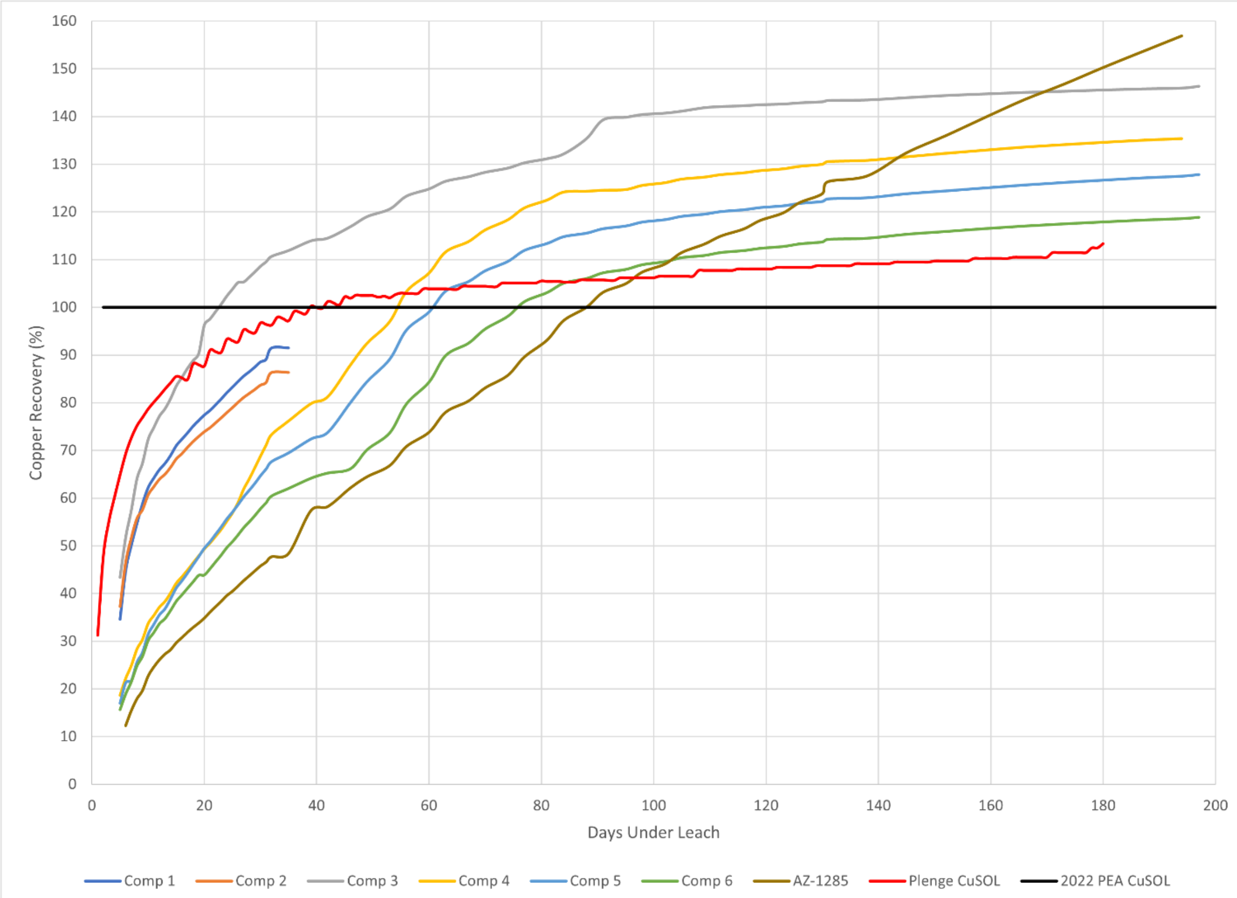


Figure 13.44: 12.7mm Supergene Column Soluble Copper Recovery

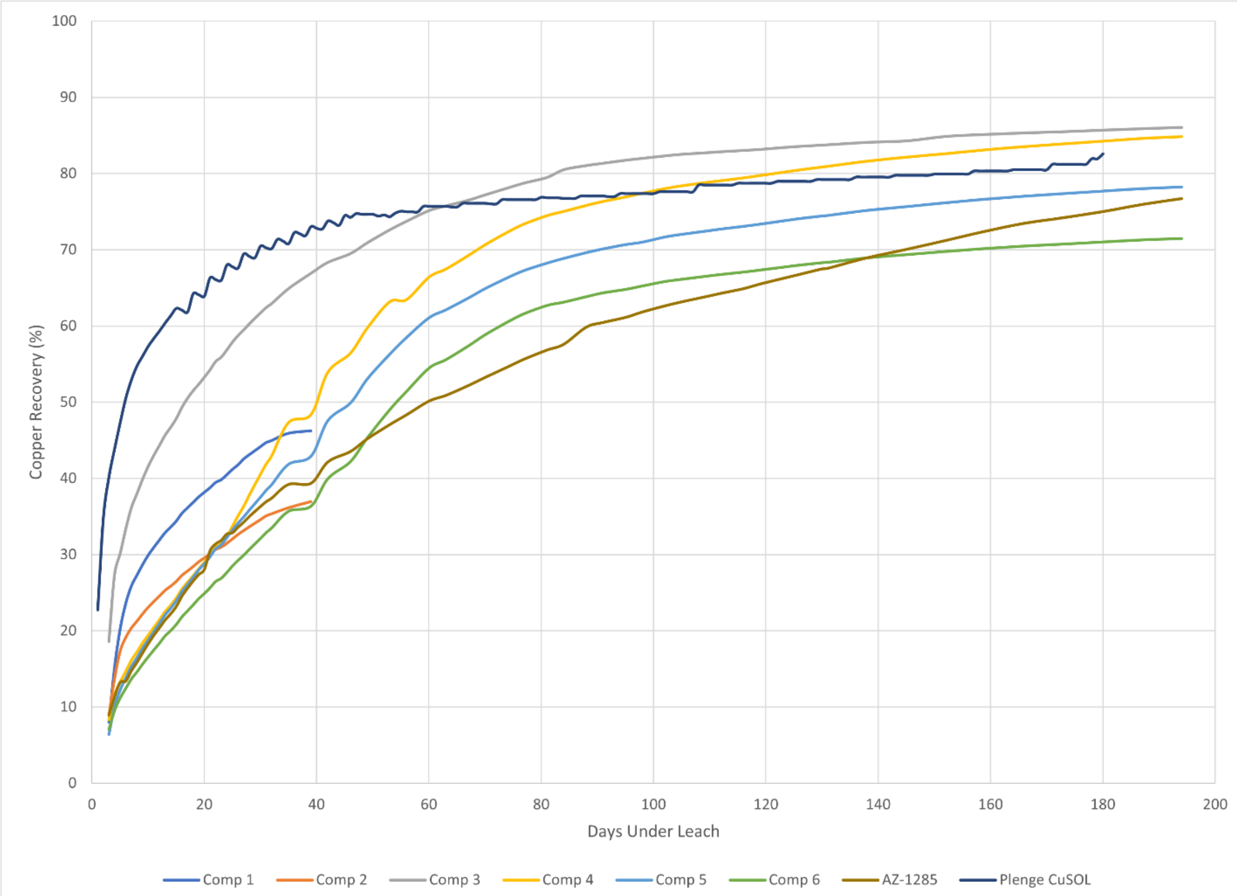


Figure 13.45: 19mm Supergene Column Total Copper Recovery

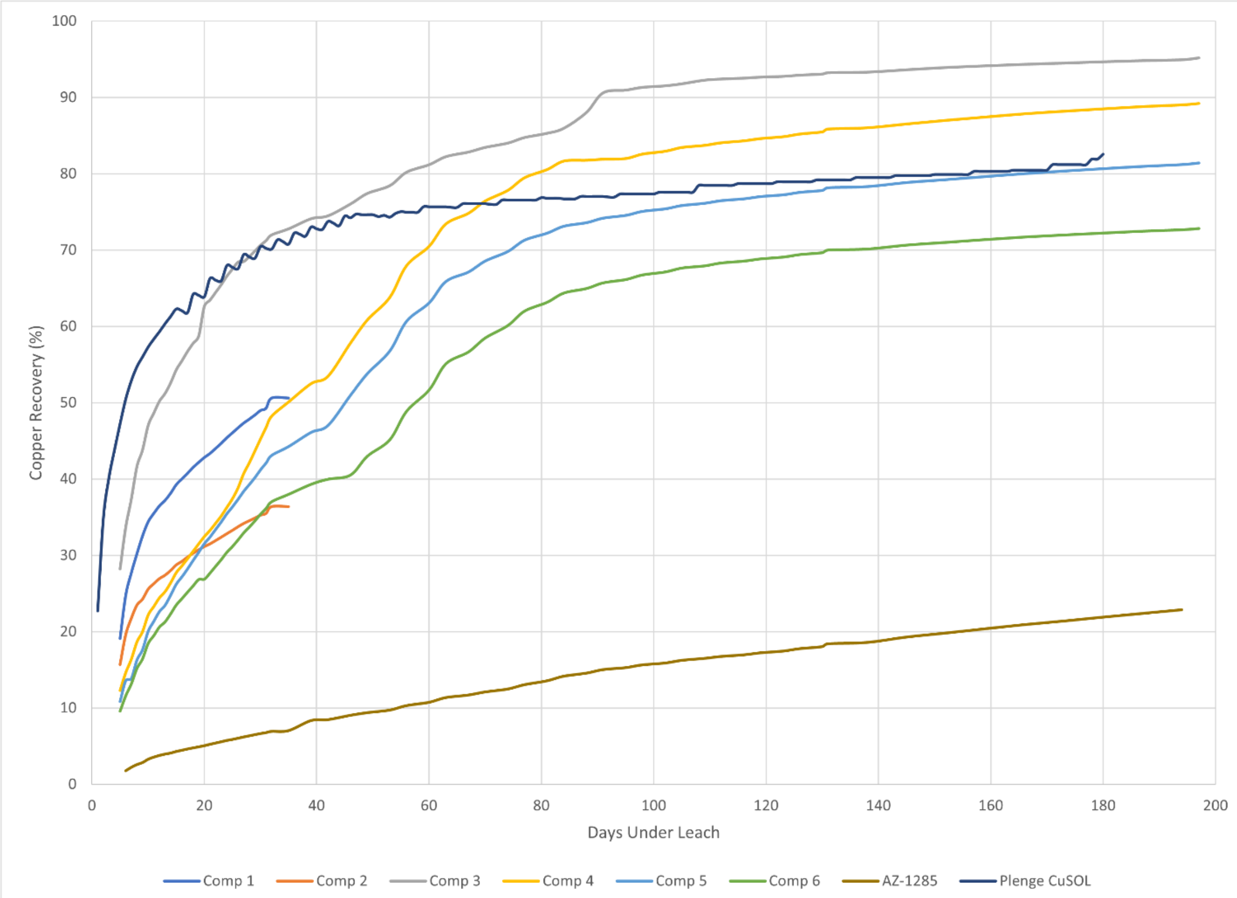


Figure 13.46: 12.7mm Supergene Column Total Copper Recovery

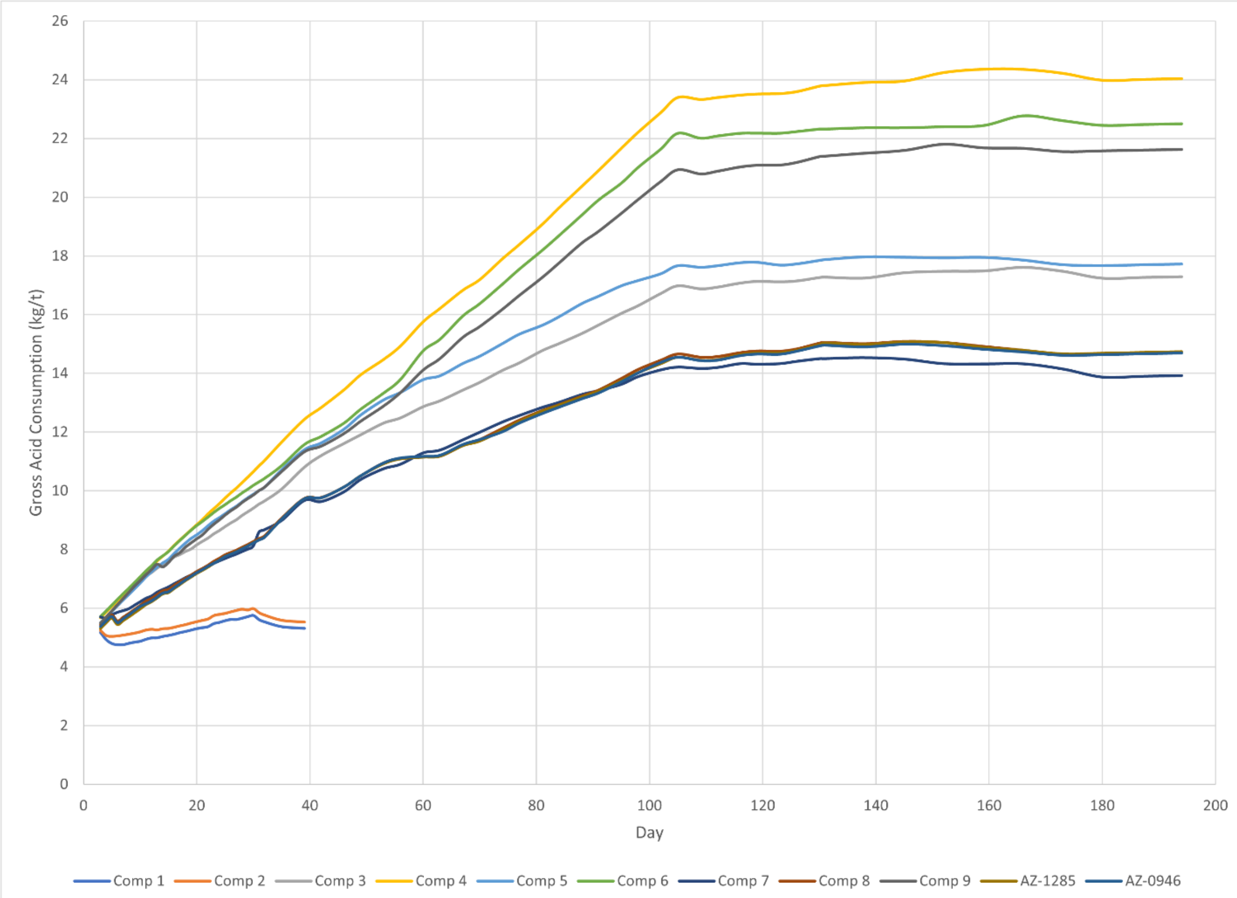


Figure 13.47: 19mm Column Gross Acid Consumption

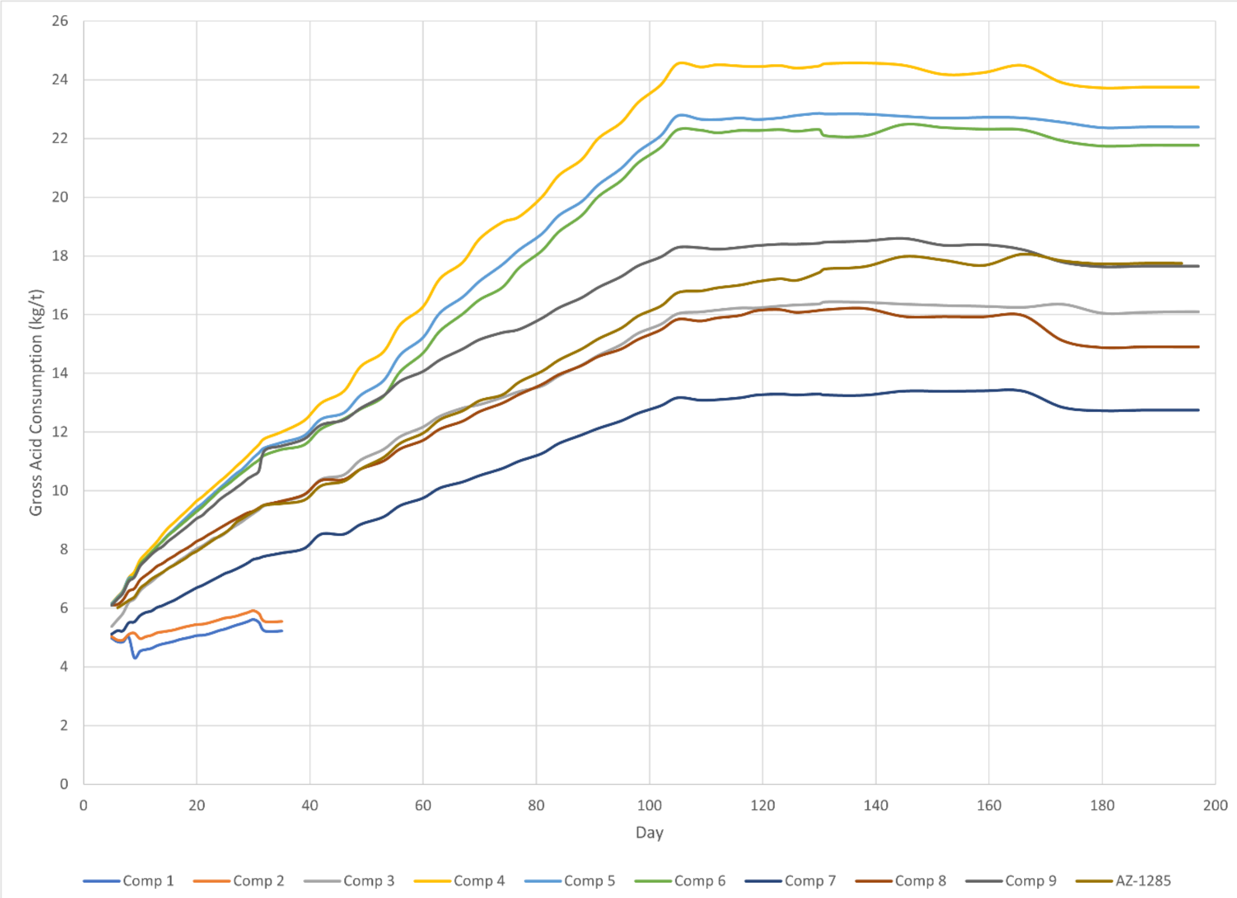


Figure 13.48: 12.7mm Column Gross Acid Consumption

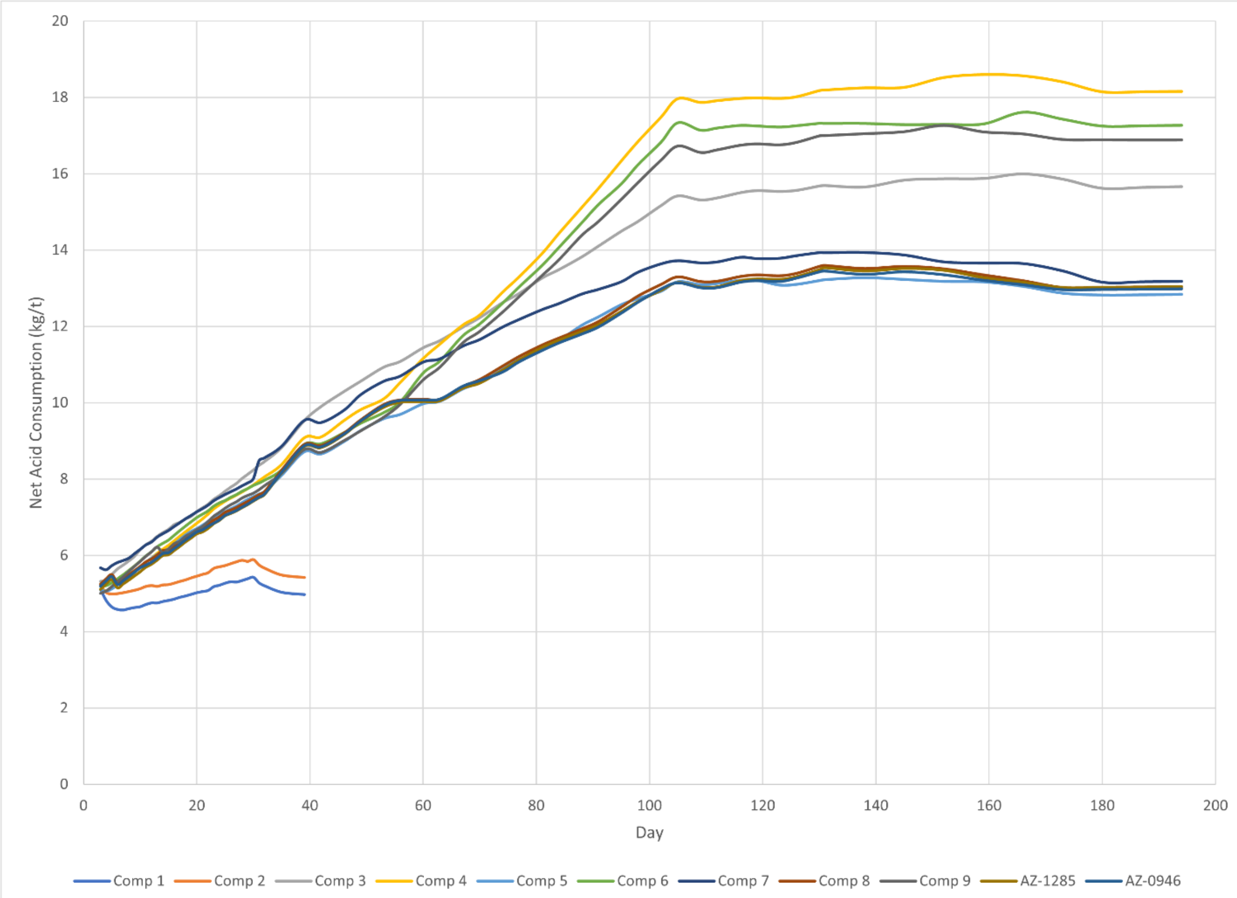


Figure 13.49: 19mm pH

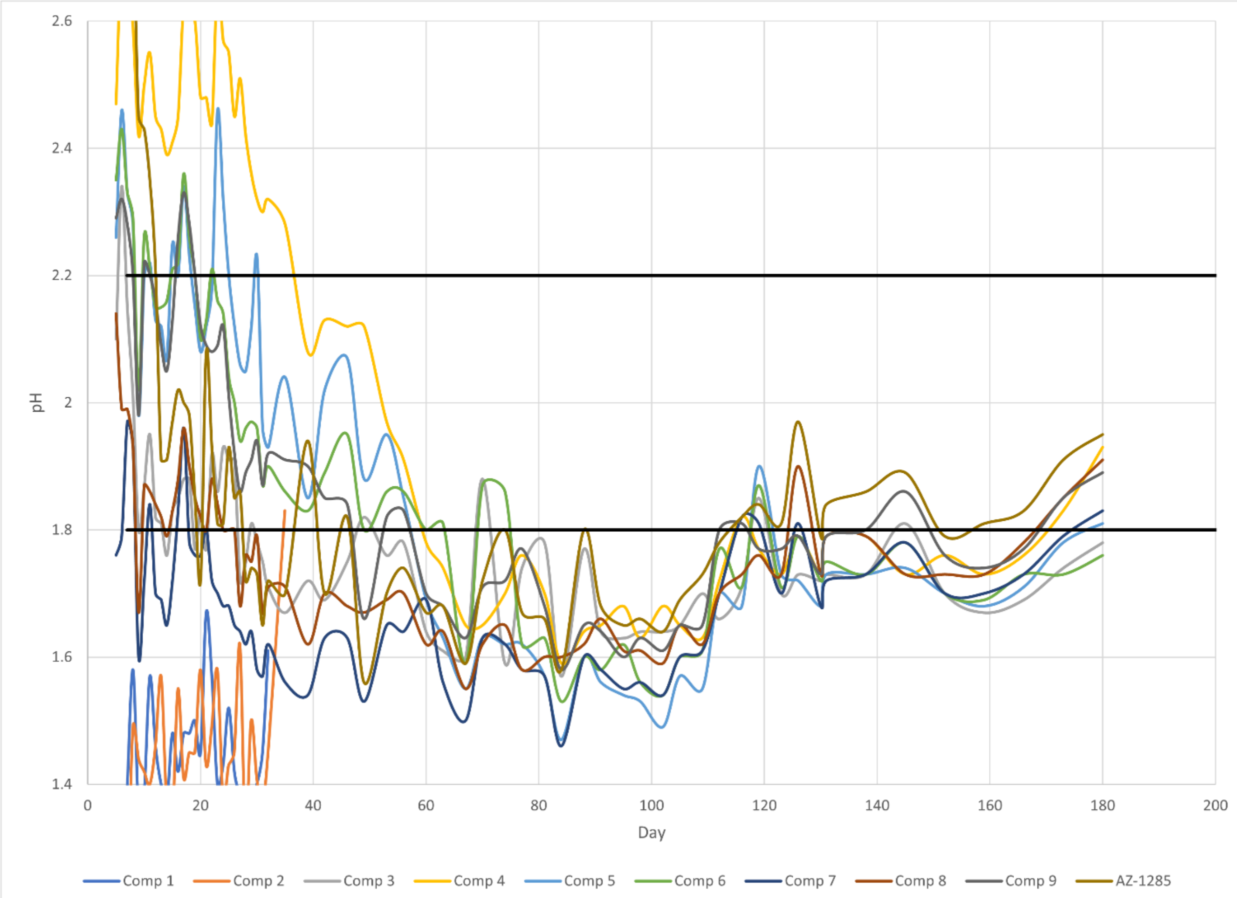


Figure 13.50: 12.7mm pH

13.3.6 Solid-Liquid Settling and Filtration

Initial results of 11 solid-liquid settling tests were completed on flotation tailings samples (125-, 175-, 250-, and 350-micron tests) at the effective date of this report. The flotation tailings were diluted with water to 12% solids by weight and dosed with RHEOMAX 1050 at 10 g/t in 1,000 mL cylinders. The average results of the 11 samples (44 total) are summarized in Table 13.46.

Table 13.46: Average Solid-Liquid Settling Results

	Initial % Solids	Velocity (mm/s)	Final % Solids	$D_{initial}$	D_{final}	Unit Area M^2hr/t
125-micron	12%	5.99	57.53%	7.33	0.72	0.31
175-micron	12%	6.53	57.52%	7.34	0.66	0.29
250-micron	12%	6.96	60.36%	7.33	0.63	0.27
350-micron	12%	7.12	61.00%	7.33	0.61	0.26

At the time of this report, no filtration data has been reported from SGS.

13.3.7 Mineralization (XRD, CLAY ANALYSIS, XRF, TIMA-X PMA)

The mineralogical characterization tests of 74 drill core head assays were completed in X-Ray Fluorescence (XRF), X-Ray Diffraction (XRD) with clay analysis. TESCAN Integrated Mineral Analyzer (TIMA-X) uses both Energy Dispersive X-Ray (EDX) and backscattered electron (BSE) signals to identify minerals and compare them to entries in the SGS mineral library, an upgrade in technology from the traditionally accepted QEMSCAN technology.

Each sample for the TIMA-X was analyzed over four (4) size fractions at +150-micron, -150-micron / +74-micron, -74-micron / +37-micron, and -37-micron for the best understanding of the complex mineralogy of the material deposit. At the effective date of this report, the TIMA-X results have not been fully reported.

XRD results of the Supergene, Primary and Oxide/LIX materials and respective lithologies are reported in Table 13.47, Table 13.48, and Table 13.49 respectively.

XRF results of the Supergene, Primary and Oxide/LIX materials and respective lithologies are reported in Table 13.50, Table 13.51, and Table 13.52 respectively.

Of significance is the level of potassium in the gangue minerals. With dissolution this could drive iron precipitation as potassium jarosite particularly at elevated temperatures. Additionally, significant biotite and chlorite levels are also observed in some composites, which can lead to higher acid consumption, particularly at higher leaching temperatures and lower pH in the leach solutions. Continued analysis and monitoring of these features is planned in the future workplan.

Table 13.47: Supergene XRD and Clay Results

	Dacitic Porphyry			Diorite			Hydrothermal Magmatic Breccia			Porphyry Diorite			Quartz			Rhyodacite Porphyry		
	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max
Albite	28.64	22.66	32.63	24.56	8.22	45.12	20.14	18.12	23.45	23.14	6.36	41.55	2.64	0.53	6.53	19.37	14.42	25.96
Alunite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.16	0.16	0.16	0.00	0.00	0.00
Biotite	0.00	0.00	0.00	2.66	0.15	4.44	1.09	0.21	1.97	2.64	0.64	4.64	0.00	0.00	0.00	0.00	0.00	0.00
Chalcocite	0.23	0.21	0.24	0.65	0.19	1.26	1.00	0.74	1.28	0.28	0.12	0.37	0.34	0.20	0.47	0.47	0.38	0.59
Chalcopyrite	0.53	0.08	0.89	0.55	0.14	1.49	0.69	0.43	1.14	0.58	0.10	1.02	0.71	0.29	1.14	0.56	0.48	0.65
Chlorite	4.60	3.52	5.73	4.42	1.67	8.26	3.76	1.36	6.25	5.74	2.15	9.15	1.65	1.16	2.13	4.56	1.28	7.83
Galena	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.23	0.23	0.23	0.00	0.00	0.00
Goethite	0.54	0.54	0.54	0.47	0.16	1.37	0.00	0.00	0.00	0.41	0.21	0.78	0.00	0.00	0.00	0.22	0.22	0.22
Gypsum	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	2.27	2.27	2.27	0.00	0.00	0.00	0.00	0.00	0.00
Hematite	0.11	0.11	0.11	0.71	0.24	2.99	0.77	0.74	0.81	0.39	0.13	0.62	0.31	0.13	0.50	0.17	0.14	0.19
Kaolinite	0.00	0.00	0.00	1.19	0.48	2.15	0.89	0.82	0.96	1.15	1.14	1.17	0.39	0.14	0.63	0.57	0.43	0.80
K-Feldspar	7.13	5.27	10.44	5.42	0.55	12.27	9.76	1.08	15.74	9.03	3.84	13.02	2.86	2.51	3.48	3.96	0.44	9.65
Molybdenite	0.02	0.02	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Montmorillonite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Muscovite	20.84	17.61	23.16	19.20	8.38	33.14	18.46	12.73	24.88	17.67	9.66	33.76	17.64	11.04	21.06	27.75	12.09	35.08
Oligoclase	6.01	4.46	8.89	6.80	3.19	12.15	3.67	2.63	4.90	10.55	2.79	23.15	3.87	2.42	6.26	3.71	0.09	11.45
Pyrite	0.56	0.33	0.71	1.00	0.09	1.78	0.94	0.44	1.86	1.54	0.14	5.47	1.00	0.85	1.21	1.51	0.29	2.44
Quartz	30.97	27.89	33.86	34.16	18.83	50.20	37.87	32.27	42.26	31.80	20.49	49.57	69.24	64.42	74.55	38.48	32.33	43.96
Rutile	0.35	0.24	0.42	0.44	0.31	0.56	0.46	0.32	0.62	0.40	0.26	0.58	0.18	0.15	0.23	0.30	0.23	0.42
Sphalerite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Tourmaline	0.00	0.00	0.00	1.75	0.80	3.69	4.21	4.21	4.21	0.00	0.00	0.00	0.55	0.55	0.55	2.32	0.56	4.08
Jarosite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00

Table 13.48: Primary XRD and Clay Results

	Dacitic Porphyry			Diorite			Hydrothermal Magmatic Breccia			Porphyry Diorite			Quartz			Rhyodacite Porphyry		
	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max
Albite	27.47	18.17	37.64	23.12	7.47	36.63	26.52	22.96	30.81	26.80	23.68	29.91	12.77	6.12	19.42	30.78	18.04	40.67
Alunite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Biotite	0.00	0.00	0.00	1.49	0.13	4.05	3.27	3.15	3.39	0.59	0.11	1.07	0.00	0.00	0.00	1.15	1.15	1.15
Chalcocite	0.99	0.99	0.99	0.39	0.27	0.49	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.21	0.21	0.21
Chalcopyrite	0.70	0.30	0.98	0.89	0.27	3.00	2.51	1.10	4.18	0.74	0.70	0.78	1.25	0.85	1.66	0.69	0.40	0.93
Chlorite	3.58	0.98	5.70	6.46	2.32	9.93	5.72	1.55	8.15	3.80	2.68	4.92	2.33	0.39	4.26	4.05	1.97	7.47
Galena	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.74	1.74	1.74	0.00	0.00	0.00
Goethite	0.00	0.00	0.00	0.49	0.31	0.67	0.00	0.00	0.00	0.00	0.00	0.00	1.66	1.66	1.66	0.30	0.30	0.30
Gypsum	1.93	1.29	2.57	5.30	1.93	10.08	5.05	0.13	9.38	5.98	4.56	7.40	15.15	6.29	24.01	1.06	0.19	1.93
Hematite	0.31	0.11	0.51	0.45	0.12	1.00	0.75	0.56	0.93	0.59	0.59	0.59	0.28	0.28	0.28	0.19	0.19	0.19
Kaolinite	0.93	0.93	0.93	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.42	0.42	0.42	0.00	0.00	0.00
K-Feldspar	7.44	1.70	17.22	8.91	2.36	17.54	12.39	1.72	23.75	7.41	6.94	7.88	5.93	0.71	11.15	9.79	4.54	16.01
Molybdenite	0.00	0.00	0.00	0.03	0.03	0.03	0.03	0.03	0.03	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Montmorillonite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Muscovite	17.81	6.60	23.51	13.82	6.33	21.37	12.55	2.92	28.35	20.83	20.08	21.58	12.70	6.80	18.59	18.59	7.18	27.79
Oligoclase	4.95	2.95	8.70	15.15	1.28	26.92	6.03	3.34	10.31	6.96	4.20	9.72	1.41	0.62	2.21	4.96	2.96	8.08
Pyrite	1.18	0.21	2.68	0.70	0.13	2.53	0.57	0.23	1.15	0.33	0.10	0.56	2.91	1.64	4.19	1.83	0.68	2.98
Quartz	33.60	21.38	44.63	27.05	17.96	39.17	24.44	17.92	31.88	25.95	24.02	27.88	40.74	30.72	50.77	28.27	21.07	40.51

Table 13.48: Primary XRD and Clay Results																		
	Dacitic Porphyry			Diorite			Hydrothermal Magmatic Breccia			Porphyry Diorite			Quartz			Rhyodacite Porphyry		
	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max
Rutile	0.41	0.31	0.46	0.46	0.31	0.61	0.34	0.27	0.38	0.32	0.29	0.35	0.21	0.16	0.26	0.33	0.28	0.38
Sphalerite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	2.06	0.29	3.83	0.00	0.00	0.00
Tourmaline	2.20	2.20	2.20	1.54	1.33	1.75	3.60	3.60	3.60	0.00	0.00	0.00	0.99	0.99	0.99	0.00	0.00	0.00
Jarosite	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00

Table 13.49: Oxide/LIX XRD and Clay Results			
	Oxide/LIX		
	Average	Min	Max
Albite	15.30	1.07	26.88
Alunite	0.00	0.00	0.00
Biotite	4.11	4.11	4.11
Chalcocite	0.29	0.16	0.48
Chalcopyrite	0.36	0.36	0.36
Chlorite	3.36	1.36	6.55
Galena	0.00	0.00	0.00
Goethite	1.52	0.52	3.65
Gypsum	0.51	0.51	0.51
Hematite	0.93	0.11	3.19
Kaolinite	0.81	0.16	2.00
K-Feldspar	4.09	0.29	8.35
Molybdenite	0.00	0.00	0.00
Montmorillonite	0.52	0.35	0.79
Muscovite	27.80	15.31	39.94
Oligoclase	5.13	0.86	14.49
Pyrite	0.67	0.08	2.04
Quartz	41.47	31.92	58.01
Rutile	0.39	0.24	0.61
Sphalerite	0.00	0.00	0.00
Tourmaline	3.21	2.44	5.00
Jarosite	1.84	1.77	1.89

Table 13.50: Supergene XRF Results

	Dacitic Porphyry			Diorite			Hydrothermal Magmatic Breccia			Porphyry Diorite			Quartz			Rhyodacite Porphyry		
	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max
Fe 0.01-70%	2.01	1.75	2.30	2.80	2.12	4.70	2.20	2.04	2.31	2.64	2.06	3.27	1.34	0.79	1.64	2.24	2.19	2.27
Fe2O3 (%)	2.88	2.50	3.29	4.01	3.04	6.72	3.15	2.92	3.30	3.78	2.95	4.68	1.91	1.13	2.34	3.20	3.13	3.25
Si 0.04-47%	32.04	31.54	32.31	31.27	30.28	33.0	31.18	30.72	31.75	31.29	30.05	32.29	38.23	36.49	40.29	32.51	31.95	33.16
SiO2 (%)	67.96	67.31	69.11	67.25	60.43	70.9	66.70	65.71	67.91	67.22	65.06	69.09	81.78	78.06	86.19	68.57	65.14	70.93
Ca 0.04-40%	0.46	0.22	0.93	0.44	0.14	2.71	0.20	0.15	0.25	0.50	0.13	1.35	0.14	0.11	0.18	0.37	0.14	0.74
CaO (%)	0.64	0.30	1.30	0.62	0.19	3.79	0.28	0.21	0.35	0.70	0.19	1.88	0.20	0.16	0.25	0.51	0.19	1.04
Mn 0.02-77%	0.05	0.05	0.05	0.05	0.02	0.07	0.02	0.02	0.02	0.04	0.02	0.07	0.04	0.03	0.04	0.03	0.03	0.03
MnO (%)	0.05	0.02	0.07	0.04	0.02	0.10	0.03	0.03	0.03	0.05	0.02	0.09	0.03	0.02	0.05	0.03	0.03	0.03
Al 0.02-53%	8.39	7.90	9.01	8.24	6.90	9.27	8.34	7.52	8.86	8.57	8.10	9.05	4.72	3.62	5.66	8.40	7.96	9.04
Al2O3 (%)	15.86	14.94	17.03	15.56	13.04	17.5	15.76	14.22	16.74	16.19	15.31	17.10	8.92	6.83	10.70	15.87	15.04	17.08
Ti 0.03-60%	0.27	0.23	0.29	0.33	0.22	0.39	0.33	0.30	0.37	0.31	0.23	0.39	0.15	0.12	0.20	0.28	0.22	0.37
TiO2 (%)	0.45	0.39	0.49	0.54	0.37	0.65	0.55	0.50	0.62	0.51	0.38	0.66	0.26	0.19	0.34	0.47	0.37	0.61
Mg 0.03-30%	0.70	0.51	0.88	0.63	0.32	1.05	0.73	0.62	0.92	0.63	0.33	1.23	0.34	0.13	0.56	0.58	0.32	0.82
MgO (%)	1.16	0.84	1.47	1.04	0.54	1.75	1.22	1.03	1.52	1.05	0.54	2.04	0.57	0.22	0.93	0.96	0.54	1.37
P 0.01-9%	0.04	0.03	0.05	0.05	0.02	0.08	0.04	0.02	0.05	0.06	0.04	0.10	0.06	0.06	0.06	0.05	0.02	0.07
P2O5 (%)	0.09	0.08	0.12	0.12	0.05	0.17	0.09	0.05	0.12	0.15	0.09	0.22	0.08	0.02	0.14	0.11	0.04	0.15
S 0.01-24%	0.62	0.56	0.66	1.07	0.11	2.54	0.96	0.66	1.44	1.16	0.22	3.07	0.79	0.48	0.95	0.92	0.38	1.60
SO3 (%)	1.53	1.40	1.64	2.68	0.28	6.35	2.40	1.64	3.59	2.89	0.56	7.66	1.96	1.20	2.37	2.29	0.94	4.00
K 0.01-2%	2.93	2.67	3.27	2.82	2.22	3.75	2.94	2.19	3.53	2.85	2.61	2.98	1.92	1.53	2.13	2.91	2.74	3.07
K2O (%)	3.52	3.22	3.93	3.40	2.68	4.52	3.55	2.64	4.26	3.44	3.15	3.58	2.30	1.84	2.56	3.51	3.30	3.70
Cu 0.01-80%	0.33	0.23	0.43	0.65	0.04	1.25	1.10	1.02	1.25	0.32	0.13	0.62	0.36	0.17	0.49	0.47	0.22	0.63
CuO (%)	0.41	0.29	0.54	0.81	0.05	1.57	1.38	1.28	1.57	0.39	0.16	0.78	0.45	0.21	0.61	0.59	0.28	0.79
Zn 0.01-16%	0.03	0.02	0.03	0.02	0.01	0.05	0.05	0.04	0.05	0.03	0.01	0.10	0.17	0.07	0.36	0.02	0.01	0.03
ZnO (%)	0.02	0.01	0.03	0.03	0.01	0.06	0.06	0.05	0.06	0.04	0.01	0.13	0.22	0.09	0.45	0.03	0.02	0.04
As 0.01-23%	0.02	0.01	0.02	0.02	0.01	0.02	0.00	0.00	0.00	0.02	0.01	0.02	0.01	0.01	0.01	0.00	0.00	0.00
As2O3 (%)	0.02	0.02	0.02	0.02	0.01	0.03	0.01	0.01	0.01	0.01	0.01	0.02	0.02	0.02	0.02	0.01	0.01	0.01
Pb 0.01-18%	0.00	0.00	0.00	0.01	0.01	0.03	0.00	0.00	0.00	0.07	0.07	0.07	0.09	0.01	0.24	0.02	0.01	0.02
PbO (%)	0.00	0.00	0.00	0.02	0.01	0.03	0.00	0.00	0.00	0.07	0.07	0.07	0.10	0.01	0.26	0.02	0.01	0.02
Na 0.06-17%	3.23	2.80	3.62	2.06	0.92	3.39	2.14	1.98	2.27	2.41	0.69	3.52	0.53	0.24	0.85	2.23	1.39	3.38
Na2O (%)	4.35	3.77	4.87	2.77	1.24	4.57	2.89	2.66	3.06	3.25	0.93	4.74	0.72	0.33	1.15	3.01	1.88	4.56
Mo (%) 0.01-1%	0.01	0.01	0.01	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MoO3	0.02	0.01	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.00	0.00	0.00	0.00	0.00	0.00
LOI	3.39	2.53	4.42	4.94	2.74	8.71	5.91	4.98	6.54	3.96	2.94	5.86	3.53	2.89	4.40	3.88	3.29	4.54

Table 13.51: Primary XRF Results

	Dacitic Porphyry			Diorite			Hydrothermal Magmatic Breccia			Porphyry Diorite			Quartz			Rhyodacite Porphyry		
	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max	Average	Min	Max
Fe 0.01-70%	2.15	1.71	2.60	2.68	1.88	3.42	2.94	1.59	4.28	1.90	1.87	1.93	2.96	2.07	3.85	2.14	1.70	2.73
Fe2O3 (%)	3.07	2.45	3.72	3.83	2.68	4.89	4.20	2.27	6.11	2.73	2.68	2.77	4.24	2.96	5.51	3.06	2.43	3.90
Si 0.04-47%	31.75	31.61	32.03	30.46	27.92	32.76	29.88	28.25	33.00	30.61	30.22	31.00	28.81	26.73	30.88	32.27	31.46	32.72
SiO2 (%)	67.93	67.62	68.53	65.16	59.72	70.09	63.93	60.43	70.61	65.49	64.65	66.32	67.33	66.06	68.59	68.32	67.31	70.01
Ca 0.04-40%	0.52	0.16	0.73	1.14	0.21	3.48	1.63	0.23	2.71	1.69	1.46	1.91	1.09	0.35	1.83	0.62	0.20	0.93
CaO (%)	0.73	0.22	1.03	1.60	0.29	4.87	2.28	0.33	3.79	2.36	2.04	2.68	1.53	0.49	2.56	0.87	0.28	1.30
Mn 0.02-77%	0.03	0.02	0.04	0.03	0.02	0.07	0.08	0.07	0.08	0.04	0.03	0.04	0.07	0.07	0.07	0.02	0.02	0.02
MnO (%)	0.04	0.03	0.05	0.04	0.02	0.09	0.10	0.09	0.10	0.05	0.04	0.06	0.09	0.09	0.09	0.03	0.02	0.03
Al 0.02-53%	8.18	7.98	8.33	8.43	7.31	9.07	7.59	6.90	8.30	8.08	7.98	8.18	6.36	4.22	8.50	7.96	7.90	8.00
Al2O3 (%)	15.44	15.07	15.74	15.93	13.82	17.13	14.35	13.04	15.69	15.27	15.08	15.46	12.02	7.98	16.06	15.04	14.94	15.12
Ti 0.03-60%	0.27	0.24	0.31	0.35	0.23	0.42	0.29	0.25	0.32	0.27	0.23	0.31	0.18	0.12	0.24	0.25	0.23	0.29
TiO2 (%)	0.45	0.40	0.53	0.59	0.39	0.70	0.49	0.41	0.54	0.45	0.38	0.52	0.31	0.20	0.41	0.42	0.38	0.49
Mg 0.03-30%	0.50	0.42	0.55	0.90	0.45	1.20	0.78	0.50	1.05	0.56	0.47	0.65	0.38	0.26	0.49	0.60	0.38	0.88
MgO (%)	0.83	0.69	0.92	1.49	0.75	1.98	1.30	0.83	1.75	0.93	0.78	1.08	0.62	0.43	0.81	1.01	0.63	1.47
P 0.01-9%	0.05	0.05	0.05	0.06	0.04	0.08	0.05	0.04	0.08	0.05	0.03	0.06	0.05	0.05	0.05	0.04	0.03	0.05
P2O5 (%)	0.12	0.11	0.12	0.14	0.09	0.18	0.11	0.08	0.17	0.11	0.08	0.13	0.07	0.02	0.11	0.09	0.08	0.11
S 0.01-24%	1.19	0.64	1.63	1.01	0.38	2.27	2.03	1.00	2.54	1.61	1.13	2.09	3.23	0.80	5.66	1.24	0.63	1.80
SO3 (%)	2.96	1.59	4.07	2.52	0.94	5.68	5.07	2.51	6.35	4.02	2.82	5.21	8.06	1.99	14.12	3.09	1.56	4.49
K 0.01-2%	2.71	2.49	3.10	2.66	2.09	3.29	2.94	2.47	3.75	2.75	2.66	2.83	2.23	2.04	2.41	2.92	2.49	3.27
K2O (%)	3.27	3.00	3.74	3.21	2.52	3.96	3.55	2.98	4.52	3.31	3.21	3.41	2.69	2.46	2.91	3.52	3.00	3.93
Cu 0.01-80%	0.54	0.31	0.97	0.37	0.12	1.05	0.92	0.42	1.53	0.26	0.24	0.28	0.34	0.09	0.59	0.32	0.28	0.34
CuO (%)	0.68	0.39	1.22	0.47	0.15	1.32	1.16	0.53	1.92	0.33	0.30	0.35	0.43	0.11	0.74	0.39	0.35	0.42
Zn 0.01-16%	0.03	0.01	0.04	0.02	0.01	0.05	0.04	0.02	0.05	0.04	0.02	0.05	1.47	0.01	2.93	0.03	0.01	0.04
ZnO (%)	0.04	0.02	0.05	0.03	0.02	0.06	0.04	0.02	0.06	0.05	0.03	0.06	1.84	0.02	3.65	0.02	0.01	0.05
As 0.01-23%	0.02	0.02	0.02	0.02	0.01	0.04	0.02	0.01	0.02	0.01	0.01	0.01	0.02	0.02	0.02	0.02	0.01	0.02
As2O3 (%)	0.02	0.02	0.02	0.02	0.01	0.05	0.03	0.02	0.03	0.01	0.01	0.01	0.02	0.02	0.02	0.02	0.01	0.02
Pb 0.01-18%	0.00	0.00	0.00	0.01	0.01	0.01	0.00	0.00	0.00	0.01	0.01	0.01	1.62	1.62	1.62	0.00	0.00	0.00
PbO (%)	0.00	0.00	0.00	0.02	0.02	0.02	0.00	0.00	0.00	0.01	0.01	0.01	1.74	1.74	1.74	0.00	0.00	0.00
Na 0.06-17%	2.84	2.11	3.63	2.91	2.06	3.62	2.79	2.31	3.68	2.78	2.74	2.82	2.00	0.80	3.19	2.56	1.62	3.28
Na2O (%)	3.83	2.84	4.89	3.93	2.78	4.88	3.76	3.11	4.96	3.76	3.70	3.81	2.70	1.09	4.30	3.46	2.18	4.42
Mo (%) 0.01-1%	0.00	0.00	0.00	0.02	0.01	0.02	0.02	0.02	0.02	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
MoO3	0.01	0.01	0.01	0.02	0.01	0.03	0.03	0.03	0.03	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.01	0.01
LOI	4.53	3.54	5.36	4.40	2.51	8.19	6.24	3.59	8.71	6.13	4.68	7.57	5.78	4.22	7.34	4.35	3.23	5.36

Table 13.52: Oxide/LIX XRF Results			
	Oxide/LIX		
	Average	Min	Max
Fe 0.01-70%	2.75	3.96	1.92
Fe2O3 (%)	3.93	5.66	2.75
Si 0.04-47%	32.15	33.88	30.56
SiO2 (%)	68.05	72.49	64.35
Ca 0.04-40%	0.36	1.77	0.12
CaO (%)	0.51	2.48	0.17
Mn 0.02-77%	0.04	0.05	0.02
MnO (%)	0.05	0.07	0.03
Al 0.02-53%	8.28	8.89	6.99
Al2O3 (%)	15.65	16.80	13.21
Ti 0.03-60%	0.31	0.42	0.18
TiO2 (%)	0.52	0.70	0.30
Mg 0.03-30%	0.68	0.93	0.36
MgO (%)	1.13	1.54	0.59
P 0.01-9%	0.04	0.06	0.02
P2O5 (%)	0.10	0.15	0.02
S 0.01-24%	0.53	1.44	0.02
SO3 (%)	1.32	3.59	0.04
K 0.01-2%	2.83	3.46	2.19
K2O (%)	3.40	4.16	2.64
Cu 0.01-80%	0.32	1.03	0.04
CuO (%)	0.40	1.29	0.05
Zn 0.01-16%	0.02	0.03	0.01
ZnO (%)	0.02	0.04	0.01
As 0.01-23%	0.03	0.05	0.01
As2O3 (%)	0.03	0.06	0.01
Pb 0.01-18%	0.02	0.03	0.01
PbO (%)	0.02	0.03	0.01
Na 0.06-17%	1.93	3.33	0.24
Na2O (%)	2.60	4.49	0.33
Mo (%) 0.01-1%	0.00	0.00	0.00
MoO3	0.01	0.01	0.01
LOI	4.54	6.20	3.08

13.4 SEPRO MICROWAVE TESTING

Drill core samples from the 2022 drilling season were issued to SEPRO labs in Canada. The samples arrived in October 2022 and the metallurgical test program has been developed by Samuel Engineering and supervised by Mr. Eric Hill. At the effective date of this report, the mineralogy has not been fully completed. Preliminary scoping samples were submitted from the major material types (Supergene and Primary). The microwave system uses a bench assessment of 3 kW for microwave potential of the materials. Figure 13.51 indicates the sample's microwave heating behavior.

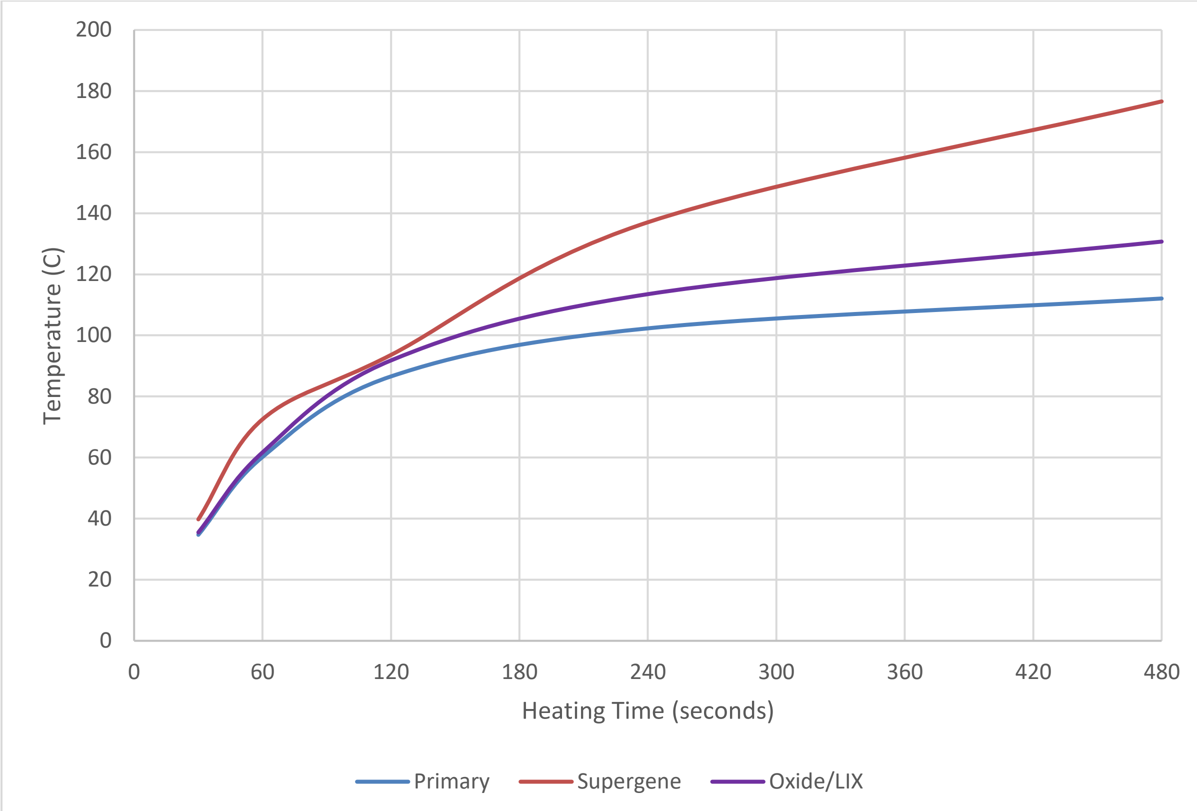


Figure 13.51: Material Response to Microwave

13.5 NUTON TESTING

Nuton LLC is a technology venture of Rio Tinto, one of the world's largest mining companies, and home to a unique integration of innovative nature-based technologies, expertise and capabilities. At the core of Nuton is a portfolio of proprietary copper bio-heap leach related technologies. Nuton aims to advance the environmental, social and governance performance of the industry whilst delivering copper growth. In August 2022, McEwen Copper and Nuton™ entered into a collaboration agreement to test the viability of Nuton™ technologies on a range of Los Azules process material types. Although Nuton has completed larger scale testing at several projects and has developed proprietary modeling techniques to predict results, there are no commercial applications of the Nuton™ technology operating at the time of this report.

From the 2022 drilling season, over 4.5 tonnes of PQ/HQ core material from Los Azules were shipped to Hazen Research in Golden, Colorado in September 2022. Nuton was in progress of setting up a laboratory facility directly at Hazen, mirroring the technology and practices from the Bundoora facility in Melbourne. The column material is currently in progress in both facilities. The 4.5 tonnes of sample represent multiple material types and lithologies as prescribed in Table 13.53.

Table 13.53: Material Shipped to Hazen to undergo Nuton Testing				
Hole #	**From**	**To**	**Material Type**	**Lithology**
AZ17131	334	378	Primary	Dacitic Porphyry
AZ17134	198	251.5	Supergene	Rhyodacite Porphyry
AZ22138	139	203	Supergene	Dacitic Porphyry
AZ22138	422	526	BN-CPY	Diorite
AZ22140	116	245	Supergene	Diorite
AZ22142	95	166	Supergene	Rhyodacite Porphyry
AZ22142	371	432	Primary	Porphyry Diorite
AZ22142	165	222	Supergene	Porphyry Diorite
AZ22146	96	188	Supergene	Hydrothermal Magmatic Breccia
AZ22149	152	236	Supergene	Diorite
AZ22149	383	428	Primary	Diorite
AZ22150	141	215	Primary	Diorite

Additionally, in November 2022, samples from Composite 6, Composite 7, Composite 8, and Composite 9 (90kg of each and described in Table 13.21) were shipped from SGS Santiago and delivered to Melbourne, Australia at the Nuton Bundoora facility. The purpose of these samples was for comparative analysis from SGS utilizing the Nuton™ technology.

Nuton has developed proprietary models to simulate the performance of Nuton™ technologies both in column leach tests and in commercial scale operations. These models have been developed over the past 25 or more years and have been validated at various scales. The Nuton Computational Fluid Dynamics (CFD) model is used to project copper recovery and reagent consumption and provide information necessary to evaluate capital and operating costs specific to the Nuton process, which will help inform an evaluation of the Nuton contribution to the Los Azules project and a notice to proceed to a next stage of the project. Through the CFD modelling, the requirement to add proprietary additives was identified to operate the columns under optimum Nuton conditions.

The columns at Hazen and Bundoora are currently in progress and results to date are considered preliminary. The test matrix for Bundoora can be found in Table 13.54 and for Hazen in Table 13.55. Bundoora column tests are being conducted in 100 mm I.D. by 1 m tall columns containing approximately 10 kg of crushed and agglomerated material. Hazen column tests are being conducted in 140 mm I.D. by 1 m tall columns containing approximately 20 kg of crushed and agglomerated material. All material was agglomerated to 4-6% moisture and loaded into the columns, resting for three (3) days before commencing irrigation with raffinate. The raffinate is being added at a rate of 10 L/hr/m2. Microbial cultures (biomass) required for leaching are added directly to the columns. The biomass was produced from proprietary cultures. Nuton™ uses three (3) different mixed cultures consisting of bacteria and archaea to enable operation over a wide temperature range: a mesophilic culture for a lower temperature range, a moderately thermophilic culture for a moderate temperature range, and an extremely thermophilic culture for a high temperature range. Additives are used to enhance the leaching process.

Table 13.54: Bundoora Column Test Matrix										
	Composite 6		Composite 7		Composite 8			Composite 9		
Operating Temperature [2]	moderate	high	high	med-high	high	high	moderate	high	med-high	moderate
pH[1]	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2
Additive Type[1]	-	-	Additive 2	Additive 2	Additive 2	Additive 2	Additive 3	Additive 2	Additive 2	Additive 3
Additive 1a[1]	-	Yes	Yes	-	-	Yes	-	-	-	-
Additive 1b[1]	-	-	-	-	Yes	-	-	Yes	Yes	-

1 - Specific information is proprietary and confidential
2 – Specific temperature settings are proprietary

Table 13.55: Hazen Column Test Matrix																
BHID	AZ22138 (422-526)				AZ22149 (383-428)		AZ22149 (152-236)	AZ22140 (116-245)	AZ22150 (141-215)	AZ17134 (198-251.5)	AZ22142 (95-166)	AZ22142 (371-432)	AZ22142 (165-222)	AZ17131 (334-378)	AZ22138 (139-203)	AZ22146 (96-188)
Material Type	BN-CPY				Primary		Supergene	Supergene	Primary	Supergene	Supergene	Primary	Supergene	Primary	Supergene	Supergene
Operating Temperature [1]	med-high	med-high	med-high	high	med-high	high	moderate	med-high	high	med-high	med-high	high	med-high	high	med-high	med-high
pH[1]	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2	< 2
Additive Type[1]	Additive 2	Additive 2	Additive 2	Additive 2	-	-	-	-	Additive 2	Additive 2	Additive 2	Additive 2	Additive 2	Additive 2	Additive 2	Additive 2
Additive 1a[1]	Yes	Yes	-	Yes	Yes	Yes	-	-	Yes	-	-	Yes	-	Yes	-	-
Additive 1b[1]	-	-	Yes	-	Yes	Yes	-	-	-	-	-	-	-	-	-	-

1 - Specific information is proprietary and confidential

Stage 1 column leaching of the composite samples at Bundoora, as well as of the primary bulk samples at Hazen, is underway and expected to complete during Q1 2024.

13.6 ADEQUACY OF DATA AND USE

The metallurgical work completed to date and ongoing at the effective date of this report provides an adequate understanding of the expected performance characteristics for a PEA level of analysis. For Los Azules the anticipated copper extraction of the CuSOL fraction of the assayed copper in each block is 100%. Additionally, approximately 15% of the residual copper component can also be extracted based on the metallurgical results to date. Copper recovered to cathodes considers a heap efficiency and inventory factor of 90% of the extractable copper based on general experience. The expected overall total copper recovery expected is approximately 73% and is distributed over a two-year timeframe from placement on the leach pad. In the opinion of the QP, the metallurgical test work and analysis support the metallurgical assumptions provided and used in the mineral resource estimation, the preliminary mine plans, and the economic analysis presented in this report.

Potential scenarios for the preferred future Phase 2 operations employing the Nuton™ bio-leaching technology are presented and discussed in Section 25.2.1 of this report. Although Nuton has completed larger scale testing at several global project sites and has developed proprietary modeling techniques to predict leaching performance results, there are no commercial applications of the Nuton™ technology operating at the time of this report. A significant testing program, including broader column testing of the project resources, site-based scale-up work will be required to validate these preliminary estimates. As such, these results are not considered suitable for inclusion at this time in the initial project cases presented are and only included as a demonstration of the potential future opportunity.

In the opinion of the QP, the metallurgical test work and reconciliation and production data support the metallurgical assumptions used in the mineral resources, the mine plans, and the economic analysis.

14.0 MINERAL RESOURCE ESTIMATES

14.1 INTRODUCTION

This section is a summary of documents presented in 2021 and 2022 detailing the items discussed here[1],[2],[3]. The late-2022/spring McEwen drilling was not available when these documents were completed. The information presented here is updated to include all data collected during the summer/fall 2021/2022 field season.

This report augments previous work with data from 52 drill holes (22,497, meters) completed by December 31, 2022. Additional data included 159 drill holes (24,704 samples) submitted to re-assay for elements not provided in the original assay suite.

The geological model was rebuilt from all available data and the complete data set reassessed to develop a new estimation strategy. The resource estimation was performed by Jeff Sullivan PhD and Silvia Satchwell of CRM-SA, LLC. The work is overseen by Allan Schappert of Stantec who serves as the qualified person (QP) for the resource estimate.

14.1.1 Resource Database and Geological Model Extent

The current database is adequate for the preparation of a long-range model that will serve as a basis for modeling associated with completing the PEA. The extent of mineralization along strike exceeds 4 km and the distance across strike is approximately 2.2 km. The deposit is open at depth. Over the approximately 2.5 km strike length where mineralization is strongest, the average drill spacing is approximately 150 meters to 200 meters but there are localized areas where drilling is on a 100m spacing. The assay database considers 162 drillholes and 56,528 meters of assay interval data. Resource estimation work was performed using Datamine Studio modeling software.

14.1.2 Summary of Controls on Mineralization

Mineralization shows strong continuity from south to north and vertically. Laterally, grades decrease moving away from a NNW striking central structure. The main control on mineralization is the modeled mineral zone which follows the typical porphyry copper pattern. Below the unmineralized overburden, a low-grade leach unit is found which overlies a well-developed zone of secondary enrichment which transitions into primary mineralization at depth. The model also contains a small oxide/sulfide mixed zone. There is only minor copper oxide mineralization and so no economic oxide zone was modeled.

A secondary control on grade is provided by lithology. In terms of copper grade, the strongest mineralization is found in a low-volume hydrothermal breccia. The remaining lithologies are intrusive rocks that are modeled according to the age of the mineralization. The pre-mineral pluton (diorite) is the background rock intruded by the relatively narrow early mineral porphyry (EMP) and inter-mineral porphyry (IMP). The EMP has elevated grades relative to the diorite and IMP. Combinations of lithology and mineral zone are used to control the estimation; however, the combinations applied differ for the models of copper, gold, and silver.

Additional observed controls on grade are:

- Within the enriched zone, proximity of a sample to the leached/enriched or enriched/primary contact is observed as a control on cyanide soluble copper grade and solubility.

[1] CRM, February 2022, Re-estimation of Copper Grades, Los Azules Project, Argentina
[2] CRM, May 2022, Estimation of Gold and Minor Elements, Los Azules Project, Argentina
[3] CRM, August 2022, Soluble Copper Estimation, Model Notes, Los Azules Project, Argentina

- The highest grades are observed along a central NNW striking, steeply dipping structure/fault. The elevated grades are due to both a higher proportion of the higher-grade lithologic units (breccia and EMP) near the structure and an increase in fracturing of the host rock. These properties of the mineralization generate a lateral grade trend (reduction in grade with increasing distance from the structure). As a result, grades are more continuous parallel, as opposed to perpendicular, to the structure.
- Copper grades are well behaved with low relative variability. Gold and silver grades are more variable. There are narrow breccia and late quartz vein occurrences that carry elevated grades (particularly for gold and silver). Grades associated with these narrow occurrences (not captured by the geological model) are volumetrically restricted during estimation. Aside from this issue, copper grades are not capped. For gold and silver, a local capping algorithm was applied to define and manage outliers.
- Over large volumes there is some correlation between average grades of the metals due to the control exerted by the central structure. Locally, however, correlation can be poor. The correlation between soluble and total copper is strong outside of the enriched zone. Within the enriched zone, correlation varies by depth in the enrichment profile.
- At the contacts between estimation units, a sharp change in grade is observed for copper and sharing samples across estimation units is not allowed (i.e., hard boundaries were utilized).

14.1.3 Spatial Correlation

Spatial correlation was modeled by mineral zone using pairwise relative variograms. Modeled variograms show the expected NNW anisotropy.

14.1.4 Block Model Validation

Copper grades were estimated by ordinary Kriging while gold and silver were estimated using inverse distance squared weighting. The checks performed to validate the estimates included:

- Comparison of drillhole data and model grades in plan and section
- Compare model and data (nearest neighbor estimates) averages over the estimation unit, over slices through the model and over large blocks.

These checks showed that the model reproduced the major features of the data while the match between the model and data averages was acceptable.

14.1.5 Resource Classification

The mineral resources have been classified according to guidelines and logic summarized within the Canadian Institute of Mining, Metallurgy and Petroleum (CIM 2019) and as defined by the Securities Exchange Commission's SK 1300 (SEC 2018). Definitions of mineral resources are slightly different in these two codes. With slight variations resources were classified as Indicated or Inferred by considering geology, sampling, and grade estimation aspects of the model. For geology, consideration was given to the confidence in the interpretation of the lithologic domain boundaries and geometry. For sampling, consideration was given to the number and spacing of composites, the orientation of drilling and the reliability of sampling. For the estimation results, consideration was given to the confidence with which grades were estimated, as Measured by the quality of the match between the grades of the data and the model.

14.1.6 Resource Summary

The indicated and inferred resource for the enriched and primary zones are presented below in Table 14.1 and Table 14.2, respectively. Mineral resources are determined using an NSR cut-off value to cover the processing cost for each recovery methodology. For supergene and primary material going to the leach pile the cutoff was $2.74/t. For supergene going to the mill the cutoff was $5.46/t and primary material going to the mill was $5.43/t. The resource is further constrained by a pit shell that demonstrates the reasonable prospects of eventual economic extraction (RPEEE) of this material.

Table 14.1: Indicated Resources for the Los Azules Project

	MTonnes	Average Grade				Contained Metal			
		In-Situ Copper Total (%)	In-Situ Copper Soluble (%)	In-Situ Gold (g/tonne)	In-Situ Silver (g/tonne)	In-Situ Copper Total Content (Blbs)	In-Situ Copper Soluble (Blbs)	In-Situ Gold (Moz)	In-Situ Silver (Moz)
Leach	944.2	0.46	0.30	-	-	9.54	6.25	-	-
Mill - Supergene	73.0	0.13	-	0.09	1.10	0.21	-	0.20	2.58
Mill - Primary	218.1	0.25	-	0.036	1.06	1.19	-	0.25	7.43
Total Leach	**944.2**	**0.46**	**0.30**	**-**	**-**	**9.54**	**6.25**	-	-
Total Mill	**291.1**	**0.22**	**-**	**0.049**	**1.07**	**1.40**	**-**	**0.46**	**10.01**
Total Measured & Indicated	**1,235.3**	**0.40**				**10.94**			

Table 14.2: Inferred Resources for the Los Azules Project

	MTonnes	Average Grade				Contained Metal			
		In-Situ Copper Total (%)	In-Situ Copper Soluble (%)	In-Situ Gold (g/tonne)	In-Situ Silver (g/tonne)	In-Situ Copper Total Content (Blbs)	In-Situ Copper Soluble (Blbs)	In-Situ Gold (Moz)	In-Situ Silver (Moz)
Leach	695.7	0.32	0.19	-	-	4.91	2.96	-	-
Mill - Supergene	525.6	0.30	-	0.05	1.44	3.45	-	0.87	24.40
Mill - Primary	3,288.0	0.25	-	0.03	1.18	18.35	-	3.37	124.67
Total Leach	**695.7**	**0.32**	**0.19**	**-**	**-**	**4.91**	**2.96**	**-**	**-**
Total Mill	**3,813.6**	**0.26**	**-**	**0.035**	**1.22**	**21.79**	**-**	**4.24**	**149.07**
Total Inferred	**4,509.3**	**0.31**				**26.70**			

Notes:

There is a reasonable prospect of eventual economic extraction of the leach resource using sulfuric acid leaching and SX/EW recovery at NSR cutoff of $2.74/t. The supergene and primary material can be treated in a float mill with NSR cutoffs of $5.46 and $5.43/t respectively. NSR values are based on a copper price of $4.00/lbs., gold at $1,700/oz., and silver at $20/oz., where applicable.

Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant factors. The quantity and grade of reported Inferred mineral resources in this estimation are uncertain in nature and there is insufficient exploration to define these Inferred mineral resources as an Indicated or Measured mineral resource; it is expected that further exploration will result in upgrading some of this material to an Indicated or Measured classification.

14.2 AVAILABLE DATA

Figure 14.1 is a plan map of the project area and shows the collar location of the drill holes used in this MRE. Some of the drilling done by the Battle Mountain Group in 2008 has been excluded because of the lack of lithologic logging. This data represents a very small fraction of the total drilling done to date.

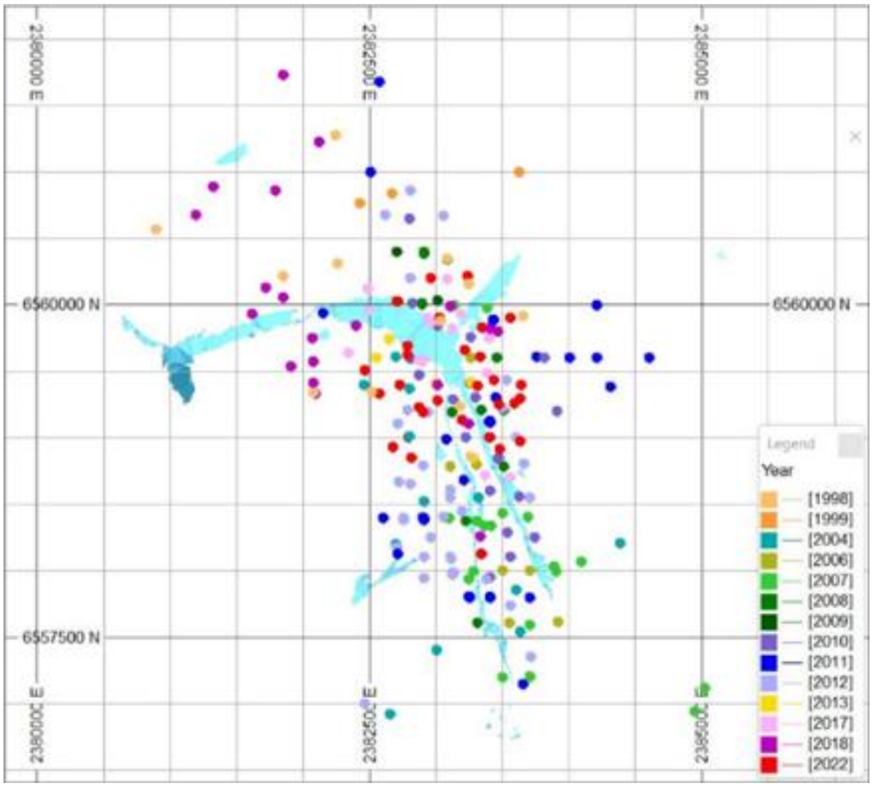


Figure 14.1: Drill Hole Location Map

14.3 GEOLOGIC MODEL

14.3.1 Introduction

The following is a summary of the understanding of the genesis, the interpretation criteria and parameters used in the geological modelling of the Los Azules deposit detailed in a report by Atticus Geoscience (Mortimer, 2022). The model will assist in ongoing exploration and is used as a base for the 2023 MRE model described herein. The 3D geological model was constructed using Leapfrog software.

14.3.2 Geological Evolution

The Los Azules deposit contains overprinting mineralization and alteration events, and it is necessary to understand the time and spatial relationships of these events to develop an integrated geological model.

The development of all models, the cross-cutting relationships and construction sequences are based on the geological evolution and known events, the evidence for which comes from direct observations of cross-cutting relationships seen in the field, and in the drill core.

14.3.3 Structural Model

The structural regime plays a fundamental role in the mineralization, alteration, mineral zonation, and the development of the intrusive lithologies.

The structural model is based largely on geological mapping and field observations (Pratt, 2010), recent discussions with the current geology team, interpretations from drilling data to define and confirm the structural controls and has been refined iteratively throughout the process. Further drilling and additional work will continue to better define the structural model.

14.3.4 Lithology Model

Lithological contacts and solids have been constructed from the integration of data from the lithology, alteration, and assay data input tables in the database. The geological map was also used to define contacts for the volcanics and quaternary cover.

The lithology model is constructed using an event modelling concept – building each contact surface following geological chronology and building each unit in sequence. Table 14.3 describes the units that are modelled and their order in the event sequencing.

Table 14.3: Chronological Geological Events used in Model Construction

Age	Event	Sub Event
Younger	Erosion (Topography)	
	Quaternary Cover	
	Inter Mineral Porphyry (IMP) + MagHydBx + HydBx	HydBx
		MagHydBx
		IMP
	Early Mineral Porphyry (EMP)	
	Diorite Pre-Mineral Pluton (PMP)	
Older	Triassic Volcanics	

All surfaces within the model, except one, were constructed using implicit modelling methods, interpolating between known contact points (drill hole intervals) with subsequent editing guided by sectional and level plan interpretations. The volcanic diorite contact surface is built using geological mapping interpretation lines as no drilling has currently intercepted the volcanics. Figure 14.2 and Figure 14.3 show examples of the lithological model in plan and section views.

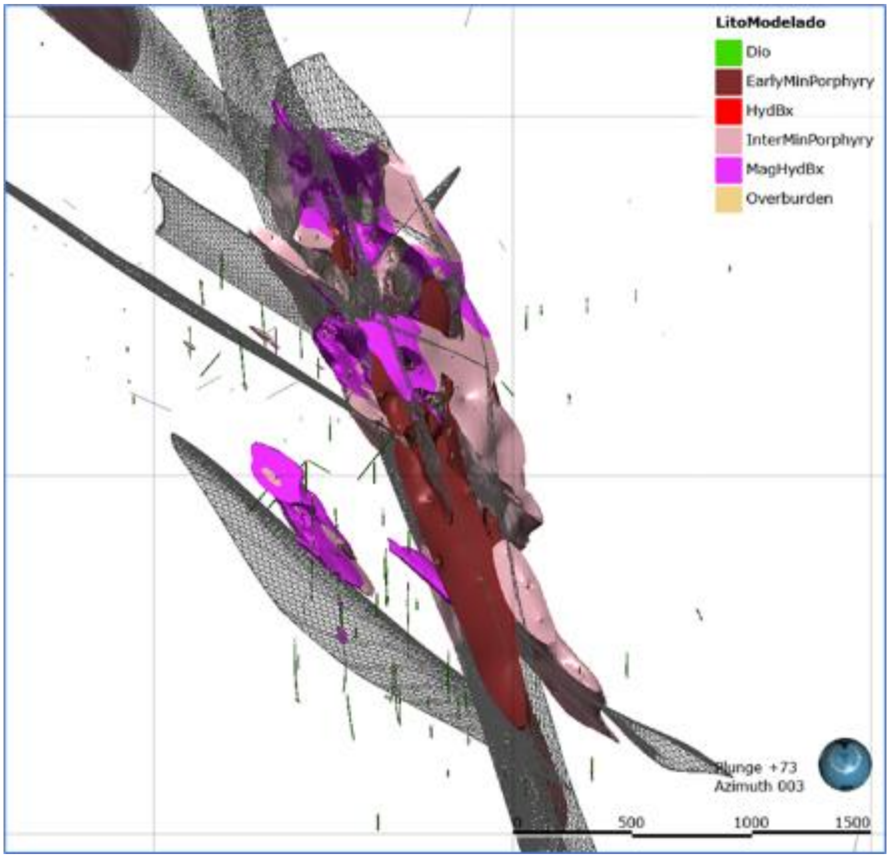


Figure 14.2: Plan view of the lithology model under construction

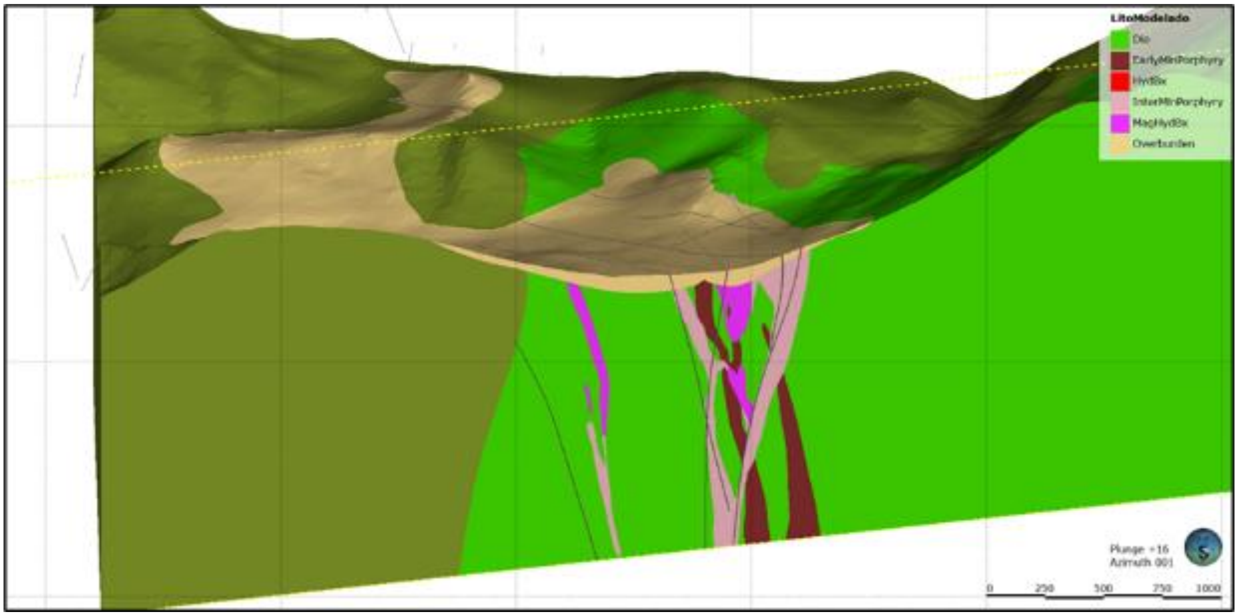


Figure 14.3: Oblique section view of the completed lithological model looking North.

14.3.5 Alteration Model

Contact surfaces and solids of the alteration model are based directly on drill data, constructed using integrated data from interval fields in the alteration, lithology, and assay data tables.

The alteration model was constructed considering the evolution of the deposit. Only the reactive lithologies present at the time of porphyritic intrusions can be affected by the alteration, so the model only considers lithologies below the quaternary cover surface.

The construction sequence considers only one principal event and is thus more aligned to the spatial distribution of alteration types rather than the temporal distribution. Overprinting of alteration types has not been considered in the model. Table 14.4 shows the sequence of alteration events that were used in the model creation.

Table 14.4: Sequence of Alteration Effecting the Los Azules Deposit.

Sequence	Event	Sub Event
	Erosion (Topography)	
	Quaternary cover	
	Silicification	
	Argillic	
	Potassic	
	Sericite	Chlorite-Sericite
		Sericite
	Chlorite (Propylitic)	

Alteration surfaces are built using implicit modelling, interpolating between known contact points (drill hole intercepts) with interpolation parameters fitting with the geological interpretation.

14.3.6 Copper Mineral Zonation Model

The copper mineral zonation model surfaces and solids were constructed using interval selections from the assay, mineralogy, and lithology drill data tables. Interval selections are based primarily on the sequential copper assay data and mathematically defining the min zone category into oxide ('OX'), mixed ('MIX'), supergene enriched ('SG'), transition ('TR'), and hypogene ('HYP'). Table 14.5 shows the criteria used to define each of these zones.

Table 14.5: Mineral Zonation Criteria.	
Sequential Copper Assays (%)	Category
Acid Soluble Copper >= 30%	OX: Oxide
Cyanide Soluble Copper >= 50%	SG: Supergene (Enriched)
Residual Copper Content >= 80%	HYP: Hypogene (Primary)
Cyanide SolCu <= 50% AND Acid SolCu >= 15%	MIX: Mixed
Cyanide SolCu <= 50% AND Residual Cu <= 80%	TR: Transition

The leached zone ('LIX') could not be defined from the sequential copper assay and was assigned based on a combination of the absence of copper and presence of iron oxides. Additional categories of primary bornite ('BN') and primary bornite-chalcopyrite ('BN-CPY') were assigned based on sequential copper assays below the base of the supergene surface and confirmed through logging and visual presence of bornite.

The mixed ('MIX') category has undergone partial oxidation and/or partial leaching of the supergene zone and exhibits repeated fluctuation between oxide and supergene. The transition zone ('TR') is a region of primary copper mineralization that has undergone partial supergene enrichment through repeated fluctuation between hypogene and supergene. Table 14.6 shows the sequence of geological events that have altered and effected the Los Azules deposit.

Table 14.6: Geologic Events Altering and Effecting the Los Azules Deposit	
Sequential	**Event**
↑	Erosion (Topography)
	Quaternary cover
	LIX (Leached)
	MIX / OX (Mixed / Oxide)
	SG (Supergene)
	TR (Transition)
	BN (Primary Bornite)
	BN-CPY (Primary Bornite-Chalcopyrite)
	HYP (Hypogene)

There is not enough material to define an oxide solid. The minor, narrow intervals and amounts of oxide lithology that are present have been grouped into the mixed ('MIX') category.

Estimation domains for the copper resource are the copper mineral zonation models except that the transition surface is eliminated and the definition of the base of the supergene has been defined using the geological logging.

The copper zonation models are bult using implicit modelling, interpolating between known contact points (drill hole intercepts) fitting with the geological interpretation and with subsequent editing guided by sectional interpretations.

14.3.7 Conclusions and Recommendations

The construction methodology of the geological models is extremely robust. It breaks the deposit down into its component events and by understanding each of the controls related to that event, yields a greater understanding of the deposit and a more robust series of inter-related models. The modelling is carried out in sequence: structure – lithology – alteration – mineralization – zonation with iterative revision and reconstruction.

Continued exploration and drilling, especially via angled holes, will serve to better define and improve confidence in the model going forward.

Overall, modelling shows that Los Azules is a large structurally controlled porphyry deposit, open towards the west, northwest and at depth. The extensive supergene enriched zone has developed down structures that transition into primary sulfide mineralization. Modelling shows multiple bornite centers within the primary zone highlighting exploration potential at depth and along the currently modelled structures.

14.4 COMPOSITING

Composites are created from irregular length sample intervals to produce equal length grade data that can be directly compared. To avoid excessively averaging or smoothing the grade data, the composite length is linked to the sampling interval. For previous drilling campaigns irregular sample lengths were sometimes used during the logging and sampling process; however, the most common sampling interval has been 2m. Over 90% of the assayed sample intervals have a sample length of 2m. To preserve the details of the original logging and minimize the amount of grade smoothing, the 2m length was selected for compositing.

Composites are of equal length beginning at the drillhole collar. Within each 2m interval, the majority-logged lithology and mineral zone over the interval are assigned to the composite. Random checks of the composited grades were performed, and no errors were detected.

14.5 EXPLORATORY DATA ANALYSIS

The exploratory data analysis (EDA) is performed to determine the important controls on grades. Use of these controls during estimation will improve model quality and better define the spatial extent of high- and low-grade volumes. Statistical analysis is a key component of the EDA; however, statistical results are only valid over volumes where the statistical distribution of grades are not dependent on location. For this reason, before statistical evaluation, an assessment of the behavior of grades in space is required.

Once the key controls on grade are determined, the deposit is sub-divided into volumes (domains) in which the statistical behavior of grades is consistent. Resource model blocks located within a domain are identified/coded and then estimated using a constant, domain-specific set of estimation parameters.

14.5.1 Behavior of Grades in Space

When viewed in plan, copper grades generally align parallel to the major NNW oriented structure associated with the wetlands/vegas. This association between grades and the structure is shown in Figure 14.4 which is a plan view of the drilling and represents the structure as a straight line.

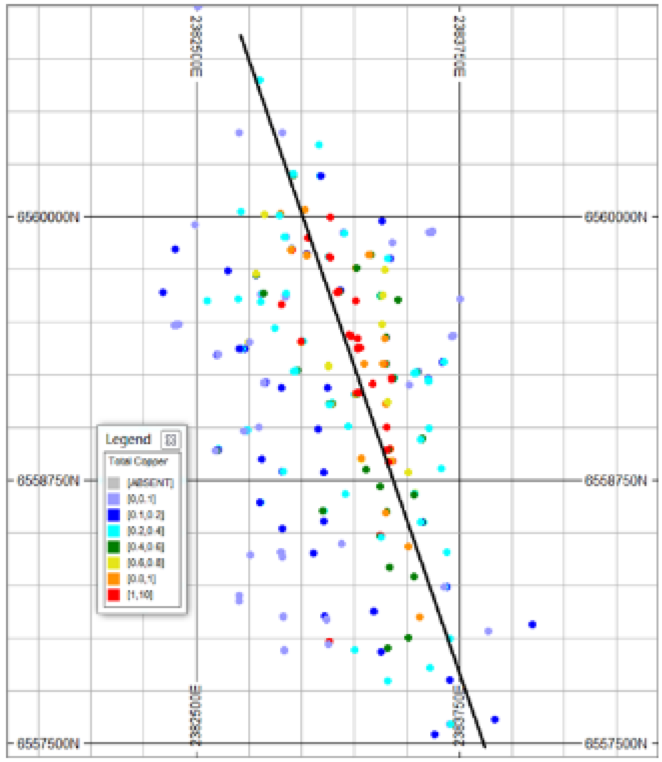


Figure 14.4: Plan Map of Drilling Showing Location of Central Structure

The spatial relationship between the grade and the line can be examined by calculating the horizontal distance from each composite to the line. Average copper can then be examined as a function of distance (Figure 14.5).

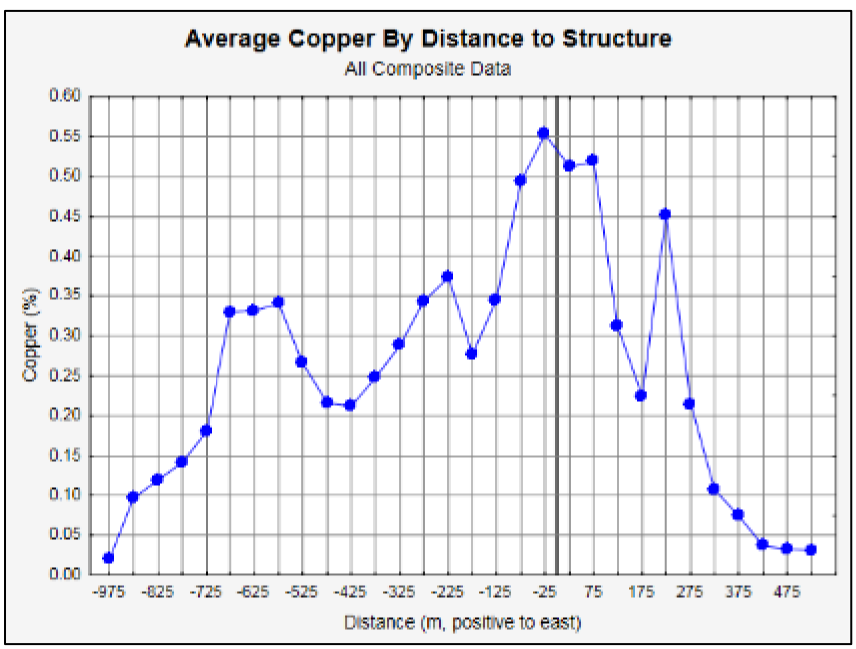


Figure 14.5: Relationship Between Composite Copper Grades and Structure

The relationship between copper grade and the distance to central structure must be considered when evaluating average grade.

The key geologic variables that are qualitatively related to copper grades are lithology and mineral zone. The plots considering lithology and mineral zone are presented in Figure 14.6 and Figure 14.7.

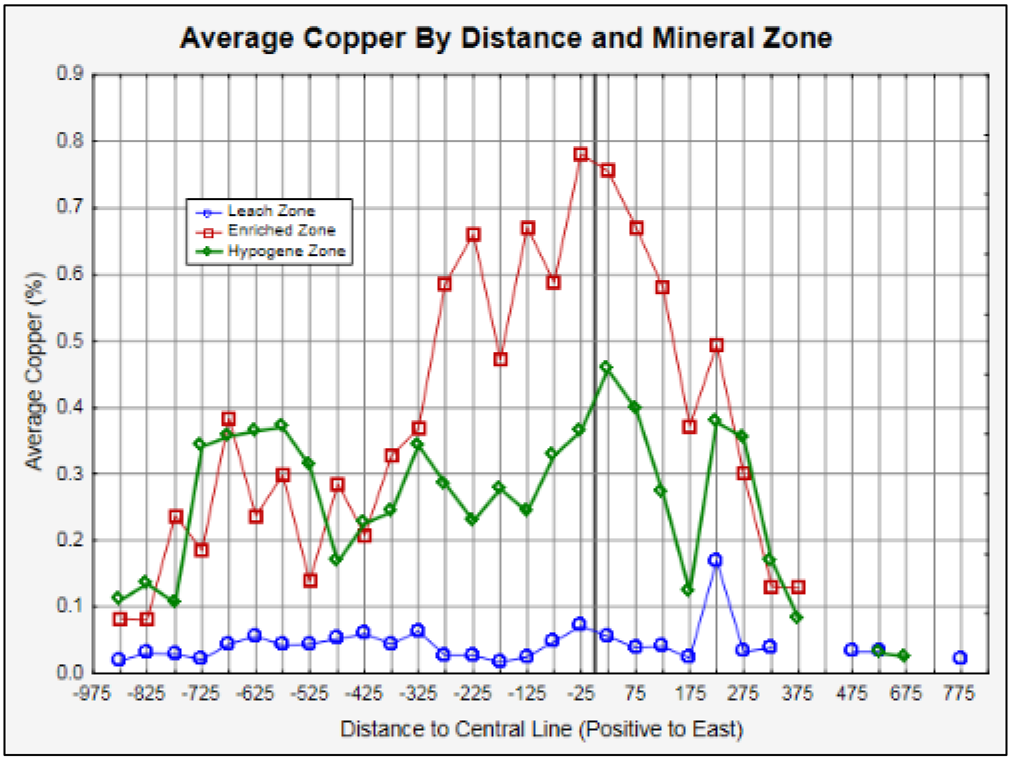


Figure 14.6: Average Copper by Distance and Mineral Zone

Within the leach zone, average total copper grades are consistent across the deposit and are not influenced by the central structure. Within the enriched and hypogene/primary zones, average copper grades are influenced by distance. Due to this dependence on location, global statistics (over all data) are not locally representative.

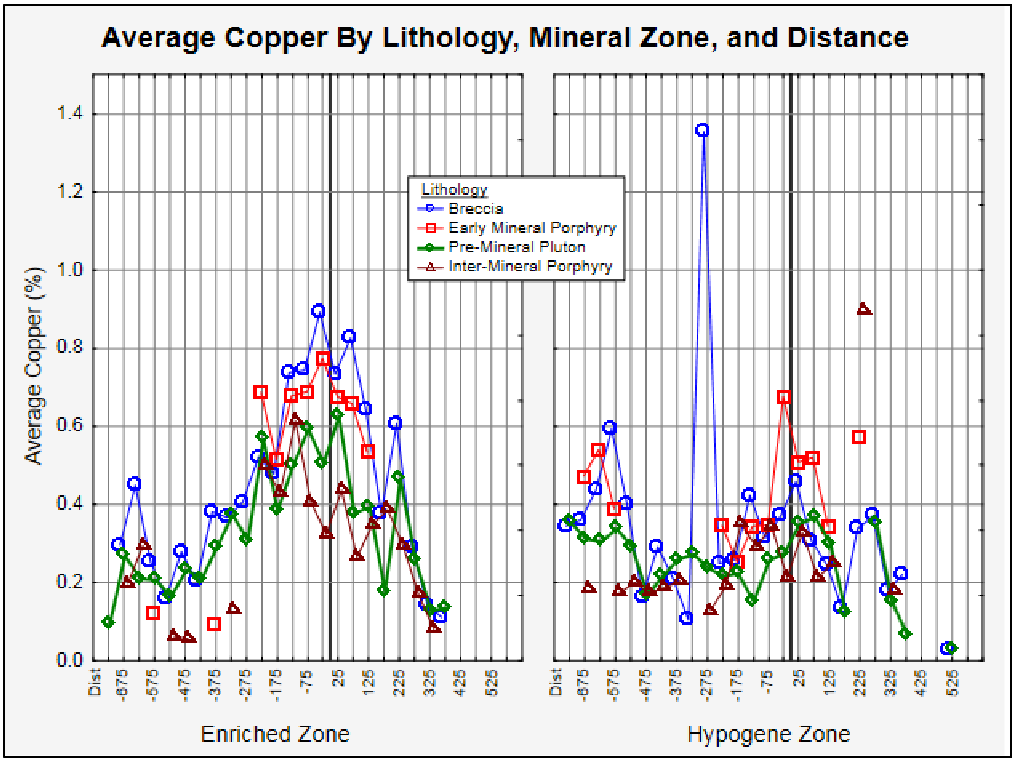


Figure 14.7: Average Copper by Mineral Zone, Lithology and Distance

For the enriched zone, grades are clearly enhanced near the structure. For each lithologic unit, grades are largest near the structure. The highest grades are found in the breccia and early mineral porphyry lithologies. Grades in the inter-mineral porphyry tend to be the lowest. Larger variability is seen in the breccia unit. This is due, in part, to the small number of samples in this unit.

For the hypogene zone, copper grades in all lithologies are elevated near the main structure indicating the continuity of the structure and mineralization at depth. A second high grade zone is found in the western portion of the deposit (distance -600m). This zone of higher grades is associated with a second hypothesized structure at depth. This structure represents a potential future drilling target.

14.5.2 Basic Statistics, Domains, and Coding

Typically, average grades and other statistics are computed by a geologic variable to define estimation domains. This type of analysis, neglecting location, will cause an overstatement of variance and could result in incorrect outlier management decisions. To remove some of the dependence of grades on location, data are divided into three distance groups: <-200m (West), >200m (East) and within 200m of the main structure (Central).

Table 14.7 below considers total copper grade statistics by location and mineral zone as defined by the geologic model.

Table 14.7: Total Copper Statistics by Location and Mineral Zone.

Sector	Mineral Zone	Number of Composites	Average (%)	Standard Deviation (%)	Coefficient of Variation	Minimum	Percentile					Maximum
							25th	50th	75th	95th	98th	
West	Overburden	225	0.017	0.015	0.875	0.003	0.009	0.011	0.017	0.051	0.065	0.086
	Leached	1,545	0.039	0.033	0.841	0.001	0.017	0.030	0.052	0.102	0.140	0.282
	Enriched	3,205	0.424	0.511	1.204	0.005	0.167	0.297	0.501	1.200	1.776	12.89
	Mixed	490	0.192	0.164	0.855	0.010	0.094	0.157	0.246	0.449	0.576	1.773
	Bn-Cpy	792	0.381	0.217	0.569	0.044	0.250	0.329	0.454	0.775	1.040	2.035
	Bornite	244	0.460	0.357	0.776	0.103	0.304	0.403	0.507	0.750	2.050	3.370
	Hypogene	7,138	0.253	0.273	1.075	0.000	0.097	0.178	0.316	0.727	1.057	3.928
Central (Near Structure)	Overburden	90	0.020	0.014	0.709	0.006	0.010	0.016	0.023	0.052	0.066	0.094
	Leached	2,479	0.029	0.032	1.105	0.001	0.012	0.020	0.033	0.078	0.140	0.298
	Enriched	6,439	0.639	0.449	0.702	0.008	0.328	0.533	0.825	1.507	1.910	4.538
	Mixed	372	0.326	0.442	1.356	0.008	0.062	0.125	0.381	1.301	1.692	2.311
	Bn-Cpy	1,123	0.445	0.389	0.874	0.011	0.209	0.345	0.541	1.099	1.500	4.422
	Bornite	346	0.371	0.328	0.883	0.076	0.213	0.293	0.396	1.090	1.447	3.171
	Hypogene	4,511	0.318	0.267	0.840	0.006	0.146	0.258	0.405	0.780	1.085	3.825
East	Overburden	106	0.044	0.027	0.623	0.010	0.024	0.036	0.054	0.091	0.127	0.158
	Leached	1,117	0.032	0.031	0.944	0.001	0.014	0.024	0.040	0.088	0.131	0.253
	Enriched	640	0.407	0.279	0.686	0.019	0.189	0.365	0.545	0.899	1.187	1.824
	Mixed	21	0.157	0.230	1.459	0.018	0.037	0.085	0.133	0.530	0.957	0.957
	Bn-Cpy	0										
	Bornite	0										
	Hypogene	607	0.183	0.182	0.997	0.005	0.037	0.137	0.242	0.576	0.745	1.000
All Data		31,490	0.334	0.384	1.149	0.000	0.080	0.228	0.444	1.049	1.442	12.89

After partially controlling for location, average grades show important differences among the various mineral zones. Some of the minor mineral zones (mixed, Bn-Cpy, and bornite) are less common in the east.

Basic statistics by lithology are also presented by sector in Table 14.8

Table 14.8: Total Copper Statistics by Lithology and Sector

Sector	Mineral Zone	Number of Composites	Average (%)	Standard Deviation (%)	Coefficient of Variation	Minimum	Percentile					Maximum
							25th	50th	75th	95th	98th	
West	Hydro Brx	2,111	0.320	0.337	1.054	0.002	0.120	0.230	0.396	0.952	1.258	3.392
	Early Min. Porphyry	307	0.202	0.339	1.683	0.000	0.006	0.017	0.292	0.936	1.174	1.786
	PreMineral Pluton	8,907	0.237	0.230	0.972	0.001	0.082	0.180	0.323	0.618	0.855	4.04
	Inter-Min. Porphyry	470	0.136	0.113	0.831	0.002	0.055	0.115	0.174	0.361	0.446	0.771
	Overburden	215	0.016	0.013	0.832	0.003	0.008	0.011	0.017	0.047	0.061	0.070
	Mag. Hyd. Breccia	777	0.744	0.870	1.169	0.003	0.275	0.517	0.942	2.164	2.737	12.886
	Late Qtz Veins	698	0.318	0.352	1.106	0.003	0.111	0.201	0.420	0.883	1.320	3.928
Central (Near Structure)	Hydro Brx	2,957	0.476	0.416	0.874	0.002	0.168	0.376	0.680	1.329	1.627	3.058
	Early Min. Porphyry	2,194	0.436	0.413	0.947	0.002	0.046	0.366	0.650	1.182	1.448	4.422
	PreMineral Pluton	5,070	0.280	0.287	1.025	0.001	0.085	0.205	0.381	0.792	1.108	4.021
	Inter-Min. Porphyry	2,027	0.289	0.232	0.801	0.001	0.129	0.264	0.404	0.681	0.936	2.521
	Overburden	94	0.020	0.014	0.709	0.006	0.010	0.016	0.023	0.048	0.066	0.094
	Mag. Hyd. Breccia	2,352	0.756	0.569	0.753	0.005	0.372	0.623	1.022	1.870	2.278	4.538
	Late Qtz Veins	616	0.361	0.280	0.775	0.007	0.167	0.297	0.472	0.916	1.110	1.692
East	Hydro Brx	773	0.253	0.284	1.125	0.006	0.038	0.142	0.377	0.810	1.067	1.824
	Early Min. Porphyry	7	0.573	0.094	0.165	0.476	0.510	0.543	0.600	0.766	0.766	0.766
	PreMineral Pluton	949	0.092	0.134	1.462	0.002	0.021	0.038	0.101	0.366	0.530	0.899
	Inter-Min. Porphyry	213	0.183	0.237	1.297	0.003	0.019	0.046	0.275	0.683	0.861	1.000
	Overburden	104	0.043	0.027	0.628	0.010	0.024	0.036	0.054	0.091	0.127	0.158
	Mag. Hyd. Breccia	161	0.385	0.266	0.690	0.002	0.192	0.372	0.547	0.830	1.000	1.261
	Late Qtz Veins	41	0.110	0.095	0.863	0.012	0.022	0.072	0.188	0.280	0.317	0.317
All Data		31,043	0.338	0.386	1.142	0.000	0.084	0.231	0.448	1.055	1.449	12.89

The western sector is dominated (in terms of number of data) by the pre-mineral pluton. The early mineral porphyry is more common near the structure while the inter-mineral porphyry is more common to the east of the structure. The average grade of the high-grade breccia units is clearly lower in the eastern sector.

Elevated grades are observed in the two breccia units. Based on general similarity in grades and occurrence, the hydrothermal and magmatic hydrothermal breccia were combined for purposes of estimation.

The late quartz veins can have elevated grades (particularly for precious metals); however, this unit was not defined in the geologic model due to a lack of observable continuity. Composites with this logged code were restricted during estimation.

Based on this information, estimation domains (with minor exceptions) were created by combining mineral zone and lithology. The selected domains and numerical codes for copper estimation are:

- 2101 – Leach/Breccia
- 2102 – Leach/Early Mineral Porphyry
- 2103 – Leach/Pre-Mineral Pluton
- 2104 – Leach/Inter-mineral Porphyry
- 3101 – Enriched/Breccia
- 3102 – Enriched/Early Mineral Porphyry
- 3103 – Enriched/Pre-Mineral Pluton
- 3104 – Enriched/Inter-mineral Porphyry
- 4101 – Mixed/Breccia
- 4102 – Mixed/Early Mineral Porphyry
- 4103 – Mixed/Pre-Mineral Pluton
- 4104 – Mixed/Inter-mineral Porphyry
- 5101 – Bn – Cpy/Breccia
- 5102 – Bn – Cpy/Early Mineral Porphyry
- 5103 – Bn – Cpy/Pre-Mineral Pluton
- 5104 – Bn – Cpy/Inter-mineral Porphyry
- 6101 – Bornite/Breccia
- 6102 – Bornite/Early Mineral Porphyry
- 6103 – Bornite/Pre-Mineral Pluton
- 6104 – Bornite/Inter-mineral Porphyry
- 7101 – Hypogene/Breccia
- 7102 – Hypogene/Early Mineral Porphyry
- 7103 – Hypogene/Pre-Mineral Pluton
- 7104 – Hypogene/Inter-mineral Porphyry
- 105 – Overburden
- 107 – Volcanics

14.5.3 Behavior Near Contacts

Grades within specified estimation domains can vary as a function of distance to a contact when a halo of mineralization or other type of transition occurs between domains. When grades transition across domain boundaries it may be appropriate to share samples across the boundary during estimation to preserve the transition in the model. If sharp changes in grade are observed across a contact, the sharing of samples is inappropriate.

To observe whether grades near contacts are transitional, two types of analysis were performed. First, large blocks that straddle the contact were defined and average grades on the two sides of the contact were compared. This analysis evaluates grades near the contact over the range of observed values and provides an indication of the grades that would be combined if there were free sharing of grades during estimation. The second check is more localized; the distance between each composite and the nearest model block (from a different domain) is determined. Average grades are then computed and plotted. Although more localized, this approach combines data over the entire deposit.

As an example of the analysis performed, the behavior of different lithologies in the enriched zone is considered. Figure 14.8 shows a matrix of scatter plots that show the results of analysis of composites on the two sides of contacts between the various domains within the model.

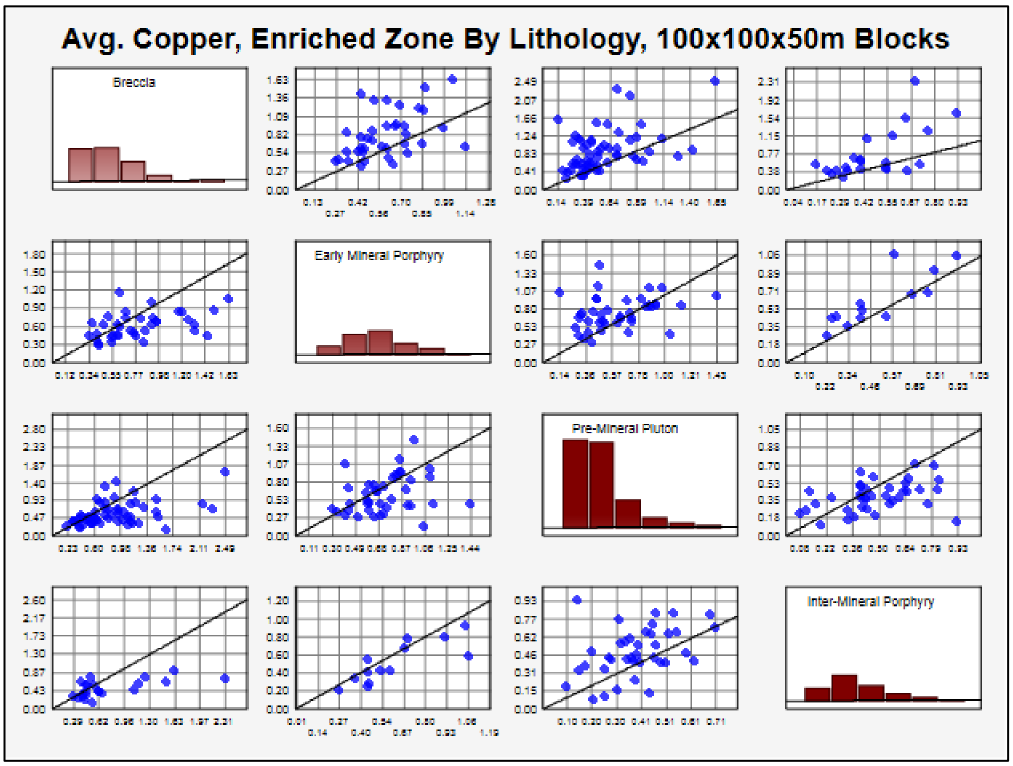


Figure 14.8: Cross Contact Composite Comparison

In this plot each point represents average grades on two sides of a contact. The labeled histograms on the diagonal define the lithology considered on the X and Y axes. Thus, the scatterplot in the upper right corner of the plot considers blocks containing breccia and inter-mineral porphyry data. In this case, the grades in the breccia are greater and independent of the grades in the porphyry. Sharing samples across this boundary is not appropriate.

For the other combinations of units, the average grades on the two sides of the contact are different. The most similar are grades on the two sides of the breccia/early mineral porphyry contact. The grades across the pre-mineral pluton and inter-mineral porphyry are also similar. For these two contacts, the detailed contact plots are shown Figure 14.9.

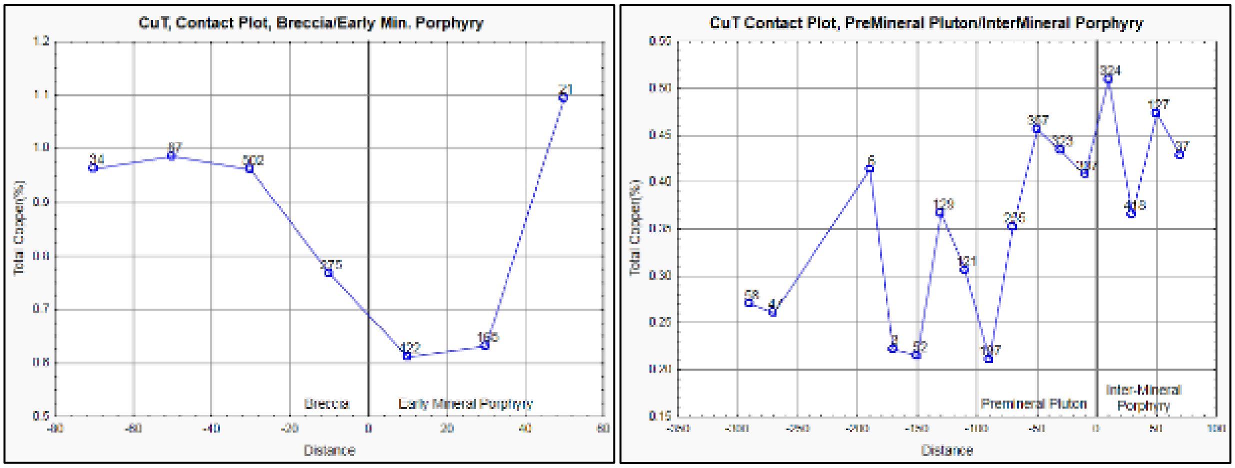


Figure 14.9: Detailed Cross Contact Composite Grade / Distance Analysis

For the Breccia/EMP contact, grades drop from about 0.8% to 0.6% at the contact. At the Pre-Mineral Pluton/Intermineral Porphyry contact grades increase from about 0.4% to 0.5%. Based on these changes in grades near the contact and the behavior of average grades in the previously presented contact scatterplots (Figure 14-8), these contacts and all other lithology contacts are treated as hard boundaries during estimation with no sharing of samples between lithology units.

14.5.4 Cyanide Soluble Copper EDA

Sequential copper determinations for acid soluble copper (CuAS) and cyanide soluble copper (CuCN) are provided by a standard sequential copper assay methodology by the Alex Stewart laboratory. The methodology is comprised of a dilute sulfuric acid-controlled leach and assaying of the resulting dissolved copper followed by a dilute sodium cyanide-controlled leach and assaying of the resulting dissolved copper. Residual copper in the final tails is processed in a four-acid digestion and assayed to determine the total copper content. Acid soluble and cyanide soluble assays are combined to determine the approximate soluble copper content (CuSOL) of each sample based on the methodology. The sequential total copper assay determination is only used for comparative purposes and the total copper assayed in the drill program procedure is used in the drilling database.

Cyanide soluble (CuCN) grades are directly linked to the expected metal production for the leach project. For this reason, a detailed evaluation of both CuCN grade and the ratio of CuCN to total copper (solubility) was performed.

A very important control on solubility is mineral zone since solubility is linked to copper mineralogy. Figure 14.10 presents box plots of CuCN by mineral zone.

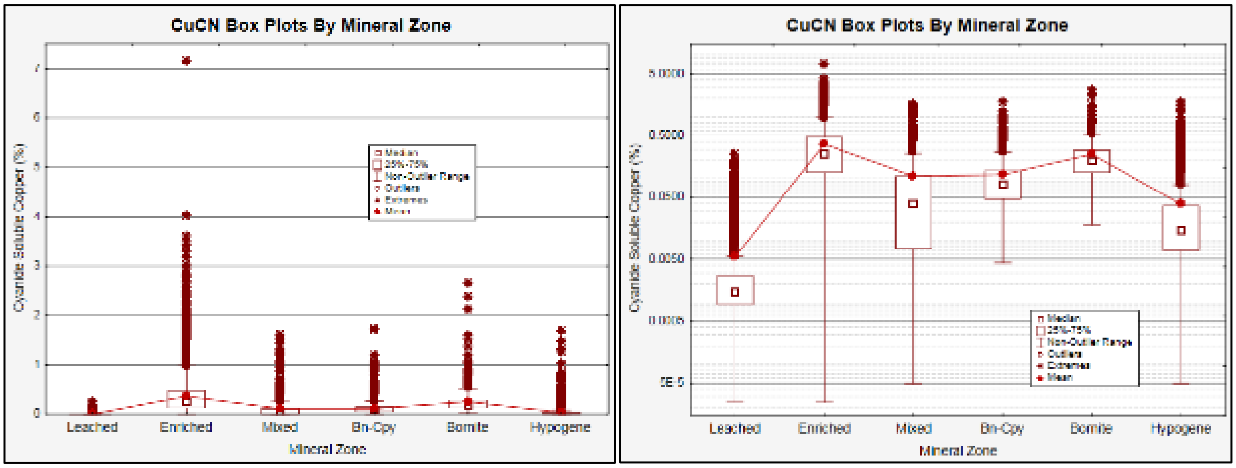


Figure 14.10: Box Plots of CuCN by Mineral Zone

The two plots are identical only the vertical scale is different. As seen, only the enriched and bornite zones show an important quantity of elevated CuCN grades. For the other mineral zones, at least 75% of the data have grades less than 0.15%.

The central structure also provides an important control on CuCN grade. Figure 14.11 presents average grades as a function of distance to the line representing the structure. The data is further broken down by mineral zone as defined in the legend.

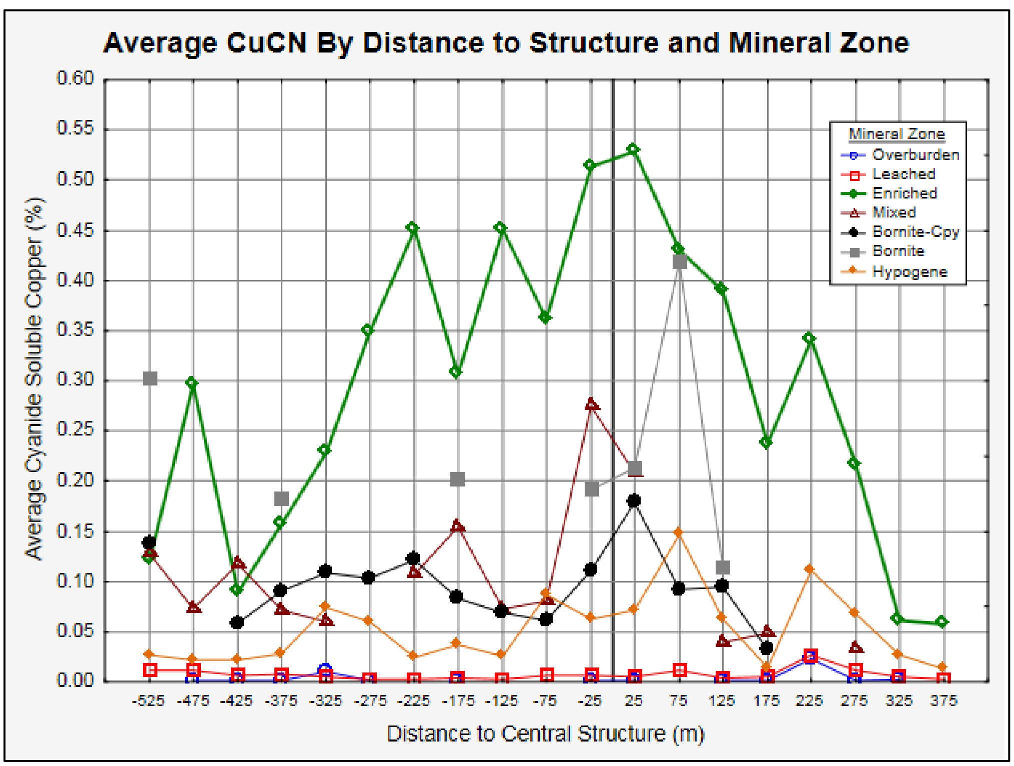


Figure 14.11: Average CuCN grades by Distance from Central Structure.

The lateral behavior of CuCN grades is similar to total copper in the enriched zone. The best primary zone grades are found in the bornite zone, unfortunately this zone is not common. Grades in the leached and hypogene zone are low; however, hypogene grades do increase slightly near and to the east of the structure. Table 14.9 presents CuCN statistics by mineral zone.

Table 14.9: Basic Statistics – CuCN by Mineral Zone.

Mineral Zone	Number of Composites	Average (%)	Standard Deviation (%)	Coefficient of Variation	Minimum	Percentile					Maximum
						25th	50th	75th	95th	98th	
Overburden	419	0.002	0.006	3.220	0.00014	0.001	0.001	0.002	0.004	0.010	0.091
Leached	4,827	0.005	0.016	2.925	0.00003	0.001	0.002	0.003	0.024	0.056	0.251
Enriched	10,190	0.362	0.365	1.008	0.00003	0.126	0.251	0.470	1.070	1.422	7.150
Mixed	864	0.112	0.209	1.868	0.00005	0.007	0.039	0.109	0.503	0.851	1.609
Bn-Cpy	1,913	0.117	0.135	1.162	0.004	0.048	0.080	0.136	0.326	0.515	1.725
Bornite	590	0.249	0.241	0.966	0.018	0.126	0.195	0.289	0.569	0.904	2.648
Hypogene	11,961	0.038	0.073	1.893	0.00005	0.007	0.015	0.036	0.160	0.267	1.697
All Data	30,764	0.151	0.271	1.794	0.00003	0.006	0.037	0.181	0.658	1.024	7.150

The number of CuCN composites is less than the number of total copper composites since CuCN assays were not consistently requested in all historical drilling programs. In the important enriched zone, there are 10,190 CuCN composites as compared to 10,284 total copper composites.

For composites assayed by both methods, Figure 14.12 shows the correlation between soluble and total copper for sulfide mineral zones.

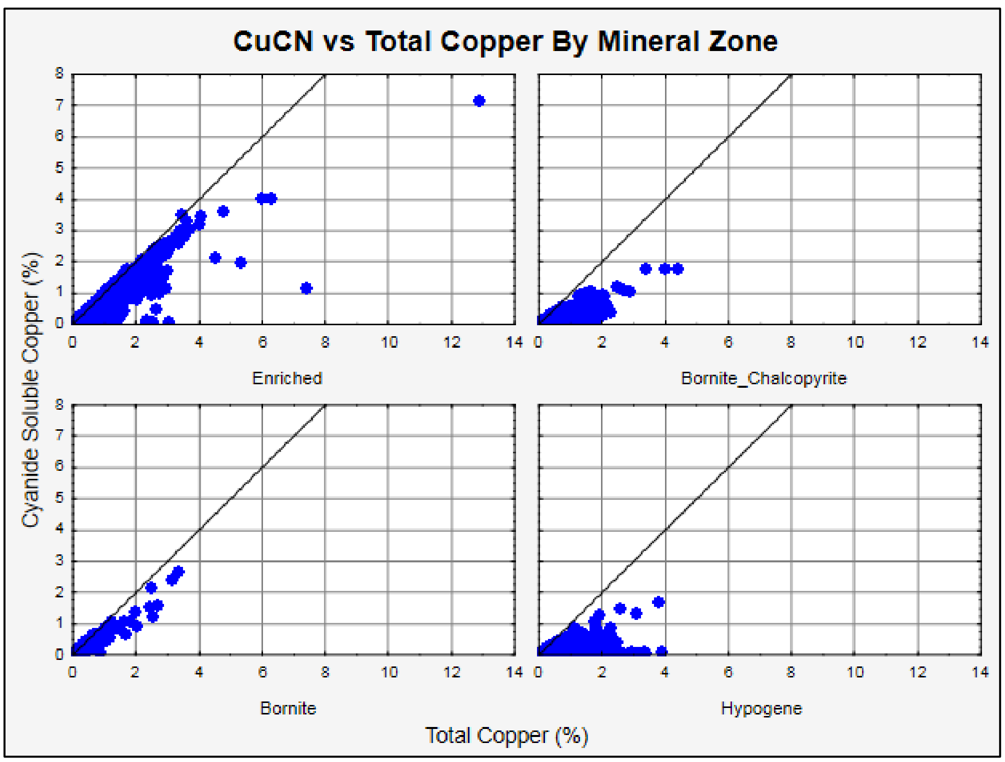


Figure 14.12: Scatter plots of Total Copper VS Cyanide Soluble Copper

For the bornite zone, the relationship between the two types of copper is close to linear. For the hypogene and bornite-chalcopyrite zone correlation is very weak. There is an important level of correlation in the enriched zone; however, the slope of the line (solubility ratio) is not constant.

The solubility ratio is dependent on copper mineralogy. From the base of the leach to the top of primary zone, a change in copper mineralogy is seen. Near the base of the leach zone, chalcocite and possibly copper oxides are observed. Near the base of the enriched zone, a mixture of chalcocite and chalcopyrite is found. Due to this change in mineralogy, solubility is expected to vary across the vertical thickness of the enriched zone.

To check for a vertical trend in solubility, the relative depth of each composite (depth below top of enriched/total vertical thickness) was determined and statistics by relative depth interval were calculated.

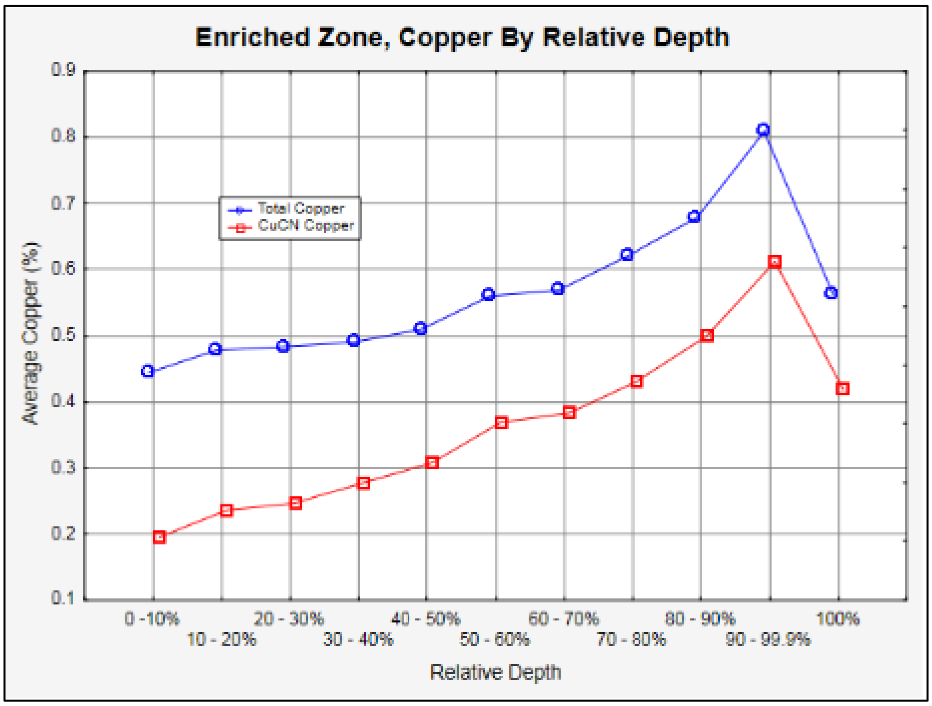


Figure 14.13: Graph Showing Relationship Between Copper Grades and Depth.

Moving from a relative depth of 0 to 100% equates to moving from the base of the enriched zone to the top. Due to model resolution, the uppermost group in some cases captures samples that could be partially leached; as the enriched zone thins (associated with lower grade) more samples report to the 100% bin (as an example, if the enriched zone thickness is only 1 block, all composites from the drillhole report to the 100% group).

The plot clearly shows that average CuCN grades decrease more rapidly with depth than total copper in the enriched zone. This unequal decrease in average grades results in a reduction in CuCN solubility with depth.

14.5.5 Gold and Silver

The spatial distribution of gold and silver grades is like that of copper; however, the precious metals show high grade values associated with assumed localized structures and veins (Late-Stage Quartz Veins identified in logging) that are independent of copper. Scatterplots of average grades over large blocks show these features of the correlation.

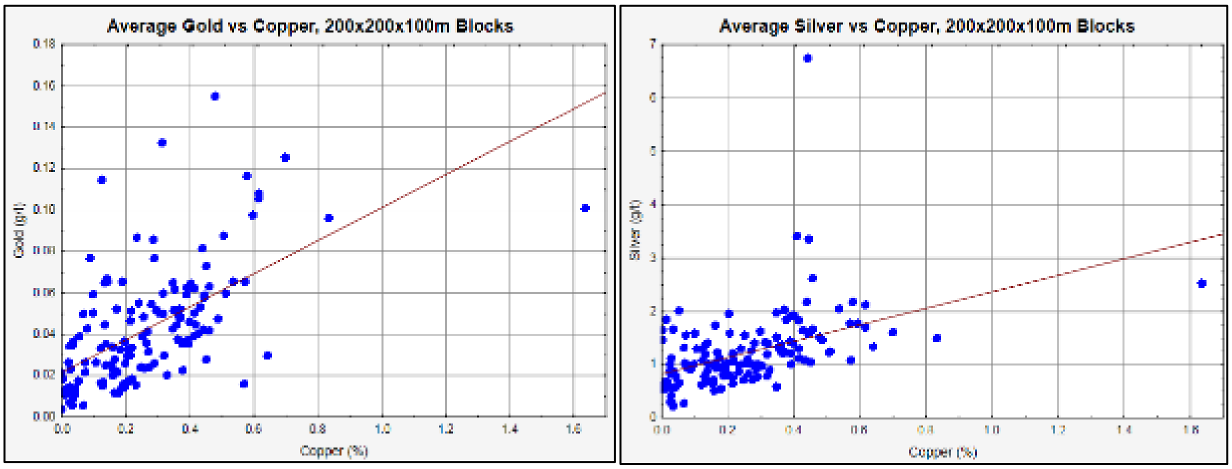


Figure 14.14: Scatter plots of Precious Metals vs Total Copper.

Grade capping of precious metal grades will be required to address the local grade variability.

Table 14.10: Basic Statistics for Gold Grades by Mineral Zone.

Mineral Zone	Number of Composites	Average (%)	Standard Deviation (%)	Coefficient of Variation	Minimum	Percentile					Maximum
						25th	50th	75th	95th	98th	
Overburden	421	0.030	0.032	1.055	0.005	0.010	0.020	0.040	0.100	0.120	0.220
Leached	5,140	0.052	0.103	1.961	0.003	0.010	0.030	0.070	0.150	0.190	3.590
Enriched	10,284	0.057	0.121	2.132	0.003	0.020	0.040	0.070	0.150	0.220	8.720
Mixed	883	0.053	0.089	1.667	0.003	0.010	0.030	0.070	0.150	0.220	1.290
Bn-Cpy	1,915	0.063	0.106	1.677	0.005	0.020	0.040	0.070	0.160	0.240	2.290
Bornite	590	0.060	0.080	1.335	0.005	0.030	0.040	0.070	0.140	0.210	0.980
Hypogene	12,256	0.043	0.172	4.036	0.003	0.010	0.020	0.040	0.120	0.190	9.160
All Data	31,489	0.051	0.138	2.735	0.003	0.010	0.030	0.060	0.140	0.200	9.160

Table 14.11: Basic Statistics for Silver Grades by Mineral Zone.

Mineral Zone	Number of Composites	Average (%)	Standard Deviation (%)	Coefficient of Variation	Minimum	Percentile					Maximum
						25th	50th	75th	95th	98th	
Overburden	421	0.601	0.758	1.262	0.25	0.25	0.25	0.70	1.50	2.40	8.23
Leached	5,141	0.940	1.987	2.115	0.10	0.50	0.50	0.83	2.50	4.38	65.5
Enriched	10,284	1.518	10.115	6.661	0.15	0.50	0.70	1.50	4.30	6.60	954
Mixed	883	1.471	3.170	2.156	0.15	0.50	0.55	1.30	4.90	10.1	49.7
Bn-Cpy	1,915	1.908	2.876	1.507	0.15	0.60	1.20	2.20	5.30	8.50	53.8
Bornite	590	1.739	1.730	0.995	0.25	0.80	1.30	2.10	4.10	6.80	19.6
Hypogene	12,256	1.418	3.456	2.437	0.10	0.50	0.60	1.30	4.50	7.70	172
All Data	31,490	1.399	6.294	4.499	0.10	0.50	0.60	1.40	4.20	7.00	954

As compared to total copper, precious metal grades show more variability, and the mineral zone provides a weaker control on grades.

14.6 BULK DENSITY

Density was estimated using the same 2190 density measurements (by the water immersion method) available for the 2017 PEA model. No density data was collected during the 2022 field program.

Density data was coded for lithology and mineral zone using the block model and a statistical analysis was performed. High and low density outliers were defined. Density values less than 2 g/cc were set to 2 g/cc (1 sample) and values larger than 2.9 g/cc were set to 2.9 g/cc (1 sample).

Density was estimated by mineral zone. Summary statistics are presented in Table 14.12.

Table 14.12: Basic statistics of Density by Mineral Zone

Basic Statistics - Density By Mineral Zone						
Mineral Zone	Number of Data	Average (g/cc)	Standard Deviation (g/cc)	Coefficient of Variation	Minimum	Maximum
Overburden	0					
Leached	418	2.49	0.142	0.057	2.00	2.90
Secondary Enrichment	607	2.53	0.101	0.040	2.00	2.80
Mixed	63	2.55	0.106	0.042	2.00	2.70
Bn-Cpy	120	2.59	0.078	0.030	2.40	2.80
Bn	30	2.63	0.080	0.030	2.40	2.80
Hypogene	952	2.59	0.086	0.033	2.20	2.90
All Data	2,190	2.55	0.110	0.043	2.00	2.90

A value of 2.4 g/cc was assigned to overburden.

Density was estimated separately for each mineral zone. Inverse distance squared estimation was used for each estimate. The search was anisotropic aligned parallel to the main structures (N20W) with radii of 150m along strike, 100 m across strike, and 100m vertically. A minimum of 4 and a maximum of 10 samples were used with the additional restriction that a maximum of 3 samples per drillhole were used. This restriction requires that data come from at least 2 drillholes to estimate a block. For un-estimated blocks, the search was expanded by a factor of 2 and then 3. Blocks that remained un-estimated were assigned the average density of the appropriate mineral zone.

14.7 EVALUATION OF OUTLIER GRADES

Grades per estimation domain show an important spatial association with the central NNW structure. As a result, the distribution of grades, including the values of the upper percentiles, vary by location; standard approaches to defining outliers (such as examining the global distribution of grades) are thus not appropriate. When average grades per domain are variable in space, outliers must be identified and managed based on local statistics. The capping approach taken compares local grades with and without each composite grades. Those composites associated with a substantial change in local grade are identified as outliers. Local outliers are capped to a value that is consistent with the neighboring grades. Local capped grades were defined for copper, gold, and silver per estimation domain. Table 14.13 presents the potential metal removal due to capping.

Table 14.13: Potential Effect of Capping on Copper, Gold, and Silver Content.

Mineral Zone	Number of Composites	Average Copper (%)		
		Uncapped	Capped	Metal Removal
Leached	5,155	0.033	0.033	0.0%
Enriched	10,284	0.558	0.556	0.3%
Mixed	869	0.250	0.250	0.0%
Bn-Cpy	1,915	0.418	0.416	0.6%
Bornite	590	0.408	0.408	0.0%
Hypogene	12,256	0.274	0.274	0.0%
All Data	31,069	0.339	0.338	0.2%

Mineral Zone	Number of Composites	Average Gold (g/t)		
		Uncapped	Capped	Metal Removal
Leached	5,154	0.052	0.050	3.8%
Enriched	10,284	0.057	0.053	6.4%
Mixed	869	0.053	0.049	7.9%
Bn-Cpy	1,915	0.063	0.059	6.1%
Bornite	590	0.060	0.056	7.5%
Hypogene	12,256	0.043	0.037	12.1%
All Data	31,068	0.051	0.047	7.9%
Mixed Zone capped to 0.1 g/t				

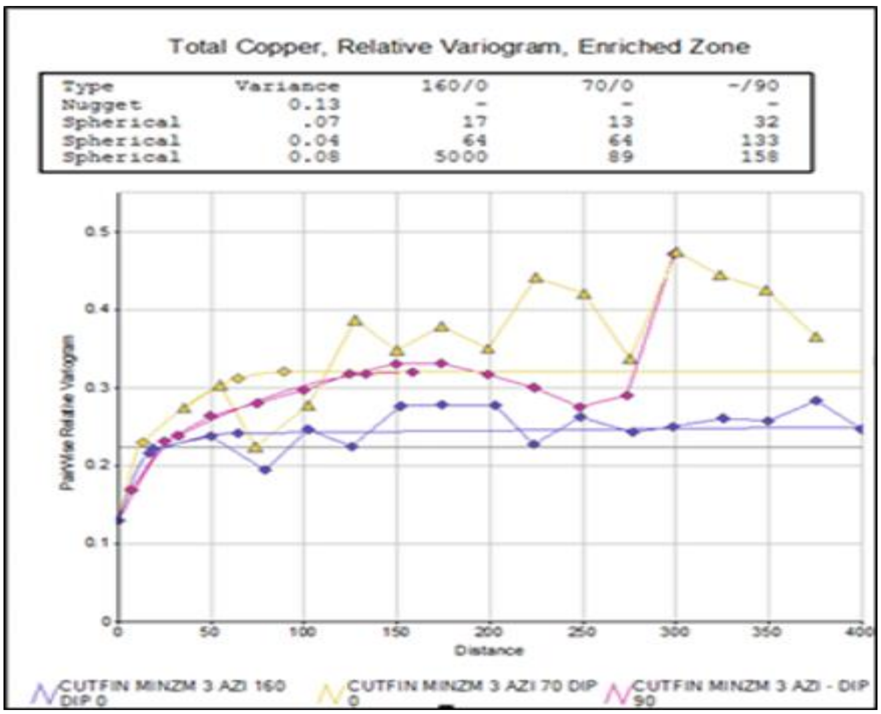
Mineral Zone	Number of Composites	Average Silver (g/t)		
		Uncapped	Capped	Metal Removal
Leached	5,155	0.938	0.828	11.7%
Enriched	10,284	1.518	1.298	14.5%
Mixed	869	1.488	1.325	11.0%
Bn-Cpy	1,915	1.908	1.868	2.1%
Bornite	590	1.739	1.726	0.7%
Hypogene	12,256	1.418	1.309	7.7%
All Data	31,069	1.410	1.268	10.0%
Mixed Zone capped to 9.0 g/t				

Based on the very low metal removals seen for total copper, the resource estimate is based on uncapped copper grades. Local capped grades are used for gold and silver except for the mixed zone. Due to the small number of data, the mixed zone composites were capped at approximately the 99th percentile of the grade distribution.

In addition to capping, a small number of composites were restricted for all elements. Two types of composites requiring restriction were identified: logged late quartz veins and breccia composites more than 22.5 m (1.5 bench heights) from a breccia block. The restricted composites only participated in the estimation of the block containing the composite.

14.8 VARIOGRAPHY

Experimental variograms were computed by mineral zone. Given the observed spatial grade trends, variability is expected to be dependent on direction and the variograms will show a zonal anisotropy. To best show the anisotropy, variograms rather than correlograms are preferred. To reduce noise associated with local variability, pairwise relative variograms were computed and modeled.



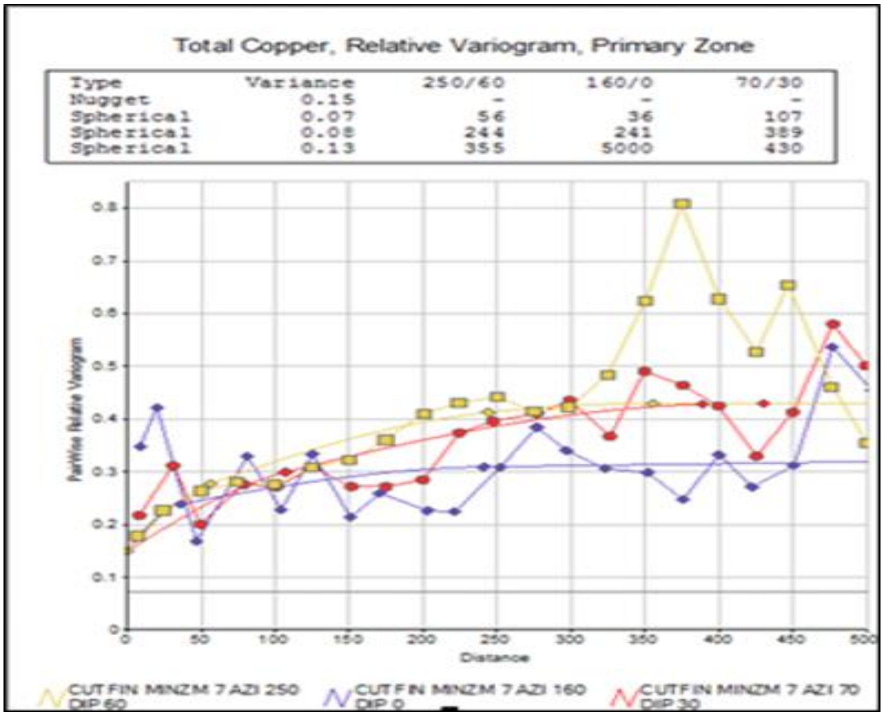


Figure 14.15: Experimental Data and Modeled Variogram.

For the enriched zone the strongest correlation is observed parallel to the strike direction of the steeply dipping central structure. The largest variability is seen perpendicular to the plane due to the grade trend in this direction. Experimental variograms parallel to possible dip directions showed less correlation than the vertical variogram.

For the primary/hypogene zone, the strongest correlation remains parallel to the NNW structure. The downdip and perpendicular-to-the-plane variograms show similar levels of correlation.

Table 14.14: Presents the Variogram Model Parameters for Copper, Gold, and Silver.

Copper

Mineral Zone	Axis Orientation (trend/plunge) Major	Minor	Semi-Major	Nugget Effect	Spherical Structure 1 Variance (C Value)	Distances (m) Major	Minor	Semi	Spherical Structure 2 Variance (C Value)	Distances (m) Major	Minor	Semi	Spherical Structure 3 Variance (C Value)	Distances (m) Major	Minor	Semi
Enriched	N20W/0	N70E/0	0/90	0.13	0.07	17	13	32	0.04	64	64	133	0.08	5000	89	158
Leach	N20W/0	N70E/0	0/90	0.1	0.1	32	88	34	0.07	320	133	69	0.05	5000	133	69
Primary	N20W/0	250/60	70/30	0.15	0.07	36	56	107	0.08	241	241	389	0.13	5000	355	430

Gold

Mineral Zone	Axis Orientation (trend/plunge) Major	Minor	Semi-Major	Nugget Effect	Spherical Structure 1 Variance (C Value)	Distances (m) Major	Minor	Semi	Spherical Structure 2 Variance (C Value)	Distances (m) Major	Minor	Semi	Spherical Structure 3 Variance (C Value)	Distances (m) Major	Minor	Semi
Enriched	N20W/0	N70E/0	0/90	0.15	0.08	29	23	13	0.1	175	111	82	0.12	5000	187	5000
Leach	N20W/0	N70E/0	0/90	0.14	0.06	25	84	23	0.05	69	98	211	0.2	5000	122	5000
Primary	N20W/0	70/30	250/60	0.2	0.2	131	117	63	0.1	1200	437	482	0.1	5000	491	1200

Silver

Mineral Zone	Axis Orientation (trend/plunge) Major	Minor	Semi-Major	Nugget Effect	Spherical Structure 1 Variance (C Value)	Distances (m) Major	Minor	Semi	Spherical Structure 2 Variance (C Value)	Distances (m) Major	Minor	Semi	Spherical Structure 3 Variance (C Value)	Distances (m) Major	Minor	Semi
Enriched	N20W/0	N70E/0	0/90	0.23	0.1	42	41	35	0.12	55	91	266	0.07	5000	200	700
Leach	N20W/0	N70E/0	0/90	0.15	0.05	44	27	33	0.05	89	36	115	0.1	133	51	5000
Primary	N70E/0	N20W/0	250/60	0.2	0.1	22	22	12	0.1	53	39	64	0.1	318	207	196

14.9 MODEL SETUP AND LIMITS

The resource block model was developed in DATAMINE Studio software. Table 14.15 presents the dimensions and limits of the model. The Posgar 94 coordinate system was used. The block size of 20x20x15m is consistent with the typical selective mining unit (SMU) used for this type of copper deposit.

Table 14.15: Block Model Origin and Dimensions.

Direction	Minimum	Maximum	Block Dimension	Number of Blocks
West - East	2,380,711.108	2,385,711.108	20	250
South - North	6,556,193.157	6,562,093.157	20	295
Elevation	2605	4390	15	119

Model blocks are assigned lithology and mineral zone codes based on the block centroid and the wireframe models from the geologic model. Sub-blocks were not used. The model contains a field defining the percentage of the block below surface topography.

14.10 INTERPOLATION PARAMETERS

The block model grades were estimated using a combination of ordinary Kriging and inverse distance squared weighting. Inverse distance weighting was used for the smaller domains where a variogram model could not be developed (mixed zone and overburden). The estimation search uses multiple passes with decreasing restrictions to allow estimation of a large proportion of the model blocks. The search pass where each block is estimated is stored in the model output file.

Different, but similar, searches were used for copper and the precious metals. The search strategy for copper is presented to illustrate the approach. The first three search passes use an octant restriction requiring that 4 of the octants surrounding the block contain data. This increases the likelihood that data will surround the block. If a block cannot be estimated in the first three passes, estimation based on a minimum number of drillholes near the block is performed. Three drillhole restriction search passes are considered yielding a total of six estimation searches, the first three search passes with octants.

Table 14.16: Search Strategy for Copper Estimation, Pass 1 to 3														
Mineral Zone	Search Distance By Direction			Octant Restriction			# of Composites		Search Expansion 1			Search Expansion 2		
	N20W/0	N70E/0	0/90	Octants Filled	Min. Per Octant	Max. Per Octant	Minimum	Maximum	Factor	Minimum	Maximum	Factor	Minimum	Maximum
Overburden	100	100	100	4	1	7	15	30	2	15	30	3	8	20
Leach	100	100	100	4	1	7	15	30	2	15	30	3	8	20
Enriched	150	75	25	4	1	7	15	30	2	15	30	3	8	20
Mixed	150	75	50	4	1	7	15	30	2	15	30	3	8	20
Bn-Cpy	150	75	50	4	1	7	15	30	2	15	30	3	8	20
Bornite	150	75	50	4	1	7	15	30	2	15	30	3	8	20
Hypogene	150	75	50	4	1	7	15	30	2	15	30	3	8	20

The second three searches (search passes 4 through 6) are presented in Table 14.17.

Table 14.17: Search Strategy for Copper Estimation, Pass 4 to 6												
Mineral Zone	Search Distance By Direction			Max. Samples Per Hole	# of Composites		Search Expansion 1			Search Expansion 2		
	N20W/0	N70E/0	0/90		Minimum	Maximum	Factor	Minimum	Maximum	Factor	Minimum	Maximum
Overburden	300	300	300	5	10	24	2	10	24	3	10	24
Leach	300	300	300	8	12	24	2	12	24	3	12	24
Enriched	450	225	75	8	12	24	2	12	24	3	12	24
Mixed	450	225	150	8	12	24	2	12	24	3	12	24
Bn-Cpy	450	225	150	8	12	24	2	12	24	3	12	24
Bornite	450	225	150	8	12	24	2	12	24	3	12	24
Hypogene	450	225	150	8	12	24	2	12	24	3	12	24

Anisotropic searches parallel to the major NNW structure are used outside of the overburden and leach. A smaller vertical search is used in the enriched zone due to the observed vertical trend in copper solubility.

The same search strategy and variogram models were used for both total and soluble copper to avoid generating solubility artifacts in the estimates. Not all composites have both total and soluble copper grades since soluble copper assays were not performed in some of the past drilling campaigns. To account for the missing data, copper is estimated in two passes:

- Estimate grades using only the composite data with both soluble and total copper.
- Estimate grades using all total copper grades (final total copper estimate).
- To obtain the final soluble copper grade, compute the ratio of soluble to total copper (from the first estimate), then multiply this ratio by the final total copper estimate.

In this approach there is an implicit assumption of solubility stationarity (invariability of the average in space). Within the enriched zone, solubility is clearly a function of proximity to the upper and lower contacts of the unit. To account for this natural change in mineralogy/solubility, the enriched zone copper estimates are restricted to relative depth bands and the vertical search is reduced.

14.11 VALIDATION

To validate the model, comparisons of the estimated block grades with the grades of the samples are undertaken to assess whether the model honors the data. To remove spatial clustering and define the volume of influence of each composite, nearest neighbor models were created. The height of the model blocks is 15m while the composite length is 2m. If an NN model were created using this information, only every 7th or 8th composite would be nearest to a block centroid (i.e., most data would not participate in the validation). To address this issue, the resource model blocks were subdivided into 2.5m high sub-blocks for the NN model.

The validation steps performed were:

- Visual inspection of model and data grades on section and plan
- Comparison of average model and sample grades per estimation domain
- Comparison of average model and sample grades over large blocks
- Comparison of average model and Sample grades over slices through the model

14.11.1 Visual Inspection

The visual inspection considers both the geological model and the estimated grades. The model check confirms that the model blocks are properly coded and that the domain codes of the data match those of the model. The check of grades compares the spatial pattern of grades seen in the composite samples with that of the estimated block grades. Important features observed in the samples such as the anisotropic correlation pattern parallel and perpendicular to the central structure, the decrease in grade moving away from the data, and the vertical decrease in solubility with depth in the enriched zone are reproduced in the model. Furthermore, the grades seen in the drillholes match well with the estimated block grades.

14.11.2 Average Grades by Domain

As an overall check of the estimated grades, average model and NN grades are compared by domain and search pass for the leach, enriched and hypogene zones. For each comparison, the number of estimated blocks is shown along with the relative difference in the two estimated grades.

Both average total and cyanide soluble copper grades are compared. It is noted that for soluble copper the resource model used a two-stage estimation to account for the missing CuCN data while the NN estimate is a simple estimate using only the available data.

Examining the number of blocks estimated per pass provides information on drillhole density per mineral zone. The enriched zone was typically estimated in search pass 2 or 3 while the bulk of the hypogene zone could not be estimated until search pass 4.

Table 14.18: Comparison of Resource and NN Estimates in The Block Model.

Comparison of Model and Data Averages By Mineral Zone and Search Pass (only blocks with Nearest Neighbor Estimates)

Search Pass	Mineral Zone	Tonnes (000,000)	Total Copper			Cyanide Soluble Copper			Acid Soluble Copper		
			Model Avg (%)	Data Avg (%)	Relative Difference	Model Avg (%)	Data Avg (%)	Relative Difference	Model Avg (%)	Data Avg (%)	Relative Difference
1	Cover	3.2	0.025	0.026	1.8%	0.001	0.001	4.1%	0.008	0.008	0.5%
1	Leach	61.2	0.035	0.036	3.0%	0.005	0.005	4.1%	0.009	0.010	9.0%
1	Enriched	124.6	0.617	0.610	-1.1%	0.406	0.400	-1.6%	0.053	0.054	1.8%
1	Mixed	1.9	0.281	0.257	-9.1%	0.156	0.131	-18.9%	0.071	0.075	5.2%
1	Cpy-Bn	0.4	0.342	0.315	-8.6%	0.110	0.085	-28.9%	0.021	0.019	-7.7%
1	Bornite	0.1	0.324	0.421	23.1%	0.261	0.325	19.6%	0.012	0.016	25.9%
1	Hypogene	43.8	0.296	0.301	1.6%	0.059	0.060	1.7%	0.016	0.016	3.0%
1	Volcanics	0.6	0.003	0.003	-5.4%	0.001	0.001	-5.3%	0.001	0.000	-68.9%
2	Cover	47.2	0.021	0.022	3.3%	0.001	0.002	14.1%	0.006	0.006	-2.9%
2	Leach	344.5	0.034	0.035	3.8%	0.005	0.006	7.4%	0.008	0.009	3.8%
2	Enriched	416.5	0.482	0.479	-0.7%	0.307	0.303	-1.4%	0.049	0.050	3.1%
2	Mixed	1.5	0.398	0.375	-6.1%	0.228	0.208	-9.7%	0.093	0.092	-0.7%
2	Hypogene	168.1	0.273	0.275	1.0%	0.062	0.061	-1.4%	0.015	0.016	5.2%
2	Volcanics	20.9	0.003	0.003	-3.9%	0.001	0.001	-4.0%	0.001	0.000	-11.2%
3	Cover	78.2	0.032	0.039	18.1%	0.002	0.002	18.6%	0.008	0.009	8.9%
3	Leach	217.6	0.035	0.034	-0.7%	0.006	0.006	6.7%	0.009	0.006	-38.4%
3	Enriched	257.6	0.405	0.398	-1.8%	0.238	0.232	-2.6%	0.043	0.044	1.9%
3	Mixed	0.1	0.267	0.146	-82.6%	0.138	0.053	-161.6%	0.060	0.026	-132.7%
3	Hypogene	192.2	0.232	0.245	5.3%	0.045	0.046	2.5%	0.013	0.015	13.0%
3	Volcanics	18.5	0.003	0.003	-5.2%	0.001	0.001	-4.7%	0.001	0.000	-70.6%
4	Cover	58.9	0.028	0.029	3.1%	0.001	0.002	5.5%	0.007	0.006	-9.7%
4	Leach	222.4	0.030	0.029	-1.6%	0.005	0.005	1.0%	0.008	0.006	-24.0%
4	Enriched	206.1	0.369	0.347	-6.3%	0.216	0.197	-9.6%	0.041	0.041	1.4%
4	Mixed	1.0	0.363	0.325	-11.7%	0.192	0.150	-28.3%	0.101	0.112	9.3%
4	Cpy-Bn	342.0	0.359	0.347	-3.4%	0.099	0.095	-4.8%	0.020	0.020	-0.9%
4	Bornite	86.1	0.357	0.360	1.0%	0.214	0.224	4.4%	0.024	0.024	2.9%
4	Hypogene	2218.8	0.254	0.253	-0.1%	0.024	0.025	7.4%	0.008	0.008	6.5%
4	Volcanics	52.6	0.004	0.004	-7.3%	0.001	0.001	-14.1%	0.001	0.000	-32.4%

For cyanide soluble copper, the most appropriate comparisons are over the enriched zone. Outside of this zone, very low soluble copper grades are observed, and small differences can generate large relative difference values.

Except in those domains where data is sparse, the model and data average grades match acceptably well for total copper. For soluble copper, the two estimates of average grades match well for the enriched zone domains in passes 1 through 3. This similarity in average grades provides an indication that the two step Kriging estimate, used for soluble copper, has not generated an estimation bias. In search pass 4, the difference in averages grades increase for both copper species indicating that estimate quality is reduced.

14.11.3 Average Grades Over Large Blocks

Model and sample data (NN model) grades are averaged into larger 60x100x60m blocks and then compared in a scatterplot. A larger block dimension is considered in the North-South direction due to the stronger grade correlation parallel to the NNW structure. Averaging block grades over a larger volume compensates for the difference in the variances of block and data grades and allows a local, numerical, comparison of the two estimates over the entire deposit. Groups of blocks with important differences in the two estimates can be identified in space and reviewed.

Figure 14.16 presents validation scatterplots for total and cyanide soluble copper for blocks estimated in search pass 1 and search passes 1 and 2.

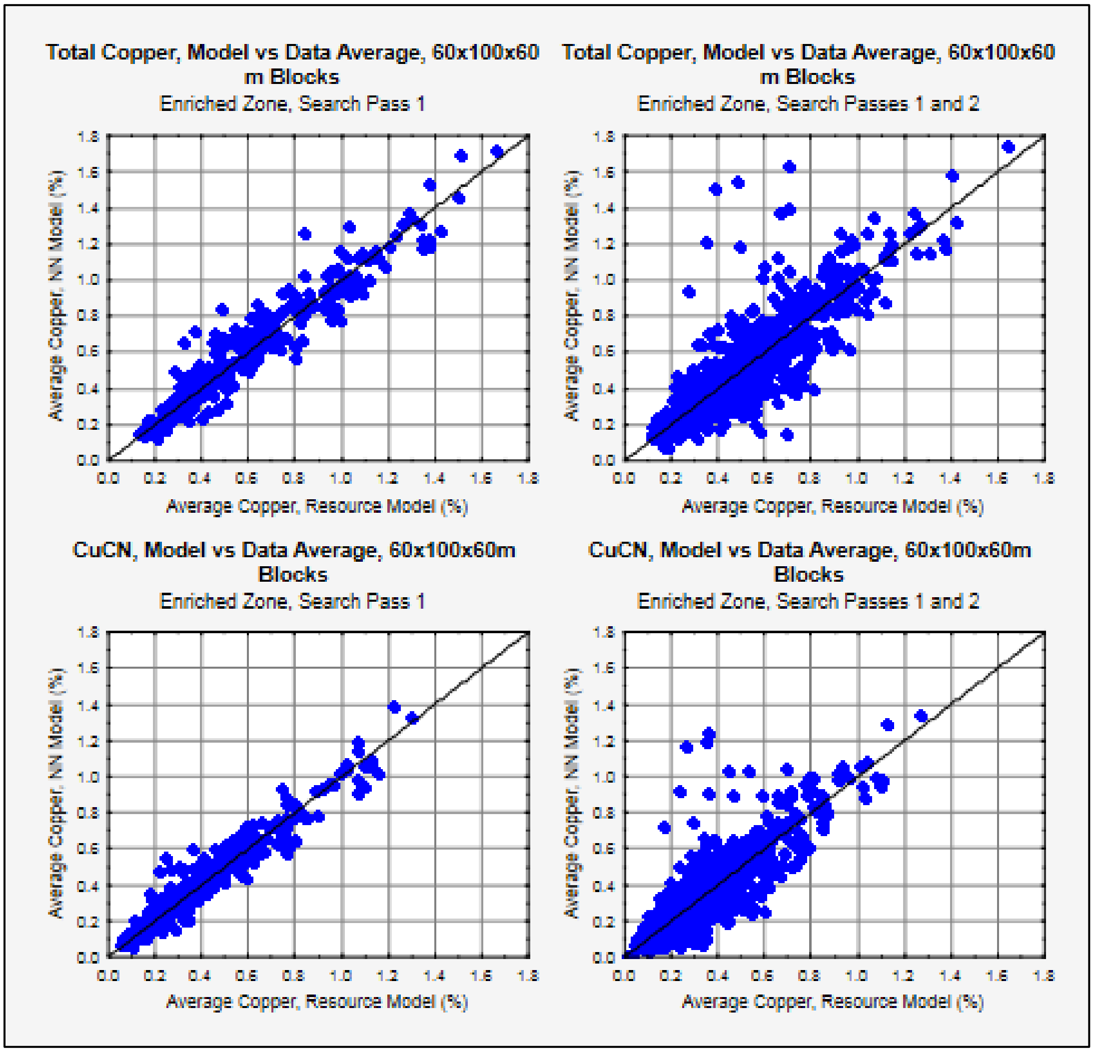


Figure 14.16: Scatter Plots of Large Block Comparison of Samples VS Model Grades.

The upper scatterplots are for total copper while the lower plots are for cyanide soluble copper. The plots on the left side consider model blocks estimated in search pass 1. Blocks estimated in passes 1 and 2 are shown on the right. In all four scatterplots the points are well centered on the (black) line Y=X. For blocks estimated in search pass 1 the scatter is less; however, the comparison for pass 1 and 2 estimates is acceptable. For cyanide soluble copper, the NN estimate is more variable and indicates the possibility of larger grades. If true, the model is conservative.

14.11.4 Validations Over Slices Through the Model

Slices with widths of 80m (NS) and 60m (EW) are cut through the resource model and average model and sample averages are compared. Figure 14.17 presents average total and soluble copper grades for the enriched and hypogene zones for blocks estimated in passes 1 or 2.

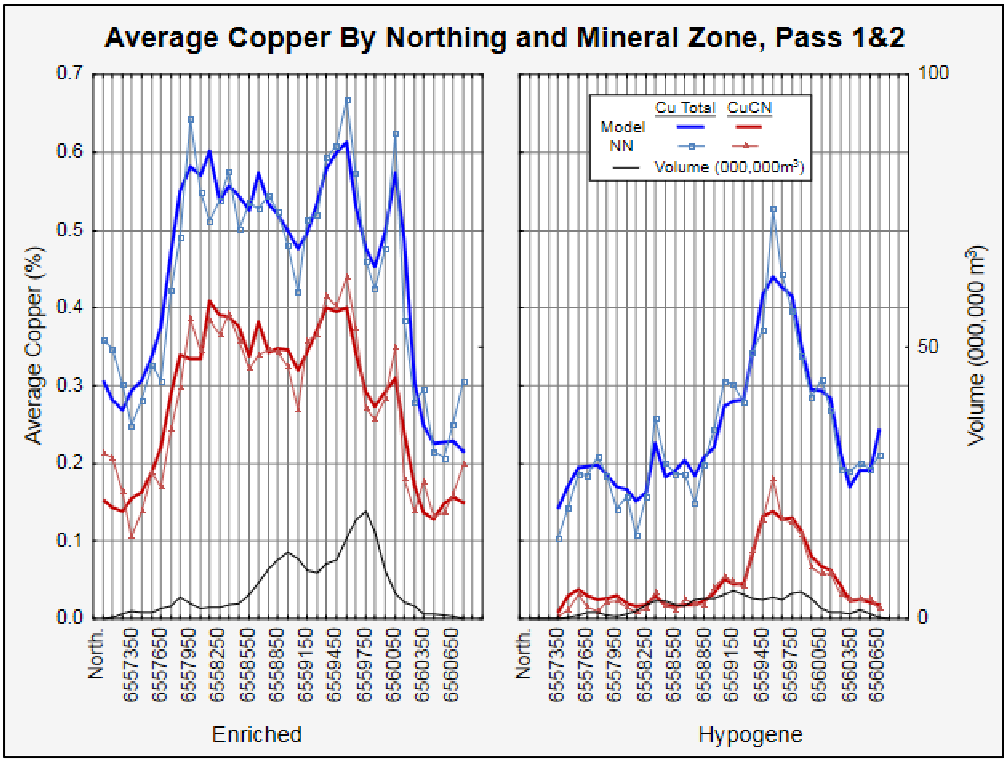


Figure 14.17: East-West Swath Plots of Total and Cyanide Soluble Copper Grades in the NN Model

The total volume per slice is shown on the right axis. As seen, the two sets of average grades are very similar across the deposit with the NN average showing larger variance.

The same plot considering grades by easting is shown in Figure 14.18.

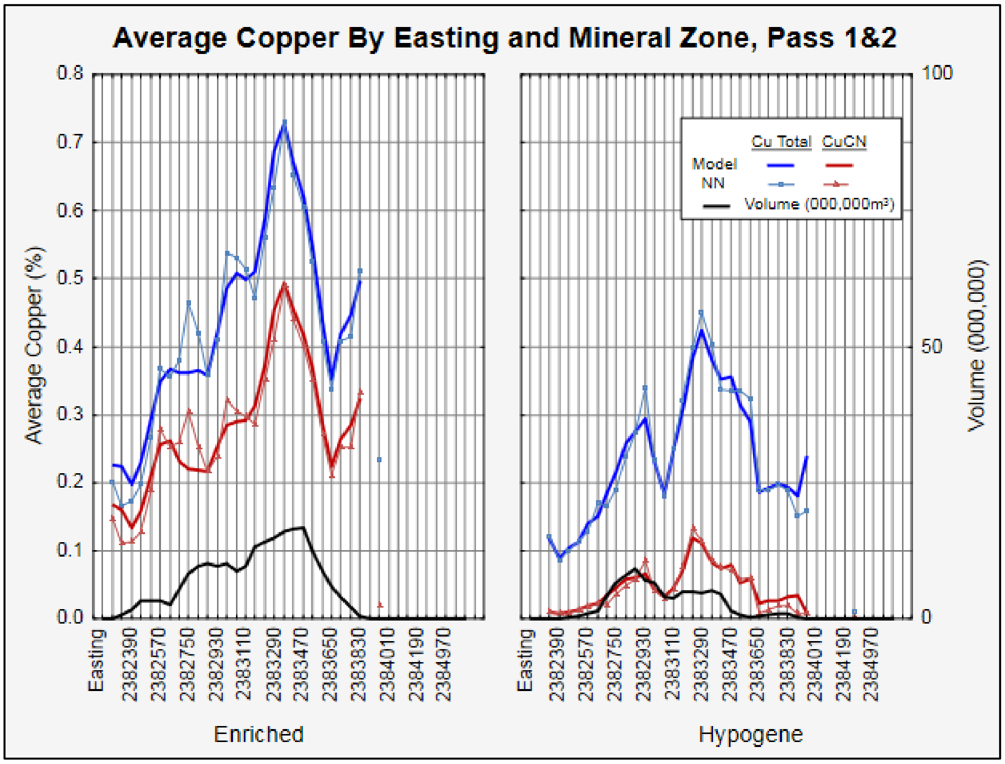


Figure 14.18: North-South Swath Plots of Total and Cyanide Soluble Copper Grades in the NN Model

When considered as a function of easting, average grades show a peak associated with the central NNW structure and decrease rapidly to both the west and east. The peak grade is seen for total copper in both mineral zones, but the peak is sharper in the enriched zone. For both total and soluble copper, in each of the two mineral zones, the resource model and NN averages match well.

In the vertical direction, a vertical solubility trend was observed in the data. To best reproduce this trend the estimation approach utilized a relative depth coordinate transform to control the estimation. This estimate is compared against an untransformed NN estimate.

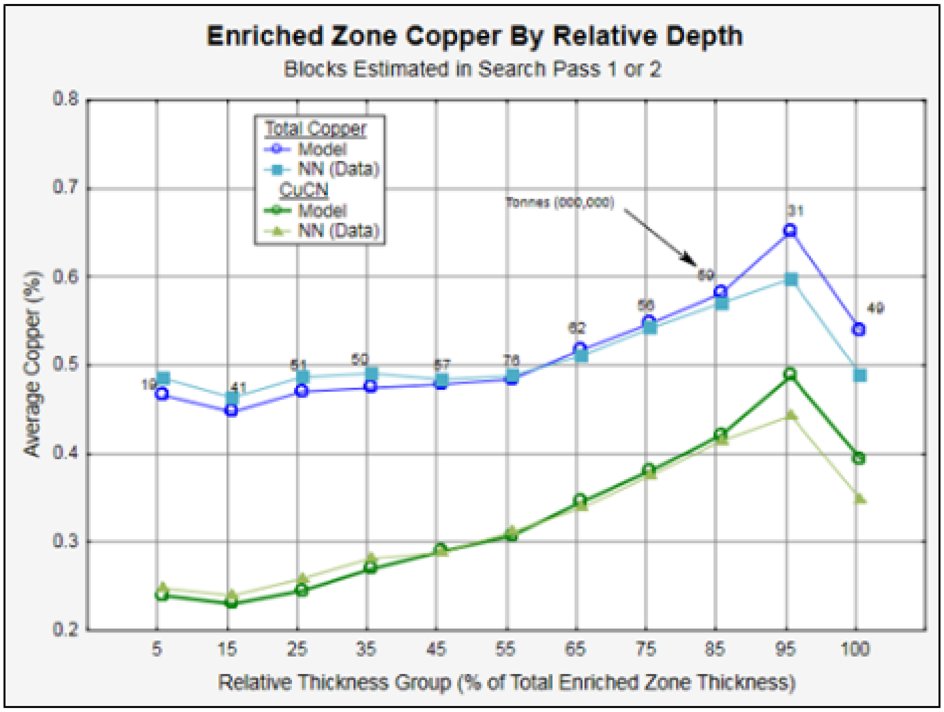


Figure 14.19: Total and Cyanide Soluble Copper Related to Depth from the Top of the Enriched Zone

This plot shows that the resource model finds lower grades approaching the base and slightly larger grades near the top of the enriched zone. This type of average grade difference is an indication that the NN model (based on untransformed coordinates) is smoothing the vertical trend relative to the resource model. Given the small differences observed and the differences in the model and validation approach, the model is validated by the NN results.

14.12 RESOURCE CLASSIFICATION

The mineral resources at the Los Azules deposit have been classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2019) and the SEC's SK 1300 regulations (2018). Studies supporting the 2017 PEA model (Hatch, 2017) found that drilling on a 150m grid was sufficient to define Indicated resource. This result was validated by CRM[4].

The resource classification approach taken recognizes that the Los Azules deposit is not drilled on a regular grid. To obtain a measure of the local drilling density, a block-by-block computation was performed for blocks located outside of the leached or overburden zones. For each block, the average distance to the nearest 3 drillholes was determined. To select the most relevant dimension, plan and section images were created and the following were reviewed:

- The location of the drillholes
- The continuity of grades in space
- Images of the estimated distance to the nearest 3 drillholes
- The extent of the validation volume (blocks estimated in search passes 1 and 2).

4 Consultores de Recursos Minerales (CRM), Memo to Antonio Samaniego (SRK), Review of Los Azules Drill Spacing and Classification, 4 June 2021

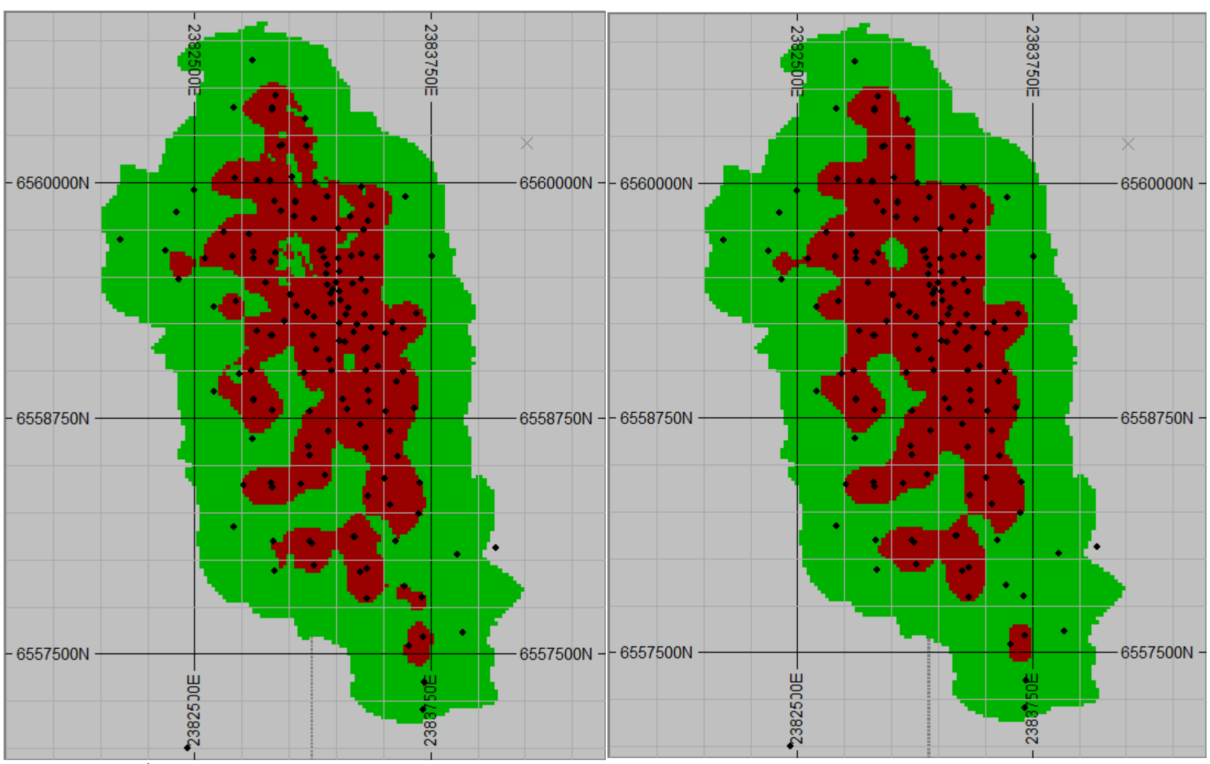
Based on this assessment, it was determined that non-breccia blocks where the average spacing to the nearest 3 drillholes was less than 120 m were defined as Indicated resource. The reduction from the previously stated 150 m grid to 120 m is to allow for the differences between a regular grid spacing and the distance to 3 irregularly spaced drillholes. For breccia blocks, 75 m was selected due to the reduced continuity of this lithology. Using a similar evaluation, blocks with an average distance to the nearest 3 drillholes between 120 m and 400 m were considered inferred resource.

To satisfy the requirement for reported mineral resources to have a reasonable prospect of eventual economic extraction (RPEEE) an open pit was evaluated using the resource model. The economic value of each block was calculated based on the metal content, the price of each metal, processing costs, and other downstream costs associated with having a final saleable product. This value is stored for each block of the model as Net Smelter Return (NSR). The parameters used to calculate NSR in the models are detailed in Section 14.13.1.

The NSR values were used to generate an open pit with variable cutoff values to cover the material types and recover methodology. The parameters used to evaluate the leach versus potential floatation mill recovery are detailed in Section 14.13 .2.

The initial Indicated resource blocks were subsequently modified as follows:

- Indicated blocks below elevation 3340 m were converted to Inferred resource. This elevation range is investigated by three deep drillholes which identified a mineralized zone that is offset from the main structure. The geologic controls on mineralization in this volume are not well understood.
- The initial classification was smoothed to remove some of the isolated blocks. An example of the smoothed and unsmoothed classification is presented in Figure 14.20.



Level 3500 +/- 8m, The Initial (L) and Smoothed Classification (R); Data in northwest are leach and are not considered.

Figure 14.20: Comparison of Indicated and Inferred limits before and after Smoothing.

About 1/3 of the resource is located underneath some rock glacial features. Figure 14.21 shows the plan outline of the resource pit and the location of the cryogenic geoforms identified with a 50 m standoff line around each. Given the environmental consideration of these features the indicated material below and to the west of the geoforms within the resource pit shell has been downgraded to inferred pending further investigations.

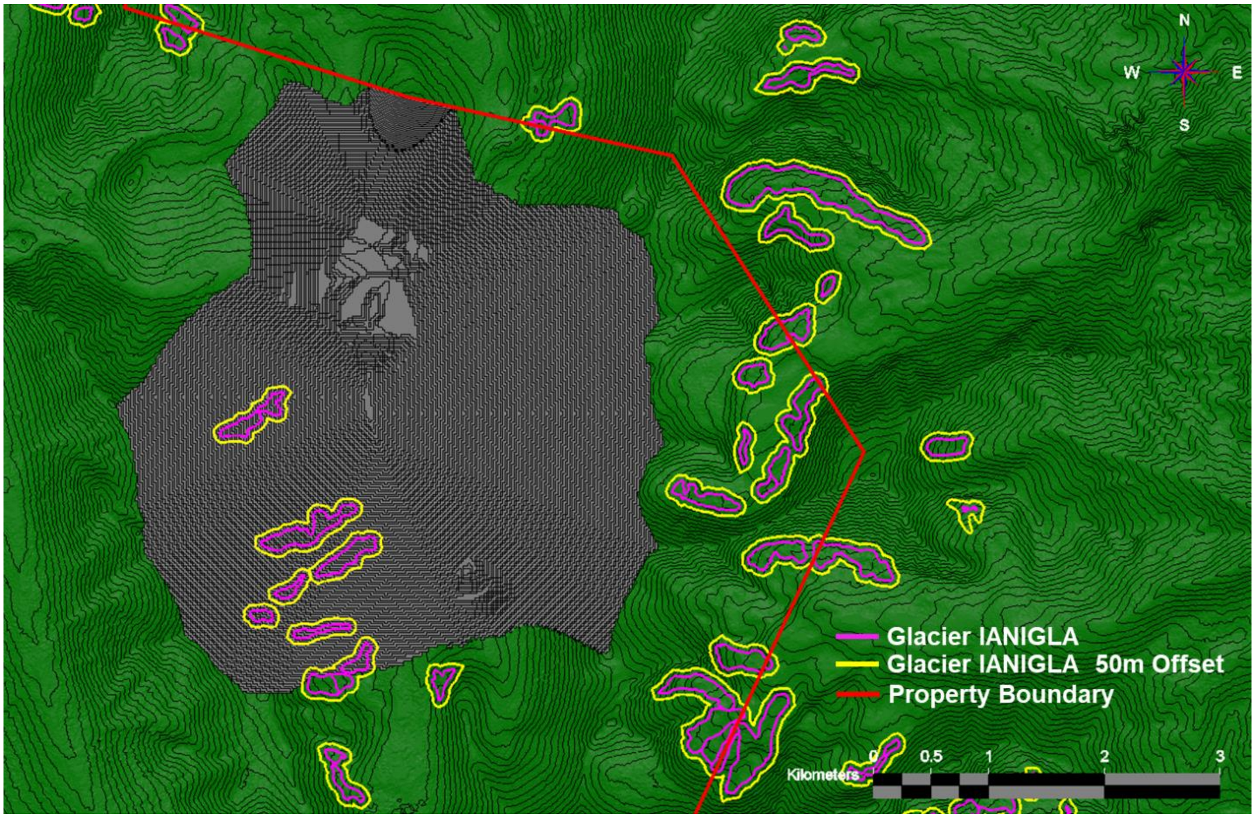


Figure 14.21: Plan View of the Resource Pit with Geoforms Outlines

14.13 MINERAL RESOURCES

Table 14.19 reports the Indicated and Inferred resources from the 2017 NI 43-101 PEA Technical Report, which were reported at a 0.2% copper cut-off. These are now obsolete and are replaced with the 2023 PEA Mineral Resource Estimate in this report. They are presented here only for comparative purposes.

Table 14.19: 2017 Estimate of Los Azules Mineral Resources

	Average Grade				Contained Metal			
Mtonnes	**Cu %**	**Au g/t**	**Mo %**	**Ag g/t**	**Cu Blbs**	**Au Moz**	**Mo Mlbs**	**Ag Moz**
Indicated								
962	0.48	0.06	0.003	1.8	10.2	1.7	57.3	55.7
Inferred								
2,666	0.33	0.04	0.003	1.6	19.3	3.8	194.0	135.4

Note: The mineral resources do not have demonstrated economic viability

14.13.1 NSR Parameters

The NSR calculation varies according to the recovered metal and the mineral zone being processed. Table 14.20: NSR Parameters for Leach Recovery shows the prices and costs associated with using an acid leach method to recover copper from the supergene enriched material in the model.

Table 14.20: NSR Parameters for Leach Recovery		
Constant Item	**Value**	**Units**
Metal Prices		
Copper Price	$ 4.00	USD$/lbs.
Gold Price	$ 1,700.00	USD$/Oz.
Silver Price	$ 20.00	USD$/Oz.
Selling Costs (insurance & marketing)		
Copper Selling Cost	$ 0.02	USD$/lbs.
Gold Selling Cost	$ -	USD$/Oz.
Silver Selling Cost	$ -	USD$/Oz.
Processing Cost		
Processing Tonnes	4.17	$/tonne
Transportation Costs		
Land Freight (truck)	100.00	USD$/mt
Ocean Freight (ship)	50.00	USD$/mt
Total Freight	150.00	USD$/mt

Note:

- Only copper is recovered in the leach method and has a constant recovery of 95% applied.
- With applied solvent extraction / electro-winning (SX/EW) cathode production there are no smelter nor related costs incurred.

With the potential for froth flotation as a recovery method the NSR values were calculated for both high-grade enriched and primary material in a mill. This has the added benefit of also recovering the gold and silver present in the resource. Table 14.21 details the parameters used to calculate NSR for this case.

Table 14.21: NSR Parameters for Mill/Flotation Process		
Constant Item	**Value**	**Units**
Metal Prices		
Copper Price	$ 4.00	USD$/lbs.
Gold Price	$ 1,700.00	USD$/Oz.
Silver Price	$ 20.00	USD$/Oz.
Selling Costs (insurance & marketing)		
Copper Selling Cost	$ 0.02	USD$/lbs.
Gold Selling Cost	$ -	USD$/Oz.
Silver Selling Cost	$ -	USD$/Oz.
Recoveries are specified in the rock type variable tabs		
Concentrate Terms		
Min. Copper feed grade	0	% Cu
Cu Con. Grade for Supergene	28.53	% Cu
Cu Con. Grade for Primary	31.96	% Cu
Payable Metal		
Copper Payability	96.5	%
Gold Payability	90.0	%
Silver Payability	90.0	%
Minimum Con. Grade for Credit Payability		
Gold	1	g/t
Silver	30	g/t
Refining Charges		
Copper Refining Charges	$ 0.080	USD$/lbs.
Gold Refining Charges	$ 8.000	USD$/Oz.
Silver Refining Charges	$ 0.500	USD$/Oz.
Treatment Charges		
Con. Treatment Charge	$ 80.00	USD$/dmt
Transportation Costs		
Land freight – trucking rate		$USD/wmt/km
Land freight – trucking distance		km
Land freight – trucking	$ 100.00	$USD/wmt
Ocean freight	$ 50.00	$USD/wmt
Con. Moisture spec	8	%
Total freight	$ 162.00	$USD/dmt
Copper Treatment and Transport	$ 0.1098	$/lbs

Table 14.21: NSR Parameters for Mill/Flotation Process		
Constant Item	**Value**	**Units**
gold/silver Treatment and Transport	$ 0.0075	$/oz
Total Selling Cost		
Copper	$ 0.21	$/lbs
Gold	$ 8.01	$/oz
Silver	$ 0.51	$/oz
Recovery Factors	**Enriched**	**Primary**
Copper	89.3	93.2
Gold	65.6	62.9
Silver	54.0	68.8
Copper Concentrate Spec.	28.53	31.96
Payable Metal	**Value**	
Copper	96.50	
Gold	90.00	
Silver	90.00	
Recovery with Payable Metal	**Enriched**	**Primary**
Copper	0.8617	0.8994
Gold	0.5904	0.5661
Silver	0.4860	0.6192

14.13.2 Pit Design Parameters.

The calculated NSR value in each block was used to evaluate an open pit within the resource model. Both Inferred and Indicated blocks were used to create the pit. The original pit was unconstrained and extended to use all the available data in the model. This pit had to be adjusted to avoid an area of cryogenic geoforms on the south-west corner of the resource area. Table 14.22 details the parameters used to create these shapes.

Table 14.22: Open Pit Design Parameters.		
Pit Depth	Inter Ramp Slope Angle	Slope Code
Surface to 600m	42°	
600m to 800m	38°	
800m to 1,000m	34°	
1,000 to 1,200m	32°	
Overburden	30°	

Figure 14.22 below shows the areas in the leach and mill pits where the pit slope angles shown in Table 14.22 above are applied.

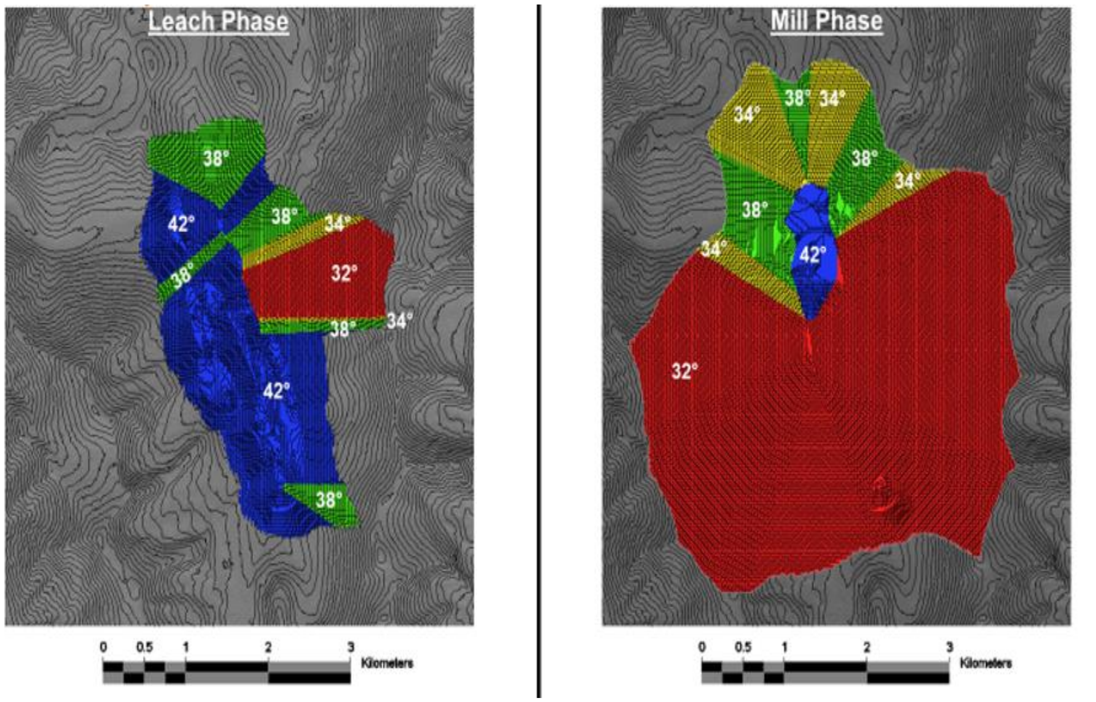


Figure 14.22: Plan View of the Resource Pits with Slope Angles

14.13.3 Mineral Resource Table

The resource pit and NSR cutoffs detailed above were used to define an economic MRE as reported in Table 14.23 and Table 14.24. The resources exclude leach and overburden material and include only Indicated and Inferred blocks.

Table 14.23: Indicated Resources for the Los Azules Project									
	MTonnes	Average Grade				Contained Metal			
		In-Situ Copper Total (%)	In-Situ Copper Soluble (%)	In-Situ Gold (g/tonne)	In-Situ Silver (g/tonne)	In-Situ Copper Total Content (Blbs)	In-Situ Copper Soluble (Blbs)	In-Situ Gold (Moz)	In-Situ Silver (Moz)
Leach	944.2	0.46	0.30	-	-	9.54	6.25	-	-
Mill - Supergene	73.0	0.13	-	0.09	1.10	0.21	-	0.20	2.58
Mill - Primary	218.1	0.25	-	0.036	1.06	1.19	-	0.25	7.43
Total Leach	**944.2**	**0.46**	**0.30**	**-**	**-**	**9.54**	**6.25**	**-**	**-**
Total Mill	**291.1**	**0.22**	**-**	**0.049**	**1.07**	**1.40**	**-**	**0.46**	**10.01**

Table 14.24: Inferred Resources for the Los Azules Project									
	MTonnes	Average Grade				Contained Metal			
		In-Situ Copper Total (%)	In-Situ Copper Soluble (%)	In-Situ Gold (g/tonne)	In-Situ Silver (g/tonne)	In-Situ Copper Total Content (Blbs)	In-Situ Copper Soluble (Blbs)	In-Situ Gold (Moz)	In-Situ Silver (Moz)
Leach	695.7	0.32	0.19	-	-	4.91	2.96	-	-
Mill - Supergene	525.6	0.30	-	0.05	1.44	3.45	-	0.87	24.40
Mill - Primary	3,288.0	0.25	-	0.03	1.18	18.35	-	3.37	124.67
Total Leach	**695.7**	**0.32**	**0.19**	**-**	**-**	**4.91**	**2.96**	**-**	**-**
Total Mill	**3,813.6**	**0.26**	**-**	**0.035**	**1.22**	**21.79**	**-**	**4.24**	**149.07**

Notes:

1. There is a reasonable prospect of eventual economic extraction of the leach resource using sulfuric acid leaching and SX/EW recover at NSR cutoff of $2.74/t. The supergene and primary material can be treated in a mill with NSR cutoffs of $5.46 and $5.43/t respectively. NSR values are based on a copper price of $4.00/lbs., gold at $1,700/oz., and silver at $20/oz., where applicable.
2. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, socio-political, marketing, or other relevant factors.
3. The quantity and grade of reported Inferred mineral resources in this estimation are uncertain in nature and there is insufficient exploration to define these Inferred mineral resources as an Indicated or Measured mineral resource; it is expected that further exploration will result in upgrading some of this material to an Indicated or Measured classification.

Table 14.25 details the indicated material in the environmentally sensitive area under the cryogenic geoforms that was downgraded to Inferred in the MRE.

Table 14.25: Inferred Material under the Cryogenic Geoforms									
	MTonnes	Average Grade				Contained Metal			
		In-Situ Copper Total (%)	In-Situ Copper Soluble (%)	In-Situ Gold (g/tonne)	In-Situ Silver (g/tonne)	In-Situ Copper Total Content (Blbs)	In-Situ Copper Soluble (Blbs)	In-Situ Gold (Moz)	In-Situ Silver (Moz)
Leach	-	-	-	-	-	-	-	-	-
Mill - Supergene	2.5	0.22	-	0.02	0.94	0.01	-	0.002	0.07
Mill - Primary	79.9	0.28	-	0.026	1.46	0.49	-	0.067	3.74
Total Leach	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Total Mill	**82.3**	**0.28**	**-**	**0.026**	**1.44**	**0.51**	**-**	**0.069**	**3.82**

Soluble copper was not reported in the 2017 PEA resource statement and is therefore not compared.

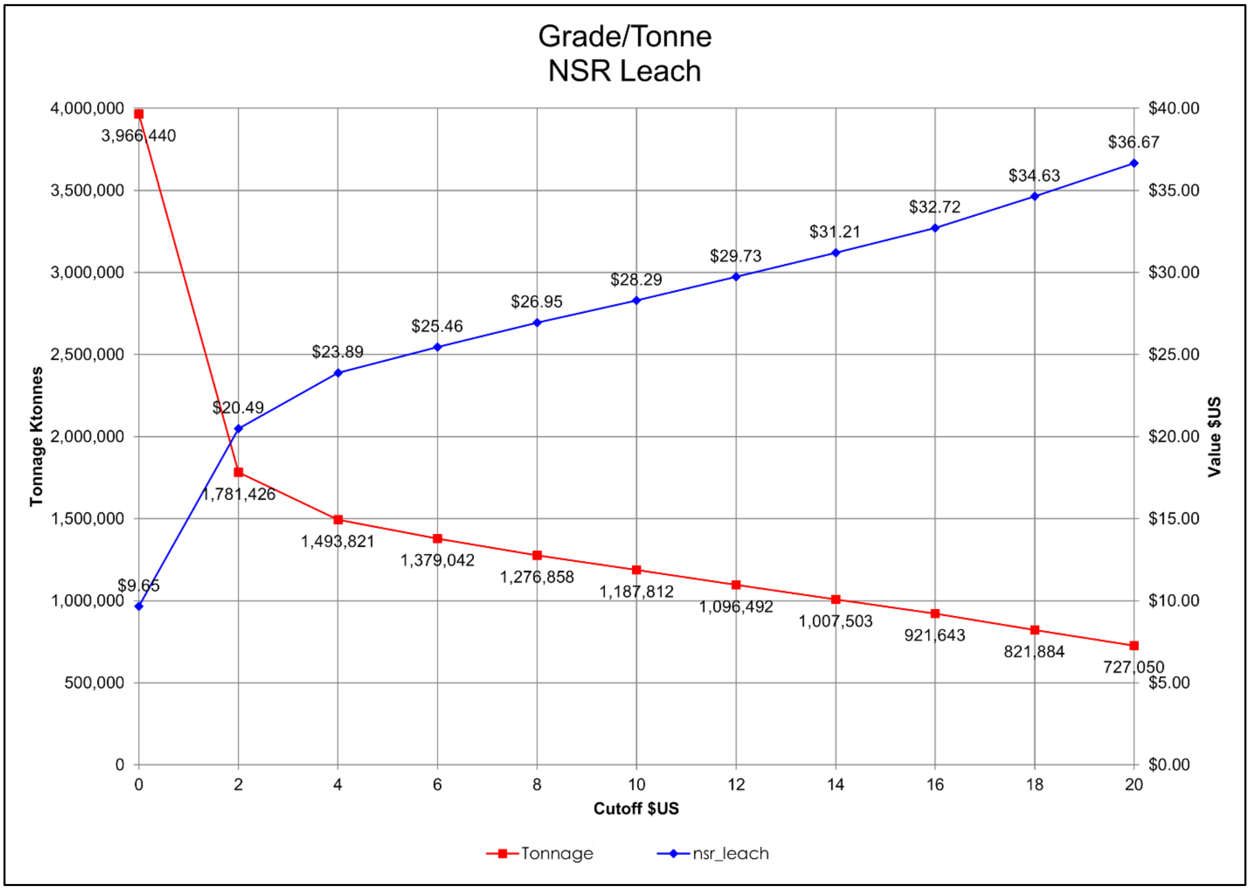
The tonnage above cutoff shows an increase of the tonnes of Inferred and Indicated material in the current model. The current model predicts 19% lower overall copper grades.

The major factors contributing to these differences are:

- The current model treats the high-grade breccia and early mineral porphyry units as separate domains with relatively small volumes. The estimation considers these lithology domains as statistically different and no sharing of samples between the high-grade lithologies and the surrounding (lower grade) domains is allowed. For the 2017 model, data from all lithologies are combined. As a result, the 2017 model laterally smears grades outward from the high-grade lithologies while, at the same time, smooths (reduces) the grades within the high-grade zones. This change in modeling can be expected to increase copper grade above cutoff while reducing tonnes above cutoff.
- The 2017 PEA model did not consider the important control on grades provided by the central NNW structure or the lateral grades trend (lower grades) moving away from the structure. As a result, a large, isotropic, circular search was used during estimation. This type of search can be expected to smooth the observed lateral grade trend. The current model uses an anisotropic search oriented at N20W and the model validations show that the lateral grade trend is well reproduced. As a result, it is expected that the 2017 model would provide a larger estimate of tonnes above cutoff at a lower grade.
- The 2017 PEA model contains large lateral extensions of the enriched zone which enhanced the enriched zone volume. Drilling in 2022 targeted some of these extensions and reduced the lateral volume of the enriched zone.
- The tonnage of Indicated resource stated in 2017 was ostensibly based on a 150m drill spacing. Review of the extent of the volume of the 2017 Indicated resource shows that it often extends into volumes where the drill spacing is larger than 150m. The drill spacing, and methodology used in the 2017 PEA led to a higher indicated resource tonnage estimate than using the 120m drill spacing applied in the 2023 estimate.
- Drilling performed in 2022 provided additional details on the shape and volume of the enriched zone adding confidence to the local modeling of the estimation domains and estimation of grades.

14.13.4 Cutoff-Grade Sensitivity

The NSR cutoff used to develop the leach pit was based on total soluble copper plus the added value of 15% of the residual copper in the block. The actual cutoff value was set at $2.74/t. Figure 14.23 is a grade-tonnage curve showing the NSR values for leachable material in the block model. The NSR value used to determine the outline of the mill pit was based on total copper and credits for gold and silver. The mill cutoff was set at $5.46 for supergene material and $5.43 for primary. Figure 14.24 is a grade-tonnage chart of the NSR value for material destined for the mill.



Note: These figures will be updated to report only resources confined by the resource pits to better compare the reported resource results.

Figure 14.23: Grade / Tonnage Curves for Leach NSR

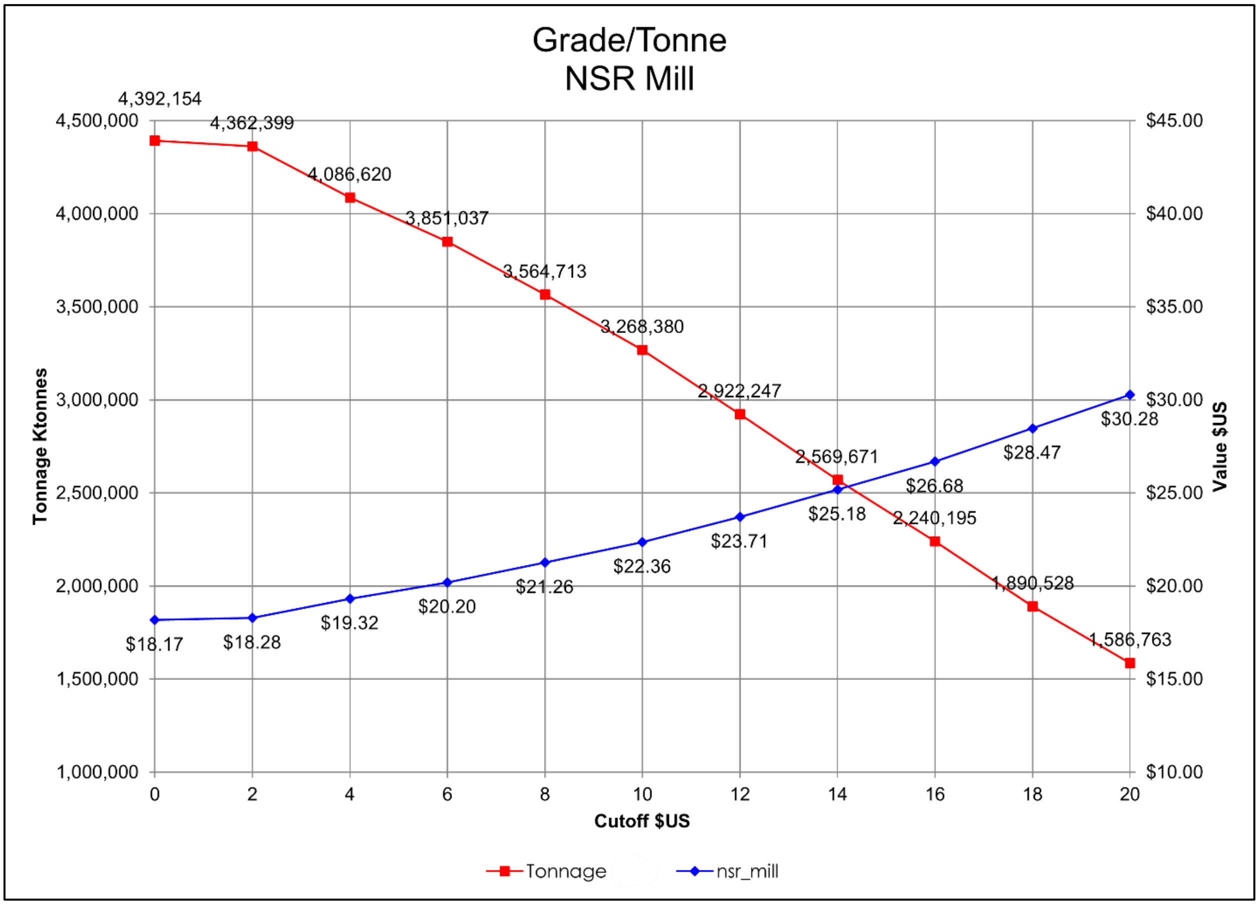


Figure 14.24: Grade / Tonnage Curves for Mill NSR

14.14 ADEQUACY STATEMENT ON SECTION 14

The QP believes that the EDA, mineralogical wireframing, grade capping, and grade estimation methodologies used in the creation of this MRE followed sound industrial standards and conforms to the requirements of an NI 43-101 Technical Report. The outlined resource has shown itself to be quite robust with recent added drilling having little effect on the tonnes and grade but improving confidence. Major factors affecting the MRE include the decision to not declare any Measured resource until the lithologic boundaries are better understood and the downgrading of resource under the identified cryogenic geoforms to Inferred for potential environmental concerns.

15.0 MINERAL RESERVE ESTIMATES

No Mineral Reserves are estimated in this Preliminary Economic Assessment.

16.0 MINING METHODS

This section discusses aspects of the mine that includes Inferred Resources. Inferred Resources are considered geologically too speculative to have the economic considerations applied to them that would enable them to be categorized as mineral reserves as part of further detailed evaluation at the prefeasibility or feasibility level. In addition, the reader is reminded that the mine plan was developed using both Inferred and Indicated material; therefore, there is no guarantee that the presented PEA based on these combined resource tonnages will be realized.

16.1 INTRODUCTION

This 2023 PEA for Los Azules is based on a revised and updated geological block model with an effective date of December 2022. The characteristics of the block model and the mineral resource estimation process that was followed to build it are described in detail in Section 14.

In many respects, the Los Azules deposit is a classic Andean-style porphyry copper deposit. The surface overburden cover lies on top of a barren leached zone, which in turn overlies a zone of secondary supergene enrichment of variable copper grades and thickness, and below it the primary or hypogene mineralization extends to at least 1,000 m below the present surface. The Los Azules hydrothermal alteration system is at least 5 km long and 4 km wide and is elongated in an NNW direction along a major structural corridor.

The mineralized system disappears below volcanic cover to the north; therefore, the ultimate extent is unknown. Los Azules has been geologically mapped over many years. The entire area comprising the Los Azules deposit is covered by thick talus or valley fill, so none of the mineralized materials are exposed in outcrop, although some near-surface exposures have been exposed in trenching at the crest of the La Ballena ridge that defines the long axis of the deposit. Consequently, the interpretation of the structures and intrusive bodies is based almost entirely upon drill hole data.

An altered zone surrounds the Los Azules deposit, which is approximately 4 km long by 2.5 km wide. The limits of the mineralization along strike and at depth have not been entirely constrained by drilling. Many of the holes in the core resource area have been terminated in mineralization.

Hypogene minerals include chalcopyrite and to a lesser extent, bornite. However, chalcopyrite is the most important copper mineral in the upper levels of the deposit, while hypogene bornite appears at deeper levels together with chalcopyrite. Copper sulfides rarely exceed 2% to 3% of rock volume. Intervals of 0.1% to 0.35% copper are common in the hypogene mineralization.

Circulation of meteoric ground water leached the primary sulfides (pyrite and chalcopyrite) from the host rocks over the past several million years, and the leached copper was redeposited below the water table in a sub-horizontal zone, or blanket, of supergene enrichment as secondary chalcocite and covellite. The intensity of secondary enrichment diminishes with depth, except along major structures where it may extend to great depth.

Starting at the boundary between the barren leached zone and the supergene mineralization, secondary enrichment mineralization gradually transitions to predominantly hypogene mineralization at depth. Figure 16.1 and Figure 16.2 show a long section and cross section, respectively, through a representative portion of the orebody along with copper grade zones and pit phase limits.

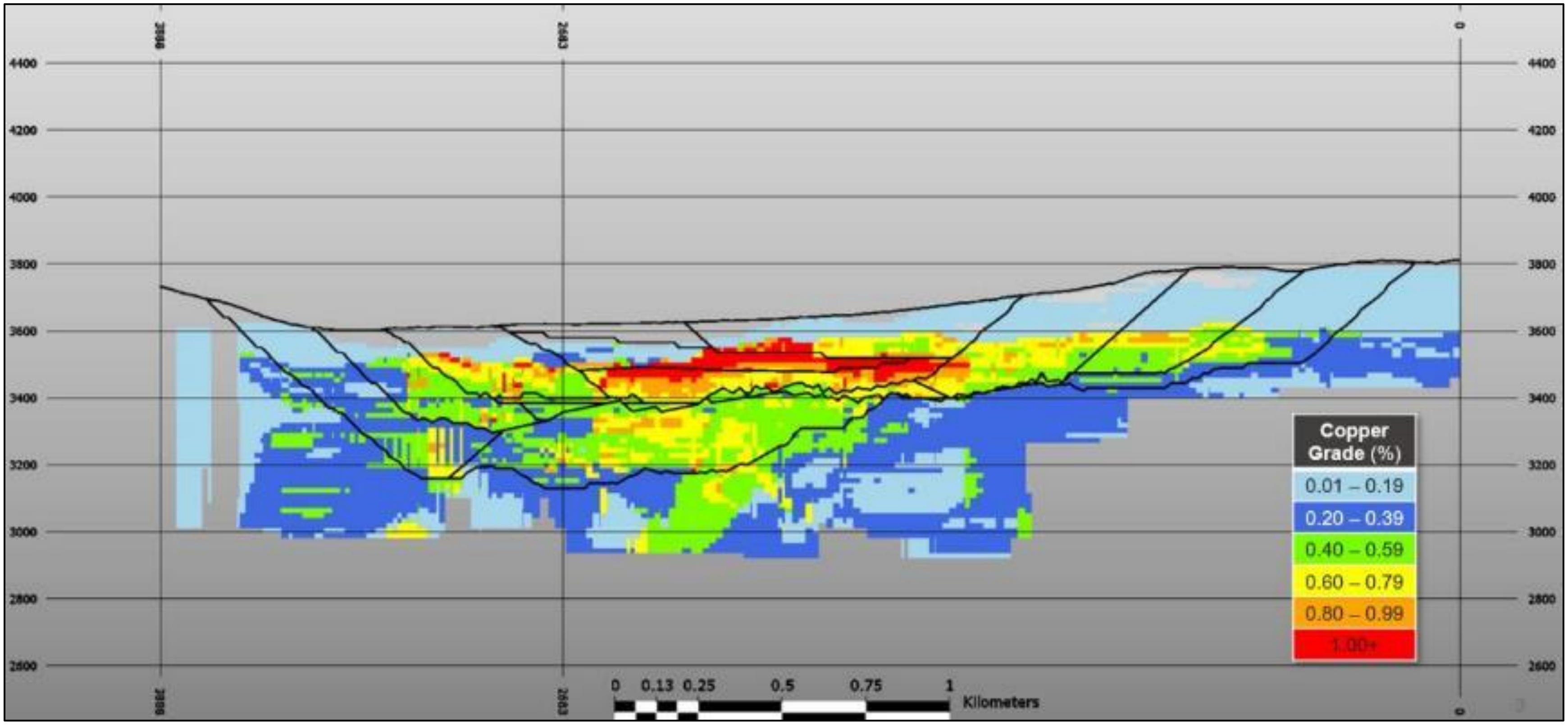


Figure 16.1: Long Section through the Los Azules Mineralization Looking East – with Mining Phase Outlines

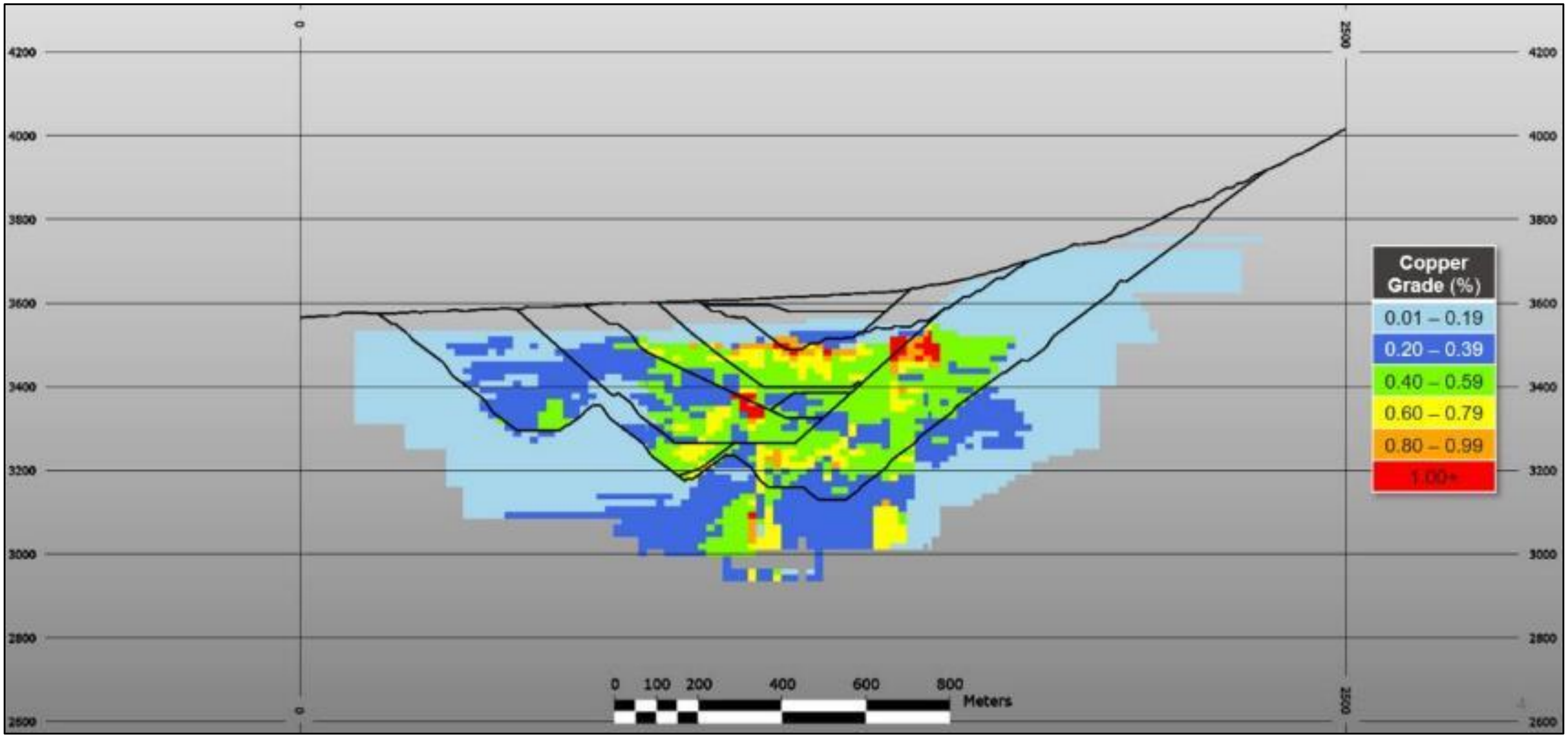


Figure 16.2: Cross Section through the Los Azules Mineralization Looking North West– with Mining Phase Outlines

Los Azules can be described as a typical copper porphyry deposit, with large tonnages of disseminated copper minerals with the usual succession of leached, supergene, and hypogene mineralization.

Mineralized material is present from relatively shallow elevations and therefore large-scale, conventional, open pit mining methods are considered the most appropriate method to mine the Los Azules mineralization. This said, considerable effort was taken to evaluate mining methods, and specifically equipment, which would lead to an operation with a reduced carbon footprint. Work related to decarbonization of the mine has included trade-off studies between conveyors and trucks, in-pit crushing and conveyor (IPCC) systems, battery electric trucking and trolley-assist truck haulage systems.

Although most, if not all, of these technologies have merit and could be incorporated into future mine designs, for the purpose of this PEA, electrification of the mine to a reasonable practical extent was the goal. The mine plan therefore incorporates a combination of electric-hydraulic front shovels and large mine haul trucks that will be equipped with pantographs from the start of operations to allow for the early use of trolley-assist infrastructure. Several sections of the pit and external haul roads have been designed to be equipped with trolley-assist electrical infrastructure.

16.2 ECONOMIC PIT LIMIT EVALUATIONS

The ultimate pit limit and intermediate pit shells were developed with the use of Geovia Whittle™ pit optimization software. Within this software, the pseudo-flow pit optimization algorithm was used to develop the incremental pit shells and associated shell values. The Whittle software also provided guidance for the selection of the optimal mining phases.

To obtain the ultimate pit limit and intermediate pit shells, the block model was first updated to include the net smelter return (NSR), surface restrictions / constraints, and pit slope geotechnical guidance. The block model was then imported into Whittle™ pit optimization software, with mining parameters and production rates added within the software. All these subjects are discussed in the following subsections.

A series of economic pit optimizations were evaluated to define pushbacks and the ultimate pit. The pit-optimization presented in this section was created for a mine that produces copper cathode using a heap-leach process (the base case pit optimization). The Los Azules deposit also contains mineralized material that may be economic for producing a milled copper concentrate product, but this material was not considered for base case pit optimization.

16.2.1 Block Model

The geological block model named "Model in Ascii modpea_03-03-23" was received from CRM-SA in a CSV format on March 3, 2023. The model contains data items that define mineral zones, lithologies, metal grades and resource classifications, amongst other items. It is composed of blocks measuring 20 m long, 20 m wide, and 15 m high. The block model is situated in the POSGAR 94 coordinate system and is oriented along the north-south axis.

Stantec made a series of additions to the model, using HxGN MinePlan™ 3D, to support the pit optimization process. The following are the tasks completed by Stantec.

- Coding the percentage of each block below topography (based on a survey surface dated 04 April 2022.)
- Calculating the soluble copper grade, which was the sum of the cyanide and acid soluble copper grades.
- Creating metal grade items with undefined values to zero.

- Creating NSR values for the PEA mine plan as well as the resource pits. This is discussed further in subsequent sections.
- Adding ore code items that defined which material could have been considered as mineralized material for the pit optimization.
- Adding pit slope zone definition items (discussed further in subsequent sections.)
- Adding flag codes to identify material that could have been stockpiled for future copper concentrate production.

The pit optimization presented in this PEA was based on Indicated and Inferred resources. No material in the block model was classified as a Measured resource.

16.2.2 Net Smelter Return

NSR is the revenue estimated to be earned when a tonne of mineralized material is mined and processed to produce a saleable product. NSR values were calculated for the Los Azules project on a US$/t basis using Python-based scripts. The NSR calculations for Los Azules took the following parameters into consideration.

- Metal Prices
- Process / Leaching Recoveries
- Selling Costs, such as Insurance and Marketing
- Payability Rates
- Refining Charges (where applicable)
- Treatment Charges (where applicable)
- Freight Costs (only for saleable products)

Refining and treatment charges were only applied to material processed to produce a concentrate. NSR values were calculated for producing copper cathode and copper concentrate (with gold and silver credits). The NSR for concentrate production was calculated to identify the amount of mineralized material that could be processed in the future to produce a concentrate. Revenue from processing this mineralized material and producing a concentrate was not considered during the pit optimization for the PEA base mine plan as this plan considers the leachable resource tonnage.

The following expenses and parameters are not incorporated into the NSR calculation.

- Capital Expenditures
- General and Administrative Costs
- Ore Loss and Dilution
- Cost of Mining mineralized material or Waste
- Mineral Processing Costs
- Interest Expenses
- Taxes

There are two sets of NSR calculations coded into the block model. One for the leach process and one for the mill process.

16.2.2.1 Leach Process NSR Calculations

Relevant inputs for the heap-leach NSR calculation are presented in Table 16.1. Based on the provided information, 100% of the soluble copper was recoverable with the heap-leach process, in addition to 15% of the non-soluble copper grade (total copper grade minus the soluble copper grade). These recovery rates were provided by Samuel Engineering and are discussed in Section 13. The remainder of the inputs were developed by the Stantec project team.

Table 16.1: Heap-Leach Net Smelter Return Inputs for the Preliminary Economic Assessment Mine Plan

Parameter	Values	Units
Copper Selling Price	3.75	US$/lbs.
Copper Selling Cost *(Producers Brokerage Fee)*	0.02	US$/lbs.
Soluble Copper Recovery	100.00	%
Residual Copper Recovery*	15.00	%
Freight Costs	150.00	US$/dmt
**Applied to the difference between total copper and soluble copper grades*		

The first step required for calculating NSR values for the heap-leach operation was to determine the tonnes of copper recovered, as presented in Equation 1 and Equation 2. The soluble recovery rate was applied to the soluble copper grade. The additional copper recovery was applied to the remainder by subtracting the soluble copper grade from the total copper grade. This was then multiplied by the respective block mass to determine how many tonnes of copper cathode could be recovered.

Equation 1: Heap-Leach Recoverable Copper Calculation

$$\begin{aligned} Recoverable\ Cu_1 \\ &= (Copper\ Grade\ \%_{Soluble} \times Recovery_{Soluble}) \\ &+ [Recovery_{Additional} \times (Copper\ Grade\ \%_{Total} - Copper\ Grade\ \%_{Soluble})] \end{aligned}$$

Equation 2: Recoverable copper calculation for the whole block.

$$Recoverable\ Cu_2 = Block\ Volume\ \times SG\ \times Recoverable\ Cu_1$$

The value of the copper recovered in the block was the product of the metal value and the difference between the copper selling price and the copper selling cost; as shown in Equation 3. Similarly, the freight cost was the product of the weight of the cathode and the unit transportation cost, as shown in Equation 4.

Equation 3: Heap-Leach Metal Value Calculation.

$$Metal\ Value\ (\$) = Recoverable\ Cu_2\ \times (Selling\ Price - Selling\ Cost)$$

Equation 4: Freight Cost Calculation.

$$Freight\ Cost\ (\$) =\ Freight\ Rate\ \times Recoverable\ Cu_2$$

The NSR of the whole block was the difference between the metal value and the freight cost as seen in Equation 5. Finally, the NSR was calculated on a US$/t basis by dividing the block value by the product of the block volume and SG as shown in Equation 6.

Equation 5: Heap-Leach Whole Block Value Calculation.

$$Block\ Value\ (\$) = Metal\ Value - Freight\ Cost$$

Equation 6: Dollar-per-tonne NSR Calculation for the heap-leach process.

$$NSR\ (\$)_{Leach} = \frac{Block\ Value}{Block\ Volume\ \times SG}$$

16.2.3 Surface Restrictions

The economic pit optimization for Los Azules was conducted while honoring surface restrictions with the key constraint being areas identified as cryogenic geoforms (zones of subsurface frozen water). The extents of the geoforms were provided in DXF format by McEwen Mining and was obtained from the Instituto Argentino de Nivología, Glaciología (Alday, 2022). No disturbance was permitted within 50 m (measured horizontally) of the geoforms. Geoforms in the vicinity of the deposit were mostly located on the valley slopes, as shown in Figure 16.3.

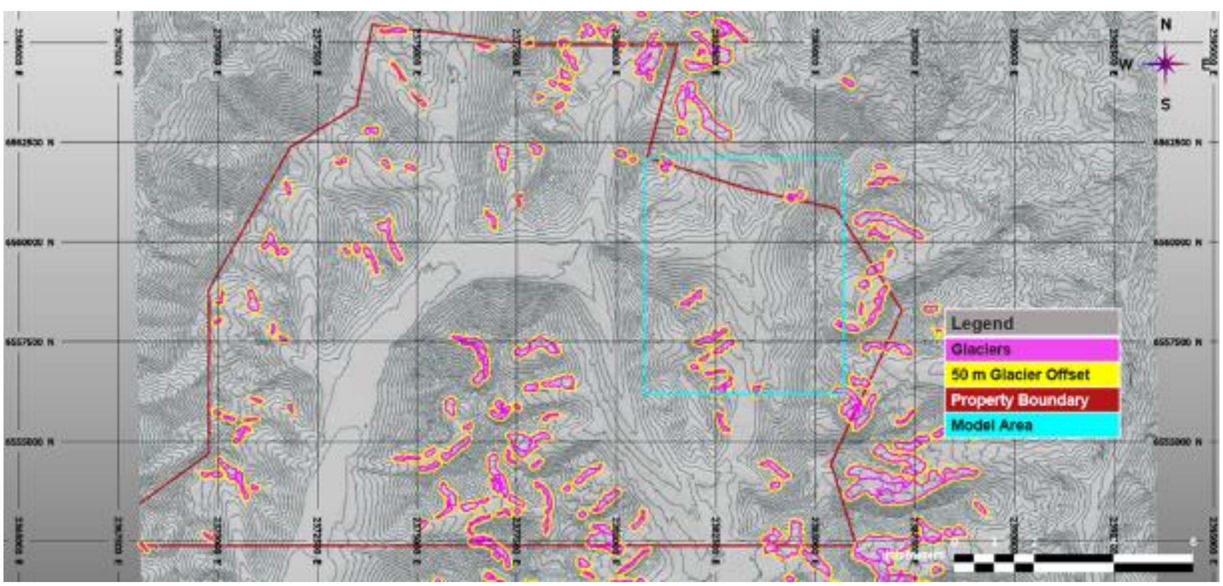


Figure 16.3: Cryogenic Landform Locations on the Los Azules Property, with 50 m Topography Contours

16.2.4 Pit Geotechnical Stability

Based on available information and ongoing iterative changes to pit plans as pit planning advances, Stantec adopted the following approach to the PEA geotechnical pit stability assessment.

- Characterize overburden and rock mass conditions using existing data.
- Conduct limit equilibrium stability analyses based on rock mass parameters.
- Seepage modeling was used to inform the phreatic surface behind the pit wall, based on assumed

boundary conditions which includes allowing for pit dewatering.

- Assessment of different pit slope heights to support interim pit phase design and allow exploitation using steeper initial pit slopes, as appropriate.
- Review of different slope angles to build relationships between factors of safety (FOS), slope angle and slope height.
- Use the results of the assessment to create slope angle guidance to inform pit design/planning.

Investigations to date have focused on defining the resource, involving generally vertical holes without orientation of core, borehole televiewer logging or triple tube drilling to facilitate core recovery. Some resource holes to date have included assessment of rock quality designation (RQD) and point load testing. Select holes have recorded assessment of rock mass rating (RMR) and inputs to RMR. At the time of doing the pit optimization and designs, no geotechnically focused drilling had been completed (geotechnical holes are being drilled during the current drilling campaign and this information will be included in future studies). As there are no boreholes with discontinuity information and existing resource holes are clustered near the center of the pit, Stantec has not created geotechnical domains but has applied the analyses across the entire pit; this approach was informed by review of rock quality data and strength parameters across the pit area. Therefore, there is no structural information for the pit wall assessment at this stage, which limits the stability assessment to rock mass controlled rather than structurally controlled (kinematic) stability.

This is a key limitation of the study, especially in review of existing pit examples including Los Pelambres (Eggers, 2016), but is not uncommon for PEA level studies, where evaluation for the early stages of pit viability mining can be based on judgment or experience in similar environments (Read & Stacey, 2009).

The results of a Limit Equilibrium Analyses undertaken are Stantec is presented in Figure 16.4, where the green field shows the allowable factors of safety.

Anticipated Factor of Safety - Based on Slope Face Angle and Pit Depth						
Angle of Slope Face	Depth of Pit (m)					
	400	500	600	800	1000	1200
30°	-	-	-	1.46	-	1.26
32°	-	-	1.53	1.38	-	1.20
34°	-	-	1.46	1.32	1.23	1.14
36°	-	-	1.40	1.26	1.16	1.09
38°	-	1.43	1.34	1.20	-	1.04
40°	1.50	1.37	1.28	1.14	-	0.98
42°	1.44	1.31	1.22	-	-	-

Figure 16.4: Factor of Safety with Slope Angle and Pit Depth

Based on the 2022 geotechnical characterization, assumptions and analyses, the recommendations for the Los Azules PEA overall slope angles are presented in Table 16.2.

Table 16.2: Overall Suggested PEA Pit Slope Angles, by Overall Pit Depth	
Total Pit depths	**Angle of slope in rock**
up to 600m	42°
600m to 800m	38°
800m to 1000m	34°
1000m to 1200m	32°

Notes:

Pits 800 m to 1200 m are very deep for the rock quality indicated by the available data; there is little empirical experience in this range and the slope angles remain uncertain at this stage.

Seismic analysis has not yet been undertaken.

Hydrogeology is based on assumptions and dewatering is likely required for the angles given.

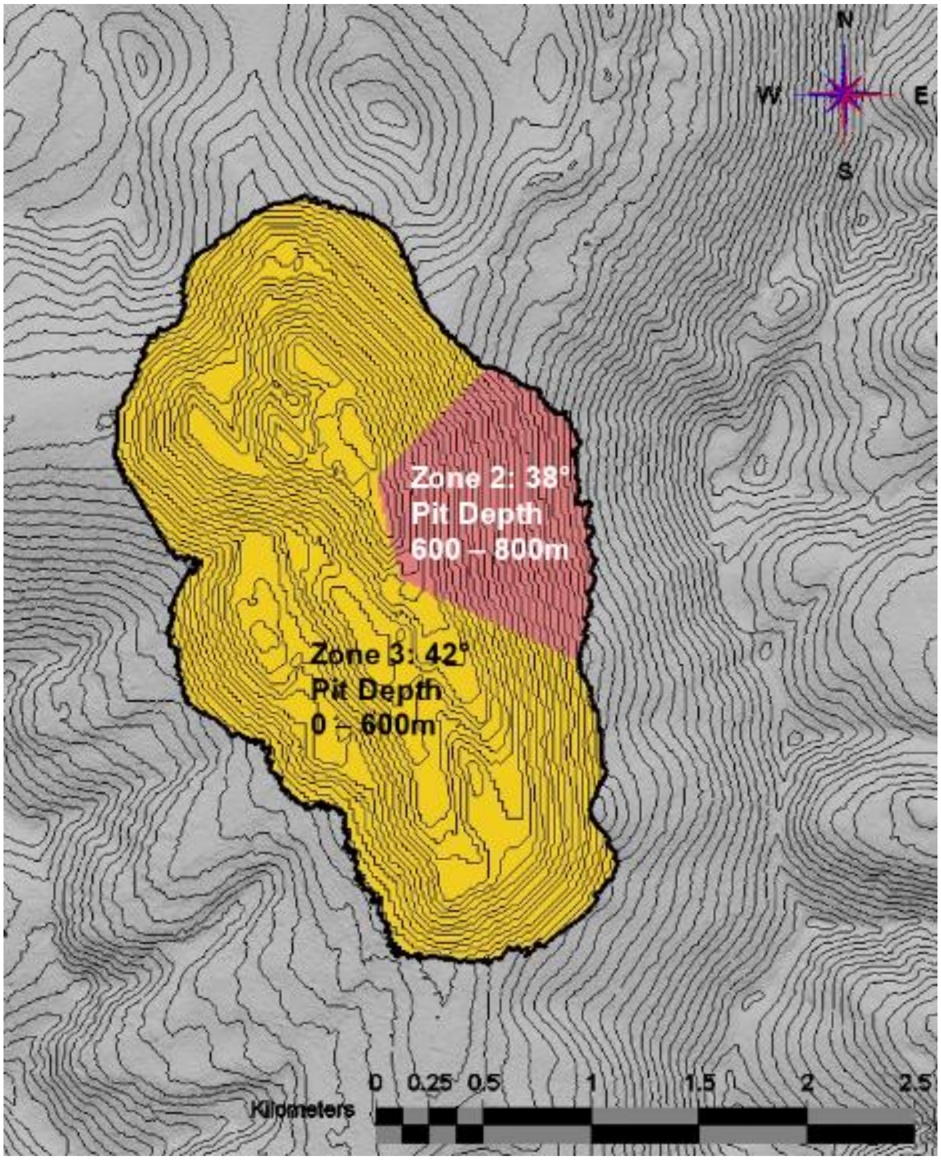
Pit depths allowed for 30 m overburden (OB); 2 x 15 m bench heights at 34 degrees BFA, 8 m catch bench within OB, 15 m catch bench between OB and rock. Giving inter-ramp angle of 30 degrees. Overburden depth will vary which may affect the combined rock/overburden overall slope angles.

Failure surfaces completely within the blast damaged zone are not included; Stantec assumes bench design will account for managing bench scale stability.

Pit slope angles are not incremental, i.e., a 700 m pit slope requires the entire slope to be at 38° degrees, not stepped down.

The pit slope angle guidance for the Los Azules pit optimization was based on the overall height of the slope and the overall wall angle. The slope angles provided in Table 16.2 were applied to the entire slope, not incrementally as the pits progressed deeper. The pit optimization was refined through multiple iterations where each slope angle was applied to certain parts of the pit wall. This refining process was repeated to ensure that the geotechnical guidance was applied accurately.

The overall slope angles presented above only apply to pit walls excavated through rock. An inter-ramp angle of 30° was assumed for pit slopes excavated through overburden. A plan of the model slope zones is presented in Figure 16.5. Slope zones correspond to overall slopes angles presented in Table 16.2. Slope zones 2 and 3 were coded vertically through the entire block model. Slope zone 1 (the overburden zone) has an irregular shape and was coded over zones 2 and 3 where present.



Note: Zone 1 is overburden and used where present in the block model.

Figure 16.5: Slope Zones for ultimate selected pit shell

It is assumed that the future bench design guidance will account for managing bench scale stability including potential failure surfaces completely within the blast damaged zone. Design guidance for benches has not been included at this stage.

16.2.5 Mining Parameters and Production Rates

Samuel Engineering, Whittle Consulting, and Stantec collaborated to develop a target copper cathode production rate that seeks to maximize project value. Several cases with different production capacities were analyzed over time, and each case had a different capital requirement, revenue stream and ultimate value. A heap leach stacking plan was also developed for each case.

Two pit optimization scenarios were carried forward: the first at a production rate of 175 ktpa of copper cathode (base case), and the second a rate of 125 ktpa (alternative case). Input parameters common to both cases are presented in Table 16.3. These two scenarios were selected based on high-level trade-off studies that were conducted and identified an optimal range of copper cathode production of between 125ktp to 175ktpa. Production rates below 125kpta had difficulty supporting the capital costs required for fixed site costs (power, access, and services). Production rates above 175ktpa of copper cathode are hard to sustain throughout the mine life and does not justify increased crushing, stacking, and SX/EW capacity for only a few years.

Table 16.3: Pit Optimization Input Parameters

Parameter	Value	Units
Processing Cost	3.48	US$/t
Dilution Rate	5	%
Mining Recovery Rate	95	%
Default In Situ Density*	2.53	t/m^3
Stockpile Reclaim Cost	1.00	US$/t
Stockpile Grade Recovery Rate	90	%
Maximum Stockpile Size	100	Mt
Bench Height	15	m
Maximum Vertical Advance Rate	12	Benches Per Year
Pre-Stripping Rate	50	Mt/a
Pre-Stripping Period Length	1	Year
Discount rate	8	% Per Year

*Only applied to blocks without a density value defined.

The estimated processing cost was supplied by Samuel Engineering. Dilution and mine recovery rates were specified by Stantec. High recovery rates for mineralized material and low dilution rates are common assumptions for bulk mining in massive-type copper-porphyry deposits. Rock density values were included in the block model. The default density stated above was only used if a density value was not defined in a particular block and is not expected to influence the final optimization in a material manner.

Stantec is of the opinion that a large stockpile could be constructed north of the pit to allow low-grade mineralized material to be stored while higher grade material is processed during the early years of the project. The stockpile size for design and optimization purposes was limited to 100 million tonnes. Rehandling of mineralized material for processing will require loading and hauling to a primary crusher. Stantec estimated a re-handle cost of US$1.00/t for pit optimization purposes, which consisted of known haulage and loading costs. The stockpile rehandle costs were then included in equipment productivity calculations to derive the total mining cost.

A bench height of 15 m was selected because it is suitable for bulk mining with large mining equipment and the block model was built using 15 m in the z direction. The various stages of the mining cycle (drilling, blasting, loading, hauling, and auxiliary activities) tend to limit the vertical advance rate (VAR) for large pits and a VAR limit of 12 benches per year, per operating phase has been applied at Los Azules.

Discounted cash flow evaluations for both production scenarios were discounted at a rate of 8% per year, which is deemed appropriate for the PEA level of evaluation.

The 175 ktpa production case benefits from greater revenues and a reduced mine operating cost because of higher copper cathode production in the early years of the mine plan and the use of larger class of haul trucks and loading equipment. The maximum mining rate and mineralized material stacking rates are also greater for the 175 ktpa case to support the increased demand for process material. Parameters are summarized in Table 16.4.

Table 16.4: Pit Optimization Parameters Specific to Each Production Case.

	Value		
Parameter	**175 ktpa (Base Case)**	**125 ktpa (Alternative Case)**	**Units**
Mining Cost*	1.90	2.05	US$/t
Maximum Mining Rate	160	145	Mt/a
Maximum mineralized material Stacking Rate	100	80	Mt/a
**Includes General & Administrative Costs*			

16.2.6 Pit Optimization Pits and Phase Selection

The pit optimization produces a series of nested pit shells at revenue factors in 0.02 increments. Revenue factors are incremental multipliers of the metal selling prices. For example, a revenue factor of 0.5 represents the pit shell generated at a metal value of 50% of the base case prices. It should also be noted that the pit shells generated by this process are not designs as they do not consider access, catch benches, safety berms and other features incorporated into the final pit design. Pit phasing increases the estimated discounted value during the pit optimization process by mining mineralized material earlier and delaying some waste mining.

Pit shells were generated for revenue factors 0 – 1.00 with slope codes designed to support pit shells up to 800 m deep. The first revenue factor to yield a pit shell for the 125 ktpa case was 0.18 because it was only at this revenue factor that the pit shell was able to overcome the early-stage costs of mining the non-mineralized waste cover. As discussed in the resource geology section, there is a flat-lying layer of non-mineralized material in the valley bottom that is approximately 75 m thick and acts like a fixed mining cost that needs to be overcome to create the first economic pit shell that releases mineralized material.

The optimization analyses produced shells up to a revenue factor of 1.00, as defined. However, the largest pit shell with slopes that adhered to the slope design criteria was generated at revenue factor 0.72. Pits created beyond the 0.72 revenue factor were greater than 800 m deep; at which point the entire slope would have to be shallowed from 38° to 34° to support slopes up to 1,000 m high.

The same pattern was seen in the 175 ktpa case, but pits with valid slopes were produced for revenue factors from 0.18 to 0.70. The geometry of the 175 ktpa pit shells was slightly different than the 125 ktpa case because they were created with different input mining costs. Despite the difference, the revenue factor 0.72 pit shell created for the 125 ktpa case was effectively the same configuration as the revenue factor 0.70 pit shell produced for the 175 ktpa case. The overall difference in tonnage was less than 1%.

The pit-by-pit graphs shown in Figure 16.6 and Figure 16.7 summarizes the results of the mineralized material and waste tonnage of each pit generated, along with indicative discounted value estimates for the 125 ktpa and 175 ktpa cases, respectively. It is important to note that the values presented should be viewed as relative comparisons to one another and they do not represent net present values for the overall project. These indicative value estimates are discounted at a rate of 8% per year and do not include capital, royalties, taxes, or any other costs than have not been discussed in this section of the report. Indicative value estimates are created with input operating costs that are based on the best available information. True operating costs were calculated once a full detailed mine plan was developed. Variance between the cost inputs for optimization and final mining costs following mine design and schedule are to be expected.

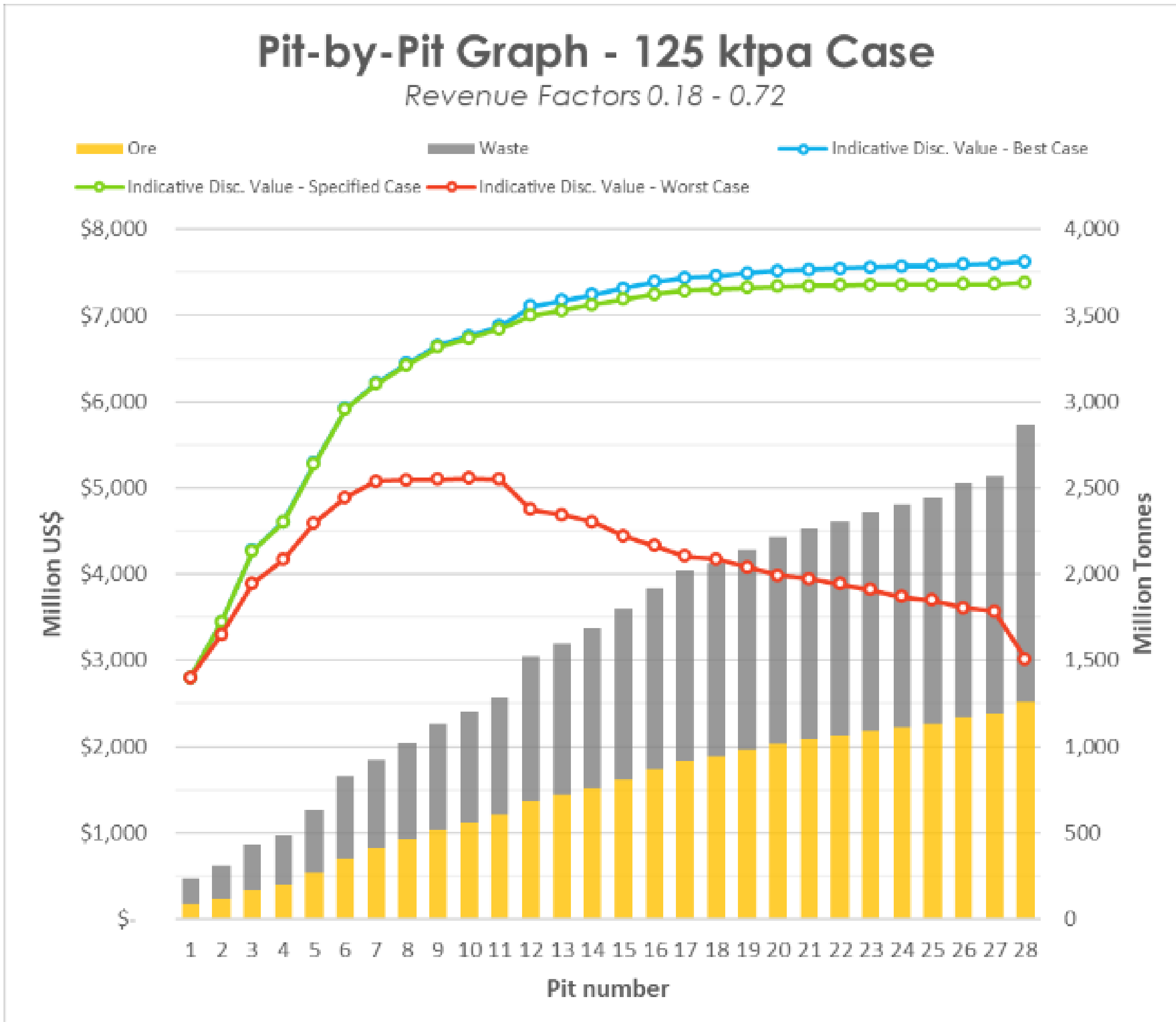


Figure 16.6: Pit-by-Pit graph Economic Pit Optimization for the 125 ktpa production alternative case

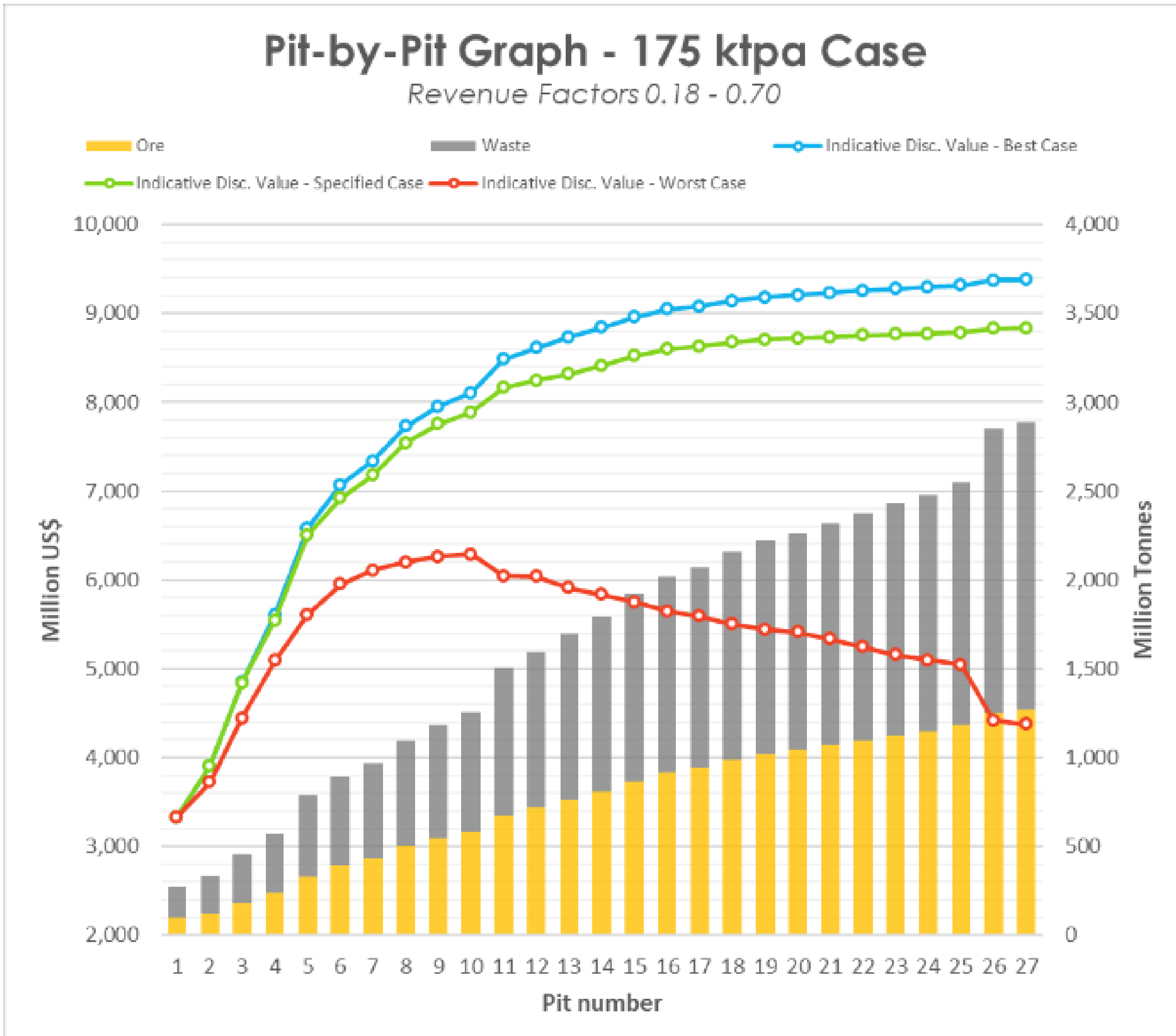


Figure 16.7: Pit-by-Pit graph Economic Pit Optimization for the 175 ktpa production base case.

The discounted value for each pit shell is calculated for the best, the specified, and the worst-case scenario. All three scenarios are described as follows:

- **Best case –** Discounted value is determined by scheduling every preceding pit as a pushback to create as many mining phases as possible.
- **Specified case –** Discounted value is determined by scheduling the pit with selected pit phases.
- **Worst case –** Discounted value is determined by mining the largest selected pit without any mining phases.

Given the close-spacing of the revenue increments used (0.02), the best-case scenario is unachievable in practice because it often requires mining in very narrow pushbacks (usually 20-40 m wide) which are not feasible for large-scale bulk mining.

The worst-case scenario is achievable if a pit is mined without any phasing, but this would not represent a logical mining strategy. The specified case is intended to represent an achievable mining scenario which can serve as the basis for developing a mining strategy and sequence. The best and worst cases can be considered bounding cases. The value of the specified case is always between the best and worst-case scenarios.

The specified case for the 125 ktpa production scenario used pit shells 1, 3, 5, 7, and 12 as interim pushbacks to reach the final pit shell 27 (the largest pit selected). Pushbacks were selected on their ability to add value, based on the pit-by-pit graph, their available working space (minimum mining width of 120 m), and the length of time required to mine a pushback. A substantial pushback in a mine of this scale should take at least three years to mine. A smaller pushback may not effectively utilize the pushback given the VAR restriction of 12 benches per year. The specified case for the 175 ktpa production scenario used pits 1, 4, 6, and 13 as interim pushbacks to reach the final pit shell 25 (the largest pit selected). Neither largest selected pit shell encroached on the 50 m offset zone around the cryogenic geoforms.

The largest pit shells were selected because they achieved the peak discounted value using the optimization parameters. The deposit may be able to support a longer life project given there are still resource tonnes not included, but the effect of the discount rate effectively reduces the value of further pushbacks to the point that it would erode the project's present value. The pit-by-pit graphs for both cases show the specified case offers incrementally more value by selecting one or two more pushbacks, however mining those incrementally larger shells is deemed unfeasible given the pushback width required for a bulk mining fleet.

Interim pushbacks were selected based on the value added and the practicality of mining them using a large bulk mining fleet. The actual mining sequence will be different (discussed in 16.4), but it will be guided by the phases produced from the pit optimization. The 125 ktpa specified case achieved a peak indicative discounted value of US$7.4 billion and the 175 ktpa case achieved US$8.8 billion in peak indicative value (excluding capital). The 175 ktpa case benefits in terms of discounted value by producing more revenue over a mine life of 25 years as opposed to 32 years for mining essentially the same pit in the 125 ktpa case. While the 175 ktpa case generates a higher discount value, it will also require more capital to develop and operate a mine at that level of production.

It is important to note that these values are indicative estimates calculated during the pit optimization to be used to guide selection of pit limits and phasing. The values cannot be construed as net present value estimates for the overall Los Azules project. They do not consider actual capital requirements, development timelines and are estimated from coarse mining schedules with simplified inputs.

16.3 MINING PHASES AND PIT DESIGN

The pit shells generated and selected during the pit optimization process were used as the basis for the pit design. The pit shells follow the geotechnical guidance for overall slope angles but do not have detailed bench, access, and ramp designs. Given the scale of the pit, it was assumed that the pit shell inter-ramp slope angles could be steepened to accommodate highwall ramps while still following the overall slope angle guidance. Many areas in the selected pit shells are mined down to a V-shaped point along the bottom of the pit since the optimization process is based on cone-shaped pits. Mining down to a point is impossible in practice given the need for equipment access, which is why Stantec adopted a minimum mining width of approximately 60 m for final pit benches. Interim pit shell phases were not truncated to simplify scheduling for the PEA but the small variance in tonnage between pit phases does not materially influence the schedule.

Stantec determined that the material quantities in the selected interim pit shells are representative for mine scheduling at a PEA level. A more refined definition of the phased quantities should be addressed at a more detailed level of study. Both production scenarios presented (the 125 and 175 ktpa cases) share the same ultimate pit shell limit and, therefore, will have the same final pit design. Overall mined quantities by resource classification are shown in Table 16.5. Quantities assume mineralized material loss equals dilution and are based on using a leaching scenario NSR cutoff of $3.48/t.

Table 16.5: Pit Quantities by Resource Classification.

Resource Classification	Material/Grade Item	Value	Units
Indicated	Leachable Material	804.4	Mt
	Copper Grade	0.50	%
	Soluble Copper Grade	0.34	%
	Recoverable Copper	2.91	Mt
	Leach NSR	27.26	US$/t
Inferred	Leachable Material	378.0	Mt
	Copper Grade	0.38	%
	Soluble Copper Grade	0.26	%
	Recoverable Copper	1.03	Mt
	Leach NSR	20.60	US$/t
Total	Leachable Material	1,182.4	Mt
	Copper Grade	0.46	%
	Soluble Copper Grade	0.31	%
	Recoverable Copper	3.94	Mt
	Leach NSR	25.13	US$/t
	Waste	1,366.2	Mt
	Stripping Ratio	1.16	Waste t/Leachable material t

Pre- and post-mining plan views of the ultimate pit shell are shown in Figure 16.8 and Figure 16.9, respectively. The selected pit is approximately 3.7 km long, and 2.1 km wide (measured in plan). The highest point in the pit is at an elevation of 3,960 m and the lowest point is at 3,130 m giving a maximum wall height of approximately 830 m.

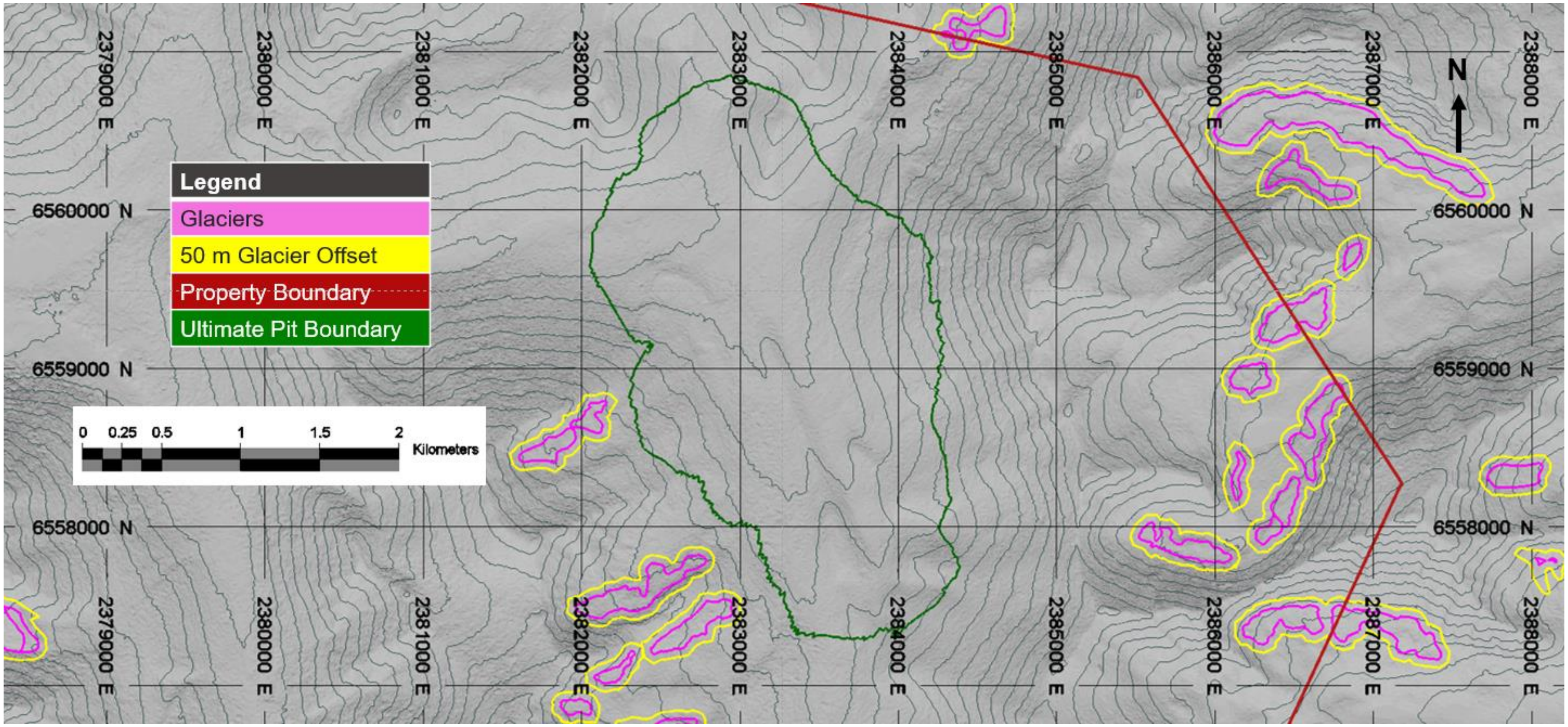


Figure 16.8: Pit Area Prior to Mining, with 50 m Topography Contours

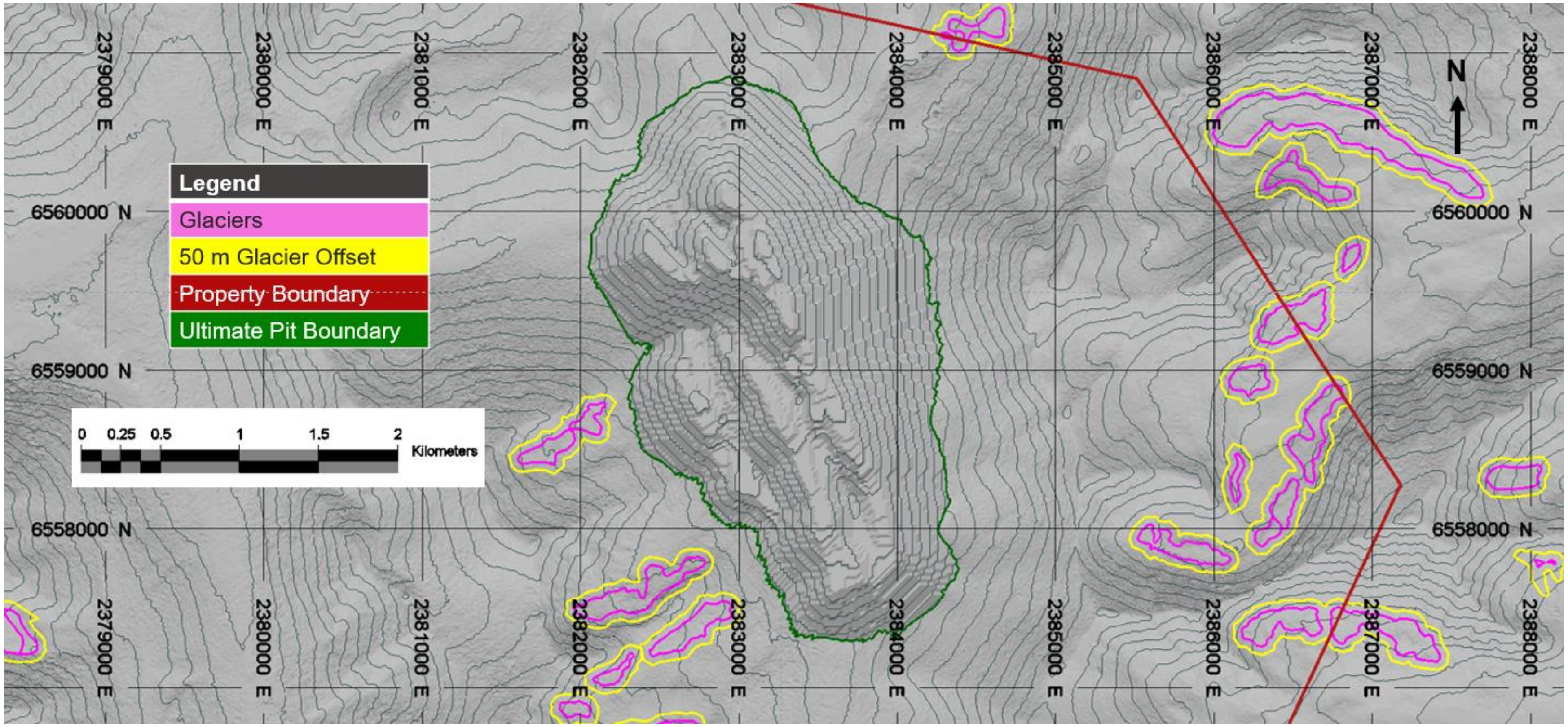


Figure 16.9: Largest Selected Leach Only Pit, with 50 m Topography Contours

16.3.1 Phasing Plan for the 175 ktpa Production Scenario (Base Case)

The phase 1 pit is the smallest pit shell generated during the pit optimization (revenue factor 0.18.) It is subdivided into four operating areas, phases 1A1, 1A2, 1B1, and 1B2, that allow for early access to mineralized material and deferral of waste stripping. Phase 1A1 is designed to facilitate early mineralized material access with minimal waste stripping. Phase 1A2 is primarily a waste stripping phase, whereas phases 1B1 and 1B2 contain most of the phase 1 mineralized material. Phase 1B1 was underneath 1A2 and is not visible in the plan view.

Phase 2A expands the north end of the phase 1 pit, behind phases 1A2, 1B1, and 1B2. Phase 1B2 allows for stripping waste from the east slope, approximately 120 m above the valley floor. Phase 2A allows for waste stripping on the north slope approximately 120 m above the valley floor.

Phase 3A1 and 3A2 expand the pit to the south, through the valley floor. The stripping ratio through this area of the pit (phases 2B-3A2) is relatively consistent, ranging from 2.2 - 2.6:1. The stripping ratio drops to 0.5:1 in phase 3B, which expands the north end of the pit to the west.

Phase 4 is split into three sub-phases: 4A, 4B, and 4C. Phase 4A is designed to release a balanced amount of mineralized material and waste by expanding the pit to the north. Phase 4B is designed to be a higher strip (1.8:1) pushback that expands the pit into the eastern hillside above the Los Azules deposit. Phase 4C expands the opposite side of the pit into the western hillside at a strip ratio of 0.6:1. All three sub-phases release a similar amount of mineralized material (46-51 Mt), but at varying metal grades.

Phase 5 is split into five sub-phases: 5A1, 5A2, 5A3, 5B, and 5C. Phase 5A1 is designed to remove waste overlying 5A2 and 5A3, and it does not contain any processable material. Phases 5A2 and 5A3 expand the pit to the north and west with balanced stripping ratios. Phase 5B is connected across base of the pit (under phase 1 and 2) but is composed of separate east and west working areas above an elevation of 3,400 m. The final pushback, phase 5C, expands the pit to the south. Unlike the balanced stripping ratios from phase 5A2-5B, the stripping ratio of phase 5C roughly doubles to 2.2:1. Despite the greater waste stripping requirement, phase 5C contains less mineralized material and waste than phase 5B, and the grade of the mineralized material is higher as well. On a US$/t basis, NSR values for mineralized material in phase 5C are 46% more valuable than the mineralized material in 5B.

Cross sections of deposit and phases are presented in Figure 16.10 to Figure 16.13. Note that NSR values are only shown for blocks that are classified as a mineral resource, and with NSR values greater than the processing cost ($3.48/t). Table 16.6 presents the pit quantities by phase for the 175 ktpa case.

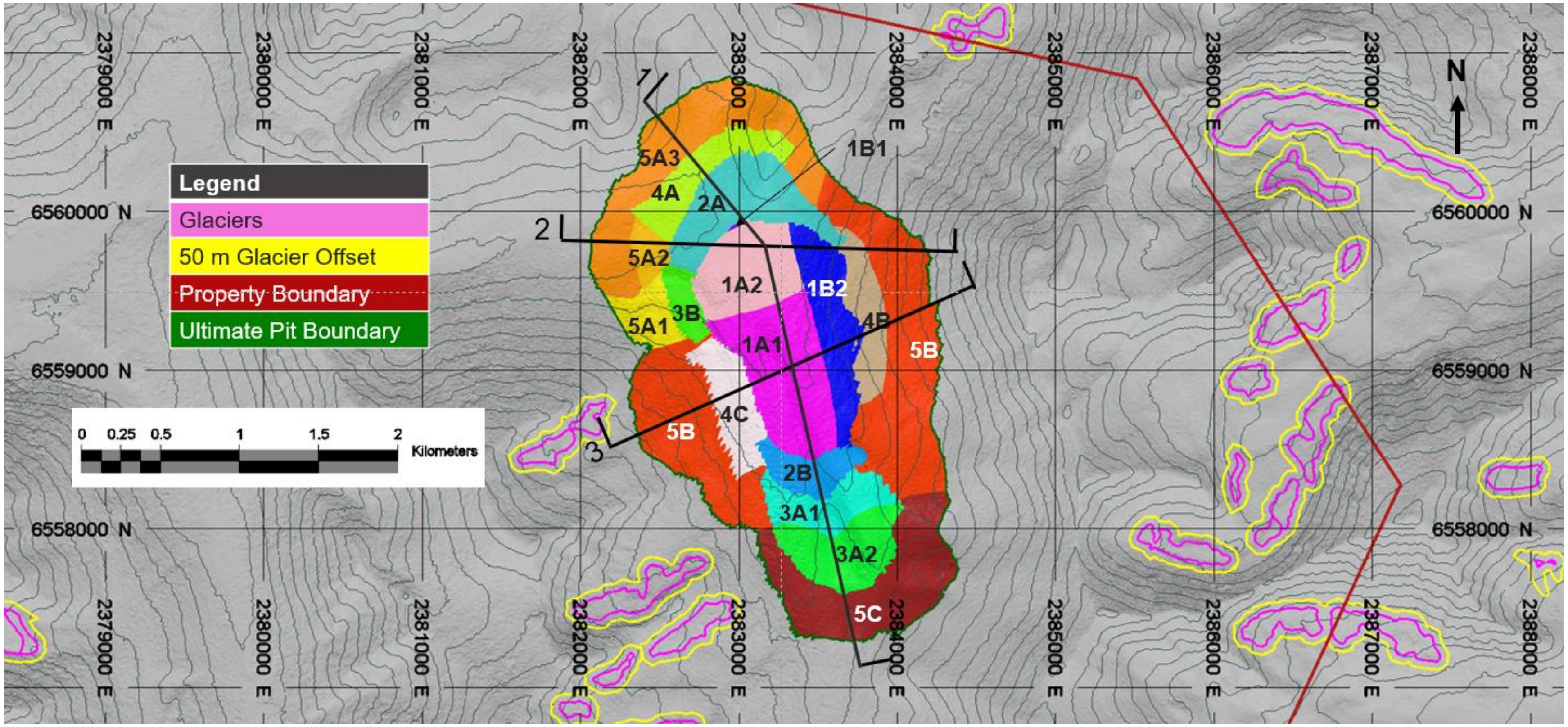


Figure 16.10: Cross Section Plan

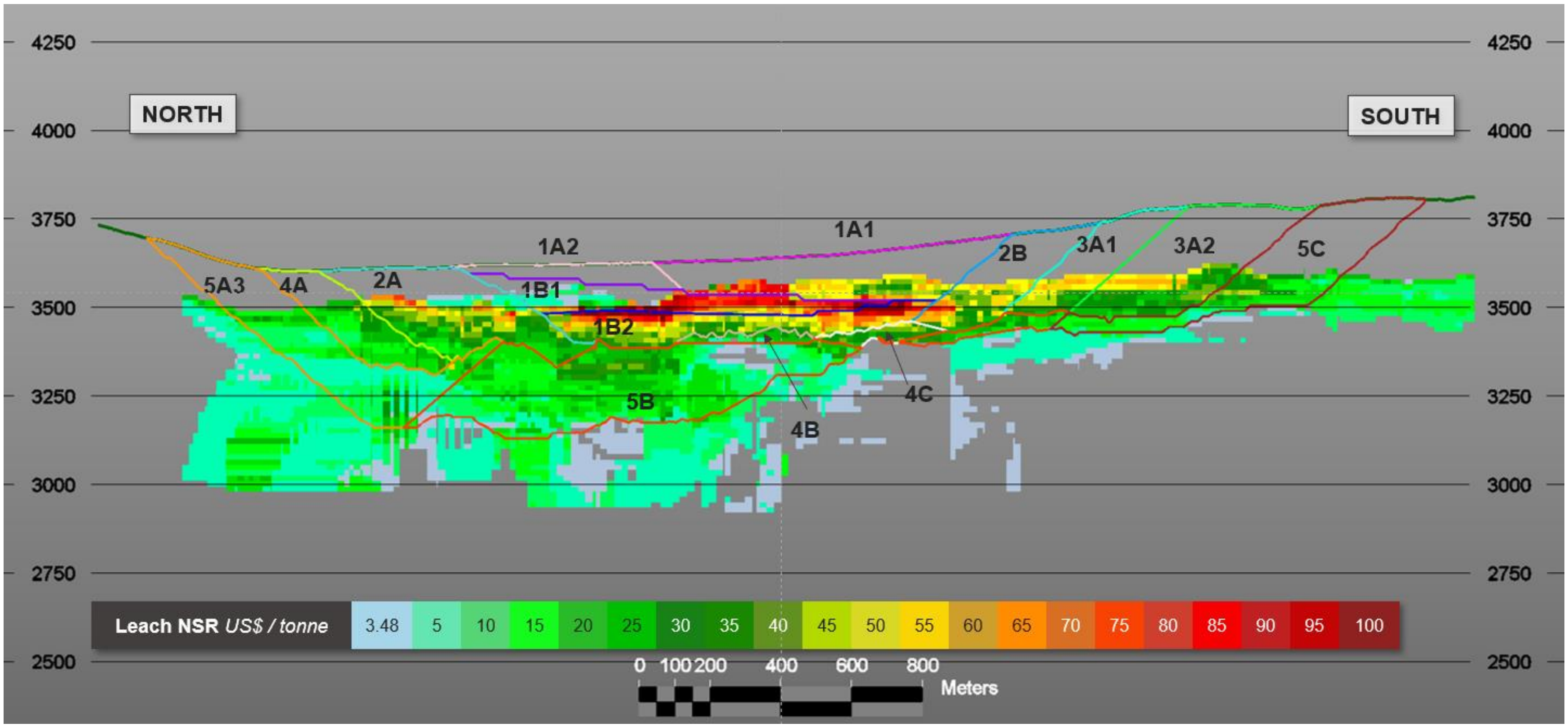


Figure 16.11: Cross Section 1 with Pit Phasing and Leach NSR Values

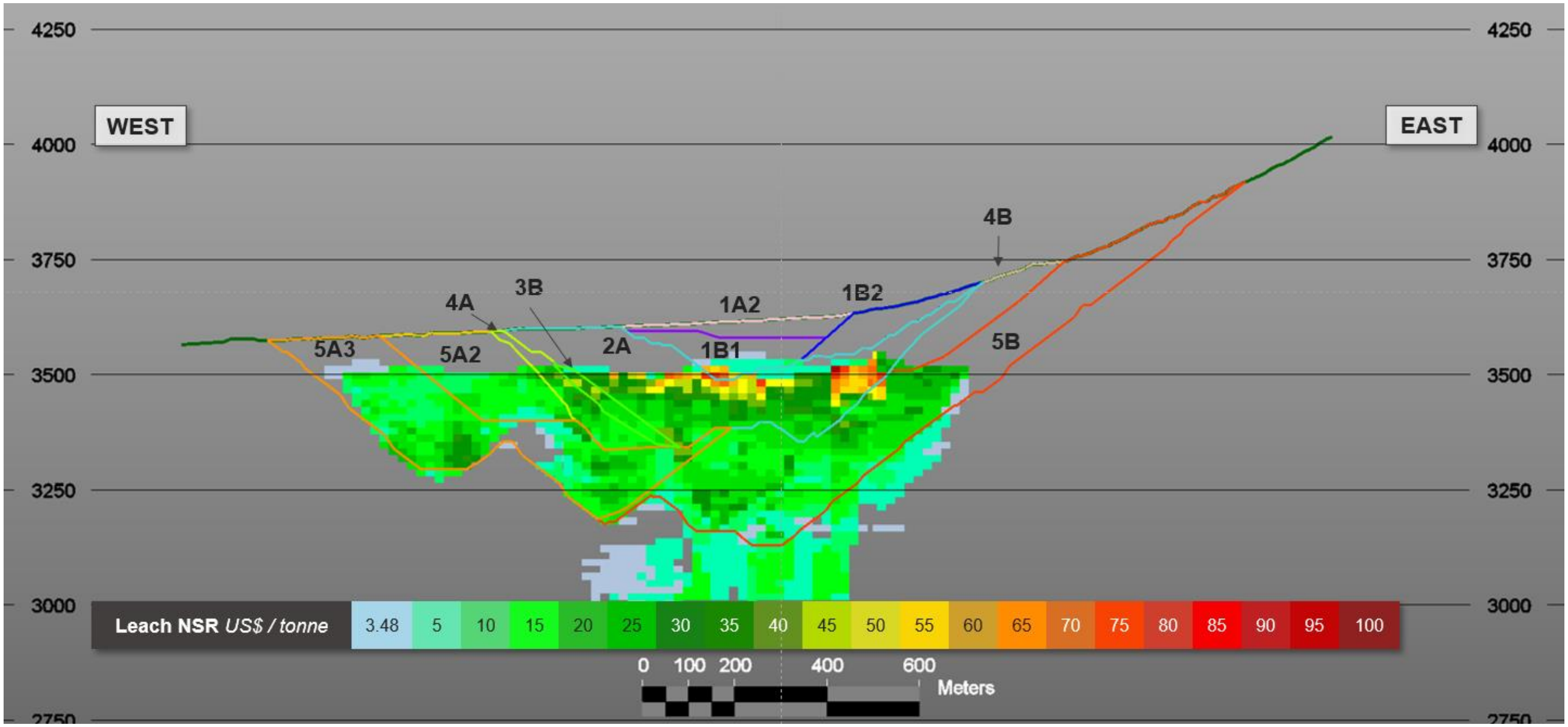


Figure 16.12: Cross Section 2 with Pit Phasing and Leach NSR Values

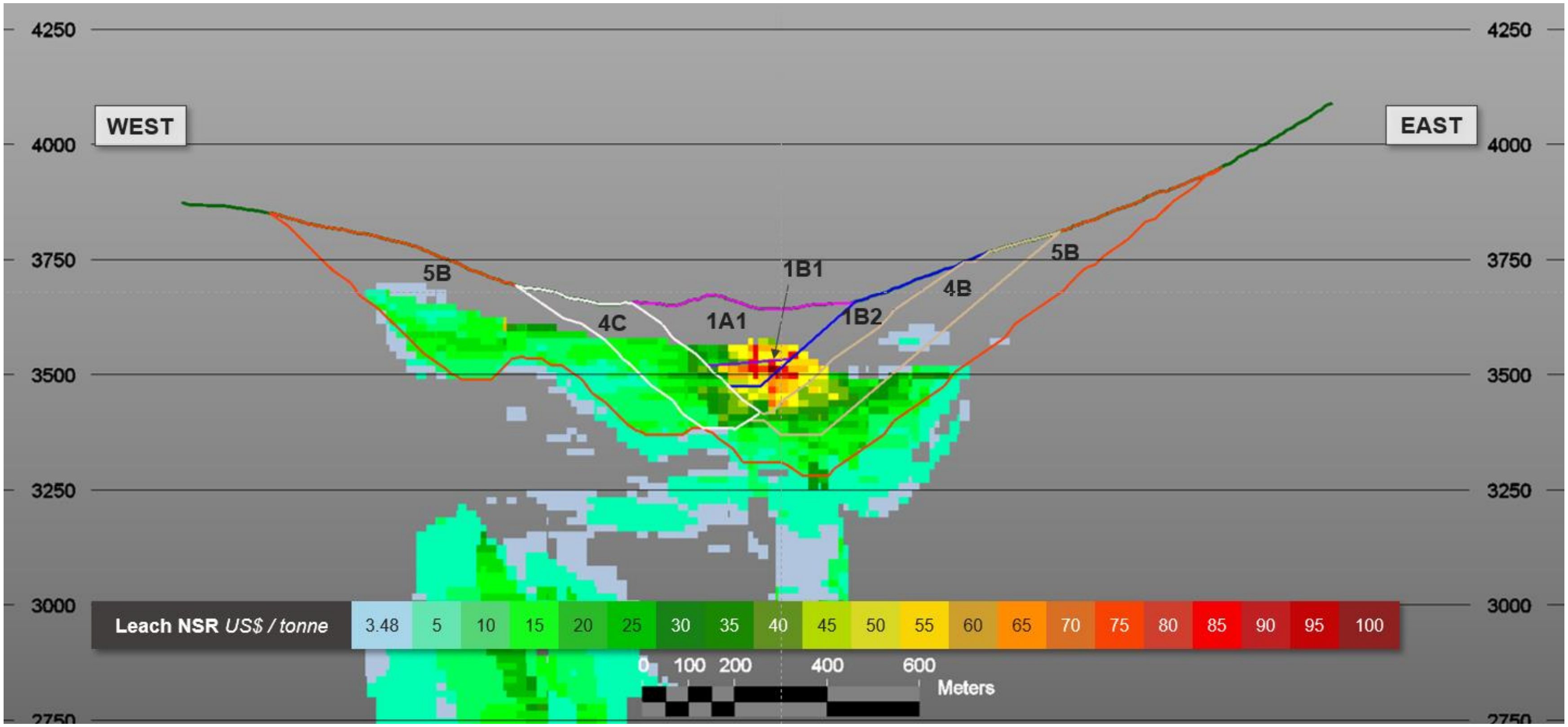


Figure 16.13: Cross Section 3 with Pit Phasing and Leach NSR Values

Table 16.6: Pit Quantities by Phase for the 175 ktpa Case										
			Waste Material		Leachable Material					
Mining Phase	**Total**	**Strip Ratio**	**Waste**	**Potential Mill Material[1]**	**Tonnage**	**Copper Grade**	**Soluble Copper Grade**	**Gold Grade**	**Silver Grade**	**NSR**
-	*Mt*	*(Waste t/Leach t)*	*Mt*	*Mt*	*Mt*	*%*	*%*	*g/t*	*g/t*	*US$/t*
1A1	85.0	4.0	64.4	3.8	16.9	0.61	0.69	0.06	1.29	49.53
1A2	34.4	60.9	32.7	1.1	0.6	0.49	0.55	0.06	1.66	39.58
1B1	55.3	0.5	13.7	3.4	38.1	0.63	0.76	0.07	1.54	51.06
1B2	97.5	1.3	50.3	3.9	43.3	0.75	0.92	0.08	1.51	60.50
2A	139.1	1.1	67.5	5.5	66.0	0.45	0.61	0.06	1.72	36.10
2B	37.5	2.5	26.9	0.1	10.6	0.59	0.70	0.06	1.06	47.54
3A1	45.9	2.6	30.4	2.6	12.9	0.49	0.59	0.06	0.96	39.18
3A2	93.4	2.2	49.8	14.3	29.3	0.49	0.62	0.09	1.18	39.60
3B	58.9	0.5	17.6	1.7	39.7	0.36	0.50	0.04	1.31	28.70
4A	95.6	1.1	48.9	0.8	45.9	0.29	0.44	0.04	1.42	23.43
4B	128.0	1.8	75.1	6.9	46.0	0.40	0.53	0.06	1.20	32.21
4C	83.0	0.6	31.4	0.1	51.4	0.29	0.39	0.03	0.92	23.19
5A1	8.0	0.0	8.0	0.0	0.0	0.00	0.00	0.00	0.00	0.00
5A2	57.9	1.0	27.5	1.7	28.7	0.25	0.30	0.02	0.81	19.89
5A3	379.4	0.8	153.3	18.6	207.5	0.20	0.32	0.03	1.10	16.02
5B	894.8	0.9	382.1	45.9	466.7	0.23	0.40	0.05	1.28	18.82
5C	254.9	2.2	157.2	18.8	78.9	0.34	0.49	0.05	1.07	27.52
Total	**2,548.5**	**1.16**	**1,237**	**129.3**	**1,182.4**	**0.31**	**0.46**	**0.05**	**1.25**	**25.13**

[1]Average in situ grade of potential mill material: Total Copper (0.107%), Gold (0.069 g/t), Silver (0.910 g/t). Determined with mill NSR cutoff values of $5.46/t for supergene material, and $5.43/t for primary material. Due to rounding, values may not sum.

16.3.2 Phasing Plan for the 125 ktpa Production Scenario (Alternative Case)

The phasing plan for the 125 ktpa production case is presented in Figure 16.14. The pit has been divided into 13 operating phases to target release of high-grade material early in operations, and to manage waste rock stripping. The phasing sequence was guided by the pit optimization process to maximize value as described in the previous section. Mining quantities for each phase are summarized in Table 16.7.

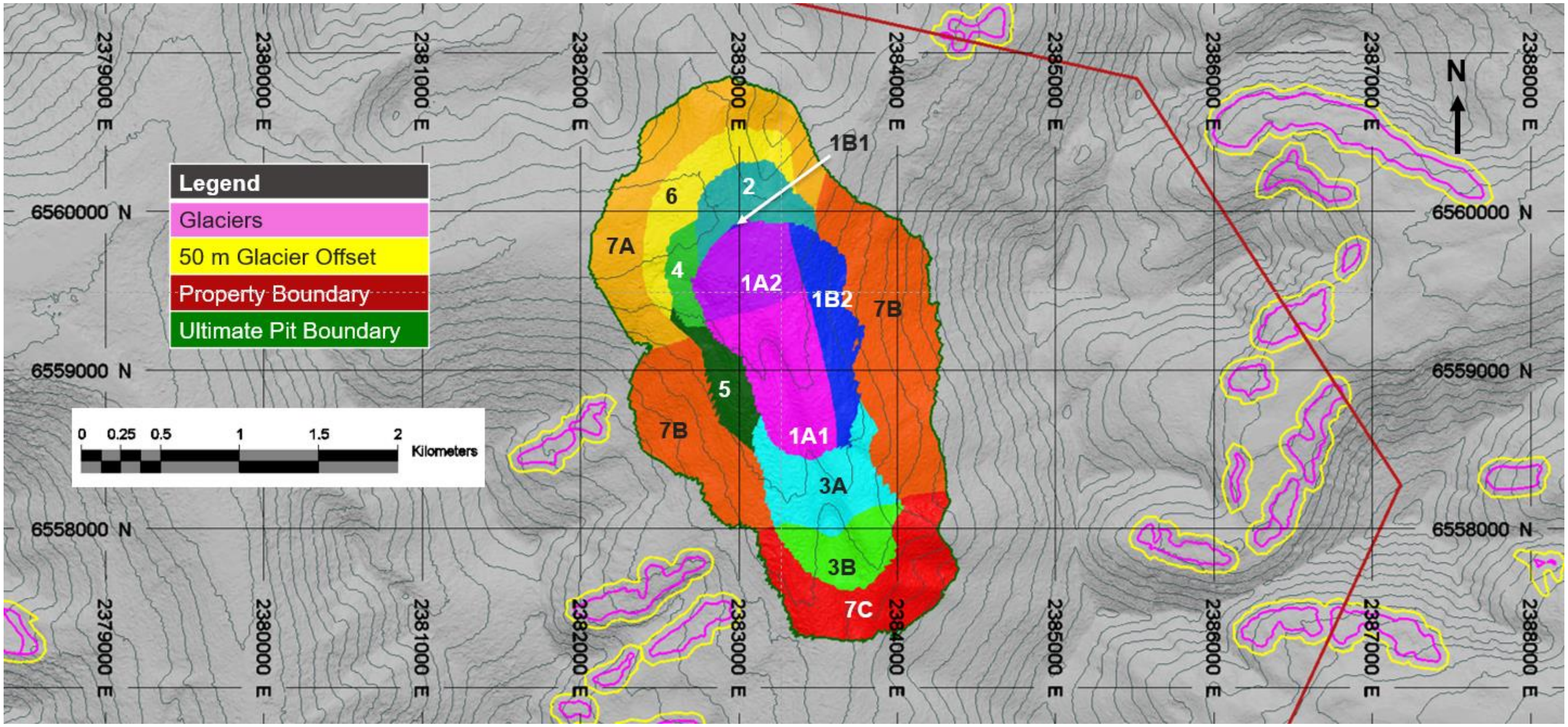


Figure 16.14: Phasing Plan for the 125 ktpa Case.

The phase 1 pit was the smallest pit shell generated during the pit optimization (revenue factor 0.18). It was sub-divided into four operating areas (phases 1A1, 1A2, 1B1, and 1B2) to allow for early access to mineralized material and deferral of waste stripping. Phase 1A1 is set up to facilitate early mineralized material access with minimal waste stripping. Phase 1A2 is primarily a waste stripping phase, whereas phases 1B1 and 1B2 contain most of the phase 1 process material. Phase 1B1 is underneath 1A2 and does not show in the plan view.

Phase 2 expands the north end of the phase 1 pit. Phase 1B2 allows for stripping waste from the east slope, approximately 120 m above the valley floor. Phase 2 allows for waste stripping on the north slope approximately 100 m above the valley floor.

Phase 3 expands the pit to the south, across the valley floor. Phase 3 is split into two portions: 3A and 3B. Transitioning from phase 3B to phase 4, the stripping ratio drops from 2.35:1 to 0.76:1 because much of the waste the valley bottom was mined by prior phases. Phase 4 pushes the north end of the pit further to the west. It is followed by phase 5 immediately to the south, and phase 6 to the north and west. The stripping ratios for phases 4 to 6 are all less than 1:1.

Phase 7 is split into three areas: 7A, 7B, and 7C. Phase 7A begins in the north end of the pit, followed by 7B in the middle, and 7C to the south. Phase 7B is connected through the base of the pit (under phase 5) but includes separate east and west working areas above an elevation of 3,385 m. Phases 7A and 7B have equivalent stripping ratios. On a US$/t NSR basis, the mineralized material in phase 7B is 25% more valuable than the mineralized material in 7A, however phase 7B has approximately 2.4 times more waste material than 7A.

The average mineralized material NSR value increased from $19.95/t to $27.55/t from phases 7B to 7C, and the stripping ratio doubled. The overall amount of waste stripping required is much lower in phase 7C than in 7B.

Cross sections of resource block model and phases are presented in Figure 16.15 to Figure 16.18. Note that NSR values are only shown for blocks that are classified as a mineral resource and with NSR values greater than the processing cost ($3.48/t).

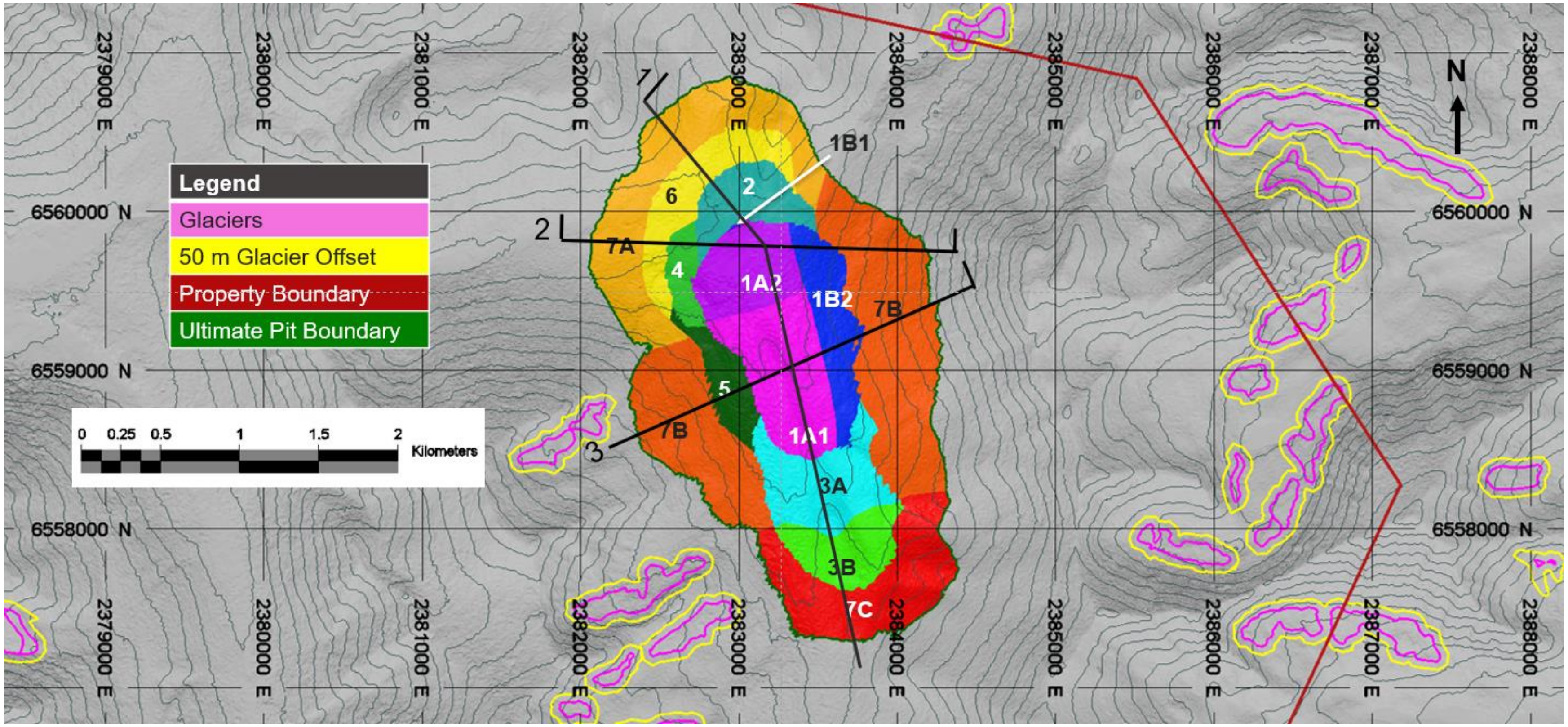


Figure 16.15: Cross Section Plan

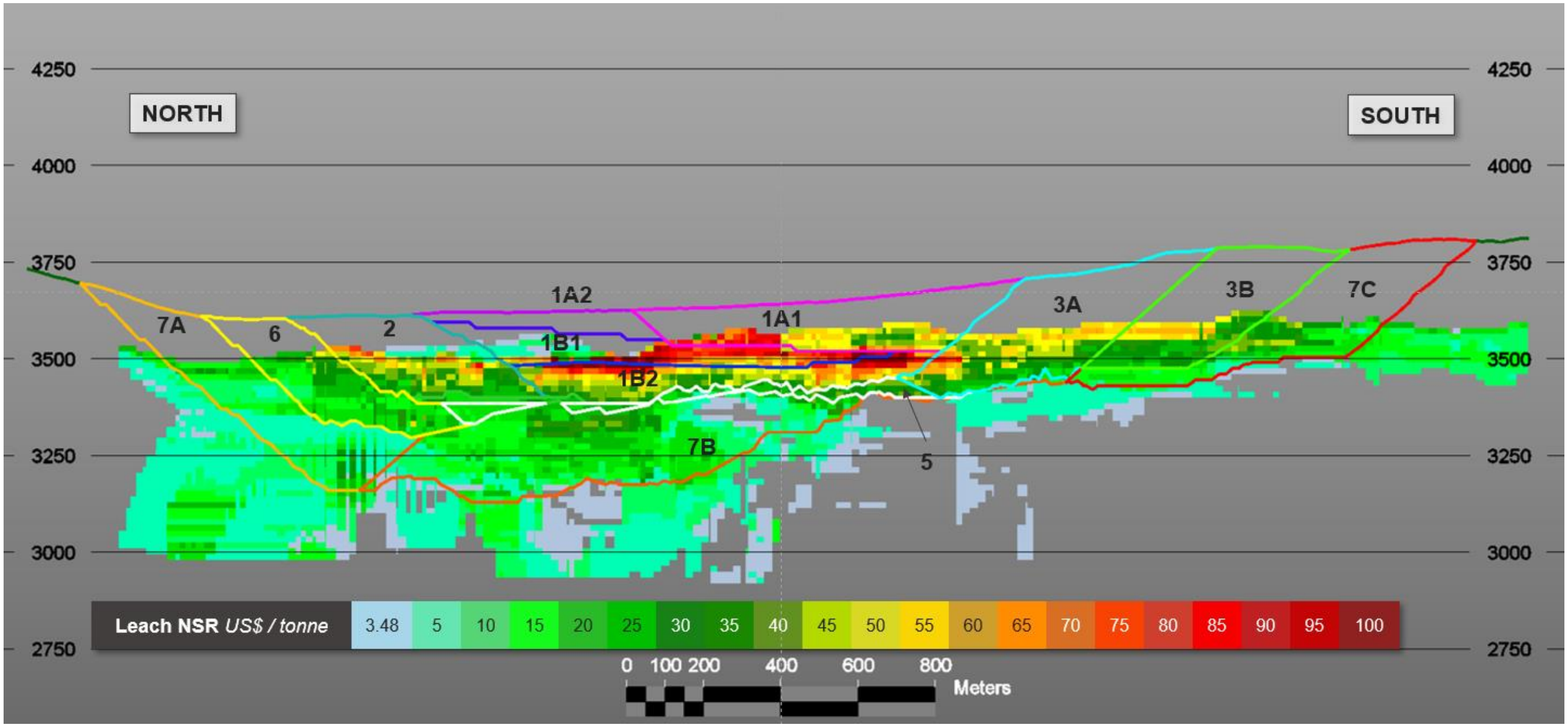


Figure 16.16: Cross Section 1 with Pit Phasing and Leach Net Smelter Return Values

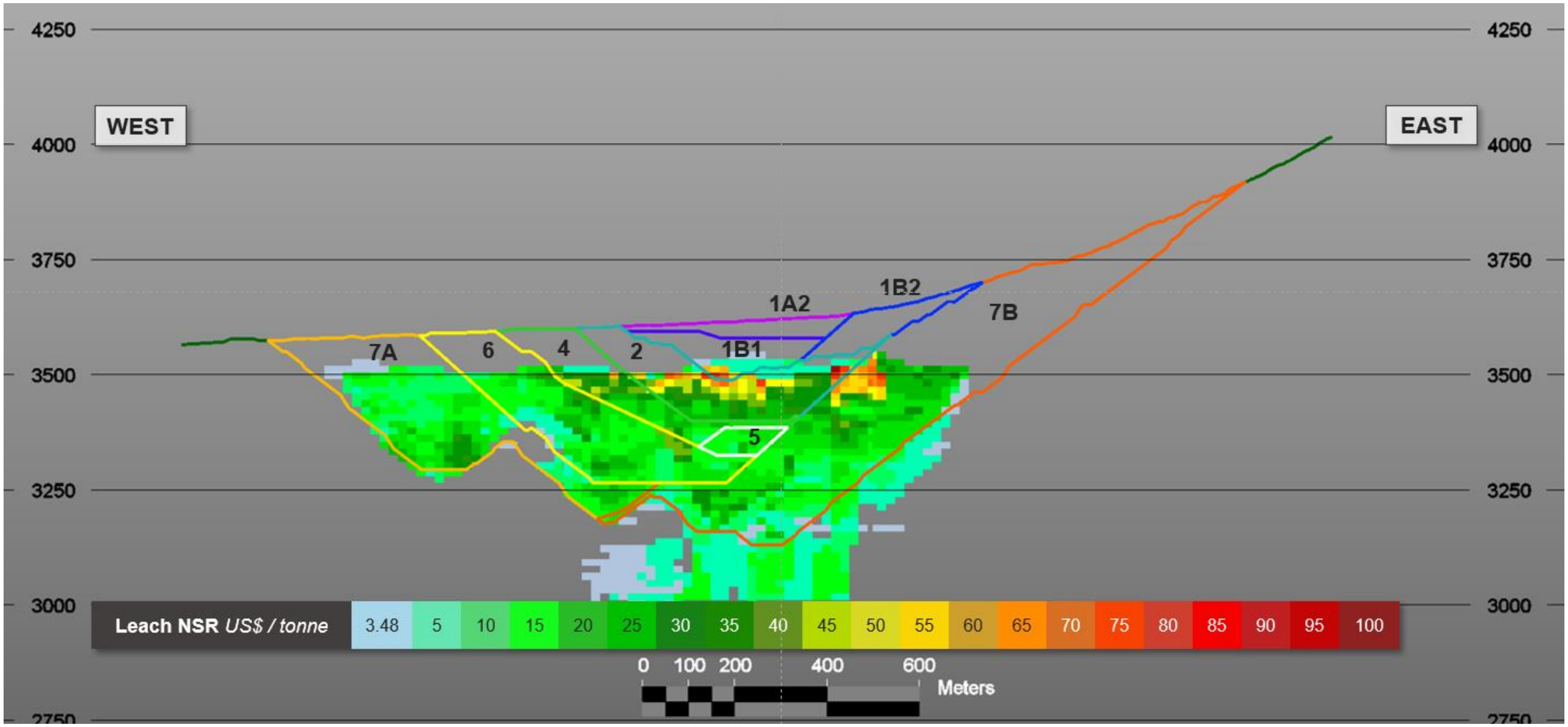


Figure 16.17: Cross Section 2 with Pit Phasing & Leach NSR Values.

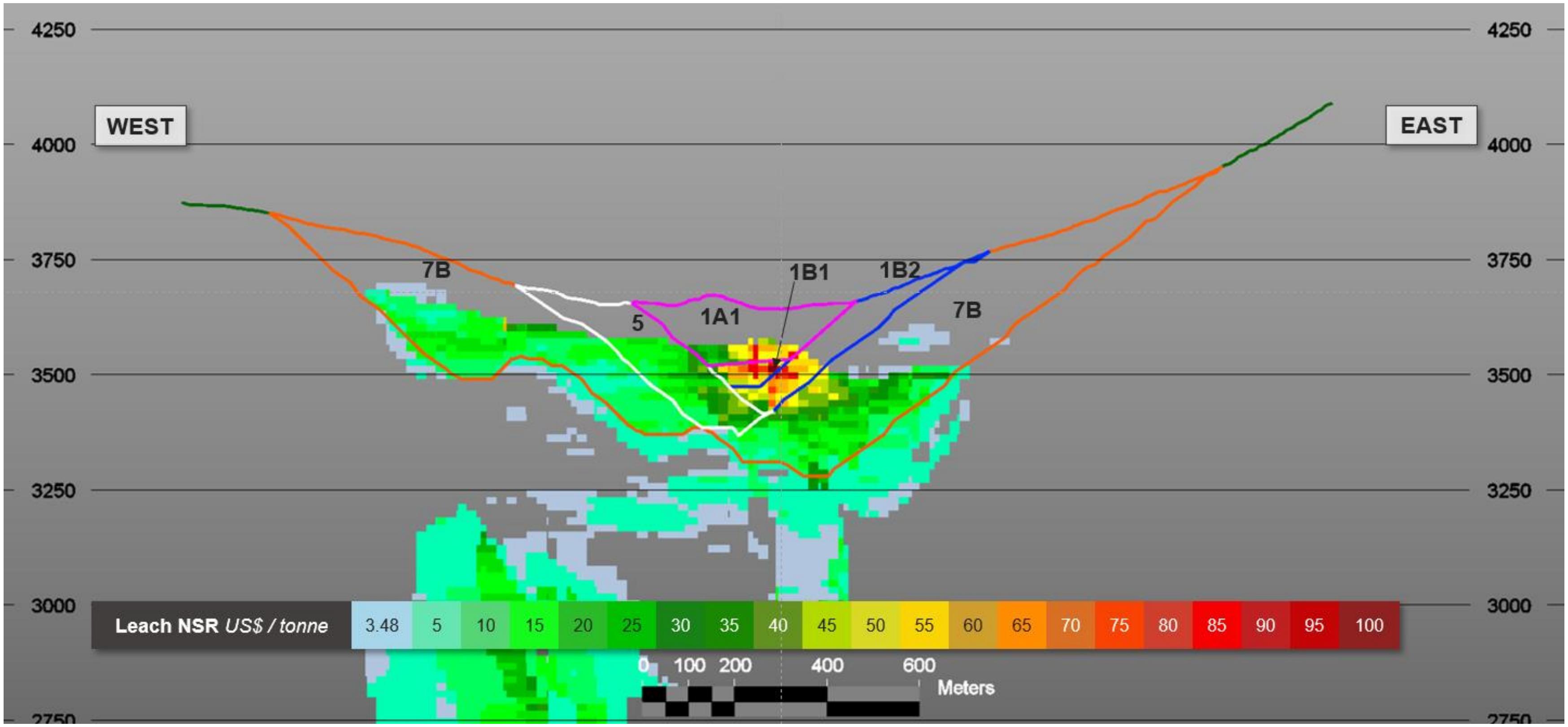


Figure 16.18: Cross Section 3 with Pit Phasing & Leach NSR Values

Table 16.7: Selected Pit Quantities by Phase for the 125 ktpa Case										
			Waste Material		Leachable Material					
Mining Phase	Total	Strip Ratio	Waste	Potential Mill Material[1]	Tonnage	Copper Grade	Soluble Copper Grade	Gold Grade	Silver Grade	NSR
-	*Mt*	*(Waste t / Leach t)*	*Mt*	*Mt*	*Mt*	*%*	*%*	*g/t*	*g/t*	*US$/t*
1A1	85.0	4.0	64.4	3.8	16.9	0.69	0.61	0.06	1.29	49.53
1A2	34.4	60.9	32.7	1.1	0.6	0.55	0.49	0.06	1.66	39.58
1B1	55.3	0.5	13.7	3.4	38.1	0.76	0.63	0.07	1.54	51.06
1B2	97.5	1.3	50.3	3.9	43.3	0.92	0.75	0.08	1.51	60.50
2	86.8	1.4	47.1	3.8	35.9	0.63	0.48	0.07	1.78	38.59
3A	110.4	2.1	72.2	2.7	35.5	0.62	0.50	0.06	1.03	40.37
3B	85.7	2.3	46.3	13.8	25.6	0.63	0.51	0.09	1.21	41.22
4	50.8	0.8	19.0	2.9	28.9	0.54	0.38	0.04	1.46	30.52
5	125.8	0.4	33.3	1.3	91.2	0.41	0.29	0.04	1.05	23.08
6	187.7	0.8	79.0	2.7	105.9	0.43	0.28	0.04	1.37	22.91
7A	388.3	0.9	169.3	18.4	200.6	0.32	0.20	0.03	1.07	15.97
7B	978.3	1.0	448.9	52.1	477.3	0.41	0.25	0.05	1.27	19.95
7C	262.6	2.2	160.7	19.4	82.5	0.49	0.34	0.06	1.07	27.55
Total	**2,548.6**	**1.16**	**1,237**	**129.3**	**1,182.4**	**0.46**	**0.31**	**0.05**	**1.25**	**25.13**

[1]Average in situ grade of potential mill material: Total Copper (0.107%), Gold (0.069 g/t), Silver (0.910 g/t). Determined with mill NSR cutoff values of $5.46/t for supergene material, and $5.43/t for primary material. Due to rounding, values may not sum.

16.4 LOS AZULES MINE PRODUCTION SCHEDULE

16.4.1 Base Case – 175 ktpa Copper Cathode Case

The Los Azules mine schedule was created using the Hexagon MinePlan Schedule Optimizer. The pit phases described in section 16.3 were split into 15 m benches and then imported into the schedule optimizer. The schedule was set up to maximize NSR while allowing for access. Other constraints are listed below.

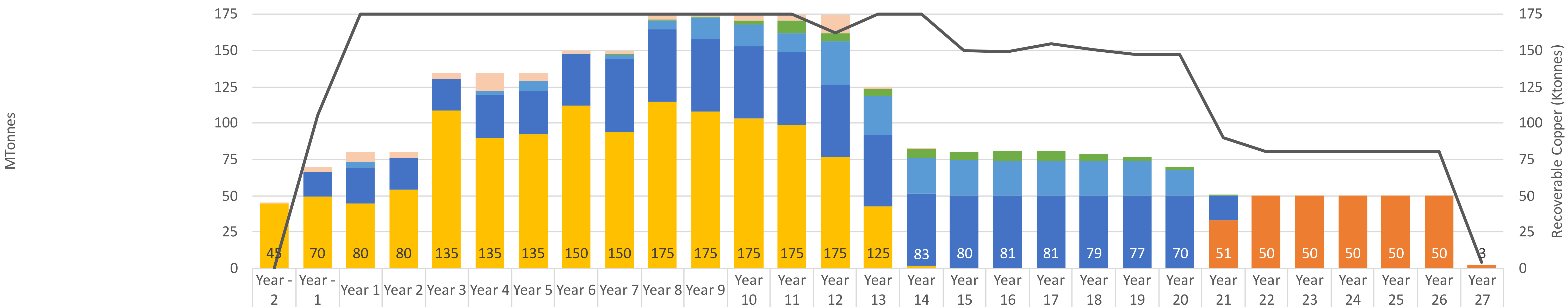
- A maximum of 12 benches per phase per year (VAR limit).
- Maximum crusher throughput of 18Mt/yr (year -1).
- Maximum crusher throughput of 25Mt/yr (year 1 to 3).
- Maximum crusher throughput of 30-35 Mt/yr (year 4 to 6).
- Maximum crusher throughput of 50 Mt/yr (year 7 to end of mine life).
- A recoverable copper limit of 175 kt/yr.

A stockpiling strategy will be implemented based on mining capacity, crusher capacity, and copper cathode production. In years where direct feed requires higher grade material to produce 175 kt, lower grade material is stockpiled to release higher grade material. Stockpiled material remaining at the end of mining will be reclaimed after mining has been completed in the pit.

The mine schedule does not include a mill processing stream, as additional work is required to determine the feasibility of a mill. Potentially millable material is segregated in the stockpile area to keep it available should a mill be developed in the future.

Based on the mining phase sequence and the constraints listed above, the 175 kt cathode production schedule is shown in Figure 16.19.

Los Azules Production Schedule: Higher Mining Rate 175 ktpa Case



	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27
To Mill Supergene Stk (Mtonnes)	0.4	3.5	6.9	3.9	4.1	12.8	5.7	2.2	2.6	3.7	1.2	4.3	4.4	13.3	1.4	0.2	-	-	-	-	-	-	-	-	-	-	-	-	-
To Mill Primary Stk (Mtonnes)	-	-	-	-	-	0.0	0.0	0.0	0.0	0.4	0.9	2.3	8.7	5.1	4.9	6.1	5.9	6.9	7.1	5.0	3.0	1.8	0.6	-	-	-	-	-	-
To Leach Stk (Mtonnes)	-	-	3.6	-	-	2.3	7.0	0.4	3.2	6.1	15.0	15.0	13.1	30.0	27.0	25.1	24.0	23.9	24.0	24.0	23.9	18.2	-	-	-	-	-	-	-
Leach - From Pit (Mtonnes)	-	16.9	25.0	22.2	22.1	30.0	30.0	34.9	50.0	50.0	50.0	50.0	50.0	50.0	49.1	49.6	50.0	50.0	50.0	50.0	50.0	50.0	16.9	-	-	-	-	-	-
Leach - From Stk (Mtonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	33.1	50.0	50.0	50.0	50.0	50.0	2.7
Waste (Mtonnes)	44.6	49.7	44.5	53.9	108.8	89.9	92.4	112.5	94.1	114.8	107.9	103.4	98.8	76.6	42.8	1.7	0.5	0.1	0.0	-	-	-	-	-	-	-	-	-	-
Recoverable Copper (Ktonnes)	-	105	175	175	175	175	175	175	175	175	175	175	175	163	175	175	150	149	155	151	147	147	90	80	80	80	80	80	4
Crusher Throughput (Mtonnes)	-	17	25	22	22	30	30	35	50	50	50	50	50	50	49	50	50	50	50	50	50	50	50	50	50	50	50	50	3
Max Bench Advancement Rate	8	7	10	12	7	12	12	6	12	9	6	6	5	7	9	5	3	4	4	3	11	7	7	-	-	-	-	-	-

[1]Waste: Potential Mill material includes supergene and primary material above a mill NSR cutoff of $5.46/t and $5.43/t, respectively.
Note: Recoverable copper values are reported at the time of mining and do not include leach recovery timelines. Refer to Section 13 for actual annual copper cathode production.

Figure 16.19: Los Azules Base Case Mine Production Schedule

As shown in Figure 16.19, pre-stripping begins in year -2, lasting for two years. In year -1, leachable material (16.9 Mt) is crushed and stockpiled on the pad to be ready for the leaching process in the following year. Mine production for direct feed to the leaching operation begins in year 1 at a maximum of 25 Mt per year until end of year 3. Mine production increases to a maximum of 30 to 35 Mt of leachable material between years 4 and 6. The final expansion of leachable material production starts in year 7 going up to 50 Mt per year.

Recoverable copper cathode production starts in year -1 with the mine delivering 105ktpa of recoverable copper. From year 1 to 14 copper production from the mine produces approximately175ktpa of recoverable copper. From years 15 to 20 recoverable copper cathode mined reduces to approximately 150ktpa. The pit reaches the end of development in year 20. After year 20, only the low-grade stockpile material is reclaimed, resulting in a drop of recoverable copper production to 80 kt by year 26 (end of mine life). Recoverable copper is the amount of copper that could be recovered from the leaching process but has not been leached yet. Please see Section 13 for the actual amount of copper leached and cathode produced on an annual basis.

16.4.2 End of Period Maps

Following is a series of period plans depicting the progression of the mine over time. End of period maps include annual periods up to year 5, then in five-year increments up to year 20. Years 21 to 27 (end of mine life) is when the low-grade stockpile is rehandled, and the material is processed.

Mine Site Plan

Non-mineralized material from the PEA pit is hauled to either the North MRSF or the South MRSF. Mineralized material is hauled to either the Leach Stockpile or directly to the crusher for processing. The North MRSF has approximately 835 Mt of storage while the South MRSF has approximately 530 Mt of material storage. The South MRSF is the primary destination for non-mineralized material in the initial ~5 years of mining. The remaining non-mineralized material is hauled to the North MRSF. A small amount of mineralized material is sent to the Leach Stockpile starting in year 1 (3.6 Mt), with more significant amounts of material being sent in years 9 through 19.

Figure 16.20: Mine Site Plan

Year -2

Mining commences in phase 1A1 down to elevation El. 3610 m. Pre-stripping activity involves only waste material (45 Mt) being placed at the south and north MRSF at El. 3860 m and El. 3600 m, respectively. End-of-period year -2 is illustrated in Figure 16.21.

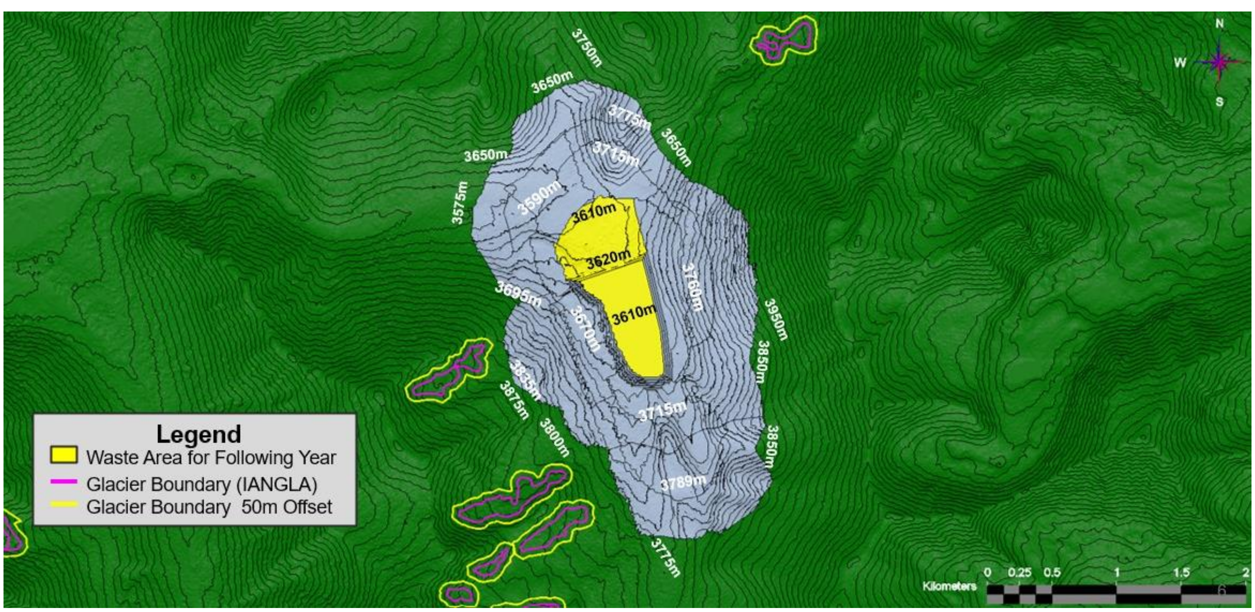


Figure 16.21: End of Period Year -2

Year -1

Mining activity continues in phase 1A1 (completed) and 1A2 to El. 3520 m and El. 3565 m, respectively with 16.9 Mt of leachable material sent to the crusher and 49.7 Mt of waste material is placed in the south at El. 3890 m. Access is established to the leach stockpile, phase 1B2 (El. 3610 m), phase 2 (El. 3700 m), and phase 3A (El. 3775 m) in year -1 in preparation for use in year 1. One front-end loader (bucket capacity ≈25 m^3) and 2 hydraulic shovels (bucket capacity ≈42 m^3) are used in this period. End of period year -1 is illustrated in Figure 16.22.

Figure 16.22: End of Period Year -1

Year 1

Phase 1A2 is mined to completion at El. 3350 m. Phases 1B1, 1B2,2A, 2B, and 3A are mined down to El. 3490 m, El. 3670 m, El. 3685 m, 3700 m, 3685 m, and 3745 m respectively. 25 Mt of leachable material is sent to the crusher and 3.6 Mt of low-grade leachable material is stockpiled to El. 3610 m. 44.5 Mt of waste material is placed in the at El. 3890 m. Access is developed to phases 3B (El. 3775 m), and 4 (El. 3745 m) in preparation for mining activity in year 2. One front-end loader (bucket capacity ≈25 m^3) and 2 hydraulic shovels (bucket capacity ≈42 m^3) are used in this period. Pit configuration at the end of period year 1 is illustrated in Figure 16.23.

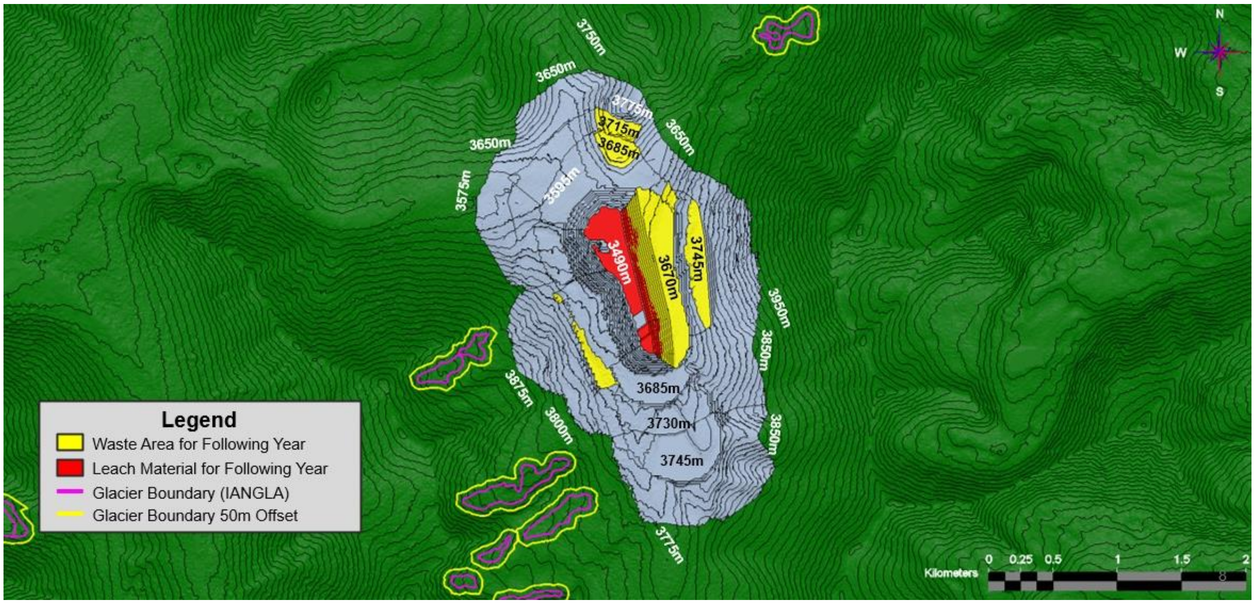


Figure 16.23: End of Period Year 1

Year 2

Most of the material being mined in year 2 comes from Phase 1B1down to El. 3490 m. Phases 1B2, 2, 3A, 3B and 6 are mined to El. 3505 m, El. 3670 m, El. 3685 m, El. 3715 m, and El. 3655 m, respectively. 22.2 Mt of leachable material is sent to the crusher. 54 Mt of waste is placed in the south at El. 3920 m. Access is established to Phase 4 at El. 3715 m in preparation for mining activity in year 3. One front-end loader (bucket capacity ≈25 m^3) and 2 hydraulic shovels (bucket capacity ≈42 m^3) are used in this period. With a large electric rope shovel (bucket capacity ≈62 m^3) brought on in this period as well. End of period year 2 is illustrated in Figure 16.24.

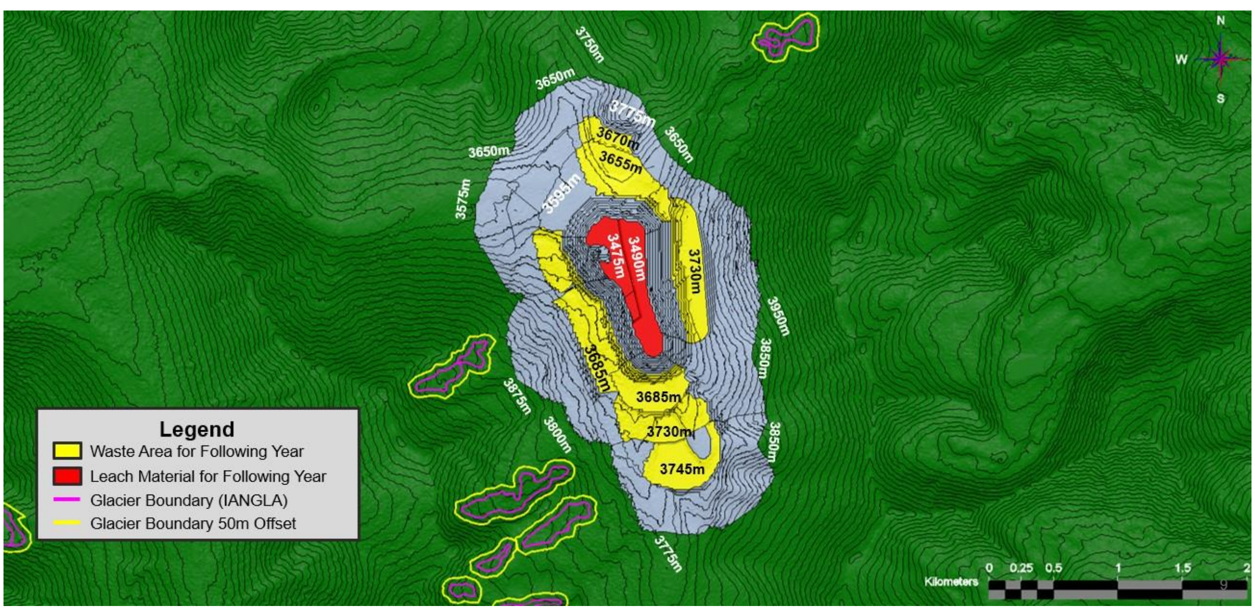


Figure 16.24: End of Period Year 2

Year 3

Phases 1B1 and 1B2 are mined to El. 3445 m. Phases 2 and 4 are mined to El. 3595 m and El. 3655 m, respectively. Phases 3A and 3B are mined to El. 3625 m and El. 3610 m, respectively. Phase 4 is mined to El. 3625m. 22.2 Mt of leachable material is sent to the crusher. 108.8 Mt of waste is placed in the south at El. 3950 m. One front-end loader (bucket capacity ≈25 m³) and 2 hydraulic shovels (bucket capacity ≈42 m³) are used in this period. End of period year 3 is illustrated in Figure 16.25.

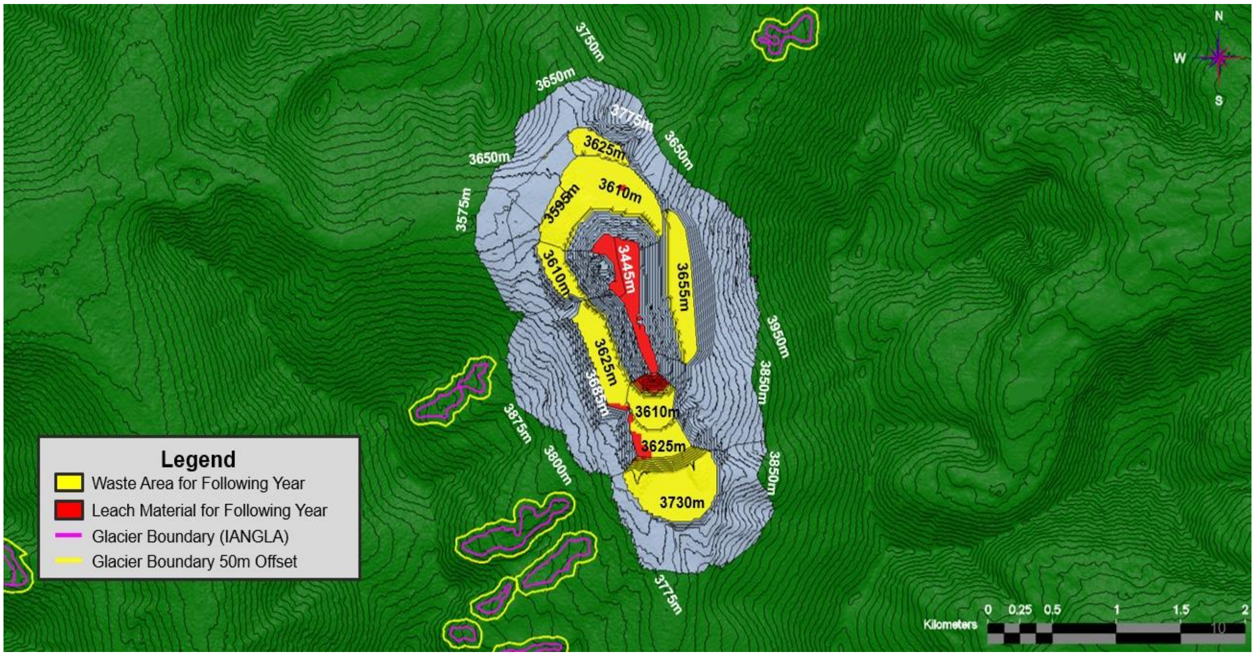


Figure 16.25: End-of-Period Year 3

Year 4

Mining in several phases continues to ensure an adequate release of mineralized material throughout the mine life. Phase 1B1 is completed at El. 3430 m and phase 1B2 is mined to El. 3430 m. Phases 2A, 2B, and 3A are mined to El. 3580 m, El. 3445 m, El. 3565 m, respectively. Phases 4A, 4B, and 4C are mined down to El. 3610 m, El. 3625 m, El. 3595 m. 30.0 Mt of leachable material is sent to the crusher. 89.9 Mt of waste material is placed in the south MRSF to at El. 3950 m. Access is established to phase 5A and phase 5B at El. 3775 m and El. 3955 m in preparation for mining activity in year 5. Two front-end loaders (bucket capacity ≈25 m³), two hydraulic shovels (bucket capacity ≈42 m³), and an electric rope shovel (bucket capacity ≈62 m³) are used in this period. The pit configuration at the end of period year 4 is illustrated in Figure 16.26.

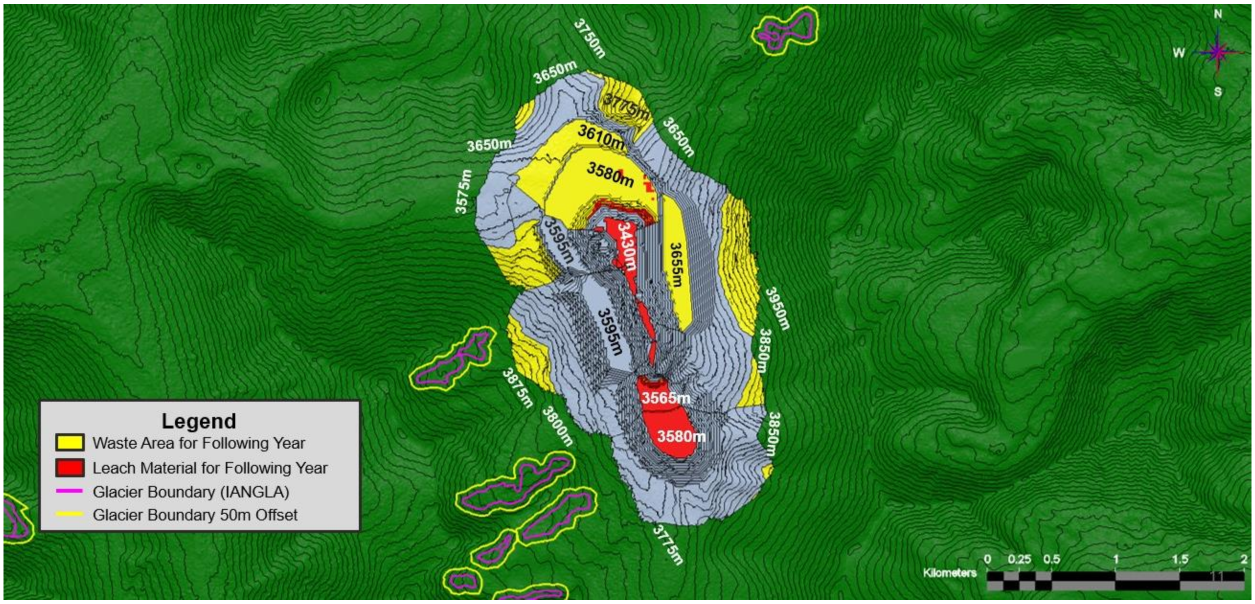


Figure 16.26: End-of-Period Year 4

Year 5

Phase 1B2 continues to be mined down to El. 3400 m. Phase 2A is mined down to El. 3490 m, phase 3A is mined to 3520m. Phases 4A and 4B are mined to El. 3580 m and El. 3610 m, respectively. While phases 5A and 5B are mined to El. 3610 m, and El. 3790 m, respectively. 30.0 Mt of leachable material is sent to the crusher. 92.4 Mt of waste is placed on the south and north MRSF at El.-+ 3950 m and El. 3660 m, respectively. An additional hydraulic shovel (bucket capacity ≈42 m^3) is added to the fleet in period 5. This brings the loading fleet to a total of; Two front-end loaders (bucket capacity ≈25 m^3), three hydraulic shovels (bucket capacity ≈42 m^3), and an electric rope shovel (bucket capacity ≈62 m^3) are used in this period. The pit configuration at the end of year 5 is illustrated in Figure 16.27.

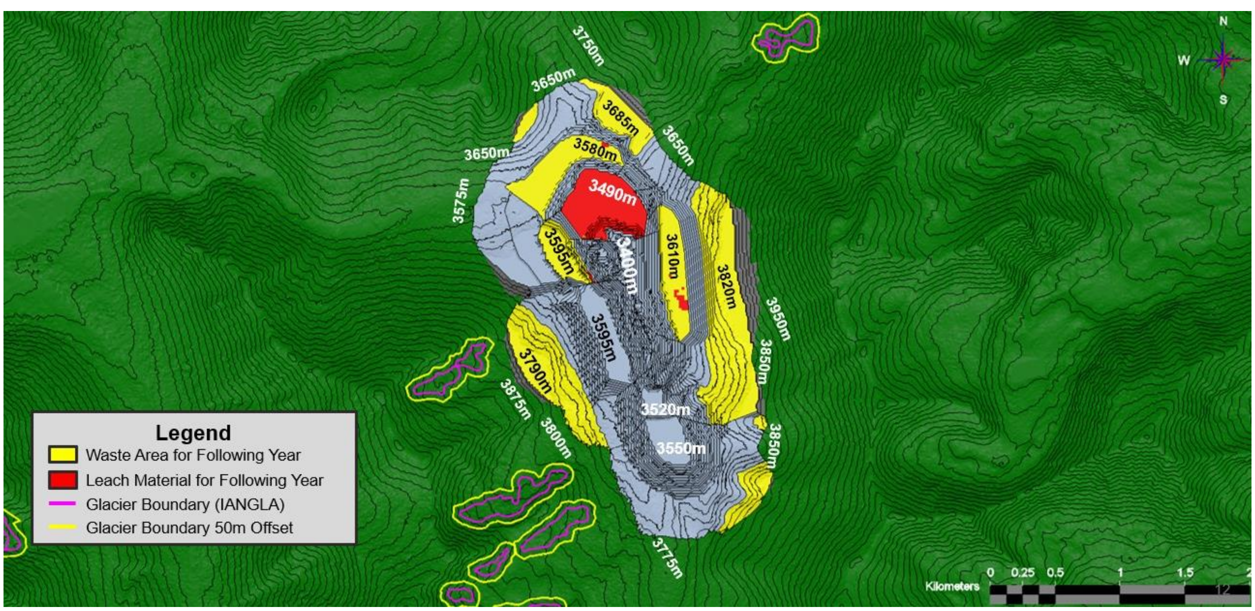


Figure 16.27: End of Period Year 5

Year 6 to 10

Mining continues is the outer phases as the mine life approaches the halfway point. Phases 1B2, 2A, and 2B are completed at El. 3385 m, El. 3325 m, and El. 3415 m, respectively. Phases 3A and 3B are mined to El. 3460 m. Phases 4A and 4B are mined to El. 3400 m, while phases 4C is mined to El. 3430 m. Phases 5A, 5B, and 5C are mined to El. 3490 m, El. 3565 m, El. 3760 m, respectively. In the 5-year period, a total 234.9 Mt of leachable material is sent to the crusher and 39.7 Mt of low-grade leachable material is stockpiled. 532.7 Mt of waste is placed on the south and north MRSF to El. 4010 m and El. 3750 m, respectively. Two front-end loaders (bucket capacity ≈25 m^3), three hydraulic shovels (bucket capacity ≈42 m^3), and an electric rope shovel (bucket capacity ≈62 m^3) are used in this period. The pit configuration at the end of year 10 is illustrated in Figure 16.28.

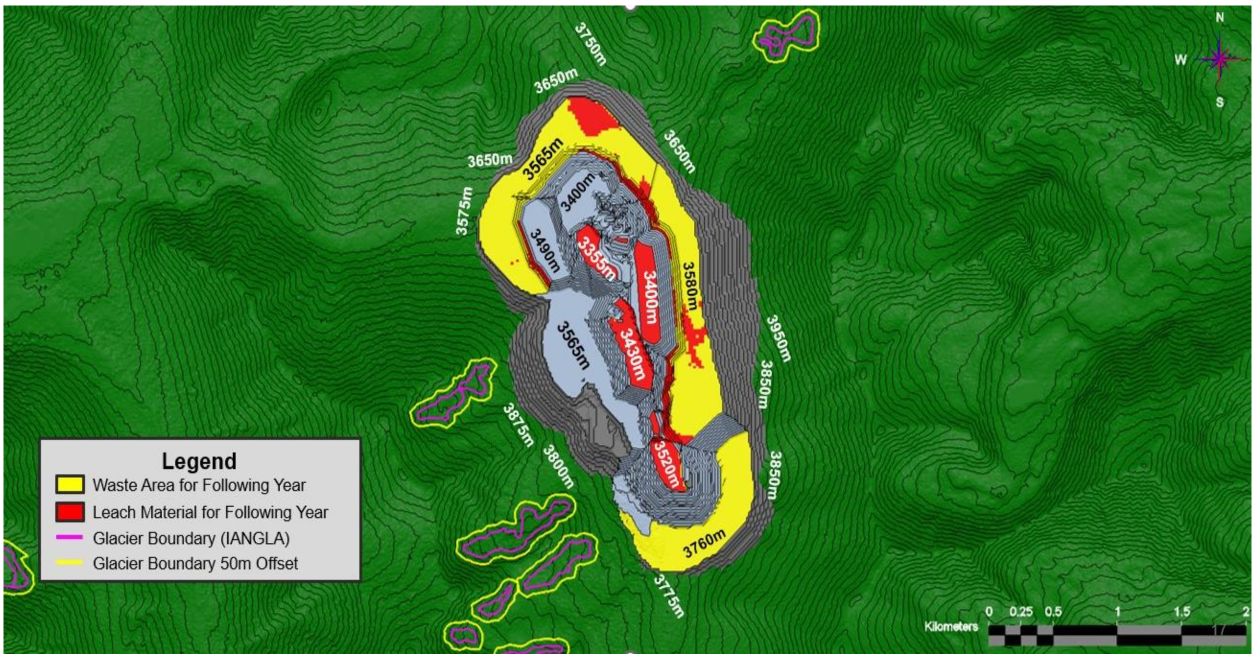


Figure 16.28: End of Period Year 10

Year 11 to 15

Mining continues as all sides of the pit are brought down to maintain access and allow for productive headings. Most of the waste material has been mined out by the end of year 15 with production failing from 175 Mt in year 11 to approximately 80 Mt in year 15. Phases 3A,3B, 4B and 4C are completed at El. 3445 m, El. 3280 m, El. 3370 m, and El. 3370 m, respectively. Phases 5A, 5B, and 5C are mined to El. 3415m, El. 3430 m, and El. 3445 m, respectively. In the five-year period, a total 248.7 Mt of leachable material is sent to the crusher and 119.2 Mt of leachable material is stockpiled. 3220.3 Mt of waste is placed on the south and north MRSF. One front-end loader (bucket capacity ≈25 m³), one hydraulic shovel (bucket capacity ≈42 m³), and an electric rope shovel (bucket capacity ≈62 m³) are used by the end of this period. The end of period pit position for year 15 is illustrated in Figure 16.29.

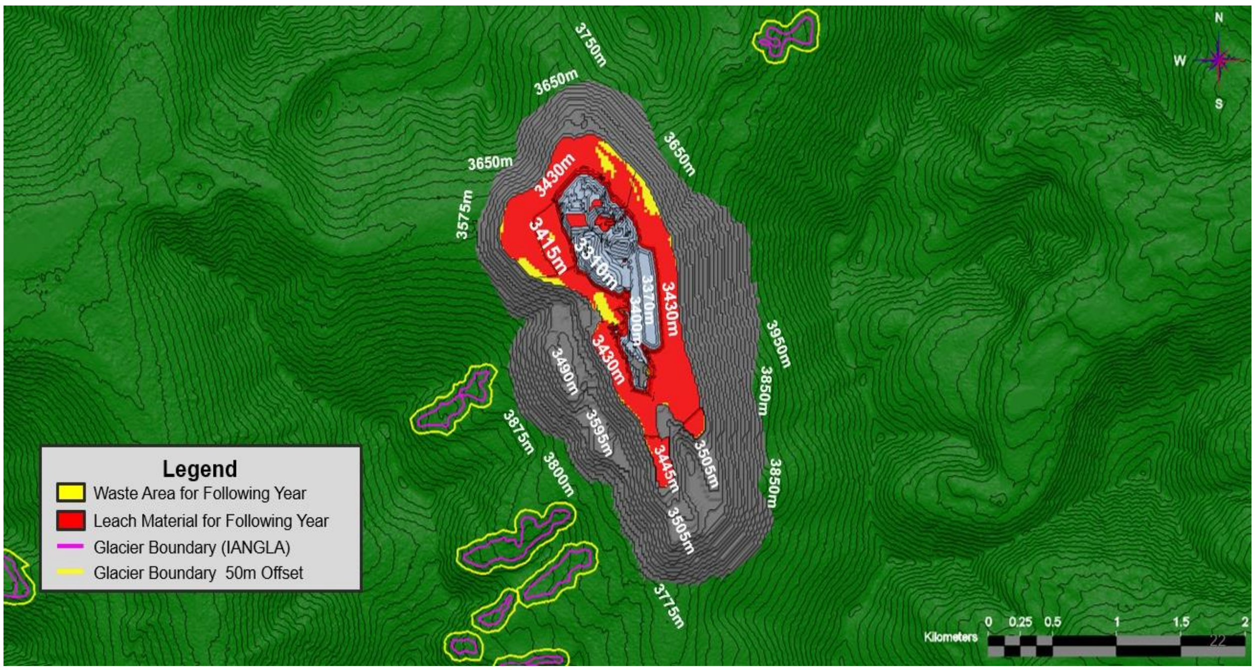


Figure 16.29: End of Period Year 15

Year 16 to 20

During this period mining of the PEA pit is almost completed, with a small amount of primarily mineralized material remaining in phase 5B and 5C. Phases 4A and 5A are completed at El. 3280 m and El. 3160 m. Phases 5B and 5C are mined to El. 3205 m and El. 3385 m, respectively. In the five-year period, a total 250.0 Mt of leachable material is sent to the crusher and 113.9 Mt of leachable material is stockpiled.0.06 Mt of waste is placed on the south (completed to El. 4310 m)) and north MRSF at El. 4840 m. One hydraulic shovel (bucket capacity ≈42 m^3) and an electric rope shovel (bucket capacity ≈62 m^3) are used by the end of this period. The end of period pit configuration for year 20 is illustrated in Figure 16.30.

Figure 16.30: End of Period Year 20

Year 21 to 27 – End of Mining

Mining of the PEA pit is completed by year 21 with the last of the remaining material mined from phases 5B and 5C, the pit bottom is at El. 3130 m. A total 50.0 Mt of leachable material is sent to the crusher with 16.6 Mt coming from the pit and 33.1 Mt coming from the low-grade stockpile in year 21. There is no waste mined during this period and both the south and north MRSF are completed. From years 22 to 27 material is reclaimed from the low-grade stockpile at a rate of 50 Mt per year. One front-end loader (bucket capacity ≈25 m^3 and one hydraulic shovel (bucket capacity ≈42 m^3) are used by the end of this period. The end of mining configuration for year 27 is illustrated in Figure 16.31. This shows the MRSF to full design, along with the pit being mined out and the low-grade stockpile being reclaimed.

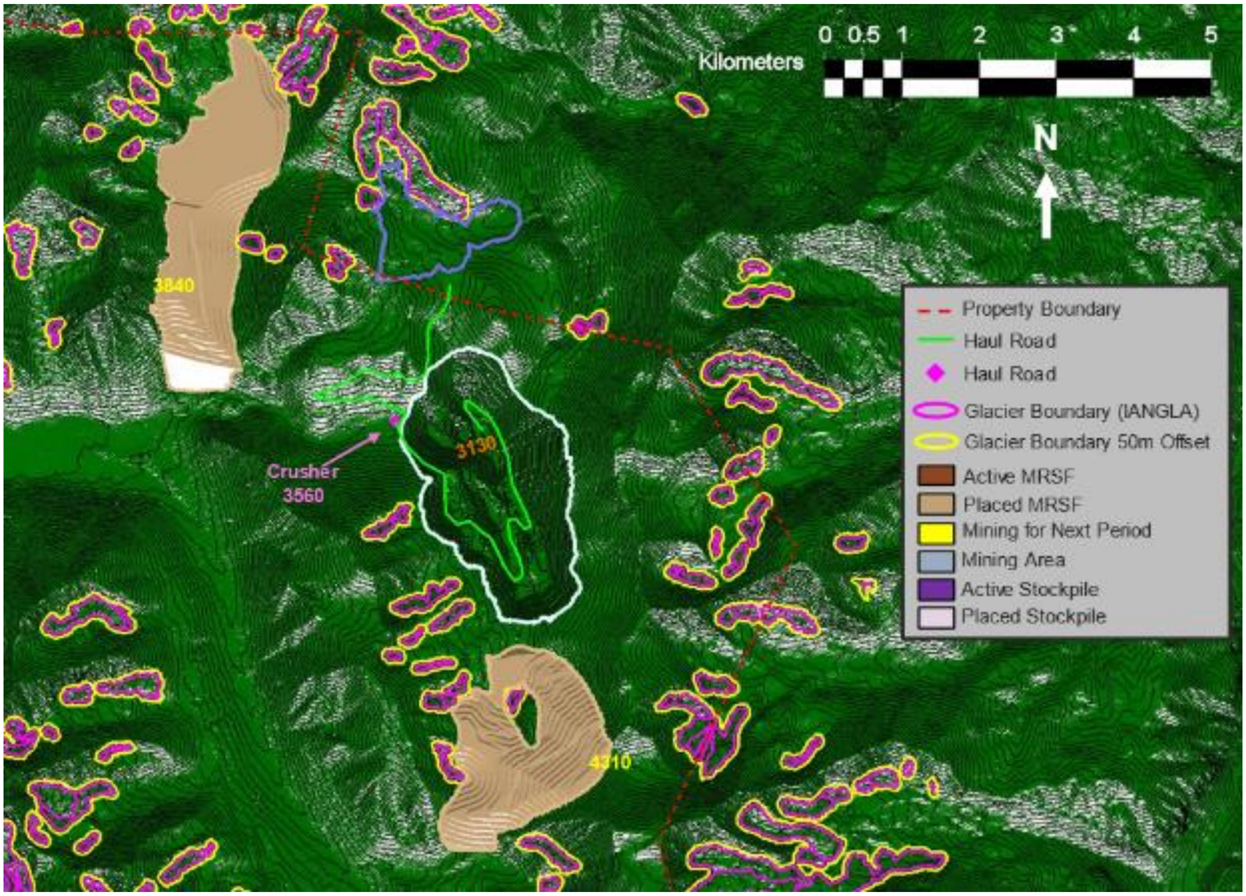


Figure 16.31: End of Period Year 27

16.4.3 Alternative Case – 125 ktpa Copper Cathode Case

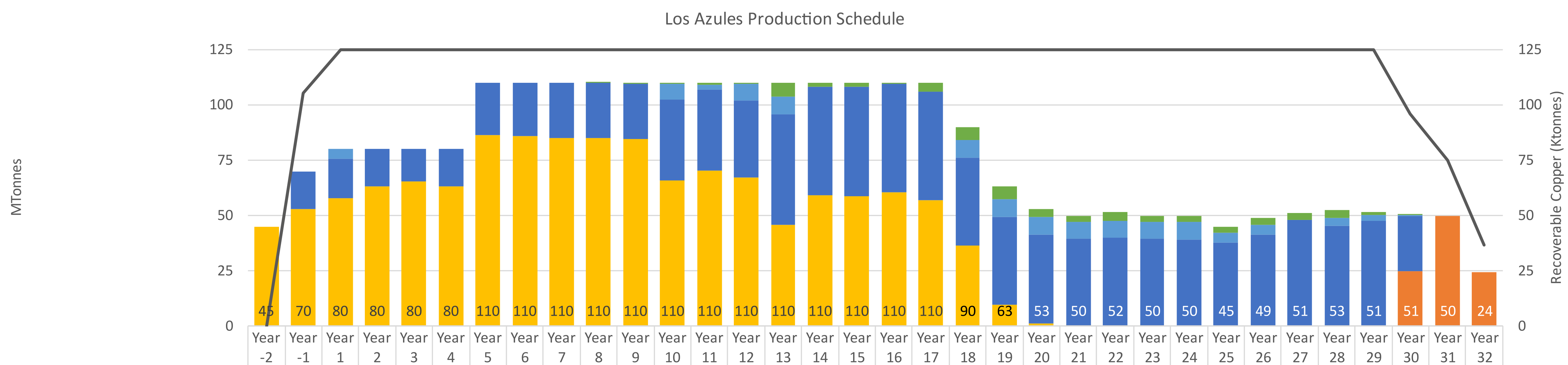
An alternative production case was completed to show the potential opportunity for mining to the same ultimate pit limit at a lower rate (with lower capital requirements). Phases used in this alternative 125 ktpa copper cathode case are shown in Section 16.3. Phases were also split into 15 m benches and then imported into Hexagon MinePlan Schedule Optimizer. The schedule was set up to maximize NSR while allowing for access. Additional constraints are listed below.

- A maximum of 12 benches per phase per year (VAR limit).
- Maximum crusher throughput of 20Mt/yr (year -1 to 3).
- Maximum crusher throughput of 25 Mt/yr (year 5 to 9.
- Maximum crusher throughput of 35 Mt/yr (year 10 to 12).
- Maximum crusher throughput of 50 Mt/yr (year 13 to end of mine life.
- A recoverable copper limit of 125 kt/yr.

A stockpiling strategy is implemented based on mining capacity, crusher capacity, and recoverable copper production. In years where direct feed requires higher grade material to produce 125 kt, lower-grade material is stockpiled to release higher grade material. Stockpiled material will be reclaimed after mining has been completed in the pit.

The mine schedule does not include a mill processing stream, future work is required to determine the feasibility of a mill. Potentially millable material is segregated within the waste dump area to allow for future recovery if a mill is constructed.

The resulting production schedule is shown in Figure 16.32.



	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14
To Primary Stk (Mtonnes)	-	-	-	-	-	-	-	-	-	0.0	0.1	0.3	0.7	0.3	6.2	1.6
To Leach Stk (Mtonnes)	-	-	4.1	-	-	-	-	-	-	-	-	7.0	2.1	7.6	7.9	-
Leach - From Pit (Mtonnes)	-	16.9	18.0	16.7	14.6	16.7	23.5	23.9	25.0	25.0	25.0	37.0	36.9	35.0	50.0	49.0
Leach - From Stk (Mtonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste (Mtonnes)	45.0	53.1	57.9	63.3	65.4	63.3	86.5	86.1	85.0	85.0	84.9	65.7	70.3	67.1	45.9	59.3
Recoverable Copper (Ktonnes)	-	105	125	125	125	125	125	125	125	125	125	125	125	125	125	125
Crusher Throughput (Mtonnes)	-	17	18	17	15	17	24	24	25	25	25	37	37	35	50	49
Max Bench Advancement Rate	8	7	12	9	6	9	6	9	9	4	5	7	5	5	6	4

	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32
To Primary Stk (Mtonnes)	1.6	0.5	4.1	5.9	5.7	4.0	2.9	4.3	2.9	2.8	2.9	3.0	3.4	3.6	1.3	0.9	-	-
To Leach Stk (Mtonnes)	-	-	-	7.9	8.0	8.0	7.8	7.3	7.6	8.0	4.3	4.9	-	3.9	2.6	0.0	-	-
Leach - From Pit (Mtonnes)	49.8	49.2	49.0	39.8	39.8	40.0	39.1	40.0	39.5	39.2	37.8	41.1	47.8	45.2	47.4	25.3	-	-
Leach - From Stk (Mtonnes)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	24.7	50.0	24.4
Waste (Mtonnes)	58.6	60.3	56.9	36.5	9.7	1.2	0.2	0.1	0.0	-	-	0.0	-	-	-	-	-	-
Recoverable Copper (Ktonnes)	125	125	125	125	125	125	125	125	125	125	125	125	125	125	125	96	75	37
Crusher Throughput (Mtonnes)	50	49	49	40	40	40	39	40	40	39	38	41	48	45	47	50	50	24
Max Bench Advancement Rate	4	7	3	3	4	3	3	5	3	3	3	6	3	5	5	7	0	0

[1]Waste: Potential mill material includes supergene and primary material above a mill NSR cutoff of $5.46/t and $5.43/t, respectively.
Note: Recoverable copper values are reported at the time of mining and do not include leach recovery timelines. Refer to Section 13 for actual annual copper cathode production.

Figure 16.32: 125 ktpa Recoverable Copper

As shown in Figure 16.32, pre-stripping begins in year -2, lasting for two years. In year -1, leachable material (16.9 Mt) is crushed and stockpiled on the pad for the leaching process in the following year. Mine production for direct feed to the leaching operation begins in year 1 at a maximum of 18 Mt per year until end of year 4. Mine production increases to a maximum of 25 Mt of leachable material between years 5 and 9 and 35 Mt of leachable material between years 10 and 12. The final expansion of leachable material production starts in year 13 at 50 Mt per year lasting until the end of mine life in year 32.

Recoverable copper is the amount of copper that can be recovered from the leaching process but has not completed the full leaching cycle. Recoverable copper production starts in year -1 with the mine delivering 105 kt of recoverable copper. From years 1 to 29, recoverable copper production from the mine releases 125 ktpa. From years 30 to 32, the recoverable copper) is drawn from the low-grade stockpile producing between 96 ktpa to 37 ktpa of recoverable copper in the final year.

The schedule also aims to utilize the capacity of the selected loading equipment. In year -2, one front-end loader (bucket capacity ≈25m^3) and one hydraulic shovel (bucket capacity ≈42 m^3) are sufficient. Starting in year -1, an additional hydraulic shovel (bucket capacity ≈42 m^3) is added to the loading fleet. The loading fleet configuration remains the same until year 5, where a third hydraulic shovel is added (bucket capacity ≈42 m^3) for a peak total movement capacity of 110 Mt per year. The loading fleet starts to ramp down in year 19 to one hydraulic shovel and one front-end loader until the end of mine life in year 32. See Section 16.6 for more details.

16.5 MINE ROCK STORAGE FACILITIES

The mine plan requires mining and placement of 1.37 billion tonnes of waste mine rock. Two locations for establishing Mine Rock Storage Facilities (MRSFs) have been identified, the North MRSF and the South MRSF. In addition, a location for the storage of stockpiled low-grade material for the leaching process has been identified. Both the 125 ktpa case and the 175 ktpa case have the same MRSF layouts apart from the leach stockpile. Refer to Figure 16.33 for the MRSF layout of the 175 ktpa base case and Figure 16.34 for the 125 ktpa alternate case, respectively.

Figure 16.33: Location of Mine Rock Storage Facilities and Leach Stockpile for 175 ktpa Base Case

Figure 16.34: Location of Mine Rock Storage Facilities and Leach Stockpile for 125 ktpa Alternative Case

16.5.1 MRSF Configuration and Stability

Both the North and South MRSF follow the design parameters shown in Table 16.8.

Table 16.8: MRSF Parameters	
MRSF Parameters	
Lift Height (m)	30
Bench Width (m)	35
Slope Face Angle (°)	37
Road Width (m)	50
Maximum Road Grade (%)	10
Overall Slope (H:V)	2.5H:1V

The primary low-grade stockpile has an overall slope of 2.5H:1V.

Refer to Section 18.4 for more details.

16.6 MINING EQUIPMENT

To achieve the highest possible productivity while mining 15 m benches, the operation will be required to use large-scale surface mining equipment. For the 125 ktpa case, 305 t load capacity class trucks were selected and for the 175 ktpa case 363 t load capacity class trucks were selected as the optimal truck sizes. This was based on loading equipment size and fleet composition, required truck productivity, and controlling the overall number of trucks required.

Equipment and personnel requirements will be based on the mine operating 24 hours per day, 365 days per year giving a possible operating hour total of 8,760 hours. Stantec assumed typical allowances for maintenance, downtime, and stand-by to develop the equipment productive hours which were then adjusted for utilization to provide the estimates in Table 16.9.

Table 16.9: Annual Mine Equipment Productive Hours	
Annual Mine Equipment Productive Hours	
Haul Trucks	6,284
Shovels	6,073
Front-End Loaders	5,863
Support Equipment	5,242
Auxiliary Equipment	3,669
Other Equipment	2,621

The primary mining fleet consists of loading and hauling units, while support equipment consists of dozers, drills, and graders. The fleet size and composition were developed from first principles by building up the equipment productivities for the primary loading fleet. The MS-Haulage software package was used to estimate haul cycle times for the various mining phases to develop waste, mineralized material, and stockpile haulage times. The MS Haulage inputs used are shown in Table 16.10.

Table 16.10: MS Haulage Inputs	
Rolling Resistance Near Pit/Dump	2%
Rolling Resistance Typical Road	2%
Max Speed Typical	50 km/hr
Max Speed Switchback/Corners	10 km/hr
Max Speed near Pit/Dump	50 km/hr
Max Speed Downhill Loaded	20 km/hr
Max Speed Downhill Unloaded	20 km/hr

Haul cycle profiles were developed in MS Haulage and linked with the mining schedule to determine the most optimal routing of material. Cycle times calculated by MS Haulage were then linked to the mine production schedule to determine the truck requirements by mining period. The resulting fleet composition for the 125 ktpa and 175 ktpa mine schedules is shown in Figure 16.35 and Figure 16.36, respectively.

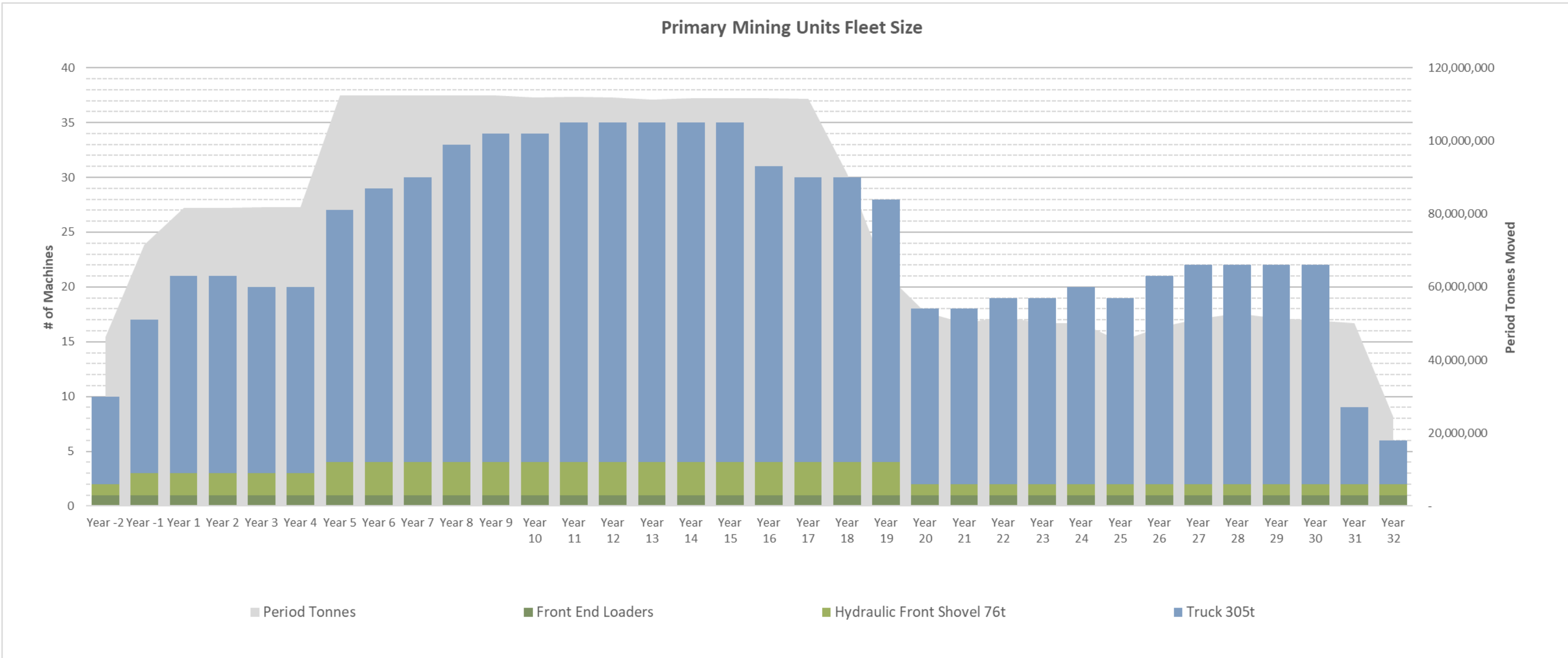


Figure 16.35: Fleet Size for 125 ktpa Alternative Case Primary Mining Fleet

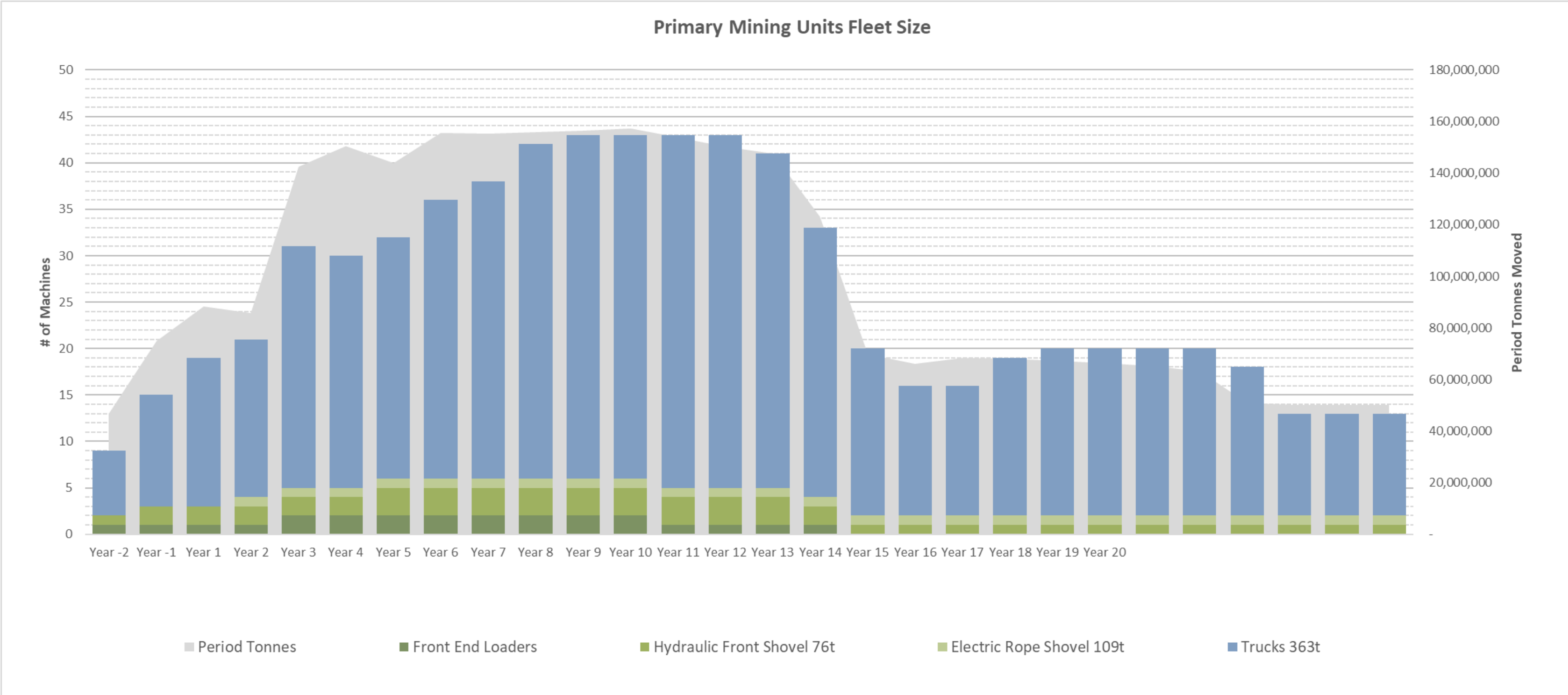


Figure 16.36: Fleet Size for 175 ktpa Base Case Primary Mining Fleet

The 125 ktpa case primary mining fleet requires a peak of 30 each 305-t class haul trucks, 1 each 35 m^3 front-end loader, and 3 each 40 m^3 hydraulic shovels. There is a ramp up of the fleet through the first 5 years and then a drop off once peak mining rates drop off after approximately year 17.

The 175 ktpa case primary mining fleet requires a peak of 37 each 363-t class haul trucks, 2 each 35 m^3 front-end loaders, 3 each 40 m^3 hydraulic shovels, and 1 each 61 m^3 rope shovel. There is a ramp up of the fleet through the first 5 years and then a drop off once peak mining rates drop off after approximately year 13.

16.6.1 Equipment Ownership

For the PEA, it is assumed that all mining operations, including pre-stripping phases, road work, site development, fuel and lube delivery, transportation of equipment, maintenance, and any other miscellaneous tasks will be completed with the mining equipment described in this section. It is also assumed that the equipment will be owned and operated by the owner. However, it should be noted that as the project develops, there is potential that contract mining would be a suitable option for special projects especially in accessing and developing the upper bench areas.

16.6.2 Trolley-Assist Strategy

To reduce operating costs and GHG emissions, the mine plan assumed a trolley-assisted haulage model. The life-of-mine (LOM) average operating cost (US$/t) using trolley assisted haulage is approximately $0.8/t, whereas a conventional haulage scenario would cost approximately $1.10/t. This translates to an estimated undiscounted LOM operating cost savings of approximately $800M. Furthermore, a trolley assisted haulage model cuts GHG emissions by about 2M t over the entire LOM. Assuming a carbon cost of $50/t, this GHG reduction provides an undiscounted LOM savings of $105M.

As a criterion for implementation of trolley infrastructure for a given haul road network was a 2-year minimum operational life. Therefore, the trolley systems were designed in areas where it could be installed for a minimum of 2 years on a sustained uphill grade. While trucks were on trolley assist, the obtainable uphill speed was increased to 22.5 km/hr. Over the LOM, 38% of the loaded distance will be under trolley assist.

16.6.3 Drilling and Blasting

Based on the bench height, limited rock strength data and first principles were used to determine pattern spacing and subsequent powder factor. It was assumed that 80% of the blasting is in hard rock and 20% is in weaker rock. The spacing used for hard rock is 7.5m by 8.5m resulting in a powder factor of 0.41 kg/m^3 and the spacing used for weaker rock is 9m by 10.5m resulting in a powder factor of 0.26 kg/m^3. Stantec modelled the primary production drilling fleet after Epiroc's Pit Viper 271 (PV271). The PV271 drill is commonly used in surface mines around the world and has a benchmark penetration rate of 0.5m/min in hard rock. It has the capability of drilling single pass holes up to 18 m deep ranging from 171 mm to 270 mm in diameter. During the final years of operation all material movement is rehandled from the stockpile and no drills will be required. Note that this section only focuses on drilling units used for production drilling and blasting operations. Geological or geotechnical drilling requirements were not considered.

16.6.4 Support Equipment

Support and other equipment used for auxiliary jobs was also specified based on the production schedule and therefore follows a similar profile. Like the primary equipment, the support equipment will be sized to support large-scale surface mining operations. There will be two dozer models on site, Caterpillar D10 and

Caterpillar D8 size machines, and they are planned to be used for MRSF operations, pit and road maintenance, and other miscellaneous jobs.

The support excavators are modelled after the Caterpillar 390. Excavators are primarily used for road maintenance, for digging sumps and ditches, and will therefore have a low demand.

There are two motor-grader models considered, the Caterpillar 24M and the Caterpillar 16M class machines. The motor-grader demand is a function of truck hours. The other equipment category consists of a wide variety of machines. It includes water trucks, maintenance trucks, crew pit trucks and buses, fuel and lube trucks, tire manipulators, mine rescue vehicles, light plants, generators, and other auxiliary equipment. The peak fleet size for each piece of equipment is shown in Table 16.11and Table 16.12.

Table 16.11: Peak Support Equipment 175 ktpa Case	
Equipment Type	**Maximum Number of units**
Drills	8
Dozers	12
Excavators	2
Graders	7
Other	103

Table 16.12: Peak Support Equipment 125 ktpa Case	
Equipment Type	**Maximum Number of units**
Drills	7
Dozers	11
Excavators	2
Graders	7
Other	89

16.7 MINE WORKFORCE

This section discusses the mining related labor required to develop and operate the Los Azules open pit, including all technical, operations, and maintenance personnel. Operations and maintenance labor and supervision has been allocated to cover the planned 24 hours per day, 365-work days year schedule. A 12-hour operating shift was assumed that requires two crews per day. The shift schedule is based on having four crews on a multi-week rotation. The mine workforce size was determined based on the production schedule presented in this PEA.

Table 16.13 shows the required mine labor for the 175 ktpa case from a pre-development year (year -2), a peak production year (Year 16) and the final year (year 27). Table 16.14 shows the required mine labor for the 125 ktpa case from a pre-development year (year -2), a peak production year (Year 16) and the final year (year 27).

Table 16.13: Mine Labor Complement 175 ktpa Base Case			
Positions	**Year -2**	**Peak Year 11**	**Year 22**
Mine Operations: Supervision & Labor	210	598	488
Mine Maintenance: Supervision & Labor	115	375	310
Technical Services	55	74	65
Total Mine Department	**380**	**1,074**	**863**

Table 16.14: Mine Labor Complement 125 ktpa Alternative Case			
Positions	**Year -2**	**Peak Year 13**	**Year 27**
Mine Operations Supervision & Labor	234	562	448
Mine Maintenance Supervision & Labor	125	348	298
Technical Services	55	74	65
Total Mine Department	**414**	**984**	**811**

16.8 HYDROGEOLOGY AND PIT DEWATERING

An initial hydrogeological investigation was completed in 2010 and a more extensive hydrological and hydrogeological investigation of the proposed pit area was completed in 2011 (Ausenco Vector, 2011). A total of 8 standpipe piezometers and 6 vibrating wire piezometers have been installed, 16 in situ permeability tests have been performed, and groundwater and surface water quality samples were collected and analyzed by an off-site laboratory. Fourteen additional in situ permeability tests were conducted in 9 exploration bore holes in 2018. These studies led to a conceptual understanding of the hydrology and hydrogeology around the proposed open pit.

During pit development, saturated overburden and Tertiary volcanics including porphyritic dacite, dacite, and rhyolite tuff will be encountered. The overburden includes thick deposits of glacial outwash and alluvial materials in the valleys and the northern sectors of the pit, where the thickness can be over 80 m. The permeability of these materials is very high, although the spatial extent, as defined by borehole drilling and a seismic investigation, are limited. Once groundwater is removed from storage the flows from the overburden would be limited to the rate from abutting materials and infiltration.

Groundwater flow in the volcanic bedrock is primarily controlled by ubiquitous fracturing of the porphyritic diorite and geologic structures in the area. The degree of fracturing of the porphyritic diorite and the permeability associated with the hydrothermal breccia and fault zones suggest that groundwater inflow to the pit will be high. Numerical groundwater flow modeling from the 2011 study suggested that during later stages of pit development the groundwater inflow to the pit will exceed 600 L/s. Updating these results based on the larger pit size and depth of the current PEA suggests groundwater inflows to the pit on the order of 800 L/s.

Given the shallow depth to groundwater and the high permeability of the geologic units in the pit area, high-capacity vertical dewatering wells both in-pit and outside the pit boundaries will be necessary. The in-pit wells will be used to remove groundwater occupying the pores and fractures in the rock mass within and surrounding the pit shell. These wells would be operated in advance of mining and are wells that would be consumed by the ultimate pit configuration. Likely, these wells will include both overburden and bedrock pumping wells. Overburden pumping could also be supplemented with pumping from shallow excavations (drains) in the areas where the depth to groundwater is shallow. Pit perimeter wells will also be needed to intercept water flowing to the pit from the surrounding groundwater system and to lower the water table behind the pit slopes. These wells will target primary groundwater flow paths and be targeted such that they intercept major fault zones (e.g., Piquenes fault south of the pit). A sump or series of sumps will also be used to pump water out from the pit bottom accumulating from pit wall runoff and/or groundwater inflow not captured by wells.

Prior to discharge of mine water, pit water would be used in the process plant, or if not, routed either to a sediment pond or rapid infiltration basin. Additional geochemical studies are necessary to evaluate the geochemical characteristics of the pit wall rocks and the potential for acid rock drainage, which could result in the need for treatment of in-pit waters. Most groundwater will be intercepted prior to seeping into the pit using wells. Initial water quality data suggest that this approach may permit discharge of these waters without treatment (e.g., pH, metals, sediment etc.).

Additional hydrogeologic data collected outside the area of mineralization and at greater depths will refine the long-term dewatering requirements and cost estimates. Long-term high-rate pumping tests should also be completed to determine the large-scale hydraulic properties of the geologic materials in the pit area and evaluate boundary conditions in the flow system that may exert strong controls over groundwater flow in the area.

17.0 RECOVERY METHODS

17.1 INTRODUCTION

This 2023 PEA incorporates an updated development strategy for the potential resource processing sources include Oxide/LIX, Supergene (copper sulfide enriched material) and Primary (sulfide material containing primary copper mineralization). LIX mineralization is the Spanish acronym for "leached" or leach cap. This updated development strategy includes two phases of development: Phase 1 considers mining and processing resources associated with the oxide and supergene copper mineralization in the near surface portion of the deposit using heap leaching methods. Two production rates (125 kt and 175 kt per annum of copper) are considered for Phase 1.

A Phase 2 of the project considers the continued development of the deposit's primary copper mineralization found beneath the supergene copper layer. The focus of this PEA is the initial Phase 1 project with limited concepts presented for Phase 2. For clarity, the economic outcomes for the development cases presented in this 2023 PEA include only Phase 1.

The Supergene and Oxide/LIX material is believed to be suitable for treatment in a conventional lined leach pad suitable for commercially proven sulfuric acid bio-leaching technology with a conventional SX and EW process facilities to produce copper cathodes meeting London Metals Exchange (LME) Copper Grade A quality standards or ASTM B115-10 – Cathode Grade 1.

The Phase 1 implementation scheme for the Project is an open pit mine initially processing materials with crushing, bio-heap leaching and solvent extraction and electrowinning (SX/EW) facilities to produce LME Grade A copper cathodes for sale in Argentina or for export. Phase 1 mining extracts a total of 11.90 billion lbs. (5,404 kt) of contained copper from the resource, of which 8.68 billion lbs (3.938 kt) is considered recoverable copper as cathodes. The total copper recovery expected is approximately 73% and considers scale-up efficiencies and production distribution over a two-year timeframe from placement of material on the leach pad.

The Phase 1 Base Case includes processing facilities to produce 175,000 tonnes per annum (tpa) of copper cathodes from higher grade, heap leachable copper content materials. The processing facility will function through to the completion of mining of Phase 1 in Year 21 with stockpile reprocessing and residual leaching operations to Year 28 (the "Base Case"). Mining operations for the Base Case ramp up over the proposed mine life from approximately 70 million total tonnes per annum moving up to 175 million tonnes per annum through the life of the Project as copper grades decrease, and waste stripping increases.

To demonstrate the scalability of the project at lower initial capital requirements, an alternative case was also developed at a lower copper production rate of 125,000 tonnes per annum of copper cathodes. The processing facility will function through to the completion of mining of Phase 1 in Year 30 with stockpile reprocessing and residual leaching operations to Year 33 (the "Alternative Case"). Mining operations for the Alternative Case ramp up over the proposed mine life from approximately 70 million total tonnes per annum to 110 million tonnes per annum through the life of the project as copper grades decrease, and waste stripping increases.

Phase 1 project development for either case is expected to take approximately 33 months to mechanically complete from notice to proceed and point of project financing. Construction development will prioritize the initial leach pad and ponds, crushing and stacking systems to facilitate the placement of leach materials on the pad during pre-striping and prior to starting the rest of the facilities start. Ramp up to full leaching capacity is expected to take six to nine months from plant start-up and placement of mineralized material on the pad with commercial production of copper from the SX/EW plant is expected to be achieved in approximately 12 months from start of leaching assuming the mining schedules shown. Finalization of the necessary permits to begin work is expected to be completed during the proposed feasibility study timeframe. Early works will commence, once project funding is available, with access road upgrades, site preparation, construction infrastructure and power line development.

Metallurgical characterization testing has been completed in the form of sequential assay (sulfuric acid and cyanide steps) for the resources considered, bottle roll testing and column testing. The sequential assay method provides acid soluble copper (CuAS) and cyanide soluble Copper (CuCN), both assays combined (CuAS + CuCN) provide a number for readily leachable/soluble copper (CuSOL); compared to the total copper assay (CuT).

A summary of the resources considered for processing by source is provided in Table 17.1. Primary copper materials with sufficient leachable copper content are included in the Phase 1 leaching resources considered. Stockpiled material is limited to the resources contained in the Phase 1 project mining plans. Additional primary copper resource materials related to the Phase 2 processing options are not represented.

Table 17.1: Potential Process Materials Distribution – Leach Only Pit Shell

Material Type	Potential Process Material (Mt)	Grade (CuT) (%)	Grade (CuSOL) (%)	Gold Grade (g/t)	Silver Grade (g/t)
Heap Leach Total	**1,182**	**0.457**	**0.311**	**0.05**	**1.25**
Oxide/LIX Material	22.2	0.079	0.051	0.04	0.80
Supergene Material	1,015	0.472	0.341	0.05	1.19
Primary Material	145	0.409	0.143	0.06	1.71
Potential Mill/Leach Material (Stockpiled)	**129.3**	**0.107**	**0.023**	**0.07**	**0.91**
Primary Material	58.6	0.178	0.027	0.04	0.87
Oxide/LIX/Supergene Material	70.7	0.047	0.019	0.10	0.94

The primary copper mineralization dominant material (129.3 million tonnes in the current plan) will be stockpiled for future processing routes, which may include a concentrator or alternative leaching technologies.

17.2 PROCESS PLANT LOCATIONS

The entire site plan includes the locations of the process plants (heap leach, SX/EW facilities, acid plant and copper concentrator) and ancillary facilities through Figure 5.5. The locations were based on the following factors:

- Phase 1 and Phase 2 processing options and long-term expansion considerations.
- Favorable topography allows for gravity flow within and between process facilities and to minimize mass earthworks and pumping requirements.
- Preference for proximity of acid plant, agglomeration, heap leach and SX/EW.

It is important to minimize material transportation distance from the mine to the process plant. The primary crusher would be located as close to the mine as possible (i.e., pit rim configuration) to minimize truck haulage distance, thereby dictating conveyance of crushed mineralized material to the process plants. The ancillary facilities would be located close to the main facilities for purposes of convenience, e.g., the truck shop adjacent to the pit and the primary crusher, or isolation, e.g., permanent camp located to minimize plant noise impact.

17.3 HEAP LEACH (SX/EW) PROCESS FLOWSHEET

The Project envisions processing the Oxide/LIX and Supergene resources mined. The conceptual block flow diagram for the processing facilities included in the Project is presented in Figure 17.1.

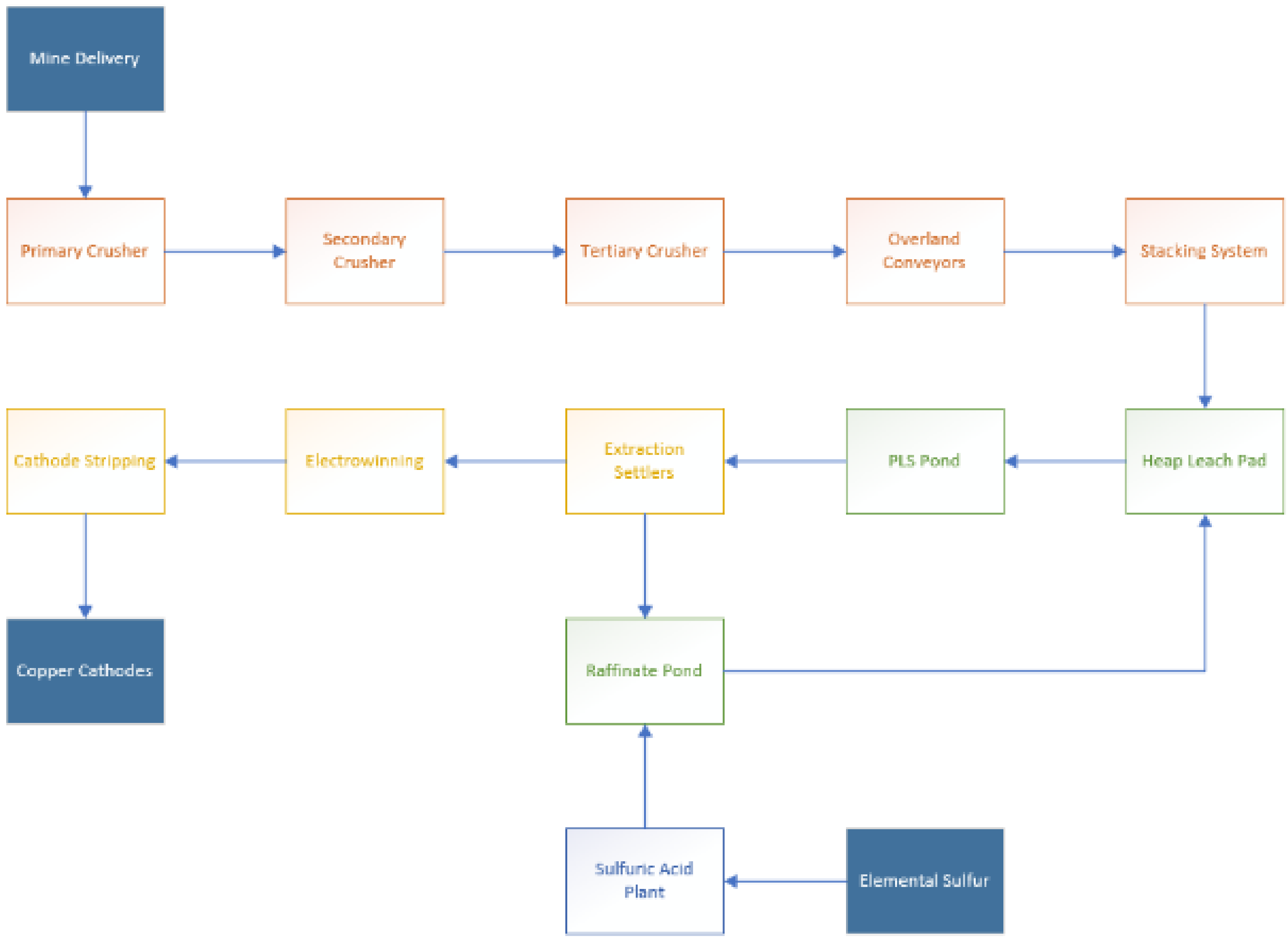


Figure 17.1: Heap Leach Process Flowsheet

17.3.1 Base Case Processing Facilities (175,000 tonnes per annum Cu Cathode)

An initial Phase 1 Base Case was developed at a nominal copper production rate of 175,000 tonnes per annum of copper cathodes. An expansion of the mining rates and materials handling facilities is required by Year 4 and again in Year 7 to maintain copper production as the copper grade drops. This initial processing facility will function through to the completion of mining for the initial project phase in Year 21 with low grade stockpile reclaim and reprocessing in Years 21-27 and residual leaching operations to Year 28.

Mining operations ramp up over the proposed mine life from approximately 80 million total tonnes per annum moved up to 150 million tonnes per annum moved through the life of the project as copper grades diminish, and waste stripping requirements increase. Processing feed rates begin at 22 million tonnes per annum and increase to 35 million tonnes per annum in Year 4 and ultimately to 50 million tonnes per annum in Year 7 through the life of the Phase 1 project.

Initial capital costs for the Base Case project includes camps and site infrastructure, mining equipment to support the initial mining rate of 80 million tonnes per annum, crushing and heap leach stacking systems to support the initial 22 million tonnes per annum throughput rate, SX/EW facilities to support 175,000 tonnes per annum nominal copper production with an EW rectifier turn-up capacity of 12%, and heap leach facility (Phases 1A and 1B) to support placement of 75.9 million tonnes.

An initial on-site sulfuric acid production at 200,000 tonnes per annum and heap leach facility to support placement of 75.9 million tonnes. The sulfuric acid plant includes power generation from steam generated that off-sets 15%-20% of the processing demand. The waste heat carried by the sulfuric acid plant cooling water will also be used to heat SX/EW electrolytes and leaching system make-up water.

A 220kV site substation and power line will be provided by the regional utility YPF Luz to include 100% renewable power supply. Site infrastructure also includes emergency power (20MW) via diesel generation The costs are reflected in the cost of power and included as process operating costs in the financial model.

Sustaining capital for the Base Case includes mine equipment additions, heap leach pad expansions approximately every 2-3 years, mine equipment additions as the pit matures, crushing plant expansions in years 4, and crushing/stacking expansion in 7, an additional SX train for increased flows in year 7, and on-site acid plant expansions in years 4, 6 and 7 to an ultimate capacity of 700,000 tonnes of acid per year as processing rates and mining rates increase.

17.3.2 Alternative Case Processing Facilities (125,000 tonnes per annum Cu Cathode)

To demonstrate the project at lower initial capital requirements, an alternative case was also developed at a lower copper production rate of 125,000 tonnes per annum of copper cathodes. Options considered as alternatives ranged from 50,000 tpa to 125,000 tpa copper production rates.

This initial Phase 1 project Alternative Case option considers a processing facility to nominally produce 125,000 tonnes per annum of copper cathodes from higher grade, highly leachable (soluble) copper content materials. This initial processing facility will function through to the completion of mining for the initial project phase in Year 30 with lower grade stockpile reprocessing in Years 30-32 and residual leaching operations to Year 33.

Alternative Case mining operations ramp up over the proposed mine life from approximately 80 million total tonnes per annum moved up to 110 million tonnes per annum moved through the life of the project as copper grades diminish, and waste stripping requirements increase. Processing feed rates begin at 18 million tonnes per annum and increase to 25 million tonnes per annum in year 7 and ultimately to 50 million tonnes per annum in year 13 through the life of the Phase 1 project.

Initial capital costs for the Alternative Case project includes camps and site infrastructure, mining equipment to support the initial mining rate of 80 million tonnes per annum, crushing and heap leach stacking systems to support the initial 18 million tonnes per annum throughput rate, SX/EW facilities to support 175,000 tonnes per annum nominal copper production with a turn-up capacity of 12%, sulfuric acid production at 150,000 tonnes per annum and heap leach facility (Phase 1A) to support placement of 42.4 million tonnes. A 220kV substation and power line will be provided by the regional utility to include renewable power supply. The costs are reflected in the cost of power.

Sustaining capital for the Alternative Case includes mine equipment additions, heap leach pad expansions approximately every 2-3 years, mine equipment additions as the pit matures, crushing plant expansions in years 5 and 10, and crushing/stacking expansion in 13, an additional SX train for increased flows in year 10, and acid plant expansions in years 6, 10 and 13 as processing rates and mining rates increase.

17.3.3 Crushing and Stacking

Run-of-Mine (ROM) Supergene material mined from the open pit resource will be delivered directly to the primary gyratory crusher; located at the closest and safest site in proximity to the lowest elevation section of the ultimate pit rim. This location minimizes both horizontal and vertical haulage for transport of material from the mine to the primary crushing stage using employing an MMD Sizer to produce a minus 150 mm discharge product.

Crushed material will cycle through a series of secondary/tertiary screens and crushers, capable of producing a yearly maximum tonnage rate initially of 20-25M tonnes per annum (tpa) for the Base Case and 18-20M tpa for the Alternative Case. The crusher and stacking facilities will be operated 365 days/year and 24 hours/day at an operating availability of 70% or 6,132 operating hours using a two-shift rotation for process personnel. The crushing circuit produces a product at 80% passing 16.4 mm with an expected moisture content of 3%.

The crusher and stacking facilities will be operated 365 days/year and 24 hours/day using a two-shift rotation for process personnel. The crushing circuit produces a product at 80% passing 16.4 mm with an expected moisture content of 3%.

The crusher product will be transported to the heap leach pad by a series of overland conveyors. The overland conveyors will discharge the material into agglomeration drums, bringing the moisture up to 5% using raffinate solution directly from the raffinate pond, outside of the heap leach boundaries.

The agglomerated material will be discharged through a tripper conveyor, a series of portable conveyors, and finally a telescoping radial stacker to place the material directly on the heap leach pad.

For the Base Case option, the crushing plant will have three (3) development phases during the Project, with the initial Phase 1 allowing production of a maximum of 25M tpa of material. In Year 4 of operation, a secondary and tertiary crusher is added to increase the capacity to 36M tpa. In Year 7, a final addition of a secondary and tertiary crusher is installed to increase the capacity a maximum of 51.4M tpa of material.

For the Alternate Case option, the crushing plant will have four (4) development phases during the Project, with the initial Phase 1 allowing production of a maximum of 18M tpa of material. In Year 5 of operation, a secondary and tertiary crusher is added to increase the capacity to 25M tpa. In Year 10, a second line will be operational, duplicating the crushing circuit in the initial project and allowing production capacity of 43M tpa with a final addition of a secondary and tertiary crusher is installed to increase the capacity a maximum of 51.4M tpa of material.

17.3.4 Heap Leach Pad and Ponds

Crushed material from Oxide/LIX and Supergene material types that is mined will be stacked on the heap leach pad; intending to be built in 9 m lifts to a maximum elevation of approximately 150 m. Material grade is determined by sequential assay and any leach material is stacked on the pad if it is above the soluble copper cut-off grade.

The total initial pad design has a capacity of 956.6M tonnes, a future pad expansion is anticipated accommodate an additional 225M tonnes for the final total material requirement of 1,182M tonnes. The possibility to increase the nominal heap height is also under consideration. The leach pad develops over time inside a valley with an upstream growth as prescribed in Figure 17.2 and Figure 17.3. Pad ultimate height of 150m is not considered extreme for design purposes and collection system integrity.

Raffinate leach solution is pumped directly from the Raffinate pond and applied to the surface of the heap through irrigation. The acid solution will percolate through the heap leach material dissolving copper and some impurities. The resulting pregnant leach solution (PLS) will drain from the bottom of the heap and will be collected in the PLS pond. From the PLS pond, the solution will be pumped to the feed tank for solvent extraction. The leaching sequence for the pad is planned as follows in Table 17.2.

Table 17.2: Average Leach Cycle Times

Leach Cycle Component	**Time (days)**
Pad Loading	14
Surface Preparation / Piping	7
Active Solution Application	180
Drain Down and Decommissioning	9
Minimum Total Cycle Time	**210**

Leaching solutions (raffinate), containing dilute sulfuric acid (5-10 g/L H_2SO_4) will be pumped from the raffinate pond and applied to the top of each lift and allowed to percolate though the copper leach material. Solution application is planned to be by a combination of sprinklers and drip emitters.

The raffinate is being added at a rate of 6 L/hr/m^2 with biomass inoculated into the raffinate allowing for slower, bioleaching of sulfide minerals. The active biomass will be a product of the column test work currently being completed at SGS on the Project material.

Soluble copper (CuSOL) recovery is estimated based on the SGS and Plenge test results is 100%, with residual copper (CuRES) of 15% for a combined total copper (CuT) recovery expectation of approximately 73%. As discussed in Section 13.0, the column work completed at Plenge did not include a bacterial leaching component that would improve overall leaching performance.

Since mineralized material placement occurs over a year's time in the mine production plan, the last quarter of the year (3 months) is not expected to contribute to the production in the year mined. Recovery has been shifted to the following year to account for the placement and preparation time required in the current estimations. Only 60% of the recoverable copper is considered in the year placed with the remaining 40% coming is the following year.

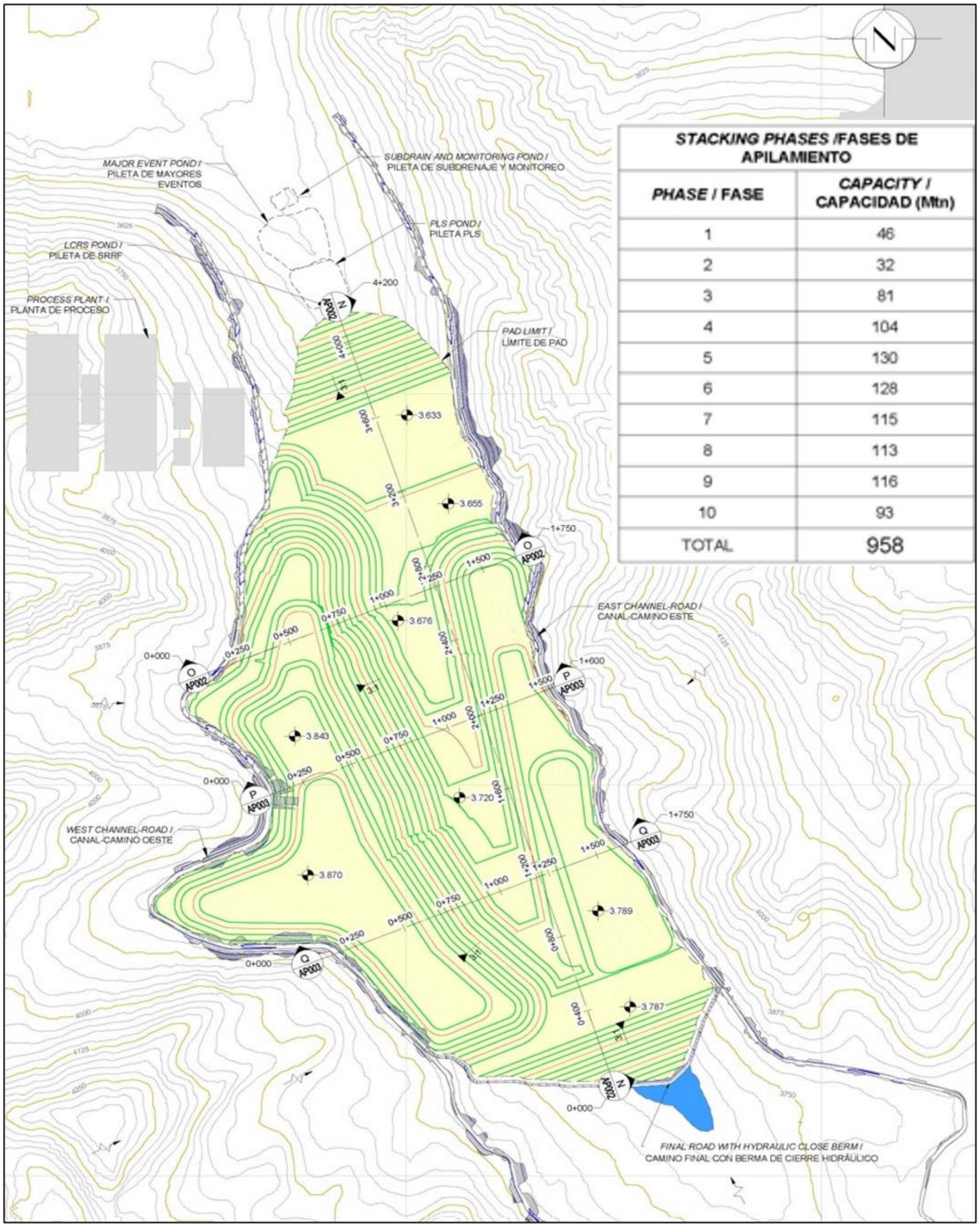
For the Base Case option, the total initial pad design capacity of 956.6M tonnes is completed over 21 years and a total of 8 expansions after the initial construction period is over. The initial build out of Phase 1A&B has a capacity of 75.9M tonnes of crushed/agglomerated material. The remaining material capacity that is capable of being placed for Phase 2 (77.5M tonnes), Phase 3 (96.3M tonnes), Phase 4 (120M tonnes), Phase 5 (120M tonnes), Phase 6 (120M tonnes), Phase 7 (120M tonnes), Phase 8 (120M tonnes), and Phase 9 (106.9M tonnes).

A future pad expansion is anticipated in Years 21 and 23 to accommodate an additional 225M tonnes for the final total material requirement of 1,182M tonnes. The possibility to increase the nominal heap height is also under consideration.

For the Alternate Case option, the total initial pad design capacity of 956.6M tonnes is completed over 31 years and a total of 9 expansions after the initial construction period is over. The initial build out of Phase 1A has a capacity of 42.4M tonnes of crushed/agglomerated material. The remaining material capacity that is capable of being placed for Phase 1B (33.5M tonnes), Phase 2 (77.5M tonnes), Phase 3 (96.3M tonnes), Phase 4 (120M tonnes), Phase 5 (120M tonnes), Phase 6 (120M tonnes), Phase 7 (120M tonnes), Phase 8 (120M tonnes), and Phase 9 (106.9M tonnes).

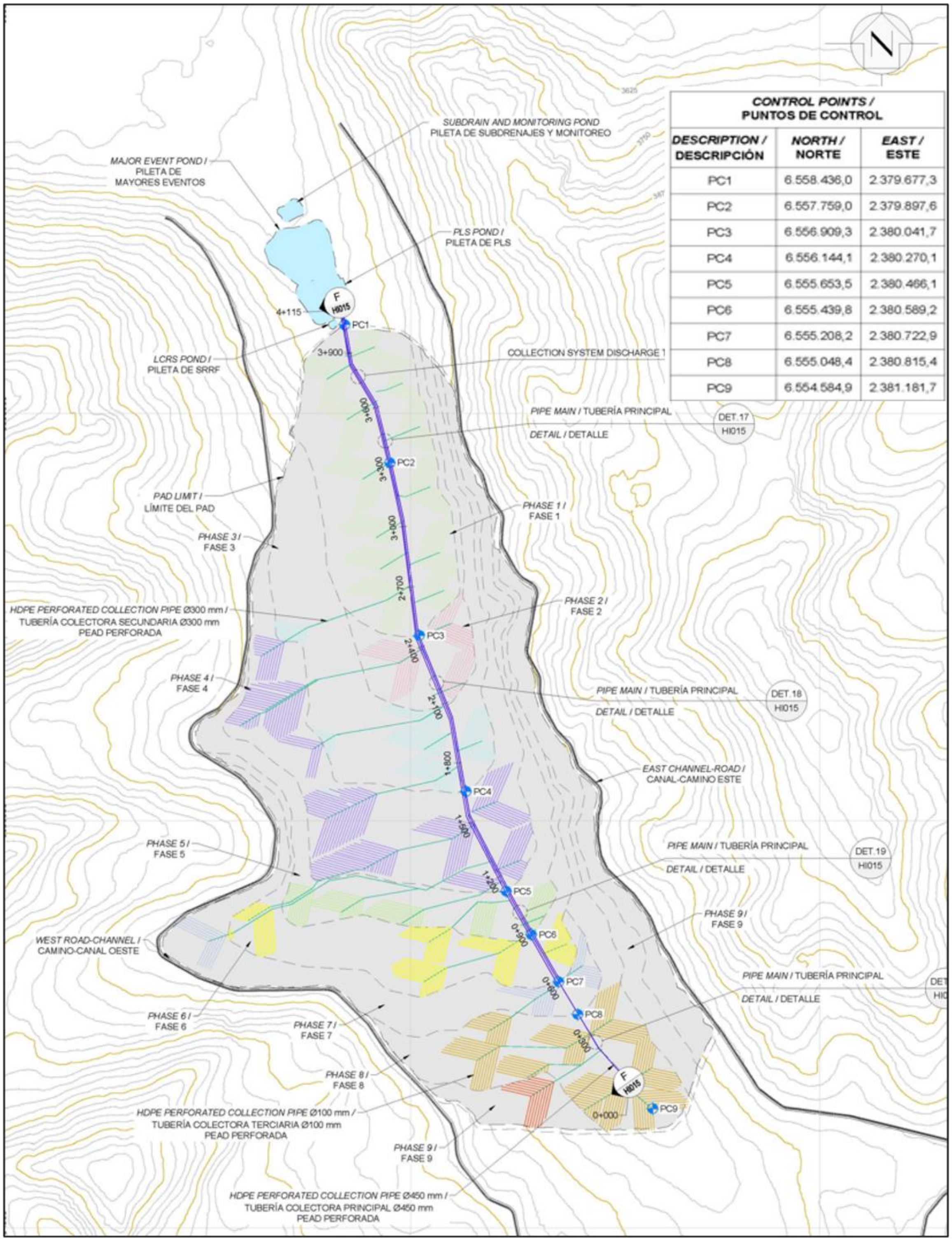
A future pad expansion is anticipated in Years 28 and 30 to accommodate an additional 225M tonnes for the final total material requirement of 1,182M tonnes. The possibility to increase the nominal heap height is also under consideration.

The expected materials placement and copper production schedule is presented Table 17.5 for Years 1 through 17 and Table 17.6 for Years 18 through life of mine. Copper recovered to cathodes considers a heap efficiency (5%-7% impact) and solution inventory (3%-5% impact) factor of 90% of the extractable copper based on general experience for large leach pads.



STACKING PHASES /FASES DE APILAMIENTO	
PHASE / FASE	CAPACITY / CAPACIDAD (Mtn)
1	46
2	32
3	81
4	104
5	130
6	128
7	115
8	113
9	116
10	93
TOTAL	958

Figure 17.2: Heap Leach Pad General Layout



DESCRIPTION / DESCRIPCIÓN	NORTH / NORTE	EAST / ESTE
PC1	6.558.436,0	2.379.677,3
PC2	6.557.759,0	2.379.897,6
PC3	6.556.909,3	2.380.041,7
PC4	6.556.144,1	2.380.270,1
PC5	6.555.653,5	2.380.466,1
PC6	6.555.439,8	2.380.589,2
PC7	6.555.208,2	2.380.722,9
PC8	6.555.048,4	2.380.815,4
PC9	6.554.584,9	2.381.181,7

Figure 17.3: PLS Collection

Table 17.3: Base Case Leach Pad Plan and Estimated Copper Production – Year 1 through Year 17

Heap Leaching Production Plan		Units	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Mined Tonnes	Tonnes	tonnes	**1,182,412,578**	-	16,868,298	24,999,498	22,169,116	22,119,587	30,000,000	29,989,025	34,898,803	50,000,000	50,000,000	50,000,000	50,000,000	50,000,000	50,000,000	49,058,494	49,631,418	50,000,000	50,000,000	50,000,000
	In-situ Soluble Copper Grade	%	**0.00**	-	0.61	0.68	0.77	0.76	0.56	0.56	0.48	0.33	0.33	0.33	0.33	0.33	0.31	0.34	0.33	0.28	0.28	0.29
	Contained Soluble Copper	tonnes	**3,678,833**	-	103,444	170,813	170,701	168,856	169,129	168,565	166,336	165,592	167,042	166,104	164,814	166,074	154,234	166,906	165,453	140,866	138,390	144,516
	Residual Copper Grade	%	**0.00**	-	0.08	0.11	0.13	0.19	0.13	0.14	0.17	0.13	0.11	0.12	0.14	0.12	0.11	0.11	0.13	0.12	0.14	0.14
	Contained Residual Copper	tonnes	**1,727,024**	-	13,469	27,917	28,661	40,960	39,137	42,901	57,757	62,723	53,057	59,306	67,903	59,507	56,286	53,962	63,649	62,103	71,406	70,446
	In-situ Total Copper Grade	%	**0.00**	-	0.69	0.79	0.90	0.95	0.69	0.71	0.64	0.46	0.44	0.45	0.47	0.45	0.42	0.45	0.46	0.41	0.42	0.43
	Contained Total Copper	tonnes	**5,405,857**	-	116,913	198,729	199,361	209,816	208,266	211,466	224,093	228,315	220,098	225,410	232,718	225,581	210,520	220,868	229,101	202,969	209,796	214,962
Heap Leaching - Soluble Copper Available		tonnes	**3,678,833**		62,067	205,932	170,746	169,594	169,020	168,791	167,228	165,889	166,462	166,479	165,330	165,570	158,970	161,837	166,034	150,701	139,381	142,066
Heap Leaching - Residual Copper Available		tonnes	**259,054**		1,212	4,533	4,254	5,406	5,980	6,209	7,772	9,111	8,538	8,521	9,670	9,430	8,636	8,234	8,966	9,408	10,153	10,625
Total Recovered Copper to Cathodes LME Grade A		tonnes	**3,937,886**		N/A	192,500	192,500	175,465	175,000	175,000	175,000	175,000	175,000	175,000	175,000	175,000	167,606	170,071	175,000	160,109	149,533	152,690
		lbs	**8,681,464,009**			424,385,500	424,385,500	386,829,151	385,805,000	385,805,000	385,805,000	385,805,000	385,805,000	385,805,000	385,805,000	385,805,000	369,504,115	374,937,743	385,805,000	352,976,138	329,661,126	336,620,534
			Copper In-Process Inventory		tonnes	17,965	465	-														

Total Copper Recovery **72.8%**
Soluble Copper Recovery **107.0%**

Table 17.4: Base Leach Pad Plan and Estimated Copper Production - Years 18 - LOM

Heap Leaching Production Plan		Units	Total	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28
Mined Tonnes	Tonnes	tonnes	**1,182,412,578**	50,000,000	50,000,000	50,000,000	50,000,000	50,000,000	50,000,000	50,000,000	50,000,000	50,000,000	2,678,339	
	In-situ Soluble Copper Grade	%	**0.00**	0.27	0.25	0.25	0.15	0.14	0.14	0.14	0.14	0.14	0.14	
	Contained Soluble Copper	tonnes	**3,678,833**	137,242	127,361	124,502	76,442	70,337	70,337	70,337	70,337	70,337	3,768	-
	Residual Copper Grade	%	**0.00**	0.18	0.26	0.30	0.18	0.13	0.13	0.13	0.13	0.13	0.13	
	Contained Residual Copper	tonnes	**1,727,024**	89,684	131,218	150,243	89,601	66,315	66,315	66,315	66,315	66,315	3,552	-
	In-situ Total Copper Grade	%	**0.00**	0.45	0.52	0.55	0.33	0.27	0.27	0.27	0.27	0.27	0.27	
	Contained Total Copper	tonnes	**5,405,857**	226,926	258,579	274,746	166,043	136,652	136,652	136,652	136,652	136,652	7,320	-
Heap Leaching - Soluble Copper Available		tonnes	**3,678,833**	140,151	131,314	125,646	95,666	72,779	70,337	70,337	70,337	70,337	30,395	1,507
Heap Leaching - Residual Copper Available		tonnes	**259,054**	12,298	17,191	21,395	17,079	11,344	9,947	9,947	9,947	9,947	4,299	213
Total Recovered Copper to Cathodes LME Grade A		tonnes	**3,937,886**	152,450	148,504	147,041	112,745	84,123	80,284	80,284	80,284	80,284	34,694	1,720
		lbs	**8,681,464,009**	336,090,721	327,392,491	324,166,237	248,556,720	185,457,909	176,994,117	176,994,117	176,994,117	176,994,117	76,486,251	3,792,403

Total Copper Recovery **72.8%**
Soluble Copper Recovery **107.0%**

Table 17.5: Alternate Case Leach Pad Plan and Estimated Copper Production – Year 1 through Year 17

Heap Leaching Production Plan		Units	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17
Mined Tonnes	Tonnes	tonnes	**1,182,423,093**	-	16,868,298	17,999,666	16,745,806	14,585,029	16,706,125	23,539,177	23,851,117	25,000,000	25,000,000	25,000,000	37,000,000	36,885,009	34,981,832	50,000,000	49,046,148	49,752,839	49,212,661	49,039,643
	In-situ Soluble Copper Grade	%	**0.311%**	-	0.61	0.68	0.73	0.83	0.72	0.51	0.50	0.49	0.48	0.48	0.32	0.33	0.34	0.24	0.24	0.23	0.23	0.24
	Contained Soluble Copper	tonnes	**3,678,863**	-	103,444	122,030	122,078	121,653	120,553	120,360	119,500	121,277	120,247	119,133	117,547	120,112	118,361	119,664	116,485	113,394	113,087	118,132
	Residual Copper Grade	%	**0.146%**	-	0.08	0.11	0.12	0.15	0.18	0.13	0.15	0.10	0.13	0.16	0.13	0.09	0.13	0.07	0.12	0.16	0.16	0.09
	Contained Residual Copper	tonnes	**1,727,041**	-	13,469	19,799	19,479	22,316	29,648	30,935	36,665	24,820	31,685	39,114	49,688	32,589	44,262	35,575	56,766	77,370	79,422	45,786
	In-situ Total Copper Grade	%	**0.457%**	-	0.69	0.79	0.85	0.99	0.90	0.64	0.65	0.58	0.61	0.63	0.45	0.41	0.46	0.31	0.35	0.38	0.39	0.33
	Contained Total Copper	tonnes	**5,405,903**	-	116,913	141,829	141,558	143,969	150,200	151,295	156,165	146,097	151,933	158,247	167,235	152,700	162,623	155,239	173,252	190,765	192,509	163,918
Heap Leaching - Soluble Copper Available		tonnes	**3,678,863**			155,973	142,748	121,823	120,993	120,437	119,844	120,566	120,659	119,579	118,181	119,086	119,061	119,142	117,756	114,631	113,210	116,114
Heap Leaching - Residual Copper Available		tonnes	**259,056**			3,398	3,345	3,177	4,007	4,563	5,156	4,434	4,341	5,421	6,819	5,914	5,939	5,858	7,244	10,369	11,790	8,886
Total Recovered Copper to Cathodes LME Grade A		tonnes	**3,937,919**		N/A	140,000	140,000	140,000	135,465	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000
		lbs	**8,681,536,222**			308,644,000	308,644,000	308,644,000	298,645,151	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000
			Copper In-Process Inventory		tonnes	19,372	25,465	10,465														

Total Copper Recovery **72.7%**
Soluble Copper Recovery **107.0%**

Table 17.6: Alternate Case Leach Pad Plan and Estimated Copper Production - Years 18 - LOM

Heap Leaching Production Plan		Units	Total	Year 18	Year 19	Year 20	Year 21	Year 22	Year 23	Year 24	Year 25	Year 26	Year 27	Year 28	Year 29	Year 30	Year 31	Year 32	Year 33
Mined Tonnes	Tonnes	tonnes	**1,182,423,093**	39,763,863	39,821,942	40,000,000	39,075,545	40,000,000	39,527,570	39,238,927	37,844,668	41,121,798	47,821,537	45,188,554	47,399,033	50,000,000	50,000,000	24,406,305	
	In-situ Soluble Copper Grade	%	**0.311%**	0.30	0.30	0.30	0.30	0.29	0.30	0.29	0.30	0.28	0.23	0.24	0.22	0.16	0.13	0.13	
	Contained Soluble Copper	tonnes	**3,678,863**	117,871	120,084	119,109	118,857	117,007	116,975	114,409	114,937	113,419	108,921	107,716	104,158	80,592	65,688	32,064	-
	Residual Copper Grade	%	**0.146%**	0.12	0.08	0.10	0.10	0.13	0.14	0.18	0.18	0.19	0.22	0.25	0.29	0.21	0.12	0.12	
	Contained Residual Copper	tonnes	**1,727,041**	47,526	32,774	39,276	40,954	53,284	53,502	70,605	67,085	77,209	107,192	115,224	138,949	102,972	61,216	29,881	-
	In-situ Total Copper Grade	%	**0.457%**	0.42	0.38	0.40	0.41	0.43	0.43	0.47	0.48	0.46	0.45	0.49	0.51	0.37	0.25	0.25	
	Contained Total Copper	tonnes	**5,405,903**	165,397	152,858	158,384	159,811	170,291	170,477	185,014	182,022	190,628	216,113	222,940	243,107	183,563	126,904	61,945	-
Heap Leaching - Soluble Copper Available		tonnes	**3,678,863**	117,975	119,199	119,499	118,958	117,747	116,988	115,435	114,726	114,026	110,720	108,198	105,581	90,018	71,650	45,514	12,826
Heap Leaching - Residual Copper Available		tonnes	**259,056**	7,025	5,801	5,501	6,042	7,253	8,012	9,565	10,274	10,974	14,280	16,802	19,419	17,604	11,688	6,362	1,793
Total Recovered Copper to Cathodes LME Grade A		tonnes	**3,937,919**	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	125,000	107,623	83,337	51,876	14,619
		lbs	**8,681,536,222**	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	275,575,000	237,264,633	183,725,589	114,365,865	32,227,984

Total Copper Recovery	**72.7%**
Soluble Copper Recovery	**107.0%**

17.3.5 Solvent Extraction and Electrowinning (SX/EW)

Copper leached in the heap leaching operation will be recovered as a saleable copper cathode product in a solvent extraction (SX) and electrowinning (EW) facility. This facility will be operated continuously with an effective availability of 98%.

Solvent Extraction (SX)

The SX plant is designed to be modular and processes pregnant leach solution (PLS) in three extraction (E) units typically in a series-parallel configuration for each processing train with a single stage of stripping (S) as indicated by: E1×E2×E1P×1S. Two minutes mixing time per mixer-settler unit is anticipated. No wash stages or after-settlers are anticipated or included in the current design. A loaded organic tank and diluent storage tank are collocated with the SX mixer settlers. The SX plant is designed for an initial two processing trains to be modular construction and have a fabric structure cover over each train for weather protection.

The SX plant is designed to operate in several configurations and organic to aqueous (O:A) flow ratios to accommodate the varying tonnage rates to the heap leach and resulting surface areas under leach with decreasing process PLS feed grades from the mining operations over time. The PLS flow capacity for various extraction stage configurations per SX train is show below.

Table 17.7: SX Processing Train Operating Capacity Options

SX Train PLS Flow Capacity		Extraction Units Configuration	Units	PLS Flow
O:A Ratio =	1	Series	m3/h	1,700
		Series Parallel	m3/h	3,400
		Parallel	m3/h	5,100
O:A Ratio =	2	Series	m3/h	850
		Series Parallel	m3/h	1,700
		Parallel	m3/h	2,550

For the Base Case option, initially two trains of SX will be installed. PLS copper grades will vary from 7.88 g Cu/L in the initial years to 0.90 g Cu/L at the end of the mine life. As leach system flows increase with increasing material to the pad with dropping copper grades, a third SX train is added in Year 7. Each train is provided with a tensioned fabric structure covering to minimize the potential impacts of snow and wind.

The PLS from the heap leach ponds will be pumped for processing to a copper SX/EW plant capable of nominally producing 175,000 tpa of copper cathodes (design maximum of approximately 201,210 tpa) with a plant availability of 95%. Any copper in the PLS solution not plated as cathode will continue to recirculate in inventory until EW available capacity is available.

The EW plant nominal capacity will be 175,000 tpa copper production, with rectified turn-up able to accommodate a maximum design production up to 201,250 tpa of copper cathodes. Copper EW is expected to require 258 cells in three processing bays of 86 cells with two parallel lines of 43 cells per bay. EW cells are constructed of polymer concrete and containing 84 cathodes (1.15 m2/side plating area per cathode) and 85 anodes each, operating in series and connected to rectifier transformer units. Expected current efficiency is 92% operating at a nominal 310 A/m2 current density (design maximum 356.5 A/m2). Cathode stripping from the permanent stainless-steel blanks will be done with two (2) stripping machine that are of a semi-automatic, robotic design positioned in-line, one each between the three processing bays on either side of the center bay.

For the Alternate Case option, initially two trains of SX will be installed. PLS copper grades will vary from 6.18 grams Cu/L (g Cu/L) in the initial years to 0.79 g Cu/L at the end of the mine life. As leach system flows increase with increasing material to the pad with dropping copper grades, a third SX train is added in Year 13.

The PLS from the heap leach ponds will be pumped for processing to a copper SX/EW plant capable of nominally producing 125,000 tpa of copper cathodes (design maximum of approximately 143,750 tpa) with a plant availability of 95%. SX extraction efficiency based on an average 92% copper recovery from PLS to cathodes. Any copper in the PLS solution not plated as cathode will continue to recirculate in inventory until EW available capacity is available.

The Electrowinning (EW) plant nominal capacity will be 125,000 tpa copper production, with rectified turn-up able to accommodate a maximum design production up to 143,750 tpa of copper cathodes. Copper EW is expected to require 184 cells in two processing bays of 92 cells with two parallel lines of 46 cells per bay. EW cells are constructed of polymer concrete and containing 84 cathodes (1.15 m^2/side plating area per cathode) and 85 anodes each, operating in series and connected to rectifier transformer units. Expected current efficiency is 92% operating at a nominal 310 A/m^2 current density (design maximum 356.5 A/m^2). Cathode stripping from the permanent stainless-steel blanks will be done is a stripping machine that is of a semi-automatic, robotic design positioned in-line between the two processing bays.

Copper cathode bundles of up to 2,000 kg to 2,500 kg each will be sampled, weighed, labeled, and strapped then placed in a secure area for pick up by a copper broker for transport and sale free of board (FOB) at the Los Azules site.

The EW operation will be housed in a pre-engineered steel building fitted with an overhead crane for copper production material handling. Siding will be fabric or fiberglass construction. The process office and process control room will be in a prefabricated building including a small wet laboratory for process control assays and mine grade control sample assays.

The facilities also include a tank farm area composed of electrolyte solution tanks, electrolyte filters, crud handling system, and a solution management holding tank.

17.3.6 Sulfuric Acid Plant

Rather than trucking concentrated sulfuric acid up to the Project, elemental sulfur produced at the YPF refinery in Lujan de Cuyo, Argentina, can be utilized to create sulfuric acid at site. This decreases transportation, environmental, and health and safety risks from transportation of sulfuric acid to the project site. Initial analysis of the YPF sulfur product at the Nuton lab at its Bundoora lab does not indicate any deleterious components to the leaching systems are present.

Elemental sulfur prill is brought into the facility and melted into liquid sulfur prior to combustion with oxygen. This produces gaseous sulfur dioxide (SO_2) and sulfur trioxide (SO_3). The SO_2/SO_3 is used to convert to sulfuric acid (H_2SO_4); at normal operation the acid plant will produce sulfuric acid concentration of 94.5%. The SO_3 is absorbed into a circulating sulfuric acid with a concentration of 98% to help form more sulfuric acid.

For the Base Case option, the Phase 1 acid plant will produce approximately 200,000 tonnes of sulfuric acid a year to be used in the heap leach process. The Phase 2 acid plant train is expected to be operational in Year 4 and 100,000 tonnes per annum, bringing total acid production capacity to 300,000 tonnes of sulfuric acid a year. A Phase 3 acid plant train of 100,000 tonnes per annum is expected to be operational in Year 6, this brings the total acid production capacity to 400,000 tonnes of sulfuric acid a year. A Phase

4 acid plant of 300,000 tonnes per annum is expected to be operational in Year 7, this brings the total acid production capacity to 700,000 tonnes of sulfuric acid a year.

For the Alternate Case option, the Phase 1 acid plant will produce approximately 150,000 tonnes of sulfuric acid a year to be used in the heap leach process. The Phase 2 acid plant train is expected to be operational in Year 6 and be a duplicate of the Phase 1 operation, bringing total acid production capacity to 300,000 tonnes of sulfuric acid a year. A Phase 3 acid plant train is expected to be operational in Year 10, this brings the total acid production capacity to 600,000 tonnes of sulfuric acid a year. A Phase 4 acid plant is expected to be operational in Year 19, this brings the total acid production capacity to 700,000 tonnes of sulfuric acid a year.

Any excess heat and heated water can be utilized in the process plant to fill necessary gaps. Steam will be captured and put through a combined cycle turbine generator to produce power.

17.3.7 Reagents, Water, and Power

Projected reagent and operating consumables requirements for the LOM project are summarized as follows.

- Energy: 1.097 kWh/lb Cu produced
- Makeup fresh water: 270 liters/second (lps) average
- Sulfuric Acid: 1,381 tonne/day average, net of SX/EW credits
- Elemental Sulfur: 409 tonne/day average, net of SX/EW credits
- SX Reagents
  - Extractant: 1200 - 1600 kg per day
  - Diluent: 2272 – 3000 liters per day
- EW Reagents
  - Cobalt Sulfate: 0.025 kg per tonne Cu produced
  - Guar: 0.005 kg per tonne Cu produced
  - Mist Suppressant: FC-1100

17.3.8 Water

Heap leach water make-up requirements are the largest consumer of fresh water at the Los Azules site. Moisture retention is estimated to be 5% by weight for the mineralized material in the leaching areas. This is calculated from an initial moisture content of approximately 3% by weight of non-agglomerated stacked material and an estimated terminal moisture content of 8% after leaching and complete drain down. Evaporation losses are also significant, averaging 830 mm per year for the exposed active leaching areas.

The heap leach pad area is also the largest precipitation collector on the site as well.

Knight Piesold completed a conceptual water balance around the heap leach circuit as dictated in Table 17.8 and further shown in Table 17.8. The requirements for the conceptual water balance are as follows:

- Heap Leach Pad Area: Divided into two types of areas, an active pad area, and an inactive pad area. The active pad area is defined as the area that is under leach/irrigation. The inactive pad area is the remaining area of the pad, whereas leach/irrigation was already completed, or is waiting to go under leach.

- Pond Areas:
  - Pregnant Leach Solution (PLS) Pond
  - Raffinate Pond
  - Event Pond

Table 17.8: Leach Pad Flow In / Flow Out		
Type	**Flow In**	**Flow Out**
Active Pad Area	Precipitation; Placed Material Moisture; Raffinate Leach Solution	Evaporation; Residual Material Moisture; Loss due to irrigation
Inactive Pad Area	Precipitation	Evaporation
Pond Area	Precipitation	Evaporation

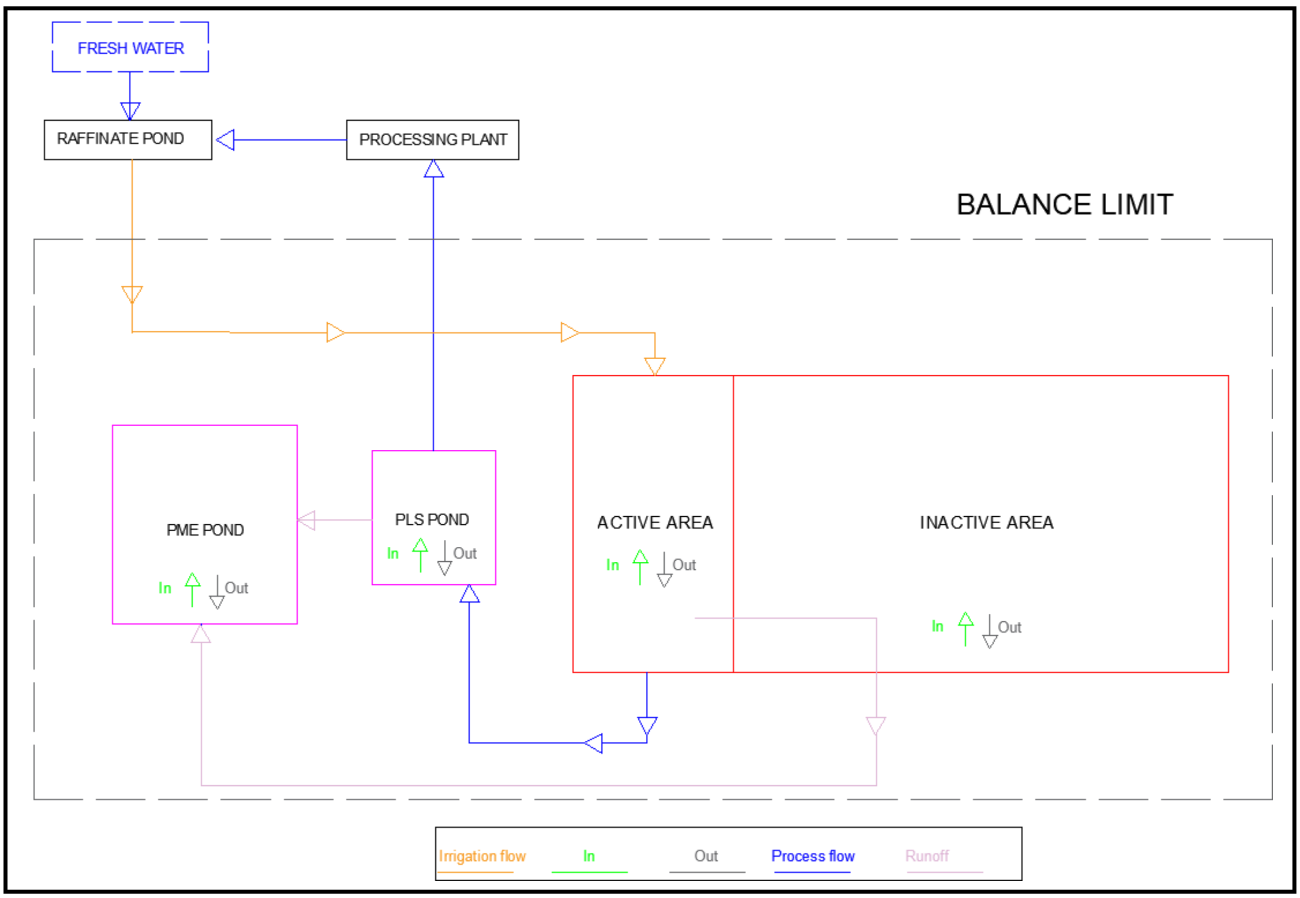


Figure 17.4: Leach Pad Flow In / Flow Out

The management of water in the Pad requires different systems for its operation:

- Subsurface water management and subdrainage system
- Geomembrane liner system
- Leak Recovery and Recirculation System

- Solution collection system
- PLS Pond, Raffinate Pond, Event Pond, and subdrainage
- Surface water management and diversion channels

Water usage for the site was estimated at two different time frames, one from Years 1 through 10 and the other from Years 11 through the life of the operations. This includes maximum and minimum rainfall events to provide an average water usage and make-up estimates for the process plant and leaching areas in Table 17.9.

Table 17.9: Average Process Make-up Water Requirements		
Flow (liters / second)	**Years 1 – 10**	**Years 11 - LOM**
Base Case Option		
Minimum	53.1	69.8
Maximum	125.6	119.9
Average	**87.8**	**111.7**
Alternative Case Option		
Minimum	42.3	79.8
Maximum	93.7	131.2
Average	**56.9**	**106.1**

Evaporation & Solution Losses:

- An average evaporation loss of 830 mm/year over the active leach pad area was considered based on meteorological data from the site station.
- The active heap leach area will be irrigated with a combination of sprinklers and drip emitters across the active pad area pad, on a continuous 24-hour basis based on a 6.11 L/hr-m2 solution rate. Based on experience with this type of equipment and system, an average evaporation of 1% of the total flow to the pad has been considered for the solution losses in the leach pad and trench areas.

17.3.9 Power

For the Base Case, approximately 57 MW of power will ultimately be required for the process facilities. For the Alternative Case, approximately 45 MW of power will ultimately be required. However, the acid plants will provide a portion of the power through steam power cogeneration.

Table 17.10: Projected Process Facilities Average Electric Power Usage					
		Average Consumption, without acid plant		**Net Consumption, with acid plant credit**	
		kWhr/lb Cu	**MWhr/year**	**kWhr/lb Cu**	**MWhr/year**
Base Case	Total Power	1.37	430,000	0.89	273,000
Alternate Case	Total Power	1.37	370,000	0.89	238,000

If the sulfuric acid plant and power lines to site are both down, emergency power will be supplied through two (2) 10 MW diesel generators to continue power to the site for the foreseeable future to avoid weather impacts and maintain operation of critical systems.

17.3.10 Sulfuric Acid

The heap leach acid consumption estimate varies with the tonnage rates processed, types or materials leached (Oxide/LIX and Supergene) and the recovered copper content (grade). The expected average gross acid consumption per tonne of material leached considered is 18 kg sulfuric acid/tonne. Copper plated in the EW operation regenerates 1.54 tonnes of sulfuric acid per tonne of copper plated that is returned to the leaching circuit via the SX operation.

The mine average acid required for operation is based on a gross acid consumption for all materials and net of copper production credits. The life of mine net acid consumption per tonne of material leached is 13.1 kg/tonne and 1.79 kg/lb Cu produced.

All acid will be produced on site through the acid plant. Sulfur prills will be supplied by YPF from their Mendoza based natural gas refinery operations. Life of mine sulfur consumption averages 471 tonnes per day and ranges from 82 tonnes per day to 675 tonnes per day.

Start-up and minor supplemental sulfuric acid will be brought directly to site to make up the difference after maximizing the production of each phase of acid plant.

An additional 2 tpd acid is expected to satisfy electrolyte bleed make-up and all other SX/EW requirements. Most, if not all, of this acid would report to the raffinate pond and be used in the leaching operation.

17.4 ADEQUACY STATEMENT ON SECTION 17

The QP believe the facilities and descriptions of the processing areas are appropriate and consistent with other current operations and studies for similar facilities. Equipment selections are based on vendor consultations and appropriate process modeling. The information is suitable for use in establishing reasonable prospects for eventual economic extraction for the Mineral Resources, the mine plans and financial analysis.

18.0 PROJECT INFRASTRUCTURE

18.1 INTRODUCTION

Project infrastructure includes:

- Access to Los Azules
- Power Supply to Los Azules
- Mine Rock Storage Facility (MRSF) (by Stantec)
- Camp Facilities
- Employee Housing and Transportation
- Water Supply (by Stantec)
- Heap Leach Pads (by KP)

The 2017 PEA considered a bi-national approach to infrastructure for both the Los Azules development and the operational phases. Bi-national opportunities exist with road access, power supply, and cathode or concentrate logistics.

Port facilities for materials and product distribution considered in this PEA include the inland port of Rosario in Argentina and the ports of Valparaiso, Ventanas, San Antonio and Coquimbo in Chile. The Rosario port facilities can be accessed via road or via rail transport from the Cañada Honda rail depot located at the southern outskirts of San Juan.

Power is proposed to come from Tocota, Argentina, north of Villa Nueva.

The city of Mendoza, probable source of fuel and sulfur, is located approximately 200 kms south-southeast (275 road kms) from Calingasta. YPF operates a large 113,200-barrel-per-day crude oil refinery and desulfurization facility just south of Mendoza at Luján de Cuyo.

Mendoza is also the location of the nearest international airport (MDZ). Regional air service is also available from San Juan (UAQ).

The distances from the junction of the National Route (Ruta Nacional) RN 149 with the RN 153 (south of the town of Barreal) to the Chilean ports for copper cathode export are as follows:

- Ventanas: 380 km
- Valparaíso: 410 km
- San Antonio: 440 km

The entire distance is covered by paved roads, except for approximately 37 km of gravel in Argentina, on the RN 149, in Mendoza territory, between the junction with RN 153 (San Juan/Mendoza limit) and Uspallata (of the 55 km between RN 153 and Uspallata, 37 km are gravel and 18 km north of Uspallata are already paved). However, this stretch of gravel is passable all year round and it is highly likely that in the short term it will be paved prior to the project development.

Figure 18.1: Regional Infrastructure (Google 2023)

18.2 ACCESS TO LOS AZULES

The Los Azules Project is currently accessed from San Juan via traveling northward on National Route (RN) 40 for 58 km, turning west on Provincial Route (RP) 436 for 23 km, then continuing west following National Route 149 for 93 km to Calingasta.

Access continues following 120 km of gravel road with eight river crossings and two mountain passes (both above 4,100 m elevation). This route is shown as the Exploration Road in Figure 18.2, below. This access is subject to snow accumulation has not been maintained over the winter. This update describes upgrading and using an existing southern route and a potential future northern access route within a right of way requested by McEwen that is less affected by snow. Also described is an airstrip currently permitted for construction.

The city of Mendoza, probable source of fuel and sulfur, is located approximately 200 kms south-southeast (275 road kms) from Calingasta. Mendoza is also the location of the nearest international airport (MDZ). Regional air service is also available from San Juan (UAQ). Mendoza can be accessed by RN 40 from San Juan or by using RN 149 to RP 52 to reach Mendoza.

The major Chilean population center of Santiago is approximately 270 kms south-southwest (400 kms by well-developed road) from Calingasta.

Port facilities for materials and product distribution include the inland port of Rosario in Argentina and Valparaiso, Ventanas, San Antonio and Coquimbo in Chile. The regional map showing access routes and infrastructure is provided in Figure 18.2. Cathodes or concentrate would travel south on RP 149 to Uspallata and from there to Chile over RN 7 to one of the three ports. A 37 km section of RP 149 is not paved from the Mendoza province border towards Uspallata.

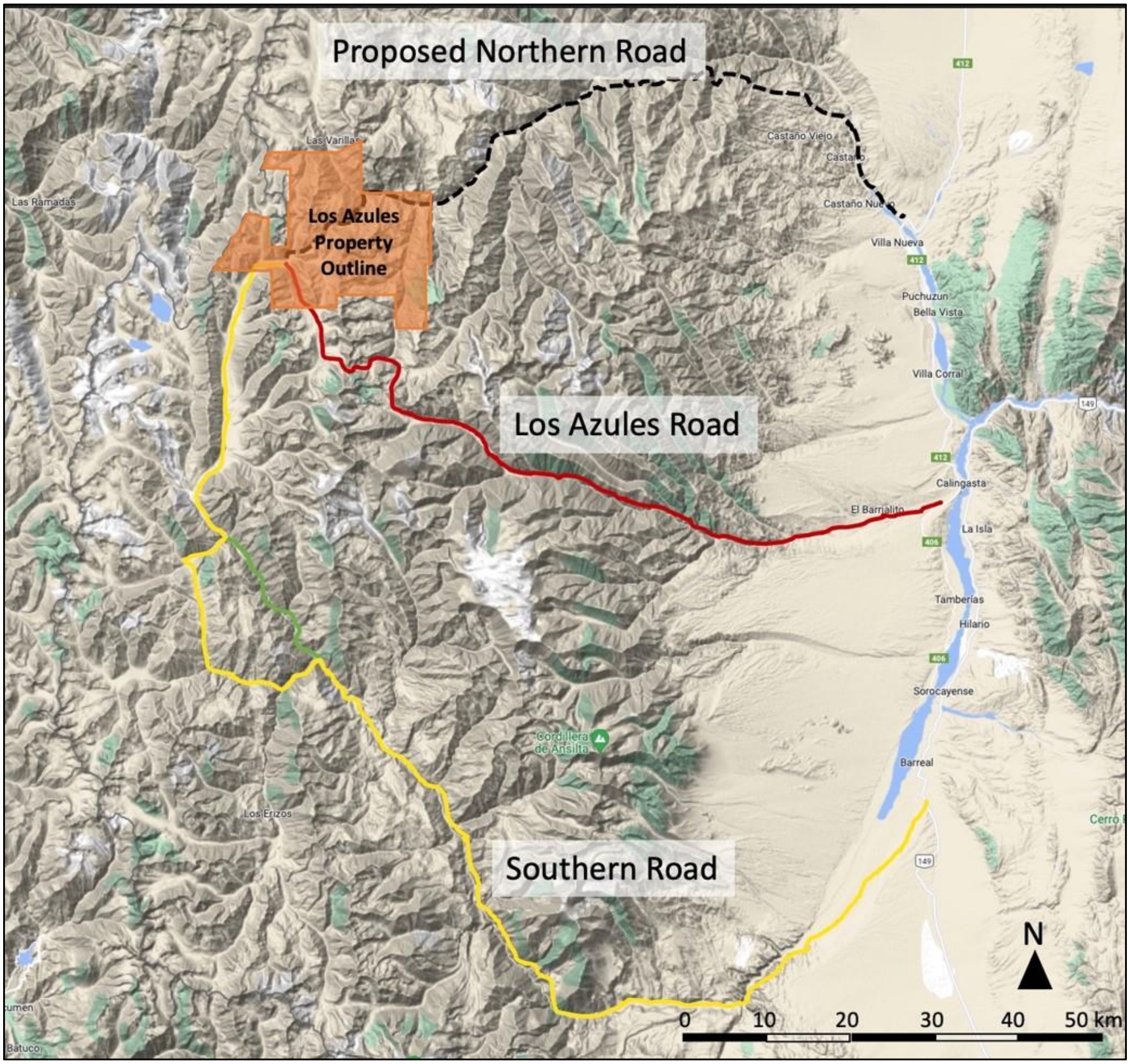


Figure 18.2: Existing Access & Infrastructure (ACMSA, 2022)

Two other access routes are shown in Figure 18.2. The Southern Access Road from Barreal to Los Azules is existing and passable but requires upgrades to make the road passable for haulage of material to site. The Southern Access Road is much longer than either the Exploration Road (124km from Calingasta) or the Northern Access Road (117 km from Villa Nueva) at 192 km from Barreal. The second route is the Northern Access Road which begins in Villa Nueva and has a preliminary design by Ruiz y Asociados of San Juan.

Figure 18.3: Access Roads Photos (McEwen, 2023)

The current site access, using the Exploration Road, which largely runs through the Calingasta river ravine, turns out to be a critical corridor in terms of altitude differences, passing through two narrow passes located at 4,170 m of altitude (Portezuelo de la Totora) and 4,300 m of altitude (Portezuelo Cabeza de León) respectively, to later arrive at the Project location is at 3,390m. This also forces a series of switchbacks with the complexities that this development causes for traffic. The Exploration Road was upgraded in 2022/2023 to allow for access by larger vehicle traffic, including trucks pulling semi-trailers and has provided safer transit as a result. The road will continue to be maintained to provide seasonal secondary site access and support the incoming high voltage powerline routing.

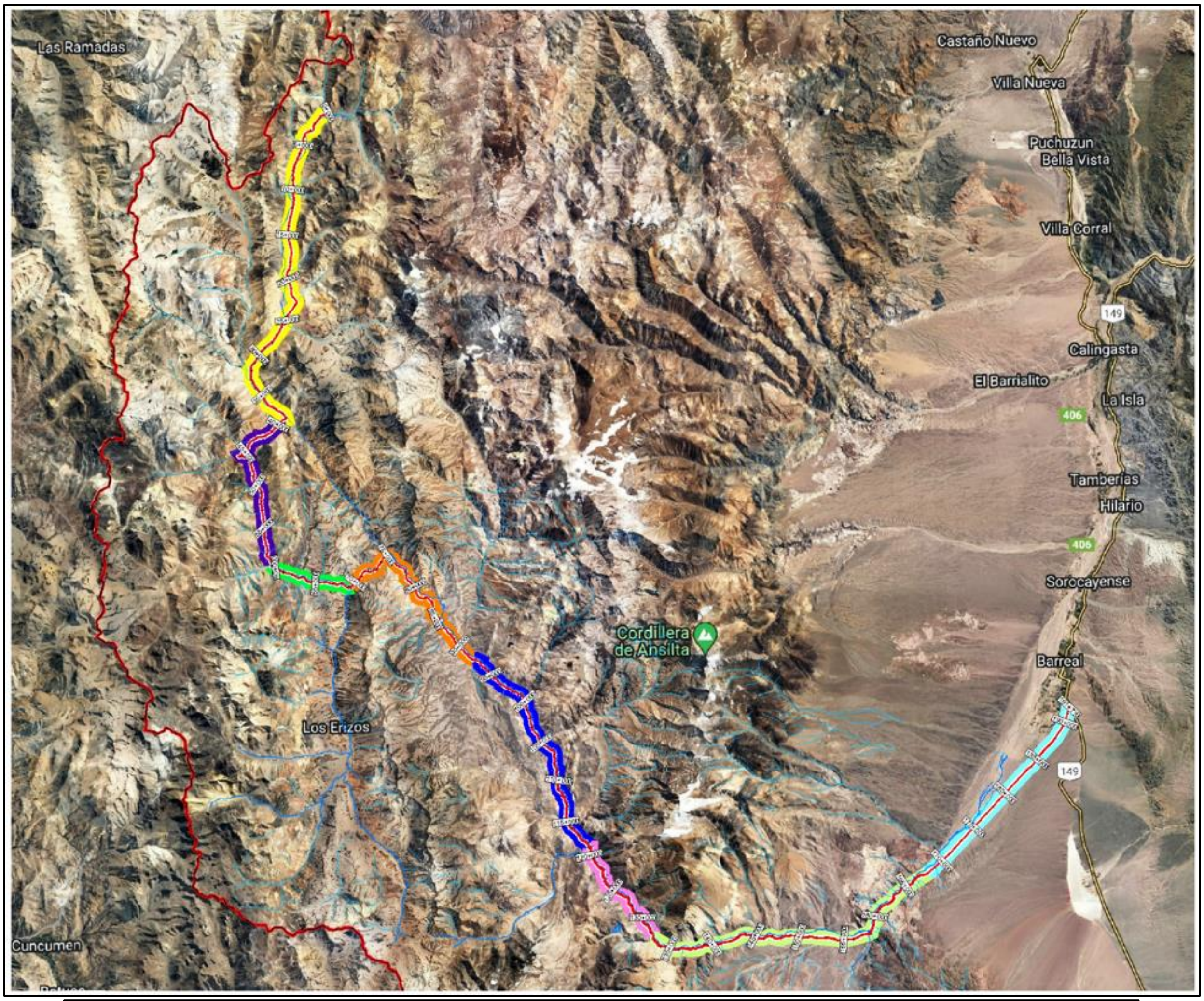
The main access will be by the Southern Road route to begin the project, this road will be upgraded for the expected year-round traffic travel to and from the site and extends approximately 192 km from the town of Barreal. The Southern Route was investigated by Ruiz y Associado Consultoras R.S.L. (RyAC) in 2023 and their preliminary findings, designs and costs were presented in their report, "MAIN REPORT LOS AZULES PROJECT PRELIMINARY CAPEX SOUTH ACCESS ROAD PROV. OF SAN JUAN – REP. ARGENTINA" dated May 11, 2023. The conceptual level designs and costs for the Southern Road upgrade were provided by Ruiz y Associados Consultoras S.R.L. (RyAC) and were estimated to be USD $138 million. These costs do not assume any contribution from other projects in the area. The route is shown in Figure 18.4 below.

The South Access Road corresponds to one of the three corridors that today is considered the Los Azules project to link the road to the state road infrastructure available, mainly to the "North-South Axis" made up of the Andean Corridor (RN 149). The South Access corridor almost entirely runs through a mountainous environment, specifically in the Andean zone of the Department of Calingasta. It presents high complexity in several sectors of its layout, being unprecedented in some sections, for which there are no predictions of its behavior over time (falling ground, instability of slopes, avalanches, etc.).

The Southern Road and Exploration Road elevation profiles are shown in Figure 18.4 below. The southern route is approximately 900 meters lower in elevation at the peaks and presents an overall better profile for continuous operations travel into and from the site.

Figure 18.4: Site Access Road Profiles (McEwen)

The route is carried out mainly in a north-south direction through the Salinas River ravine, up to the confluence of the latter with the Verde River, along about 39 km. At this point, the road turns to the south-west through the Verde River ravine, since it to the Salinas River. This provides an important series of advantages and efficiencies, which needs to be confirmed in terms of its feasibility, through further studies. Consequently, a moderate criterion was assumed, developing the road through the Verde, Colorado, Pantanosa and Santa Cruz River valleys, until reaching the confluence again with the Salinas, where both rivers make up the Blanco River.



	SOUTHERN ROAD UPGRADE PROJECT ROUTE & SECTIONS	Distance (km)
	INTERNATIONAL BORDER ARGENTINA / CHILE	
	SECCIÓN 1 - CAMPAMENTO AZULES - CONFLUENCIA RÍO SALINAS - A° VERDE	38.8
	SECCIÓN 2 - CONFLUENCIA RÍO SALINAS - A° VERDE / CONFLUENCIA RÍO COLORADO - RÍO PANTANOSA	20.7
	SECCIÓN 3 - CONFLUENCIA RÍO COLORADO - RÍO PANTANOSA / CONFLUENCIA RÍO PANTANOSA - RÍO SANTA CRUZ	9.7
	SECCIÓN 4 - CONFLUENCIA RÍO PANTANOSA - RÍO SANTA CRUZ / CONFLUENCIA RÍO BLANCO - RÍO DE LOS SOMBREROS	22.8
	SECCIÓN 5 - CONFLUENCIA RÍO BLANCO - RÍO DE LOS SOMBREROS / CONFLUENCIA RÍO BLANCO - RÍO BRAMADERO	26.8
	SECCIÓN 6 - CONFLUENCIA RÍO BLANCO - RÍO BRAMADERO / EL MOLLE	13.7
	SECCIÓN 7 - EL MOLLE / LAS CALETAS	35.5
	SECCIÓN 8 - LAS CALETAS / EMPALME RN 151	23.7

Figure 18.5: Southern Access Road Route & Design Basis Sections (RyAC, 2023)

The planned route in this sector implies a development of approximately 37 km instead of the 21km that would be required to continue along the Salinas River. To do this, it runs along the Verde River until it ascends to the Urrutia pass (3,310 masl), and then descends along the Colorado River to the confluence with the Pantanosa River, advancing in the direction of this river until its confluence with the Santa Cruz. The journey through these three rivers has the characteristic that it is carried out in general through narrow ravines, with sections of unstable slopes, presenting alternating rocky sectors and ones with uncompacted soils, and the presence of lowlands near the rivers in the lower portions of the valleys. Finally, after the confluence between Pantanosa and Santa Cruz, the path runs through a wider ravine corresponding to the latter, up to the junction of the Salinas and the Santa Cruz, which constitutes the source of the Blanco River. This last segment is already part of the access easement to El Pachón, so that in certain sectors there is a road with a more developed platform, while noting that this road in general has not been provided with protection against erosive effects, which are required in such an environment.

The corridor continues along the Blanco River, corresponding to RP 402 according to the Dirección Provincial de Vialidad (Provincial Directorate of Roads (DPV)) name, passing the El Molle area, and the crossing with the Colorado River, arriving at the "Blanco – Los Patos" confluence, where the name of the route corresponds to RP 400. From this sector, continuing parallel to the Los Patos River, in descent, and after passing through a rocky sector called "Las Caletas" you arrive at Barreal and RP 149.

To reliably estimate the cost for the developments, it was necessary to subdivide the total corridor into eight (8) "Sections" each with relatively homogeneous characteristics in relation to the surrounding environment. Section 1 to Section 4 will be the most affected by winter aspects (snowfall, avalanches, temperatures, etc.). As one descends through the corridor, the potential erosion in the river valleys is magnified due to the increase in the supply drainages and basins and the potential runoffs prone to be generated by locally common torrential rains, especially during the summer months. To complete the analysis, specific investigations must be addressed in corridors of this nature, including the imprints of the cryogenic geoforms and periglaciers of the "Cuenca Río Blanco" established by the Argentine Institute of Nivology, Glaciology and Environmental Sciences (IANIGLA), for the purpose of verifying potential interferences with the route under study, so that they do not occur.

This analysis of the route focused mainly on aspects of soil movement, structural package and the drainage requirements of the work, as they are the most significant items in these types of infrastructure, also attending to the relative stabilization works (walls, stonework) as well as safety works (escape ramps, curbs, etc.), and complementary works in general, all of which is detailed throughout the RyAC report.

Table 18.1: Southern Access Road Upgrade Estimate (RyAC, 2023)

Southern Road Upgrades	**USD $**
Direct Cost	$84,890,387
Indirect Cost	$46,881,950
Total	**$131,772,337**
Profit 5%	$6,588,617
Project Cost	**$138,360,954**
Total Length (Km) Sections 1-8	**191.7**
$/Km (Direct Cost)	**$442,829**

The northern route is a shorter, much lower route which traverses the narrow Atutia river ravine and has only one pass to traverse at 3,800 m of altitude. Construction of this route will require the construction of a pioneer road to allow more accurate surveying of the terrain and support construction of the final alignment. One advantage of constructing the northern route is that it could support a pass to Chile and for that reason, provide for the recapture of the capital cost of construction against future royalty payments. This route has been presented to the Provincial Highway Directorate, which supports additional access to the remote upper area of the mountain range.

18.3 POWER SUPPLY TO LOS AZULES

Power will be supplied from the Argentinian grid via an initial 220 kV overhead transmission line approximately 120 km long and tie into the existing utility substation at Calingasta. Due to the altitude, every overhead line must be built for a higher nominal voltage (e.g., a 500 kV line operated at 220 kV voltage level). The lower air density reduces the breakdown voltage of the air and makes it necessary to increase the distance between phases, which in turn increases the reactance of the line. Thus, voltage drop becomes the main design criterion. The transmission line route will be parallel to the existing Exploration Road to Los Azules site and construction of 220kV/175MVA GIS substation at the site.

Initially the Base Case project will require approximately 89 MW, increasing to 131 MW as the processing facilities are expanded and mine power requirements increase over time. The Alternative case requirements increase to an initial 71 MW increasing to 117 MW.

The expansion of the existing Rodeo and Calingasta 500 kV substations connected to the national grid will also be necessary. The work will be carried out under the initiative and responsibility of National power provider YPF Luz. The estimated YPF investment is USD $155 million for their installation of the powerline and substation facilities, including the site substation. The breakdown of the preliminary costs provided by YPF Luz are:

- Rodeo 500 kV Substation: USD $50 million
- Calingasta 500 kV Substation: USD $35 million
- Calingasta – Los Azules 220 kV line (120 km): USD $60 million
- Los Azules 220 kV GIS Substation: USD $10 million

YPF Luz has provided a referential rate sheet for a Power Purchase Agreements (PPA) contract to obtain power sourced exclusively from renewable resources. The power would be sourced from solar and hydroelectrical generation. The rate would depend on the contract term and would be a take or pay contract with a minimum power contract under payment. A rate of $0.065/kWh based on a minimum 10-year term is considered in this PEA.

YPF Luz has also signed a memorandum of understanding to include the installation of the sub-station at Calingasta and transmission line to the site. Cost recovery specifics are still pending, however a 3% interest on capital and overall, 5% IRR scheme has been developed to approximate this aspect of the investment cost recovery while preserving the base power rate and included in the project financials. A commercial proposal is under development by YPF Luz.

Emergency power will be supplied through two (2) 10 MW diesel generators to provide power to the site for the offices, camps and to maintain operation of critical systems. Solar power augmentation and energy storage options will be evaluated in the next phase of study.

Additionally, the on-site sulfuric acid plants will be outfitted with a steam cogeneration power plant for electricity generation from the heat and off-gasses. This will provide approximately 20% of the site requirements during operation. On-site solar power supplemental systems at the main camp and other facilities will also serve to minimize the grid power usage.

18.4 MINE ROCK STORAGE FACILITY (MRSF)

The Mine Rock Storage Facilities (MRSF's) include the North MRSF and the South MRSF which will be used to store mined out overburden and excess rock, and a low-grade stockpile. A plan view showing the locations of the proposed MRSF's is presented in Figure 18.6 with corresponding cross-sections presented in Figure 18.7. The locations of the MRSF's were selected with consideration given to the mine layout plan, haulage distances from the pit, natural slopes confinement, and foundation suitability. The locations of the MRSFs were also adjusted to avoid cryogenic geoforms and to maintain offsets from the pit and other critical infrastructure.

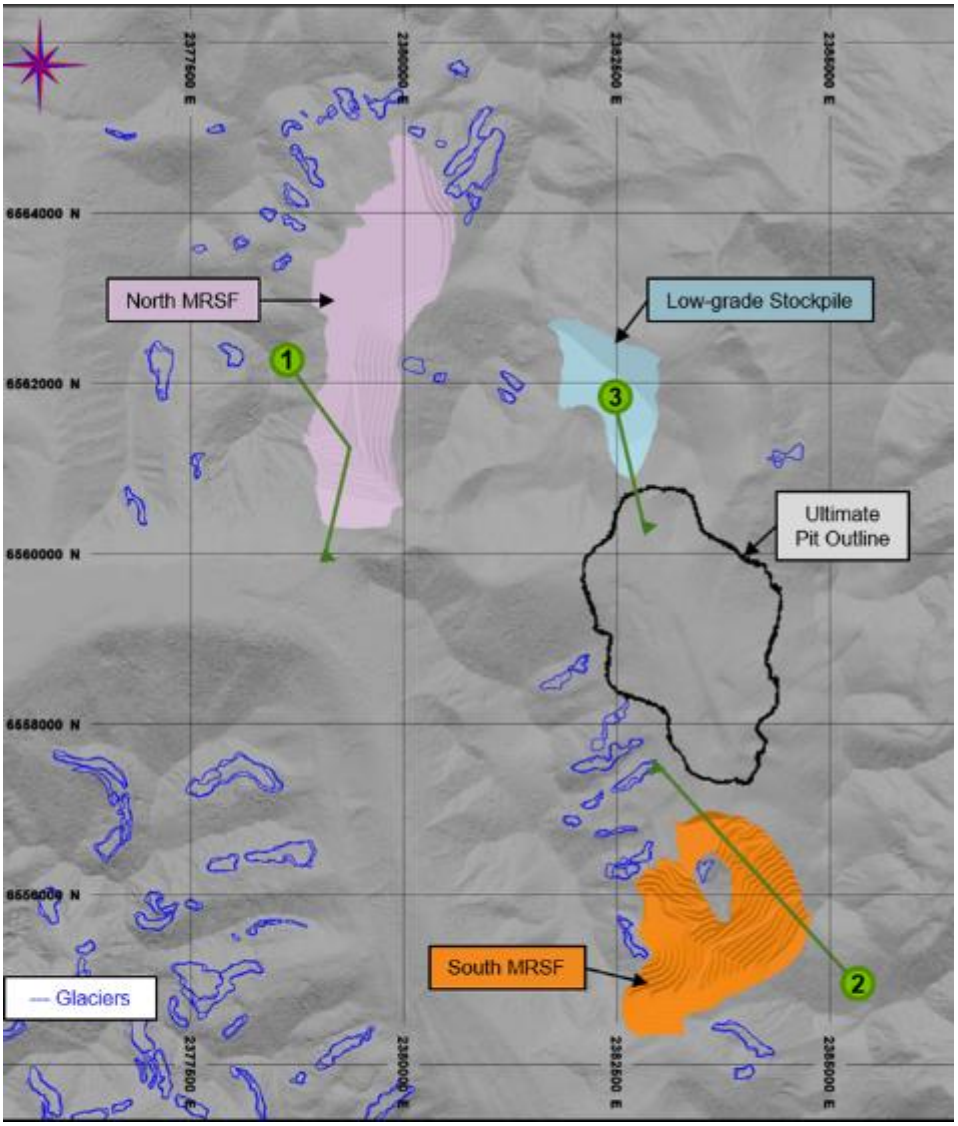


Figure 18.6: Proposed MRSF's Plan View (showing cross sections)

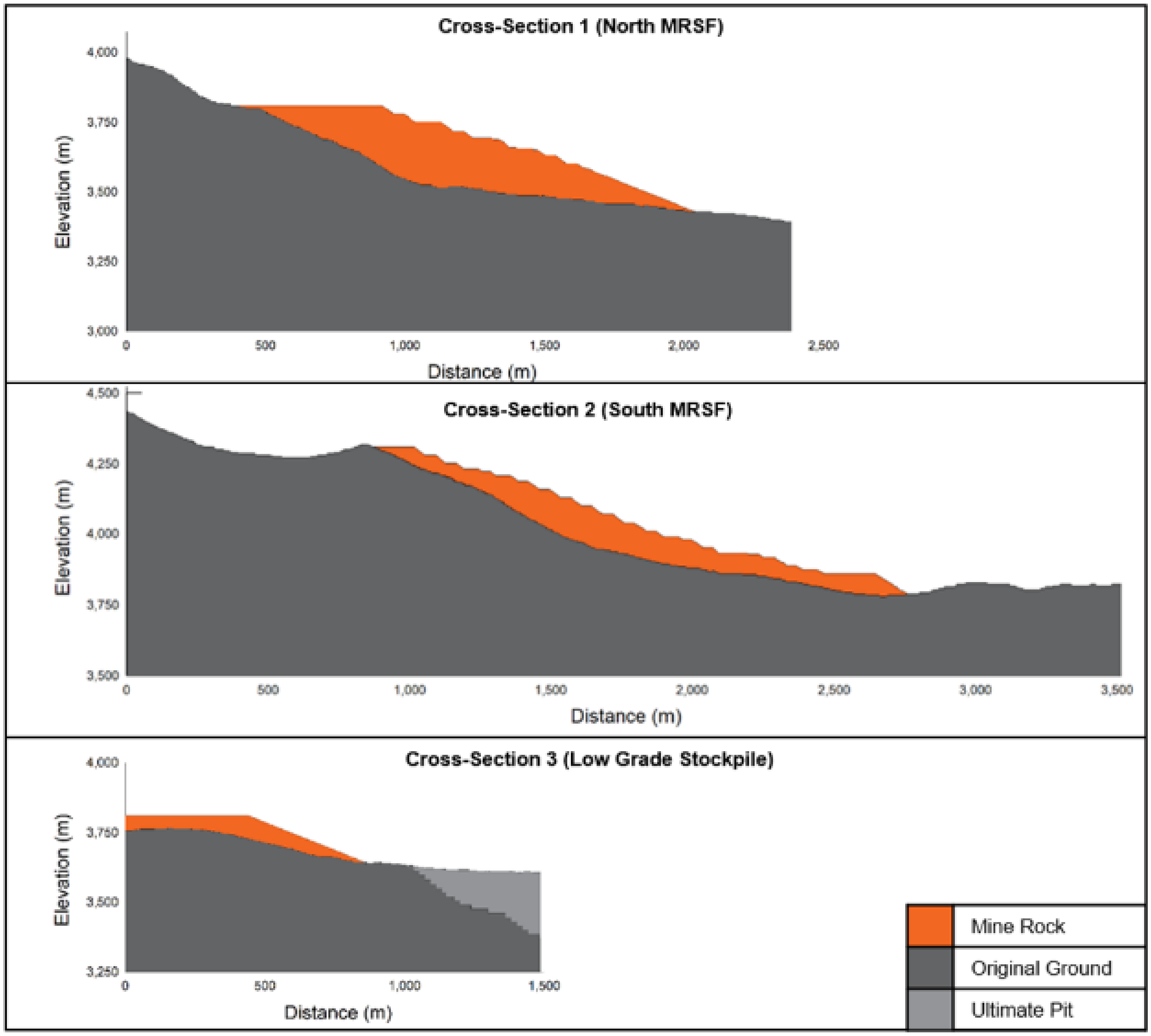


Figure 18.7: Proposed MRSF Cross-Sections

The geotechnical foundation conditions for the MRSFs were interpreted based on information available from historical drilling programs and surveys (1998 to 2018), and the on-going field investigation. The surficial conditions in the MRSF areas are dominated by thick glaciofluvial soil deposits along the valley floors which thin out towards the valley side slopes. The soil stratigraphy includes till (as moraines and other glacial deposits), colluvium and alluvium underlain by igneous rocks. Wetlands are scattered throughout the lowest areas of the valley floors.

Geotechnical slope stability analyses for the MRSFs were completed using representative cross-sections selected based upon a review of the proposed geometries, foundations, probable mine rock thickness isopachs, piezometric conditions and potential impacts of failure. The results demonstrate that with overall slopes of 2.5H:1V, the proposed MRSFs meet the target acceptable factor of safety (FoS) of 1.30 for static and 1.05 for seismic loading.

The MRSFs will primarily be founded on flat valley floors and will be advanced up the native valley slopes using the bottom-up construction method where possible. Catch benches will allow for collection of flowslide, rock rollout, deformation or other instability while protecting downslope infrastructure. Benches will further provide the required space for possible closure re-sloping and creating geomorphic landforms in the future to achieve long term stable slopes which may be revegetated if required. For adequate long-term stability, a controlled construction methodology will be adopted, such as using sectors with low quality mine rock placed at the back of the facilities and high-quality rock placed at the front (slope face). Performance monitoring will be carried out to assess and manage the MRSF stability.

To minimize MRSF contact water, surface water from catchment areas outside of the MRSF footprints will be collected and diverted around them using diversion channels when feasible. Subdrainage systems are also planned which will collect and channel surface runoff and subdrainage within and through the MRSFs and towards contact water collection ponds. Surface water from catchment areas upstream of the MRSF's footprint which cannot be diverted will drain into and through the MRSFs in the subdrainage system. Based on the current layout, one large subdrain following the alignment of the natural valley drainage will be required for the North MRSF. The South MRSF and the low-grade stockpile will each require sets of two smaller subdrains. Foundation preparation for the construction of the subdrainage systems will include pre-stripping, particularly in the wetland areas, to achieve competent foundations and acceptable grade for construction. Similarly, wetland areas located upslope of the subdrainage systems may also be pre-stripped as needed to reduce the potential for fines transport and blockage of the drains. Humic materials removed from the wetlands because of pre-stripping will be stockpiled for future closure and reclamation use.

18.5 CAMP FACILITIES

This subsection was prepared by Jason McLennan, McLennan Design. McLennan Design is a wholly owned subsidiary of Perkins + Will, Inc.

Initially, the project will use the existing site all-weather modular camps and construction camp to begin operations. These facilities can accommodate up to 2,500 workers, with seasonal constraints to 2,000 workers. Portions of the existing and future construction facilities will be periodically occupied as needed to support ongoing construction activities at the site.

A projected camp staffing is shown in Table 18.2. All employees and contractors will be housed in the Los Azules camp facilities.

Table 18.2: Projected Camp Staffing Requirements

Site Camp Planning			
	Initial	**Year 16**	**Ultimate**
Mine	240	506	1105
Process	170	218	218
G&A Site Staff	107	107	107
Contractors	150	150	150
Camp Ops/Services	249	357	560
Visitors/Executive	30	40	50
Spares	50	50	50
Total Rooms	**996**	**1428**	**2240**

The Los Azules permanent site camp will house and support approximately 1,000-2,500 workers at any time, with the initial camp for 1,000 and flexibility to accommodate larger as needed. Images of the camp are shown below in Figure 18.8 and Figure 18.9. The permanent camp is currently planned for construction to support occupancy in year 5 of the project.

Figure 18.8: Mine Camp Concept - Isometric view showing Solar Arc

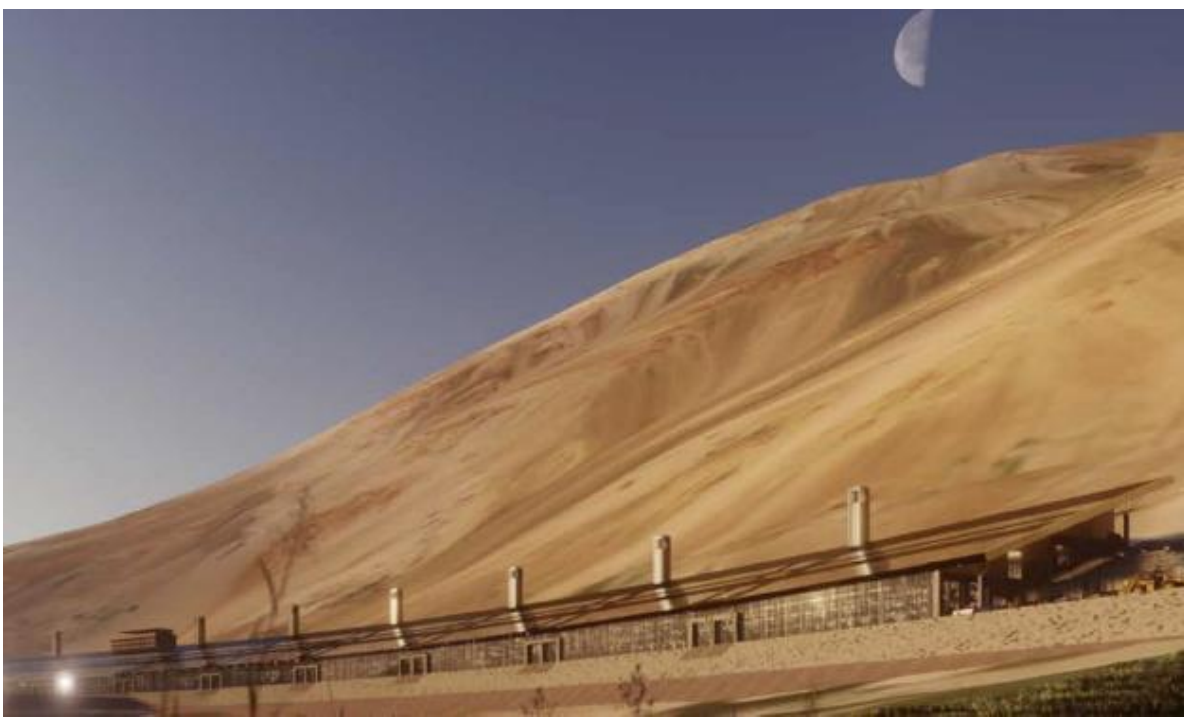


Figure 18.9: Mine Camp Concept - Oblique View

To accommodate alternate staff demands, the permanent camp is designed for scalability and can be configured to house more than 1,000 employees if needed in various ‘neighborhood’ groupings organized in a linear fashion within the facility. Creating camp conditions that are exceptional will reduce absenteeism and help McEwen maintain a productive, happy, and engaged workforce where jobs are coveted and retention high.

In addition to the regular shift workforce, the camp can expect to receive frequent visitors, including Specialists, executives, and suppliers, who will be residing within the camp as well. Future estimates on visitor counts will determine the number of surplus accommodations being designed and an on-site ‘hotel’ is provided for both flex accommodation as well as visitors who wish to experience a once in a lifetime location.

The mine camp has been strategically located to optimize multiple variables. Worker safety, comfort, well-being, as well as the distance from the mine operations and access to the main road are major considerations.

In addition, the specific layout and orientation have been selected to support passive design, and solar energy generation, which are key considerations. The selected site is located near the operations, at an elevation of 3,400 meters. The camp conforms to the north slope of a valley, to ensure full sun exposure out of the shadow of other mountain peaks for most daylight hours. By being located on the other side of a ridge from mining operations, both sound, vibrations, and dust will be minimized, thereby improving the worker’s comfort and safety.

‘Leave no trace’ and ‘pack it in, pack it out’ have been mottos of wilderness exploration for decades. Bringing this mentality to the camp’s design has played a fundamental role in shaping the material selection and end-of-life plans. Innovative design techniques have replaced conventional camp materials with historical and place-based alternatives that will mitigate the lasting impacts of the site’s presence.

The concept is rooted in eliminating the concept of waste, by turning used materials back to their original state so that they can disappear back into the ecosystem they came from.

There are a variety of parameters that must be evaluated when selecting camp materials, operational and embodied carbon analysis measures the global warming potential of each material from harvest through disposal and quantifies its performative impact during operations. Material toxicity is a chronic issue in the construction industry. Typical construction materials lack ingredient disclosure and transparency which results in the use of endocrine disrupters, carcinogens, volatile organic compounds (VOC), and many other serious risks to human and ecosystem health. At the end of the camp’s life, all biodegradable materials will be easily turned into the soil, to disappear into the landscape within a few years, or will be disassembled for re-sale and re-use.

Disassembly and re-use have been key variables during early design. Selecting materials and assemblies that can easily be churned back into the site, or disassembled into reusable parts, will enable a swift, and efficient end-of-life process. The main goal is to divert all camp materials from the landfill, thereby eliminating the concept of waste. To do this we are employing the following major construction materials and assemblies for the camp:

Strawbale Construction

This centuries-old construction technique uses agricultural byproducts as thermal insulation and structural support. A wood frame can be integrated into the strawbale or similar materials construction if desired, although substantial literature is available showing the structural integrity of strawbale construction. Once in place, an earthen plaster is applied to both sides to ensure lasting preservation and fire-resistance of the strawbales. Strawbales can be locally sourced from the San Juan region, supporting local farmers. At the end of the camp's life, the strawbale can be broken down and composted and integrated into the soil for greater soil health.

Sealing and finishing the strawbale walls will be accomplished with earth plaster. Earth with high sand and clay composition is ideal for the creation of earth plaster walls. When mixed with water - and if desired, a binding agent, the earth forms a thick paste that can be applied to interior and exterior walls for an aesthetic, plaster-like finish. Standard gypsum plastering includes materials with a high global warming potential (GWP) and potential health hazards. By using materials that can be sourced from the site itself, earth plaster has no GWP and is completely non-toxic. At the end of the camp's life, deconstruction will crumble the plaster back into sediment on the site – once again, leaving no trace.

Gabion Assemblies

One material that is practically limitless at Los Azules is rock. Gabion construction will take medium-sized rocks from around the site and turn them into easily constructed caged rock walls – called gabion walls. By using rocks found on the site, we can dramatically reduce the global warming potential (GWP) by eliminating a large quantity of concrete that would need to be extracted, manufactured, and transported to the site. When the camp closes, the metal cages will be removed and reused or recycled, leaving the rocks right where we found them. Leaving no trace at the site or for our climate.

Prefabricated Containers & Room Boxes

The camp will be utilizing prefabricated and pre-finished dorm units and other modules which will slot into the pre-laid substructure to then be enclosed by the superstructure. Prefabricated units can be made off-site using local labor and materials, these units will then be brought up to the mountain for fast, low-skill, installation. Prefabrication increases efficiency while decreasing cost and material waste.

Large Span Systems

An expansive spaceframe, ETFE, or cable-based frame is being engineered to enclose the camp, providing rainwater capture, solar energy generation, and climatization. Steel is a material with one of the highest GWPs per volumetric unit. By eliminating many of the structural columns and using minimal members, the overall quantity of steel required to span such a large distance is dramatically reduced. The large span system will be sized for standard dimensions so that at the end of the project's life it can be disassembled neatly and transported down the Andes for resale and re-use.

Net Positive Energy

Renewable energy is a core pillar of today's environmental paradigm. The Camp at Los Azules will surpass what is expected by becoming fully net-positive for energy. The camp will be designed, first, to reduce energy consumption as much as possible with passive design and efficient appliances. Next, the camp will offset consumption with the leading technology in renewable energy production and storage. Lastly, the camp will be connected to a distributed energy grid, which is powered by 100% renewables, ensuring uninterrupted power 24/7.

The Los Azules camp and mine will be forming a microgrid in one of the most remote locations on the planet. This self-sufficient energy system will serve the entire site's occupied footprint and will facilitate load stabilization through energy storage technology and carefully timed consumption. Even though the camp will be connected to offsite energy production, it is being sized for net-positive energy production, making it a candidate for International Living Future Institute's (ILFI www.living-future.org) Net Zero Energy certification. The ILFI has created the world's most rigorous green building standards.

The camp will be integrating energy-saving measures that will drastically reduce the development's energy use intensity (EUI). Relying exclusively on LED lighting and energy-star appliances will ensure that energy is being rationed responsibly. Automatic and user controls are imperative in allowing for optimal thermal comfort and efficiency. Additionally, orienting the project towards the sun, to take full advantage of the sun for daylighting, direct heat gain in the winter, water heating, and other passive strategies. All this combined with optimally engineered thermal envelopes, to reduce heat loss, will significantly cut the camp's energy loads, thus saving money on production and storage infrastructure.

The camp will reduce heating and cooling loads through passive design, which will take advantage of the sun's energy and natural ventilation to create an idyllic interior climate without energy-consuming mechanical systems. Key techniques include Passive solar, heat stack ventilation, earth tubes, thermal mass, and insulation.

The camp will use materials with a high thermal mass potential, including concrete, compacted earth floors, pools of water, and gabion walls. These materials slow the transfer of heat acting as a space temperature regulator. During the day, the sun's rays are absorbed into the mass, which at night will continue to radiate heat. During the summer, interior spaces will be thoroughly flushed with a cool night breeze. By the end of the night, the mass has cooled off again and will absorb more heat which otherwise could overheat the space.

The importance of daylight control can be found in the life support systems section. Additional heat will be provided for smaller spaces, including individual units, although their loads will be reduced.

Lower-density hot air will naturally rise in a space resulting in an upward airflow current. Enclosed space can optimize this effect by adding operable vents on the side of the structure facing the direction of the breeze. These vents will let in cooler outside air, will rise through the space as it heats up due to solar and internal gains. By the time the air reaches the top of the building, it is hot and ready to be expelled. Placing stacks on the opposite end of the building from the louvers will help flush hot air as needed.

Earth tubes draw fresh air from the outside, filter it, and then run it through tubes that are buried at least 1 meter below the ground plane. The earth acts as a thermal mass, thereby regulating the temperature of the air before it passes through the building's envelope. During the summer, the air will be cooled, and in the winter, the air will be warmed to above freezing temperatures. Earth tubes can be combined with heat stack ventilation to provide a constant supply of fresh, temperate air.

The South Wall of the Mine Camp facility will feature a super insulated, straw bale wall assembly for high thermal performance coupled with a highly air-tight assembly overall.

On-site renewable energy generation for the camp will come from an expansive solar super roof that has been integrated into the biosphere shell. Strategic placement of the PV will ensure that the right balance of sunlight is reaching the camp and greenhouse areas, while a large portion of the roof is generating electricity. Extensive solar modeling and analysis will optimize for direct, strategically located sun exposure and maximum energy efficiency.

Local utilities have provided an economically attractive source of 100% renewable energy using a combination of solar and hydroelectrical. EPSE and YPF are strategic partners for the provision of net positive energy on an annual basis. Due to weather, altitude, and transportation difficulties, installing wind turbines on the site is not optimal. By connecting to a centralized, renewable grid, the mining camp can take advantage of a wider variety of energy production, ensuring grid stabilization. While the camp is being designed to be fully self-sufficient and produce more energy than it consumes, having a connection to a diversified energy source will increase the camp's resilience.

Finally, the camp will have on-site energy storage which will allow it to store surplus energy that is generated during sunny hours. This energy can be used to regulate the camp's reliance on the grid so that camp usage does not overlap with peak mining loads and to ensure resilience in the case of an outage. An extensive investigation of energy storage systems has informed the storage strategy of the camp. Battery storage is by far the most space-efficient storage system on the market. Other attractive options for energy storage include iron or sodium batteries, flywheel energy storage, and innovative gravity storage using rock waste. Further evaluation of energy storage needs will be finalized once energy use reduction and production have been fully modeled and calculated.

The camp will also be designed to provide heating and climate control, acoustics, medical and support services including recreation and medical clinic, improved air quality using living plant systems, and water management to capture rainwater and snowmelt, retain that water and treat it for reuse using natural systems. The camp will pursue ILFI certification based on the alignment with the Living Building Certification "Water Petal".

The camp will be designed to provide space for growing food in a self-sustaining environment. Finally, the camp will provide waste management systems to provide reuse of waste materials, either through direct reuse, recycling, composting and elimination of single-use plastics and packaging.

18.6 TRANSPORTATION

Transportation of employees to the various worksites will be provided by on-site buses and light vehicles. All management will have assigned light vehicles. Lunchtime will be taken on board mobile equipment or at designated lunchroom facilities established at each major facility location during the construction phase. Transportation between the Los Azules site and the city of San Juan for staff embarking on, or returning from work rotation, will be by company aircraft or otherwise by a bus service.

A longer-term solution for Los Azules is a fly-in-fly-out operation from San Juan airport to Los Azules. Flight time is anticipated to be approximately one hour. The workers coming into rotation arrive to the San Juan airport at 05:00 and commence their shift per normal start time at site. On finishing the shift at the end of rotation the workers report to the airport and are home early that same evening.

The Los Azules development is in the high Andes mountains of western Argentina. To the southwest of Los Azules and downstream of the proposed processing facilities the Rio Salinas Valley broadens and straightens. It is of very low gradient and suitable for formation of an airstrip to service Los Azules. The airstrip will be at an altitude of approximately 3,250 masl, 8 km from the proposed camp/offices facilities along the southern access road route.

A detailed topographic survey was performed during 2017. Airstrip design was completed later in 2017. A site inspection confirmed the geotechnical condition is glacial outwash sands and gravels. The survey and geotechnical inspection indicated airstrip formation works are without complexity and suitable construction materials are immediately available at the site by screening of in-situ materials.

The airstrip permit had been applied for to the Argentina authority (ANAC) since approximately 2015 and granted in 2019 and still in force. Initially a STOL (Short Take Off and Landing) permit has been requested. The permit will allow construction of an airstrip and for planes such as a DH-6 to land at Los Azules and support the exploration, permitting and early implementation phases of the Los Azules development.

A longer-term vision is to utilize larger aircraft at Los Azules such as a Dash 8 type personnel transport aircraft and potentially a C-130 Hercules Transport. These aircraft will require a longer airstrip and a future extension to the airstrip is anticipated. The extension is to enable larger aircraft types to use the airstrip facility and will require an easement over the affected part of the property to the south of Los Azules lands.

A serviceable airstrip at Los Azules delivers the following critical and useful outcomes that, collectively, significantly enhance the exploration, implementation, and operating phases of Los Azules development. These include:

- Enables an emergency evacuation and a specialist emergency response.
- Safer personnel travel by aircraft than buses on mountain roads.
- Flying time of less than one hour from San Juan compared to at least 7 hours road travel by bus.
- Enables exploration drilling activities beyond the reliable summer 3-month snow free weather window affecting the high passes and extends the safe drilling season to between nine and 12 months.
- Will enable accelerated timely in-fill drilling and exploration drilling for the mine pit to be drilled from indicated status up to measured status.
- Can be used to supply the Los Azules development if the access road is unserviceable such as from an un-cleared snow fall event.
- Facilitates fly-in-fly-out work rotations per world best practice.
- Facilitates air freight support during development phase.
- Facilitates deliveries of critical air freight needs during development and operations.

Navigation aids, safety and security evaluations and flight simulations are pending for operations. A preliminary costing for the STOL airstrip formation at Los Azules in 2017 was less than $5M and the airstrip formation works can be completed within a single summer season.

18.7 WATER CONSUMPTION

Surface water is available on the property in adequate amounts for McEwen Mining's exploration activities. Preliminary hydrological and meteorological evaluations have indicated sufficient water exists for the proposed Los Azules mining and processing facilities and to provide the necessary fresh water needed to house employees at the mine site. For the Base Case the estimated average consumption is approximately 113 liters per second (L/s) in the initial years and 163 L/s in the later years as mining progresses and processing throughputs increase. For the Alternate Case the estimated average consumption is approximately 95 liters per second (L/s) in the initial years and 152 L/s in the later years as mining progresses and processing throughputs increase.

The leaching process for recovering copper was selected, in part, due to the lower water consumption. This was in line with our guiding principles to minimize our impact on the environment. Selecting a hydrometallurgical process option for Los Azules would reduce effective water usage by 75% to 80% over a concentration alternative. Given this context, the most appropriate technology selection for Los Azules to minimize water usage is a hydrometallurgical approach, which is the basis for the project development in this Technical Report. Additionally at Los Azules, alternatives for improving precipitation/snow capture, site dust control, reuse/recycle and passive water treatment strategies are being developed and included in the project design.

Net water consumption considers averaged annual precipitation expectations, available capture areas, evaporation losses and heap leach moisture retention. Key assumptions used in estimating water usages are shown in Table 18.3.

Table 18.3: Key Assumptions for Water Usage Estimates			
Water Assumptions	Units	Value	Source
Pond Area	m^2	219,632	Knight Piesold Design
Minimum Annual Rainfall	mm/year	63	From Stantec - 12/5/22 report
Maximum Annual Rainfall	mm/year	475	From Stantec - 12/5/22 report
Irrigation Rate	$L/h/m^2$	6	Average
Leach Material Feed Moisture	%	3%	Average
Terminal Moisture	%	8%	Test work estimate
Irrigation Rate Losses - Losses, plant, etc.	%	1%	Assumption
Evaporation Rate	mm/year	830	From Stantec - 12/5/22 report
Dust Control	L/s	43.5	From Stantec - averaged over the entire year
Camp Use			
	L/d/person	75.0	Nominal Estimate
	L/d/person	100.0	System Design Estimate

The estimated net water consumption for the project by usage source is presented in Table 18.4 below for the Base Case and Alternate Case Phase 1 project options.

Table 18.4: Life of Mine Average Water Consumption by Case			
Water Usage by Major Area During Operations - Life of Mine Average			
		Base Case	**Alt. Case**
Mine Area – Shops, truck wash	L/s	2.0	2.0
Processing			
Process Plant & Offices	L/s	2.0	2.0
Leaching	L/s	100.9	78.7
Site Areas			
Camps	L/s	1.5	1.5
Administrative	L/s	0.5	0.5
Dust Control			
On-Site Mine, Roads	L/s	3.8	3.8
Off-Site Roads	L/s	39.7	39.7
Other, Miscellaneous	L/s	3.2	3.1
Make-up Water Required	**L/s**	**153.5**	**131.3**
Year 1-5 Average	L/s	**112.9**	**95.2**
Year 1-10 Average	L/s	**137.3**	**103.9**
Year 11-27 Average	L/s	**162.9**	**151.9**

18.8 WATER SUPPLY

The fresh water available at Los Azules from natural surface streams that progressively confluence to form the Rio Salinas and groundwater from proposed pit dewatering operations exceeds the projected water consumption demands of the project development and mining operation phases.

The envisaged mine dewatering holes for lowering the level of the groundwater level around the mine pit will also deliver water. Future dewatering of the actual mine pit sumps will also be a source of water, even if it is contact water.

Based on an initial project development including copper heap leaching and copper SX/EW processing the preliminary hydrological and meteorological evaluations have indicated sufficient water exists for the proposed Los Azules mining and processing facilities and to provide the necessary fresh water needed to house employees at the mine site estimated to average approximately 113 liters per second (L/s) in the initial years and 163 L/s in the later years are mining progress and processing throughputs increase. A more detailed evaluation of available water resources will need to be undertaken for an IIA submission.

Surface water flowing from the Los Azules development is all contained within a single watershed. Meteorological and watershed analysis (Stantec, 2022) estimates average annual surface water flows 438 L/s in the Rio Salinas exiting the Los Azules watershed with calculated average monthly surface water flows ranging from low of 92 L/s in March to 1316 L/s in August. Over the recent 5-year drought period estimated average annual surface water flow exiting the Los Azules watershed is 275 L/s. These estimates include surface availability over the entire watershed. Potential losses or gains due to surface diversion structures or storage facilities are not accounted for in these estimates. A photo of the Rio Salinas is shown in Figure 18.10.

Long-term mine dewatering estimated for the current pit configuration is 525 L/s (Section 16.8). Previous dewatering estimates ranged from 600 L/s to 800 L/s (Hatch, 2017; Ausenco, 2011) A summary of potential water supply sources is shown in Table 18.5. The combined groundwater and surface water availability estimates for this PEA are in surplus to the 600 L per second required for long term make-up water and the surplus is further augmented when mine area dewatering water is considered.

Table 18.5: Estimated water supply by source.

Source	Rate	Comments
Average SW flows	438 L/s	Stantec, 2022
Estimated pit dewatering	525 L/s	Current PEA Ch. 16.8
SW flows – drought conditions	275 L/s	Stantec, 2022
Pit dewatering estimate – upper range	800 L/s	Ausenco, 2011
2017 pit dewatering estimate	600 L/s	Hatch, 2017

The Los Azules development has available surface water and groundwater resources exceeding the water demand. To manage the excess water, non-contact water such as stream flow water and dewatering bore water will need to be managed by a network of stream diversions, contour channels and pipes. This will deliver the surplus non-contact water back into the environment at a point downstream of the active project site.

Figure 18.10: The Rio Salinas at the Proposed Campsite.

It is recommended that more studies into water management are performed as a part of baseline environmental assessments, including:

- Ground-water level measurement over a whole year, including winter.
- Stream flow gauging measured over a whole year.
- Permeability testing in the area to be dewatered around the mine pit to assess probable dewatering extraction volumes and to confirm dewatering water quality is suitable as non- contact water.

A detailed contact water / non-contact water management plan needs to be developed to support the IIA permitting process including the design and location of water diversion structures, and the staged formation of any contour channels. This will be further supported by an engineered project water balance. At the same time as the IIA Application submission, Los Azules will apply for the water rights and the associated water use permits where Los Azules is granted to have beneficial use of its water rights.

18.9 HEAP LEACH PADS AND PONDS

18.9.1 Introduction

The total initial pad design has a capacity of 956.6M tonnes, a future pad expansion is anticipated to accommodate an additional 225M tonnes for the final total material requirement of 1,182M tonnes. The possibility to increase the nominal heap height is also under consideration. The leach pad develops over time inside a valley with an upstream growth as prescribed in **Error! Reference source not found.** and Figure 17.4: Leach Pad Flow In / Flow Out. Pad ultimate height of 150m is not considered extreme for design purposes and collection system integrity.

The leach pad has a subdrainage system whose objective is to evacuate the subsurface flow from groundwater and avoid the generation of pressure below the liner system and prevent damage to the geomembrane. The system is composed of perforated double-walled corrugated high-density polyethylene (HDPE) pipes of variable diameter between 200 mm to 450 mm in diameter placed in trenches with a minimum depth of 1.20 m. The pipes are arranged forming a network that joins the points of lower elevation of the foundation area of the stack. Normally the pipes are located on streams and floodplains draining the flow to a pool (subdrainage) located at the foot of the stack.

The lining system is installed above the subdrainage system and is composed of a 2 mm thick HDPE geomembrane, installed on a layer of soil of low permeability (minimum 30 cm), which also functions as a second liner. Complementary to the liner system, the pad has a potential leak recovery system, identified as the "leak recovery and recirculation system," which collects leaks that may pass through the upper lining system. The collected flows are conducted to a sump, to be repumped to the leach pad. This system keeps the potential infiltrated flows under control, minimizing the hydrostatic load on the secondary geomembrane and thus reducing the risks of leaks into the environment.

The leach pad also has an over liner collection system composed of a network of 100 mm diameter double-walled corrugated HDPE pipes (drainage pipes) and 300 mm and 450 mm diameter conduction pipes that collect the PLS and gravity flow directly to the PLS Pond.

The design criteria details are presented in Table 18.6.

Table 18.6: General Design Criteria

Parameter	Unit	Value
Initial Designed Stacking Capacity	Mt	957
Ultimate Capacity Required	Mt	1,182
Number of Stages	-	9
Stacking Height	m	150
Minimum slope of the stack base	%	2.5
Overall Slope	H:1V	3.0
Local Slope	H:1V	2.0
Bench Width	m	10.0
Stacking Layer Height	m	9
Maximum Irrigation Area	ha	128
Leach Material Permeability	cm/s	0.01
Storm Precipitation 100 years	mm/day	68.10
Storm Precipitation 1000 years	mm/day	135.30
Initial Material Moisture (from Mine)	%	3.0
Leaching Moisture	%	8.0
Residual Moisture	%	5.0
Drain Down Time	hours	100
Bomb Stop	hours	24

18.9.2 General Configuration of the Heap Leach Pad

The Initial Pad develops inside a valley with an upstream growth. It seeks to have a differentiated management of non-contacted water, both surface and subsurface runoff, and contacted water, associated with the irrigation of the stack. Differentiated water management requires different systems that make up the Pad. In relation to these set objectives, the conceptual engineering considers the following systems for the operation of the Pad:

- Subsurface water management and subdrainage system
- Geomembrane system

- Filtration recovery and recirculation system
- Solution collection system
- Rich solution basins, major events, and underdrainage
- Management of surface waters and diversion channels

The leaching pad has a subdrainage system whose objective is to evacuate the subsurface flow from groundwater and avoid the generation of sub pressures below the lining system and damage the geomembrane. The system is made up of perforated double-walled corrugated high-density polyethylene (HDPE) pipes of variable diameters between 200 mm and 450 mm in diameter, excavated in trenches with a minimum depth of 1.20 m. The pipes are arranged to form a network that joins the lowest elevation points on the foundation area of the stack. Normally the pipes are located on streams and vegas, draining the flow towards a sink (sub-drainage) located at the foot of the pile.

Above the subdrainage system, the lining system has been arranged, consisting of a 2 mm thick HDPE geomembrane, installed on a layer of low permeability soil (minimum 30 cm), which also works as a second lining. This configuration is aligned with international cyanide management regulations. It should be noted that the system covers the entire length of the Pad, including the pools.

As a complementary part of the lining system, the Pad has a potential leak recovery system, identified as the “filtration recovery and recirculation system,” which collects any leaks that may pass through the lining system. The collected flows are conducted to a sump, to be re-pumped to the leaching pad.

Any flows are collected and recirculated through a pumping system back to the stack. This system keeps potential infiltrated flows under control, minimizing the hydrostatic load on the secondary geomembrane and thus reducing the risks of seepage into the environment.

Lastly, the Pad has a collection system made up of a network of 100 mm diameter corrugated double-walled HDPE pipes (drainage pipes) and 300 mm and 450 mm diameter conduction pipes that collect the solution. rich and conduct it by gravity towards the PLS pool. From here the solution is pumped to the process plant.

For the management of surface waters, a set of channels have been arranged that border the perimeter of the Pad. These channels collect runoff water or those produced by snowmelt at the head of the basin, and on the sides of the Pad, discharging them downstream of the Pad, towards the Frío River.

18.9.3 References

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19.0 MARKET STUDIES AND CONTRACTS

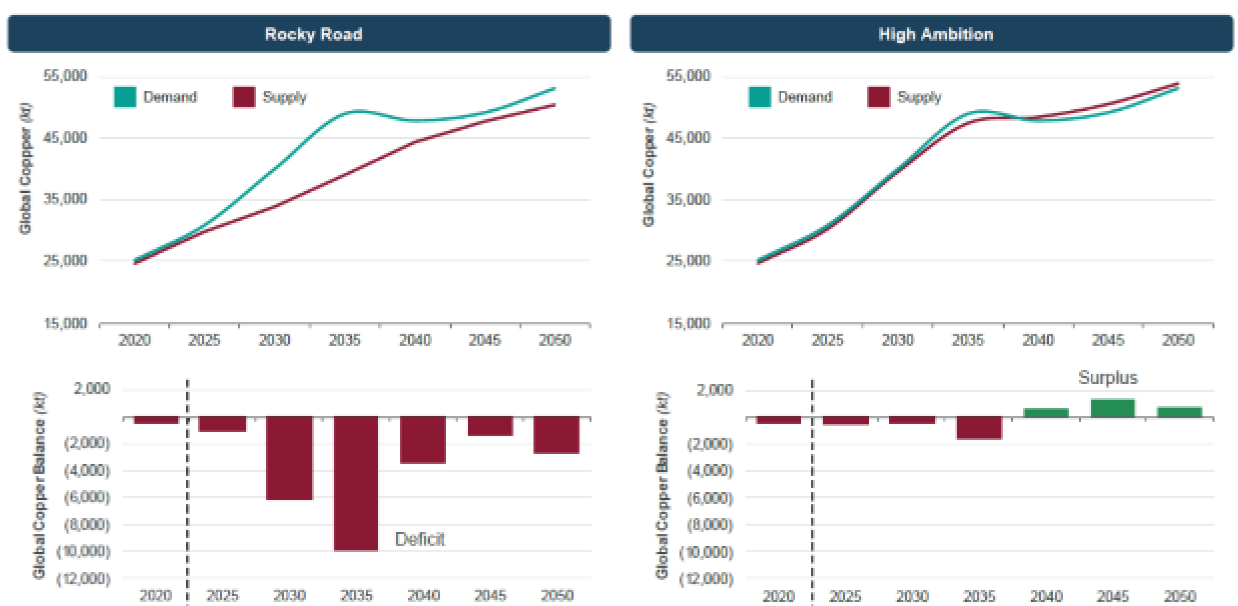
The PEA for the Los Azules deposit is based on initial production of copper cathode from heap leaching for the 31 years. Potential recovery of gold and silver depends on the use of alternate leaching techniques from the copper leaching residues or flotation to produce concentrates.

19.1 COPPER MARKET OUTLOOK – SUPPLY VS DEMAND

The global copper market continues to be one characterized by a paucity of copper orebodies in the project pipeline and worldwide demand continuing to grow. A typical project takes between 15 and 20 years to move from discovery to production and on average a decade from the completion of a feasibility study to ramp-up to full production. This delay in putting projects into production is due to technical, regulatory, or social issues.

Copper consumption is expected to increase over time, greater than the historical rate of 3% per year, due to increased consumption for renewable power technologies and fuel cell and battery electric vehicles. Renewable power from wind turbines, solar photovoltaics, fuel cells, and power storage technologies and transmission and distribution will require additions to supply to provide these increases. The increased use of fuel cell and battery electric vehicles are an additional source of demand. These vehicles use between 30 and 435 kg of copper per vehicle, versus today's internal combustion vehicle, which uses 24 kg. These changes in power generation, transmission, and distribution, as well as the end uses are critical to meet decarbonization goals set by the International Energy Agency.[5]

The market forecasts by S&P Global[6] range between a modest deficit of supply to a significant deficit, ranging between the use of current or a best-case use of technology, copper recycling, and mine/refinery capacity utilization ("Rocky Road" vs "High Ambition" forecast), The best-case scenario assumes highly optimistic advances overall to achieve that production level. A comparison of the Rocky Road and High Ambition forecasts of supply, demand, and are shown below in Figure 19.1, copper market outlook – supply.



[5] https://openknowledge.worldbank.org/bitstream/handle/10986/38160/CMO-October-2022.pdf
[6] https://www.spglobal.com/marketintelligence/en/mi/Info/0722/futureofcopper.html

Figure 19.1: Future Copper Market Demand Scenarios (from S&P Global)

19.2 COPPER MARKET OUTLOOK - PRICES

Projected copper prices to be used in the economic analysis are based on a range of prices including consensus projections and an economist view of the long-run steady state copper spot price.

Consensus projections for the long-term copper prices range between $3.25 and $4.25 per pound of copper with a mean price of $3.75 per pound of copper. This is reflected in the following table with the price projections and the date of those projections, below.

May 5, 2023

Copper *(US$/lb)*

Date	Firm	2023	2024	2025	2026	LT
01-May-23	CIBC	$3.69	$3.75	$3.85	$3.55	$3.55
28-Apr-23	JP Morgan	$3.94	-	-	-	$4.00
27-Apr-23	Canaccord	$4.02	$4.25	$4.50	$4.50	$4.25
26-Apr-23	Raymond James	$4.24	$4.00	-	-	$3.50
26-Apr-23	Paradigm	$4.02	$4.25	$4.00	$3.75	$3.75
25-Apr-23	Deutsche Bank	$3.99	$4.08	$4.54	$4.37	$4.08
25-Apr-23	RBC	$4.00	$4.00	$4.50	$4.50	$3.50
25-Apr-23	Morgan Stanley	$3.90	-	-	-	$3.42
24-Apr-23	Credit Suisse	$3.98	$3.80	$4.00	$4.20	$3.50
24-Apr-23	BofA	$4.28	$4.48	$4.76	$4.36	$3.25
24-Apr-23	BMO	$3.95	$3.63	$3.63	$3.97	$3.95
23-Apr-23	National Bank	$3.80	$3.80	$3.65	$3.65	$3.65
22-Apr-23	Jefferies	$4.10	$4.75	$5.25	$5.50	$4.00
21-Apr-23	Stifel	$4.02	$4.25	$4.50	$4.00	$4.00
21-Apr-23	TD	$4.02	$4.10	$4.25	$4.50	$3.75
20-Apr-23	Eight Capital	$4.00	$4.25	$4.50	$4.25	$3.75
20-Apr-23	Barclays	$3.88	$3.48	$3.25	$3.40	$3.30
19-Apr-23	Cantor	$3.25	$3.25	$3.25	-	-
18-Apr-23	Desjardins	$4.01	$4.00	$4.15	$4.25	$4.25
18-Apr-23	HSBC	$3.96	$3.70	$3.75	$3.80	$3.30
18-Apr-23	Scotia	$4.00	$4.00	$4.50	$5.00	$4.00
17-Apr-23	UBS	$3.89	$4.00	$4.00	$4.00	$3.50
14-Apr-23	Cormark	$3.85	$3.75	$3.75	$3.75	$3.75
14-Apr-23	Haywood	$4.00	$4.25	$4.25	$4.25	$4.25
12-Apr-23	Berenberg	$4.00	$3.97	$3.97	$3.86	-
11-Apr-23	BNP Paribas	$3.73	$3.86	$4.08	-	$3.86
28-Mar-23	Macquarie	$3.95	$3.39	$3.56	$3.86	$3.63
Average		**$3.94**	**$3.96**	**$4.10**	**$4.15**	**$3.75**
					Minimum	$3.25
					Maximum	$4.25

Figure 19.2: Long-term Copper Pricing (CIBC, May 2023)

The long-run steady state copper price as estimated by an economist uses the information in the current forward market for the metal, along with a mathematical model of how prices move towards the long-run price over time from a current base price, to project the future.[7] The steady state copper price is estimated using the Laughton-Jacoby model.

Inputs for the copper price model are weekly spot prices traded in LME, and the base price used was as of December 1, 2022. The nominal long-run price is estimated to be $3.81 per pound. The real or deflated dollar copper price in 2033 is expected to be $2.85.

The price selected to use for the financial model was $3.75 per pound.

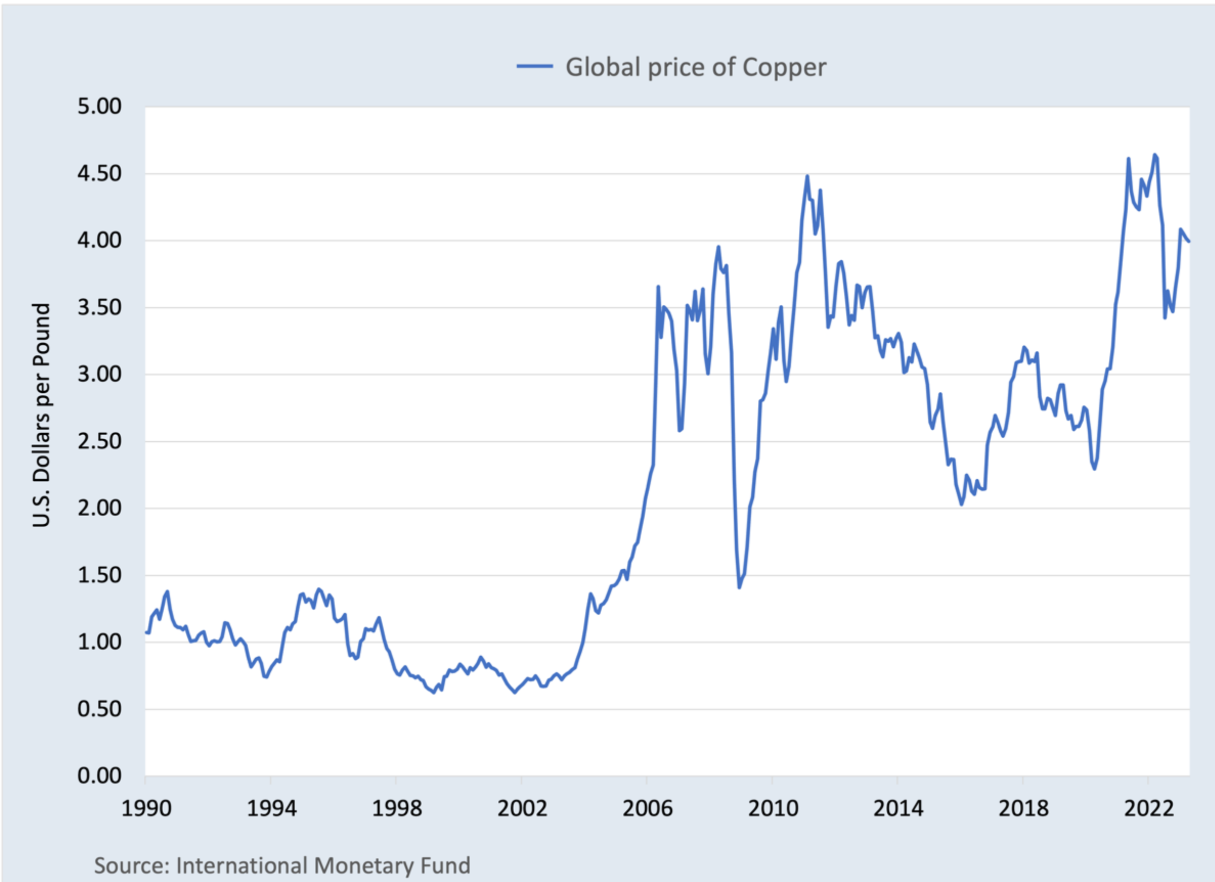


Figure 19.3: Copper Prices 1990 to Present (source: International Monetary Fund[8])

[7] , Davis, G, 2022, Copper and Energy Price Forecasting for the Los Azules Copper Project Preliminary Economic Assessment, internal report for McEwen Copper

[8] https://www.imf.org/en/Research/commodity-prices

19.3 PRECIOUS METAL PRICES

Projected precious prices to be used in the economic analysis are based on consensus projections.

Consensus projections for the long-term gold prices range between $1,355 and $2,000 per troy ounce of gold with a mean price of $1,641 per troy ounce of gold. This is reflected in the following table with the price projections and the date of those projections, below.

Similarly, consensus projections for the long-term silver prices range between $17 and $26 per troy ounce of silver with a mean price of $21.53 per troy ounce of silver.

The prices selected to be used for the Mineral Resource estimate were $1,700 per troy ounce of gold and $20 per troy ounce of silver.

May 5, 2023

Gold *(US$/oz)*

Date	Firm	2023	2024	2025	2026	LT
01-May-23	CIBC	$1,810	$1,700	$1,650	$1,650	$1,650
28-Apr-23	JP Morgan	$1,963	-	-	-	$1,600
27-Apr-23	Canaccord	$1,984	$2,100	$2,176	$2,227	$2,262
27-Apr-23	H.C. Wainwright	$1,750	$1,750	$1,750	$1,750	$1,750
27-Apr-23	M Partners	$1,952	$1,900	$1,900	$1,900	$1,900
27-Apr-23	Laurentian	$1,750	$1,750	$1,750	$1,750	$1,750
26-Apr-23	Paradigm	$1,860	$1,750	$1,650	$1,650	$1,650
25-Apr-23	Deutsche Bank	$2,009	$2,300	$2,550	$2,141	$1,800
25-Apr-23	RBC	$1,735	$1,700	$1,700	$1,650	$1,600
25-Apr-23	Morgan Stanley	$2,031	-	-	-	$1,355
24-Apr-23	BofA	$2,009	$2,011	$1,808	$1,805	$1,800
24-Apr-23	Credit Suisse	$1,888	$1,700	$1,650	$1,650	$1,500
24-Apr-23	BMO	$1,905	$1,709	$1,675	$1,615	$1,500
23-Apr-23	National Bank	$1,825	$1,825	$1,750	$1,675	$1,675
22-Apr-23	Jefferies	$1,897	$1,900	$1,900	$1,750	$1,500
21-Apr-23	Stifel	$1,961	$1,813	$1,750	$1,750	$1,750
21-Apr-23	Raymond James	$1,810	-	-	-	$1,600
21-Apr-23	Cantor	$1,960	$2,000	$2,000	$2,000	$2,000
20-Apr-23	Eight Capital	$1,900	$2,000	$1,900	$1,800	$1,700
20-Apr-23	Barclays	$1,930	$1,775	$1,650	$1,600	$1,500
19-Apr-23	TD	$1,954	$2,000	$1,850	$1,800	$1,700
18-Apr-23	Desjardins	$1,973	$2,000	$1,900	$1,775	$1,775
18-Apr-23	HSBC	$1,905	$1,820	$1,725	$1,732	$1,600
18-Apr-23	Scotia	$1,904	$1,900	$1,700	$1,700	$1,700
17-Apr-23	UBS	$1,985	$2,075	$1,956	$1,775	$1,600
14-Apr-23	Cormark	$1,800	$1,800	$1,800	$1,800	$1,800
14-Apr-23	Haywood	$1,945	$1,975	$1,900	$1,900	$1,900
11-Apr-23	BNP Paribas	$1,970	$2,060	$2,100	-	$1,600
29-Mar-23	Berenberg	$1,922	$1,950	$1,825	$1,650	-
28-Mar-23	Macquarie	$1,885	$1,775	$1,800	$1,900	$1,500
Average		**$1,906**	**$1,890**	**$1,843**	**$1,784**	**$1,690**
					Minimum	$1,355.00
					Maximum	$2,262.00

Figure 19.4: Long-term Gold Pricing (CIBC, May 2023)

19.4 PAYABLES, TREATMENT AND REFINING CHARGES

Table 19.1 shows the market assumptions that have been used for the economic analysis. The QP's believe the metal price assumptions in this report are appropriate and consistent with other current studies and are suitable for use in establishing reasonable prospects for eventual economic extraction for the Mineral Resources, the mine plans and financial analysis.

Table 19.1: Market Assumptions for Financial Analysis

Parameter	Unit	Value
Copper Concentrate Grade	%	29.5
Long Term Copper Price	$/lb	3.75
Long Term Gold Price	$/oz	1700
Long Term Silver Price	$/oz	20
Payable Cu	%	96.5
Payable Au	%	90
Payable Ag	%	90
Treatment Charge	$/t concentrate CIF	80
Penalties for impurities	$/t concentrate CIF	0 (no penalties anticipated)
Cu Refining Charge	$/lb payable Cu	0.08
Au Refining Charge	$/oz payable Au	8
Ag Refining Charge	$/oz payable Ag	0.5
Concentrate Transportation	$/wmt concentrate	125

19.5 MINERAL RESOURCE ESTIMATE

A long-term copper price of $4.00 per pound, gold price of $1,700 per troy ounce and silver price of $20 per troy ounce were used for constraining mineable shapes, input to cutoffs, and for establishing reasonable prospects of eventual economic extraction for Mineral Resources. Using a higher long-term price assumption for Mineral Resources helps ensure that the Mineral Resources within the PEA mine plans are a subset of those resource and gives more flexibility to the mining engineer when determining the cutoff used. The use of a higher price assumption for Mineral resources has become a common industry practice.

19.6 MARKETING

No copper concentrate marketing plan has been developed at this stage. It is expected that the Los Azules cathodes will be sold Free on Board at the Los Azules project site, and that concentrate will be a high-quality saleable concentrate and is intended to be exported through a port already exporting copper concentrates. Exporting the Los Azules copper concentrates through an existing copper concentrate handling port has potential for cooperation, blending and value adding, however this is not yet considered. Marketing to Europe will have an advantage in the current political environment as they are currently looking for sources to replace Russian copper concentrates.

19.7 CATHODE OR CONCENTRATE TRANSPORTATION

Copper cathode transportation is considered in Section 18. Product is expected to be sold FOB Los Azules, with the buyer taking responsibility for transportation from site. In the future, a novel leaching technology or a copper concentrator could be considered for processing the underlying primary copper mineralization.

19.8 CONTRACTS

No contracts are in place related to the refining, handling, sales and hedging, transportation of supplies or products, and forward sales contracts are currently in place.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Sections 20.1 and 20.6 have been prepared by Maria Paula Martinez, Knight Piésold A.C.S.A. (KP) based on background information from studies conducted from 2012 to date and the exploration environmental impact report along with its updates, available since 2010. Section 20.2 was prepared by Rob Bowell PhD, C. Chem, C. Geo, P. Geo (NL), SRK Consulting UK Limited. Sections 20.3 to 20.5 have been prepared by W David Tyler, Registered Member SME, McEwen Copper.

20.1 ENVIRONMENTAL BASELINE STUDIES

Baseline studies to date include surface and groundwater quality, flow measurement, climate, flora, fauna, limnology, air quality, archeology, geology, geomorphology, and glacier characterization. All baseline data collection, except for meteorological data, has been conducted during late spring, summer, and early fall due to the difficulty of accessing the site in winter.

The results of the baseline studies were documented in the exploration IIA (2010), and in subsequent updates, 1st biennial exploration IIA update (2012), 2nd biennial exploration IIA update (2014), 3rd biennial exploration IIA update (2016), 4th biennial exploration IIA update (2018), 5th biennial exploration IIA update (2021) and 6th biennial exploration IIA update (2023). The environmental and social baseline is currently being prepared to complete the IIA for the exploitation of the Los Azules Project.

In general, the study of each of the components has been considered in two parts: the first corresponds to the area containing the mineral deposit, where the mine and associated infrastructure will be developed, known as the Mine Area, and the second describes the geology of the area over which the access road to the site is laid out, from the town of Calingasta to the valley of Río Frio, called Access Road.

20.1.1 Geology and Seismicity

The baseline study for the geological component has been developed for both the Mine Area and the Access Road.

The exploration carried out at Los Azules since 1998 has resulted in the discovery of a significant porphyry copper deposit. The property is located within the highly prospective sub-belt of central Chile, which in turn is part of the Miocene porphyry copper belt of the central Andes of South America.

This deposit corresponds to a porphyry copper system formed by a series of Miocene sub-volcanic intrusive phases hosted in a contemporary diorite stock, which in turn is in a Permo-Triassic volcaniclastic complex. The Los Azules deposit is hosted by a calc-alkaline pluton at least 7 km long (north-south) and >2.5 km wide.

Porphyries of rhyodacitic composition of phyllonian type, and to a lesser extent (volume), the hydrothermal magmatic breccias, are the rocks causing the mineralization. All these bodies are located along a regional fault system referred to herein as Las Vegas- Piuquenes, which is intercepted by a northwest trending structure, known as the Los Azules System.

From a geochemical point of view, the Los Azules samples are derived from high oxidation hornblende-bearing chalcoalkaline magmas and are more mafic than the reference material. In addition, all units have typical volcanic arc signature.

On the other hand, the area defined as Access Roads, begins at the foot of the Precordillera morphostructural unit, continues westward over the Frontal Cordillera and finally enters the main mountain range. The outcrops of the geological units are mostly arranged in a north-south direction, controlled by the structuring occurring in the region and subsequent periglacial reworking.

In this area where the access road is located, there are outcrops of different lithologies, among which the Choiyoi Group volcanites and the granitic-granodioritic bodies of Triassic age predominate. These constitute the structural base of the region. The rest of the outcrops correspond to Mesozoic and Cenozoic sequences, in which sediments and volcanites are interspersed. The Mesozoic sequences are associated with the deportation and filling of back-arc basins. The Quaternary deposits cover the above by means of periglacial, colluvial, alluvial, etc. deposits.

Regarding seismicity, the baseline study shows the results of the analysis, interpretation, and description of the seismological characteristics in the two study areas, Mine Area and Access Roads. The site in question is located at the contact between the Frontal Cordillera and the Main Cordillera. The seismic activity in the Cordillera is notably less intense and frequent than in the Precordillera and Sierras Pampeanas.

The study area is surrounded by several seismic sources. The seismotectonic framework of the project area is dominated by the subduction of the Nazca plate beneath the South American plate, and the active tectonics of the Western Cordillera.

In the Western Precordillera, the most tectonically active structures are: the Barreal-Las Peñas transpressional belt (150 km long, north-northwest trending, made up of staggered mountain blocks, bounded by reverse faults and shear zones, with sinistral strike-slip components); the Jarillal-Ansilta fault, which is reverse (approximately 17 km long, with a north-south orientation, west dipping, and with a scarp to the east); and the El Alcázar fault, which is reverse (15 km long, with a general north-south strike and a sunken western block).

The convergence between the Nazca and South American plates causes numerous intermediate depth earthquakes in the horizontal subduction sector. A smaller number of earthquakes occur within the continental crust, at depths between 1 and 50 km, with events being more dangerous than the deep ones.

The main seismogenic sources located within a 100 km radius of the Mine Area and Access Road were analyzed. Some structures in Chilean territory were added (such as the Domeyko Fault System), which constitute potential seismogenic sources for the area under analysis, in terms of secondary effects such as mass wasting processes. It should be noted that, in all cases, the available data on the ages of the deposits and last movement, slip rates and recurrence intervals are extremely scarce, except for the El Tigre fault.

The faults located in the project area are strongly masked by the deposits originated by the intense cryoclastism, which totally or partially cover evidence of faulting (scarps, natural exposures, etc.), making their analysis difficult. The North and North Camp faults have sections with lengths of less than 5 km, so they are considered unlikely to generate an earthquake capable of producing a surface rupture. The Diagonal-La Ballena fault is more than 5 km long and is therefore considered to be the most important seismogenic source in the project area, within Argentine territory. About 15-20 km west of the project, in the Chilean sector (close to the international border) there are two N-S fault sections assigned by several authors to the Domeyko fault system, of sinistral character, with a subordinate vertical component (possibly west-dipping reverse). This structure is also an important seismogenic source, capable of producing a 6.4 to 6.8 magnitude earthquake. Probably the greatest damage in the event of a high magnitude earthquake will be caused by mass wasting processes (falls, landslides, and avalanches) due to the thick debris cover, steep slopes and periglacial environment which favor such instability on the slopes, and secondly by seismic shaking. Most of the important seismogenic sources (considered seismogenic sources of moderate-high magnitude events) are located at distances greater than 100 km from the Project.

Finally, four levels of seismic verification are proposed: 1) Normal Operating Earthquake for non-critical structures (NOEnc), with a 44-year return period, 2) Safety Earthquake for non-critical structures (SEnc), with a 285-year return period, 3) Normal Operating Earthquake for critical structures (NOEc), with a 144-year return period, and 4) Safety Earthquake for critical structures (SEc) with a 10000-year return period. To define levels 1 and 2, a mine operating time of 30 years has been assumed. On the other hand, critical works are deemed to be those whose collapse constitutes a threat to public safety and/or poses a risk to one or more population centers or to the environment.

20.1.2 Geomorphology and glacier study

The baseline study of this component has identified, characterized, and described the existing geoforms, landscape forms and modeling processes, distinguishing between the Mine Area and the Access Road.

Since 2012, Mountain Pass Consulting has conducted an extensive geomorphological characterization and mapping of the Los Azules area. Geomorphological features have been mapped in the Mine Area and Access Road areas, satellite images, geophysical and hydrogeochemical studies have been analyzed, topographic controls have been carried out and temperature probes have been installed to evaluate possible permafrost conditions and the overall distribution of periglacial origin.

Regarding geomorphology, on a regional scale the territory of San Juan has a predominantly west-east general gradient from the Andes Mountains to the Tulum valley, continuing towards the Bermejo valley in the southeastern end of the province, and to the east the Sierras Pampeanas of Valle Fértil and the great Bajo Oriental (Eastern Lowlands), with a succession of mountainous foothills and longitudinal intermontane depressions to the south.

The climatic characteristics of the province affect the exogenous processes occurring in the region. On the other hand, the intensive recent tectonic activity has produced a continuous rejuvenation of the landscape, accentuating the action of these processes. Mechanical weathering and aeolian accumulation are common in the Precordillera and lower parts of the Frontal Cordillera.

In the Mine Area, glaciers were the main landscape shapers during the Pleistocene, forming glacial troughs and cirques, deep valleys with high shoulders and fluvioglacial terraces of significant development, especially in the valley of the Río Salinas. The retreat of ice masses at the end of the last Pleistocene glaciation left its mark on glacial conditions. Subsequently, periglacial conditions set in and are still present today.

The climatic characteristics of the area during the Pleistocene period led to the generation of large volumes of debris, which were moved to lower areas by rainwater or snowmelt, giving rise to different fluvioglacial levels that are evidenced by the presence of stepped terraces.

Finally, the landscape forms observed today are the result of degradational and aggradational processes, associated with periglacial, fluvial, and alluvial action, as well as tectonic activity.

The Access Road, in turn, begins in the town of Calingasta, where the confluence of the river by the same name with the Río de Los Patos is located. This sector has characteristics and geoforms typical of a Fluvial environment. Further west, on the banks of the Río Totora, there are vega sectors interspersed with sectors of steep walls that confine and form the valley of the river. On these walls, it is common to observe various mass wasting geoforms produced by the gradients and loose material falling down the slopes.

Further into the mountainous region, the current landscape is the result of erosion and glacial accumulation processes, which generated geoforms such as U-shaped valleys, moraine deposits and debris glaciers. With the subsequent climate change and the partial retreat of ice at the end of the Pleistocene, the landscape was modified by fluvial erosion and mass wasting events.

Periglacial conditions are restricted to the highest areas of the sector, where the current access road is laid out. Cryoclastism is one of the factors involved in the process of generating periglacial geoforms, which by means of physical weathering, produces gelifract mantles that may creep downslope. Active and inactive rock glaciers are geoforms occurring along the road.

Regarding the cryosphere, in the Mine Area and west of the Access Road area of Los Azules, predominant characteristics of periglacial environments can be observed, such as changes in water status, the action of frost, seasonal frozen soils, permafrost and snow cover. These environments form part of the cryosphere. In addition, among the elements that make up the cryosphere and characterize periglacial environments is permafrost -or permanently frozen soils-, defined as soil that remains at or below 0°C for at least two consecutive years. This is the main non-glacial high mountain element.

From April 2011 to the present, soil temperature gauges have been installed to determine whether the temperature regime allows the presence or formation of permafrost. The purpose of installing soil temperature sensors is to determine the soil temperature regime.

The first analysis was based on the use of the Basal Temperature of Snow (BTS) method. Due to the conditions assumed for the application of this method that are not observed in the study area, the research team developed an alternative method based on the analysis of Mean Annual Ground Temperature (MAGT). In addition, and to increase the accuracy of the distribution analysis, the temperature analysis methods were applied on two Digital Elevation Models (DEM). In this way, the results of the temperature analysis methods on the ASTER and SRTM DEMs were compared.

The Los Azules Mine Area is unpopulated during the winter months and climatic data are incomplete to date, so snow thickness and distribution in the area is unknown. The absence of a snow cover, which acts as an insulator, causes ground temperatures to be significantly affected by air temperature.

The presence of permafrost confirms only ground temperature conditions but does not confirm ice content. Data taken over a decade of studies in the general Los Azules area, as well as in peripheral areas, confirm that permafrost is isolated and discontinuous in the study area. In the evaluation of the potential impacts of a mining project on permafrost in the Los Azules area, it is important to consider that the maximum elevation point around La Ballena (the area with the highest grade in the exploration area) is 3850 masl. Although permafrost may be found at this elevation, monitoring results obtained from the temperature probes indicated that permafrost would be sporadic and of limited extension, since most of the deposit of economic interest is located below this elevation.

As mentioned above, the distribution of soil temperature sensors that has been carried out since 2011 to the present makes it possible to determine the presence, or confirm the absence, of permafrost in the study area. The current distribution will enable the incorporation of areas that were previously outside the area of interest. Excavations conducted by the mining company have confirmed that the permafrost distribution prediction derived from soil temperature modeling is valid and applicable to the study area.

Climatological data taken in the Los Azules Project area indicate an increase in maximum temperatures, which adversely affects snowfall conditions, causing a decrease in snow cover thickness, a decrease in ablation water measured through surface runoff, and a decrease in soil moisture.

Although several rock glaciers have been surveyed in the Río Frío basin, only one has been found in the northern sector of La Ballena which could be under the influence of the mining project. Various monitoring data confirm that the debris glacier in the northern sector of Los Azules continues to show elevation loss at its toe, probably due to the disintegration of the ice cover. Finally, company personnel have been trained to take measurements on the glacier to collect data to help monitor its behavior.

Throughout the sections of the Access Road, the main geological hazards in the western sector that could affect the road layout are mass wasting processes that in some cases reach the riverbed plain, such as landslides, avalanches, debris flows and rockfalls. On the other hand, in the western sector, geomorphological processes of periglacial environment were recognized in several sectors, such as creeping processes on cryogenic slopes.

20.1.3 Climatology

The baseline study includes a characterization and description of the climate and meteorology in the Project area, covering the mine area and access road. The Project has a meteorological station, named Los Azules, and three additional stations have been considered, which given their location and proximity to the project, provide valuable information that complements the station located at Los Azules. The additional weather stations are: Tascadero, El Polvo and Calingasta -EEA San Juan.

In the mine study area, historical data from 2010 to 2022 are available, and it was possible to conclude that the climate of the project area is characteristically a high-altitude climate with general features of the Tundra type where the temperature responds to the spatial field of the topographic contour lines ("topo-climatic component"). During the summer months, the average temperature can reach 9.21°C at Los Azules, while in winter it reaches -3.64°C, with extreme minimum temperatures as low as -24.3°C. The average annual temperature is 2.75 °C.

Most of the precipitation in the Los Azules Mine Area occurs in the form of snow (due to its altitude) and has an annual regime with a winter maximum. On the other hand, the occurrence of the ENSO (El Niño South Oscillation) warm phase, which produces above-average precipitation values affecting the entire region, is noteworthy. The opposite is the case with the La Niña phenomenon. The inferred values to estimate annual precipitation in the Mine Area are estimated at 220 mm based on available information, which may be exceeded with the El Niño event. Evapotranspiration in the area, according to the Tascadero station, is 1348 mm.

The average relative humidity is 32.28%. June has the highest monthly average with 38.9%, followed by July, with 32.28%. The lowest levels occur in November and December (27.77 % and 26.23 %, respectively). With respect to radiation, the absolute maximum is 1423 W/m² in January, with an average of 256 W/m² for the entire year.

The average atmospheric pressure is 681.71 hPa, while the average wind speed is 2.93 m/s. The maximum speed is 14.5 m/s (52.2 km/h), with gusts that can reach 28.6 m/s (102.96 km/h). The wind comes from the north 22% of the times. Winds from the N (21.4%), W (10.98%), WNW (14.04%) and NNW (13.08%) directions add up to a total of 59.5% of the times, with southern wind accounting for 11.68%. In general, the wind follows the topography of the sector, i.e., the north-south direction of the valley, with additional influence of winds coming from the west.

In reference to the Access Road, two meteorological stations have been considered for this sector: El Polvo and Calingasta. In the El Polvo area, the average temperature is 9.93 °C, while in Calingasta it is 16.3 °C. The average relative humidity in El Polvo is 27.40% and in Calingasta it is 37.31%.

Wind speed varies between 0.5-2.1 m/s in 9.2% of the times in El Polvo. In 41% of the cases, wind speed varies between 2.1-5.7 m/s. Finally, in 45% of the cases, wind speed varies between 5.7 and 8.8 m/s. In El Polvo, the wind comes from the N and SE in most cases.

20.1.4 Hydrology and Hydrogeology

Hydrology and hydrogeology have been studied since 2011 up to the present. In the hydrology baseline study, flow measurements have been taken since 2012 and water quality was monitored between 2011 and 2022 (except for 2021). For the hydrogeology baseline study, the results of groundwater quality studies carried out between 2012 and 2022 (except 2021) have been integrated. There is also information on water levels and well head elevation that provide data on the chemical quality of water, as well as hydraulic data on aquifers that provide a perspective on their storage capacity and recharge, as well as groundwater flow directions, among other data.

In 2011, seven standpipe piezometers were installed, plus six vibrating-wire piezometers. The standpipe piezometers were of two types, shallow alluvial glacial aquifer (PAG), and bedrock aquifers (PAF). For groundwater quality monitoring, a total of 7 (seven) piezometers are available. The wells historically monitored since 2012 are those called PAG-01, PAG-03, PAF-04 and PAF-05. As of the last available campaign, carried out in April 2022, wells PAF-06, PAF-02 and PAG-03 have been incorporated and will be monitored in monthly campaigns starting in December 2022.

Field work included observation, measurement, and sampling of surface water bodies. The flow measurement network of the project consists of nineteen (19) surface water points in the MA, plus four (4) reference points located outside the Mine Area on the course of the Río Salinas. The network is completed by six (6) points located in the Access Road.

Up to the date of preparation of this report, a total of sixteen (16) hydrological campaigns had been carried out for the Los Azules Project during the period between January 2013 and November 2022. The field work was carried out by permanent staff of the Hydraulic Research Institute (Instituto de Investigaciones Hidráulicas -IDIH) of the School of Engineering of the National University of San Juan. In its execution, standard techniques were used which are widely recognized in hydrological literature. As a result of these determinations, estimates of runoff geometry were made, establishing the width of the sections and their corresponding head or water depth.

The values obtained range from a few liters per second (especially for those monitoring points corresponding to small streams in the upper sector of the basin) to flows of several cubic meters per second, the latter typically observed in lower sectors of the basin. As a general flow measurement conclusion, it was determined that the outflows of the basins are mainly surface water and that there is very little variability between maximum flows and average flows recorded.

For surface water monitoring, the results of 23 monitoring points corresponding to the Mine Area (mentioned from the sector to the west and center-west of La Ballena, La Ballena and outside the Mine Area) have been analyzed: M13, M16, M14, Salinas, Salinas-2, M15, M08-M23, M09, M10 BIS, M12, M01, M02, M03, M07, M04, M05, M17 BIS, M11, M06, M18, LA5, Salinas-3, Salinas-4 and Km 101. Plus 5 (five) points corresponding to the AR: Alumbrera, Candadito, Despensita, RCa01, Km 1001 and Los Patos.

With respect to the analysis of surface and groundwater quality in the baseline study, field data and physicochemical laboratory results, major ions (some years), and total metal concentrations are available; and in the case of surface water, bacteriological results are also available for some monitoring points. With the available information, the following analysis has been carried out: data integration and processing; hydrochemical characterization of the basins involved based on the main ions, using Piper and Stiff diagrams (using Diagrammes and Aquachem 4.0 software). The evolution of the concentrations of some parameters of interest has also been studied to determine the baseline water chemistry of each monitoring point. In addition, the parameters measured in the laboratory were compared with the water quality guidance levels of National Law No. 24585 - 1995, for different applicable uses: human drinking, livestock drinking, irrigation and aquatic life in fresh surface water (the latter use only in the case of surface water). In those cases where the regulation does not establish a guidance value, reference values established by other regulations were used for comparative purposes: National Law No. 24051 - 1991, Argentine Food Code, World Health Organization (WHO, 2004) and Canadian Environmental Quality Guidelines (2003).

20.1.5 Flora and Fauna

In the flora and fauna baseline study, each component has been described and characterized to describe the respective communities without Project intervention. The flora data available correspond to the period between 2013 and 2022. The available data for Terrestrial fauna in the project area were recorded during the period 2008-2019 and 2022-

With respect to flora, the vegetation and its floristic composition in the Los Azules Project area correspond, within the Andean-Patagonian Domain, to the High Andean Province, High Andean-Cuyo District, according to the classification proposed by Cabrera (1994). This is one of the districts whose floristic composition and community characterization are least known. The High Andes ecoregion is represented floristically by grassy steppes (predominantly on mountain slopes with angular clasts) and/or shrubs (predominantly at the bottom of valleys and at the foot of slopes) adapted to climatic stress. This environment is called zonal or hillside, and presents low coverage caused by the high infiltration that generates dry and unstable soils. Likewise, in the High Andes there are so-called azonal vega environments, which comprise flood meadows with permanent water supply, where cushion species of the Cyperaceae and Juncaceae families dominate. They have a scattered spatial distribution due to topographic, geomorphologic, and hydrologic factors and to local soil conditions, humidity (degree of saturation) or any characteristic such as micro-relief, salt concentration, degree, and intensity of grazing.

Vegas are the most important ecosystems in terms of biodiversity, productivity, and percentage of vegetation cover in relation to other environments in the central Andes. High Andean vegas are systems that are highly dependent on water availability.

Signs of intense trampling and overgrazing resulting from activities led by Chilean herders have been recorded and documented in the study area. On the other hand, for several years there has been a process of natural desiccation due to the drought affecting the region, with a reduction of snow precipitation in the Andes from 1998 to the present.

From historical data, a total of 138 plant species have been identified in the various high Andean environmental units sampled throughout the different seasons in the Mine Area. Species of the Asteraceae, Poaceae, Cyperaceae and Juncaceae families predominate. The most dominant species in the study area are: Eleocharis pseudoalbibracteata, Carex gayana, Oxyclöe bisexualis, Oxyclöe haummanian, Deyeuxia curvula, Deyeuxia velutina, Poa holciformis, Adesmia subterránea, Anarthrophyllum gayanum, Pappostipa frígida, Chuquiraga oposittifolia, Azorella trifoliolata, Juncus acutus. These species constitute the different plant communities found in the Mine Area, which, in general terms, were identified in the respective rivers and by the monitoring activities carried out.

In general terms, a decrease in the biologically active surface area of vegas was observed. The results indicate a continuing shift from oversaturated to saturated environments and an increase in the area of dry environments.

It can be said that, at present, of all the impacts observed, overgrazing is seen as one of those with the greatest impact on the physiognomy of the vega systems analyzed. On the other hand, the process of desiccation of the water bodies and the drought affecting the system should also be mentioned. Rehabilitation and restoration practices are therefore required.

Regarding the vegetation along the Access Road between the Candadito camp and the Los Azules exploration camp, it is important to highlight the presence of communities of Adesmia pinifolia and accompanying species that are in areas near the road, this being characteristic of the shrub steppe environment.

With respect to fauna, as mentioned above, in the study area we have observed high plant consumption and biomass loss from browsing, soil and vegetation degradation from trampling, significant accumulations of herbivore excretions, and numerous dust bathing/wallowing sites with total loss of plant communities. This shows significant overgrazing pressure by domestic livestock, which far exceeds the environment's carrying capacity.

During a good part of the year (November to March), cattle raising in the high Andean region relies on exclusive and extensive grazing on high forage value systems such as the vegas. This intensive and ancient activity, which dates to colonial times, is one of the main threats and factors in the degradation of these natural systems.

Also, the presence of dogs (Canis familiaris) associated with human activities (crianceros) could have an important impact on bird populations, mainly due to predation of eggs and nest-bound off-spring of Larus serranus (Andean gull), Chloephaga melanoptera (piuquén), Thinocorus orbignyianus (collared snipe), and of passerines that build nests in ground level bushes. The reptile populations of Liolaemus fitzgeraldi could be affected, since they are found in habitats with very low cover, which makes them very vulnerable to these predators. During the campaigns, it was possible to record and observe the predatory behavior of dogs on a European hare (Lepus europaeus).

During all fauna monitoring carried out in the study area, a total of 79 species were recorded, consisting of 64 birds, 11 mammals and 4 reptiles. When comparing habitats of vegas, steppes and slopes, the greatest richness, abundance, and diversity is found in vega environments. In other words, the most productive habitats concentrate the greatest biomass of terrestrial fauna.

The main refuge, feeding and nesting site for terrestrial fauna in the Mine Area is the vega known as La Ballena, which is considered a priority site for the conservation of terrestrial fauna. Andean vegas are the sites with the highest plant productivity, and many organisms depend on them.

It is safe to say that there are no records of amphibian species in the entire study area. In relation to the reptiles found, Liolaemus fitzgeraldi (Aconcagua lizard) is suitable as an indicator of ecological quality, given its close relationship with the stunted vegetation that it uses as a refuge and food source. It is highly abundant and is frequent in almost the entire study area; its absence could be indicative of disturbances due to anthropic activity (Dr. Juan Carlos Acosta 2013).

Among the bird species, the Andean goose and the Agachona Chica are proposed to be regarded as indicators. The Andean goose (Oressochen melanopterus) is regarded as such because it consumes aquatic plants, and its abundance depends mainly on the presence of vegas in good environmental conditions. At the same time, it nests and breeds in the "mine area", where several stages of chick development have been detected. The species was observed in greater abundance in the slopes and lagoons of "La Ballena" and the "Río de Las Salinas ". On the other hand, the Agachona Chica (Thinocorus rumicivorus), being one of the main granivorous species that feeds on seeds, is considered key in terms of dispersal and increased probability of germination of plant species found in the steppe environment. Chicks have been recorded in the area, which means that it would be a breeding site for the species. Finally, the presence of the condor (Vultur gryphus), a species of great cultural value and categorized as "near threatened" on a national scale, is also important.

Regarding mammals, one of the species of greater interest as an emblematic species of the region is the guanaco (Lama guanicoe), which frequently feeds in vegas. The abundance of the European hare (Lepus europaeus), an invasive exotic species, is also high in steppe and hillside environments.

20.1.6 Limnology, Ichthyology and Bioaccumulation

The development of the baseline included the description and characterization of the limnology, ichthyology, and bioaccumulation component without project interventions. Data from five limnological campaigns carried out in 2007, 2008, 2019, 2021 and 2022 were considered. For the ichthyological component, data from 2007, 2010 to 2015 and 2022 were integrated. For the study of metal bioaccumulation in biotic components (fish and plants), a campaign was conducted in 2022.

Limnology. The close relationship between the properties of the environment and the presence of limnological organisms leads to the use of the latter as a bioindicator representative of the conditions at a given time under study. The bioindicator-based methodology has the advantage that it enables detection and evaluation of the intensity and extent of a contamination process. Therefore, for the baseline study, limnology, ichthyology, and bioaccumulation analyses have been carried out.

One of the groups of organisms studied in the baseline includes organisms with benthic habits, i.e., those that live at the bottom of water bodies on different substrates (sand, silt, rocks, submerged plants), such as phytobenthos and benthic macroinvertebrates. On the other hand, planktonic organisms were surveyed, which are defined as the group of living beings, animals or plants, adults, and larvae, which float passively in bodies of water, or which, if able to swim, cannot resist the movement of weak currents.

In the Phytoplankton communities, diatoms were the most representative in every year except for 2022, when cyanobacteria predominated. A significant number of water bodies was monitored, with the highest richness observed in the Ballena sector in 2007 and the lowest in the lower sub-basin of the Río de Las Salinas.

In general, the algae species identified in the vegas of the La Ballena sector are typical of cold waters and have a pH between 6 and 8. They are indicative of eutrophic (presence of nutrients) and polysaprobic (high degradation of organic matter) environments. These characteristics are typical of the vega systems, given the high production of organic matter, both by terrestrial and aquatic flora.

With respect to phytobenthos, the most characteristic were bacillariophyceae in all campaigns except for year 2022, when cyanobacteria predominated. These organisms are favored by the submerged vegetation in the water bodies.

Protozoa were the most abundant of the zooplankton group analyzed in all sectors and mainly in the 2019 campaign.

The 2022 sampling corresponded to the fall and in 2021 to the summer, which resulted in the decrease in taxa richness in the macroinvertebrate community and algae density. Ephemeroptera of the Leptopheliidae family, which are indicators of good water quality, were scarcely represented in the Río de Las Salinas and Río Frío in the summer, while in the fall they were only recorded in the Río Frío. This family was also observed at different monitoring points in the La Ballena sector.

Ichthyology. In the Puna and Andean region there are species of small fish (Order Siluriformes) which are endemic and, therefore, their localization is of great importance. Such species belong to the genera Hatcheria and Trichomycterus. However, although their presence is not ruled out, no specimens have been found, unlike the rainbow trout which was found in all years surveyed.

The study of introduced species such as the rainbow trout, Oncorhynchus mykiss (Salmonidae), is of great importance not only because they are organisms that modify aquatic ecosystems, impacting directly by predation on other vertebrates such as fish or amphibians, but also because they consume the invertebrates that are part of these prey's diet. The great phenotypic plasticity of rainbow trout with respect to native species (aggressive predation, great migration capacity, wide trophic spectrum, high reproductive potential, etc.) is also well known, which increases its danger as an invasive species, causing the extinction or decrease of native species.

According to data recorded during the campaigns, Oncorhynchus mykiss (rainbow trout) specimens have been observed in the sectors of Río de La Embarrada, Río Frío, Río de Las Salinas and Arroyo La Totora. In addition, a total of 4 individuals were captured in the 2013 and 2014 campaigns of the species Hatcheria macraei (eagle catfish) in Río Calingasta (La Alumbrera). It is concluded that there were no significant differences in the years studied.

The identification of the rainbow trout, an introduced species, demonstrates its great plasticity to the environment studied. It was found in sectors with little water and little food. Its presence could displace the communities of native species found or potentially present, such as the Trichomycterus genus.

Bioaccumulation. Bioaccumulation is the storage of a contaminant in an organism from any source of exposure including air, water, and food. Many organisms such as fish, crustaceans and aquatic plants have the capacity to bioaccumulate metals in their tissues, which would imply their entry and mobility within the trophic chains with the consequent risk to the environment, human health and the alteration of population dynamics and trophic cascades. The complex relationships that develop in the ecosystems associated with the Los Azules project must be understood to determine the bioaccumulation of metals in biotic components (fish and plants) and for various related waterways, considering the presence of the project in advanced exploration, not yet in operation.

Of the 18 bioaccumulation analyses performed on Myriophyllum chytene and Oncorhynchus mykiss, the values that were above the established thresholds are: Aluminum, Arsenic, Barium, Cobalt, Chromium, Iron, Manganese, Molybdenum, Lead, Zinc and Selenium. These represent the first data generated with respect to bioaccumulated metals in the biota sampled.

20.1.7 Edaphology

The edaphology study for the baseline is currently being prepared. Its objective is to reflect the results of the identification and characterization of the soil units present in the study areas considered, the Mine Area and the Access Road.

The soils of this region are characterized by being shallow and in some sectors skeletal. In general, they are located on fluvial terraces, alluvial fans, in active watercourses, on slopes, on bodies of disintegrated rocky material (for example, originated by mass wasting), in sectors where water surfaces, etc. The central and distal parts of alluvial fan systems, in general, are characterized by the deposition of fine material necessary for soil development and plant growth.

In December 2022, soil sampling was carried out for subsequent analysis, and prior to this, a background review of the area was carried out, together with analysis of satellite images and radar-derived digital terrain models. This information was used to identify homogeneous physiographic units and soil cartographic units. Based on this information, the different sampling points were then determined. The geographical position of the profiles was established using a GPS satellite positioning system, and they were expressed in plane x and y coordinates using the GK coordinate system for strip 2 and the POSGAR 98 datum, corresponding to the sampling area.

For the classification of soil units in the Mine Area, previous soil characterization works in that area have been considered, and those sectors without sampling have been completed to obtain a more detailed knowledge of the characteristics of the soils present in the area. During the field campaign for the development of the baseline conducted in May 2020, test pits were dug to describe the soil profile and obtain samples, which were subjected to physical-chemical and agrological analysis. This work will be later complemented with data from recent sampling.

The taxonomic classification of soils was based on the North American Soil Taxonomy system of the Soil Survey Staff, United States Department of Agriculture (SSS-USDA, 1975) and the second edition of Soil Taxonomy, A Basic System of Soil Classification for Making and Interpreting Soil Surveys (1999). Classification has also been based on land use capacity, using the Land Capability classification by Klingebield and Montgomery (1961). This system demarcates the potential agricultural areas and identifies land-use limitations.

The basic parameters will be determined in the field. The description during sampling includes the identification of the test pit or sample (numeric or alphanumeric code), its location, climatic conditions prior to and at the time of sampling, landscape, vegetation, water table depth, type of relief, physiographic position, and source material. The description of the soil profile covers the aspects (contemplated in the edaphological data sheet used), corresponding to characteristics such as stoniness and rockiness, runoff, slope, permeability, risk of flooding/waterlogging, erosion, pH, presence of salts or alkalis, moisture distribution, drainage, and land use). These data are later complemented by laboratory analytical results obtained from physical-chemical and agrological analyses of soil samples extracted from the various horizons.

Regarding the access road, the soils in the region have developed over deposits resulting mainly from erosion and depositional activity of rivers and on slopes, from mass wasting, etc., and show evidence of recent development, thus presenting conditions of high vulnerability that make them very susceptible to degradation.

The cryoclastism affecting rocky outcrops has given rise to numerous rockfalls, which, when deposited at the foot of the slope, have formed debris cones and extensive slopes, among other geoforms. These materials invade the watercourses and are reworked, transported, and deposited in sectors of decreasing energy of the transport agent, where they constitute the original soil-forming material.

The low proportion of fine material that accumulates in some sectors of the valleys, particularly in areas with water availability, favors the installation of typical vegetation (vegas) and the weak development of soil observed in the valleys.

The fluvial and mass wasting processes (Holocene) that took place after the retreat of glaciers, together with the severe climate conditions, give a distinctive character to the soils, whose profile has very poorly developed horizons, generally defined by an organic horizon a few centimeters thick and another mineral horizon, having, in general, a thickness of no more than one meter.

Soils in the Access Road study area are in equilibrium with the prevailing conditions in the region, which suggests that the current weak degree of development will continue, if the conditioning factors remain unchanged.

Further edaphological information will be obtained once the results of the December 2022 sampling have been analyzed and integrated into the baseline study.

20.1.8 Archaeology

The baseline study of the archaeology component has integrated all the archaeological verification, prospecting and monitoring work carried out by Dr. Catalina Teresa Michieli and Carlos E. Gómez O. in the Mine Area and the Access Road from 2012 to the present.

The archaeological prospecting and monitoring works carried out were authorized, without exception, by Resolution issued by the Secretariat of Culture of the Ministry of Tourism and Culture of the Province of San Juan, as the Enforcement Authority for heritage laws, Provincial Law N° 571-F and National Law N° 25743, and they were duly reported. These works were aimed at locating possible pre-Hispanic (indigenous) or historical archaeological sites in the areas of potential impact, and the subsequent and periodic review of their condition (or monitoring).

From the Baseline study, it was possible to conclude that, in general, the entire area of influence of the Los Azules project does not have significant pre-Hispanic (or Indigenous) archaeological sites, except for some isolated findings of undefined lithic material (remains of stone carvings) or unfinished tools that were duly reported and rescued.

However, on the project's access road (Cuesta del Gringo), there are two large rocks with unique petroglyphs (indigenous engravings on rock).

The cultural-historical sites are well represented both on the access road and in the mine area and are all related to the management of livestock of European origin after the Spanish conquest (post-Hispanic).

On the western slope of the Cordillera de La Tortora and within the Project area, these sites -simple lodges or night shelters- are linked to the settlement and transit of Chilean goat herders ("crianceros"). Both are still in use today.

On the eastern slopes of the Cordillera de La Totora, these sites - simple lodges or night shelters - are linked to cattle trafficking.

On the western slope of the Cordillera de La Tortora and within the Project area, these sites are linked to the settlement and transit of Chilean goat herders ("crianceros"). Both are still in use today.

The most significant Chilean herders' posts, considering their very likely ancient age, their constructive complexity, their size and their preservation in an almost original state without additions of current materials, are: 'Puesto Colorado' (on the right bank of the Río Frío), 'Puesto Redondo' (on the left bank of the Río Frío), 'Puesto Gris' (on the right bank of the Arroyo de la Embarrada), and 'Puesto de La Coipa' (on the right bank of the Río de Las Salinas at the mouth of the Arroyo Estero de la Coipa).

Also significant are the following lodges or night shelters used by Chilean herders in transit across the Andean passes: Alojo-A, on the right bank of the Estero Verde (now called Arroyo Azules), and Alojo Escondido, on the left bank of the Río de Las Salinas.

20.1.9 Protected Areas

For the development of the baseline for this component, the various publications, regulations, and classifications on Protected Areas within the system of natural protected areas of Argentina and the province of San Juan were consulted, as well as categories designed by supranational organizations. The Mine Area and Access Road were considered for the study.

The situation of Natural Protected Areas (NPAs) at both the national and provincial levels was contextualized in detail, and the analytical criteria were established. The proximity criterion was defined as representative since it provides specificity in the analysis of the characteristics of the study area. On this basis, the natural protected areas included were considered, as well as areas surrounding each defined sector.

The result of the baseline analysis indicated that the mine area does not interact with NPAs within a study area of a 45 km radius. Considering conservation interest sites, as defined by the Territorial Regulation of

Native Forests (Law No. 26331), no areas with Conservation Categories have been identified in the study area.

The Access Road does not interact with NPAs within the study area, considered with a 10 km buffer. Considering the conservation interest sites, as defined by the Territorial Regulation of Native Forests (Law No. 26331), Conservation Category II zones (yellow) are identified in the first 20 km of the initial section (east) of the access road.

The closest NPAs are Los Morrillos wildlife refuge, Cerro Alcázar natural monument and Estancia Don Carmelo, at approximately 80 km from the Mine Area.

In conclusion, no interactions were observed between the study areas (Mine Area and Access Road) and protected areas, nor with areas of conservation interest based on native forest criteria. Also, there are no RAMSAR sites, Biosphere Reserves or other protected sites recognized by the Nation, the Province of San Juan and/or international treaties.

20.1.10 Socioeconomic aspects

Located in the northwest of the Province of San Juan, the Department of Calingasta is one of the 19 municipalities, and it is the department with the largest surface area. Entering the department and following National Route 149, which crosses the department from north to south, the towns of Villa Nueva, Puchuzun, Villa Corral, Villa de Calingasta, Alcaparrosa, Barrialito, La Isla, Tamberías, Hilario, Sorocayense, Barreal and Villa Pituil are located.

This form of human settlement constitutes the main population settlement feature on the longitudinal valley of Calingasta, which also follows the course of the Río Los Patos from the south and the Río Castaño Viejo from the north, converging in the vicinity of the town of Calingasta to form the Río San Juan.

The limits of governmental action of the municipality coincide with the departmental boundaries, with the mayor exercising the political power and, together with the Deliberating Council, being responsible for municipal governance.

It has three important population centers: Villa Calingasta, Tamberías and the conglomerate formed by Barreal and Villa Pituil. The distances between the human settlements entail a decentralization of municipal services. Delegations are established in these three towns ensuring a certain territoriality of the Municipality, exercising governance, control, collection of fees and taxes, performing works, providing waste collection services, and establishing a communal presence, with Calingasta constituting a polycentric department in the management of the municipal services.

In 2010 the department had a population of 8,588 people (National Statistics Institute - INDEC) ranking fifteenth in the province with a 5%, inter-census growth and with relative importance at the provincial level that decreased to 1.26% with respect to the previous census (1.32%) due to higher population growth in other departments.

The population structure shows 53% of males and 47% of females (National Statistics Institute - INDEC 2010) having a progressive pyramidal structure with a lower percentage of young people in the group composed by the population from 0 to 4 years old and a slight increase in the older age groups, thus showing a slight aging process. The 15 to 64 age group is noteworthy, with a higher number of males than females (33.35% and 29.35 respectively), totaling 62.70% of the department's population, with an average age of 28.6 years and a masculinity index of 113 men for every 100 women.

20.1.11 Future environmental and social work plan

During December 2022 and from early 2023, environmental monitoring activities will focus on expanding the data set already available to capture seasonal environmental variations at the site that are relevant to support the preparation of an environmental impact analysis and report (IIA).

In addition, the baseline that is currently being prepared will serve to provide a complete description of the site and the social environment and economic activities in the area of influence before the beginning of any project development activities and will also provide the company with a reference framework in the event of any future complaints or claims of adverse impacts.

Once the baseline is fully established, and the site is authorized and moving forward with construction, baseline data collection will be maintained for all components that show seasonal or inter-annual variations throughout the permitting process until project development begins. Upon commencement of project development, the established Environmental and Social Management Plan (ESMP) will be complied with. This plan is an instrument whose main objective is to develop measures aimed at the Project's sustainable development-

The ESMP presents the measures for prevention, mitigation, rehabilitation, restoration or re-composition of the environment affected by the environmental impact, organized according to stages and chronology of execution. Law 24585 addresses these measures for the following components: geomorphology, water, atmospheric conditions and air, soil, flora and fauna, ecological processes, and sociocultural environment.

On the other hand, the actions for implementation, follow-up and monitoring of such measures are developed specifically for each component in the Monitoring Plan and in the Project Closure Plan.

As part of the environmental and social management plans, the following aspects will be considered and monitored as part of the environmental and social work plan for the future:

Table 20.1: Summary of future environmental and social work plan

Environmental and/or social component	Sub-component	Project Stage		
		Construction	Operation	Closure
Geomorphology	Control of active geological processes	x	x	x
	Cryogenic geoforms	x	x	x
Water	Water resources	x	x	x
Atmospheric conditions	Greenhouse gas emissions	x	x	x
	Meteorology and air quality	x	x	x
	Noise and vibration	x	x	x
Soil	Soil	x	x	x
Flora and fauna	Terrestrial biota protection	x	x	x
	Aquatic biota protection	x	x	x
	Control of exotic species	x	x	x
Ecological processes	Compensation of ecological impacts	x	x	x
	Ecological restoration of disturbed areas	x	x	x
Socio-cultural environment	Transport	x	x	x
	Heritage protection	x	x	

Table 20.1: Summary of future environmental and social work plan				
Environmental and/or social component	Sub-component	Project Stage		
		Construction	Operation	Closure
	Employment program	x	x	x
	Training and education	x	x	x
	Consultation and communication	x	x	x
	Visual impact	x	x	x
	Contribution to local development	x	x	x

20.2 GEOCHEMISTRY

20.2.1 Introduction

SRK (UK) have undertaken an assessment of the environmental geochemistry of the Los Azules copper mine as part of the Preliminary Economic Assessment (PEA). The assessment has considered the geology and geochemistry of the process feed material, and mineral storage facilities with respect to acid rock drainage and metal leaching potential (ARDML).

The geochemical evaluation is in its early stages. A review of the available geological information pertinent to the study has been undertaken, and over 100 samples have been collected and analysis started for geochemical characterization, but not all the data was available for inclusion in the PEA. This section describes the work undertaken to date and is sufficient for the purposes of a PEA assessment. The geochemical characterization work planned for the Feasibility Study is also described, and preliminary geo-environmental models are presented for the deposit and mine rock storage facilities. Based on the available information and the geo-environmental conceptual models, the main issues and potential environmental risks as they relate to environmental geochemistry are outlined.

20.2.2 Site Description

The Los Azules copper project is in the Calingasta department of San Juan province, Argentina. The project is located within the Blanco River basin, which is one of the tributaries of the San Juan River. The site location and layout are shown in Figure 5.5. The main mine site facilities that should be assessed with respect to ARDML risks include:

- Open pit
- North Mine Rock Storage Facility and South Mine Rock Storage Facility
- Low grade stockpile
- Heap leach pad (HLP)

20.2.2.1 Geological Setting

The main geological summary of the deposit is included in Section 7.

Typically, Porphyry Copper Deposits have a well-defined geological zonation reflecting the formation and alteration of the porphyry intrusion and subsequent mineralization events (Lowell and Gilbert, 1970; John et al., 2010; Sillitoe, 2010). The deposits have a strong hydrothermal alteration pattern and ore distribution that influences the environmental geochemistry of the deposit (Figure 20.1). Understanding the controls is important to ensuring the geochemical characterization covers all aspects of the deposit and potential lithological and alteration types. As there is overlap and variability the resulting combinations are grouped as material types.

It is not just the primary element dispersion that is important; for many deposits the zonation produced by oxidation is also a key influence on geochemical behavior. Thus, it is critical to define the different material types that constitute the deposit as they will have different acid generation characteristics and metal leaching characteristics.

At Los Azules because of the history and evolution of the deposit, most of the materials demonstrate propylitic and phyllic (sericite) alteration.

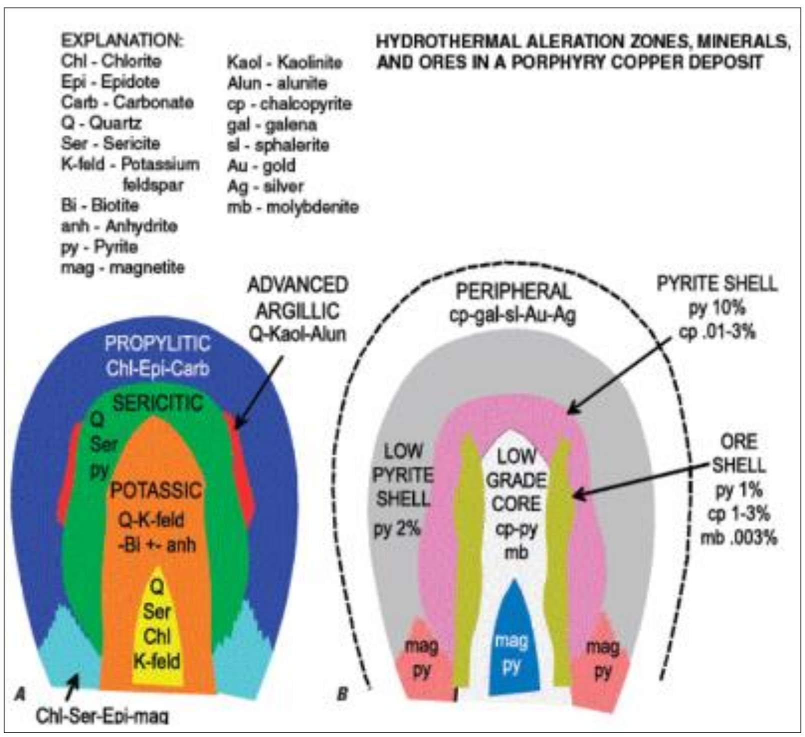


Figure 20.1: Schematic cross section of Porphyry Copper Deposit (John et al., 2010 modified from Lowell and Guilbert, 1970)

20.2.2.2 Climate

Climate information has been obtained from the 2017 PEA.

"Using weather data from the January 2013 to May 2016 period, the climate is categorized as semi-arid. *Most of the precipitation occurs during the winter months as moderate snowfalls during the winter and temperatures as low as -24°C and a large diurnal temperature range of approximately 20 degrees."*

20.2.2.3 Baseline Hydrochemistry and Applicable Water Quality Standards

Hydrochemical monitoring of surface and groundwater in tributaries and monitoring wells in and around the project area has been undertaken by the Instituto de Investigaciones Hidraulicas (Institute of hydraulic investigations) (IDIH) (IDIH, 2022). Previous monitoring campaigns have been undertaken in 2011 and in 2007-2008. The project's monitoring network comprises 30 surface points, 7 groundwater points and 8 points for bacteriological monitoring.

20.2.2.4 Water Quality Standards

One of the main objectives of the environmental geochemistry program will be to estimate contact water chemistry and the water chemistry that may be discharged from the site to the environment, or that may migrate from the site as seepage in groundwater and that may report to environmental receptors. The estimates of contact water chemistry will be compared to baseline water quality results for the receiving waterbodies and will also need to be assessed against appropriate water quality objectives to assess whether the mine contact waters could present a potential risk to the environment.

It is currently assumed that the water quality objectives for discharge of mine contact waters will adopt the San Juan province water quality standards (Table 20.2), as established in Decree No. 1.426 Law no. 24.585 The San Juan province water quality standards comprise four different standards: for human consumption; aquatic life in fresh surface water; crop irrigation; and for stock drinking water. For total iron, the law does not define guide values, however; the Argentinean Food Code No. 18284 and its Article 982 modification defines the tolerable limit for drinking water to be 0.3 mg/l.

The specific water quality standards that will apply to the mine contact waters will be dependent upon the established used of the water and should be evaluated further and discussed and agreed in consultation with the appropriate regulators.

Table 20.2: Water Quality Standards from Decree 1.426 Law 24.585					
Parameter	**Unit**	**Sources of water for human consumption**	**For protection of aquatic life in fresh surface water**	**For irrigation**	**For stock Drinking water**
pH	pH units	6.5 - 8.5	6.5 - 9.0	6.5 - 8.5	6.5 - 8.5
Total Dissolved Solids (TDS)	mg/l	1000	1000	1000	1000
Aluminium	mg/l	0.2	-	5	5
Antimony	mg/l	0.01	0.016	-	-
Arsenic	mg/l	0.05	0.05	0.1	0.5
Barium	mg/l	1	-	-	-
Boron	mg/l	-	0.75	0.5	5
Beryllium	mg/l	0.000039	-	-	0.1
Cadmium	mg/l	0.005	0.0002	0.01	0.02
Cyanide	mg/l	0.1	0.005	-	-
Zinc	mg/l	5	0.03	2	0.05
Cobalt	mg/l	-	-	0.05	1
Copper	mg/l	1	0.002	0.2	1
Chromium	mg/l	0.05	0.002	0.1	1
Chromium (hexavalent)	mg/l	0.05	-	-	-
Fluoride	mg/l	1.5	-	1	1
Mercury	mg/l	0.001	0.0001	0.002	0.002
Manganese	mg/l	-	0.1	-	-
Molybdenum	mg/l	-	-	0.01	0.5
Nickel	mg/l	0.025	0.025	0.2	1
Nitrate	mg/l	10	-	-	-
Nitrite	mg/l	1	-	-	-
Gold	mg/l	1	-	-	-
Palladium	mg/l	-	-	5	-
Silver	mg/l	0.05	0.0001	-	-
Lead	mg/l	0.05	0.001	0.2	0.1
Selenium	mg/l	0.01	-	0.02	0.05
Uranium	mg/l	0.1	0.02	0.01	0.2
Vanadium	mg/l	-	0.1	0.1	0.1

20.2.3 Mine Facility Geoenvironmental Conceptual Models

Interaction of exposed mined materials with precipitation (as rain or snowmelt) and other contact waters can release solutes that will determine the contact water chemistry released from each of the mine waste facilities. Release of contact waters into the environment with elevated solute concentrations has the potential to impact on local water quality.

The climate and water demand for the process feed materials is such that water availability may constrain mine production, and it is likely that water management during operations will seek to re-use and re-circulate as much water as possible. Therefore, the surface discharge of mine contact waters to the environment may be limited although potential seepage of contact waters to groundwater will need to be considered.

Post-closure there is the potential that contact waters could discharge to the environment as run-off or seepage to groundwater. Contact waters could be associated with any mine facilities and include the Mine Rock Facilities, the open pit, and the heap leach pad. For this preliminary assessment, conceptual models for these components have been developed based on SRK's experience of similar projects, using available design details and knowledge of the general site conditions based on background information.

Overall, the composition of contact waters is complex and is influenced by a range of factors. These include:

- Geochemical weathering behavior of the rock materials based on the mineralogy.
- Temperature effects that influence reaction rates.
- Particle size distribution of the rock material that will influence the exposed reactive surface area of the materials.
- Rainfall and hydrology that will dictate the water flow and the degree of contact/flushing of the weathering products form zones within the waste rock, with some zones being regularly flushed by water and some zones being effectively isolated from mobile water.

Geoenvironmental conceptual models are presented to highlight the main environmental risks associated with the main mine waste facilities and ARDML.

20.2.3.1 Mine Rock Storage Facilities

Areas of stored mine rock can be subject to weathering and release of solutes in contact waters.

The project will comprise of two MRSFs. The North MRSF will occupy an area of 7.55 km^2 and will receive 2,790 Mt of waste rock. The South MRSF is smaller and will occupy an area of 3.36 km^2 and will receive 705 Mt of waste rock. Waste rock will be placed at the South MRSF between year 1 of operation and up to year 10 when commencement of waste rock placement will occur in the North MRSF until the end of mining operations. Collectively the MRSFs will receive waste rock in the proportions detailed in Table 20.3, with the majority (70.8%) of the waste rock comprising the DIO-PMP lithology material.

Table 20.3: Combined Lithology Breakdown to MRSF (from Stantec)

Lithology	Mass (tonnes)	Proportion (%)
HBX	29,622	0.001
EMP	69,733,300	1.99
DIO-PMP	2,475,910,143	70.8
IMP	95,019,332	2.7
OVB	306,790,628	8.78

Table 20.3: Combined Lithology Breakdown to MRSF (from Stantec)		
Lithology	**Mass (tonnes)**	**Proportion (%)**
MAG-HYD-BX	2,956,985	0.08
VOLCS	545,547,133	15.6
Total	3,495,987,143	100

- Weathering of waste rock will release solutes that will be mobilized in rainfall/snow/snowmelt that falls on the MRSF. Detailed water balance information and hydraulic characteristics for the MRSF are not yet known, but the climate is categorized as semi-arid and as such there is the potential that much of the rainfall will be lost to evaporation (particularly for smaller rainfall events). However, water is likely to percolate/infiltrate into the MRSF when rainfall is higher or during seasonal snowmelt.
- Overall, snowmelt and high rainfall events will increase retained moisture in the Main Rock Storage Facility (MRSF) rock mass and result in seepage at the base of the MRSF. Rainfall may also result in the generation of surface run-off from the MRSF, but this is mostly likely only to be significant during storm events which would have high levels of dilution and relatively short contact times with the mine rocks.
- Water that does infiltrate into the MRSF will eventually migrate downwards to the base of the MRSF or until it meets a low permeability horizon. Infiltrated water at the base of the MRSF may migrate laterally to emerge from the base of the MRSF as toe seepage at a downgradient point of discharge and migrate further to groundwater and mix and migrate with advective groundwater flow.
- The water that migrates through the MRSFs are likely to mobilize sulfide oxidation products (acidity, sulfate, metals). Dissolution of oxidation products could result in the toe seepage and seepage to groundwater being of poor water chemistry, with acidic pH and elevated solute content (i.e., trace elements and sulfate).

20.2.3.2 Open Pits

<u>During Operations</u>

During operations pits are dewatered to ensure the pit floor remains dry and to dewater the pit walls for stability. Dewatering typically occurs throughout the mine life. This water will comprise abstracted groundwater, rainfall, and runoff from the pit walls. The chemistry of the abstracted water will therefore be a mixture of groundwater plus any solutes mobilized from the wall rocks. In addition, explosives used in mining typically comprise nitrogen-based compounds (e.g., ANFO), and therefore mine contact water can contain nitrogen species (primarily nitrate and ammonium) because of spillage, leaching and/or incomplete detonation of explosives during the blasting activities.

<u>Post-closure: Pit Lake</u>

At the end of mining the dewatering of the pit will cease and the water table will rebound, which may lead to the formation of a pit lake, depending upon the climate and hydrogeology. The water table rebound will be driven by a combination of direct rainfall and snowfall, and groundwater and surface water inflows to the pits along with surface runoff and direct rainfall to the pit lake.

The degree of groundwater rebound post-closure has not yet been assessed. The pit will fill with water from rainfall and snowmelt as surface runoff from the catchments surrounding the pit and from groundwater ingress. As water levels increase within the pit there may be groundwater outflows from the pit. Depending on site layout there may potentially also be runoff and seepage from nearby MRSFs that could enter the pit. Should the pit lake water level rise to the lowest point on the pit rim, further water ingress would result in the discharge of pit waters via the decant point.

The key components of the pit lake Conceptual Study Model and solute sources that will determine pit lake quality include:

- Evaporation will be a key influence on the pit water quality given the semi-arid climate of the project site. Evaporation will remove water and leave behind the solute load, resulting in the evapoconcentration of solutes within the remaining water.
- Direct rainfall/snow into the pit will contain a minor solute load. Overall, rainfall/snow will be limited but will act to dilute in-pit waters.
- Groundwater inflows will contribute solute load. Should groundwater begin to outflow from the pit lake, this would remove water mass and solute loads and potentially impact local groundwater. Groundwater may also include solute loading from other sources, such as the MRSFs or the Low-Grade Stockpile, depending upon the hydrogeology and the direction of groundwater flows.
- Pit wall and talus weathering is the largest source of solute loadings to the pit lake. Rainfall to the pit wall surface will mobilize solute released from weathering of the wall rock and talus on the pit benches. SRK have estimated the areas that each of the lithologies will occupy in the final pit walls, as indicated in Table 20.4 and shown visually in.

Table 20.4: Pit wall areas by lithology

Material Type	Pit Wall Area (m²)	% Pit Wall Area
Unassigned	1,040,000	8%
Cover	81,900	1%
Volcanics	2,380,000	18%
LIX	4,050,000	30%
SG	522,000	4%
MX	0	0%
TR	721,000	5%
PRI BN	8500	0%
PRI BN-CPY	131,000	1%
HY	4,460,000	33%
Total	**13,394,400**	**100%**

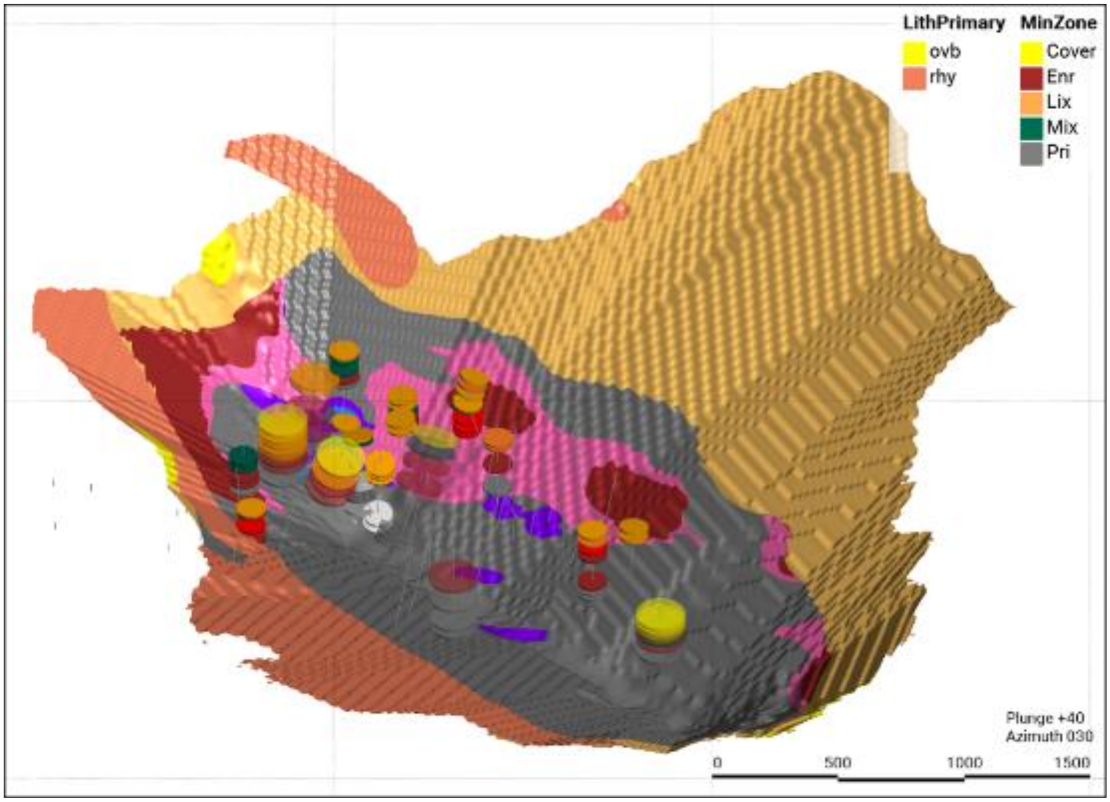


Figure 20.2: Pit wall lithologies (discs denote the position of the geochemical characterization samples

20.2.3.3 Tailings Storage Facility

20.2.3.4 Heap Leach Pad

During operations, it is expected that the HLP will be operated in such a way that there is not release of contact water to the environment.

Post-closure there will be an inventory of acidic heap leach solutions with elevated trace metal content and solute load, a feature that will persist for an unknown length of time. Post-closure, it is likely that the HLP will be closed progressively which may include the use of direct precipitation and or rinsing. Rinsing can be carried out by recirculating existing heap leach solutions or by using fresh water – with the aim of reducing parameters of concern to an acceptable level. Solutions will drain from the HLP to the process solution pond via infiltration and drain down. Water will leave the pond to downstream compliance points and the surroundings, once it is compliant with environmental standards, which may require some form of water treatment.

20.2.3.5 Low Grade Process Stockpile

Low grade material will be dumped at the north stockpile starting when primary mineralization is encountered until it is needed to supplant feed in the concentrator.

The geo-environmental conceptual model for low grade process stockpiles is the same as for MRSF. Although the material would be anticipated to be more reactive than waste rock, operational measures would mean that discharge of contact water to the environment should not occur.

Post-closure, the low-grade process stockpile is not anticipated to be present.

20.2.4 Geochemical Characterization

20.2.4.1 Site Visit

SRK Geochemists Rob Bowell and Brooke Clarkson visited the Los Azules core warehouse facility in Calingasta, November 7-9, 2022, and were accompanied by Hugo Bracamonte from McEwen Copper. The focus of the visit was to examine the drill core from the intervals selected for geochemical characterization. Field logging included a description of the lithology, alteration, mineralogy, and structure. Abundance of key mineral species, i.e., sulfides and sulphates, was approximated based on volumetric percentages visible in hand sample. Information from the detailed review of mineralogy related to alteration and copper mineralization will be applied to interpret the results of geochemical characterization test work. All available core intervals were photographed wet for later reference.

In addition to reviewing the core samples selected in August 2022 for the first phase of geochemical characterization, additional samples were targeted to represent all material types predicted in the conceptual pit design. The main gap in the Phase 1 sample group was the volcanic lithology and was addressed with additional samples selected from available core. The rest of the additional samples were selected to fill spatial gaps or to capture the variability within each material type.

20.2.4.2 Sample Selection

Geochemical characterization with respect to ARDML is currently ongoing. SRK is completing sample selection and collection over several phases. For all phases of sampling, consideration was given to the main zones of lithology, alteration, and mineralization that define the material types within the open pit, as waste rock and process feed material as follows:

- Lithologies – three main lithology groups were considered, as unconsolidated sediments (Cover or

Overburden), Volcanics including andesite, dacite, and rhyolite, and the porphyry complex including dacite porphyry (dacp), diorite, (dio), magnetite hydrothermal breccia (mag), porphyritic diorite (pordio), and rhyodacite porphyry (rydacp).

- Alteration – in the porphyry complex, chlorotic and sericitic alteration are the dominant alteration types, with minor amounts of potassic, phyllic and silic alteration.
- Mineralization – five main zones of mineralization have been considered, including the leached cap (LIX, Lixivada), a zone between the leached cap and supergene zone (MIX), Enriched Supergene zone (SG or ENR), a transitional zone (TR) between supergene and hypogene zones, and a Primary (PRI, hypogene) zone.

The cut-off copper-grade used to define waste rock was 0.2 weight % copper.

The first phase of sample collection was undertaken in August 2022 when a total of 65 samples were collected. Sample numbers and distribution throughout the considered aspects are shown in Table 20.5. The first phase of sample collection was based on a now outdated pit shell design. The samples were sent for static testing at the SGS Minerals Services laboratory in Santiago, Chile, which comprised of acid base accounting, NAG testing and whole rock assay.

Table 20.5: Waste Rock and Process Feed Samples Collected During Phase 1

Lithology	Alteration		
Cover (Alluvium), 4% of pit wall area, 9% of total tonnage.			
dacp	Chloritic	Lixiviada (Leached Cap), 34% of pit wall area, 29% of total tonnage.	3
dio			12
mag			1
pordio			3
dacp	Sericitic		3
dio			
mag			1
pordio			
dacp	Chloritic	Enriched/ SG, 13% of pit wall area, 22% of total tonnage	3
dio			4
mag			
pordio			4
rydacp			1
dacp	Sericitic		3
dio			
mag			
pordio			1
rydacp			1
dacp	Chloritic	Mixed/ TR, 8% of pit wall, 15% of total tonnage	--
dio			5
pordio			3
dacp	Chloritic	Primary/ Hypogene, 42% of pit wall, 26% of total tonnage	1
dio			7
mag			
pordio			1
dio	Sericitic		2

Table 20.5: Waste Rock and Process Feed Samples Collected During Phase 1			
Lithology	**Alteration**		
mag			4
dio	Potassic		2
		Total	**65**

Based on the first phase of sampling, a sub-set of samples were intended to be selected for long term kinetic humidity cell testing (HCT). However, sample processing errors by the testing laboratory meant that none of the samples from the first phase of testing could be utilized in a HCT program (all samples were pulverized to too fine a grain-size for HCT methods). Following this, additional samples were selected for HCT from core that was available from the Geometallurgical test work program.

A further phase of sample selection and collection is currently underway. This includes for the collection of an additional 143 samples which will undergo the same test work as specified above and fills in data gaps identified in the phase 1 samples as well as lateral and vertical variability of the main rock types in the proposed pit.

At the time of writing, results were only available from the testing of samples obtained as part of the first phase of sample collection. This includes results of ABA, NAG testing and whole rock assay results.

20.2.4.3 Results

At the time of writing, only results of ABA and NAG testing were available for samples obtained in the first phase of sample collection. This includes samples of waste rock and process feed material, as indicated in Table 20.3.

The results of the ABA and NAG tests are presented in Table 20.6 and Table 20.7: ABA and NAG testing Results for Process Feed Material for the mine rock and low-grade process feed samples respectively.

<u>Sulfur Content</u>

Results for total sulfur for all samples, inclusive of waste rock and process feed material, ranged from 0.04% to 2.9%. There is generally a strong correlation between total sulfur and sulfide sulfur (as shown in Figure 20.3), except for some samples and for samples associated with LIX materials. In the LIX samples, sulfate sulfur is the more dominant form of sulfur. As shown in Figure 20.3, higher sulfur concentrations are associated with process feed grade material, of the ENR mineralization type.

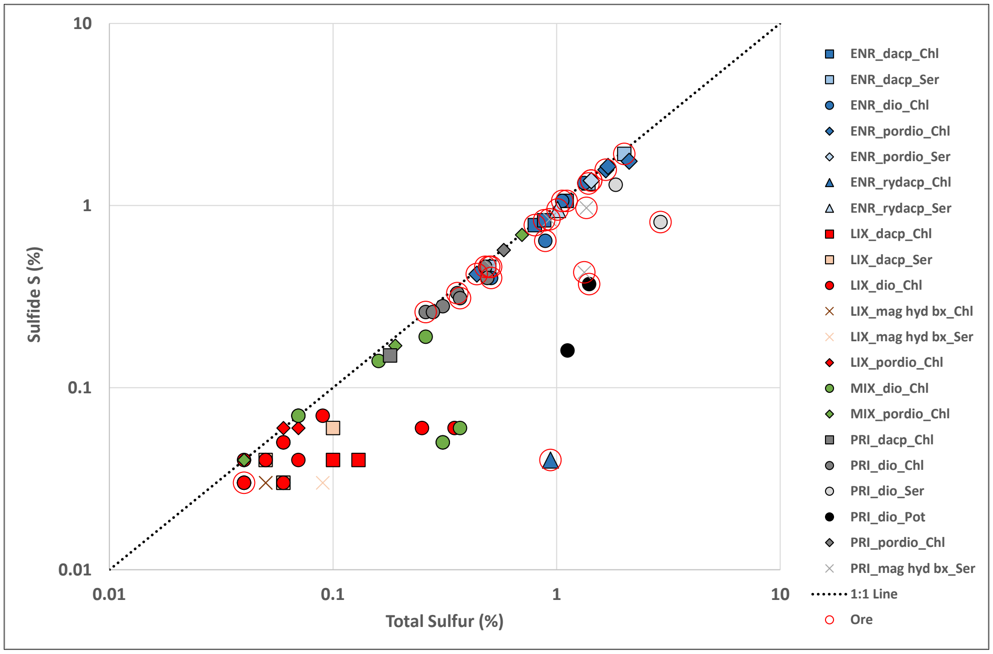


Figure 20.3: Sulfide sulfur plotted against total sulfur

Paste pH

Paste pH results were mostly above 6.5, as shown in Figure 20.4. For samples associated with ENR and PRI (waste rock and process feed material), paste pH is negatively correlated with sulfide content, with lower paste pH values associate with higher sulfide sulfur. Four waste rock samples from the LIX and MIX mineralization zone (each of the lithology diorite and chloritic alteration) had lower paste pH values of between 4.5 and 5.5.

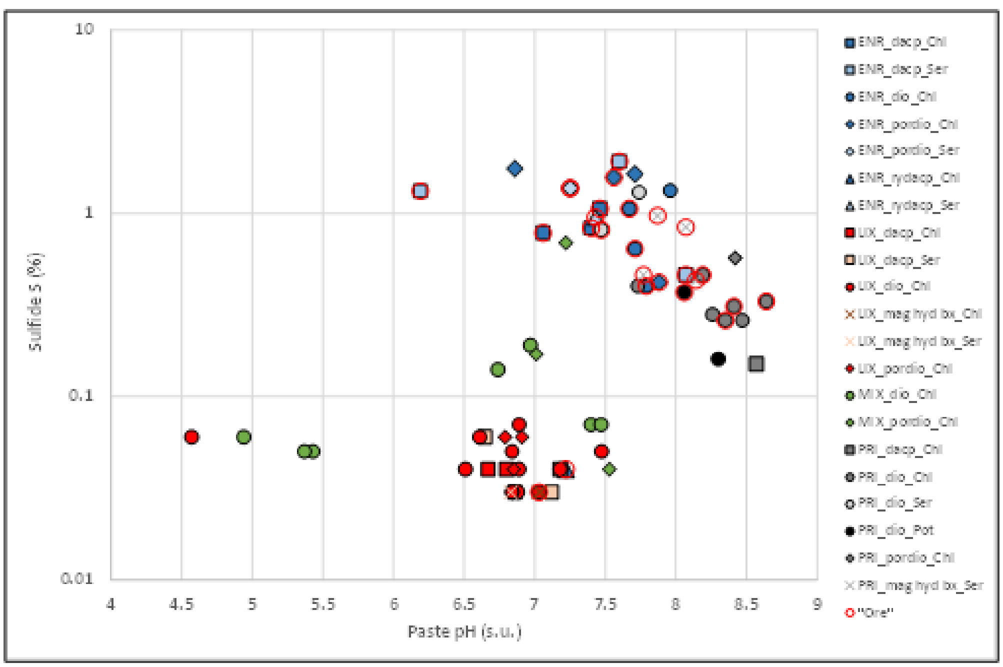


Figure 20.4: Sulfide sulfur plotted against paste pH

Neutralization potential

The Neutralization potential (NP) of the samples was evaluated by two methods: analysis for Modified NP and using Total CO_3 (both as kg $CaCO_3$/t).

Results for Modified NP were low across all samples for all material types, ranging from 1.1 to 29 kg $CaCO_3$/t. NP calculated from CO_3 (%) was also low, between 0.83 kg $CaCO_3$/t and 16 kg $CaCO_3$/t. Figure 20.5 shows that the Modified NP exceeds NP calculated from CO_3 (%) indicating that NP from slower reacting silicate is more important than from faster reacting carbonate minerals. This finding is supporting by Fizz test results, which indicated a fizz rating of 1, "no fizz" for all materials indicating an absence of fast reacting carbonate minerals.

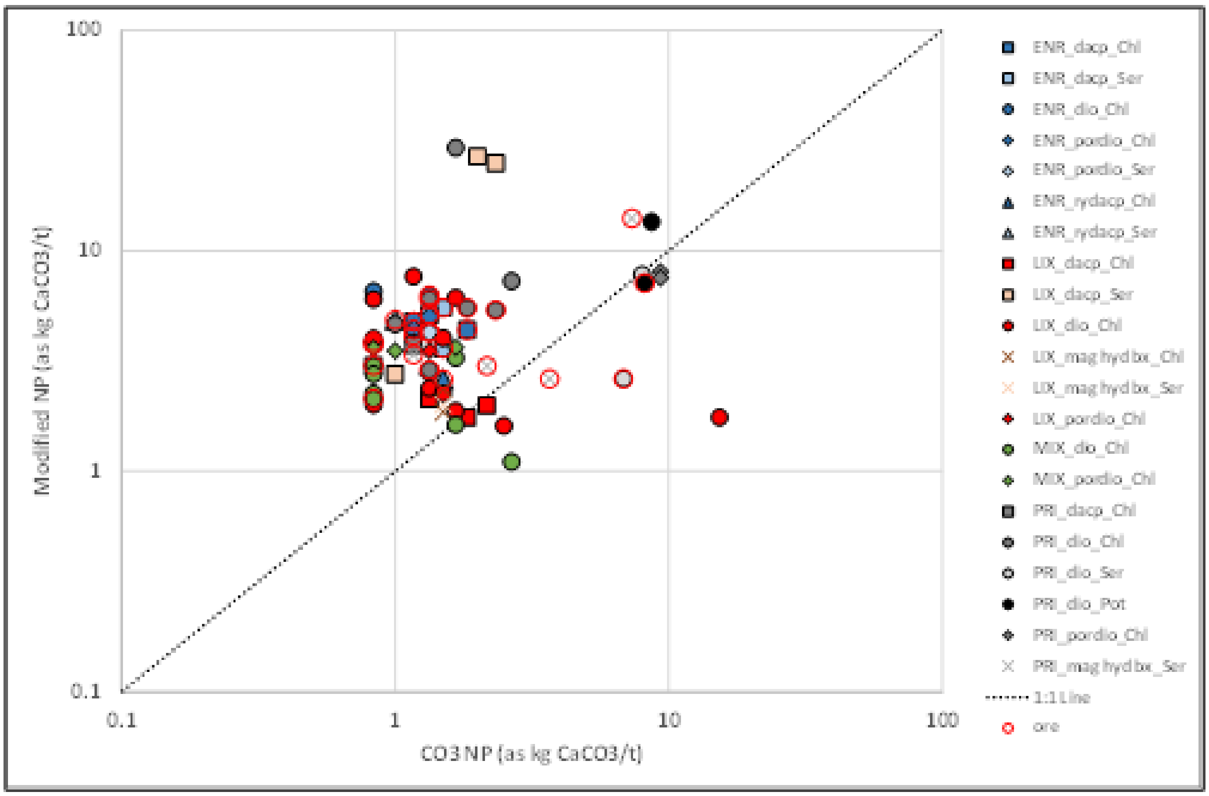


Figure 20.5: Modified NP plotted against carbonate NP

Acid rock drainage potential

Figure 20.6 shows sulfide sulfur plotted against NPR. A plot of NAG pH and NAG acidity testing is shown in Figure 20.7. These figures show that most samples are classified as either having an uncertain acid generating potential or are potentially acid generating irrespective of material type. There are few samples classified as non-PAG, and there is no or negligible carbonate present in the samples. The ARD potential of waste rock samples is:

- Three out of three ENR samples are PAG, comprising of dio and pordio litholgies and chloritic alteration.
- 6 out of 8 PRI samples covering three lithologies (dacp, dio and pordio) and chloritic potassic and sericitic alteration were PAG with the remaining samples classified with uncertain acid generating

characteristics.

- 6 out of 10 MIX samples (all dio or pordio lithology and chloritic alteration) were PAG with the remaining samples classified with uncertain acid generating characteristics.
- 16 out of 23 LIX samples, irrespective of lithology have uncertain acid generating characteristics. One LIX samples was classified as PAG (LIX_dio_chl), with the remaining samples classified as non-PAG (dio and dacp lithologies). All samples were of chloritic alteration.

NAG pH testing of waste rock samples was supportive of the NPR findings, although many of the samples classified as having an uncertain acid generating potential had NAG pH values that indicating non-PAG characteristics. NAG acidity testing showed that the ENR samples were all PAG.

For the process feed samples, 23 out of 26 samples were classified as PAG, with three uncertain acid generating characteristics. NAG testing supported the NPR classification, with 24 out of 26 samples indicating acid generating conditions. NAG acidity results showed that 15 of the 26 samples could be classified as having a low capacity to generate acid.

In terms of reactivity the samples indicate slow reactivity but potential for long term acid generation indicating ARD is more likely to be a closure issue than operational (Figure 20.8).

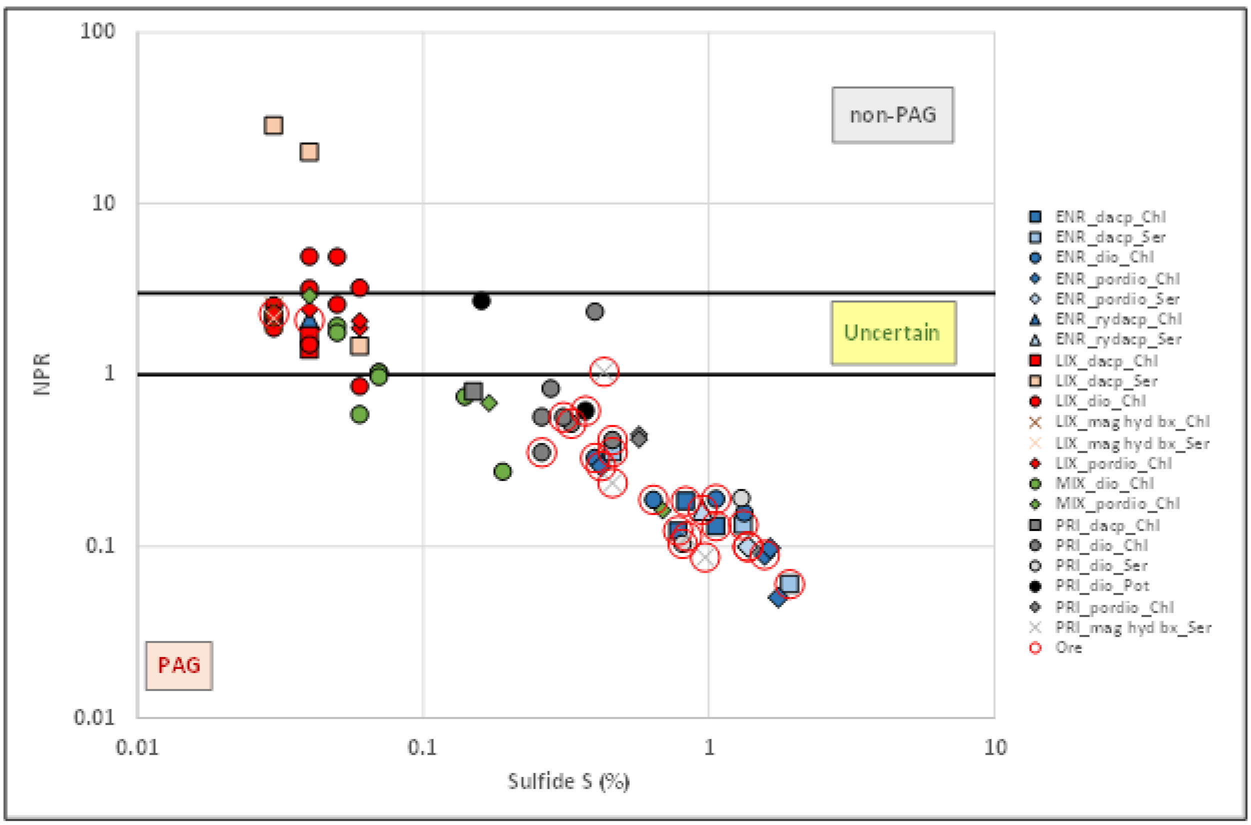


Figure 20.6: NPR plotted against sulfide sulfur

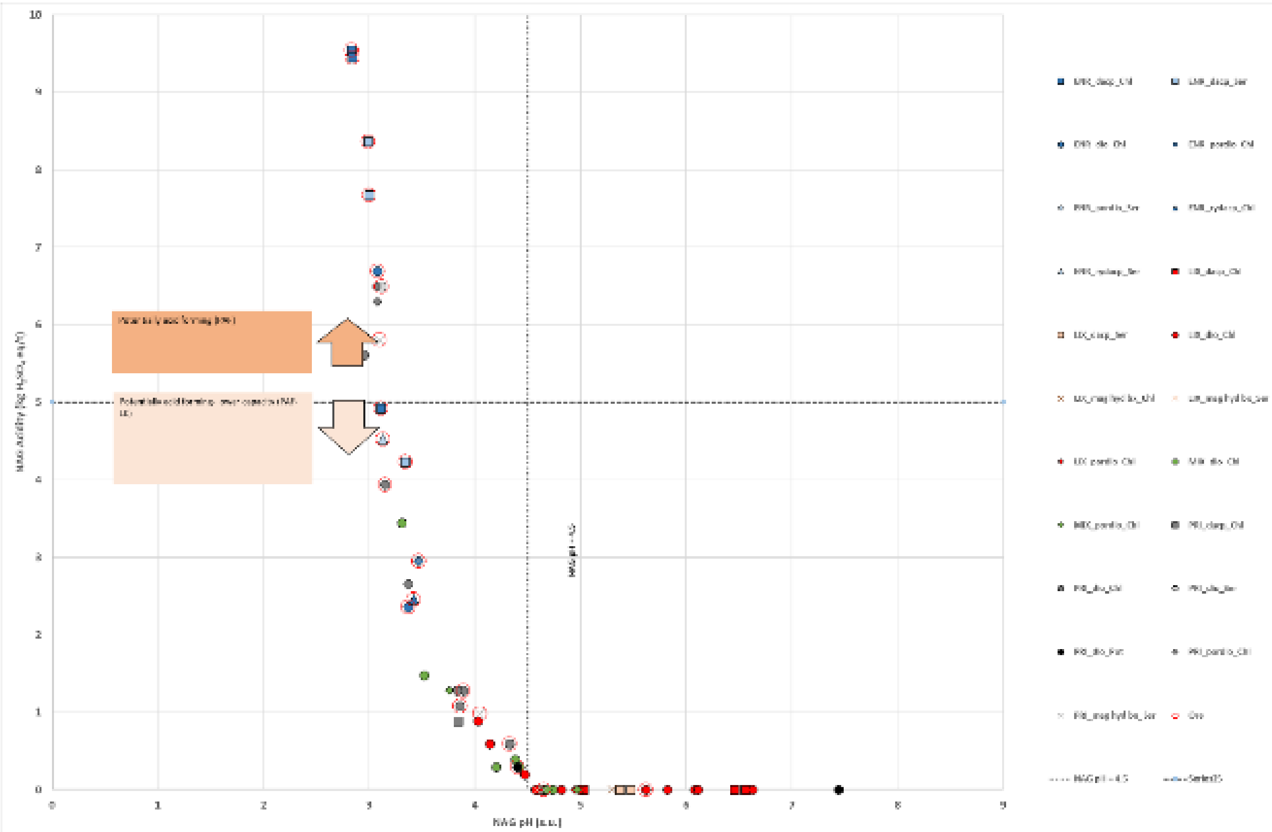


Figure 20.7: NAG pH plotted against NAG acidity

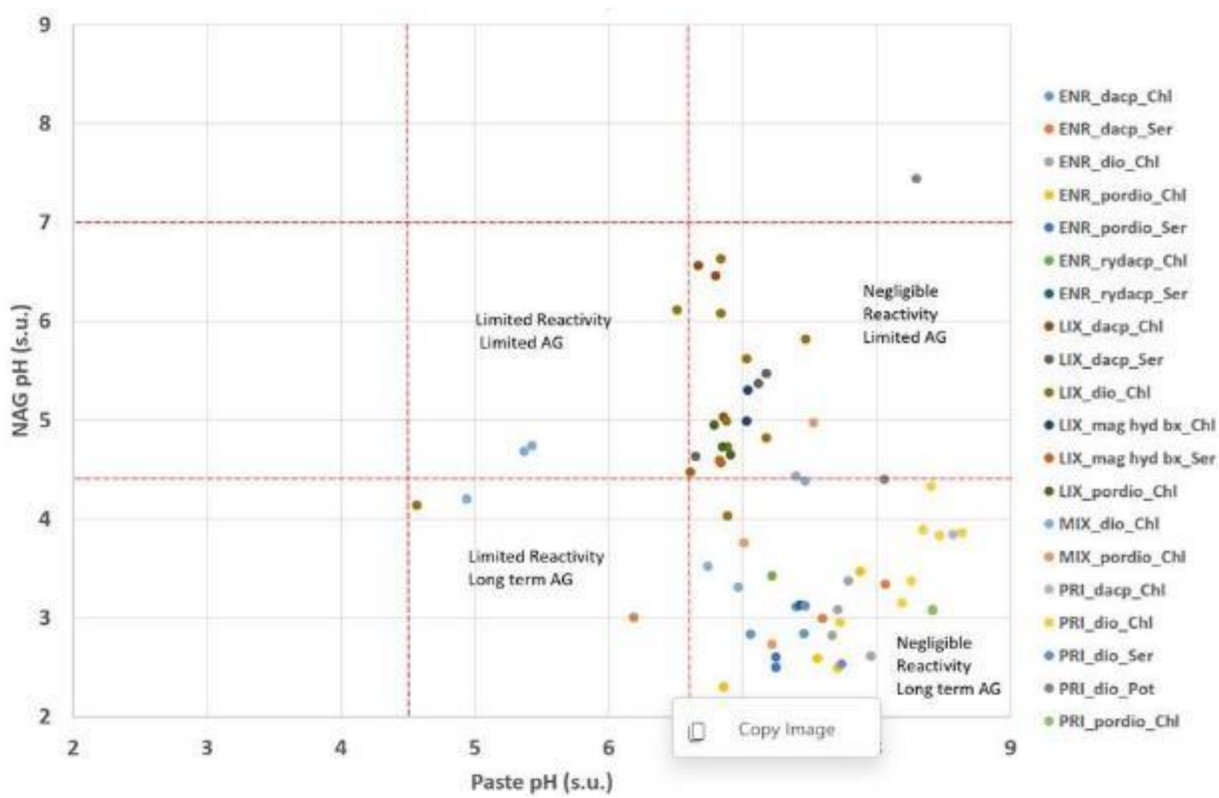


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Figure 20.8: paste pH plotted against NAG final pH

Table 20.6: ABA and NAG testing Results for Waste Rock													
Sample ID	Combined Lith_Alt_Min	Paste pH	Fizz	Total Sulfur	Sulfide Sulfur	Carbonate Carbon	Sulfate	Modified NP	CO3 NP	Acid Potential	NPR	NAG pH	NAG Acidity
		s.u.	-	%	%	%	%	kg $CaCO_3$/t	kg $CaCO_3$/t	kg $CaCO_3$/t	-	s.u.	kgH_2SO_4/t
AZ17121 (208-218)	ENR_dio_Chl	8.00	1	1.30	1.30	0.05	0.07	6.50	0.83	42.00	0.16	2.60	18.00
AZ17121 (68-76)	ENR_pordio_Chl	6.90	1	2.10	1.80	0.08	0.11	2.80	1.30	55.00	0.05	2.30	30.00
AZ17123 (183-193)	ENR_pordio_Chl	7.70	1	1.70	1.60	0.08	0.04	5.00	1.30	51.00	0.10	2.50	22.00
AZ17130 (88-98)	LIX_dacp_Chl	6.90	1	0.06	0.03	0.13	0.10	2.00	2.20	0.94	2.10	5.00	-
AZ17134 (40-50)	LIX_dacp_Chl	6.70	1	0.10	0.04	0.11	0.20	1.80	1.80	1.30	1.40	6.60	-
AZ17134 (80-90)	LIX_dacp_Chl	6.80	1	0.13	0.04	0.08	0.28	2.10	1.30	1.30	1.70	6.50	-
AZ17120 (46-56)	LIX_dacp_Ser	6.70	1	0.10	0.06	0.06	0.25	2.80	1.00	1.90	1.50	4.60	-
AZ17129 (60-70)	LIX_dacp_Ser	7.10	1	0.06	0.03	0.12	0.08	27.00	2.00	0.94	28.00	5.40	-
AZ17129 (80-90)	LIX_dacp_Ser	7.20	1	0.05	0.04	0.14	0.05	25.00	2.30	1.30	20.00	5.50	-
AZ17120 (6-16)	LIX_dio_Chl	6.60	1	0.35	0.06	0.05	0.64	6.00	0.83	1.90	3.20	4.50	0.20
AZ17132 (90-100)	LIX_dio_Chl	6.90	1	0.05	0.04	0.10	0.08	1.90	1.70	1.30	1.50	4.70	-
AZ18136 (12-22)	LIX_dio_Chl	6.90	1	0.06	0.03	0.08	0.09	2.40	1.30	0.94	2.50	5.00	-
AZ18136 (80-90)	LIX_dio_Chl	6.80	1	0.04	0.03	0.92	0.06	1.80	15.00	0.94	1.90	4.60	-
AZ18136 (132-142)	LIX_dio_Chl	6.90	1	0.09	0.07	0.09	0.01	2.30	1.50	2.20	1.00	4.00	0.89
AZ17133 (60-70)	LIX_dio_Chl	6.80	1	0.04	0.03	0.05	0.03	2.00	0.83	0.94	2.10	6.10	-
AZ17123 (51-61)	LIX_dio_Chl	6.80	1	0.06	0.05	0.09	0.12	4.00	1.50	1.60	2.60	6.60	-
AZ17128 (28-38)	LIX_dio_Chl	4.60	1	0.25	0.06	0.15	0.32	1.60	2.50	1.90	0.86	4.10	0.59
AZ17130 (60-70)	LIX_dio_Chl	6.50	1	0.07	0.04	0.10	0.09	6.10	1.70	1.30	4.90	6.10	-
AZ17126 (70-80)	LIX_dio_Chl	7.50	1	0.06	0.05	0.07	0.07	7.60	1.20	1.60	4.90	5.80	-
AZ17127 (90-100)	LIX_dio_Chl	7.20	1	0.04	0.04	0.05	0.04	4.00	0.83	1.30	3.20	4.80	-
AZ18136 (46-56)	LIX_mag hyd bx_Ser	6.80	1	0.09	0.03	0.09	0.17	2.00	1.50	0.94	2.10	4.60	-
AZ17130 (130-137.5)	LIX_mag hyd bx_Chl	7.00	1	0.05	0.03	0.09	0.15	1.90	1.50	0.94	2.00	5.30	-
AZ17130 (130-137.5)	LIX_mag hyd bx_Chl	7.00	1	0.05	0.03	0.09	0.15	2.40	1.50	0.94	2.50	5.00	-
AZ17121 (55-64)	LIX_pordio_Chl	6.80	1	0.06	0.06	0.08	0.10	3.50	1.30	1.90	1.90	5.00	-
AZ17131 (50-60)	LIX_pordio_Chl	6.90	1	0.04	0.04	0.05	0.03	3.00	0.83	1.30	2.40	4.70	-
AZ17127 (60-70)	LIX_pordio_Chl	6.90	1	0.07	0.06	0.07	0.04	3.90	1.20	1.90	2.10	4.70	-
AZ17123 (65-75)	MIX_dio_Chl	6.70	1	0.16	0.14	0.10	0.12	3.30	1.70	4.40	0.75	3.50	1.50
AZ17132 (110-120)	MIX_dio_Chl	7.40	1	0.07	0.07	0.05	0.03	2.30	0.83	2.20	1.00	4.40	0.30
AZ17132 (110-120)	MIX_dio_Chl	7.50	1	0.07	0.07	0.05	0.03	2.10	0.83	2.20	0.97	4.40	0.39
AZ18136 (150-160)	MIX_dio_Chl	7.00	1	0.26	0.19	0.10	0.05	1.60	1.70	5.90	0.27	3.30	3.40
AZ17128 (54-64)	MIX_dio_Chl	5.40	1	0.31	0.05	0.05	0.80	3.00	0.83	1.60	1.90	4.70	-
AZ17128 (54-64)	MIX_dio_Chl	5.40	1	0.31	0.05	0.05	0.80	2.80	0.83	1.60	1.80	4.70	-
AZ17128 (110-120)	MIX_dio_Chl	4.90	1	0.37	0.06	0.16	0.55	1.10	2.70	1.90	0.59	4.20	0.30
AZ17123 (77-87)	MIX_pordio_Chl	7.00	1	0.19	0.17	0.10	0.21	3.60	1.70	5.30	0.68	3.80	1.30
AZ17123 (103-113)	MIX_pordio_Chl	7.20	1	0.70	0.69	0.06	0.12	3.50	1.00	22.00	0.16	2.70	13.00
AZ17127 (126-136)	MIX_pordio_Chl	7.50	1	0.04	0.04	0.05	0.04	3.60	0.83	1.30	2.90	5.00	-
AZ17129 (310-320)	PRI_dacp_Chl	8.60	1	0.18	0.15	0.07	0.04	3.70	1.20	4.70	0.80	3.80	0.89
AZ17120 (322-332)	PRI_dio_Chl	8.30	1	0.31	0.28	0.16	0.07	7.30	2.70	8.80	0.83	3.40	2.70
AZ17131 (264-274)	PRI_dio_Chl	8.50	1	0.28	0.26	0.06	0.08	4.60	1.00	8.10	0.57	3.80	1.30
AZ17128 (174-184)	PRI_dio_Chl	7.70	1	0.49	0.40	0.10	0.07	29.00	1.70	13.00	2.30	3.00	5.60
AZ17132 (420-430)	PRI_dio_Pot	8.30	2	1.10	0.16	0.52	3.00	14.00	8.70	5.00	2.70	7.40	-
AZ17121 (240-250)	PRI_dio_Ser	7.70	1	1.80	1.30	0.48	1.20	7.80	8.00	41.00	0.19	2.50	21.00
AZ17123 (221-231)	PRI_pordio_Chl	8.40	1	0.58	0.57	0.56	0.06	8.00	9.30	18.00	0.45	3.10	6.30
AZ17123 (221-231)	PRI_pordio_Chl	8.40	1	0.58	0.57	0.56	0.06	7.50	9.30	18.00	0.42	3.10	6.50

	indicates PAG characteristics
	indicates uncertain acid generating characteristics
	indicates non-PAG characteristics

Table 20.7: ABA and NAG testing Results for Process Feed Material													
Sample ID	Combined Lith_Alt_Min	Paste pH	Fizz	Total Sulfur	Sulfide Sulfur	Carbonate Carbon	Sulfate	Modified NP	CO3 NP	Acid Potential	NPR	NAG pH	NAG Acidity
		s.u.	-	%	%	%	%	kg $CaCO_3$/t	kg $CaCO_3$/t	kg $CaCO_3$/t	-	s.u.	kgH_2SO_4/t
AZ17120 (124-134)	ENR_dacp_Chl	7.10	1	0.80	0.78	0.05	0.10	3.00	0.83	24.00	0.12	2.80	9.50
AZ17120 (268-278)	ENR_dacp_Chl	7.40	1	0.88	0.83	0.07	0.07	4.80	1.20	26.00	0.18	3.10	4.90
AZ17127 (200-210)	ENR_dacp_Chl	7.50	1	1.10	1.10	0.11	0.08	4.40	1.80	33.00	0.13	2.80	9.40
AZ17120 (86-96)	ENR_dacp_Ser	6.20	1	1.40	1.30	0.09	0.19	5.50	1.50	41.00	0.13	3.00	7.70
AZ17126 (136-146)	ENR_dacp_Ser	7.60	1	2.00	1.90	0.09	0.09	3.60	1.50	60.00	0.06	3.00	8.40
AZ17126 (176-186)	ENR_dacp_Ser	8.10	1	0.50	0.46	0.08	0.04	5.10	1.30	14.00	0.36	3.30	4.20
AZ17121 (196-206)	ENR_dio_Chl	7.70	1	1.10	1.10	0.08	0.07	6.30	1.30	33.00	0.19	2.80	11.00
AZ17127 (180-190)	ENR_dio_Chl	7.70	1	0.89	0.64	0.05	0.05	3.80	0.83	20.00	0.19	3.10	6.70
AZ17128 (152-162)	ENR_dio_Chl	7.80	1	0.51	0.40	0.07	0.10	4.10	1.20	13.00	0.33	3.40	2.40
AZ17123 (153-163)	ENR_pordio_Chl	7.60	1	1.70	1.60	0.07	0.06	4.40	1.20	49.00	0.09	2.60	18.00
AZ17127 (154-164)	ENR_pordio_Chl	7.90	1	0.44	0.42	0.05	0.06	3.90	0.83	13.00	0.29	3.50	3.00
AZ17121 (138-148)	ENR_pordio_Ser	7.30	1	1.40	1.40	0.08	0.11	4.30	1.30	43.00	0.10	2.60	16.00
AZ17121 (138-148)	ENR_pordio_Ser	7.30	1	1.40	1.40	0.08	0.11	4.30	1.30	43.00	0.10	2.50	15.00
AZ17129 (190-200)	ENR_rydacp_Chl	7.20	1	0.94	0.04	0.09	0.07	2.60	1.50	1.30	2.10	3.40	2.50
AZ17126 (202-212)	ENR_rydacp_Ser	7.40	1	1.00	0.95	0.06	0.10	4.90	1.00	30.00	0.16	3.10	4.50
AZ17133 (100-110)	LIX_dio_Chl	7.00	1	0.04	0.03	0.05	0.02	2.10	0.83	0.94	2.30	5.60	-
AZ17120 (296-306)	PRI_dio_Chl	8.20	1	0.48	0.46	0.08	0.08	6.00	1.30	14.00	0.42	3.20	3.90
AZ17131 (244-254)	PRI_dio_Chl	8.40	1	0.26	0.26	0.08	8.40	2.90	1.30	8.10	0.35	3.90	1.30
AZ17131 (322-332)	PRI_dio_Chl	8.60	1	0.36	0.33	0.14	0.06	5.40	2.30	10.00	0.52	3.90	1.10
AZ17132 (364-374)	PRI_dio_Chl	8.40	1	0.37	0.31	0.11	0.05	5.50	1.80	9.70	0.57	4.30	0.59
AZ17131 (390-400)	PRI_dio_Pot	8.10	1	1.40	0.37	0.49	3.30	7.10	8.20	12.00	0.62	4.40	0.30
AZ17131 (402-412)	PRI_dio_Ser	7.50	1	2.90	0.81	0.41	7.00	2.60	6.80	25.00	0.10	3.10	6.50
AZ17130 (440-450)	PRI_mag hyd bx_Ser	8.10	2	1.30	0.43	0.44	2.70	14.00	7.30	13.00	1.00	4.70	-
AZ17130 (496-506)	PRI_mag hyd bx_Ser	7.90	1	1.40	0.97	0.22	1.20	2.60	3.70	30.00	0.09	3.10	5.80
AZ17132 (286-296)	PRI_mag hyd bx_Ser	7.80	1	0.51	0.46	0.07	0.02	3.40	1.20	14.00	0.23	4.10	0.98
AZ17132 (306-316)	PRI_mag hyd bx_Ser	8.10	1	0.93	0.84	0.13	0.08	3.00	2.20	26.00	0.11	3.50	3.00

	Indicates PAG characteristics
	Indicates uncertain acid generating characteristics
	Indicates non-PAG characteristics

Whole Rock Assay

A multi element analysis of the waste materials was completed to provide an upper limit of available metals in the samples. The results of the multi element analysis were assessed using the Geochemical Abundance Index (GAI) (Mason, 1996), which compares the concentration of an element in each sample to its average crustal abundance. GAI values are particularly useful in determining the relative enrichment of elements based on lithology and may be used to identify elements enriched above average crustal concentrations. GAI values are calculated as follows:

$$\mathrm{GAI_i} = \log\left[\frac{\mathrm{C_i}}{1.5\ \mathrm{S_i}}\right]$$

Where C is the concentration of an element as determined from the multi element assay and S is the average crustal abundance of the element of interest (Mason, 1966). Materials are then assigned a GAI value between 0 and 6, depending on the degree of enrichment. According to the protocol given by Förstner et al (1993), a GAI value greater than three indicates significant enrichment.

Table 20.8: Interpretation of GAI values for multi element assay data

GAI Value	Interpretation
0	< 3 times average crustal concentrations
1	3 to 6 times average crustal concentrations
2	6 to 12 times average crustal concentrations
3	12 to 24 times average crustal concentrations
4	24 to 48 times average crustal concentrations
5	48 to 96 times average crustal concentrations
6	> 96 times average crustal concentrations

A summary of the whole rock assay results for the mine rock and process feed samples is shown in Table 20.9 and Table 20.10: Whole Rock Assay Results for Process Feed Materials. In summary, the results show that:

- For the mine rock, arsenic, bismuth, antimony, and selenium have GAI of >3 in most samples, although for selenium this is affected by an insufficiently low limit of detection in half of the samples. Occasionally, copper, molybdenum, lead, and tungsten have GAI values of >3.
- For the process feed samples, arsenic, bismuth, copper, antimony, molybdenum, and selenium have GAI of >3 in most samples, although for selenium this is also affected by an insufficiently low limit of detection in 14 out of 25 samples. Occasionally, tungsten is recorded with GAI values of >3.

Table 20.9: Whole Rock Assay Results for Waste Rock																	
Sample ID	Combined Lith_Alt_Min	As	Ba	Bi	Ca	Cu	Fe	K	Mg	Mn	Mo	Pb	Sb	Se	Sn	W	Zn
		ppm	ppm	ppm	%	%	%	%	%	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
AZ17121 (208-218)	ENR_dio_Chl	4	570	11	0.76	0.14	2.2	2.5	0.6	150	6	12	46	12	5	6	64
AZ17121 (68-76)	ENR_pordio_Chl	9	510	12	0.2	0.11	3	2.6	0.51	69	4	25	80	36	8	240	62
AZ17123 (183-193)	ENR_pordio_Chl	1	590	7	0.34	0.15	2.2	2.2	0.55	130	2	67	36	12	5	3	140
AZ17130 (88-98)	LIX_dacp_Chl	61	800	3	0.05	0.03	2	3.5	0.43	180	40	32	62	12	5	6	30
AZ17134 (40-50)	LIX_dacp_Chl	79	960	10	0.07	0.032	2.6	2.4	0.33	2400	63	70	47	23	5	6	50
AZ17134 (80-90)	LIX_dacp_Chl	84	920	7	0.09	0.019	2.1	2.4	0.3	830	39	63	32	19	5	7	56
AZ17120 (46-56)	LIX_dacp_Ser	55	600	9	0.09	0.021	3	2.5	0.38	49	27	22	62	39	9	20	63
AZ17129 (60-70)	LIX_dacp_Ser	100	890	11	0.17	0.04	2.2	2.6	0.44	83	7	23	66	12	5	12	86
AZ17129 (80-90)	LIX_dacp_Ser	43	790	13	0.18	0.055	2.1	2.6	0.67	270	5	110	67	12	5	6	84
AZ17120 (6-16)	LIX_dio_Chl	69	710	9	0.1	0.049	3.9	2.4	0.4	73	17	32	98	53	6	9	58
AZ17132 (90-100)	LIX_dio_Chl	49	610	9	0.15	0.076	2.3	2.5	0.59	320	28	51	37	12	5	4	100
AZ18136 (12-22)	LIX_dio_Chl	7	910	1	0.59	0.037	2.4	2.3	0.41	410	12	22	5	12	5	2	170
AZ18136 (80-90)	LIX_dio_Chl	12	1000	1	0.27	0.041	3.2	2.5	0.45	110	15	66	5	12	5	2	76
AZ18136 (132-142)	LIX_dio_Chl	1	520	2	0.24	0.072	3.7	2.3	0.74	130	25	150	7	14	5	3	150
AZ17133 (60-70)	LIX_dio_Chl	38	570	8	0.13	0.045	2.3	1.9	0.61	350	36	40	57	18	5	6	84
AZ17123 (51-61)	LIX_dio_Chl	37	770	8	0.17	0.074	3.5	2.7	0.31	920	30	430	68	18	5	4	55
AZ17128 (28-38)	LIX_dio_Chl	9	660	13	0.14	0.025	2.8	3	0.45	49	61	24	84	14	5	6	30
AZ17130 (60-70)	LIX_dio_Chl	70	1200	26	0.56	0.11	5	2.4	1	790	30	110	150	24	6	2	190
AZ17126 (70-80)	LIX_dio_Chl	1	420	13	0.16	0.036	3.1	2.1	0.79	310	5	28	51	12	5	2	130
AZ17127 (90-100)	LIX_dio_Chl	1	960	9	0.14	0.028	2.2	3	0.49	130	13	11	33	12	5	5	100
AZ18136 (46-56)	LIX_mag hyd bx_Ser	18	740	1	0.15	0.02	1.6	2.5	0.23	65	11	100	4	12	5	11	120
AZ17130 (130-137.5)	LIX_mag hyd bx_Chl	11	690	20	0.14	0.073	6.5	2.8	0.56	110	10	63	170	28	7	5	67
AZ17121 (55-64)	LIX_pordio_Chl	1	630	1	0.23	0.042	3.1	2.9	0.37	110	2	1	110	12	5	2	110
AZ17131 (50-60)	LIX_pordio_Chl	110	660	8	0.2	0.037	2.4	2.7	0.61	250	28	31	50	12	5	2	80
AZ17127 (60-70)	LIX_pordio_Chl	6	580	13	0.11	0.03	3.4	2.1	0.8	140	11	13	60	12	5	8	110
AZ17123 (65-75)	MIX_dio_Chl	7	730	4	0.06	0.059	2.9	3.4	0.28	150	22	120	58	15	5	4	18
AZ17132 (110-120)	MIX_dio_Chl	26	620	11	0.27	0.12	2.5	2.3	0.92	330	13	46	46	12	5	4	120
AZ17132 (110-120)	MIX_dio_Chl	26	620	11	0.27	0.12	2.5	2.3	0.92	330	13	46	46	12	5	4	120
AZ18136 (150-160)	MIX_dio_Chl	1	660	2	0.18	0.096	3.6	2.9	0.89	99	22	180	6	13	6	2	210
AZ17128 (54-64)	MIX_dio_Chl	4	590	15	0.11	0.015	3.2	2.3	0.56	49	22	15	57	22	5	3	25
AZ17128 (110-120)	MIX_dio_Chl	64	1100	18	0.17	0.025	4.7	3.2	0.5	110	88	29	150	28	6	5	41
AZ17123 (77-87)	MIX_pordio_Chl	1	830	5	0.06	0.06	3	3.6	0.29	200	28	130	49	14	5	3	21
AZ17123 (103-113)	MIX_pordio_Chl	30	780	5	0.06	0.064	2.4	3.6	0.31	33	17	45	42	14	5	5	25
AZ17127 (126-136)	MIX_pordio_Chl	14	810	10	0.15	0.044	2.4	3	0.65	150	10	45	41	12	5	4	110
AZ17129 (310-320)	PRI_dacp_Chl	2	820	6	0.4	0.13	1.6	2.9	0.42	300	13	29	48	12	5	3	160
AZ17120 (322-332)	PRI_dio_Chl	1	620	10	0.55	0.081	1.9	2.9	0.55	280	3	34	53	22	7	23	100
AZ17131 (264-274)	PRI_dio_Chl	1	770	12	0.56	0.16	2.4	2.4	0.79	310	82	43	44	12	5	4	180
AZ17128 (174-184)	PRI_dio_Chl	170	730	24	0.55	0.18	2.9	2.6	1.1	210	19	18	67	17	5	10	94
AZ17132 (420-430)	PRI_dio_Pot	15	640	18	2.3	0.093	2	1.9	0.75	480	3	23	41	17	5	8	110
AZ17121 (240-250)	PRI_dio_Ser	5	630	9	1.1	0.075	2.5	2.5	0.6	350	4	19	43	12	5	5	110
AZ17123 (221-231)	PRI_pordio_Chl	11	730	9	0.72	0.059	2.6	3.1	0.78	910	5	87	41	12	5	5	380

	Indicates element concentration with a GAI >3

Table 20.10: Whole Rock Assay Results for Process Feed Materials

Sample ID	Combined Lith_Alt_Min	As ppm	Ba ppm	Bi ppm	Ca %	Cu %	Fe %	K %	Mg %	Mn ppm	Mo ppm	Pb ppm	Sb ppm	Se ppm	Sn ppm	W ppm	Zn ppm
AZ17120 (124-134)	ENR_dacp_Chl	1	670	13	0.12	0.32	1.5	2.3	0.25	30	20	36	27	18	6	26	19
AZ17120 (268-278)	ENR_dacp_Chl	250	690	20	0.2	0.68	2.2	2.8	0.57	110	8	20	42	29	7	39	120
AZ17127 (200-210)	ENR_dacp_Chl	1	700	14	0.15	0.46	2.1	2.9	0.57	120	24	15	36	12	5	8	190
AZ17120 (86-96)	ENR_dacp_Ser	11	710	16	0.1	0.8	2	2.2	0.29	35	14	84	55	24	6	41	57
AZ17126 (136-146)	ENR_dacp_Ser	1	370	21	0.11	0.7	2.9	2.5	0.6	160	8	100	57	12	5	7	160
AZ17126 (176-186)	ENR_dacp_Ser	1	650	14	0.26	0.25	2.2	2.4	0.64	370	15	120	40	12	5	5	130
AZ17121 (196-206)	ENR_dio_Chl	2	310	5	0.25	0.12	0.9	1.1	0.26	65	4	8	21	12	5	3	39
AZ17127 (180-190)	ENR_dio_Chl	10	730	15	0.16	0.36	2.2	1.9	0.64	150	25	1	45	21	5	8	68
AZ17128 (152-162)	ENR_dio_Chl	3	610	35	0.25	0.41	2.6	2.2	1.1	210	68	24	93	18	5	7	87
AZ17123 (153-163)	ENR_pordio_Chl	6	670	10	0.21	0.25	2.3	2.4	0.59	88	3	55	38	12	5	4	69
AZ17127 (154-164)	ENR_pordio_Chl	1	340	1	0.02	0.36	2	1.7	0.09	52	28	30	14	12	5	2	76
AZ17121 (138-148)	ENR_pordio_Ser	1	690	14	0.22	0.37	2.1	3	0.49	110	3	21	73	26	8	60	64
AZ17129 (190-200)	ENR_rydacp_Chl	29	570	13	0.12	0.78	1.6	2.8	0.38	130	25	76	48	12	5	12	150
AZ17126 (202-212)	ENR_rydacp_Ser	1	430	16	0.18	0.68	2.1	2	0.57	250	17	140	31	12	5	8	200
AZ17133 (100-110)	LIX_dio_Chl	56	570	15	0.11	0.25	2.1	2.1	0.72	360	30	20	45	16	5	8	79
AZ17120 (296-306)	PRI_dio_Chl	1	710	9	0.28	0.25	1.8	2.6	0.53	120	10	14	31	22	5	25	80
AZ17131 (244-254)	PRI_dio_Chl	1	710	7	0.23	0.26	1.3	2	0.55	290	76	18	18	12	5	2	230
AZ17131 (322-332)	PRI_dio_Chl	180	630	16	0.43	0.31	2	2.2	0.75	270	61	18	66	12	6	11	80
AZ17132 (364-374)	PRI_dio_Chl	220	820	20	0.64	0.34	2	2.6	0.75	130	43	22	60	18	5	23	140
AZ17131 (390-400)	PRI_dio_Pot	63	550	17	1.6	0.28	2.3	2.2	0.33	510	20	10	33	12	6	4	66
AZ17131 (402-412)	PRI_dio_Ser	310	460	12	2.3	0.44	2.2	2.3	0.46	460	93	14	42	12	5	7	72
AZ17130 (440-450)	PRI_mag hyd bx_Ser	1	630	7	1.4	0.3	1.4	2.1	0.38	330	27	33	18	12	5	2	200
AZ17130 (496-506)	PRI_mag hyd bx_Ser	86	940	17	0.66	0.74	2	3.6	0.71	330	68	66	62	12	5	4	380

AZ17132 (286-296)	PRI_mag hyd bx_Ser	30	350	16	0.17	0.55	1.6	1.7	0.52	210	36	63	28	16	5	11	140
AZ17132 (306-316)	PRI_mag hyd bx_Ser	17	450	20	0.12	0.89	2	2.2	0.52	270	60	30	52	19	5	7	160

	Indicates element concentration with a GAI >3

20.2.4.4 Future Workplan

Laboratory Testing

The first phase of samples has been completed and much of the analysis of those samples is complete. Additional sampling has been conducted and sample collection continues, with additional mine rock and process feed material samples currently being collected for geochemical testing. In general, all future samples will undergo the following test work:

- Acid base accounting (ABA)
- Net acid generation (NAG) testing.
- Whole Rock Analysis/Multi-element analysis that includes 4-acid digestion followed by inductively coupled plasma mass spectrometry (ICP-MS). Note, that results of this testing have yet to be received for any samples.

It is also proposed that a sub-set of samples will undergo the following test work:

- Shake Flask Extraction (SFE) or Meteoric water mobility procedure (MWMP – E2242-13) testing
- Mineralogical analysis
- Humidity Cell Testing

Overall, the sample numbers for the additional testing are subject to change as the geochemical characterization is still in its preliminary stages. A similar approach will be applied to tailings or to heap leach residues, dependent on the process route adopted. The samples for this would be taken from metallurgical programs although it may be appropriate to use process feed samples as analogues for process tails. This approach will be finalized during the next stage of planning for the geochemical testing.

Numerical predictions

Numerical predictions of contact water quality will be carried out for the various mine waste facilities, combining mine site development and water balance information as well as any additional geochemistry during the FS.

The predictions will allow assessment of the potential impact of the mine facilities on surface water and groundwater receptors. The resulting model outputs will be compared to environmental water quality criteria to determine if a potential impact will result from the mining activities and proposed closure scenario.

The geochemical modelling process will involve a series of solution mixing, chemical reactions, and mineral surface adsorption steps to predict the surface water and groundwater composition at selected locations. The United States Geological Survey (USGS) geochemical code PHREEQC (Parkhurst and Appelo, 2010) will be used for all geochemical speciation, mixing, and reaction calculations conducted for this assessment.

The results of the impact assessment will be used to identify management methods to mitigate against potential negative impacts that have been identified for the Los Azules project and particularly interaction with district wide water resources.

20.2.5 Summary of Geoenvironmental Risk

The main geochemical risk associated with the Los Azules Project is the potential generation of acidic, metal- and sulfate-rich waters from surficial mine rock storage facilities and subsequent migration to surface water and/or groundwater environments. Specific to each mine facility, the main risks are:

- Mine Rock Storage Facilities – potential for ARD and ML as seepage to ground and surface waters during operations and post-closure.
- Open pit – potential for ARD and ML to impact the quality of water pumped from the open pit during operations. Post-closure, there is the potential for a pit lake to be impacted by ARD and ML.
- Low grade process stockpile - potential for ARD and ML as seepage to ground and surface waters during operations. Unlikely to be an issue post-closure on the assumption that a low-grade material stockpile will have been processed.
- Heap leach pad – Pregnant Leach Solution during operations will be captured for processing. Post-closure the solute inventory will need to be reduced and measures put in place to prevent any discharge of acidic and/or high solute load solutions from discharging to the environment until the acidity and solute loads reduce to appropriate levels.

Initial indications for process feed and mine rock samples are:

- Process feed material – limited acid generation from oxidized and enriched process materials but the primary process materials have a high potential for acid generation. The extension of this is that with little buffering in the process feed material, the heap may have some potential for acid generation, but it is likely higher in the tailing's material.
- Mine rock material - although some mined rock lithologies have potential for acid generation, reactivity is low and so any acid generation is likely to occur after some time, potentially decades (based on experience elsewhere) into the future, such that acid generation and potential metal leaching are more likely issues for closure rather than operations for the pit and waste rock. Much of the mined rock can be classed as non-acid generating in potential but have limited buffering capacity so perhaps best termed inert. These materials could potentially be used in construction on site.

The potential risks identified will be assessed through ongoing geochemical characterization test work and numerical predictive geochemical modelling. Risks that are identified will then be used to inform the design process and mine planning to include appropriate mitigation measures.

20.3 ENVIRONMENTAL MANAGEMENT AND MONITORING PLANS

The environmental management and monitoring plans required to protect the biophysical and social environments are identified in the Fifth Environmental Impact Assessment Update (EIA) prepared for the project by Ausenco and approved by Resolutions No. 317-MM-2021 and No. 352-MM-2021 issued by the provincial mining authority. It is anticipated that detailed environmental management plans will be required for future Project planning and development. Protection measures are identified in the EIA for the following activities or facility/equipment operation:

- Development and operation of access roads, tracking and drill rigs.
- Development and operation of camp facilities.
- Vegetation and wildlife.
- Water quality and Use.
- Protection of Sites/Areas of Cultural and Natural Heritage.
- Operation of Machinery and Equipment.
- Disturbance of Soil.

20.4 PROJECT PERMITTING

Project permitting is addressed in Section 4.

20.5 SOCIAL/COMMUNITY

The Project is in the Province of San Juan and the Department (municipality) of Calingasta. The Department of Calingasta consists of three principal communities: Barreal, Villa Calingasta and Tamberías. Their combined population was estimated at 9,641 in 2022. The population of the capital, San Juan, was estimated at 526,000 (INDEC, 2015) based on the 2010 census and projected population growth.

The local economies of Barreal, Calingasta and Tamberías are based on tourism, mining and agricultural respectively, although tourism is quickly increasing its contribution to the local economies.

The Province of San Juan and the Department of Calingasta have benefited substantially from the exploration and initial development of various large mining projects and thus remains strongly pro-mining as mine development and exploration continues in the area. No organized anti-mining or anti- development groups or organizations were identified by the Los Azules team in the Department of Calingasta.

The 2011 study conducted by Asesoría Ambiental found that there was a broad appreciation for mining as the principal economic activity of the region at the time and the main driver of economic growth. Most people interviewed during this study were generally supportive of mining development with favorable expectations for the future economic development of the region. There has not been any additional social or community work completed since the 2011 study for the Project (pers comm. McEwen Mining).

20.6 CLOSURE PLANNING

20.6.1 Introduction

The general guidelines for the closure of the Los Azules project are presented below. The Closure Plan (CP) has been developed in accordance with the current level of development and knowledge of the Project. It discusses activities to achieve physical and chemical stability and considers the future use of the land after completion of mining and processing activities. The philosophy of the closure plan considers permanent McEwen involvement during the development of the closure and post-closure of the project.

Early closure design and planning is considered of paramount importance to ensure that the environmental and social objectives defined for closure and post-closure are met. Early development of rehabilitation and closure strategies at the conceptual level is a stage of overall Project planning and provides the basis for increased development of rehabilitation and closure strategies as the Project progresses and the end of operation approaches.

The CP will be updated with each subsequent phase of the project to validate the assumptions on which this CP is based, and to incorporate further project details and/or modifications as they become available, considering the increased knowledge of environmental effects because of project operation and scientific and technological advances associated with mine closure.

The main components of the Los Azules project are as follows:

Table 20.11: Project Facilities	
Type of facility	**Facility / components**
Mine facilities	Pit Mine Rock Storage Facilities (MRSF) Stockpile
Processing facilities	Heap Leach Facility Crusher system Process plant
Ancillary and support facilities	Camp Site roads and Airstrip Power line and substations Maintenance Workshop and Warehouse Fuel Tanks and Fuel Loading Facilities Services (Water, Sewage effluents)

20.6.2 Objectives

The overall objective of closure is to ensure long-term physical and chemical stability, establishing a safe, stable, and predictable condition, capable of mitigating the changes and impacts generated by the operation of the project, with low or no maintenance required after post-closure. In this sense, the Closure Plan is intended to improve, in its post-closure stage, the environmental conditions generated during the operation of the Mine, with a view to achieving an environmental condition compatible with the surroundings, in a manner consistent with traditional land uses.

The closure plan considers a "closure" stage, which is related to the execution of the closure measures and actions, and a post-closure stage, which contemplates a period of active monitoring and maintenance of the closure works and environmental conditions, with an assessment of the environmental performance and the effectiveness of the closure measures implemented, and if necessary, the establishment of corrective actions.

For this purpose, the following closing and post-closing objectives have been defined:

- Comply with environmental obligations, international standards, and applicable legislation.
- Protect human health and the environment.
- Recondition the mine site considering the traditional use of the land.
- Develop, during the operation of the project, the progressive rehabilitation of the facilities that have reached the end of their useful life.
- Rehabilitate disturbed areas to achieve long-term physical and chemical stability, including revegetation with native species (wherever possible).
- Restore natural surface water drainage in disturbed sectors.
- Eliminate or minimize requirements for active site care and maintenance during the post-closure period (e.g., water treatment).
- Develop closure plans that include information obtained from public consultations with local communities and regulatory authorities.
- Minimize local and regional socio-economic impacts.
- Optimize closure and post-closure costs.

20.6.3 Standards and Regulations

In the province of San Juan, there is no mine closure legislation, and the province does not adhere to the National regulation - Res. 161/2021- General Guidelines for Mine Closure with Financial Guarantees in the Argentine Republic.

In this sense, the applicable legal framework is mainly provided by Law No. 24,585 on Environmental Protection for Mining Activities, incorporated to the National Mining Code under Title XIII, Section Two, which establishes in its Article 4 that the activities included in this Title correspond to: a) prospecting, exploration, exploitation, development, preparation, extraction, and storage of mineral substances covered by the Mining Code, including all activities aimed at mine closure. Annex III establishes that the Environmental Management Plan must include actions related to: "cessation and abandonment of exploitation and post-closure monitoring of operations." Provincial Law No. 6,571, on Environmental Impact Assessment, as amended by Provincial Law No. 6,800, is also applicable.

The Los Azules Project Conceptual Closure Plan, and its future updates, will be based on provincial and national legislation, international standards, and guidelines (ICMM - IFC, among others), McEwen policies and standards, and industry best practices.

20.6.4 Geo-Environmental Risk

The development of this section is based on the information prepared by SRK Consulting (UK) Limited - Geochemical Environmental Evaluation of the Los Azules Copper Project– External Memorandum, November 2022.

In general terms, there is the potential for acid rock drainage (ARD) generation. Reactivity is expected to be slow, with a potential for long-term acid generation, indicating that ADR is more likely to have an effect during closure and post-closure rather than during project operation.

The main geochemical risk associated with the Los Azules Project is the potential generation of acidic, metal- and sulfate-rich water from surface waste rock storage facilities and its subsequent migration to surface and/or groundwater environments. Specifically, for each mining facility, the main risks are:

- Mine Rock Storage Facilities (MRSF): potential for Acid Rock Drainage (ARD) and Metal Leaching (ML) in the form of seepage to groundwater and surface water during operations and after closure.
- Open Pit: The potential for ARD and ML to affect the quality of water pumped from the open pit during operations. After closure, there is the potential for the pit lake to be affected by ARD and ML.
- Low-grade process stockpile: potential for ARD and ML in the form of seepage to ground and surface water during operations. This is unlikely to be a problem after closure assuming that the low-grade material stored in the stockpile will have been processed.
- Heap Leach Facility: The pregnant leach solution during operations will be captured for processing. After closure, it will be necessary to reduce the solute inventory and take measures to prevent any discharge of acidic and/or high solute laden solutions to the environment until the acidity and solute load levels are reduced to adequate levels.
- Initial observations of the process feed and mine rock samples are:
    - Process feed materials: limited acid generation from oxidized and enriched ores, but primary mineralization material has high acid generation potential.

- Mine rock: although some mine rock lithologies have acid generation potential, reactivity is low and therefore any acid generation is likely to occur after some time, potentially decades into the future (based on experience elsewhere). Therefore, acid generation and potential metal leaching are more likely issues for closure than for pit and mine rock operations.

Future studies

Potential risks identified will be evaluated by means of geochemical characterization tests and predictive geochemical numerical models.

20.6.5 Closure Considerations / Criteria

Physical Stability

Post-closure, the remaining facilities (pit, mine rock storage facilities, heap leaching facility) will be physically stable in the long term, meeting international standards.

Chemical Stability

Air, surface water and groundwater quality after post-closure will not be affected and will reach a condition like that described during the environmental baseline, prior to the operation of the Project.

The contact of natural runoff water with any remaining facilities will be avoided by means of a differential management system for contact and non-contact water. In the event of contact water drainage, the water will be captured and stored in evaporation pools, with no discharge to the environment.

The heap leach facility will be detoxified to a concentration of 0.5 mg/l of WAD cyanide in the drainage effluent, in line with international standards (International Finance Corporation, International Cyanide Management Institute).

Hydrological Stability

A differential water management system will be implemented for contact and non-contact water. Surface water flows coming from upstream the facilities will be diverted through open channels to prevent them from entering any remaining project facilities (pit, mine rock storage facilities and heap leach facility), without altering the quantity of drainage water coming from upstream, or the natural hydrochemical quality of the water.

Contamination Control

Measures will be taken to ensure that the rehabilitation works of the disturbed areas will be based on technically effective and proven engineering practices, efficient methods, and ecologically appropriate practices. Soil affected by contaminants (chemicals and hydrocarbons) will be disposed of as hazardous waste by an authorized operator/processor, in accordance with the Mine's waste management plan.

Revegetation

Techniques will be implemented to facilitate natural revegetation using native species in the affected areas. The methodology to be used will be analyzed and defined in subsequent studies.

Landscape

Efforts will be made to ensure that the final landscape and ecosystem conditions are like those of the surrounding natural landscape. Disturbed areas will be re-profiled and/or graded to restore the natural conditions of the site.

Waste Management

Urban solid waste will be disposed of off-site at a facility designated for this purpose, with no waste remaining in the mine area after closure.

Non-hazardous waste will be disposed of in accordance with the mine's waste management plan, with no waste to be left in the area after closure.

Hazardous wastes and chemical agents and reagents from the operation will be disposed of off-site by an authorized operator.

Dismantling and Demolition of Infrastructure and Equipment

Machinery and equipment, including mobile equipment, conveyor belts, pumps, processing equipment and other equipment, will be decontaminated and washed, and subsequently dismantled. In the case of equipment and machinery with future usability, McEwen might choose to sell and/or use them for other Projects, or else transfer them to other interested parties. Equipment with no future usability could be sold as scrap or disposed of as waste in accordance with the mine's waste management plan.

All the project's surface facilities will be dismantled/demolished, and surfaces graded. The dismantling and demolition of the structures will be carried out in accordance with the environmental, health and safety measures in force, seeking the commercialization (reuse) of those elements that do not represent a risk to human health or the environment, if feasible.

Monitoring and Maintenance

For the post-closure period, compliance with the site's maintenance and environmental monitoring plan will be ensured, considering a term of at least 10 years, or until closure objectives and physical and chemical stability are achieved.

The maintenance plan will be geared towards verifying the correct functioning of the works and closure measures.

20.6.6 Closure Strategy

The following closure and rehabilitation strategy has been developed at a conceptual level. Final closure activities will commence upon completion of processing for economic purposes. The closure strategy will be carried out considering the following stages:

- Progressive Rehabilitation Stage: This stage will be developed during the operation stage of the project and includes the closure of the facilities that have reached the end of their useful life, or of the disturbed areas in disuse.
- Final Closure Stage: This stage is developed at the end of the operation stage when economic processing is completed. It comprises the closure of all the Project facilities not included in the progressive closure.
- Post-closure stage: This stage involves monitoring and checking compliance with closure objectives and the effectiveness of closure measures, identifying deviations from the objectives to adjust closure measures as required.

Details of the progressive closure plan, and the closure and post-closure monitoring and maintenance plan will be covered in the CP updates.

20.6.6.1 Assumptions

The development of the Conceptual Closure and Rehabilitation Plan is based on the following assumptions:

- The timeframe for closure and post-closure is 3 and 10 years respectively.
- Final closure activities will commence upon completion of processing.
- Progressive closure measures will require detailed engineering work and complementary studies, compatible with final closure measures. Planning for progressive closure activities will be addressed and developed in future CP updates.
- The pit will be flooded to the level of the equilibrium water table. The pit would function as a sump (evaporation exceeds water inflows), with no groundwater input from the pit to the surrounding environment.
- Low-grade process stockpile: this material is assumed to be processed in its entirety during the operation.
- There will be no water input from the heap leach facility to the environment after drying of this facility. If there is any, it will be retained in a storage and evaporation pool.
- A priori, the placement of a cover on the heap leach facility is being considered to facilitate revegetation.
- The construction of the mine rock storage facilities is assumed to include the encapsulation of mine rock having ARD potential with non-ARD generating mine rock. This will be achieved by segregation of the mine rock during pit mining.
- It is assumed that there will be stored topsoil from the stripping stage of project construction, which will be used for revegetation purposes in the disturbed sites and in the remaining facilities (pit, mine rock storage facilities, heap leach facility).
- All infrastructure will be dismantled during closure activities.
- All hazardous waste, including contaminated soil, will be removed from the site by a licensed carrier and disposed of in a duly licensed facility.
- Demolition debris, once decontaminated, will be disposed of in the mine rock storage facilities.
- It is not anticipated that any of the building structures or infrastructure would be handed over to the local communities for final use at the end of the life of the Project. Consultation with relevant government authorities and public stakeholders regarding end-use and/or closure plans will be conducted during later stages of the Project to confirm this assumption.

20.6.7 Temporary Closure

Temporary closure of the project could occur because of various factors, such as: metal prices, policy changes, judicial or administrative actions, or other unforeseen events. This will be a period whose duration is variable but not indefinite, because when conditions are favorable, project operation will be reactivated. Otherwise, measures will be taken to move forward with final closure.

The objectives of the temporary closure should be to minimize environmental and social impacts, maintain the project's infrastructure in good condition, comply with environmental and social commitments and obligations, maintain environmental permits in force and comply with applicable regulations.

The following are the general guidelines that should be considered for a temporary closure, but they should be addressed in detail at the time of developing the temporary closure plan. Activities or actions during temporary closure should consider, but not be limited to, the following aspects:

- Human Resources (direct employees and contractors): Actions in relation to own and contractors' employees. Employee containment plan. Relationship with labor unions and communication plan for employees.
- Operational: Dismantling of infrastructure that has reached the end of its useful life, and closure of unused tracks and roads, if applicable. Maintenance of structures and facilities that will later be reused when the project is reactivated.
- Environmental: Remediation / rehabilitation of impacted areas that will not be reused when the project is reactivated (associated with closure of facilities in disuse). Environmental and engineering monitoring plan aimed at guaranteeing physical and chemical stability during the time of temporary closure, ensuring maintenance of environmental permits. Study frequencies and components that have already been defined for the operation of the project are maintained, including property security controls and engineering inspection and maintenance activities.
- Community Affairs: Actions to mitigate social impacts derived from the closing of social agreements or commitments; management and mitigation of social impacts derived from employee exit and termination/suspension of contractors.
- Institutional Affairs: institutional management plan to prevent deterioration of relations with local institutions, such as the Church, Neighborhood Associations, Parish Councils, etc. Communication plan for Temporary Closure Plan; preparation of press releases to avoid reputational impacts; plan for meetings with government authorities, as appropriate.
- Contract management: Closure or suspension of contracts not required for the temporary closure stage, management of contracts for project monitoring and maintenance activities during temporary closure. Contractor oversight.
- Health and Safety: Actions aimed at preventing safety incidents or work accidents at the project site involving employees, contractors or affecting the assets or property of the Mining Project.
- Legal/Regulatory: monitoring of responsibilities derived from the closure process; monitoring of legal and regulatory compliance in the implementation of the early or temporary closure plan.

20.6.8 Final Closure – Closure Measures

Closure measures are intended to achieve long-term physical and chemical stability. The designs of the remaining facilities (pit, mine rock storage facilities, heap leach facility) are deemed to be designed to be physically stable over the long term; however, a stability verification process will be carried out considering the final topography of the construction of these facilities, considering the closure criteria defined based on international standards.

An ongoing maintenance plan will be required to ensure that all enclosed facilities and closure works perform to their design and specifications.

Removal of infrastructure and rehabilitation of these areas will occur once economic processing activities have been completed. CP updates will provide revegetation studies with native species and methodological alternatives to be used in the rehabilitation of these enclosed facilities /areas.

Whenever possible, priority will be given to the sale of equipment and components with resale value; otherwise, they will be sold as scrap or disposed of off-site at an authorized facility.

The following are the closure measures for the project's major components:

Open Pit

Once the pit exploitation stage is completed, a pit stability analysis will be carried out considering the actual construction topography. The purpose of this analysis is to verify that the design parameters adopted are consistent with those established in the original design and in the closure criteria to guarantee physical stability over the long term.

Development of the pit lake may take several years and therefore the potential exists for ARD generation. Methods to address any potential ARD generation during the refilling period (such as lime treatment) should be investigated early in the Project planning process.

A hydrogeological and hydrogeochemical model of the pit lake operation during this stage will also be developed to validate the assumptions and hypotheses related to the operation of the pit as a sump and the non-impact on groundwater in the surrounding area.

Closure measures include:

- Removal of auxiliary equipment and facilities.
- Natural flooding of the pit until the equilibrium water table is reached.
- Construction of perimeter channels to prevent entry of non-contact water from natural runoff, with discharge downstream the pit.
- Pit access closure: 2m high rock berms will be set up to prevent human and animal access to the pit.
- Scarification of access roads to facilitate revegetation conditions, except for roads necessary to access monitoring and control points.

Mineral Stockpile

The low-grade material stockpiles have similar characteristics to the MRSFs, although they may contain, in some cases, a higher sulfide content.

The closure plan considers that these stockpiles will be fully reclaimed by the process plant, leaving the site free of any remaining material. Therefore, the rehabilitation of these sites contemplates the extraction of a soil layer, whose thickness will depend on the results of the soil characterization analysis. The material extracted from the site will be disposed of in nearby MRSF. Then, the disturbed areas will be graded and scarified, ensuring positive drainage, seeking integration with the topography of the site, and providing adequate conditions for natural revegetation.

Process Plant and Crushing System

Closure measures include:

- Reduction of stock of products and chemical agents. Reagents and supplies will be removed and returned to suppliers, and/or sold to other operations, and/or disposed of off-site through specially authorized operators and companies.
- Decontamination of process equipment in accordance with procedures provided by process plant constructor.

- Dismantling of facilities, demolition of structures and platforms by a specialized company.
- Disassembly and dismantling of crushing, grinding and processing equipment. The activities will be carried out by specialized personnel.
- Scarification, grading and rehabilitation with native species.

Auxiliary facilities (camp, maintenance workshop and warehouse, administration, substations)

Closure measures are as follows:

- Reduction and final disposal of solution inventories (reagents, acids, and other chemicals) to reduce the amounts existing at time of closure. Substances will be removed from the site for final disposal by authorized operators. If they may be returned to the supplier, this option will be prioritized. These substances include lubricants, fuels and other chemical compounds used in these facilities.
- Hazardous and non-hazardous waste management: All waste will be managed in accordance with the mine's waste management guidelines, with no waste left on site. Hazardous waste will be managed by a licensed operator and disposed of at an authorized disposal facility.
- Decontamination of facilities, platforms, equipment, and conduction systems prior to dismantling and/or demolition.
- Dismantling of facilities and removal of modular structures and perimeter fences.
- Demolition of concrete structures and disposal in MRSFs.
- A contaminant characterization plan is envisaged to identify environmental liabilities and their subsequent remediation.

Machinery and Equipment

All fixed and mobile equipment with commercial value will be removed and sold. Non-marketable equipment will be removed from site.

Mine Rock Storage Facilities (MRSF)

Closure of MRSFs is intended to ensure long-term physicochemical stability, public safety and the protection of surface and groundwater resources.

Once the MRSFs exploitation stage has been completed, a stability analysis must be carried out for each one of them, considering the actual construction topography. This analysis is intended to verify that the design parameters adopted are consistent with those established in the original design and the closure criteria, to guarantee physical stability over the long term.

During construction of the MRSFs, encapsulation will be carried out to prevent ADR generation. At the same time, verification of the design parameters will be carried out to ensure that they are compatible with closure stability criteria. The generation of contact water is not anticipated. This assumption will have to be validated in subsequent studies based on the actual encapsulation design. Depending on the results, a system for contact water collection, storage and evaporation in pools will be designed, if necessary.

Closure measures include:

- Removal of ancillary equipment and facilities.
- Non-contact water ('NCW') management system to capture natural runoff water flows and prevent them from entering the MRSF, with discharge downstream the facility into the natural drainage

network.

- Scarification of access roads, except for those necessary to access monitoring or control points.
- If feasible, rehabilitation actions will be implemented by applying topsoil and revegetation with native species.

Fuel Storage and Supply Facilities

These facilities will receive special treatment at the time of removal, including measures to prevent spills and environmental contamination, as well as observance of specific safety standards for fuel storage tank dismantling. The facilities will be removed by a specialized company, with no components remaining on site.

A contaminant characterization plan is envisaged to identify environmental liabilities and their subsequent remediation.

Utilities and Communications Infrastructure

- All above ground utilities and communications infrastructure will be removed.
- Buried pipelines used for water supply and treatment infrastructure will be drained and removed.
- Support structures (poles) and communications cables, if present, will be removed and disposed of off-site at a licensed facility.
- Power lines will be removed.

Site Roads and Airstrip

- Service and access roads to the site will remain in place for closure activities and selected roads will be maintained to allow access during post-closure monitoring. The remaining roads and unused sites will be scarified and rehabilitated with native species.
- The airstrip will be maintained during closure to allow access to the site and to perform monitoring tasks during post-closure.
- Ditches and drainage culverts will be removed, and the roadbed will be opened or cut at watercourse crossings to restore natural drainage systems in the area.
- Upon completion of the post-closure period, airstrip remediation will be carried out. The landing surface will be removed and scarified, or re-graded to harmonize with the adjacent topography. Ditches and drainage culverts will be removed to restore the natural drainage systems in the area. The site will be rehabilitated by applying topsoil and revegetation with native species.

Site Water Management System

Surface water management systems of the operation that are not relevant for project closure will be removed, with drainage of the non-contact water redirected to natural drainage systems/watercourses.

20.6.9 Post Closure Monitoring and Maintenance

The purpose of the monitoring and maintenance plan is to verify the correct operation of the closure measures and compliance with the closure and post-closure objectives, and to identify potential deviations to make early adjustments to the closure measures and works. A period of 3 years has been considered for closure monitoring and maintenance and 10 years for the post-closure stage or until physical and chemical stability is demonstrated for the long term, in accordance with established closure objectives.

Progress reports on the closure and rehabilitation program, including the results of environmental and engineering monitoring, will be submitted to the Enforcement Authority in accordance with the requirements of the Resolutions or conditions included in the Environmental Impact Statement ("DIA") approving the project and its subsequent updates.

21.0 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATION

This section describes the basis of estimate preparation for McEwen Copper's Los Azules Project in the San Juan Province of Argentina. The Project includes the development of an open pit mine with muti-stage crushing and screening, a heap leach pad, and a copper solvent extraction-electrowinning facility capable of processing a Base Case 175,000 tpa or an Alternate Case 125,000 tpa. There is also a sulfuric acid plant and other associated infrastructure to support the operations, in general it includes the following facilities:

- Mine development and associated infrastructure
- Coarse Process Storage and Handling (Crushing, Conveying, Agglomeration)
- Heap Leach Pads and Conveyor Stacking Systems
- Solvent Extraction-Electrowinning (SX/EW) Facility
- Sulfuric Acid Plant
- On-Site Utilities and Ancillary Facilities including a Construction Camp
- Off-Site Infrastructure: Power Transmission Line, Access Roads, and Permanent Camp

The project initial capital costs are based on budgetary cost quotations estimates for major equipment, recent in-house cost information and installation factors, and regional contractor and facilities obtained between Q4 2022 and Q1 2023. The capital costs for the project are summarized below and should be viewed with an expected level of accuracy for a preliminary analysis at +40%/-20% consistent with AACE International Recommended Practice No. 47R-11 Estimate Class 5. The initial project development capital costs for the Base and Alternative case options are summarized in Table 21.1.

Table 21.1: Initial Capital Costs by Case		
Capital Cost Level 1 Summary	**Base Case 175 ktpa**	**Alt. Case 125 ktpa**
WBS Area	**Total (USD)**	**Total (USD)**
100 - Mining	*$65,600,000*	*$65,600,000*
200 - Ore Storage & Handling	*$234,500,000*	*$192,500,000*
400 - Heap Leaching	*$158,500,000*	*$142,100,000*
500 – SX/EW Facilities	*$250,400,000*	*$167,700,000*
600 - Acid Plant	*$94,900,000*	*$79,900,000*
800 - Ancillary Facilities	*$23,300,000*	*$23,300,000*
900 - Site Development & Yard Utilities	*$126,300,000*	*$112,200,000*
2000 - OffSites	*$167,400,000*	*$167,400,000*
Direct Costs	**$1,120,900,000**	**$950,700,000**
Field Distributable & Services	*$17,900,000*	*$17,000,000*
Construction Services	*$67,300,000*	*$57,000,000*
Construction Camp & Services	*$78,100,000*	*$69,900,000*
Freight	*$45,800,000*	*$39,000,000*
EPCM Services	*$170,100,000*	*$140,900,000*
Common Indirects	**$379,200,000**	**$323,800,000**
Mine Equipment	*$197,000,000*	*$202,400,000*

Table 21.1: Initial Capital Costs by Case		
Capital Cost Level 1 Summary	**Base Case 175 ktpa**	**Alt. Case 125 ktpa**
Mine Pre-Stripping	*$163,800,000*	*$171,300,000*
Owners Other Costs	*$105,900,000*	*$82,200,000*
Owners Cost	**$466,700,000**	**$455,900,000**
Subtotal	**$1,966,800,000**	**$1,730,400,000**
Contingency	$495,000,000	$423,100,000
Total Capital Cost	**$2,461,800,000**	**$2,153,500,000**

As a recommended practice of The Association for the Advancement of Cost Engineering (AACE) International, the Cost Estimate Classification System provides guidelines for applying the general principles of estimate classification to project cost estimates. The capital cost is a Class 5 Order of Magnitude estimate as defined by the AACE guidelines. Typical accuracy ranges for AACE International Class 5 estimates are -20% to -50% on the low side, and +30% to +100% on the high side, depending on the complexity of the project.

Major equipment preliminary budgetary pricing sourced for this PEA estimate is described in Table 21.2 below.

Table 21.2: Major Equipment Budget Cost Sources	
Mining Equipment – Rope Shovels/Loaders	Komatsu
Mining Equipment – Haul Trucks/Hydraulic Excavators	Liebherr
Trolley Assist System	Liebherr
Primary Crusher (Sizer) Station	MMD Mining Machinery, Metso-Outotec
Secondary Crusher (Sizer) Station	MMD Mining Machinery, Metso-Outotec
Tertiary Crusher & Screens	Metso-Outotec
Overland Conveyors Package	Terra Nova Technologies
Mobile Conveyor Stacking System	Terra Nova Technologies
Agglomeration Drums	Terra Nova Technologies
Solvent Extraction Plant Equipment Package	Metso-Outotec, Metalex
Electrowinning Plant Equipment Package	Metso-Outotec, Metalex
Sulfuric Acid Plant Package	Metso-Outotec

SX/EW installation costs and minor equipment, tanks and pumps were sourced from the SE database from recent projects completed or estimated in detail within the past year.

Knight-Piesold provided preliminary quantities and costs for the heap leach pad geo-technical, lining solutions recovery and water management systems in the heap leaching area. Costs are based on recent Argentinian projects completed in the region.

Temporary operations modular camp costs were sourced from ATCO Sabinco (Chile). McLennan Design provided preliminary estimates for the permanent camp facilities.

The incoming 220 kV powerline and substation upgrades at the existing Calingasta and Rodeo locations are provided by YPF Luz and consider the main site substation as part of their provision of energy to the project. No capital is included in this estimate and the investment costs incurred by YPF are recovered as operating cost. Preliminary capital costs for these facilities were estimated by YPF Luz to be approximately USD $155 million.

The estimate is built up by cost centers as defined by the project's WBS for Area designations, and in the case of the process facilities, by prime commodity accounts, which include earthwork, concrete, structural steel, buildings, mechanical equipment (including platework), piping, electrical and instrumentation. Owner's Costs include the initial mine fleet, preproduction stripping costs and preoperational costs for early crushing and material placement on the leach pad.

Regional contractor rates and unit costs we sourced from contractors familiar with the region and Argentinian based costs. These were applied in some instances where preliminary quantities have been estimated (e.g., civil works, leach pad).

The costs assume an EPCM type contract in which equipment and materials will be purchased on a competitive basis, and installation contracts will be awarded in defined packages on a time and materials, unit price or lump sum basis. It is also assumed that the EPCM contractor will purchase the major equipment on behalf of the Owner with no additional mark-up.

The estimate is expressed in first quarter 2023 United States dollars and all references herein are in USD. Due to the extended nature of the study budgetary pricing ranged from third quarter 2022 to second quarter 2023. No provision has been included to offset future escalation.

Most costs and equipment estimates were provided on a USD basis. Where source information was provided in other currencies, these amounts have been converted at the following rates:

- 1 USD = 0.97 Euros (EUR)
- 1 USD = 1.33 Canadian Dollar (CAD)
- 1 USD = 0.81 British Pounds (GBP)

21.1.1 Exclusions

Items not included in the capital estimates are as follows:

- YPF Luz 220 kV power supply to site (powerline, related substations)
- Sunk costs (costs prior to start of detailed design)
- Exploration cost
- Permitting cost prior to detail design (costs for EIA/IIA submittal are sunk)
- Land acquisition
- License and royalty fees for any technology or equipment considered
- Allowance for special incentives (schedule, safety, etc.)
- Interest and project financing costs
- Salvage cost credits for equipment or materials

21.1.2 Sustaining Costs

Sustaining capital is the periodic addition of capital that is required for equipment purchases or construction of additional facilities required to maintain operations (outside of the normal day-to-day operations and maintenance costs).

These capital costs which will be incurred during years when the plant is operational are not included in the initial capital cost estimate but are included with the economic model in the years that the costs are anticipated to occur for the purpose of calculating the overall economic benefits of the project.

The sustaining capital plan for the Base Case and Alternative Case projects is presented below in Table 21.3

Table 21.3: Base & Alternative Case Sustaining Capital Plans

Description	175 ktpa HG w/ YPF HV Financing	125 ktpa HG w/ YPF HV Financing
Mining Fleet Equipment Capital		
Mine Fleet Augmentation - Year 1	$ 45,908,126	$ 59,431,591
Mine Fleet Augmentation - Year 2	$ 132,672,525	$ 11,615,616
Mine Fleet Augmentation - Year 3	$ 9,558,361	$ 12,653,355
Mine Fleet Augmentation - Year 4	$ 17,781,276	$ 50,430,589
Mine Fleet Augmentation - Year 5	$ 39,836,312	$ 20,006,627
Mine Fleet Augmentation - Year 6	$ 26,391,301	$ 17,527,203
Mine Fleet Augmentation - Year 7	$ 77,923,775	$ 21,003,030
Mine Fleet Augmentation - Year 8	$ 13,715,498	$ 16,430,491
Mine Fleet Augmentation - Year 9	$ 759,798	$ 759,798
Mine Fleet Augmentation - Year 10	$ 29,589,537	$ 19,717,805
Mine Fleet Augmentation - Year 11	$ 52,270,829	$ 44,566,923
Mine Fleet Augmentation - Year 12	$ -	$ 40,646,582
Mine Fleet Augmentation - Year 13	$ 14,675,936	$ 28,970,020
Mine Fleet Augmentation - Year 14	$ 6,065,146	$ 31,531,959
Mine Fleet Augmentation - Year 15	$ 2,284,256	$ 1,539,376
Mine Fleet Augmentation - Year 16	$ 15,446,265	$ 11,855,015
Mine Fleet Augmentation - Year 17	$ 10,271,749	$ 11,016,629
Mine Fleet Augmentation - Year 18	$ 24,397,379	$ 2,668,498
Mine Fleet Augmentation - Year 19	$ 9,785,438	$ 6,194,188
Mine Fleet Augmentation - Year 20	$ 259,675	$ 15,404,278
Mine Fleet Augmentation - Year 21	$ -	$ -
Mine Fleet Augmentation - Year 22	$ 259,675	$ 259,675
Mine Fleet Augmentation - Year 23	$ 643,663	$ 7,010,366
Mine Fleet Augmentation - Year 24	$ -	$ 38,200,219
Mine Fleet Augmentation - Year 25	$ -	$ 25,466,813

Table 21.3: Base & Alternative Case Sustaining Capital Plans		
Description	**175 ktpa HG w/ YPF HV Financing**	**125 ktpa HG w/ YPF HV Financing**
Mine Fleet Augmentation - Year 26	$ -	$ -
Mine Fleet Augmentation - Year 27	$ 406,088	$ 933,725
Mine Fleet Augmentation - Year 28	$ -	$ 436,475
Mine Fleet Augmentation - Year 29	$ -	$ -
Mine Fleet Augmentation - Year 30	$ -	$ -
Mine Fleet Augmentation - Year 31	$ -	$ 408,298
Mine Fleet Augmentation - Year 32	$ -	$ 408,850
Sustaining Mine Fleet	**$ 530,902,607**	**$ 497,093,991**
Heap Leach Pad		
HLP - Phase 2	Initial Capex	$ 28,603,418
HLP - Phase 3	$ 71,181,787	$ 71,181,787
HLP - Phase 4	$ 82,441,756	$ 82,441,756
HLP - Phase 5	$ 59,096,432	$ 59,096,432
HLP - Phase 6	$ 50,400,574	$ 50,400,574
HLP - Phase 7	$ 41,851,108	$ 41,851,108
HLP - Phase 8	$ 55,071,238	$ 55,071,238
HLP - Phase 9	$ 34,336,575	$ 34,336,575
HLP - Phase 10	$ 14,702,209	$ 14,702,209
HLP - Phase 11	$ 55,071,238	$ 55,071,238
HLP - Phase 12	$ 55,071,238	$ 55,071,238
Subtotal Heap Leach Pads	**$ 519,224,155**	**$ 547,827,573**
Processing Facilities		
Acid Plant Expansion	$ 115,089,383	$ 146,787,819
Crushing Plant Expansion	$ 352,582,434	$ 76,816,709
Stacking System Expansion	$ 80,191,200	$ 80,191,200
Acid Plant Expansion	$ 115,089,383	
Crushing Plant Expansion	$ 76,816,709	$ 352,582,434
Acid Plant Expansion	$ 222,488,729	$ 146,787,819
SX Train	$ 37,218,833	$ 34,103,500
Crushing Plant Expansion		$ 76,816,709
Acid Plant Expansion		$ 199,434,031
Subtotal Process Facility Expansions	**$ 999,476,672**	**$1,113,520,221**
"Regeneration Green Design" Permanent Camp	$ 193,375,000	$ 193,375,000
TOTAL SUSTAINING COSTS	**$ 2,242,978,000**	**$ 2,351,817,000**

21.2 PROJECT DEVELOPMENT EXECUTION PLAN AND SCHEDULE

The Los Azules project execution plan and schedule is based on an Engineering, Procurement & Construction Management (EPCM) execution approach allowing for multiple specialty and local contractors to be considered. Argentina has construction companies that have constructed significant industrial facilities and heap leach pads and that are familiar with the project location and environment.

The initial project development for either case is expected to take approximately 33 months to mechanically complete from notice to proceed and point of project financing. Construction development will prioritize the initial leach pad and ponds, crushing and stacking systems to facilitate the placement of leach materials on the pad during pre-striping and prior to starting the rest of the facilities start. Ramp up to full leaching capacity is expected to take six to nine months from plant start-up and placement of mineralized material on the pad with commercial production of copper from the SX/EW plant is expected to be achieved in approximately 12 months from start of leaching. Finalization of the necessary permits to begin work is expected to be completed during the proposed feasibility study timeframe. Early works will commence, once project funding is available, with access road upgrades, site preparation, construction infrastructure and power line development.

Engineering is considered in two phases. An initial phase to confirm long lead time equipment selections and preparation of purchase agreements for immediate release upon Notice to Proceed, preliminary engineering design and development of early works contracting packages prior to project financing and execution decision leading to a formal Notice to Proceed milestone. The second phase includes detail engineering to support construction, selection, and procurement of the balance of the equipment and materials and incorporation of equipment data into the design.

Equipment is expected to be purchased nationally where practical and internationally. Materials will be sourced within Argentina where available and regionally. International transportation, customs and importation logistical considerations will be required. Sources and ports in Chile are expected to be the primary routes with some materials and equipment coming from Argentinian ports from European suppliers. A 90-day allowance for transport and importation has been included in equipment delivery time frames.

The envisioned construction approach will be a prime contractor supplemented by local and specialty contractors. A specific contracting plan is not yet developed. Construction considers development of the necessary temporary infrastructure for the construction activities and the workforce is expected to peak at 2,300 workers. Off-site pre-assembly and fabrication will be used to the extent possible to minimize the on-site staff in Calingasta.

The initial project is expected to take approximately 30-36 months to mechanically complete from the Notice to Proceed and point of project full funding.

Table 21.4 provides the long-lead equipment delivery assumptions considered. As post COVID and other global impacts to supply chain constraints ease, these delivery times are expected to improve to more traditional timeframes.

Table 21.4: Long-lead Equipment Delivery Assumptions

Long Lead Items	Lead Time (months)
Power sub-station transformers	20-24
Sulfuric acid plant design/supply	20-24
SX/EW plant design/supply equipment	16-20

Table 21.4: Long-lead Equipment Delivery Assumptions	
Long Lead Items	Lead Time (months)
Mine shovels	16-18
Stacking system & agglomerating drums	16-18
Mine trucks	12-14
Primary & Secondary Sizing Stations	12-14
Tertiary Crushing plant equipment (tertiary crushers, feeders)	12-14
Overland conveyor	10-12
In-plant conveyors	10-12

A nine-month preliminary engineering and construction period is considered to finalize funding and prepare for long-lead equipment purchase and construction contracts. Early works will commence with access roads, site preparation, construction infrastructure and power line development, much of which is initially off-site and less access and weather dependent.

Site activities consider seasonal challenges and winter conditions starting in May/June of each year with some activities stopping and recommencing in September/October. Detailed planning and winterization will be required to ensure year-round construction activities can take place. Scheduling contingencies have not been considered at this level of study.

Figure 21.1 presents a conceptual Project Execution Schedule based on regional contractor inputs and long-lead equipment and materials delivery assumptions provided by vendors. The schedule assumes that the feasibility study work is completed as described, finalization of the IIA/DIA permitting process and other necessary permits to begin work is completed during the proposed feasibility study and preliminary timeframe and financing is in place to achieve the schedule milestones.

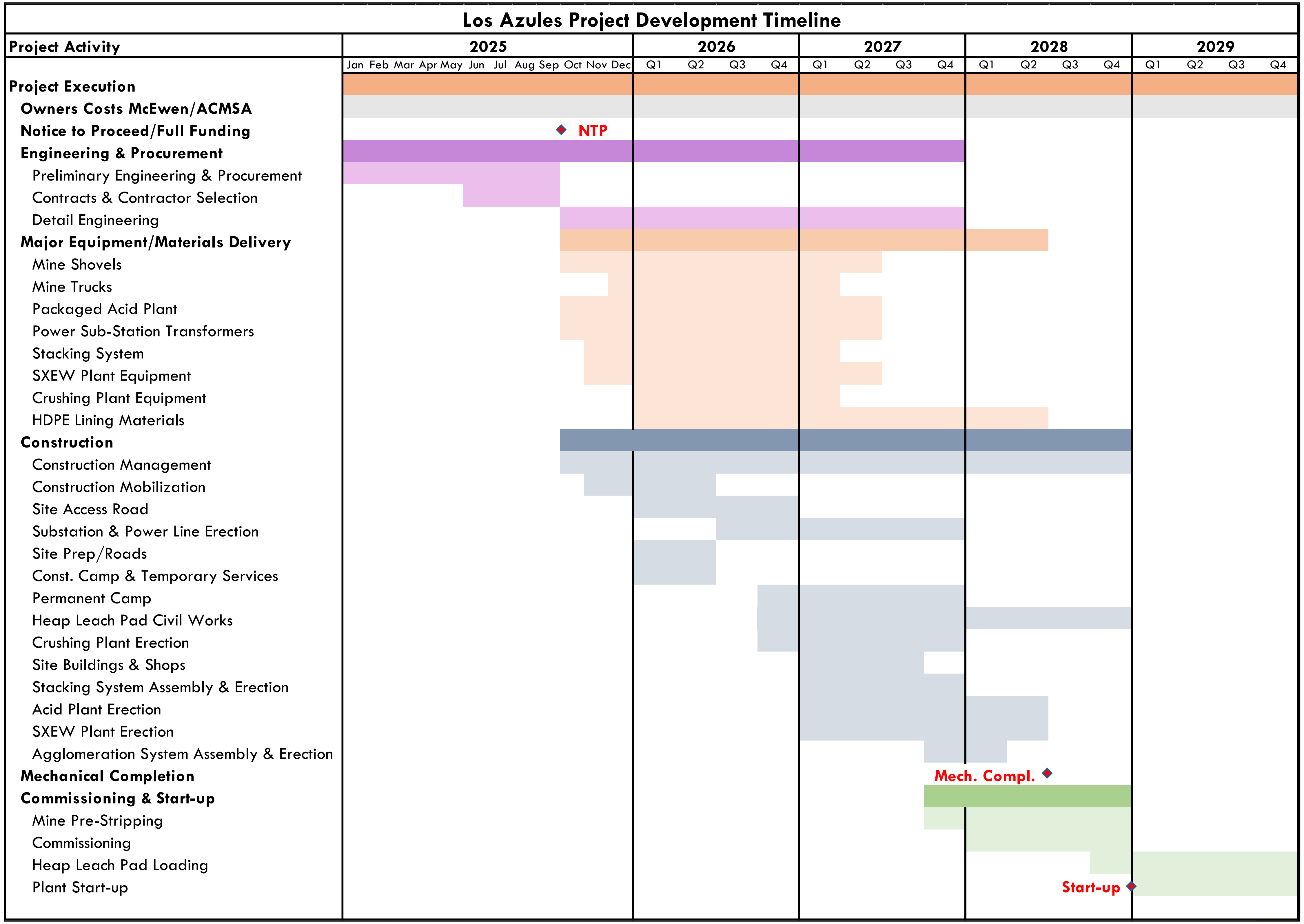


Figure 21.1: Conceptual Project Execution Schedule

The unit processes and equipment considered for the Los Azules operation are well known and highly developed, however as an integrated facility a level of complexity is also understood. Vendors considered for the supply of the major equipment are well known in the industry, regionally and in the specific process plant applications. There are no similar plants in Argentina to draw experienced work forces from, although in neighboring Chile and Peru these types of facilities are common. In terms of a McNulty Curve consideration, it is expected that this facility would fall between a Class 1 or 2 facility for throughput related aspects, with copper production expectations directly tied to the expected leaching performance assumptions.

Ramp up to full leaching tonnage capacity is expected to take six to nine months from plant start-up considering material placement timing. Copper recovery assumptions are discussed in Section 13 of this document and assume that 100% of the soluble copper can be recovered over a three-year timeframe based on conceptual leach cycles and timing. Commercial production is expected to be achieved in approximately 18 months as the leaching process matures.

21.3 OPERATING COST ESTIMATION

The project operating costs are summarized in Table 21.5 for life-of-mine (LOM) values per tonnes of material processed and per pound of copper produced.

Table 21.5: Life of Mine Operating Cost Summary

OPEX SUMMARY	Life of Mine		175ktpa Base Case	125ktpa Alt. Case
Mining OPEX	Per Eq. Lb Cu	$/lb Cu	$0.56	$0.57
	Per tonnes processed	$/t	$4.14	$4.27
Processing OPEX	Per Eq. Lb Cu	$/lb Cu	$0.37	$0.37
	Per tonnes processed	$/t	$2.73	$2.74
SG&A	Per Eq. Lb Cu	$/lb Cu	$0.15	$0.17
	Per tonnes processed	$/t	$0.94	$1.11
TOTAL OPEX (C1 Costs)*	Per Eq. Lb Cu	$/lb Cu	$1.07	$1.11
	Per tonnes processed	$/t	$7.96	$8.27

*Note: Figures may not add up exactly due to rounding

21.3.1 Mining Operating Costs

Operating cost estimates for mine equipment were developed from a combination of data from InfoMine USA, Inc's CostMine mining cost service and Stantec's experience on past projects. Labor and fuel rates were applied separately to build up costs specific for Los Azules.

Mine operating costs were built up using hourly rates for all equipment types, equipment hours, and mined tonne on an annual basis. Costs included fuel, maintenance, wear parts, maintenance labor, and operator labor. Diesel fuel price was USD $1.53/l, power costs of $0.065/kWh was provided by Samuel Engineering based on quotes from YPF, and a fully burdened operator and maintenance personnel cost of USD $50,000 was used to determine a USD cost per operating hour for each piece of equipment.

Haulage profiles were developed for each period and Hexagon's MSHaulage software was used to determine truck cycle times and subsequently the number of trucks required. Loading units were determined based on fleet match and productivity calculations and all other equipment was based on standard factors using the number of operating trucks as a basis.

Using fleet operating costs, productivities, and mine schedule tonnes this resulted in the following weighted average LOM operating costs for the Base Case option:

Table 21.6: Mine Operating Costs 175ktpa Base Case	
Description	**OPEX ($/t)**
Loading	0.13
Hauling	0.81
Drill and Blast	0.36
Dozing	0.11
Grading	0.05
Support Equipment	0.21
Mining G&A	**0.10**
Mine Support Tasks	**0.12**
Mining Total	**1.89**

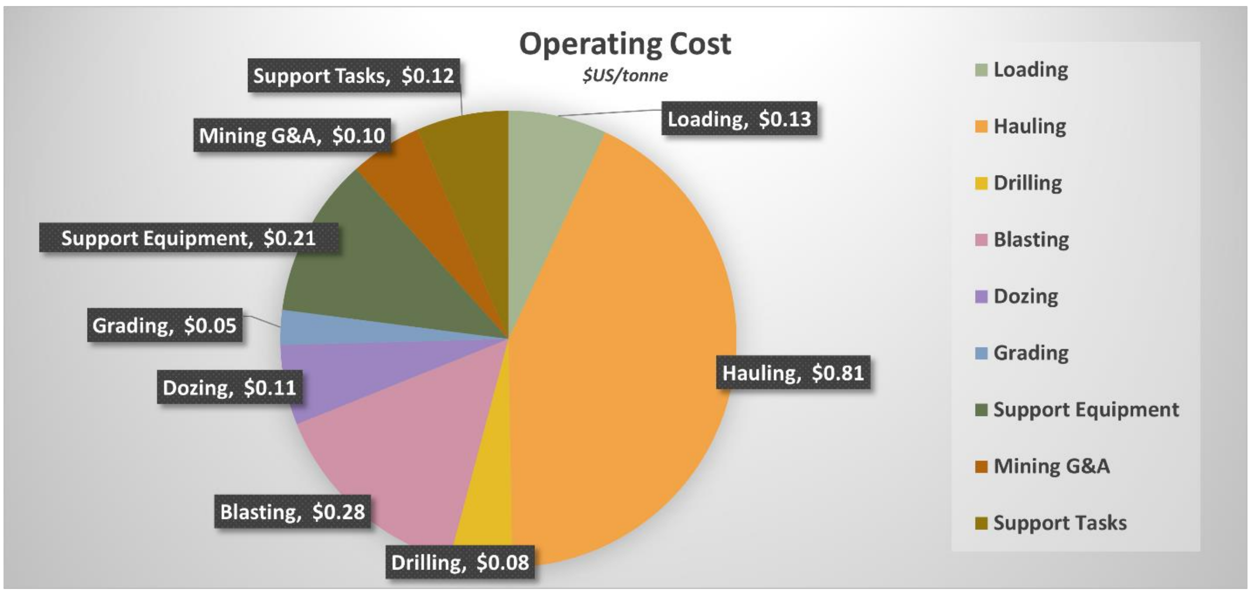


Figure 21.2: Mine Operating Cost Breakdown

For the Alternative Case, using the same methodologies for fleet operating costs, productivities, and mine schedule tonnes this resulted in the following weighted average LOM operating costs.

Table 21.7: Mine Operating Costs 125ktpa Alternative Case	
Description	**OPEX ($/t)**
Loading	0.14
Hauling	0.81
Drill and Blast	0.38
Dozing	0.15

Table 21.7: Mine Operating Costs 125ktpa Alternative Case	
Description	**OPEX ($/t)**
Grading	0.06
Support Equipment	0.27
Mining G&A	0.11
Mine Support Tasks	0.13
Mining Total	**2.05**

Total LOM operating costs ranged from $2.52/t to $1.36/t. Mine Support Tasks include road building, water pumping, snow clearing, access roads, electrical cable moves, and re-drilling.

21.3.2 Processing Operating Costs

Process operating costs (OPEX) were determined from first principals, with the following basis:

- Unit costs for consumables were provided by McEwen Copper or taken from vendor equipment quotations or based on historical data or experience.
- The exempt and non-exempt labor requirements were provided by McEwen Copper.
- Copper Recovery and Acid consumption was provided through the work within SGS-Santiago and detailed in Section 13.3.5.1.
- Reagent and fuel pricing costs were obtained by McEwen Copper.
- The electric power cost is the current rate obtained by McEwen Copper.
- Annual wear part consumption for major equipment were taken from vendor recommendations where provided. Where vendor information was not obtained, a percentage of the equipment purchase price was applied to estimate the parts costs.
- General maintenance supplies were estimated by applying a percentage of the total equipment purchase cost for a given area.
- Sulfur pricing is based on regional pricing at $315/tonne delivered to site (sulfur-price.com Q4 2022 basis).
- Power generated by the acid plant was used to off-set grid power, additional details for power can be found in Section 17.
- Additional details for sulfuric acid consumption can be found in Section 17.

The operating and maintenance unit supplies costs are summarized in Table 21.8.

Table 21.8: Unit Supply Assumptions		
Item	**Units**	**$/unit**
General		
Soluble Copper Recovery	%	100.0%
Residual Copper Recovery	%	15.0%
Crushing Operating Availability	%	70%
SX/EW Operating Availability	%	98%
Operating Days per Year	day/yr	365

Table 21.8: Unit Supply Assumptions		
Item	**Units**	**$/unit**
Shifts per Day	-	3
Hours per Shift	hr	8
Operating Hours per Year, at availability, HLF	hr/yr	6132
Convert Tonnes to Pounds	t/lb	2204.623
Operating Hours per Year, at availability, SX/EW	hr/yr	8585
Power		
Electric Power	$/kWhr	$0.0650
Primary Crusher Power	kW	600
Secondary Screens	kW	110
Secondary Crusher	kW	1866
Tertiary Screens	kW	165
Tertiary Crusher	kW	2799
Conveyors	kW	3000
Agglomeration	kW	250
EW	kWh/t Cu	2700
SX/TF Power/Utilities	kW	5250
Reagents		
Sulfuric Acid - Gross Consumption	kg/t	18
Elemental Sulfur	$/t	$315
Sulfuric Acid Cost – External Supply to Site	$/t	$200
Acid Plant Sulfuric Acid Cost – Site Production	$/t	$164.5
Cobalt ($CoSO_4$-$5H_2O$)	$/lb Cu	0.015
Guar	$/lb Cu	0.015
FC-1100	$/lb Cu	0.015
Diluent	gal/day	0.84
	$/gal	4.45
Extractant	gal/day	0.16
	$/gal	43.78
Sulfuric Acid – SX/EW	t/day	2
Maintenance		
Heap Leach Pad Maintenance Cost	$/t	$0.12
Acid Plant Cost	$/t Elemental Sulfur	$25.00
SX/EW Maintenance Cost	$/lb Cu	$0.010

The process operating costs are summarized in Table 21.9 for life-of-mine (LOM) values on a cost per tonne processed and pound of copper produced basis.

Table 21.9: Life of Mine Operating Cost Summary			
Base Case 175 ktpa Cu	**Cost USD**	**$/tonne***	**$/lb Cu**
Labor	$ 479,800,000	$0.41	$0.06
Reagents	$ 1,582,600,000	$1.36	$0.18
Power	$ 482,800,000	$0.41	$0.06
Maintenance	$ 345,600,000	$0.30	$0.04
Miscellaneous	$ 290,100,000	$0.25	$0.03
Total Processing Costs	**$ 3,180,900,000**	**$2.73**	**$0.37**
Alt Case 125 ktpa Cu	**Cost USD**	**$/tonne**	**$/lb Cu**
Labor	$ 538,700,000	$0.46	$0.06
Reagents	$ 1,572,000,000	$1.35	$0.18
Power	$ 499,500,000	$0.43	$0.06
Maintenance	$ 351,700,000	$0.30	$0.04
Miscellaneous	$ 226,800,000	$0.19	$0.03
Total Processing Costs	**$ 3,188,700,000**	**$2.74**	**$0.37**

*Per tonne processed, excluding capitalized preproduction material placement on leach pad costs

Labor costs are built up from preliminary staffing plans and include crushing, leaching, SX/EW, and acid plant staffing on 2-week rotation/12-hour shifts for operations and process maintenance staff including an allowance for absenteeism. Staffing extends from Plant General Manager level and below only.

Electric power requirements and costs are net of acid plant generation on site.

Maintenance costs include materials, consumables and supplies only, maintenance labor costs are included in the Labor estimates.

Miscellaneous costs include the recoupment of the YPF Luz powerline and Calingasta substation investment, equivalent to approximately $0.03/kWhr increase on power costs during the payback term.

21.3.3 General & Administrative (G&A) Costs

The General and Administrative (G&A) costs cover all costs associated with maintaining a regional office in San Juan, a geology and personnel staging area at the existing Calingasta facilities, and necessary site administration and general services at the Los Azules mine site. Taxes and royalties are included in the financial model separately from G&A. Labor rates assume Hays PLC 2022 survey data for Argentina and local burdens/on-costs converted to USD and provided by McEwen.

The overall combined G&A costs estimated for these areas is approximately $40 million per year with category breakouts provided below in Table 21.10.

Table 21.10: Consolidated G&A (San Juan, Calingasta, Los Azules Site)		
	Annual Costs	**Basis**
General & Admin Labor	$12,809,000	G&A Staffing Plan - all three sites
Materials & Supplies		
Admin Supplies	$144,000	Location based allowances
Safety & Health	$192,000	Location based allowances
Equipment & Materials	$342,000	Location based allowances

Table 21.10: Consolidated G&A (San Juan, Calingasta, Los Azules Site)		
	Annual Costs	**Basis**
Utilities	$264,000	Location based allowances
Fuel & Transportation	$1,546,000	Staffing plan @ $5/person/day
Rent	$72,000	Location based allowances
Telecommunications/Internet	$33,000	Location based allowances
Insurances	$60,000	Location based allowances
Sub-Contracts		
Security	$264,000	Location based allowances
Waste Management	$128,000	Location based allowances
Legal Services	$300,000	Location based allowances
Tax & Accounting	$24,000	Location based allowances
Advertising & Recruiting	$6,000	Location based allowances
Transportation (Buses)	$3,600,000	30 buses @ $10K/month for rotations & site labor distribution
Janitorial (non-camp)	$60,000	Location based allowances
Sample Transport Services	$9,000	Location based allowances
Misc. Services/Maintenance	$168,000	Location based allowances
Software & IT	$1,200,000	All locations, $100,000/month
Advertising & Recruiting	$6,000	Location based allowances
Subscriptions & Services	$6,000	Location based allowances
Travel & Entertainment	$120,000	Location based allowances
Misc. Operating Expenses	$120,000	Location based allowances
Camp Operations	$18,434,000	Based on staffing count @ $65/day/person
Consolidated G&A Total	**$39,907,000**	

Los Azules Site

Los Azules site costs are those not included in the direct mine and processing operations cost and labor for necessary administration and site services (safety, security, purchasing and warehousing, and camp operations). The overall combined G&A costs estimated for this location is provided below in Table 21.11.

Table 21.11: Los Azules Site G&A		
Site Admin Labor	$9,157,149	G&A Staffing Plan
Materials & Supplies		
Admin Supplies	$120,000	Allowance $10,000/month
Safety & Health	$180,000	0.5% of operating labor costs
Equipment & Materials	$240,000	Allowance $20,000/month
Utilities	$240,000	Allowance $20,000/month
Fuel & Transportation	$1,491,025	Ops Staffing Plan @ $5/person/day
Sub-Contracts		
Security	$240,000	Allowance $20,000/month
Waste Management	$120,000	Allowance $10,000/month
Transportation (Buses)	$3,600,000	30 buses @ $10K/month for rotations & labor dist.
Janitorial (non-camp)	$60,000	Allowance $5,000/month
Telecommunications/Internet	$18,000	Allowance $1500/month
Misc. Services/Maintenance	$120,000	Allowance $10,000/month

Table 21.11: Los Azules Site G&A		
Camp Operations	$16,536,325	Camp Staffing Plan (excl camp staff) @ $65/person/day
Los Azules Site G&A Sub-Total	**$32,122,499**	

Staffing levels were built up from typical staffing for similar types of operations by job type and includes staffing above the area general manager levels for mining and processing. The Los Azules Site staffing plan only considers the necessary site activities, minimizing the on-site requirements in favor of locations at Calingasta or San Juan.

A preliminary G&A Staffing plan for the Los Azules site contingent is shown in Table 21.12 below.

Table 21.12: Los Azules Site Based G&A Staffing and Cost

	No. Per Shift	Total Personnel	Gross Cost (Includes absenteeism)
			USD/Year
Los Azules Site			
Site General Manager	1	1	$250,000
Asst. General Manager	1	1	$200,000
Site Admin. Manager	1	2	$110,829
Admin. Assistants	4	8	$219,510
Support Staff	4	16	$497,901
Transport Services Supt	1	2	$110,829
Commercial Manager	1	2	$110,829
Contract Admins	1	4	$143,369
Procurement Specialists	2	8	$286,738
Warehouse Supervisor	1	4	$177,071
Warehouse Operators	4	16	$603,848
Transport Services Supt	1	2	$110,829
Security Superintendent	1	2	$88,535
Supervisor Security	2	8	$286,738
Security Emergency Response	1	4	$143,369
Safety Superintendent	1	2	$110,829
Safety Supervisors	2	8	$354,141
Area Safety Lead	4	16	$573,475
Support Staff	4	16	$497,901
Medical Practitioner	1	2	$110,829
Paramedic	1	4	$177,071
Ambulance Attendant	4	16	$497,901
Maintenance General Manager	1	1	$180,000
Electrical Superintendent	1	1	$74,908
Mechanical Superintendent	1	1	$74,908
Civil Superintendent	1	1	$74,908
Plant Services Supt	1	1	$74,908
Wastewater Supervisor	1	4	$221,658
Wastewater Operators	4	16	$603,848

Table 21.12: Los Azules Site Based G&A Staffing and Cost			
	No. Per Shift	Total Personnel	Gross Cost (Includes absenteeism)
			USD/Year
Waste Management Supervisor	1	4	$177,071
Utilities and Services Manager	1	2	$149,817
Maintenance Planners	2	8	$301,924
Technical Services Manager	1	1	$95,836
Project Manager	1	2	$191,671
Projects Coordinator	1	2	$71,684
Engineers	4	8	$443,316

The Los Azules Camp Services are based on site staffing estimates and allowances for temporary workers, contractors, and direct staff, excluding the camp service staff that are included in the camp rate. Camp rates are based on current camp costs per person from regional remote camp data and current camp costs at Los Azules. Camp loading for the project on average is shown in Table 21.13 below.

Table 21.13: Site Camp Planning		
	Initial	**Ultimate**
Mine	240	1105
Process	170	218
G&A Site Staff	107	107
Contractors	150	150
Camp Ops/Services	249	560
Visitors/Executive	30	50
Spares	50	50
Total Rooms	**996**	**2240**

Calingasta Site

The Calingasta site will house the geology team and core processing activities as is currently in place. A local site security contracted person and family housing is considered at this location. The location will also include accommodations for transient staff and staging areas for bussing to the Los Azules site. Calingasta site G&A costs estimated for this location are provided below in Table 21.14.

Table 21.14: Calingasta Site G&A		
Calingasta Site G&A	**Annual Costs**	**Basis**
Calingasta Admin Labor	$1,439,344	G&A Staffing Plan
Materials & Supplies		
Admin Supplies	$12,000	Allowance $1,000/month
Safety & Health	$6,000	Allowance $500/month
Equipment & Materials	$18,000	Allowance $1,500/month
Geo Equipment & Materials	$60,000	Allowance $5,000/month
Utilities	$6,000	Allowance $500/month
Fuel & Transportation	$36,500	20 local Staff @ $5/person/day
Sub-Contracts		

Table 21.14: Calingasta Site G&A		
Calingasta Site G&A	**Annual Costs**	**Basis**
Security	$12,000	Allowance $1,000/month
Waste Management	$6,000	Allowance $500/month
Sample Transport Services	$9,000	Allowance $750/month
Misc. Services/Maintenance	$24,000	Allowance $2,000/month
Rent	$12,000	Allowance $1,000/month
Telecommunications/Internet	$6,000	Allowance $500/month
Camp Operations	$1,898,000	80 staff (excl camp staff) @ $65/person/day
Calingasta Site G&A Sub-Total	**$3,544,844**	

Staffing levels were built up from current geology team staffing and projected future support requirements once a camp is in place.

A preliminary G&A Staffing plan for the Calingasta site contingent is shown in Table 21.15 below.

Table 21.15: Calingasta Staffing			
	No. Per Shift	**Total Personnel**	**Gross Cost (Includes absenteeism)**
			USD/Year
Calingasta Site			
Calingasta Admin. Manager	1	1	$44,268
Admin. Assistant	1	1	$27,439
Support Staff	2	2	$62,238
Transport Services Technician	1	2	$71,684
Security	1	2	$75,481
Technical Services Manager	1	1	$95,836
Project Geologists	2	4	$383,342
Geological Technicians	6	12	$430,107
Support Staff	4	8	$248,950

The Calingasta Camp Services are based on the site staffing estimates and allowances for temporary workers, contractors, and direct staff, excluding the camp service staff that are included in the camp rate. Camp rates are based on current camp costs per person from regional remote camp data and current camp costs at Los Azules. Camp loading for the project on average is expected to be approximately permanent and temporary staff 80 at any one time. The facilities will be constructed in advance to the project start to support exploration and mine site activities.

ACMSA/McEwen Copper San Juan Regional Office

The San Juan Regional office will be the main administrative center for the Los Azules project operations and regional interests. All administrative tasks that do not need to be located at site on a full-time basis are included in this location. G&A costs estimated for this location are provided below in Table 21.16.

Table 21.16: San Juan Office G&A		
San Juan Office G&A		
San Juan Admin Labor	$2,212,887	G&A Staffing Plan
Materials & Supplies		

Table 21.16: San Juan Office G&A		
Admin Supplies	$12,000	Allowance $1,000/month
Safety & Health	$6,000	Allowance $500/month
Equipment & Materials	$24,000	Allowance $2,000/month
Utilities	$18,000	Allowance $1500/month
Fuel & Transportation	$18,250	10 local Staff @ $5/person/day
Rent	$60,000	Allowance $5,000/month
Telecommunications/Internet	$9,000	Allowance $750/month
Insurances	$60,000	Allowance $5000/month
Sub-Contracts		
Security	$12,000	Allowance $1,000/month
Waste Management	$2,400	Allowance $200/month
Legal Services	$300,000	Allowance $25,000/month
Tax & Accounting	$24,000	Allowance $2,000/month
Advertising & Recruiting	$6,000	Allowance $500/month
Misc. Services/Maintenance	$24,000	Allowance $2,000/month
Software & IT	$1,200,000	All locations, $100,000/month
Advertising & Recruiting	$6,000	Allowance $50,000/month
Subscriptions & Services	$6,000	Allowance $500/month
Travel & Entertainment	$120,000	Allowance $10,000/month
Misc. Operating Expenses	$120,000	Allowance $10,000/month
San Juan Office G&A Sub-Total	**$4,240,537**	

Staffing levels were built up from current geology team staffing and projected future support requirements once a camp is in place.

A preliminary G&A Staffing plan for the San Juan Regional Office contingent is shown in Table 21.17 below.

Table 21.17: San Juan Office Staffing			
	No. Per Shift	**Total Personnel**	**Gross Cost (Includes absenteeism)**
			USD/Year
San Juan Offices			
Vice President	1	1	$400,000
Admin Manager	1	1	$55,415
Admin Assistants	3	3	$82,316
Social/Environmental Manager	1	1	$55,415
Community Relations Manager	1	1	$35,842
Admin Assistant	1	1	$27,439
HR Manager	1	1	$55,415
Training Supervisor	1	1	$44,268
HR Assistants	2	2	$54,878
Accounting Manager	1	1	$44,268
Accountants - General	1	1	$35,842
Accountants - AP/AR	5	5	$179,211
Health & Safety Manager	2	2	$110,829

Table 21.17: San Juan Office Staffing			
	No. Per Shift	**Total Personnel**	**Gross Cost (Includes absenteeism)**
Health & Safety Technicians	2	2	$71,684
Commercial Manager	1	1	$55,415
Contract Admins	1	1	$35,842
Procurement Specialists	4	4	$143,369
Logistics Specialist	1	1	$35,842
Technical Services Manager	1	1	$95,836
Project Manager	1	1	$95,836
Projects Coordinator	1	1	$35,842
Engineers	4	4	$221,658
IT/Communications Manager	1	1	$44,268
IT Technicians	2	2	$71,684
Support Staff	4	4	$124,475

22.0 ECONOMIC ANALYSIS

22.1 CAUTIONARY STATEMENT

Certain information and statements contained in this section and in the Report are "forward looking" in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and study parameters of the Project; Mineral Resource estimates; the cost and timing of any development of the Project; the proposed mine plan and mining methods; dilution and extraction recoveries; processing method and rates and production rates; projected metallurgical recovery rates; infrastructure requirements; capital, operating and sustaining cost estimates; the projected life of mine and other expected attributes of the Project; the net present value (NPV) and internal rate of return (IRR after-tax) and payback period of capital; capital; future metal prices; the timing of the environmental assessment process; changes to the Project configuration that may be requested as a result of stakeholder or government input to the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this Report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this Report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Report are subject to the following assumptions:

- There being no significant disruptions affecting the development and operation of the Project.
- The availability of certain consumables and services and the prices for power and other key supplies being approximately consistent with assumptions in the Report.
- Labor and materials costs being approximately consistent with the assumptions in the Report.
- Permitting and arrangements with stakeholders being consistent with current expectations as outlined in the Report.
- All environmental approvals, required permits, licenses and authorizations will be obtained from the relevant governments and other relevant stakeholders.
- Certain tax rates, including the allocation of certain tax attributes, being applicable to the Project.
- The availability of financing for the planned development activities.
- The timelines for exploration and development activities on the Project.
- Assumptions made in Mineral Resource estimate and the financial analysis based on that estimate, including, but not limited to, geological interpretation, grades, commodity price assumptions, extraction and mining recovery rates, hydrological and hydrogeological assumptions, capital and operating cost estimates, and general marketing, political, business, and economic conditions.

The production schedules and financial analysis annualized cash flow table are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. If additional mining, technical, and engineering studies are conducted, these may alter the Project assumptions as discussed in this Report and may result in changes to the calendar timelines presented.

The Project is at the advanced exploration stage of investigation; consequently, this study is at the scoping level of accuracy, preliminary in nature, and includes Inferred mineral resources in the conceptual mine plan and the mine production schedule. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves under the standards set forth in NI 43-101. There is no certainty that the results, estimates or projections in this Preliminary Assessment will be realized.

22.2 METHODOLOGY USED

Samuel Engineering has prepared a discounted cash flow analysis of the Los Azules Project. Technical and cost inputs for the economic model were developed by Samuel Engineering with specific inputs provided by McEwen Mining. These inputs have been reviewed in detail by Samuel Engineering and are accepted as reasonable.

The discounted cash flow analysis was performed on a stand-alone project basis with annual cash flows discounted on an end-of-year basis. The economic evaluation used a real discount rate of 8% and was performed at commencement of construction (denoted as Year minus 3 of the Los Azules Project) using Q1 2023, US dollars.

All costs prior to the start of construction are considered as "sunk costs" and not considered in the economic analysis.

This economic analysis is a direct result of the capital cost estimate and is therefore considered to have the same level of accuracy minus 20% to plus 40%.

22.3 FINANCIAL MODEL PARAMETERS

Technical-economic parameters used in the model are summarized in the following sections. Table 22.1 and Table 22.2 present the model inputs used in the economic analysis based on first quarter, 2023 US dollars.

Table 22.1: Common Model Inputs

Area	Description	Units	Values
General	Tonnes Processed	Billion Tonnes	1.18
	Tonnes Waste Mined	Billion Tonnes	1.37
	Strip Ratio		1.16
	Copper Production – LOM Cu Cathode	t x 1,000	3,938
Metal pricing	Copper price	US$/lb	3.75
Cost criteria	Estimate basis	US$	first quarter 2023
	Inflation/currency fluctuation		None
	Leverage	% Equity	100%
Income tax	Argentina Corporate Income	% Profit	35%
Royalties / payments	San Juan Province	% "Mine Mouth"	3%
	TNR Royalty	% NSR	0.4%
	McEwen Royalty	% NSR	1.25%
Transportation, smelting, and refining charges	Shipping (Point of Sale – Site)	US$/tonne Copper	$44.1
	Brokerage Fee	US$/lb Copper	$0.02
Export Retentions	Argentine Export Retention	% NSR	4.5%

Table 22.2: Option Specific Model Inputs				
Area	**Description**	**Units**	**Base Case 175k tpa Cu**	**Alt. Case 125 tpa Cu**
General	Nominal Cu Cathode Production	TPY	175,000	125,000
	Construction Period	Years	3	3
	Mine Life	Years	27	32
	Operating Life	Years	28	33
	Closure Duration	Years	1	1

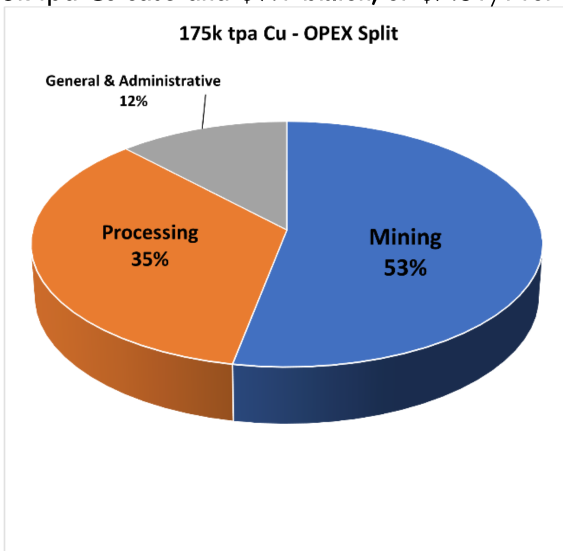
22.3.1 Capital Costs

The total capital cost is estimated at $4.71 billion, including $2.46 billion during preproduction, $244 million for working capital, and $2.24 billion in sustaining capital over the life of the mine. Table 22.3 summarizes the capital cost over the life of the mine.

Table 22.3: Life of Mine Capital Cost Summary ($000s)		
Description	**Base Case 175k tpa Cu**	**Alt. Case 125k tpa Cu**
DIRECT ON-SITE FACILITIES		
Mine Area Facilities	65,556	65,556
Mining Equipment	197,025	202,415
Mine Pre-stripping	163,782	171,279
Ore Storage & Handling	234,484	192,536
Heap Leach	158,476	142,132
SX/EW Facilities	250,367	167,720
Sulfuric Acid Plant	94,935	79,885
Ancillary Facilities	23,295	23,295
Site Development & Yard Utilities	126,323	112,249
Pre-Production Operations	30,607	18,784
OFF-SITE FACILITIES		
Power Supply	0	0
Fresh Water Supply	1,000	1,000
Permanent Camp (Camp/Offices)	26,000	26,000
Access Roads	138,361	138,361
Aviation	2,000	2,000
FIELD DISTRIBUTABLES	163,261	143,876
CONSTRUCTION MANAGEMENT / FIELD OFFICES	92,437	75,805
ENGINEERING PROCUREMENT & PROJECT MGT	**77,708**	**65,099**
OTHER INDIRECTS		
Owner's Costs	75,173	63,386
Freight	45,832	39,029
Contingency	495,248	423,050
Total Preproduction Capital	**2,461,871**	**2,153,458**
Sustaining	2,085,720	2,187,240
Working Capital (Initial)	243,910	172,456
Total LOM Capital	**4,791,501**	**4,513,153**

22.3.2 Operating Costs

The total LOM operating cost is estimated at $9.5 billion, or $8.12 per tonne of mineralized material processed for the base case 125k tpa Cu case and $9.1 billion, or $7.81/t for the 175k tpa case, as summarized in Table 22.4. The



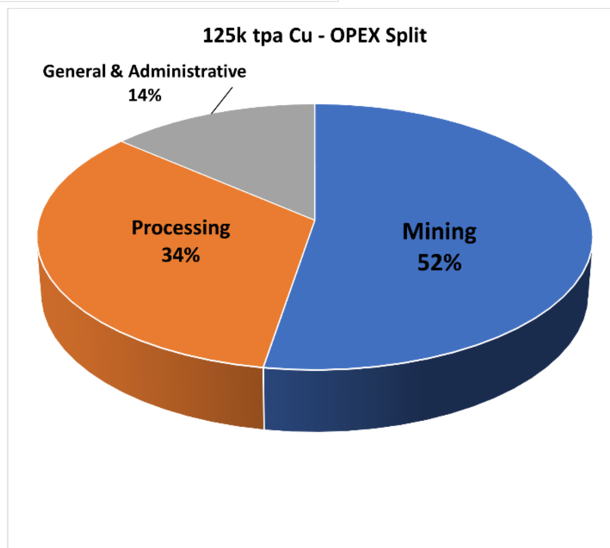
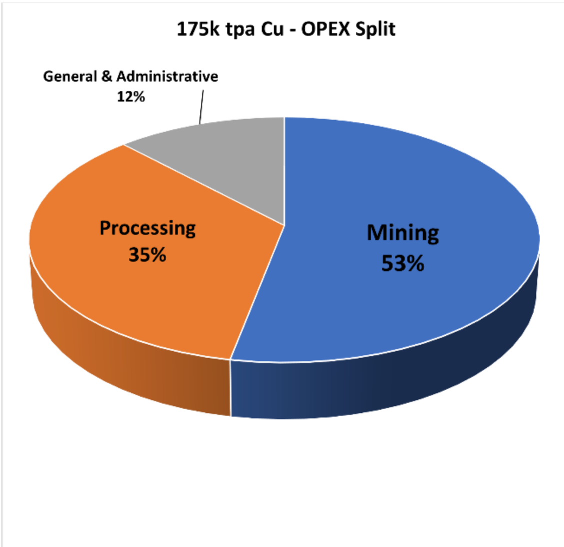


Figure 22.1 show the percentage splits of each LOM operating cost component for each of the cases.

Table 22.4: Life of Mine Operating Cost Summary			
Description			
Base Case 175k tpa Cu			
Mining	4,830,236	4.14	0.56
Processing	3,179,832	2.73	0.37
General & Administrative	1,097,443	0.94	0.13
LOM Operating Cost	**9,107,510**	**7.81**	**1.05**
Alt. Case 125k tpa Cu			
Mining	4,982,500	4.21	0.57
Processing	3,188,714	2.70	0.37

General & Administrative	1,296,978	1.11	0.15
LOM Operating Cost	**9,468,191**	**8.12**	**1.09**



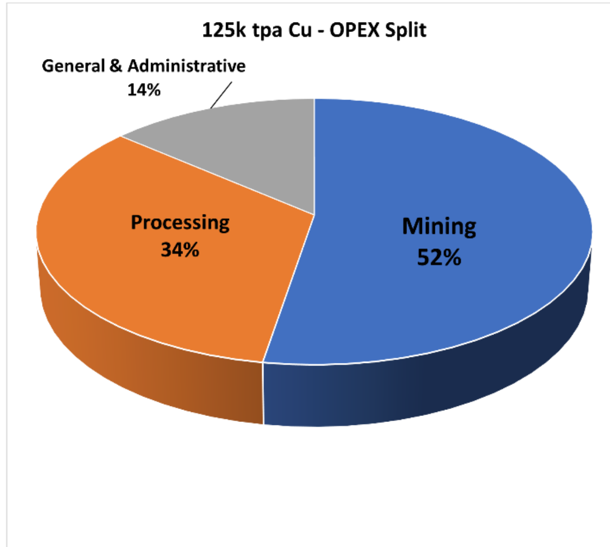


Figure 22.1: LOM Operating Costs per Tonne Mineralized Material (Samuel Engineering 2023)

Note that the processing cost is slightly higher in the economic evaluation than in the operating cost section of the report. In the economic evaluation, it has been assumed that the cost of the electrical substation and power line are included in the unit power rate. In order to approximate the increase in power cost, the annualized value of the expected capital expenditure for the electrical installations was calculated using a 10% profit on expenditure, at an annualized interest rate of 5% over the duration of plant operation. The cost for the base case plant is approximately equivalent to 3 cents per kW-hr.

22.3.3 Royalties and Taxes

The Los Azules project is charged several royalties. The Provincial (San Juan) royalty is "mine mouth" based and is calculated at 3% of the gross revenue less non-mining expenses. A reduction of the fee is allowed for capital expenditures made for the good of the public. Those expenditures are deducted from the fees up to 70% of the annual fee due. The project will also be charged two "NSR" based royalties, TNR at 0.4% and McEwen Mining at 1.25%. These royalties are calculated by deducting the costs for shipping, ocean freight, smelter treatment and refining charges, process operating costs and general and administrative costs associated with all areas of the Project except mining from the total gross revenue generated from the value of the metals to be shipped to the purchaser.

In addition to the royalties, Argentina imposes a 4.5% export retention tax on the value of the metals at the point of export. In estimating this export tax, the amount of total gross revenue less transportation, treating and refining charges are used as the cost basis. In the economic evaluation we have assumed that 10,000 tonnes of copper cathode will be sold within country not requiring export tax. The 10,000 tpy represents approximately 50% of the annual import of copper for Argentina.

Further, the project is required to pay VAT taxes on Initial capital and sustaining capital at a rate of 10.5% of the direct costs. During operations we recover 95% of the VAT on initial capital (50% in the year following, and 50% the year after) and 95% of the VAT on sustaining capital in the following year. In addition, operating VAT is charged at a rate of 21% on all non-labor operating expenses. Operating VAT is also recovered at 95%. The portion of VAT paid attributable to in country sales is collected in the current year while the balance is recovered in the following year.

In addition to the VAT taxes, the project is subject to the Argentine Corporate Profit Tax of 35%, a Debit and Credit Bank tax of 1.2% of the gross "In-Country" sales, and an Operating Bank Tax. The Operating Bank Tax is charged on non-labor related operating expenses at a rate of 1.2%. A portion of this, (0.2% of the non-labor expenses) can be recovered in the following year.

22.4 ECONOMIC RESULTS

The Project's LOM cash flow results are summarized in Table 22.5: Project Economic Summary. The Project is at the exploration stage of investigation; consequently, this study is at the scoping level of accuracy, preliminary in nature, and includes Inferred mineral resources in the conceptual mine plan and the mine production schedule. Inferred mineral resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves under the standards set forth in Canadian National Instrument 43-101. There is no certainty that the results, projects or estimates in this PEA will be realized.

Table 22.5: Project Economic Summary

Description	Units	Base Case 175k tpa Cu	Alt. Case 125k tpa Cu
Gross Revenue	**$000s**	**32,555,785**	**32,556,056**
Less Transportation, TC and RC Costs	$000s	167,640	166,357
Net Smelter Return	**$000s**	**32,388,145**	**32,389,699**
Less Royalties	$000s	1,205,692	1,203,590
Gross Income from Mining	**$000s**	**31,182,453**	**31,186,109**
Less Operating Costs	$000s	9,107,510	9,468,191
Less Export Retention	$000s	1,356,648	1,335,094
Net Profit Before Depreciation/Amortization	**$000s**	**20,718,295**	**20,382,824**

Table 22.5: Project Economic Summary			
Description	**Units**	**Base Case 175k tpa Cu**	**Alt. Case 125k tpa Cu**
Less Depreciation/Amortization	$000s	4,781,016	4,577,905
Net Profit Before Taxes	**$000s**	**15,937,279**	**15,804,918**
Less Income Taxes	$000s	5,579,398	5,520,263
Net Profit After Taxes	**$000s**	**10,357,882**	**10,284,655**
Plus Add-back Non-Cash Depreciation/Amortization	$000s	4,781,016	4,577,905
Less Sustaining Capital	$000s	2,085,720	2,187,240
Less Capital Costs	$000s	2,461,871	2,153,458
Less Working Capital	$000s	37,306	55,730
Plus Recapture Working Capital/Spares/First Fills	$000s	71,456	87,565
Less VAT	$000s	2,009,785	2,040,059
Plus Recapture of VAT	$000s	1,909,296	1,938,056
Less Mine Reclamation	$000s	179,690	179,690
Pre-Tax Cash Flow	**$000s**	**15,924,675**	**15,792,268**
IRR (Pre-Tax)	%	26.5	22.9
NPV @ 5%	$000s	7,010,791	5,697,030
NPV @ 8%	$000s	4,436,045	3,278,368
NPV @ 10%	$000s	3,286,549	2,288,731
After-Tax Cash Flow	**$000s**	**10,345,277**	**10,272,005**
IRR (Post-Tax)	%	21.2	18.4
NPV @ 5%	$000s	4,378,483	3,540,051
NPV @ 8%	$000s	2,658,535	1,928,663
NPV @ 10%	$000s	1,889,459	1,267,519
Pre-tax Pay Back Period	**Years**	**2.7**	**3.4**

22.5 SENSITIVITY ANALYSIS

Table 22.6 through Table 22.8 and Figure 22.2 through Figure 22.7 show the relative sensitivity of NPV and IRR as capital and operating costs and copper price change in the Base Case 125k tpa Cu economic model.

The sensitivity analysis shows that the Project is the most sensitive to copper price. Operating and capital costs changes have a lower impact on the Project NPV than the former variable.

Table 22.6: Copper Price Sensitivity for 175k Cu Case							
Sensitivity (%) / Item	**Metal Pricing**	**Pre-Tax**			**Post-Tax**		
	Copper Price	**NPV**	**IRR**	**Payback**	**NPV**	**IRR**	**Payback**
	Cu/lb	**$M**	**%**	**Years**	**$M**	**%**	**Years**
-50%	$1.88	($895)	3%	16.01	($970)	2%	16.33
-45%	$2.06	($360)	6%	12.80	($557)	5%	13.18

Table 22.6: Copper Price Sensitivity for 175k Cu Case

Sensitivity (%) / Item	Metal Pricing	Pre-Tax			Post-Tax		
	Copper Price	NPV	IRR	Payback	NPV	IRR	Payback
	Cu/lb	$M	%	Years	$M	%	Years
-40%	$2.25	$175	9%	9.71	($167)	7%	10.26
-35%	$2.44	$708	12%	7.88	$206	9%	8.35
-30%	$2.63	$1,242	14%	6.64	$570	11%	7.16
-25%	$2.81	$1,775	16%	5.65	$926	13%	6.24
-20%	$3.00	$2,307	18%	4.83	$1,277	15%	5.48
-15%	$3.19	$2,840	21%	4.12	$1,624	17%	4.84
-10%	$3.38	$3,372	23%	3.46	$1,969	18%	4.24
-5%	$3.56	$3,904	25%	2.97	$2,314	20%	3.68
0%	$3.75	$4,436	27%	2.73	$2,659	21%	3.18
5%	$3.94	$4,968	28%	2.52	$3,003	23%	2.90
10%	$4.13	$5,499	30%	2.34	$3,346	24%	2.75
15%	$4.31	$6,031	32%	2.18	$3,689	25%	2.61
20%	$4.50	$6,563	34%	2.05	$4,032	27%	2.49
25%	$4.69	$7,094	35%	1.94	$4,375	28%	2.38
30%	$4.88	$7,625	37%	1.85	$4,717	29%	2.28
35%	$5.06	$8,157	39%	1.77	$5,060	30%	2.18
40%	$5.25	$8,688	40%	1.70	$5,403	32%	2.10
45%	$5.44	$9,219	42%	1.63	$5,746	33%	2.02
50%	$5.63	$9,751	43%	1.57	$6,089	34%	1.95

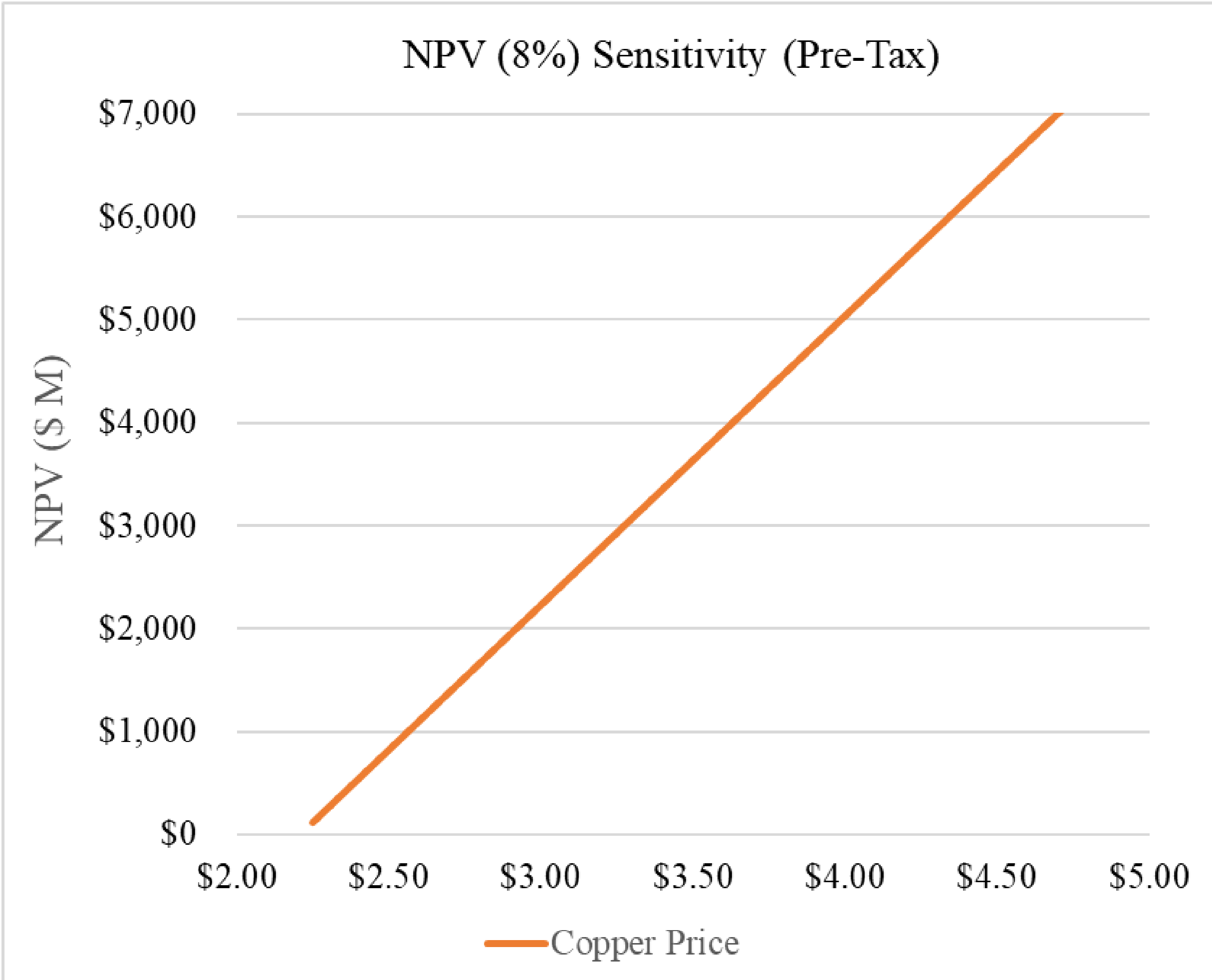


Figure 22.2: Copper Price per Pound Sensitivity on NPV @ 8% (Pre-tax, 175k Cu Case) (Samuel Engineering 2023)

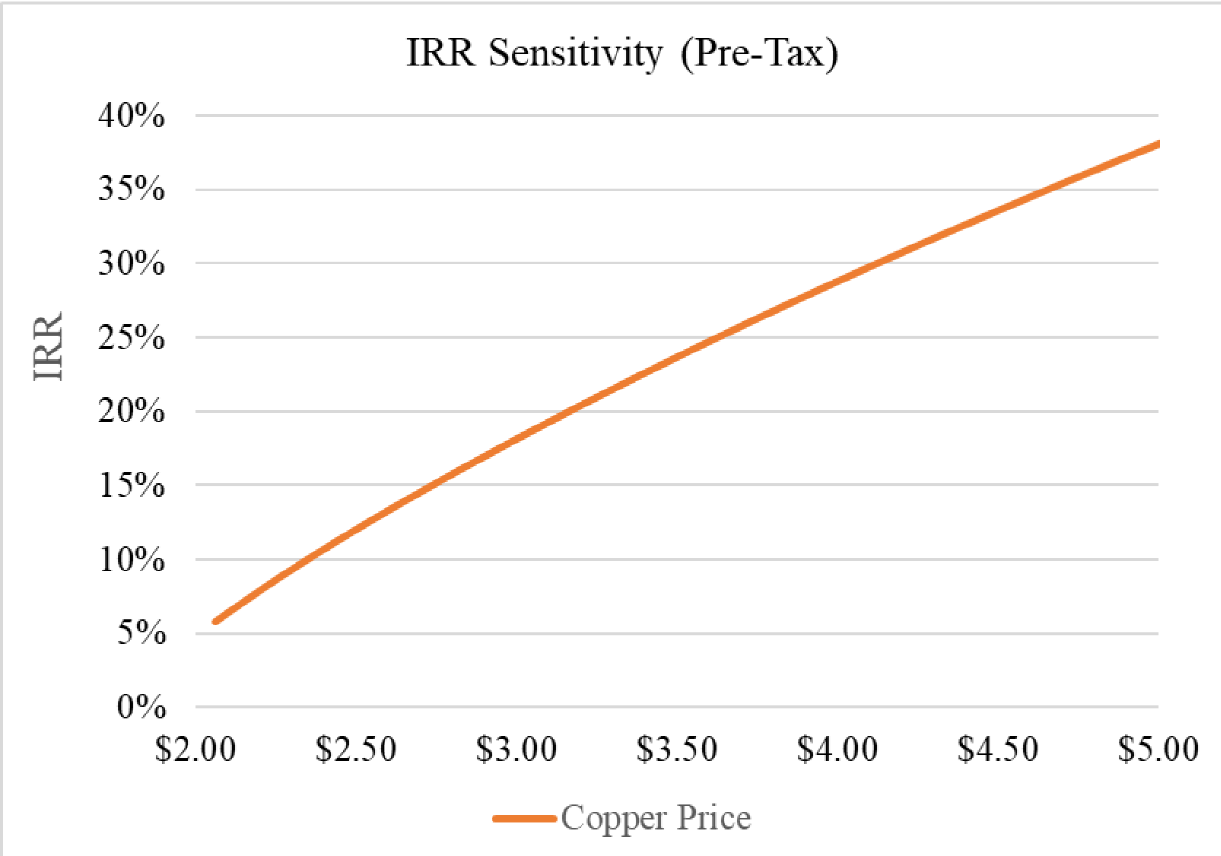


Figure 22.3: Copper Price per Pound Sensitivity on IRR (Pre-tax, 175k Cu Case) (Samuel Engineering 2023)

Table 22.7: CAPEX Sensitivity (Initial + Sustaining) for 175k Cu Case						
Sensitivity (%) / Item	Pre-Tax			Post-Tax		
	NPV	IRR	Payback	NPV	IRR	Payback
	$M	%	Years	$M	%	Years
-25%	$5,128	35%	1.91	$3,152	28%	2.35
-20%	$4,970	33%	2.05	$3,041	26%	2.50
-15%	$4,813	31%	2.21	$2,930	25%	2.64
-10%	$4,656	29%	2.38	$2,819	23%	2.79
-5%	$4,498	28%	2.55	$2,708	22%	2.93
0	$4,341	26%	2.73	$2,597	21%	3.18
5%	$4,184	25%	2.91	$2,484	20%	3.54
10%	$4,026	24%	3.15	$2,372	19%	3.94
15%	$3,869	23%	3.48	$2,260	18%	4.25
20%	$3,711	21%	3.85	$2,148	17%	4.56
25%	$3,554	20%	4.18	$2,036	17%	4.88

Table 22.8: OPEX Sensitivity for 175k Cu Case						
Sensitivity (%) / Item	Pre-Tax			Post-Tax		
	NPV	IRR	Payback	NPV	IRR	Payback
	$M	%	Years	$M	%	Years
-25%	$5,101	29%	2.52	$3,098	23%	2.89
-20%	$4,949	28%	2.56	$2,998	23%	2.93
-15%	$4,797	28%	2.60	$2,898	22%	2.96
-10%	$4,645	27%	2.64	$2,797	22%	3.00
-5%	$4,493	27%	2.68	$2,697	21%	3.09
0	$4,341	26%	2.73	$2,597	21%	3.18
5%	$4,189	26%	2.77	$2,496	21%	3.28
10%	$4,037	25%	2.82	$2,396	20%	3.38
15%	$3,885	25%	2.87	$2,295	20%	3.49
20%	$3,733	24%	2.92	$2,195	19%	3.62
25%	$3,580	24%	2.98	$2,095	19%	3.75

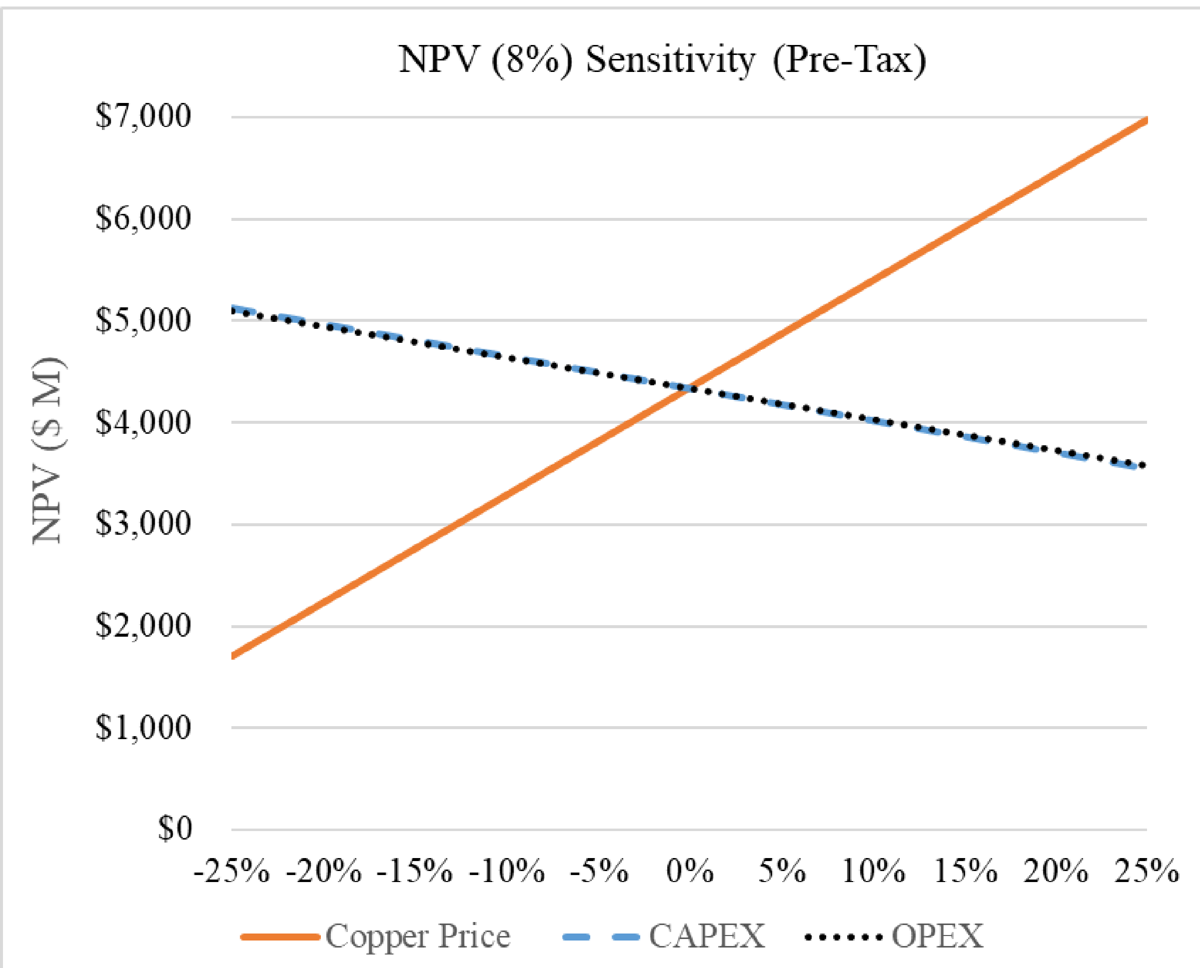


Figure 22.4: Multiple % Sensitivity on NPV @ 8% (Pre-tax, 175k Cu Case) (Samuel Engineering 2023)

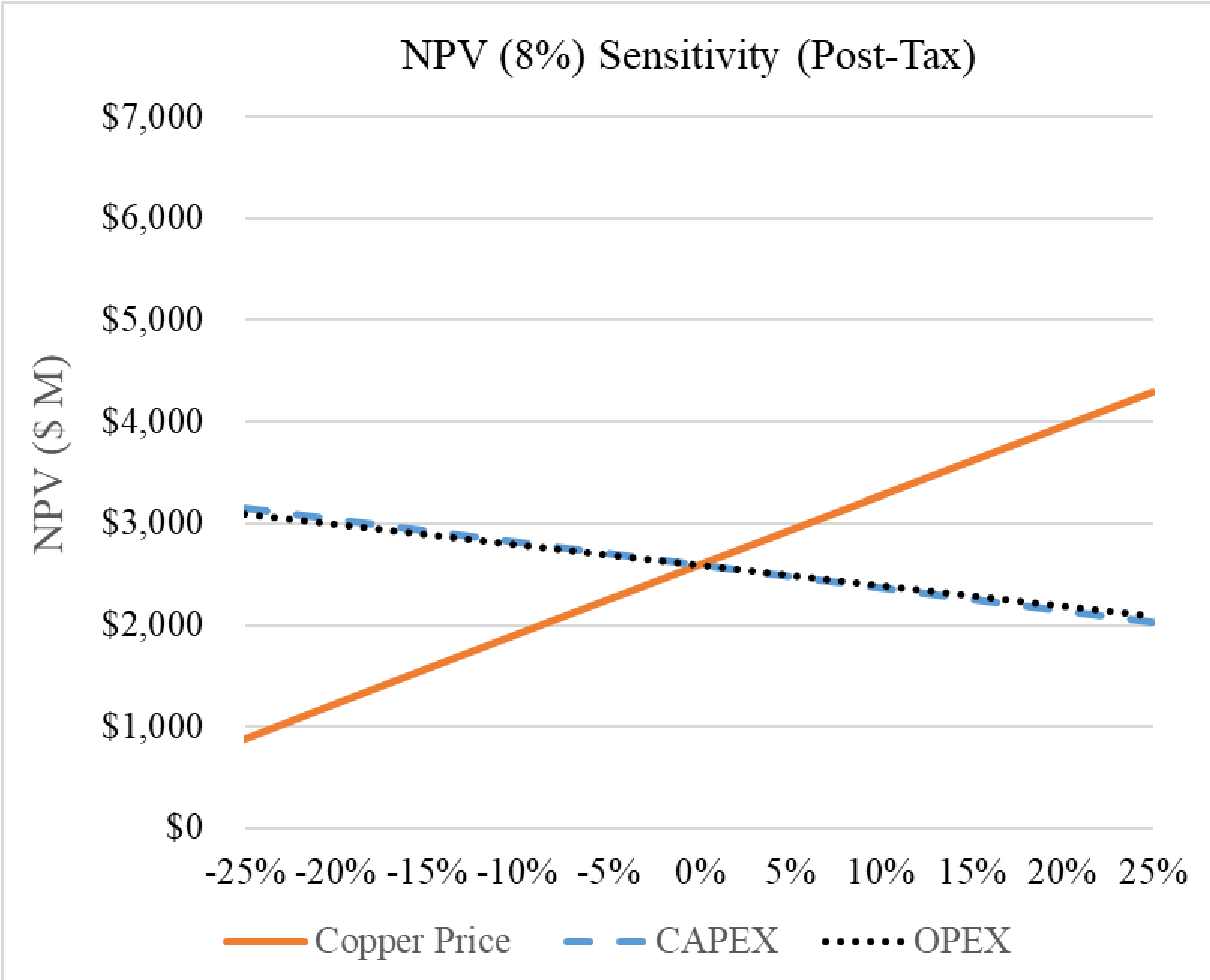


Figure 22.5: Multiple % Sensitivity on NPV @ 8% (Post-tax, 175k Cu Case) (Samuel Engineering 2023)

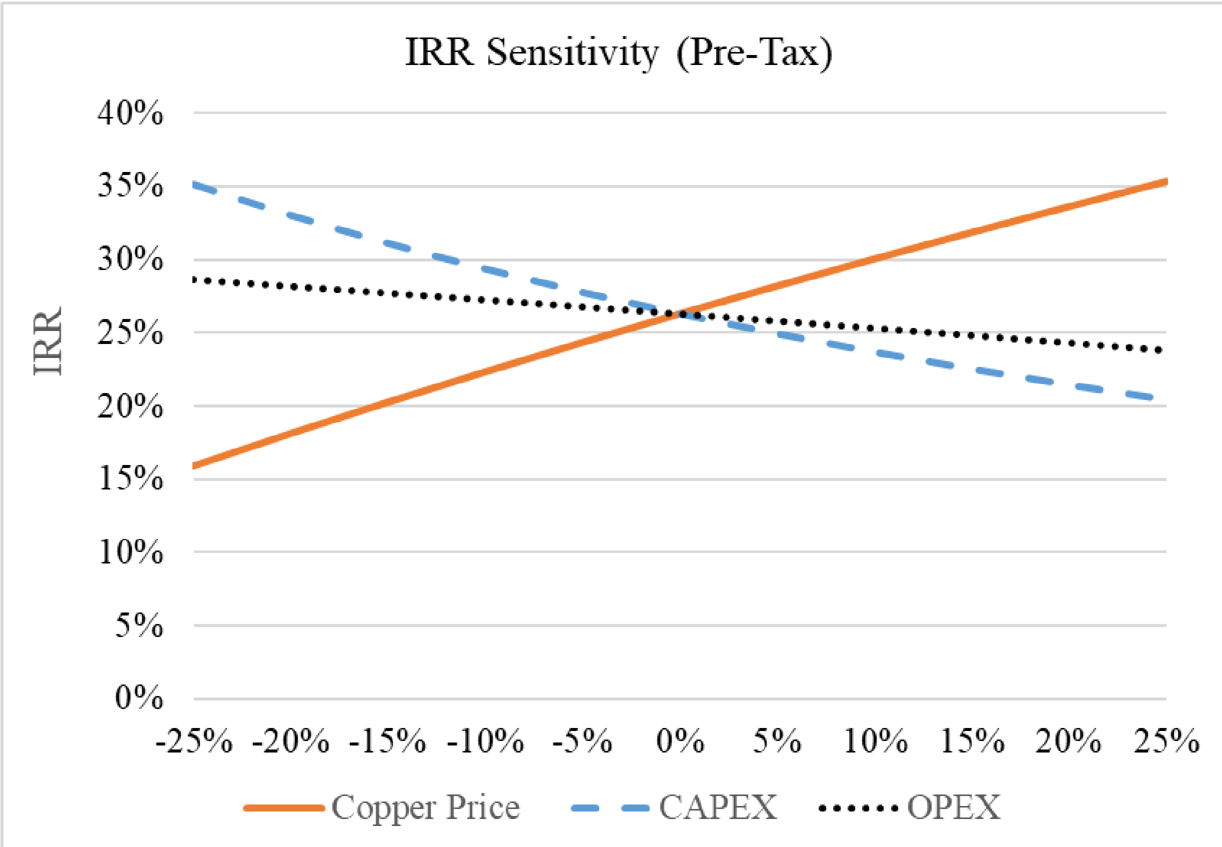


Figure 22.6: Multiple % Sensitivity on IRR (Pre-tax, 175k Cu Case) (Samuel Engineering 2023)

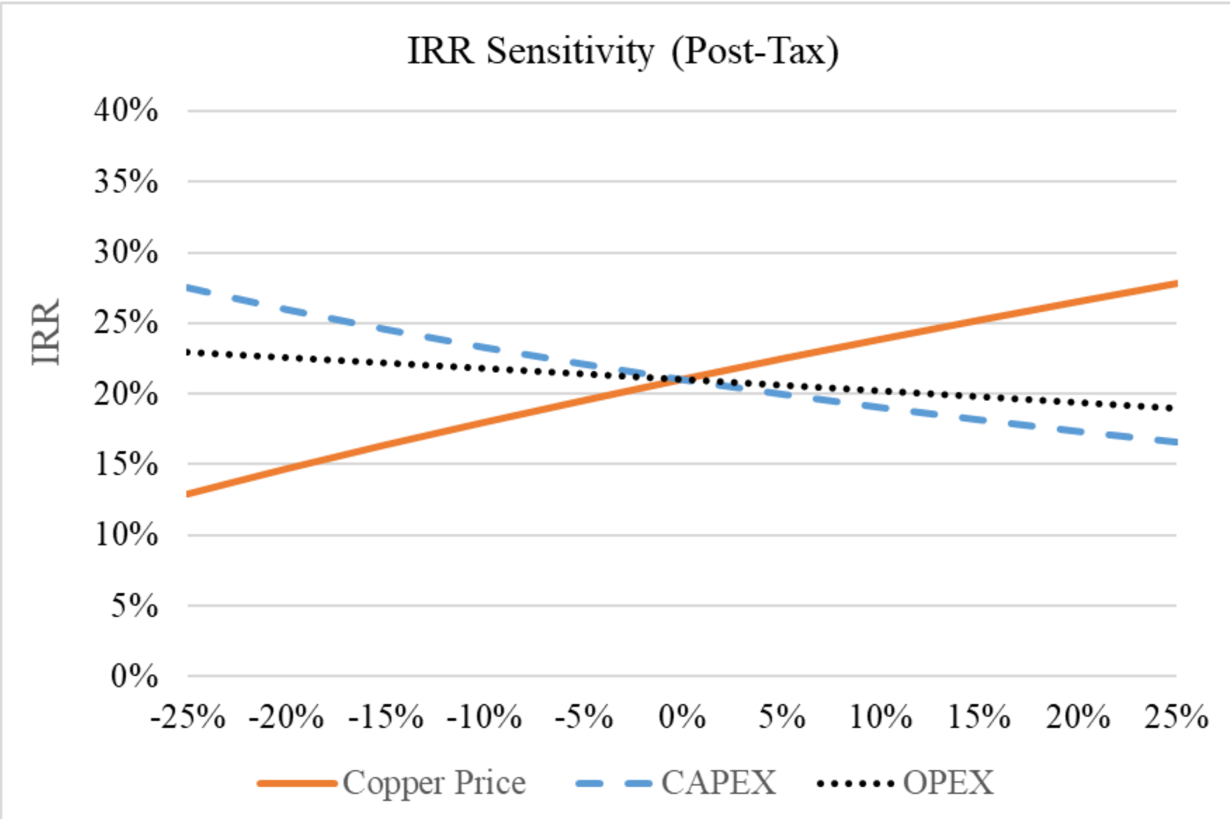


Figure 22.7: Multiple % Sensitivity on IRR (Post-tax, 175k Cu Case) (Samuel Engineering 2023)

Table 22.9: Copper Price Sensitivity for 125k Cu Case							
Sensitivity (%) / Item	Metal Pricing	Pre-Tax			Post-Tax		
	Copper Price	NPV	IRR	Payback	NPV	IRR	Payback
	Cu/lb	$M	%	Years	$M	%	Years
-50%	$1.88	($1,072)	2%	24.52	($1,104)	1%	24.88
-45%	$2.06	($635)	5%	19.47	($738)	3%	19.91
-40%	$2.25	($198)	7%	15.18	($394)	6%	15.71
-35%	$2.44	$237	9%	10.77	($80)	8%	12.45
-30%	$2.63	$673	11%	6.72	$215	9%	9.35
-25%	$2.81	$1,108	13%	5.72	$506	11%	6.18
-20%	$3.00	$1,542	15%	5.09	$793	13%	5.54
-15%	$3.19	$1,977	17%	4.32	$1,079	14%	5.06
-10%	$3.38	$2,411	19%	3.74	$1,364	16%	4.41
-5%	$3.56	$2,845	21%	3.32	$1,647	17%	3.87
0%	$3.75	$3,278	23%	2.99	$1,929	18%	3.44
5%	$3.94	$3,712	25%	2.82	$2,209	20%	3.10
10%	$4.13	$4,145	26%	2.66	$2,490	21%	2.92
15%	$4.31	$4,579	28%	2.52	$2,771	22%	2.80
20%	$4.50	$5,012	30%	2.40	$3,051	23%	2.69

Table 22.9: Copper Price Sensitivity for 125k Cu Case							
Sensitivity (%) / Item	Metal Pricing	Pre-Tax			Post-Tax		
	Copper Price	NPV	IRR	Payback	NPV	IRR	Payback
	Cu/lb	$M	%	Years	$M	%	Years
25%	$4.69	$5,445	31%	2.29	$3,332	25%	2.60
30%	$4.88	$5,878	33%	2.19	$3,611	26%	2.51
35%	$5.06	$6,311	34%	2.10	$3,891	27%	2.42
40%	$5.25	$6,744	36%	2.01	$4,170	28%	2.35
45%	$5.44	$7,177	37%	1.94	$4,449	29%	2.27
50%	$5.63	$7,610	38%	1.87	$4,728	30%	2.21

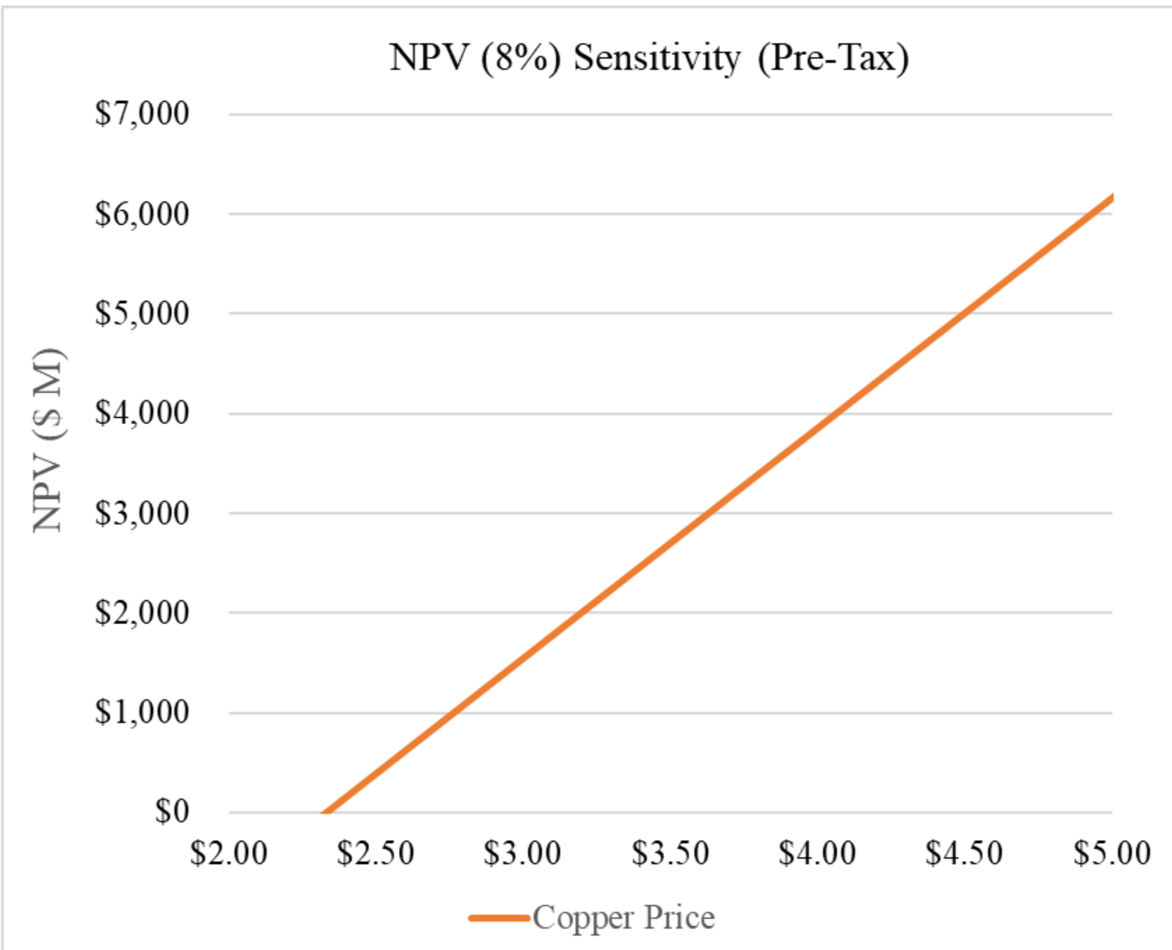


Figure 22.8: Copper Price per Pound Sensitivity on NPV @ 8% (Pre-tax, 125k Cu Case) (Samuel Engineering 2023)

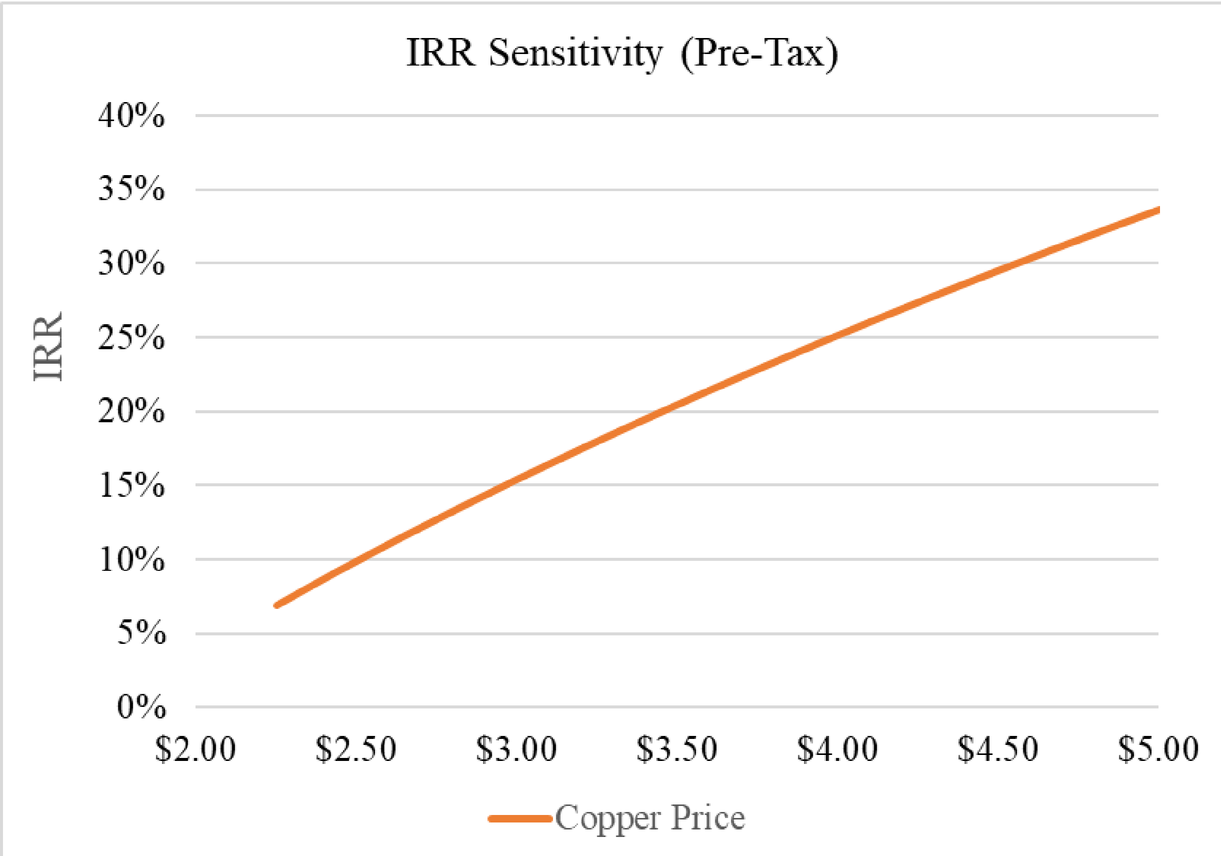


Figure 22.9: Copper Price per Pound Sensitivity on IRR (Pre-tax, 125k Cu Case) (Samuel Engineering 2023)

Table 22.10: CAPEX Sensitivity (Initial + Sustaining) for 125k Cu Case						
Sensitivity (%) / Item	Pre-Tax			Post-Tax		
	NPV	IRR	Payback	NPV	IRR	Payback
	$M	%	Years	$M	%	Years
-25%	$3,953	31%	2.29	$2,411	24%	2.60
-20%	$3,818	29%	2.43	$2,315	23%	2.72
-15%	$3,683	27%	2.57	$2,218	22%	2.84
-10%	$3,548	26%	2.71	$2,122	20%	2.96
-5%	$3,413	24%	2.85	$2,025	19%	3.16
0	$3,278	23%	2.99	$1,929	18%	3.44
5%	$3,143	22%	3.22	$1,832	17%	3.74
10%	$3,009	21%	3.47	$1,734	17%	4.06
15%	$2,874	20%	3.73	$1,636	16%	4.39
20%	$2,739	19%	4.01	$1,538	15%	4.72
25%	$2,604	18%	4.33	$1,438	14%	5.05

Table 22.11: OPEX Sensitivity for 125k Cu Case						
Sensitivity (%) / Item	Pre-Tax			Post-Tax		
	NPV	IRR	Payback	NPV	IRR	Payback
	$M	%	Years	$M	%	Years
-25%	$3,950	25%	2.78	$2,372	20%	3.02
-20%	$3,816	25%	2.82	$2,283	20%	3.10
-15%	$3,681	24%	2.86	$2,195	20%	3.18
-10%	$3,547	24%	2.91	$2,106	19%	3.26
-5%	$3,413	23%	2.95	$2,017	19%	3.35
0	$3,278	23%	2.99	$1,929	18%	3.44
5%	$3,144	22%	3.06	$1,840	18%	3.54
10%	$3,010	22%	3.14	$1,751	18%	3.64
15%	$2,875	21%	3.22	$1,662	17%	3.75
20%	$2,741	21%	3.30	$1,573	17%	3.87
25%	$2,607	20%	3.39	$1,484	16%	3.99

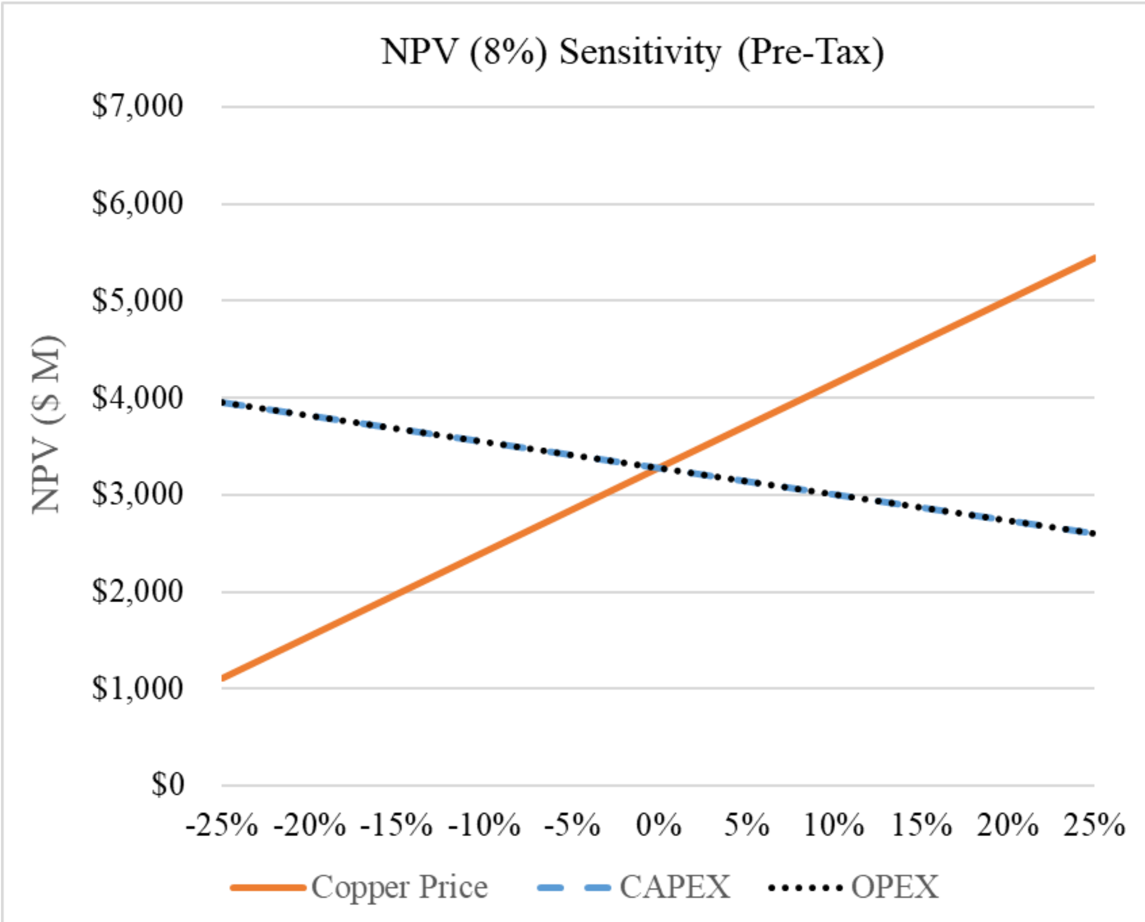


Figure 22.10: Multiple % Sensitivity on NPV @ 8% (Pre-tax, 125k Cu Case) (Samuel Engineering 2023)

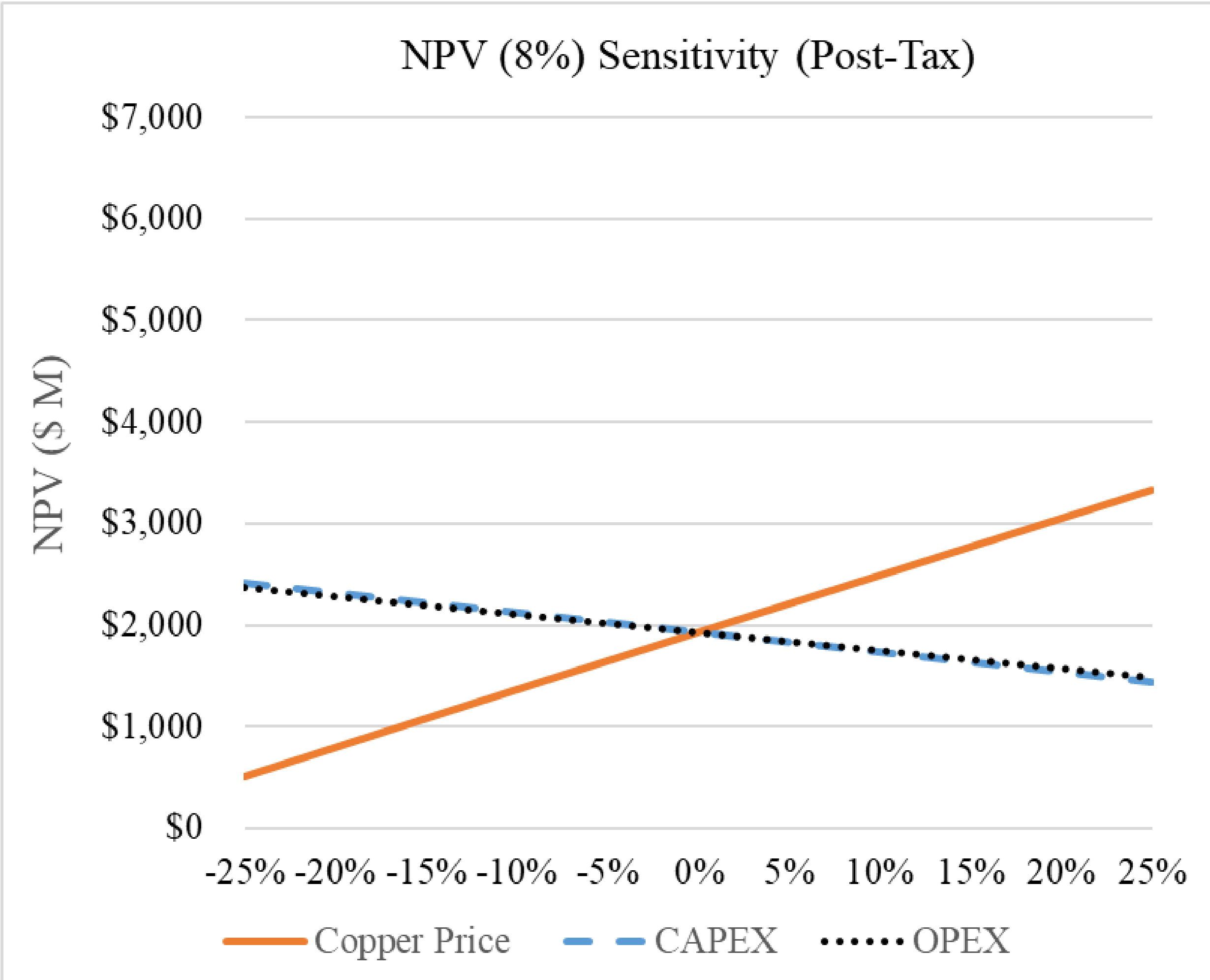


Figure 22.11: Multiple % Sensitivity on NPV @ 8% (Post-tax, 125k Cu Case) (Samuel Engineering 2023)

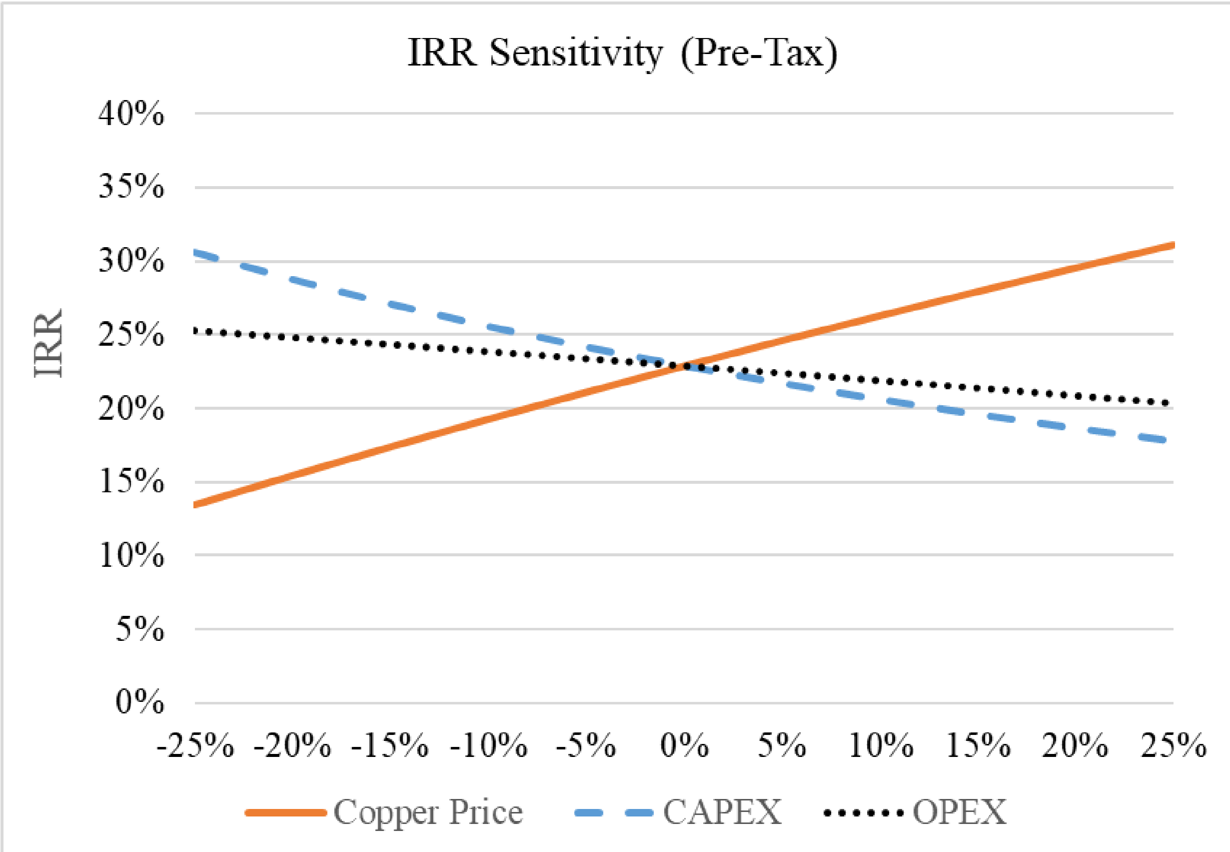


Figure 22.12: Multiple % Sensitivity on IRR (Pre-tax, 125k Cu Case) (Samuel Engineering 2023)

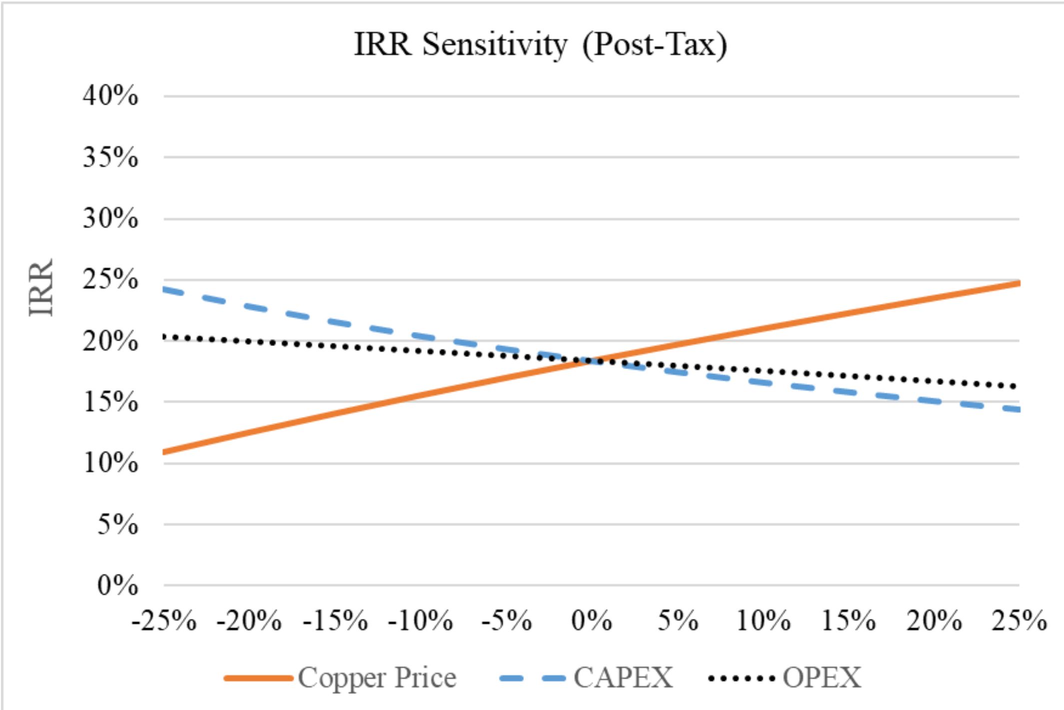


Figure 22.13: Multiple % Sensitivity on IRR (Post-tax, 125k Cu Case) (Samuel Engineering 2023)

22.6 MINE LIFE AND CAPITAL PAYBACK

The operating life of Los Azules Base Case is estimated at 28 years assuming a nominal production rate of 175,000 tonnes per day of copper. This excludes a 3-year construction and preproduction stripping period. At 175k tpa processing rate, and at a copper price of $3.75 per pound, the initial capital pre-tax payback period is projected to be 2.7 years after the start of commercial mining.

The operating life of Los Azules Alternative Case is estimated at 33 years assuming a nominal production rate of 125,000 tonnes per day of copper. At 125k tpa processing rate, and at a copper price of $3.75 per pound, the initial capital pre-tax payback period is projected to be 3.0 years after the start of commercial mining.

23.0 ADJACENT PROPERTIES

Adjacent properties material to the Los Azules project include the mining properties of Fortescue, with their Rincones de Araya project, and the Altar project held by Aldebaran, which are both south of the project, The Soberanía property is surrounded by the Azul 3 and 4, Escorpio I, and the Azul Norte mining rights and a piece of property released by Los Azules within Escorpio IV mining right to maintain the mining right itself within 3,500 ha.

The Soberanía property is currently claimed by ACMSA and three other parties. It is north of the facilities laid out currently and is not anticipated to impact future mining at Los Azules. The western boundary of the property is the border with Chile. Figure 23.1 shows the current property rights owners within and adjacent to the Los Azules project.

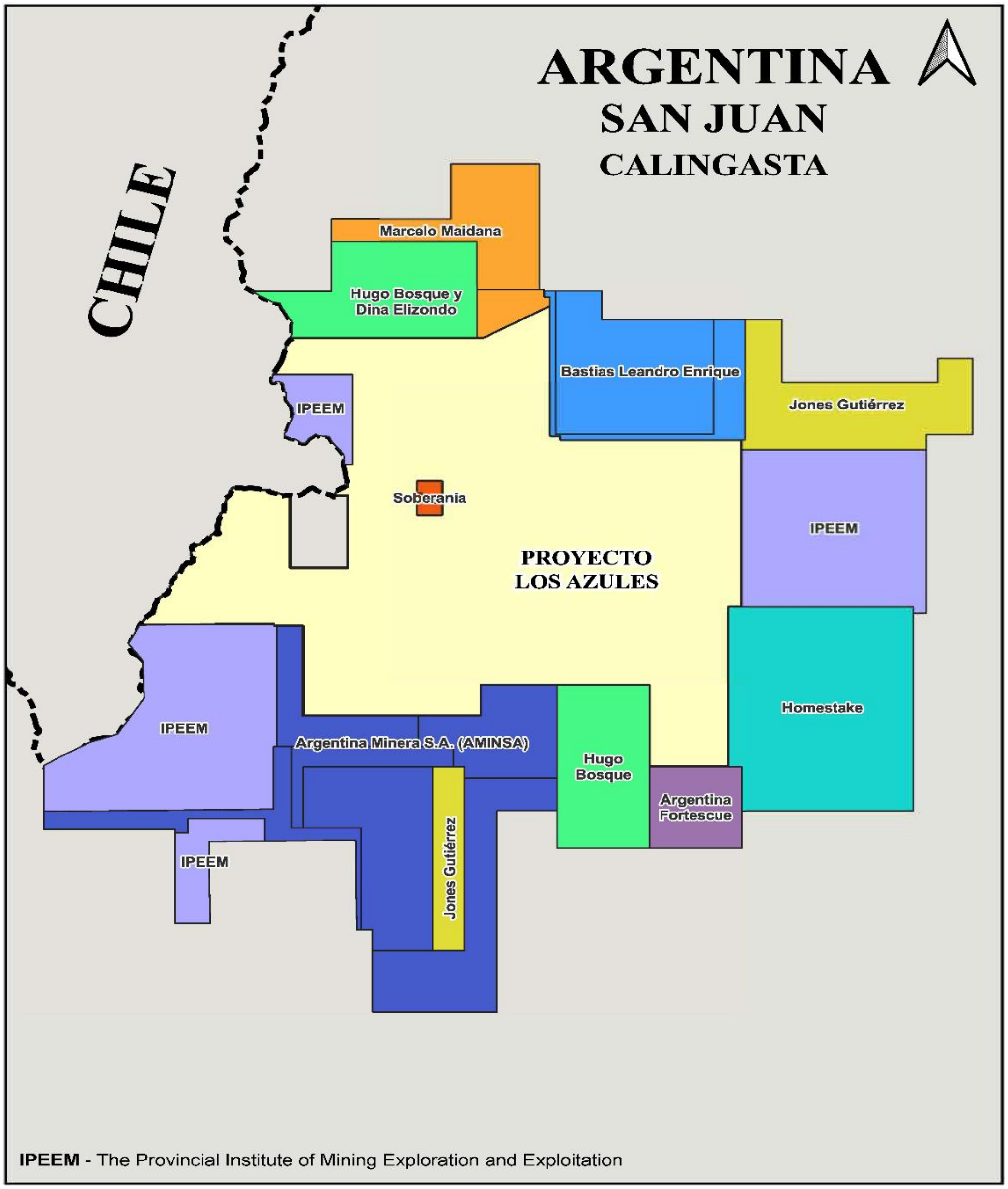


Figure 23.1: Regional Adjacent Properties

24.0 OTHER RELEVANT DATA AND INFORMATION

There is no additional information outside of that which has already been referenced and included within the report.

25.0 INTERPRETATION AND CONCLUSIONS

The Technical Report is prepared in accordance with the requirements set forth by Canadian National Instrument 43-101 (NI 43-101) for the required disclosure of material information and is intended to meet the requirements considered for a Preliminary Economic Assessment (PEA) level of study and disclosure as defined in the regulations and supporting reference documents.

Based on the results of this PEA, the contributing authors and QP's have identified important interpretations, conclusions, and recommendations to advance the Project. A complete description of these is provided in the following sub-sections. These include what is believed to be the most significant risks and opportunities to the future development of the Los Azules project.

25.1 OVERALL RISKS AND OPPORTUNITIES SUMMARY

25.1.1 Risks

- The Project is at the exploration stage of investigation; consequently, this study is at the scoping level of accuracy, preliminary in nature, and includes Inferred mineral resources in the conceptual mine plan and the mine production schedule. Inferred mineral resources are considered too speculative geologically and in other technical aspects to have the economic considerations applied to them that would enable them to be categorized as mineral reserves under the standards set forth in NI 43-101. Reserves would be required to establish a sound project basis.
- Significant additional investigation and work is required to improve the confidence level of the analysis to support a project development decision. There is no certainty that the results, project development plans or estimates in this PEA will be realized.
- Maintaining the necessary community and government engagement, planned project development work and investment plans in the Los Azules project are necessary to maintain the mining and surface rights described.
- Potential laws under consideration concerning the disturbance of vegas in Argentina is a significant risk if enacted prior to project permitting completion. The established permitting processes consider impacts and mitigations on a case-by-case basis within each Province, whereas a national law could restrict case by case and Provincial laws and processes.
- The requirement to avoid impacting localized rock glaciers poses a risk to longer term mining opportunities, including some of those in the Phase 2 options considered in this report. Site investigations to confirm the characterization of the known geomorphologic structures should be completed in continued field programs to appropriately evaluate them and determine if avoidance impact constraints should apply.
- Geologic modeling of the deposit rock types, lithologies and other aspects have been improved in the work described in this report. However, a more robust geologic modeling effort is required to support an adequate understanding of the deposit and next stage of study.
- Limited information is available on the geotechnical characteristics and hydrology/hydrogeologic conditions affecting the open pit design and pit slopes, leach pad foundation design, water resources and management, and other site facilities. These areas pose both a risk to the facilities considered in this document and areas for potential opportunity.
- Significant additional work is required to improve the metallurgical confidence level of the analysis to support a project development decision, given the preliminary nature of the metallurgical and

geo-metallurgical aspects of the deposit tested and analyzed thus far.

- The exact location of the project development surface facilities is yet to be finalized and requires hydrogeologic and geotechnical site investigations to support final location selections and engineering design work to be performed.
- Pricing and delivery estimates for equipment and materials reflect budgetary estimates considering current conditions and impacts related to inflation, geopolitical factors, and supply chain disruptions. While some of these impacts are easing, future impacts to market conditions are not considered in this analysis.
- Inflation and future impacts are not considered in this analysis due to volatility globally and in Argentina. Inflationary impacts may negatively impact the economic outcomes presented in this PEA.
- Metal price assumptions were considered based on current market conditions at the time of the report and pose both a risk and opportunity to future economic expectations.

25.1.2 Opportunities

- The resource is presently limited by the drilling completed at the time of this report and associated information developed to date. Resources with limited drilling information due to access in the areas under the vegas represent an opportunity to increase the near surface Indicated resource base within the current deposit. Additionally, opportunities for expansion of the resource base peripherally and at depth are apparent from the work completed. These should be investigated during the feasibility study drilling program.
- The Phase 1 open pit is presently constrained by the requirement to avoid impacting localized cryogenic geoforms. Site investigations to confirm these geomorphologic structures should be completed during field programs. A longer-term opportunity may exist to reclassify areas where no evidence of glacial activity is found.
- Within the cryogenic geoform constraints, limited information is available on the geotechnical characteristics and hydrogeologic conditions affecting the open pit design and pit slopes. Generalized technical parameters include a variable pit slope between 28° and 40° depending on depth. Additional work to better understand these key areas represents opportunities to reconsider the mine design parameters, potentially reducing stripping requirements and allow access to more of the deposit resources by extending and deepening the open pit.
- Materials inaccessible due to the surface glacier constraints should be considered in the context of alternative mining methods to the open pit.
- The project execution schedule assumes that the feasibility study work is completed as described, finalization of the IIA/DIA process and other necessary permits to begin work is completed during the proposed feasibility study and preliminary timeframe and financing is in place to achieve the schedule milestones. Should these happen earlier than planned, an earlier project start date could be considered.
- Pricing and delivery estimates for equipment and materials reflect current conditions and impacts related to inflation, geopolitical factors, and supply chain disruptions. If deliveries return to more typical durations, the current execution timeframe may be improved by 6 to 12 months.

25.2 UPSIDE POTENTIALS

Potential scenarios for future operations beyond the initial phase of the project considering the primary copper sulfide materials were developed. Two approaches were considered, one employing Nuton™ bio-leaching technology and secondly, a conventional copper concentrator is presented and discussed below.

25.2.1 Nuton™ Technology Opportunity

Nuton LLC is a technology venture of Rio Tinto, one of the world's largest mining companies, and home to a unique integration of innovative nature-based technologies, expertise and capabilities. At the core of Nuton is a portfolio of proprietary copper heap leach related technologies. Nuton aims to advance the environmental, social and governance performance of the industry whilst delivering copper growth.

As the mine progresses, there is an opportunity to process the Primary material on a heap leach pad, utilizing the Nuton™ technology. Ongoing metallurgical work considering new Nuton™ bio-leaching technological approaches is being developed to potentially replace the need for a future milling operation in favor of continued leaching and copper cathode production for the life of the mining operations. Potential scenarios for the future operations employing the Nuton™ bio-leaching technology are presented and discussed below.

The information contained within section 25.2.1 below was written with input from the Nuton team. As such, any recovery information detailed in this sub-section is with reference to Total Copper (CuT) recovery.

Although Nuton has completed larger scale testing at several global project sites and has developed proprietary modeling techniques to predict results, there are no commercial applications of the Nuton™ technology operating at the time of this report. Based on preliminary small-scale testing and modeling, the technology provides the opportunity to extend the mine life to more than 50 years in some instances while adding significant additional value. A significant testing program, including site-based scale-up test work, will be required to verify these preliminary estimates; therefore, these results are not considered suitable for inclusion at this time in the initial project cases presented and only included as a demonstration of the potential future opportunity.

Based on preliminary scoping testing, the Nuton™ technology offers the potential for copper recoveries of over 80% from predominantly chalcopyrite materials, depending on the specific mineralogy make-up of the deposit. At Los Azules Nuton™ has the potential to economically process the large primary sulfide copper resource as an alternative to a concentrator, with low incremental capital following the oxide leach, no tailings requirement, and a smaller environmental footprint. Producing a copper cathode from Nuton™ on-site also has the advantage of simplifying outbound logistics for copper concentrates and offers a finished product to the domestic and regional market.

Nuton has developed several proprietary models to simulate the performance of Nuton™ technologies both in column leach tests and in large-scale operations. These models have been developed over the past 25 or more years and have been validated at various scales.

Current Nuton practice is to further validate modelled data with actual column leach tests. Stage 1 column leach testing of the composite samples at Bundoora, as well as of the primary bulk samples at Hazen, is underway and expected to be completed during Q1 2024. Validation of the modelled results should, however, be possible much sooner, depending on the trends provided by the actual column leach curves.

The unoptimized outcomes for the four composites at commercial scale and under standard Nuton operating conditions, as predicted by the Nuton Computational Fluid Dynamics (CFD) model, are shown in Table 25.1.

Table 25.1: CFD Modelled Outcomes for Bundoora Lab Samples				
Simulation Sample	**Cu Grade** (%)	**Cu deported to Chalcopyrite** (%)	**CuT Recovery*** (%)	**Acid Consumption** (kg/t)
Comp 6 (+ Nuton™ additives)	0.44%	21%	80%	12
Comp 7 (+ Nuton™ additives)	0.23%	94%	79%	10
Comp 8 (+ Nuton™ additives)	0.33%	72%	73%	14
Comp 9 (+ Nuton™ additives)	0.55%	45%	74%	13
* Scale-up factor applied, but not optimized				

Typical leach recovery curves for Composites 6 to 9 for a period of 10 years are shown in Figure 25.1. It should be noted that the recoveries are not optimized yet, with further test data expected to provide opportunities for optimization, both with regards to copper recovery and reagent consumption.

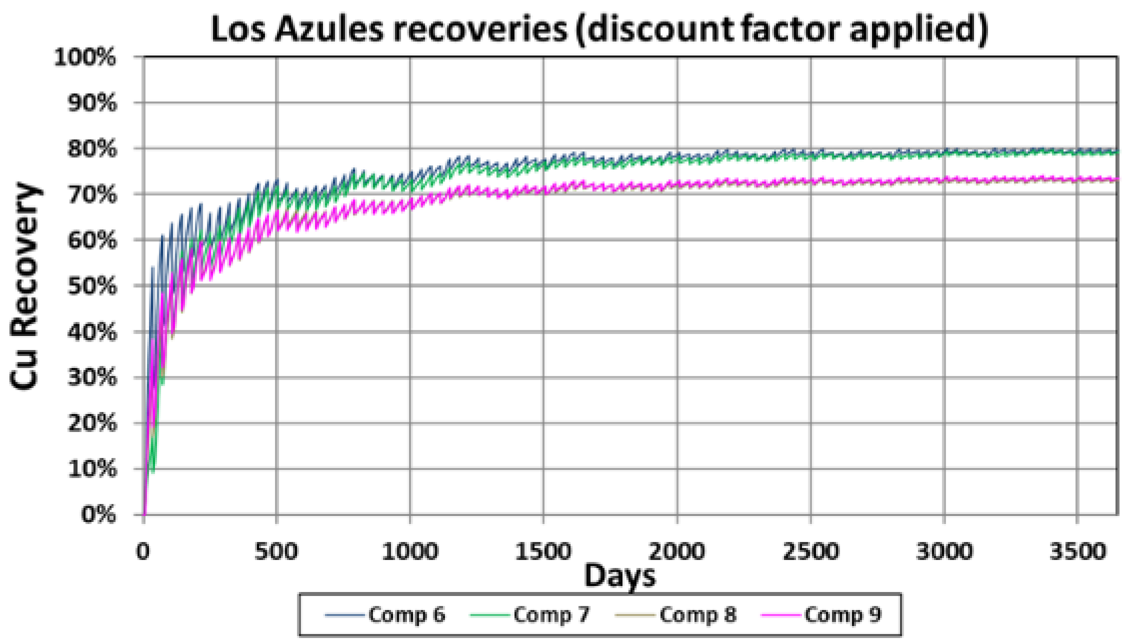


Figure 25.1: Composite Samples - Modelled Total Copper Recovery Curve

The CFD model was also applied to model the commercial scale outcomes for the four bulk samples representing the diorite zone in the deposit, which contributes to about 50% (wt.) of the mineralized zone. The unoptimized outcomes for the five primary process material types, operated under standard Nuton™ conditions at a reference commercial scale, again for a period of 10 years, are shown in Table 25.2.

Table 25.2: CFD Modelled Outcomes for Hazen Lab Samples				
Simulation Sample	**Cu Grade** (%)	**Cu deported to Chalcopyrite** (%)	**CuT Recovery*** (%)	**Acid Consumption** (kg/t)
AZ22138 (Nuton™ additives)	0.37%	71%	78	12.1
AZ22140 (Nuton™ additive)	0.14%	23%	85	5.0
AZ22149 (Nuton™ additives)	0.43%	12%	86	12.3
AZ22150 (Nuton™ additives)	0.15%	74%	77	16.4
* Scale-up factor applied, but process parameters not optimized				

Typical leach curves generated by the CFD model for the primary bulk samples over the period of 10 years are shown in the below figure.

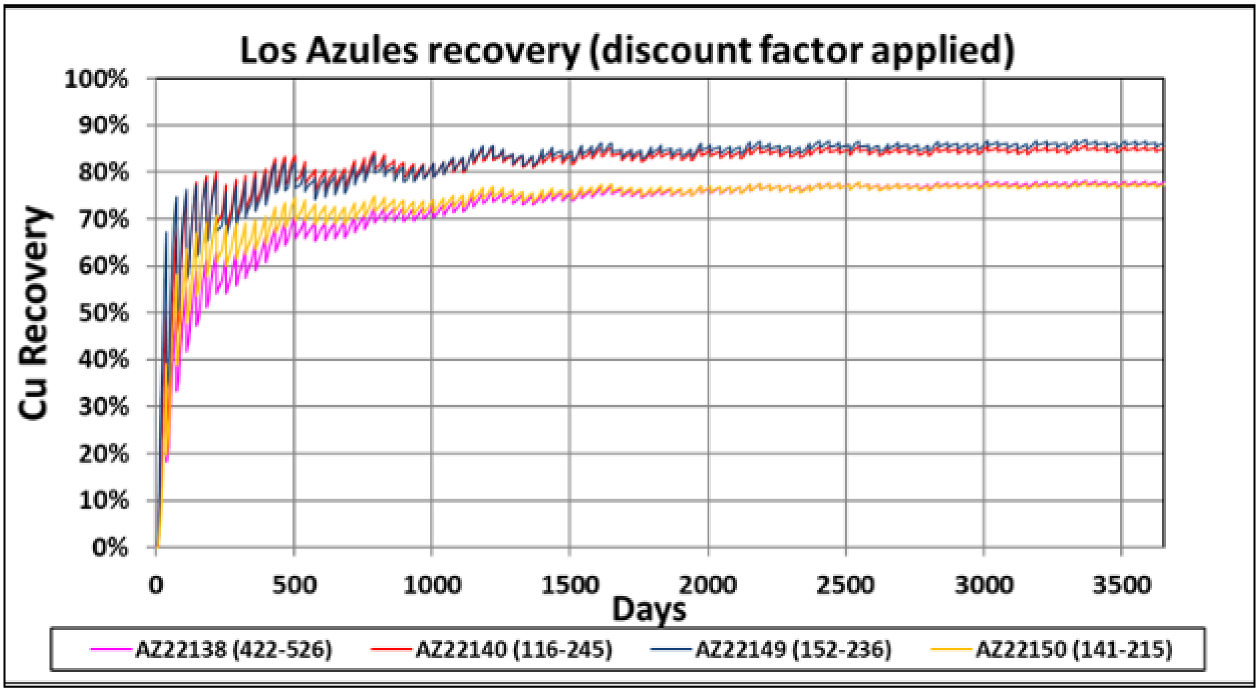


Figure 25.2: Primary Bulk Samples - Modelled Total Copper Recovery Curves

The modelled outcomes, using the Nuton proprietary CFD model, are very encouraging and indicate that unoptimized copper recovery to cathode from primary material should range from 73% to 79%. Furthermore, Nuton recovery of secondary material is high, ranging from 80% to 86%. This provides a significant opportunity to de-risk the mine plan and the need for selective mining, as simultaneous stacking of both secondary and primary mineralization material will not impact on the copper recovery from either material type. Based on the current resource estimate, this could have a significant positive impact on the expected life of mine, without significantly increasing the initial capital investment required.

25.2.1.1 Nuton Economic Opportunity

Based on the information and recommendations from the Nuton team, preliminary cases for the potential economic opportunity represented by adding this leaching approach were developed. The Nuton™ Economic Opportunity for the Project's LOM cash flow results are summarized in Table 25.5: Nuton™ Opportunity Economic Summaries.

Table 25.3 summarizes the incremental capital cost for a 35M tpa processing rate over the life of the mine to implement the Nuton™ technology as provided by Nuton.

Table 25.3: Nuton Opportunity Capital Cost Summary for 35Mtpa Case

Description	LOM Cost ($000s)
NUTON™ ON-SITE FACILITIES	
Nuton Additives Make-up Facility	8,000
Biomass Growth Facility	20,000
Raffinate Conditioning	30,000
Raffinate Conditioning Residue Storage	60,000
Additional Aeration Infrastructure (every 3 years)	16,000

The Nuton™ process operating costs provided by Nuton are summarized in Table 25.4 for life-of-mine (LOM) values.

Table 25.4: Nuton Opportunity Operating Cost Summary for 35Mpta case

	OPEX $/t Placed
Leaching Augmentation Additives	$0.10
Biomass Growth Facility	$0.05
Raffinate Conditioning Costs	$0.77
Sulfur / Pyrite Addition	$0.06
Total Additional OPEX	**$0.98**

To apply the Nuton copper recovery estimates given the mixed nature of some parts of the deposit, SE considered the chalcopyrite recovery achieved and assumed a similar result for the residual component of the sequential assaying methodology. The SE averaged estimate for this was 78% extraction of copper associated with the residual assay component. The extraction for the soluble copper component was assumed to be the same as the conventional bio-leach at 100% of copper associated with the soluble copper assay component. The expected extraction was also assumed to be slightly quicker to achieve versus the conventional bio-leach and the two-year distribution improved to 75% of the expected long-term copper extracted in year 1 of placement and 25% in year two.

Nuton provides an opportunity to leach both primary and secondary copper sulfides, providing significant opportunity to optimize the mine plan and the overall mining and processing operations. In addition, Nuton provides significant other benefits, such as lower overall energy consumption, allowing earlier conversion to renewable energy sources, lower water consumption than conventional sulfide material treatment processes, as well the potential to restore and reclaim mine sites by reprocessing mined waste.

To assess the strategic potential for inclusion of the Nuton™ technology, Whittle Consulting Pty Ltd (WCPL) undertook analysis on the Los Azules project in Argentina in support of the PEA study. WCPL were asked to evaluate the economic feasibility of alternative processing options using their proprietary mining optimization software tools and methods to assess. Three cases were analyzed which included adding the Nuton™ technology to both the Base Case and Alternative Case options and a hybrid 3rd Case allowing for expansion of the copper production from the 125 tpa rate to 175 tpa when significant primary copper mineralization was mined and the Nuton™ technology would be best deployed.

WCPL undertook pit optimisation of the deposit outside of the PEA pit, consistent with the Stantec geotechnical recommendations, for strategic evaluation and developed a series of shells for a potentially larger pit (4.23 billion rock tonnes) that includes the minable primary copper materials not processed in the initial project development options. The March 2023 block model was used, along with the PEA pit shells produced by Stantec.

Nuton Cases – Whittle Optimization

As an initial case, one scenario leverages the PEA Base Case 175k tpa Heap Leach (HL) / Solvent Extraction (SX)/ Electrowinning (EW) Plant operating for four years, before introducing Nuton™ leaching technology from year five onwards.

Key physical metrics for this case can be seen in Figure 25.3.

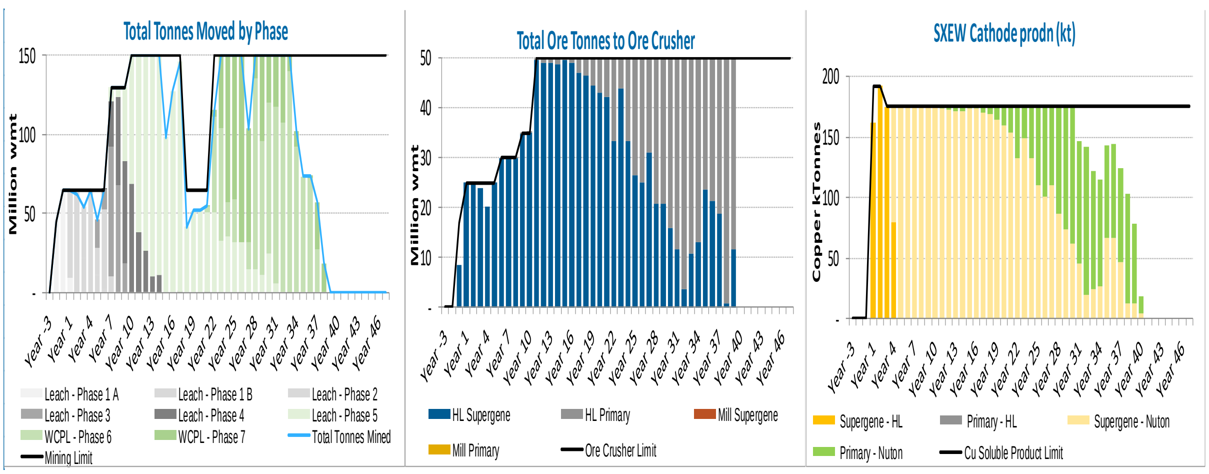


Figure 25.3: 175ktpa Heap leach to Nuton transition case

Mining capacity starts at 65 Mtpa and ramps to 150 Mtpa year ten, with the crusher progressively stepping up from an initial 25 Mtpa to 50 Mtpa to keep the larger SX EW filled as grades decrease over time. This schedule focussed on keeping the Crusher/Mill circuit capacity sufficient to keep the SX/EW utilised fully to maximise economic value through the system.

An observed outcome of this mining schedule again was a preference to process Supergene material, using stockpiling and mining schedule to bring forward higher grade material whenever possible. The schedule processes 4.23 billion tonnes of rock and feeds 1.7 billion tonnes of material containing 7.1 million tonnes of copper of which 6.41 million tonnes of copper is recovered (89.9%), life of mine – owing to slightly lower overall operating costs per tonne and shorter mine life for the higher production rates.

Over the life of operations of 40 years, mining OPEX averaged US$1.82/t mined and Heap leach/Nuton OPEX averaged US$3.01/t processed. Overall operating costs averaged $1.02/lb of copper.

A second scenario leverages the PEA Alternative Case125k tpa Heap Leach (HL) / Solvent Extraction (SX)/ Electrowinning (EW) Plant operating for ten years, before introducing Nuton™ leaching technology from year eleven onwards.

Key physical metrics for this case can be seen in Figure 25.4.

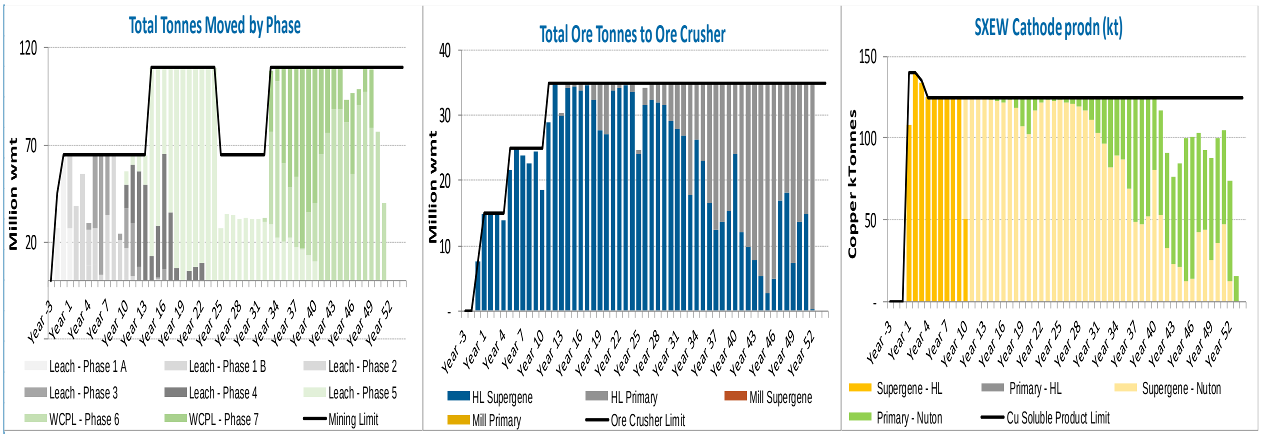


Figure 25.4: 125ktpa Heap leach to Nuton transition case

Mining capacity starts at 65 Mtpa and ramps to 110 Mtpa as Nuton™ leaching technology comes online in year seven, with the crusher progressively stepping up from an initial 15 Mtpa to 25 Mtpa to 35 Mtpa to keep the SX/EW filled as grades decrease over time. This schedule focussed on keeping the Crusher/Mill circuit capacity sufficient to keep the SX/EW utilised fully to maximise economic value through the system.

An observed outcome of this schedule was a preference to process Supergene material, using stockpiling and mining schedule to bring forward higher grade material whenever possible. Supergene material has higher soluble copper grades and thus has higher copper recovery in a conventional leaching than primary material. Later in the LOM, primary copper material is fed onto the leach using the Nuton assumptions which augments the copper recovery for this material in leaching.

The schedule mines 4.23 billion tonnes of rock and feeds 1.65 billion tonnes of material containing 6.94 million tonnes of copper of which 6.16 million tonnes of copper is recovered (88.7%), life of mine.

Over the life of operations of 53 years, mining OPEX averaged US$2.08/t mined and Heap leach/Nuton OPEX averaged US$3.16/t processed. Overall operating costs averaged $1.18/lb of copper.

After allowing an estimated US$134 million capex for the Nuton conversion and US$16 million every three years thereafter for aeration costs, the economic potential was evaluated using the information provided by Whittle.

A third scenario (3rd Case) considered leverages the PEA Alternative Case 125ktpa Heap Leach (HL) / Solvent Extraction (SX)/ Electrowinning (EW) Plant operating for six years, before introducing Nuton™ leaching technology from year seven onwards whilst expanding the SX/EW facilities to 175ktpa utilising the increased copper recovery from Nuton.

Key physical metrics for this case can be seen in Figure 25.5.

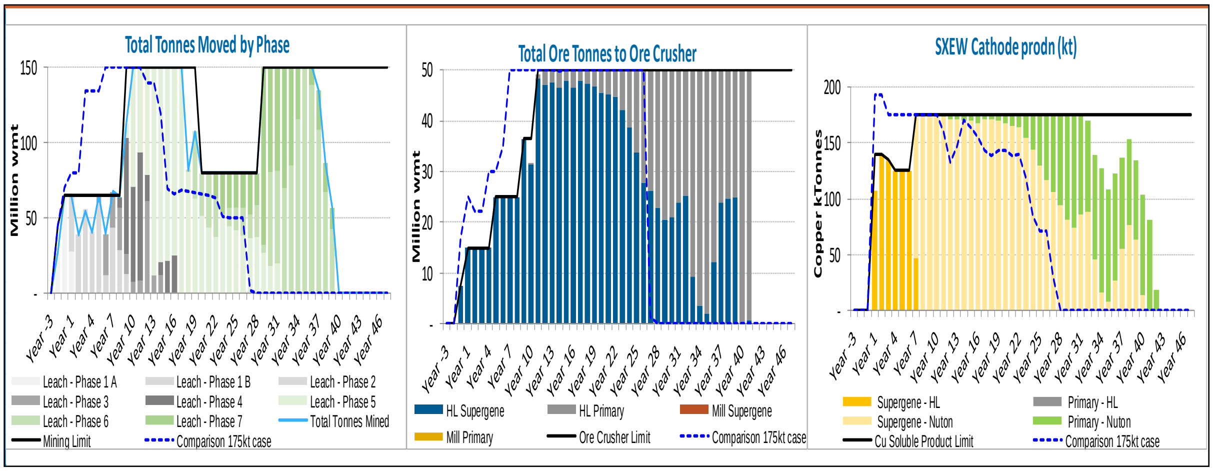


Figure 25.5: 125ktpa Heap leach to 175ktpa Nuton transition case

Mining capacity starts at 65 Mtpa and steps to 150 Mtpa in year seven, with the crusher progressively stepping up from an initial 15 Mtpa to 50 Mtpa to keep the larger SX EW filled as grades decrease over time. This schedule focussed on keeping the Crusher circuit capacity sufficient to keep the SX/EW utilised fully to maximise economic value through the system.

An observed outcome of this schedule was a preference to process Supergene material, using stockpiling and mining schedule to bring forward higher grade material whenever possible. Supergene material has higher soluble copper grades and thus recovers better on a leach than primary material. Later in the LOM, primary material is fed onto the leach.

The schedule processes 4.23 billion tonnes of rock and feeds 1.78 billion tonnes of material containing 7.19 million tonnes of copper of which 6.46 million tonnes of copper is recovered (89.5%).

Over the life of operations of 42 years, mining OPEX averaged US$1.86/t mined and Heap leach/Nuton OPEX averaged US$2.93/t processed. Overall operating costs averaged $1.03/lb of copper.

To assess and more directly compare the indicative economic potential for the Whittle options developed with the project cases, the Whittle mining information was incorporated into the SE processing and financial models for each option. A summary of the economic cases developed based on the three Whittle mine optimization runs is present Table 25.5 below.

Table 25.5: Nuton™ Opportunity Economic Summaries

Project Metric	Units	Base Case-Nuton 175k tpa Cu	Alt. Case-Nuton 125k tpa Cu	3rd Case-Nuton 125k/175k tpa Cu
Mine Life	Yr	39	52	41
Strip Ratio		1.43	1.56	1.37
Tonnes Processed	ktonnes	1,737	1,651	1,784
Copper Grade (Total)	% Cu	0.409	0.420	0.403
Copper Production – cathode Cu	ktonnes	6,411	6,165	6,461
Initial Capital Cost	USD Millions	$2,444	$2,203	$2,203
Sustaining Capital Cost	USD Millions	$2,793	$2,789	$2,793
C1 Costs (Life of Mine)	USD/lb Cu	$1.04	$1.20	$1.05
All-in Sustaining Costs (AISC)	USD/lb Cu	$1.54	$1.70	$1.55
After Taxes				

Table 25.5: Nuton™ Opportunity Economic Summaries				
Internal Rate of Return (IRR)	%	23.9%	19.2%	21.1%
Net Present Value (NPV) @ 8%	USD Millions	$3,701	$2,306	$3,214
Pay Back Period	Yr	2.7	3.5	3.5

25.2.2 Copper Concentrator Opportunity

The future Phase 2 project modification anticipates processing materials with predominantly primary copper mineralization and considers two potential scenarios for operations beyond Phase 1 of the project, a preferred option employing Nuton™ bio-leaching technology and secondly, a conventional copper concentrator and tailings storage facility that produces a copper concentrate as the final product for export. A conventional mill and flotation/concentrator option was considered to process primary copper mineralization to demonstrate economic viability employing conventional methods and support reserves estimation confidence.

The 2017 Hatch PEA included a copper concentrator. This PEA study includes Primary material with low soluble copper grades to be stockpiled. There is an opportunity that as the mine progresses, a copper concentrator could be built on site to process the Primary material through a mill and concentrator to produce a copper concentrate.

The option leverages the smaller Alternative Case option to avoid oversizing the hydrometallurgical processing facilities, to process primary copper material. The milling facilities are brought online in Year 7 at a processing rate of 120,000 tonnes per day through completion of the project in Year 41. Crushing and stacking systems initially commissioned for use in the heap leaching process could be repurposed to provide ball mill feed and enable filtered tailings transport/storage options. Tailings storage management design would provide for a lined facility with filtered tailings (dry stacked) deposition for the applicable life of mine operations to minimize environmental impacts and freshwater usage.

Stantec was retained to develop a Preliminary Economic Assessment (PEA) for the Los Azules Filtered Tailings Storage Facility (FTSF) located in the San Juan Province of Argentina. This section presents the conceptual design associated with the FTSF. The FTSF footprint is established based on the property limits of the Los Azules site and the surface mining limits identified. Alternatives within the property limits and outside the surface mining limits were identified in the alternatives evaluation and potentially provide additional capacity to the facility; however, for the purposes of the PEA document, the entire facility considered is within the surface property and mining rights of the project.

The previous heap leach project will continue to drain down and go into closure over time. All mined Supergene material and Primary material will be fed directly to the copper concentrator or be stacked directly on the pad when the concentrator is brought online depending on profitability. The conceptual block flow diagram for the processing facilities for the copper concentrator is presented in Figure 25.6. The Los Azules copper concentrator is to be a conventional copper flotation circuit with a daily throughput of 120,000 tpd, or 43.8M tpa at an operating availability of 85% or 7,353 operating hours.

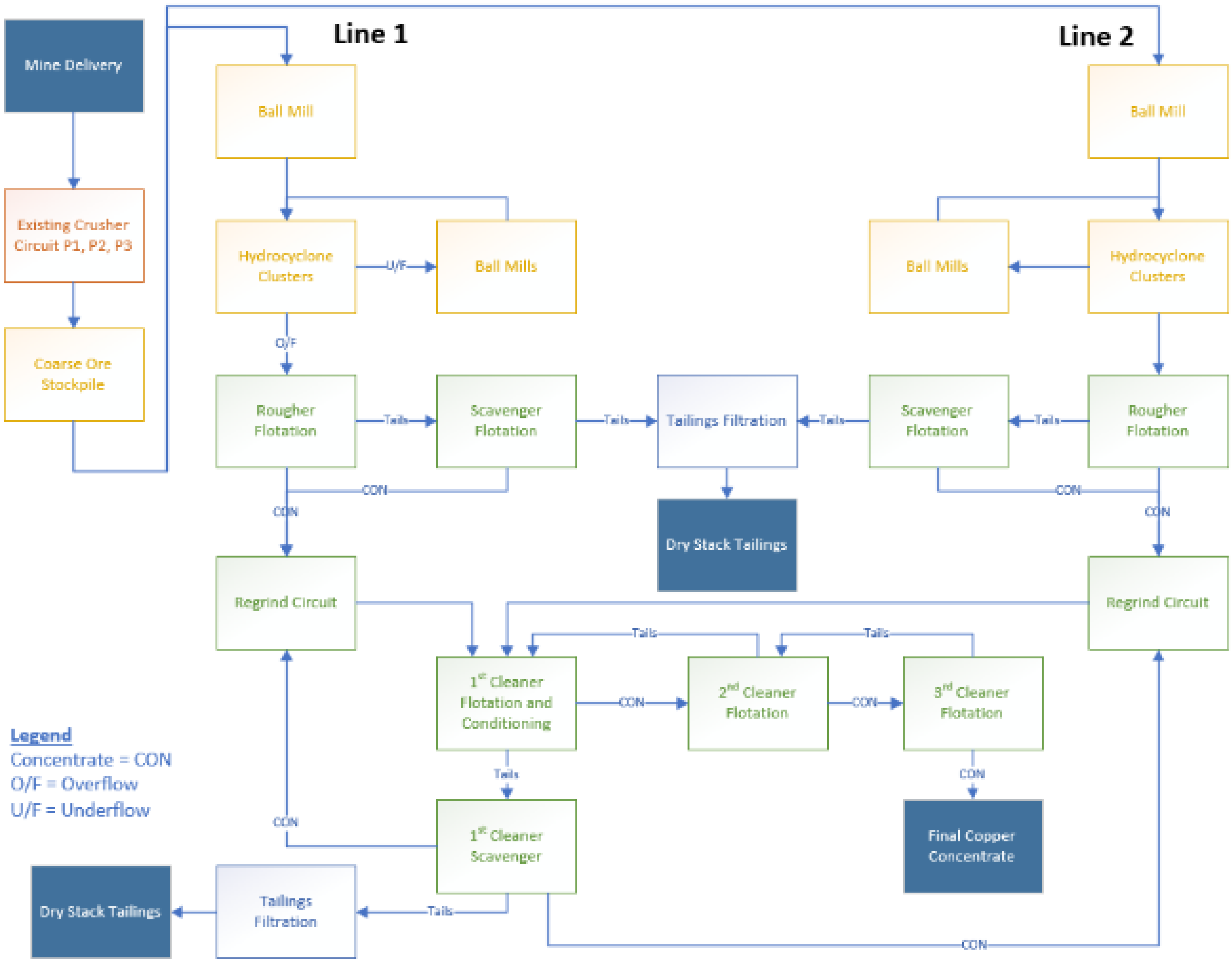


Figure 25.6: Copper Concentrator Process Flowsheet

The final copper concentrator will produce grades and total recovery as prescribed in Table 25.6, matching lock-cycle testing from Section 13.2.3.

Table 25.6: Optimized Lock-cycle Flotation Results							
Composite	**Concentrate Wt. % Of Feed**	**Copper Concentrate Grade**			**Recovery %**		
		Ag g/t	**Au g/t**	**Cu %**	**Ag**	**Au**	**Cu**
Primary	**1.45**	97	3.79	**31.96**	68.8	62.9	**93.2**
Supergene	**2.1**	28.6	3.58	**28.53**	54.0	65.6	**89.3**

25.2.2.1 Copper Concentrator Economic Opportunity

Initial project capital costs are consistent with the project Alternative Case 125 ktpa Cu leach-SX/EW project. Table 25.7 summarizes the initial capital cost for the addition of a 120,000 tpd copper mill/concentrator. Much of the costs shown were developed considering a 50% overall escalation from the costs expressed in the 2017 PEA for this evaluation.

Crushing and stacking systems initially commissioned for use in the heap leaching process will be repurposed to provide ball mill feed and enable filtered tailings transport/storage options. Tailings storage management design would provide for a lined facility with filtered tailings (dry stacked) deposition for the applicable life of mine operations to minimize environmental impacts and freshwater usage. Tailings storage facility expansions and lining for filtered tails was included at $0.50/tonne placed over the life of the milling operations.

Table 25.7: Copper Concentrator Opportunity Capital Cost Summary		
AREA		**Total**
Incoming Powerline Upgrade	USD	179,000,000
Utilities and Power	USD	52,000,000
Construction Camp/permanent camp	USD	20,000,000
SUBTOTAL Direct Cost Fixed Items	**USD**	**251,000,000**
Crushing	USD	Existing
Milling	USD	607,400,000
Tailings Storage Facility – Initial Area	USD	187,600,000
Dry Stack Conveying & Stacking	USD	120,500,000
SUBTOTAL Direct Cost Variable Items	**USD**	**915,600,000**
Total Direct Costs	**USD**	**1,166,600,000**
TOTAL INDIRECT COST	**USD**	**648,600,000**
Contingency	USD	453,800,000
Subtotal Indirect Owner & Contingency	**USD**	**1,102,400,000**
Total Project Cost Excluding Mining Equipment	**USD**	**2,268,900,000**

Heap Leach expanding to Mill Flotation

One scenario leverages the Alternative Case 125ktpa Heap Leach / Solvent Extraction/ Electrowinning Plant operating for six years, before introducing a four-ball mill circuit feeding a flotation plant and producing copper concentrate from year seven onwards.

Key physical metrics for this case can be seen in Figure 25.7.

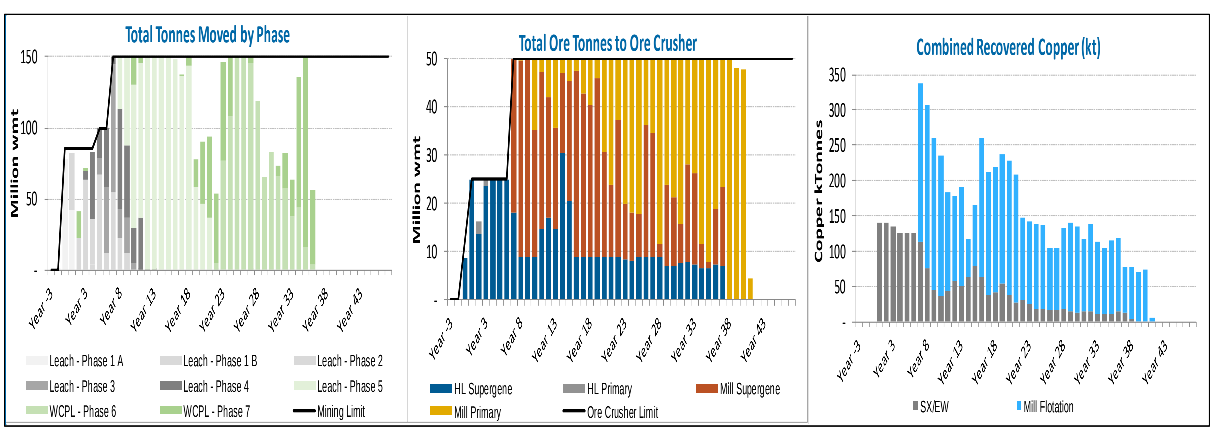


Figure 25.7: Key physicals from the Heap Leach to Mill case

Mining capacity starts at 85 Mtpa and ramps to 150 Mtpa as the Mill comes online in year seven, which is the same timing as the crusher capacity ramps from an initial 25 Mtpa to 50 Mtpa. This schedule focused on keeping the Crusher/Mill circuit utilized fully to maximize economic value through the system.

An observed outcome of this schedule was to continue Heap Leach operations of around 10 Mtpa after the mill starts, to process lower grade Supergene material as the unit cost to Heap Leach is lower than Mill-Flotation. The Mill processes higher grade Supergene and Primary material types at a finer grind of 200µm with a throughput of ~40 Mtpa. When the Heap Leach runs out of low-grade Supergene and closes, the Mill moves to a coarser grind and processes at 50 Mtpa.

The schedule processes 4.23 billion tonnes of rock and feeds 1.85 billion tonnes of material containing 7.29 million tonnes of copper of which 6.32 million tonnes of copper is recovered (86.8%) with 6,415 tonnes payable after concentrate deductions.

Over the life of operations of 41 years, mining OPEX averaged USD $1.80/t mined, a summary of the operating costs for the leach and milling option are shown in Table 25.8 ($/t as tonne processed).

Table 25.8: Life of Mine Leach/Mill OPEX ($/t processed)			
Mining OPEX	LOM	$ Millions	$7,798
	Annual Average	$ Millions	$190
	Per ton processed	$/t	$3.61
	Per Eq. Lb Cu	$/lb Cu	$0.55
Processing OPEX	LOM	$ Millions	$8,626
	Annual Average	$ Millions	$210
	Per ton processed	$/t	$4.00
	Per Eq. Lb Cu	$/lb Cu	$0.61
SG&A	LOM	$ Millions	$1,616
	Annual Average	$ Millions	$39
	Per ton processed	$/t	$0.75
	Per Eq. Lb Cu	$/lb Cu	$0.11
TOTAL OPEX	LOM	$ Millions	$18,040
	Annual Average	$ Millions	$440
	Per ton processed	$/t	$8.36
	Per Eq. Lb Cu	$/lb Cu	$1.28

After allowing an estimated US$2.3 billion capex in years 4-6 for the Mill/Flotation circuit and initial tailings storage facility, in addition to ongoing leach SX/EW costs, an economic analysis was completed, and summary results are presented in Table 25.9 below. Further work needs to be undertaken at a PEA level to determine the viability of this option.

Table 25.9: Copper Concentrator Opportunity Economic Summary

Project Metric	Units	Base Case-Mill 125k tpa Cu
Mine Life	Yr	41
Tonnes Processed	Ktonnes	2,167
Strip Ratio		1.10
Copper Production – cathodes	ktonnes	1,861
Copper Production - concentrate	ktonnes	4,554
Gold Production	Moz	1.11
Silver Production	Moz	34.2
Initial Capital Cost	USD Millions	$2,182
Sustaining Capital Cost	USD Millions	$4,686
C1 Costs (Life of Mine)	USD/lb Cu	$1.63
All-in Sustaining Costs (AISC)	USD/lb Cu	$2.25
Before Taxes		
Internal Rate of Return (IRR)	%	23.5%
Net Present Value (NPV) @ 8%	USD Millions	$4,528
After Taxes		
Internal Rate of Return (IRR)	%	19.2%
Net Present Value (NPV) @ 8%	USD Millions	$2,685
Pay Back Period*	Yr	6.8

*Note: Mill investment in Years 4-6 extends Pay Back Period

25.3 METALLURGY AND MINERAL PROCESSING

25.3.1 INTERPRETATIONS AND CONCLUSIONS

While reviewing and interpreting the prior and current test work and developing the process design for the Project, the author has developed the following conclusions:

- Extensive metallurgical testing (batch) has been conducted on representative samples from the Resource indicating that the soluble copper is able to be recovered by conventional heap leaching technology with the addition of biomass. Previous Plenge data provided an above 100% total soluble copper recovery, which is supported by the on-going SGS test work with over 120 days of leaching. The current test work on-going at SGS backs this data up as presented in Section 13.0.
- On-going SGS test work is only preliminary and subject to changes when final tail assays are completed, and a calculated head grade is determined.
- Extensive metallurgical testing (batch) has been conducted on representative samples (material types and major lithologies) from the Resource indicating the Primary and Supergene material is recoverable to a rougher flotation concentrate as presented in Table 13.28 and Table 13.29.
- Gross acid consumption has been set at 18 kg/t material based on Plenge data and backed up by the current bottle rolls completed at SGS.
- The plant crushing/comminution circuits were designed based on 70+ samples from variability

testing. This provides enough basis for hardness calculations on the plant through the variability of time.

- Multiple assays have not been completed or integrated into the Resource model. This includes the incorporation of arsenic and other deleterious materials.
- Current projections in LeapFrog and the Resource model have Pre-Mineral Pluton without breaking up Diorite and Porphyritic Diorite lithologies. Current flotation results show Diorite minerals respond poorly, when compared to Porphyritic Diorite.
- Copper mineralization is complex and varied at Los Azules, as to be expected for deposits of this type. Metallurgical characterization testing has been completed as part of this study in the form of sequential assay (sulfuric acid and cyanide steps) for the resources considered. There are several sequential assay methods in use in the industry, the methodology selected for Los Azules maintains consistency with historic assay methods with the resource data set. The sequential assay method used at Los Azules for both the resource assay and metallurgical programs provides an indication of the copper mineralization present in the form of acid soluble copper (CuAS) and cyanide soluble copper (CuCN), both assays combined provide an approximation for leachable/soluble copper (CuSOL) component of the total copper assay (CuT). Modifications to the sequential assay method could be considered in future to better approximate soluble copper leaching recovery for process control.

25.3.2 Risks

- Initial metallurgical test work to date on the supergene and primary mineralized material has not included a full variability program. The range of metallurgical performance is therefore not completely defined and average performance expectations used in this report may not be achieved.
- Phase 1 Metallurgical columns were only conducted in 3 m height columns, the heap leach pad will have 9 m lifts. This poses a risk that the leach kinetics will slow over time.
- Of significance is the level of potassium in the gangue minerals. With dissolution this could likely drive iron precipitation as potassium jarosite particularly at elevated temperatures. Additionally, significant biotite and chlorite levels are also observed in some composites, which can lead to higher acid consumption, particularly at higher leaching temperatures and lower pH in the leach solutions. Continued analysis and monitoring of these features is planned in the future test work.
- The metallurgical test work program at SGS is still in progress for flotation concentrate for deleterious elements that will appear in the concentrate; this includes As, Bi, Cd, Ga, In, Pb, Re, Sb, Se, Te, Ti, U, and Hg. Concentrate samples were shipped to SGS Lakefield in November 2022 for testing to help characterize and minimize the risk.
- While test work is still in progress on rougher flotation samples at SGS, no previous test work was utilized to characterize the tailings. This poses a risk to closure and reclamation of the dry stack tailings, if applicable. Early in the next phase of the Project, study, and testing of the materials to help the optimization of the proposed dry stack will help minimize the risk.

25.3.3 Opportunities

- The opportunity to process primary sulfides directly through a heap leach rather than building a traditional copper concentrator in the future is the envisioned approach to the Los Azules development plan. Emerging technologies for improved leaching of sulfide copper ores are being developed, a proprietary catalytic bio-heap leaching technology that may provide an alternative approach to improving the leach performance of primary sulfide content in the leach materials considered in this report. The primary sulfides are presently not considered economically suitable for commercial heap leaching operation. Nuton™ technology is currently being evaluated in this capacity. This would also have the potential to unlock the primary copper resources more economically with less environmental impact versus a mill/concentrator alternative and negate the need for a tailing storage facility. This work is currently in progress in Melbourne, Australia and at Hazen in Golden, Colorado.

- Previous analysis of the hardness and comminution parameters showed a correlation between the on-going geotechnical work with Schmidt Hammer and point load testing (PLT). An estimated 40 to 50 additional samples would need to have hardness and comminution parameters completed on them that also have Schmidt Hammer and PLT to provide a correlation of data. This has an opportunity to be applied directly to the geometallurgical block model for applications such as abrasion index, bond work index, and hardness to name a few.
- Opportunity to provide thermal blanketing on top of the heap leach pad, especially during winter months to protect drip emitters of raffinate and the inoculated biomass. This may also mean digging trenches or burying the drip emitters.
- Direct inclusion of sulfur with the leaching material going to the leach pad to support acid consumption requirements, and limit acid plant expansions is being investigated.
- Further investigation of coarse particle size flotation could reduce primary grinding costs.
- Newer, more efficient, copper flotation collectors, such as emulsions, are being developed and should be included in future testing programs to assess their value.
- Opportunity to process copper concentrates at the Project site utilizing the existing SX/EW facility instead of shipping the concentrate. This would produce copper cathodes at site.
- The process plant and equipment layout has been designed to take advantage of the existing terrain of the project site utilizing gravity flow of fluids where possible. The opportunity exists to optimize the plant layout to take advantage of gravity flow between unit operations, which could potentially remove some pumps from the process reducing capital and operating costs of the project.
- Incorporation of developing leaching technologies has the potential to improve copper recovery, reduce leaching times and minimize acid consumption requirements.
- Strategies for recovery of contained gold may add value. Preliminary leaching tests indicate that a significant amount gold is recoverable with conventional technology. Recovery techniques should consider separate processing of higher-grade gold bearing materials considering pre and post copper leaching extraction methods.
- Additional testing of microwave technology will be beneficial to understanding micro-fracturing of material, power consumption of crushers, and acid consumption.
- Primary copper mineralization resources to be mined in future can be processed by conventional mill/concentrator methods to produce copper concentrates for sale if a suitable heap leaching technology cannot be developed.

25.4 SAMPLE PREPARATION, ANALYSES, AND SECURITY

25.4.1 INTERPRETATIONS AND CONCLUSIONS

Results from the control sample analysis indicate that the copper and gold assay processes are under sufficient control to produce reliable sample assay data for resource estimation and release of drill hole assay results. Inadequate standards from early field seasons were eliminated. The use of only one lab to produce assay results improves the consistency of results. Material that was assumed to be blank but contained low copper values was replaced. Later types of blank material improved the monitoring of potential contamination of the samples.

All past deficiencies in the QC program have been addressed. The Los Azules sampling and assaying program appears to be producing sample information that meets industry standards for copper and gold accuracy and reliability. The assay results are sufficiently accurate and precise for use in resource estimation and the release of drill hole results on a hole-by-hole basis.

25.5 MINERAL RESOURCE ESTIMATES

25.5.1 INTERPRETATIONS AND CONCLUSIONS

The construction methodology of the geological models is robust. It breaks the deposit down into its component events and by understanding each of the controls related to that event, yields a greater understanding of the deposit and a more robust series of inter-related models. The modelling was carried out in Leapfrog software and was influenced by structure – lithology -alteration – mineralization – zonation with iterative revision and reconstruction.

Overall, modelling shows that Los Azules is a large structurally controlled porphyry deposit, open towards the west, northwest and at depth. The extensive supergene enriched zone has developed down structures that transition into primary sulfide mineralization. Modelling shows multiple bornite centers within the primary zone highlighting exploration potential at depth and along the currently modelled structures.

25.5.2 Risks

No portion of the present resource is classified as Measured. A key objective for the next phase of study is to have a large portion of the material mined during the payback period to be in the Measured Category. Once the project capital is paid back, the risk posed by carrying lower confidence material in the mine plan lessens.

25.5.3 Opportunities

Additional factors preventing drilled areas from being classified as Indicated or Measured are the presence of the vegas and cryogenic geoforms. McEwen Mining is continuing to conduct site investigations exploring the possibility of getting the known geoforms reclassified or identified as a non-water source for the region. If this is successful, material presently classified as Inferred due to environmental concerns can be upgraded to Indicated and Measured based on other factors.

The deposit is very much drill limited. It is open to the north, north-west, and at depth. Recent drilling has been concentrated on infill to increase Inferred tonnes to Indicated. More needs to be done to address areas under the Vegas and some internal “islands” of Inferred material, but at the same time a step out program of exploratory drilling would almost certainly bring additional resource into the project.

The structural controls of the original porphyry intrusions and subsequent enrichment of mineralization are well understood and used for the creation of the geological model on which the resource is based. With multiple periods of exploratory work by several firms the logging and sampling procedures have varied significantly. Re-visiting previously logged core and using a consistent code system would strengthen knowledge of the geologic nuances of the deposit and potentially increase confidence of our understanding to a Measured level.

25.6 PIT GEOTECHNICAL

25.6.1 Interpretations and Conclusions

The suggested slopes for the pit are based on very limited geotechnical information. Drilling to date has focused on the mineral resource and not the pit walls and has typically been within vertical holes. This is common at this stage of study. Below surface discontinuity and structural data is not available and so, kinematic assessments, and inclusion of structure behind the pit walls in rock mass stability assessment have not been possible.

Gaps in the knowledge base are anticipated to be addressed through investigation programs leading up to the feasibility study, including, inclined holes oriented into the proposed pit walls, televiewing of holes, geotechnical logging, laboratory testing, triple tube coring and hydrogeological investigations.

Slope angles presented assume of a low consequence of failure, with an associated target factor of safety of 1.2. This assumption is based on the planned outwardly extending pit shell, which gives opportunity to adapt the pit wall slope angles based on prior performance. The assumption is that pit wall failures may not sterilize significant processable materials, as failures can be excavated, and post failure pit walls adapted accordingly. Outwardly expanding walls can be replanned based on actual conditions encountered.

25.6.1.1 Risks

- Investigations reveal the rock (intact strength, rock mass strength, alternation, structure, weathering) influencing stability of the pit walls is worse than assumed, resulting in shallower design slopes.
- Faults and other structure causes issues with some orientations of the pit walls.
- The phreatic surface/pore pressure behind the pit walls is worse than assumed, due to regional hydrogeology or ineffective dewatering of the pit walls. This may result in potential for shallower slopes.
- A seismic event triggers slope failure (seismic stability will be to be reviewed in feasibility once there is adequate information regarding the pit wall rock, however there could be an event larger than the design event).
- Slope failure due to structural or rock mass concerns, not identified during any investigations. Whilst impending slope failure may be identified through planned pit wall monitoring and performance reviews during operation, and therefore may not affect worker safety, instabilities can sterilize processable materials and/or slow down operations within the pit.
- Natural geohazards, such as landslides, adversely affect the pit walls, or interfere with safe pit operation. There has been some debate to date over interpretation of natural slides or glacial features on the mountain forming the east wall.
- Poor blasting techniques result in bench scale issues.
- Low consequence of failure is no longer appropriate, and a higher target factor of safety is required, resulting in shallower pit slopes.

- Pit slope failure damages infrastructure. The latest PEA infrastructure plan shows the primary crusher very close to the pit crest; this may not be accounted for in the agreed low consequence designation and associated target factor of safety. Regardless of the consequence classification, and FOS adopted, pit failures do occur in practice and factor of safety does not eliminate probability of failure. A pit failure at this location could impact the primary crusher.
- Overburden is thicker than the average assumed by the pit designers. This would reduce the overall slope angles further. Overburden thickness is currently under investigation; estimates to date are largely based on commencement of coring rather than bedrock interface resulting in some uncertainty.
- Waste rock facility (WRF) failure results in flow of waste rock into the pit impacting safety and production. The WRF run-out distance for the PEA has been assumed from an empirical average for dry slopes. Whilst there is some buffer off-set in addition to the predicted run-out zone, empirical data shows scatter and wet/partially wet conditions may exist which may increase run-out distance.
- The WRF triggers pit wall instability. Stability analyses conducted by the WRF team indicate target FOS are achieved for WRF to pit failure.
- The pit slopes are potentially very high, and the rock is poor quality. There is little global published empirical performance data for similar rock conditions and pit slope heights, which is a risk for the project.

25.6.1.2 Opportunities

- Investigations reveal the rock (intact strength, rock mass strength, alternation, structure, weathering) influencing stability of the pit walls is better than assumed, resulting in steeper design slopes. Pit wall rock may be less fractured and/or stronger than rock hosting the resource (investigated to date) and /or enhanced geotechnical investigations may promote better recovered rock quality.
- Televiewer data indicates the structure is not adversely oriented and/or the rock is less fractured in-situ than recovered in core, allowing steeper slopes.
- Laboratory testing reveals the intact rock is stronger than previously assumed. To date, there have only been 24 UCS tests and reliance for intact compressive strength has been on point load testing with the absence of a site-specific, unit-specific conversion to UCS.
- Overburden is thinner than the average assumed by the pit designers. This would reduce the overall slope angles further. Overburden thickness is currently under investigation; estimates to date are largely based on commencement of coring rather than bedrock interface.
- The phreatic surface/pore pressure behind the pit walls is lower than assumed, due to regional hydrogeology or more effective dewatering of the pit walls. This may result in potential for steeper slopes.
- Investigations reveal the rock (intact strength, rock mass strength, alternation, structure, weathering) influencing stability of the pit walls is better that assumed, and further review by the WRF team reduces the run-out zone of the WRF, resulting in the potential to reduce the WRF offset to the pit.

25.7 MINE PLAN AND MINING METHODS

25.7.1 Interpretations and Conclusions

Pit optimization and mine planning includes Inferred and Indicated material; therefore, there is no guarantee that the presented PEA based on these combined resource tonnages will be realized. Mine planning and scheduling is on an annual (yearly) basis and requires more detailed analysis to ensure it is achievable with the given geological block model.

25.7.1.1 Risks

In addition to the general risks encountered by a mining operation of this scale, the following risks were identified specific to Los Azules.

Very poor rock quality: this risk was addressed in detail in the Geotechnical section but also poses significant safety and economic risks from the perspective of the mining operation. If, for example, the target bench face angle cannot be achieved, over time the design slope angle will not be achieved either. This will either lead to material loss during a specific phase (or at the end of mine life) or result in significantly more waste rock that would have to be stripped to expose the planned processable material. Both scenarios would impact the economics of the operation.

High altitude: although not unusual in mining in the high Andes, Los Azules is located at a high altitude, and this will certainly affect people and machines. To counter the loss in efficiency in diesel equipment, certain adjustments will have to be made and a percentage of power will still be forfeited. The proposed incorporation of battery and electrical equipment will mitigate this risk to the extent that it is implemented.

The mine schedule relies on a stockpiling strategy that requires more detailed analysis to see if it is achievable on a shorter time scale. This could result in periods within a year where there is not enough mineralized material being mined and periods within a year where there is more than planned mineralized material being mined. Additionally, the grade distribution at a bench scale needs to be better understood to have full confidence in the stockpiling strategy used for the PEA.

The mine design and plan are based on pit shells and not detailed pit designs that include interim and final access ramps. The inclusion of this information can potentially lead to more waste needing to be mined in order to establish or maintain access to mining areas within the pit.

Waste material characterization has not been taken into consideration and potentially acid generating material may require certain placement considerations that could impact the mine plan and schedule.

25.7.1.2 Opportunities

Identify ways to mine the open pit with low carbon material handling options to reduce emissions and greater productivity and reduce operating costs.

Improvements to the geological block model with better grades and/or more mineralized material tonnes. This could also impact changes to pit optimization inputs such as costs, selling price, recoveries, and slope angles.

25.8 PROJECT INFRASTRUCTURE

25.8.1 Access & Transportation

Access to the site can be achieved with upgrades to existing roads to support the project initially. Routes have been identified that provide access to ports in both Chile and Argentina. All routes will need some upgrades to fully allow traffic for construction or to support operations.

As a longer-term solution for employee transport, an area for an airfield has been identified to minimize personnel travel times once the project is put into operation. This can be constructed at some point in the future, if required.

25.8.2 Power Supply

Sufficient renewable power is available from the Argentine national power provider, YPF Luz. Initial power will be provided from the Calingasta substation and a 220kV power line to site will be constructed by YPF using the Exploration Road for support. A project milestone was the recent Memorandum of Understanding we signed with YPF Luz on April 14, 2023. YPF is one of the biggest companies in Argentina and is a majority state-owned group of national oil, gas, and energy companies. This memorandum signed between McEwen Copper and YPF Luz sets out the framework to deliver appropriate solutions to provide 100% renewable energy for the operation of Los Azules and San Juan, aimed to prepare Los Azules for carbon neutrality by 2038. A formal agreement and terms are pending from YPF Luz at the time of this report.

In addition, approximately 20% of the site power will be generated as a byproduct of the sulfuric acid plant. Finally, solar power will be used at the main camp and other facilities to minimize grid power usage.

25.9 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

No interactions were observed between the study areas (Mine Area and Access Road) and protected areas, nor with areas of conservation interest based on native forest criteria. Also, there are no RAMSAR sites, Biosphere Reserves or other protected sites recognized by the Nation, the Province of San Juan and/or international treaties.

25.10 PIT DEWATERING AND WATER AVAILABILITY

25.10.1 Groundwater

25.10.1.1 Interpretations and Conclusions

Pit dewatering and groundwater availability estimates are based on limited site-specific data. Hydraulic conductivity for these estimates is based on the following testing.

- Short duration pneumatic slug testing conducted in five piezometers.
- Air lift testing conducted in one exploration boring.
- Limited Lugeon packer testing.

This testing provides hydraulic conductivity estimates in the near vicinity of the borehole and may not provide representative estimates of aquifer storage parameters such as specific storage and specific yield.

The current conceptual model for groundwater flow in the region of the proposed pit suggests that groundwater flow may be highly influenced by faulting. Long duration pumping testing will be required for

DFS design to provide more reliable estimates of hydraulic conductivity and other hydrogeologic parameters. This testing will also show the degree to which faulting influences groundwater flow, pit dewatering, and potential groundwater supply.

25.10.1.2 Risks

Given the current uncertainty in hydrogeologic parameters and influence of faulting on groundwater flow, dewatering requirements may be larger than estimated.

Given the current uncertainty in hydrogeologic parameters and influence of faulting on groundwater flow, available groundwater may not be sufficient to meet process supply requirements.

25.10.1.3 Opportunities

Given the current uncertainty in hydrogeologic parameters and influence of faulting on groundwater flow, dewatering requirements may be smaller than estimated.

25.10.2 Surface Water

25.10.2.1 Interpretations and Conclusions

Surface water availability estimates are based on local and regional meteorological parameters extended and completed until 2021, which may vary in the future. The meteorological data from the Los Azules station has some anomalous readings indicating quality issues with the data. A QA/QC check of the data should be performed for all parameters.

Surface water availability estimates flowing from the Los Azules development is all contained within a single watershed. Potential losses or gains due to surface diversion structures or storage facilities are not accounted for in these estimates. Site-specific surface flow monitoring data is limited periodic measurements. To reduce uncertainty, continuous stream flow gauging stations are suggested to be installed in the project area.

Over the recent 6-year drought period (2016-2021) estimated average annual surface water flow exiting the Los Azules watershed is 60% of historical water availability (2001-2021). Climate change projections indicate that rainfall will decrease, and temperature will increase in the following years diminishing surface runoff. It was further observed that monthly surface water availability varies frequently over seasonal ranges with minimal or nonexistent flows occurring in some months over the available measurement periods.

25.10.2.1.1 Risks

Climate change may decrease the availability of surface water for future mining operations.

25.10.2.1.2 Opportunities

Current surface water availability estimates assume that all winter snowpack sublimates and is not included in the estimates. Snowmelt runoff may add significantly to water available for mine processes and potable water supply.

Climate change may increase the availability of surface water for future mining operations.

25.11 MINE ROCK STORAGE FACILITIES

25.11.1 Interpretations and Conclusions

Based on preliminary geotechnical assessments, overall end slopes of 2.5H:1V for the mine rock storage facilities (MRSFs) were found to meet the geotechnical stability criteria. The slope stability results showed that stability was controlled by seismic loading. The preliminary geotechnical assessments were based on limited site information available at this stage of the study, comprised mainly of data from test pitting, geotechnical index testing, historical drilling, and surface mapping. Gaps in the data are anticipated to be addressed through the ongoing geotechnical site investigation, which includes boreholes, test pits, surface geophysics and laboratory testing tailored to characterize the foundations of the proposed MRSFs.

25.11.1.1 Risks

The geotechnical analyses were completed based on limited data available.

Flow slide runout, particularly from high un-benched MRSFs, has the potential to impact downslope infrastructure.

Rock rollout is expected to be a common occurrence that requires management particularly during active construction of the MRSFs.

Wetlands typically host fine-grained sediments with low shear strengths, high groundwater surfaces, and potential for liquefaction.

With vertical thickness up to 240 m, the materials at the base of the proposed MRSFs will experience high stresses. At such stress levels, materials can experience particle crushing and deformation, potentially resulting in reduced shear strength and decreased permeability. These factors will influence the performance of the MRSFs in the long term, including post-closure.

As the proposed MRSFs are constructed to their design elevation, snow deposited over steep slopes could potentially create additional avalanche risk in the downstream areas. Intermediate berms / benches could help mitigate this risk.

As the MRSFs in some areas are planned to fill valleys with major natural drainages, the potential exists for buildup of pore pressures and/or groundwater in the MRSFs, which would generate instability in the short or long term. Inversion channels are often used for sidehill fill and heaped rock storage facilities; however, these are usually difficult to incorporate into valley or cross-valley fills unless topography and gradients are such that most of the stream flows can be intercepted upstream of the dump and channelized on the valley slope beside the dump.

25.11.1.2 Opportunities

The selection of 2.5H:1V overall slopes was based on the results of the preliminary slope stability assessment, which showed that the stability was controlled by seismic loading. The seismic loading cases were modelled using the pseudo-static loading method, which serves as a screening method. If it is controlling, it should be supplemented by a more comprehensive evaluation of the seismic effects using a deformation analysis. Preliminary deformation analyses indicate that acceptable factors of safety (FoS) are likely to be returned for overall slopes that are shallower than 2H:1V but steeper than 2.5H:1V.

Potential also exists that optimized MRSF designs may include lower seismic FoS stability cases that manage seismic deformation risk using larger infrastructure offsets, additional catch benches, toe berms, catchment trenches, or other approaches.

Initial site investigation results indicate that the phreatic surface / pore pressure is lower than assumed, which may result in improved stability.

To account for the effects of high vertical stresses, the strength of the mine rock was modelled using a conservative shear strength function. A comprehensive review supported by site investigation and laboratory testing results may bring about an improvement of the shear strength for the mine rock resulting in potentially steeper design slopes.

Generally, the most economic means of disposal is adopted for the construction of MRSFs. However, for adequate long-term stability, a controlled construction methodology should be adopted such as using sectors with low-quality mine rock placed at the back of the facilities and high-quality rock placed at the face slope. Implementation of regular UAV photogrammetry could be an innovative solution for tracking the PSD in the MRSF over its construction period. Controlled handling and placement of mine rock can also be used to separate materials that are potentially acid forming from those that are non-acid forming.

26.0 RECOMMENDATIONS

26.1 OVERALL RECOMMENDATIONS

This subsection was prepared by J. Sorensen, FAusIMM, Samuel Engineering and reviewed by the respective QP's for each subsection area.

Based on the results of this Preliminary Economic Assessment, contributing authors recommend that McEwen complete additional work to further de-risk the Project, including more advanced stages of drilling to complete the work necessary for a Feasibility Study based Technical Report.

Given the maturity of the resource development and project technical and permitting basis to date, a Preliminary Feasibility Study is considered an optional step and a Feasibility level of project definition is recommended to expedite the project development timeline to comply with the requirements of the property ownership agreements with the Mining Ministry. As of the effective date of this report, initial Feasibility resource drilling and metallurgical test work were started and in progress.

It is recommended that McEwen now focuses on further de-risking the Los Azules Project by moving to a more robust knowledge base in several critical areas. The Priority Next Steps should be:

- Enhance the definition of the mineralized material by infill drilling programs over two years supported by geological and geophysical work to develop a significant "Measured resource" at Los Azules with particular focus to the initial 5-year mine pit.
- The performance of all studies, monitoring and engineering related to interactions between the Los Azules Project and naturally occurring water.
- The performance of the environmental baselining work in conjunction with specific engineering enabled the IIA submission to the San Juan authorities in April 2023 with the objective of receiving the permitting for the development of Los Azules during the second half of 2024.

A Feasibility Study level of definition is estimated to take approximately 18 to 20 months to complete, overview provided in Figure 26.1. Based on current information from work in progress, the estimated cost is approximately $232 million including estimates for McEwen Copper/ACMSA costs (Table 26.1). The recommended technical program to complete the work deemed necessary to support the completion of a Feasibility Study is as follows:

- Complete an in-fill resource definition drilling program targeting Measured resource classification for the initial five years of the project and areas within the initial project supergene resource to Indicated classification as considered in this PEA. The program delineated for execution includes an additional 32,000 meters of diamond drilling.
- Complete the site geotechnical, seismic, glacier, hydrology and hydrogeologic investigations to a feasibility study level of definition. The program delineated included 16,000 meters of geotechnical drilling, 9,250 meters of hydrogeologic drilling, 9,700 meters of condemnation and other miscellaneous drilling, reestablishment of local surface water monitoring and field surveys.
- Complete confirmatory metallurgical test work and geometallurgical definition for the initial project process. The program delineated includes approximately 15,000 meters of additional metallurgical PQ core drilling and sampling of approximately 90 tonnes of materials, additional column leaching metallurgical testing for both conventional and augmented bio-leaching technologies. The metallurgical work includes site testing of the leach concepts with materials from the bulk sampling

campaign. Additional testing on primary mineralization materials for potential milling options is also considered.

- Update resource/geologic models and estimations, mine plans and schedules based on the additional data collected.
- Update leach pad, processing and site/off-site infrastructure facilities designs to feasibility level development and support ongoing permitting requirements. Finalize concepts for power supply, site access and logistics.
- Update execution plans, costs and financial estimates and assumptions based on the updated project definition.

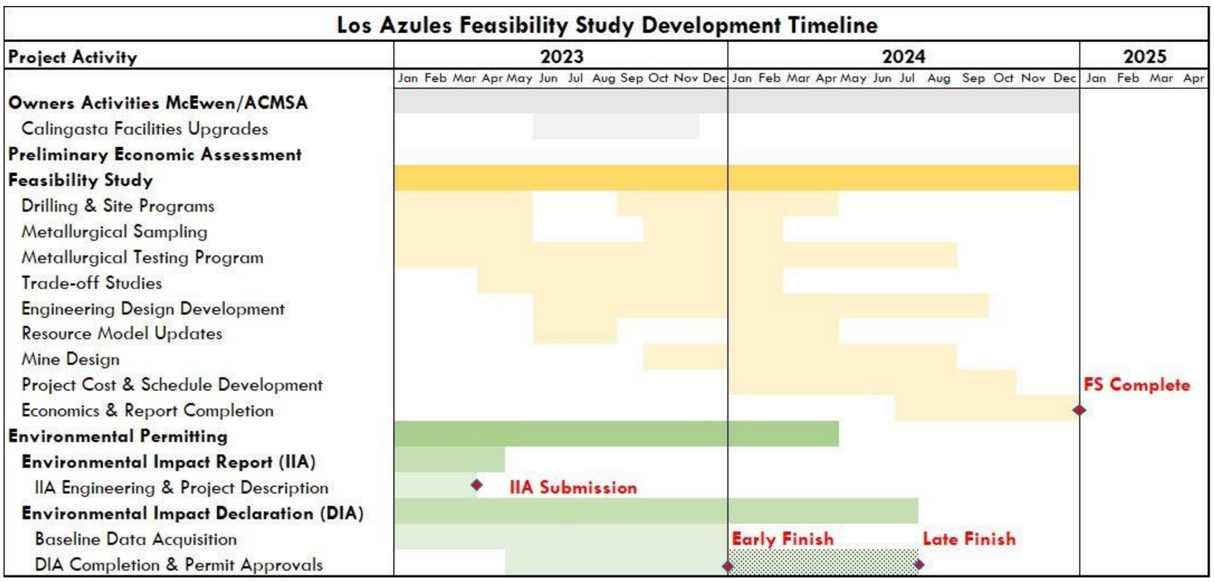


Figure 26.1: Feasibility Study Development Timeline

Table 26.1: Expected Costs for Feasibility Study Development			
Cost Category (USD Millions)	**2023**	**2024**	**TOTAL**
Corporate	$5.1	$3.1	**$8.2**
Roads*	$15.8	$4.2	**$20.0**
McEwen Copper/ACMSA	$16.6	$13.4	**$30.0**
Camp/Site Services*	$15.1	$5.8	**$20.9**
ESG/Permitting	$3.7	$3.8	**$7.4**
Exploration*	$48.3	$32.8	**$81.1**
Feasibility Study - Engineering	$9.8	$13.5	**$23.3**
Calingasta Development	$1.5	$0.1	**$1.6**
Contingency	$2.0	$8.0	**$10.0**
Cost	**$117.9**	**$84.7**	**$202.6**
Estimated VAT*	**$18.8**	**$10.9**	**$29.8**
Total	**$136.8**	**$95.6**	**$232.4**
*** Items account for costs only attributable to the Feasibility Study and do not extend through December 2024.**			

In addition to the above there are facilities and infrastructure areas to optimize, improve and de-risk through further on-site work, off-site work and by performing detailed trade off studies as described in the following text.

- If the Northern Road route is deemed constructible and feasible then perform a comprehensive trade off against the existing Exploration Road and Southern routes and define the preferred site access route.
- Trade-Off Project Infrastructure – Power Transmission to Los Azules.
  Update the power supply and renewable energy supply PEA studies for the finalized project execution scope of work. As the power supply and associated transmission line is a major critical component of the project, many aspects must be focused to in the immediate future to deliver a definitive robust logical and viable solution in which all the stakeholders have confidence and will be supported through the permitting processes and implementation.
  It is recommended all the following aspects are further developed as a priority:
  - Confirm the suitability and availability of a cost-effective, renewables-based power supply from Argentina. Develop preliminary terms and conditions for power supply.
  - Performance of power grid load flow studies.
  - Conduct a preliminary selection of conductor and transmission tower types suitable to the various climatic environments any transmission line will pass through.
  - Define the legal easements acquisition challenges and alternative processes.
  - Define permitting requirements. The Argentina authorities have advised the transmission line is to be included in the main IIA for Los Azules.
  - Include in the trade off study any benefits from a transmission line also aligned to the potential Northern Route which is expected to be relatively snow free.
- When the preferred power source and the transmission route are defined then perform environmental, geotechnical, and preliminary engineering such as access for overhead cable pulling and tower construction.
  - Evaluation of micro-hydro and solar generation for the power supply to advance camps and isolated access control sites.
  - Trade-off solar verses diesel generation at the site during the project implementation phase.
- Project Infrastructure – Sulfuric Acid and Sulfur Supply
- Further refine the acid consumption requirements based on updated metallurgical testing. Develop the sulfur supply and logistics based on off-take and expected consumption term sheet with YPF and other potential suppliers.
- Project Infrastructure – Logistics and Import/Export through Argentina and Chile
- Further refine the logistics and transport study completed for the PEA based on off-take and consumption brokerage term sheet.
- A Heads of Agreement term sheet should be developed with the ports of Rosario, Valparaiso, San Antonio, and or Coquimbo.

26.2 METALLURGY AND MINERAL PROCESSING

26.2.1 Recommendations

Based on the results to date and positive economic potential of the project, the QP recommends advancing the project to a Feasibility study level of definition.

26.2.2 Further Work

Continued and further metallurgical testing should be done to refine acid consumption and copper recoveries by source (lithologies and spatial variability) to support the metallurgical and geo-metallurgical understanding to support a Feasibility Study. The current metallurgical program is developed in three (3) phases based on sample availability to support this PEA and continuing in parallel to support future study and objectives. The Phase 1 program is complete and pending some final analysis at the time of this report and considered for the PEA analysis along with the historical information. The Phase 2 and 3 programs are started, and metallurgical sampling and sample preparation is in progress.

A substantial metallurgical sampling program has been delineated to support the Feasibility Study metallurgical program. Drill core programs includes 6,000 meters of additional metallurgical PQ core drill core (and/or equivalent HQ core) and sampling to obtain approximately 90 tonnes of material, additional column leaching metallurgical testing for both conventional and augmented bio-leaching technologies. The metallurgical work includes site testing of the leach concepts with materials from the bulk sampling campaign. Additional testing on primary mineralization materials for potential milling options is also considered.

This would include:

- The testing as outlined in this report is required to advance the level of confidence in leaching performance criteria such as recoveries, acid consumption, leach flow rates and hydrodynamic flow.
- Continued the Phase 2 metallurgical testing to be completed on additional lock-cycle column testing on the variety of material types and lithologies.
- Continued sample collection and testing in a Phase 3 metallurgical testing program, using large amounts of bulk samples for the anticipated heap lift height (columns of 10 m height) to simulate leaching of material types and lithologies to support a feasibility study.
- Test the potential benefits to direct addition of elemental sulfur to the leach materials to generate acid and heat within the leach pad.
- Use data from the variability testing program together with block model and mine production schedule to better define and optimize metal production.
- Continued metallurgical testing of gold leaching technology for low copper / high gold concentrations or the use of thiocyanate leaching / washing to recover gold on small portions of the heap leach pad.
- Continued and further metallurgical testing of copper sulfide mineral recoveries could also demonstrate alternate process facilities thereby resulting substantial expansion of production rates.

Continued and further metallurgical testing should be done to refine parameters for Nuton™ bio-leaching technology and to define acid consumption and copper recoveries by source (lithologies and spatial variability). This would include:

- Completion of the Nuton™ technology testing at their Bundoora and Hazen testing facilities on small columns and characterization testing.
- Continued testing in a Phase 2 metallurgical testing program, using large amounts of bulk samples

for the anticipated heap lift height (columns of 10 m height) to simulate leaching of material types and lithologies to support a feasibility study inclusion.

- Test the potential benefits for direct addition of elemental sulfur to the leach materials to generate acid, if required.
- Future larger scale testing of the technology and selected parameters at the Los Azules site to demonstrate the process at altitude and site conditions.

In the event a mill/concentrator option is contemplated for the primary copper sulfide materials, the following additional work should be completed to better define the viability of that option. This would include:

- Comminution variability testing be performed on an additional 40 to 60 drill hole interval samples to develop enough data to allow a correlation between PLT and Schmidt Hammer testing for incorporation into the resource model to predict hardness over the life of the mine.
- Review the copper precipitate process design to ensure features have been incorporated to ensure production of a concentrate with minimal arsenic and other impurity levels. This includes pyrite suppression techniques for supergene ores that may be processed through the copper concentrator instead of the heap leach circuit and assay of lock-cycle material for deleterious minerals and smelter penalties.
- McEwen should start discussions defining exact copper concentrate sales terms, requirements, and shipping destinations.

26.3 PIT GEOTECHNICAL

26.3.1 Recommendations

A geotechnical open pit drill program is recommended. The program should include:

- Inclined holes into the proposed pit walls.
- Televiewing of all holes.
- Geotechnical logging.
- Laboratory testing.
- Triple tubing to enhance recovery.
- Hydrogeological investigations.

Further resource drilling should also help support future geotechnical pit assessments.

26.3.2 Further Work

The following work items are recommended for the next stages of geotechnical pit assessment once further data is obtained through investigation.

- Review of newly acquired data.
- Inclusion of newly acquired data into pit slope geotechnical assessment.
- Further development of a structural model.
- Inclusion of faulting / structure into pit slope geotechnical assessment.
- Establishment of geotechnical pit domains and sectors.
- Review of failure modes including structurally controlled and rock mass failure modes.

- Consideration of bench scale, inter-ramp scale and overall scale for stability assessment.
- Inclusion of modeled hydrogeology to input the phreatic surface behind the pit walls.
- Consideration of pit dewatering to achieve pit wall slope optimization.
- Seismic assessment of slope stability.
- Development of a geotechnical risk register.
- Inclusion of waste dump loading and waste dump in pit slope geotechnical assessment.
- Development of monitoring plans.
- Geohazard assessment of the potential natural landslides that may affect pit wall stability and / or impact the pit.
- Review of overburden thickness and nature across pit.
- Additional investigation, as required, depending on findings of first geotechnical investigation. Questions / uncertainties / points of clarification may emerge during the assessment, which may need additional drilling the following season.

26.3.3 Pit Wall Monitoring and Performance Recommendations

Monitoring plans should be developed as the design evolves into FS. Monitoring plans should be based on best monitoring practice (Sharon, 2020). This should include:

- Verification of the slope design. This is an iterative process whereby slopes are monitored, actual conditions determined, as-built plans generated and compared to predicted conditions and mine plan assumptions. Slope designs may be modified throughout operation.
- Surface Deformation monitoring using prisms and radar and including development of Trigger Action Response Plans.
- Consideration of InSAR monitoring to review overlying geohazards.
- Sub-surface deformation monitoring, which may include Shape Accel Arrays (SAAs) and Time Domain Reflectometers (TDRs).
- Piezometric monitoring.
- Monitoring of climatic conditions.
- Formal inspections, performance reviews and third-party review.
- Ongoing Pit survey capture to include topography and pit wall mapping.

26.4 PIT DEWATERING AND WATER AVAILABILITY

26.4.1 Groundwater

26.4.1.1 Recommendations

Dewatering test wells and associated monitoring wells should be installed in the northeast area of the proposed pit where faults may influence groundwater flow, and in the central and southern areas of the proposed pit. Long-term, high-capacity pumping tests should be conducted in these wells to reduce the uncertainty in hydrogeologic parameters.

26.4.2 Surface Water

26.4.2.1 Recommendations

It is recommended that monitoring continue with the Los Azules meteorological station since it is the only representative station in the study area. It is also recommended that an additional weather station be installed at the site to have backup data in case of Los Azules station malfunction.

The project water balance should be updated to include potential surface water gains and losses due to diversion structures.

Continuous recording surface water flow monitoring stations should be installed on the Rio Salinas where it exits the project and in the major sub watersheds.

The calculation of water availability should be developed using a more refined and discretized methodology and considering climate change, the project facilities, water management structures and water demands.

A detailed contact water / non-contact water management plan needs to be developed, including the design and location of water diversion structures and the staged formation of any contour channels. This needs to be further supported by an engineered project water balance.

26.5 MINE ROCK STORAGE FACILITIES

The geotechnical analyses should be revisited based on the results of the ongoing geotechnical site investigations.

Flow slide runout assessment should be completed for the proposed MRSFs and stockpiles as the designs are refined at the DFS stage.

Guidelines should be established for operational management of rock rollout using catch benches, berms, trenches, offsets, and/or clear and close procedures.

The wetlands within the footprints of the proposed MRSFs should be characterized in the site investigation and laboratory testing.

It is recommended that the degradation of the material properties be considered in the geotechnical design and be confirmed by advanced laboratory characterization methods such as large scale, high pressure triaxial testing.

The haulage routes to the proposed MRSFs should be assessed for snow avalanche risk.

The applicability of diversion channels and/or underdrainage systems should be evaluated during later stages of design.

26.6 TAILINGS STORAGE FACILITY

If the mill/concentrator requirements are further developed, the design basis and criteria should be updated to include site specific information.

Detail geochemistry of the process feed material and tailings will be required during the next design stage to determine the PAG tailings.

Comprehensive testing should be performed on tailings materials to define geotechnical and index parameters. Additionally, definition of the P80 and mill capacity to define the baseline tailings parameters should be established before as tailings management methodology is largely dependent on these parameters. Compaction of tailings will be directly related to the tailing's properties. Improper compaction estimates may result in inaccurate overall storage volume estimates of the FTSF.

Further development of hydrotechnical studies is required for the FTSF. This includes water balance, water management, and return water structures as it is understood that the process water makeup will be required to rely on the FTSF water balance. The overall project success relies on the supply of water; therefore, this task is considered the priority for the next design stage. For de-risking the project, a consideration should be made to collect all surface contact water and the seepage bypassing the buttresses structures and forwarding for process.

A detailed dynamic water balance for closure needs to be developed and alternative discharge routes should be evaluated. These may include tunneling or pumping over the hills. Buried pipeline conduit under the FTSF is not recommended due to limited longevity and potential failure mode related to the pipeline collapse.

There is a 50-70 m thick alluvium / colluvium deposit along the base of the FTSF channel, which will have a significant impact on the overall water balance of the FTSF and open pit. A three-dimensional hydrogeology model for the FTSF and open pit must be developed for the operation and closure conditions. This stratum could be also susceptible to liquefaction, which could impact slope stability of embankment during earthquake events.

Non-contact water diversion ditches will be built in the highly hilled terrain, often founded on steep slopes. The slopes should be inspected and surveyed, whether they are feasible to install ditches on the slopes or not. A field investigation is also recommended to assess soil conditions for proper design. Alternatively, non-contact water ditches could be installed on the edge between the FTSF and the hills; however, their operation would be often interrupted due to constant FTSF surface raising. Inadequate design or failing ditches could result in additional impact flow to the FTSF and possible overflow over the buttresses crest.

Contact water ditches will be installed on the FTSF surface routing the surface runoff downstream of the FTSF. The ditches will be in constant reconstruction and relocation due to the raising of FTSF surface. Detailed schedule of ditch construction with the tailings filling plan and decant tower raising must be worked out during next study phase.

Detailed sizing of all ditches and ponds will be completed during the next design stage.

A trade-off study is recommended for detail scheduling of decant tower raising, and if replacement decant may be required at higher elevations if stresses will exceed the recommended target.

It was assumed that after year 10 of FTSF operation, a collection pond for the Heap Leach will not be required and the SE Buttress can be constructed in its place. It is understood that the Heap Leach leachate will not be collected after 5 years of ceasing its operation. It is recommended to consider if the entire space between the FTSF and Heap Leach can be filled during late FTSF operation. Significant volume can fit in the space resulting in buttress construction savings. Similarly, the space between the North MRSF and the FTSF could be also filled.

The FTSF is located downstream of the possible leach pad, pit and North MRSF locations. A sitewide water management strategy that includes the non-contact water management interaction between those facilities for operation and closure stages must be developed during the next design stage.

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28.0 APPENDICES

28.1 APPENDIX A – UNITS OF MEASURE AND ABBREVIATIONS AND ACRONYMS

28.1.1 Units of Measure

Table 28.1: Units of Measure		
Above Mean Sea Level	-	**amsl**
Ampere	-	A
Amperes per Square Meter	-	ASM
Annum (Year)	-	a
Argentine Peso	-	AR$
Billion	-	B
British Thermal Unit	-	BTU
Centimeter	-	cm
Cubic Centimeter	-	cm^3
Cubic Feet Per Minute	-	cfm
Cubic Feet Per Second	-	ft^3/s
Cubic Foot	-	ft^3
Cubic Inch	-	in^3
Cubic Meter	-	m^3
Cubic Yard	-	yd^3
Coefficients Of Variation	-	CVs
Day	-	d
Days Per Week	-	d/wk
Days Per Year (Annum)	-	d/a
Dead Weight Tonnes	-	DWT
Decibel Adjusted	-	dBa
Decibel	-	dB
Degree	-	°
Degrees Celsius	-	°C
Diameter	-	ø
Dollar (American)	-	US$
Dollar (Canadian)	-	CDN$
Dry Metric Ton	-	dmt
Foot	-	ft
Gallon (US)	-	gal
Gallons Per Minute (US)	-	gpm
Gigajoule	-	GJ
Gigapascal	-	GPa
Gigawatt	-	GW
Gram	-	g
Grams Per Litre	-	g/L
Grams Per Tonne	-	g/t
Greater Than	-	>
Hectare (10,000 M2)	-	ha
Hertz	-	Hz

Table 28.1: Units of Measure	
Horsepower	- hp
Hour	- h
Hours Per Day	- h/d
Hours Per Week	- h/wk
Hours Per Year	- h/a
Inch	- in
Kilo (Thousand)	- k
Kilogram	- kg
Kilograms Per Cubic Meter	- kg/m^3
Kilograms Per Hour	- kg/h
Kilograms Per Square Meter	- kg/m^2
Kilometer	- km
Kilometers Per Hour	- km/h
Kilopascal	- kPa
Kiloton (1,000 Tonnes)	- kt
Kilovolt	- kV
Kilovolt-Ampere	- kVA
Kilovolts	- kV
Kilowatt	- kW
Kilowatt Hour	- kWh
Kilowatt Hours Per Tonne	- kWh/t
Kilowatt Hours Per Year	- kWh/a
Less Than	- <
Liter	- L
Liters Per Minute	- L/m
Liters Per Second	- L/s
Megabytes Per Second	- Mb/s
Megapascal	- MPa
Megavolt-Ampere	- MVA
Megawatt	- MW
Meter	- m
Meters Above Sea Level	- masl
Meters Per Minute	- m/min
Meters Per Second	- m/s
Micron	- µm
Milligram	- mg
Milligrams Per Liter	- mg/L
Milliliter	- mL
Millimeter	- mm
Million	- M
Million Bank Cubic Meters	- Mbm3
Million Bank Cubic Meters Per Annum	- Mbm3/a
Million Tonnes	- Mt
Minute (Plane Angle)	- '
Minute (Time)	- min
Month	- mo

Table 28.1: Units of Measure	
Ounce	- oz
Pascal	- Pa
Centipoise (MPa·S)	- cP
Parts Per Million	- ppm
Parts Per Billion	- ppb
Percent	- %
Pound(S)	- lb
Pounds Per Square Inch	- psi
Revolutions Per Minute	- rpm
Second (Plane Angle)	- "
Second (Time)	- s
Short Ton (2,000 Lb)	- st
Short Tons Per Day	- st/d
Short Tons Per Year	- st/y
Specific Gravity	- SG
Square Centimetre	- cm2
Square Foot	- ft2
Square Inch	- in2
Square Kilometre	- km2
Square Metre	- m2
Three-Dimensional	- 3D
Tonne (1,000 Kg) (Metric Ton)	- t
Tonnes Per Day	- t/d
Tonnes Per Hour	- t/h
Tonnes per annum	- t/a
Tonnes Seconds Per Hour Metre Cubed	- ts/hm3
United States Dollar	- USD
Volt	- V
Week	- wk
Weight/Weight	- w/w
Wet Metric Ton	- wmt
Year	- yr

28.1.2 Abbreviations and Acronyms

Table 28.2: Abbreviations and Acronyms	
Acid Generating	- **AG**
Acid Rock Drainage	- ARD
Alternating Current	- AC
Ammonium Nitrate Fuel Oil	- ANFO
Association for the Advancement of Cost Engineering	- AACE
Andes Corporación Minera S.A.	- ACMSA
Autogenous/Ball Mill/Crushing	- ABC
Battle Mountain Gold	- BMG
Bond Ball Mill Work Index	- BWi

Table 28.2: Abbreviations and Acronyms	
Inductively Coupled Plasma	- ICP
Canadian Institute of Mining, Metallurgy and Petroleum	- CIM
Certificate Of Approval	- CofA
Close-Circuit Fully Autogenous Grinding Milling	- FAC
Conceptual Closure and Rehabilitation Plan	- CRP
Construction Quality Assurance	- CQA
Direct Current	- DC
Diorite (Pre-Mineral Pluton)	- DIO / PMP
Enrichment Ratio	- ER
Environmental Impact Assessment	- EIA
Environmental Impact Review	- EIR
Environment, Social & Government	- ESG
Exploratory Data Analysis	- EDA
Early Mineral Porphyry	- EMP
Ground Engaging Tools	- GET
Hydrothermal Breccia	- HBX
Hypogene (Primary Zone)	- HYP
Induced Polarization	- IP
Internal Rate of Return	- IRR
International Organization for Standardization	- ISO
In-The-Hole	- ITH
Inverse Distance-Weighted	- ID
Inter Mineral Porphyry	- IMP
Leach Zone	- LIX
Lerchs-Grossman	- LG
Life-Of-Mine	- LOM
Load-Haul-Dump	- LHD
Los Azules Mining, Inc	- LAMI
Magmatic Hydrothermal Breccia	- MAG HYD BX
Magneto Telluric	- MT
Million Years Ago	- Mya
Mine Block Intrusion	- MBI
Minera Andes S.A.	- MASA
Minimum Environmental Protection Standard Laws	- MEPSL
Mount Isa Mines	- MIM
Canadian National Instrument 43-101	- NI 43-101
Nearest Neighbor	- NN
Net Acid Generating/Generation	- NAG
Net Present Value	- NPV

Table 28.2: Abbreviations and Acronyms	
Net Smelter Return	- NSR
New York Stock Exchange	- NYSE
Ordinary Kriging	- OK
Overburden Zone	- OVB
Portable Infrared Spectrometer	- Pima
Preliminary Economic Assessment	- PEA
Primary Zone	- PR
Qualified Persons	- QP's
Quality Assurance	- QA
Quality Control	- QC
Relative Bulk Strength	- RBS
Reverse Circulation	- RC
Rock Quality Designation	- RQD
Run-Of-Mine	- ROM
Selective Mining Unit	- SMU
Semi-Autogenous	- SAG
Semi-Autogenous/Ball Mill/Crushing	- SABC
SGS Lakefield Research Ltd.	- SGS
Solitario Argentina S.A.	- SASA
Specific Gravity	- SG
Standard Reference Material	- SRM
Supergene Zone	- SS
Tailings Storage Facility	- TSF
Toronto Stock Exchange	- TSX
Unidirectional Solidification Texture	- UST
United Nations Development Program	- UNDP
Volcanics	- VOLCS
Waste Rock Storage Facility	- WRSF
World Meteorological Organization	- WMO

28.2 APPENDIX B ACMSA MINERAL CLAIM LOCATION (POSGAR 1994 COORDINATE SYSTEM)

Table 28.3: ACMSA Mineral Claim Location (Posgar 1994 Coordinate System)

No	Mina	Vertex	Designation	EAST (X) Posgar '94	NORTH (Y) Posgar '94
1	Agostina	1	NO-V1	2380911.06	6569793.15
		2	NE-V2	2384611.04	6569793.15
		3	SE-V3	2384611.04	6566593.16
		4	SO-V4	2380911.06	6566593.16
2	Azul 1	1	NO-V1	2379911.06	6558693.20

Table 28.3: ACMSA Mineral Claim Location (Posgar 1994 Coordinate System)					
No	Mina	Vertex	Designation	EAST (X) Posgar '94	NORTH (Y) Posgar '94
		2	V2	2383749.01	6558693.19
		3	V3	2383749.01	6559043.19
		4	NE-V4	2384611.04	6559043.19
		5	SE-V5	2384611.04	6554293.21
		6	SO-V6	2379911.06	6554293.21
3	Azul 2	1	NO-V1	2376911.07	6558693.20
		2	NE-V2	2379911.06	6558693.20
		3	SE-V3	2379911.06	6554293.21
		4	SO-V4	2376911.07	6554293.21
4	Azul 3	1	NO-V1	2382920.83	6565443.17
		2	NE-V2	2384734.04	6565443.17
		3	V3	2384734.04	6565026.17
		4	V4	2383366.05	6565026.17
		5	SE-V5	2383366.05	6563568.18
		6	SO-V6	2382428.05	6563568.18
		7	V7	2382428.05	6564100.61
		8	V8	2382920.83	6564100.61
5	Azul 4	1	NO-V1	2378669.34	6566593.19
		2	V2	2385750.04	6566593.16
		3	V3	2385750.04	6565408.17
		4	NE-V4	2388111.03	6565408.17
		5	SE-V5	2388111.03	6565063.17
		6	V6	2384734.04	6565063.17
		7	V7	2384734.04	6565443.17
		8	V8	2382920.83	6565443.17
		9	V9	2382920.83	6565779.31
		10	V10	2381850.59	6565779.31
		11	V11	2381850.59	6565443.17
6	Azul 5	1	NO-V1	2391911.01	6561127.18
		2	V2	2393625.00	6561127.18
		3	V3	2393625.00	6559960.44
		4	NE-V4	2395341.00	6559960.44
		5	SE-V5	2395341.00	6551793.22
		6	SO-V6	2391911.01	6551793.22
7	Azul Este	1	NO-V1	2384611.04	6559043.19
		2	NE-V2	2391911.01	6559043.19
		3	SE-V3	2391911.01	6555793.21
		4	SO-V4	2384611.04	6555793.21

Table 28.3: ACMSA Mineral Claim Location (Posgar 1994 Coordinate System)

No	Mina	Vertex	Designation	EAST (X) Posgar '94	NORTH (Y) Posgar '94
8	Azul Norte	1	NO-V1	2378600.55	6565443.17
		2	NE-V2	2381850.58	6565443.17
		3	SE-V3	2381850.58	6565063.17
		4	SO-V4	2378268.59	6565063.17
9	Cecilia	1	NO-V1	2385750.04	6573366.61
		2	NE-V2	2387611.03	6574371.55
		3	V3	2387611.03	6567985.13
		4	V4	2388051.03	6567985.13
		5	V5	2388051.03	6567793.13
		6	V6	2388111.03	6567793.13
		7	SE-V7	2388111.03	6565408.17
		8	SO-V8	2385750.04	6565408.17
10	Escorpio I	1	NO-V1	2380927.05	6565063.17
		2	NE-V2	2381850.58	6565063.17
		3	V3	2381850.58	6564100.61
		3	V3	2381850.58	6564100.61
		4	V4	2382428.05	6564100.61
		5	SE-V5	2382428.05	6563568.18
		6	SO-V6	2380927.05	6563568.18
11	Escorpio II	1	NO-V1	2380261.05	6563568.18
		2	NE-V2	2384661.04	6563568.18
		3	SE-V3	2384661.04	6559043.19
		4	SO-V4	2380261.05	6559043.19
		5	Exc. SO-V4	2380330.43	6559137.87
12	Escorpio III	1	NO-V1	2383366.05	6565026.17
		2	NE-V2	2384734.04	6565026.17
		3	SE-V3	2384734.04	6563568.18
		4	SO-V4	2383366.05	6563568.18
13	Escorpio IV	1	NO-V1	2373111.07	6563196.71
		2	V2	2376361.24	6565072.50
		3	V3	2376361.24	6561518.65
		4	V4	2378849.22	6561518.65
		5	V5	2378849.22	6565063.17
		6	NE-V6	2380927.07	6565063.17
		7	V7	2380927.07	6563568.17
		8	V8	2380261.07	6563568.17
		9	SE-V9	2380261.07	6559043.17
		10	SO-V10	2373111.07	6559043.17

Table 28.3: ACMSA Mineral Claim Location (Posgar 1994 Coordinate System)

No	Mina	Vertex	Designation	EAST (X) Posgar '94	NORTH (Y) Posgar '94
14	Gina	1	NO-V1	2379040.11	6570985.15
		2	NE-V2	2385750.04	6570985.15
		3	SE-V3	2385750.04	6566593.16
		4	V4	2384611.04	6566593.16
		5	V5	2384611.04	6569793.15
		6	V6	2380911.06	6569793.15
		7	V7	2380911.06	6566593.16
		8	SO-V8	2379040.11	6566593.16
15	Marcela	1	NO-V1	2391911.01	6567793.16
		2	NE-V2	2395911.00	6567793.16
		3	SE-V3	2395911.00	6559615.19
		4	V4	2395341.00	6559615.19
		5	V5	2395341.00	6559960.44
		6	V6	2393625.01	6559960.44
		7	V7	2393625.01	6561127.18
		8	SO-V8	2391911.01	6561127.18
16	Mercedes	1	NE-V1	2373111.08	6563196.71
		2	P3-V2	2373111.08	6559043.20
		3	P4-V3	2373097.09	6559043.20
		4	SE-V4	2373097.09	6558738.19
		5	SO-V5	2369835.80	6558738.19
17	Mirta	1	NO-V1	2373097.09	6559043.20
		2	NE-V2	2383749.05	6559043.20
		3	SE-V3	2383749.05	6558693.20
		4	V4	2375645.08	6558693.20
		5	V5	2375645.08	6558765.48
		6	SO-V6	2373097.09	6558765.48
18	Rosario	1	NO-V1	2376390.07	6572793.14
		2	V2	2384688.04	6572793.14
		3	NE-V3	2385750.04	6573366.62
		4	SE-V4	2385750.04	6570985.15
		5	SO-V5	2375615.42	6570985.15
19	Sofía	1	NO-V1	2388111.03	6567793.16
		2	NE-V2	2391911.01	6567793.16
		3	SE-V3	2391911.01	6559043.19
		4	SO-V4	2388111.03	6559043.19
20	Totora	1	NO-V1	2384734.04	6565063.17
		2	NE-V2	2388111.03	6565063.17

Table 28.3: ACMSA Mineral Claim Location (Posgar 1994 Coordinate System)					
No	Mina	Vertex	Designation	EAST (X) Posgar '94	NORTH (Y) Posgar '94
		3	SE-V3	2388111.03	6563568.18
		4	SO-V4	2384734.04	6563568.18
		5	SE-VE	2388071.08	6563525.97
21	Totora II	1	NO-V1	2384661.04	6563568.18
		2	NE-V2	2388111.02	6563568.18
		3	SE-V3	2388111.02	6559043.19
		4	SO-V4	2384661.04	6559043.19

28.3 APPENDIX C – QP CERTIFICATES

The QP's, whose certificates are included herein, prepared this technical report titled "CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LOS AZULES COPPER PROJECT" for the Los Azules Project, Argentina. The effective date of this Technical Report is May 9, 2023 and the report date is May 31, 2023.

CERTIFICATE OF QUALIFIED PERSON

ALLAN L. SCHAPPERT, CPG, SME-RM.

I, Allan L. Schappert. CPG, residing at 711 S. Sean Dr., Chandler, Arizona, USA, 85224, do hereby certify that:

1. I am an independent Geology Consultant, contracted by Stantec Inc.
2. This certificate applies to the technical report entitled "Canadian National Instrument NI 43-101 Technical Report Preliminary Economic Assessment, Los Azules Copper Prject" (the "Technical Report") with an effective date of May 9, 2023.
3. I am a graduate of the Lakehead University, Thunder Bay, Ontario, Canada, in 1979 with a Bachelor of Science degree in Geology. I am registered as a Certified Professional Geologist with the American Institute of Professional Geologists (No CPG-11758) and a Registered Member of the Society of Mining, Metallurgy and Exploration (SME # 04164071). I have worked as a mining geologist for a total of 44 years since my graduation.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited the Property that is the subject of this Technical Report in May of 2022 to review ongoing drilling, core logging, sampling, sample security and transport of assays to the assay lab. I have visited Alex Stewart labs in Mendoza, Argentina to review sample security, sample prep, assay methodologies, and QAQC work.
6. I am responsible for authoring Sections 7, 8, 9, 10, 11, 12, and 14 of the Technical Report along with those sections of the Summary pertaining thereto.
7. I am responsible for co-authoring Sections 1, 16, 25, and 26 of the Technical Report along with those sections of the Summary pertaining thereto.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10. As of the date of this certificate, 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.

Effective Date: 9 May 2023

Signed Date: 31 May 2023

Allan L Schappert CPG, SME-RM
Senior Resource Geologist
Stantec Inc.

CERTIFICATE OF QUALIFIED PERSON

Bruno Borntraeger

I, *Bruno Borntraeger, P.Eng.,* certify that I am employed as a Specialist Geotechnical Engineer | Associate with Knight Piésold Ltd (Vancouver), with an office address of. 1400-750 West Pender St., Vancouver, BC Canada

1. This certificate applies to the technical report titled "Project Los Azules - NI 43-101 Technical Report" (the "Technical Report") that has an effective date of May 9, 2023 prepared for McEwen Mining Inc.

2. I graduated from the University of British Columbia in Vancouver, Canada (Bachelor of Applied Science in Geological Engineering, 1990.

3. I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #20926).

4. I have practiced my profession continuously for 32 years. I have been directly involved in geotechnical engineering, mine waste and water management, heap leaching, environmental compliance, mine development with practical experience in feasibility studies, detailed engineering, permitting, construction, operations and closure. I have most recently been a QP for environmental and permitting aspects for the Filo Del Sol Project Updated PFS in 2023, and also the Josemaria Project PFS in 2019.

5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 in connection with those sections of the Technical Report that I am responsible for preparing.

6. I visited the *property on February 4th, 2023, for one day.*

7. I am responsible for sections 18.9 and 4.10, 20 of the Technical Report, as well as relevant parts in the Executive Summary of the Technical Report.

8. I am independent of McEwen Mining Inc. as independence is defined in Section 1.5 of NI 43-101.

9. I have not had prior involvement with the subject property.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument.

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

Knight Piésold CONSULTING

Dated: May 23, 2023

Bruno Borntraeger, P.Eng.
Specialist Geotechnical Engineer | Associate
Knight Piésold Ltd

KNIGHT PIÉSOLD LTD.
PERMIT NUMBER
1001011
EGBC PERMIT TO PRACTICE

CERTIFICATE OF QUALIFIED PERSON

W. David Tyler

I, W. David Tyler, Registered Member, SME, do hereby certify that:

1. I am the Project Director for:

   McEwen Copper Inc.
   Av. Ignacio de la Roza 1240, Capital, San Juan, 5400, Argentina

2. I graduated with a Bachelor of Science in Mining Engineering and a Master of Science in Environmental Science and Engineering, both from the Colorado School of Mines.

3. I am a Registered Member of the Society for Mining, Metallurgy and Exploration in good standing in the United States of America in the areas of mining and project engineering. My Member Number is 3288830.

4. I have worked as an engineer and project manager for over 40 years. My experience includes mining engineering and planning, study management, project management and project evaluations.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am a contributing author for the preparation of the technical report titled "Canadian National Instrument 43-101 Technical Report Preliminary Economic Assessment Los Azules Copper Project" (the "Technical Report") dated effective May 9, 2023, prepared for McEwen Mining Inc.; and am responsible for Sections 4, 5, 6, 18, and 19.

7. I have personally visited the project site several times in my work as Project Director for McEwen Copper.

8. I am currently working with McEwen as a Project Director for this Preliminary Economic Assessment update, and for other projects that McEwen are advancing.

9. 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.

10. I am not independent of the issuer applying all the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 31 day of May, 2023.

“signed and sealed”
Signature of Qualified Person

W David Tyler
Print Name of Qualified Person

James Lynn Sorensen, FAusIMM, QP
8450 E. Crescent Parkway Ste. 200, Greenwood Village, CO, 80111

1-303-714-4840, ext. 7613
jsorensen@samuelengineering.com

CANADIAN NATIONAL INSTRUMENT 43-101

CERTIFICATE of QUALIFIED PERSON

I, James Lynn Sorensen, (FAusIMM), do hereby certify that as the co-author of this Technical Report (Report) on the Los Azules Copper Project, San Juan Province, Argentina, titled "CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LOS AZULES COPPER PROJECT" dated May 31, 2023 with an effective date of May 9, 2023, I hereby make the following statements:

(a) I am a Metallurgical Engineer and Project Manager practicing at Samuel Engineering, Inc., 8450 E. Crescent Pkwy, Ste. 200, Greenwood Village, CO 80111, USA.

(b) I am a graduate of the University of Arizona, Tucson, Arizona, United States of America with a Bachelor of Science in Metallurgical Engineering degree conferred on May 16, 1981.

(c) I am registered in good standing as a Member (Fellow) of the Australasian Institute of Mining and Metallurgy (FAusIMM) since May 2004, Registration Number 221286.

(d) I have practiced my profession continuously as a Metallurgical Process Engineer, Operations Manager, Study Manager and Project Manager for over 40 years.

(e) I have over 25 years of collective, direct experience in the mining industry specifically related to copper hydrometallurgical operations, studies, and project development in several countries and specifically in South America.

(f) I have read the definition of "qualified person" set out in Canadian National Instrument 43-101 (NI 43-101), and do certify that, by reason of my education, affiliation with a professional association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

(g) I have not visited the Los Azules property and off-site facilities personally; however I have had representatives under my supervision visit the site and relevant facilities as well as the metallurgical laboratories.

(h) I am co-author of the Technical Report titled "CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LOS AZULES COPPER PROJECT" for the Los Azules Project, San Juna Province, Argentina. The effective date of this Technical Report is May 9, 2023.

(i) I have authored or supervised the preparation of and take responsibility for Sections 1, 2, 3, 13, 17, 18.1-3, 18.5-8, 21.1-2, 21.3.2-3, 25.1-3, 25.8.1-2, 26.1-2, and 27 of the Technical Report on the Los Azules Copper Project, San Juan Province, Argentina effective date May 9, 2023.

I have no prior involvement with the property that is the subject of this Report and I hold no interests in, nor do I expect to receive any interests, direct or indirect from McEwen Mining Incorporated or any associated or affiliated company. I am independent of McEwen Mining Incorporated applying the tests set out in Section 1.4 of NI 43-101. I have read NI 43-101 and the Report has been prepared in compliance with NI 43-101 and form 43-101F1.

As of the date of this certificate, to my knowledge, information and belief, this Report contains all the scientific and technical information that is required to be disclosed, related to its intended purposes and my areas of responsibility, to make the Report not misleading or incomplete.

I consent to the filing of this Technical Report with any stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their web sites accessible by the public.

Signed and dated at Greenwood Village, Colorado, USA, on this 31st day of May 2023.

James Sorensen
COPY

James Lynn Sorensen, Q.P.
FAusIMM (Registration #221286)

Original Signature Document Available on Request

CERTIFICATE OF QUALIFIED PERSON

Richard F. Reinke

I, Richard F. Reinke, Professional Geoscientist, do hereby certify that:

1. I am the Qualified Person for Hydrogeology and Hydrology for:

   McEwen Copper Inc.
   Av. Ignacio de la Roza 1240, Capital, San Juan, 5400, Argentina

2. I graduated with a Bachelor of Science in Applied Geophysics from Michigan Technological University, a Master of Science in Geophysics from the University of Wyoming, and a Master of Science in Geology from the Western Michigan University.

3. I am a Registered Member of the Association of Professional Engineers and Geoscientists of Alberta in good standing in the areas of geology, geophysics, and hydrogeology.

4. I have worked as a geophysicist and hydrogeologist for over 35 years. My experience includes design and planning of groundwater and surface water investigations, evaluation and analysis of groundwater and surface data, groundwater modeling, and mine dewatering evaluations and design.

5. I have not had prior involvement with the property that is the subject of the technical report.

6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

7. I am a contributing author for the preparation of the technical report titled "Project Los Azules – NI 43-101 Technical Report" (the "Technical Report") dated effective May 9, 2023, prepared for McEwen Mining Inc.; and am responsible for Section 18.7.

8. I have read National Instrument 43-101 and Form 43-101F1, and Section 18.7 of the Technical Report has been prepared in compliance with that instrument and form.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, Section 18.7 of the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 31 day of May, 2023.

Richard F. Reinke

SRK Consulting (UK) Limited
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Cardiff CF10 2HH
Wales, United Kingdom
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Tel: + 44 (0) 2920 348 150

CERTIFICATE OF AUTHOR

Robert John Bowell

Corporate Consultant (Geochemist)

SRK Consulting (UK) Ltd

Email: rbowell@srk.co.uk

I, Robert J Bowell, a Chartered Professional Chemist, Chartered Geologist and a Certified Professional European Geologist, do hereby certify that:

1. I am responsible for the preparation of the geochemistry section (20.2) of the technical report titled, CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT, PRELIMINARY ECONOMIC ASSESSMENT, LOS AZULES COPPER PROJECT "(the "Technical Report") relating to McEwen Mining Inc., Los Azules Project (the "Project") dated May 31,2023.
2. I visited the Project core site facility in November 2022.
3. I am currently employed as a consulting geochemist to the mining and mineral exploration industry, as a Corporate Consultant Geochemist with SRK Consulting (UK) Ltd, with an office address of 5th Floor Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, UK.
4. I graduated with a Bachelor's of Science Degree, First Class Honors in Geochemistry from Owen's College, Manchester University, Manchester UK, June 1988.
5. I graduated with a Doctorate in Geochemistry from Southampton University, Southampton, UK in June 1991.
6. I am a Chartered Chemist of the Royal Society of Chemistry, London, UK and have been since 1997. Membership number 332782.
7. I am a chartered Professional Geologist for the province of Newfoundland and Labrador, registration number 10809.
8. I am a Chartered Geologist and Certified Professional European Geologist through the Geological Society of London since 1997 and European Association of Professional Geologists since 2000. Registration number 1007245.
9. I am a Fellow of the Institute of Mining, Metallurgy and Materials and have been since 2010.
10. I have been employed as a geochemist in the mining and mineral exploration business and in applied academia, for the past 35 years, since my graduation from university.
11. I have read the definition of "qualified person" set out in National Instrument 43-101 of the *Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. The Technical Report is based upon my personal review of the information provided by the Issuer. My relevant experience for the purpose of the Technical Report is:

- Geochemist, SRK Consulting from 1995 to date;
- Exploration Geochemist with BHP Minerals, Hammersmith, London., 1991-1994;
- Exploration Geologist, Ashanti Goldfields, Ghana, 1988
- Experience in the above positions working with and reviewing sulfide and gold mineralogy and

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URL: www.srk.com
Tel: + 44 (0) 2920 348 150

geology, analysis, geometallurgical testwork geochemical data quality, environmental geochemistry and engineering, assurance and quality control

12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
13. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, for which the omission to disclose would make the Technical Report misleading.
14. I am independent of McEwen Mining applying the test in section 1.5 of NI 43-101.
15. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them or McEwen Mining Inc. for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public.

Dated in Cardiff, United Kingdom, May 31, 2023

("signed")

("sealed")

***Eur.Geol.* Robert Bowell PhD C.Chem. C.Geol P.Geo**
Corporate Consultant (Geochemist)
#332782, Chemist; #1007245, Geologist; 10809 PEGNFL

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CONSENT OF QUALIFIED PERSON

TO: British Columbia Securities Commission
Alberta Securities Commission
Ontario Securities Commission
TSX Venture Exchange

AND TO: McEwen Mining Inc

Dear Sirs/Mesdames:

RE: McEwen Mining Inc. (the "Company")

I, the undersigned, am an author of the technical report prepared in accordance with National Instrument 43-101 – *Standards of Disclosure for Mineral Projects* titled, "CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT, PRELIMINARY ECONOMIC ASSESSMENT, LOS AZULES COPPER PROJECT" dated effective May 9, 2023, report date May 31, 2023 (the "**Report**").

I hereby consent to the public filing of the Report, and the use of extracts from, or a summary of, the Report in the News Release.

I hereby confirm that I have read the News Release and that the News Release fairly and accurately represents the information in the sections of the Report for which I am responsible.

Dated: May 31,2023

Yours truly,

This signature has been scanned. The author has given permission to its use for this particular document. The original signature is held on file.

SRK CONSULTING (UK) LIMITED Co. No: 01575403

Robert John Bowell, PhD C.Chem.C.Geol PGeo
Corporate Consultant (Geochemist)

("Signed") ("Sealed")

CERTIFICATE OF QUALIFIED PERSON

Steven Guy Bundrock, P.Eng.

I, Steven Guy Bundrock, P.Eng., do hereby certify that:

1. I am the Qualified Person for Analysis and Design of the Mine Rock Storage Facility for:

    McEwen Copper Inc.
    Av. Ignacio de la Roza 1240, Capital, San Juan, 5400, Argentina

2. I graduated with a Bachelor of Science in Geological Engineering from Montana Tech of the University of Montana.

3. I have worked as a geotechnical engineer for over 20 years. My experience includes geotechnical site investigation, analysis and design of mine rock storage facilities.

4. I am a licensed professional engineer in British Columbia, Alberta, Manitoba, Nunavut, the Northwest Territories and the Yukon in Canada.

5. I have not had prior involvement with the property that is the subject of the technical report.

6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

7. I am a contributing author for the preparation of the technical report titled "Project Los Azules – NI 43-101 Technical Report" (the "Technical Report") dated effective May 9, 2023, prepared for McEwen Mining Inc.; and am responsible for Sections 18.4.

8. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

9. 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.

Signed and dated this 31 day of May, 2023.

Steven Guy Bundrock, P.Eng.

CERTIFICATE OF QUALIFICATIONS

I, Satjeet Pandher, P. Eng., do hereby certify that:

1. I am currently employed as Senior Mining Engineer by Stantec Consulting Ltd., 1100-111 Dunsmuir St., Vancouver, British Columbia, Canada, V6B 6A3.

2. I graduated with a Bachelor of Science degree in Mining Engineering from the University of Alberta in 2008.

3. I am a member in-good-standing of the Engineers and Geoscientists of British Columbia (Member # 37942).

4. I have worked as an Engineer for 15 years since graduating from my undergraduate degree in Mining Engineering. I have 15 years of project experience in the complete evaluation and analysis of surface mineable projects. This experience includes the evaluation of surficial precious metals, base metals, and sedimentary materials.

5. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101), and past relevant work experience, I meet the requirements to be a “qualified person” for the purposes of NI 43-101.

6. I am responsible for the preparation of portions of Section 16 and 21 of the “Los Azules Project” NI 43-101 Technical Report Preliminary Economic Assessment” dated May 31st, 2023.

7. I have no prior involvement with the property that is the subject of the Technical Report.

8. I have not personally visited the project property.

9. At the effective date of the Preliminary Economic Assessment, to the best of my knowledge, information, and belief, the Preliminary Economic Assessment contains all scientific and technical information that is required to be disclosed to make the Preliminary Economic Assessment not misleading.

10. I am not aware of any material fact or material change with respect to the subject matter of the Preliminary Economic Assessment that is not reflected in the Report, the omission to disclose which makes the Report misleading.

11. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.

Dated May 31st, 2023

“Original Signed and Sealed by Author”

Satjeet Pandher, P. Eng. Senior Mining Engineer

Steven Alan Pozder, PE, MBA, QP
8450 E. Crescent Parkway Ste. 200, Greenwood Village, CO, 80111

1-303-714-4840, ext. 4828
spozder@samuelengineering.com

CANADIAN NATIONAL INSTRUMENT 43-101

CERTIFICATE of QUALIFIED PERSON

I, Steven Alan Pozder, (PE Colorado #29144), do hereby certify that as the co-author of this Technical Report (Report) on the Los Azules Copper Project, San Juan Province, Argentina, titled "CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LOS AZULES COPPER PROJECT" dated May 31, 2023 with an effective date of May 9, 2023, I hereby make the following statements:

(a) I am a Mechanical Engineer and Senior Director practicing at Samuel Engineering, Inc., 8450 E. Crescent Pkwy, Ste. 200, Greenwood Village, CO 80111, USA.

(b) I am a graduate of the University of Denver, Denver, Colorado, United States of America with a Bachelor of Science in Engineering degree (Mechanical) conferred in 1988. I am a graduate of the University of Denver with an M.B.A. in General Business in 1994.

(c) I am registered as a Professional Engineer (P.E.) with the State of Colorado, Registration Number 29144.

(d) I have practiced my profession as a Mechanical Engineer and Project Manager in mineral processing and mining for over 34 years. My relevant experience for the purpose of the Technical Report is:

- I have worked as a consulting engineer on mining projects in roles such a mechanical engineer, project engineer, area manager, study manager, and project manager. Projects have included Scoping Studies, Prefeasibility Studies, Feasibility Studies, basic engineering, detailed engineering and startup and commissioning of new projects.
- In engineering positions, I have estimated and reviewed capital and operating costs and completed economic analyses including power requirements, reagent costs, labor requirements and costs, etc. for 27 years.

(e) I have read the definition of "qualified person" set out in Canadian National Instrument 43-101 (NI 43-101), and do certify that, by reason of my education, affiliation with a professional engineering association (as defined by NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

(f) I have not visited the Los Azules property and off-site facilities.

(g) I am co-author of the Technical Report titled "CANADIAN NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT PRELIMINARY ECONOMIC ASSESSMENT LOS AZULES COPPER PROJECT" for the Los Azules Project, San Juna Province, Argentina. The effective date of this Technical Report is May 9, 2023.

(h) I have authored or supervised the preparation of and take responsibility for Section 22 of the Technical Report on the Los Azules Copper Project, San Juan Province, Argentina effective date May 9, 2023.

I have no prior involvement with the property that is the subject of this Report and I hold no interests in, nor do I expect to receive any interests, direct or indirect from McEwen Mining Incorporated or any associated or affiliated company. I am independent of McEwen Mining Incorporated applying the tests set out in Section 1.4 of NI 43-101. I have read NI 43-101 and the Report has been prepared in compliance with NI 43-101 and form 43-101F1.

As of the date of this certificate, to my knowledge, information and belief, this Report contains all the scientific and technical information that is required to be disclosed, related to its intended purposes and my areas of responsibility, to make the Report not misleading or incomplete.

I consent to the filing of this Technical Report with any stock exchange or other regulatory authority and any publication by them, including electronic publication in the public company files on their web sites accessible by the public.

Signed and dated at Greenwood Village, Colorado, USA, on this 31st day of May 2023.

Steven Alan Pozder, PE, MBA, Q.P.
PE (Colorado Registration #29144)