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El Pilar Project

S-K 1300 Technical Report Summary

Feasibility Study

Sonora, Mexico

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EL PILAR PROJECT
S-K 1300 TECHNICAL REPORT SUMMARY
FEASIBILITY STUDY

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1 EXECUTIVE SUMMARY

1.1 OVERALL REPORT SUMMARY

Southern Copper Corporation (SCC), plans to develop the 100% wholly owned El Pilar Oxide Copper Project ("El Pilar Project" or "the Project") into one of Sonora, Mexico's new copper mines. The El Pilar Project is a greenfield project with no existing mining infrastructure or equipment at the property. The Santa Cruz River flows year round 2 km south of the Project. The railway of Ferrocarril Mexicano follows this river basin, as do local roads. Power is available in Nogales, Mexico approximately 28 km to the northwest of the Project.

Since the 2011 M3 Feasibility Study, the parent company of Stingray has changed. SCC purchased Stingray Copper, 100% owner of Recursos Stingray de Cobre, in 2015. Significant changes have been made to the design, in particular the sulfur burning plant and power plant have been eliminated and sulfuric acid will be delivered to site instead, and the crusher plant has been removed as well, two settlers were added to the solvent extraction plant.

The basis of this Technical Report Summary (TRS) at a Feasibility Study level, prepared by M3 Engineering and Technology Corp. (M3) is the use of owner's mining equipment to mine ore and stacking the ROM ore into the heap leach facility (HLF) with the same haul trucks, followed by solvent extraction and electrowinning (SX-EW) processing facilities as outlined in the process flow drawing developed in this Study. Alberto Bennett, P.E. of M3 is the Qualified Person responsible for the technical content of this TRS except where reliance upon other Qualified Persons is expressly indicated.

As estimated by Golder, Measured, Indicated and Inferred Mineral Resources are presented in Table 1-1.

Table 1-1: El Pilar Mineral Resource (Exclusive of Mineral Reserves)

Classification	Tonnes (Mt)	Total Cu (%)	Soluble Cu (%)
Measured	2.2	0.20	0.10
Indicated	81.3	0.18	0.08
Total Measured and Indicated	**83.4**	**0.18**	**0.08**
Inferred	88.6	0.12	0.06

Notes:

- The Mineral Resource estimates were prepared by Ronald Turner, P.Geo. (who is the independent Qualified Person for these Mineral Resource estimates), reported using the S-K 1300 Definition Standards adopted December 26, 2018.
- Tonnages are rounded to the nearest 100,000 tonnes.
- Resources are reported on a break-even profit basis and constrained within a pit shell outlined using a Cu price assumption of $3.795 / lb.

Copper mineralization remains open and continues to the south. Proven and Probable ore reserves planned for mining over a period of 16 years are calculated by Golder to be 317 million tonnes (Mt) averaging 0.25% total Cu (TCu). The mine-life waste to ore stripping ratio is currently estimated at 1.86:1.

The project mine design rate reaches its maximum of 27 Mt of ore per year. The Project is expected to produce 940 million pounds (Mlbs) of copper cathode over the 16-year life of mine. The cathode product should meet ASTM 115 Grade 1 copper cathode specifications (99.99+% copper) and will be purchased FOB at the El Pilar mine site.

For this TRS, Golder developed a 16 year mine schedule. At the process plant design rate, the Project is expected to produce 59 million pounds per year, which can be achieved during the first 6 years of the Project. For economical and practical reasons, the maximum mine ore capacity was designed to be 27 Mt per year. At this capacity, the average copper cathode production would be approximately 59 million pounds per year over 16 years. This production schedule forms the basis for the financial model. SCC will make a strategic decision on whether to increase mine capacity after Year 6, or accept production at a rate lower than design, depending on market conditions at that time.

The capital cost for the El Pilar Base Case Project is estimated to be US$364.9 million, including mining equipment and process facilities. Sustaining capital of $144.6 million is estimated for mining and $230.2 million for the heap leach over the life of mine, which includes heap leaching development and mine equipment.

The operating cash cost for the Project is calculated at $1.84 per pound of copper. At a base case with a copper price of $3.30 per pound and after taxes, the El Pilar Project has an IRR of 9.88%, and an after-tax net present value (NPV), discounted at 8%, of $54.2 million. The estimated payback of capital is 7.0 years.

Table 1-2: El Pilar Financial Summary

Financial Summary	
CAPEX, Processing Facility ($000)	$261,829
CAPEX, Mine ($000)	$103,070
Total CAPEX ($000)	**$364,899**
Sustaining Capital, Processing Facility ($000)	$230,201
Sustaining Capital, Mine ($000)	$144,631
Total Sustaining Capital ($000)	**$374,832**
Production Cost ($/lb Cu)	$1.84
NPV @ 8% ($000)	**$54,180**
IRR %	**9.88%**
Payback (Years)	**7.0**
Cu Reserves (kt)	**317,465**
Cu Cathode Produced (klbs)	**940,777**
Gross Revenue ($000)	**$3,104,566**

1.2 INTRODUCTION SUMMARY

SCC engaged M3 to prepare a TRS at a Feasibility Study level of the El Pilar Oxide Copper Project to assess the viability of commercial operation. Alberto Bennett, P.E., of M3 is the principal Qualified Person (QP) and author of the TRS. This Feasibility Study will be the cornerstone for project planning and construction of a copper mining operation at El Pilar.

SCC plans to develop the Project in Sonora, Mexico as an open pit mining heap leach project with a solvent extraction and electrowinning processing plant (SX-EW). The El Pilar deposit contains 317 Mt of ore with a copper content of 0.25%, containing a total of 1.7 billion pounds of copper, of which 940 Mlbs are calculated as recoverable. The process plant will be designed to produce 59 Mlbs of copper cathode per year. SCC plans to begin engineering and then construction immediately upon completion of project approval, surface acquisition and project financing.

1.3 PROPERTY DESCRIPTION AND LOCATION

The El Pilar Property is located in north central Sonora about fifteen (15) km south of the international border with United States of America as shown in Figure 1-1. The property is situated within lands of Ejido Miguel Hidalgo (also referred to as San Lazaro), in the Santa Cruz Municipality. The property is situated between UTM coordinates 3,446,000N to 3,455,000N and 526,800 E to 534,700 E.

Figure 1-1: El Pilar Location Map

The El Pilar property comprises 9,571.3771 hectares in nineteen concessions located in the state of Sonora, Mexico (See Section 3.1 for the Concession Map). These concessions are wholly owned by Recursos Stingray de Cobre S.A de C.V., the wholly owned Mexican subsidiary of SCC.

The status of mining concessions is outlined in Section 3. A total of 1,926 hectares of surface rights have been successfully negotiated with the Ejido Miguel Hidalgo, which allows for all required land ownership rights needed for project development.

1.4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

The El Pilar deposit is located at the southwest margin of the Patagonia Mountains near the base of a mountain range. The topography near the deposit permits sufficient surface space for a mining operation, leaching pads, waste disposal areas, and other facilities.

The property can be reached by road from Hermosillo, Sonora in Mexico and from Tucson, Arizona in the USA. The route from Hermosillo to Miguel Hidalgo takes about 3 1/2 hours of driving time. The route from Tucson to Miguel Hidalgo is currently a two-hour drive. The site is a green-field mining site with no existing infrastructure. Experienced mining personnel and related contractors are available within driving distance.

The project area climate allows year-round mining and processing operations. The climate is classified as semidry with a summer rainy season and limited rains the rest of the year. The average annual temperature is 17.8°C. The project site elevation ranges between 1,250 and 1,425 m above mean sea level (AMSL). The hottest months are June to September. Most of the rainfall occurs in the form of summer storms during the months of July, August and September. Mean annual precipitation is 543.6 mm.

A power line is located 3 km to the south, in the village of Miguel Hidalgo (San Lazaro) where SCC has an office and warehouse facilities, but the Project will require the construction of a high voltage power line from the site to connect with the high voltage power lines accessible in Nogales, which is 28 km northwest of the property. A railroad is located 3 km south of the deposit. Construction of a new railway spur of approximately 4 km in length is planned for the delivery of molten sulfur or sulfuric acid.

1.5 HISTORY

The history of exploration in the El Pilar area before 1992 is not well documented. However, it is known that in the 1970's, Cyprus Amax Minerals had claims in the area. According to verbal communications with local people and field evidence, it is clear that a geophysical survey and a few drill holes were completed in the northern part of the El Pilar Discovery area. Results of both the drilling and the geophysical survey are unknown.

Before Normex's direct involvement, there was no mineral resource known, only small old workings following narrow veins with erratic copper and molybdenum mineralization. Other small workings in the area were dug by local miners (gambusinos), searching for azurite and turquoise.

In 1992, Normex began acquiring ground at El Pilar. From 1992 to 1997, Normex carried out an exploration program that included regional mapping, sampling and limited geophysical surveying. From 1998 to 1999, Freeport Copper, under an agreement with Normex, carried out an exploration program that included regional mapping, rock and vegetation sampling, and some geophysical surveys. After the exploration agreement with Freeport ended in 2000, Normex continued with a short CSAMT survey and soil sampling exploration program. In addition, Normex carefully assessed the data generated by Freeport, emphasizing lab checks for validating the copper assays from the gravels. From September 2000 to March 2001, Normex completed a drill program. Following Normex's drilling campaign, resource calculations were undertaken in 2001, and again in 2003. An inferred mineral resource estimate was completed by Magri in 2003. In April 2007, a Form 43-101 F1 Technical Report on the El Pilar Property was completed by Gary Woods, P. Geo.

Preliminary metallurgical test work was carried out at Falconbridge's Lomas Bayas mine in Chile in 2003 and 2004 and by METCON in 2001 and 2005. This preliminary work included bottle roll testing and column leach tests. A scoping level economic evaluation was prepared by AMEC E&C Services Inc. of Phoenix, Arizona in 2005-2006 for Normex. The study concluded that the El Pilar Project demonstrated positive economics and was worthy of further assessment. Stingray acquired the property at that juncture. Mercator Minerals Ltd. (ML) purchased Stingray Copper in 2009. SCC acquired Stingray in 2015.

In April 2011, an NI 43-101 compliant Feasibility Study was filed on behalf of Stingray by M3. The NI 43-101 included mineral resource and a mineral reserve estimates done independently by Mike Hester of IMC in Tucson. The work carried out for the 2011 Feasibility Study served as a basis for work in the 2022 Technical Report Summary.

1.6 GEOLOGICAL SETTING & MINERALIZATION

The deposit is located along the southwest flank of the Patagonia Mountains. The geology of the El Pilar property consists of Precambrian intrusive rocks overlain by Paleozoic sedimentary rocks. These units are overlain by Tertiary sedimentary rocks. Intrusives of granitic to monzonitic composition with some pegmatitic and aplitic facies intrude all the older units. Tertiary and Quaternary alluvial fan and alluvial wash sediments cover the flanks of the ranges and the intervening valleys.

The El Pilar copper deposit occurs within unconsolidated, poorly sorted, poorly bedded, proximal facies alluvial wash deposits that are overlain by dissected younger alluvial fan deposits. The copper bearing sediments at El Pilar are comprised solely of alluvial wash gravels deposited into a paleo topographic range-front depression. At the northern boundary of the deposit, these basin-fill sediments are juxtaposed against unmineralized Precambrian granitic rocks by an east-west to northwest-trending, south dipping zone of faulting and hydrothermal brecciation. The faulting is of unknown displacement. The breccia zone comprises a multi-stage, highly silicified, copper mineralized hydrothermal breccia that is up to 100 m wide and 600 m long. The El Pilar copper deposit is interpreted to have been formed by erosion of this breccia over time into the range-front topographic depression.

Mineralization predominantly consists of the copper oxide mineral chrysocolla, which occurs as coatings on clasts of highly silicified breccia and as grains in the sedimentary gravel matrix. The main gravel sequence that hosts copper mineralization consists of poorly consolidated angular to sub-rounded fragments of breccia, intrusive rocks and minor volcanic fragments cemented in a sandy matrix. These productive gravels, referred to as Quaternary Alluvial Wash Deposits Upper (Qwu) and Quaternary Alluvial Wash Deposits Transitional (Qwt), range from 30 m to 180 m in thickness. The main zone of copper mineralization occurs within a southwest/south trending channel that extends for more than 2 km.

El Pilar lies within the Sonora-Arizona Porphyry Copper Province, about 45 km northwest of the Buenavista del Cobre copper mine owned by SCC. Buenavista del Cobre is the largest porphyry copper deposit in Mexico and one of the largest in the world. Both El Pilar and Buenavista del Cobre are situated in a highly prospective belt of copper deposits that range from La Caridad in the south through to central Arizona. The main types of copper deposits in this belt are related to porphyry copper systems and mineralization typically occurs in hydrothermal breccias pipes and as disseminations and stockworks. The El Pilar property hosts an unusual gravel hosted or transported copper resource that is atypical for the area. Copper at El Pilar is hosted by range front alluvial wash deposits with clasts of intrusive, porphyry and highly silicified rock derived from a proximal, exposed hydrothermal breccia zone. The mineralization is interpreted to have been derived at least in part from that. Reconstruction of events suggests that the breccia was mechanically weathered and eroded, transported and deposited in a channel and alluvial fan sequence that overlies a lower more indurated alluvial wash unit (Qwl).

1.7 EXPLORATION

1.7.1 Surface Exploration

Normex (Noranda) and partners began exploration at the property in the late 1990s, conducting geophysical and geochemical exploration programs over the property. A 600-tonne bulk sample was collected in 2010 for run of mine (ROM) metallurgical crib and column testing. Sample was collected from the only outcrop of mineralized Qwu in the Project area.

1.7.2 Drilling

The Stingray geologic and drill hole database was provided to the QP for review and included 316 holes that represented 71,825 m of drilling.

Pre-Stingray drilling amounts to 61 holes representing 11,988 m that were drilled by Freeport and Noranda between 1998 and 2004. All but 7 of the drill holes were fully cored, while the other 7 were reverse circulation, drilled during Freeports 1998 reconnaissance exploration program. In terms of meters of drilling, this represents 17% of the total drilling to date.

During 2007 and 2008, Stingray drilled 194 HQ core holes representing 40,822 m. In 2016, Stingray drilled an additional 61 PQ (85-mm core diameter) core holes, totaling 19,015 m. However, 14 of the 61 drill holes were used for condemnation drilling surrounding the main deposit, and 5 for regional exploration. None of these 19 holes were included in the model, or in the drill hole database provided to Golder. The Stingray drilling represents 83% of the drill hole database in terms of meters drilled. The purpose for the drilling was to: 1) validate the Freeport and Noranda drilling, 2) extend the resource base, and 3) collect samples to perform metallurgical testing over the entire deposit.

No additional drilling was conducted at El Pilar from 2017 through 2021. Geotechnical rock mass and structural data was collected from six drill holes located along the South, East, and North boundaries of the proposed pit in the spring of 2008. Core hole locations were selected in cooperation with SCC personnel to provide representative samples from the principal geologic units that will be exposed in the ultimate pit slopes. Core holes were inclined to intersect the walls of the proposed Year 12 pit design.

1.8 SAMPLE PREPARATION, ANALYSES AND SECURITY

All analytical work done for Noranda was performed by the Bondar Clegg Lab in Vancouver, B.C., following sample preparation by Bondar's affiliate in Hermosillo. For sample security, after splitting of the core, the samples were bagged by project geologists and driven to the sample preparation laboratory in Hermosillo by Noranda personnel. Sample preparation and shipment of the pulps to the Vancouver analytical laboratory was conducted by lab personnel. Noranda personnel were not involved in sample preparation.

Sample preparations and analysis for Stingray drilling was performed at two different laboratories for the 2007 and 2008 drilling (ALS Chemex and METCON Laboratory). About 60% of the drilling analyses were done by ALS Chemex laboratories and about 40%, particularly the drilling used for metallurgical testing, were done by the METCON laboratory in Tucson, Arizona. Sample preparation for ALS Chemex was done at their Hermosillo, Mexico facility, after which the pulps were shipped to Vancouver for analysis.

Sample preparations and analysis for the 2016 Stingray drilling was performed at three different laboratories. About 50% of the drilling analyses were done by Inspectorate laboratories of Hermosillo, Mexico, a contract laboratory for Bureau Veritas. About 37% were completed by Skyline laboratory in Tucson, AZ, with an additional 13% at Copper State Analytical Laboratory in Prescott, AZ. Sample preparation for Bureau Veritas was done at their Hermosillo, facility, after which the pulps were shipped to Vancouver for analysis.

As part of the QA/QC work several hundred pulps were analyzed at both the ALS Chemex and METCON laboratories in 2007 and 2008 and at the Bureau Veritas and Skyline Laboratories in 2016.

The four main laboratories used by Stingray were registered to at least ISO 9001:2000, with most to ISO 9001:2015 standards and are ISO 17025 accredited. The QP could not confirm recent certification for Copper State Analytical Laboratory.

Total copper analysis was done by a four-acid digestion, nitric, perchloric, hydrofluoric, and hydrochloric acid, followed by analysis with Inductively Coupled Plasma - Atomic Emission Spectroscopy (ICP-AES).

Stingray conducted a comprehensive program to assure the quality of its sample preparation and analysis (assaying). Standard, blank and duplicate samples from the El Pilar exploration program were subjected to quality assurance. The standard samples were prepared at METCON using interval samples provided by Stingray. The blank sample came from a monzonite outcrop at the El Pilar and is not anomalous in copper. The standard, blank and duplicate samples were generally inserted (used) as follows:

- Duplicate samples were inserted every ten samples.
- Blank samples were inserted every 20 intervals.
- Standard samples were inserted every ten interval samples.

1.9 DATA VERIFICATION

1.9.1 Mineral Resources

The QP was provided with the compiled SCC database, in Excel file format, which included survey information, downhole geological units, sample intervals and analytical results. Stingray provided the QP with a 2 m resolution topographic surface prepared from the 2008 survey, which was reviewed for consistency against the drill hole collars.

Drill hole data for the El Pilar project comprised 316 drill holes (7 RC and 309 core drill holes) totaling 71,825 m of drilling and containing 25,540 analytical samples. Supporting documentation for the Stingray and Noranda drilling data included laboratory certificates, descriptive logs, collar survey reports, and internal report documents.

Data validation was performed on the drill hole database records using available underlying data and documentation including, but not limited to, original drill hole descriptive logs, and laboratory assay certificates. Database assay values for every sample were visually compared to the laboratory assay certificates to ensure the tabular assay data was free of errors or omissions. Drill hole recovery data was also reviewed as well as QA/QC results.

The Mineral Resource QP completed a site visit on August 23, 2021. The purpose was to review the project site, geology, current, and previous exploration methods, data collection procedures and sample storage conditions. Specific intervals of cores were selected by the QP and reviewed on site.

The Golder QP, by way of the verification process, is of the opinion the data collected on the Project was generated with proper industry standard procedures, were accurately transcribed from the original source and were suitable to be used for the purpose of preparing geological models and Mineral Resource estimates.

1.9.2 Mineral Reserves

The Golder QP responsible for Mine Planning and Mineral Reserve estimates, is of the opinion the data used in the preparation of the mine design and resultant Mineral Reserve estimate, including geotechnical design criteria, cut-off grade calculations, mine modifying factors, production schedule, manpower and equipment estimates, and other test data underlying the information, contained in the written disclosure presented in this TRS are to a PFS level.

1.10 MINERAL PROCESSING AND METALLURGICAL TESTING

The El Pilar mine is an open pit, oxide copper mine, whereby copper is recovered from the heap leach pad via the application of sulfuric acid-bearing raffinate pumped from a raffinate pond. The pregnant leach solution (PLS) is then collected and copper is recovered in the plant via the process of solvent extraction and electrowinning (SX-EW) in a two-stage process that first extracts and upgrades copper ions from low-grade leach solutions into a concentrated electrolyte, and then deposits pure copper onto cathodes using an electrolytic procedure. Extensive metallurgical work has been conducted historically to best determine how much copper will be recovered from the ores and how much acid will be consumed by the process.

In 2010 and 2011, ML conducted additional metallurgical testing on mineralized material from El Pilar for the following reasons:

1. To evaluate the metallurgical viability of using run of mine (ROM) leaching, as compared to the higher cost crushing program proposed by Stingray in 2009;
2. To develop additional metallurgical copper recovery and acid consumption information from copper-bearing materials with a wide range of copper acid solubility assays;
3. To investigate the metallurgical recovery of copper and consumption of acid from and by lower grade mineralized material near or at the lower end of copper cut-off grade.

The primary objective of points #2 and #3 above was to attempt to develop a grade recovery relationship at El Pilar based on a normalized equation using the soluble copper ratio SCu/TCu. A wide range of solubility ratios were used to predict a more accurate recovery basis than was done previously.

- ROM particle size distribution at 80 percent passing 1 ¼" did not negatively impact copper extraction and gangue acid consumption compared to 80 percent passing 37.5 mm, 19 mm and irrigation flow rates of 6.1 and 7.8 L/h//m^2, respectively, using a sulfuric acid cured dosage of 4 kilogram per tonne of material.
- The copper extractions on the two composites, C-01 and C-02 ranged from 67.9% to 63.7% and gangue acid consumption ranged from 5.7 to 5.0 kilogram per kilogram of copper extracted (kg/kg Cu), and 20.7 to 17.7 kilogram acid per tonne of material (kg/ tonne) after a total leach cycle of 166 days (including 7 days of cure, 150 days of leach, 7 days of wash and 2 days of drain cycles).
- The highest copper extraction of approximately 71% was achieved at the size distribution of minus 3/4 inches on the crib C-01.
- The lowest copper extraction of approximately of 43.5% was achieved at the size distribution of plus 3/4 inches on the crib C-02.
- Percolation problems were not observed in the cribs during the leach cycle.
- There is a good correlation between the calculated head and assay head for copper and iron.

Combined metallurgical tests from the Stingray 2009 program and from the bulk sample and additional 13 column composites tested in 2010 and 2011 result in the following conclusions:

1. El Pilar copper deposit consists of gravels that are poorly cemented and disaggregate almost completely into a "pre-crushed" size distribution on mining.
2. As a result of the above and based on the crib results, ROM leaching should attain recoveries comparable to the column test averages.
3. A 120 day leach cycle should be assumed initially, although real operating conditions may show that a different leach cycle is viable. Stacking plan must be adapted accordingly to accommodate these cycles.
4. Copper recoveries at El Pilar are at least initially a function of copper solubility, although mineralogical studies suggest that over longer periods of time a considerable amount of the residual copper may be recovered.
5. There is a grade recovery relationship for 120 days of leaching as defined by the formula, Recovery % (of TCu) = 33.49ln(X) + 79.49, where X is the Ratio (%ASCu/%TCu).
6. An initial precure rate of about 10 kg per tonne acid is recommended.
7. LOM acid consumption should average approximately 22 kg acid per tonne of ore.

Several potential project upsides are suggested by the metallurgical testing program and results. While these potential project enhancements are not fully quantified in this study, they are significant and include the following:

1. Using a ~10 kg per tonne precure, rather than the 4 kg per tonne precure used in the metallurgical tests, will likely result in faster copper recovery rates that could positively impact project economics and allow for a shortened leach cycle, as well as potentially better copper recoveries over the life of mine.
2. Curing method will also be important, since ROM ore will be used. Trickle down curing will cause a lag in recovery, a thorough wetting of the ore will be crucial to achieve the projected recovery curve.

1.11 MINERAL RESOURCE ESTIMATES

This sub-section contains forward-looking information related to Mineral Resource estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including geological and grade interpretations and controls and assumptions and forecasts associated with establishing the prospects for economic extraction.

Table 1-3 below presents the Mineral Resource estimates for the El Pilar Project as estimated by Ronald Turner (MAusIMM, CP (Geo)), the QP responsible for these estimates. They are exclusive of the Mineral Reserves. The effective date of the Mineral Resource Estimate is December 31, 2021.

Table 1-3: El Pilar Mineral Resource (Exclusive of Mineral Reserves)

Classification	Tonnes (Mt)	Total Cu (%)	Soluble Cu (%)
Measured	2.2	0.20	0.10
Indicated	81.3	0.18	0.08
Total Measured and Indicated	**83.4**	**0.18**	**0.08**
Inferred	88.6	0.12	0.06

Notes:

- The Mineral Resource estimates were prepared by Ronald Turner, P.Geo. (who is the independent Qualified Person for these Mineral Resource estimates), reported using the S-K 1300 Definition Standards adopted December 26, 2018
- Tonnages are rounded to the nearest 100,000 tonnes.
- Resources are reported on a break-even profit basis and constrained within a pit shell outlined using a Cu price assumption of $3.795/lb.

It is the QP's opinion that the Mineral Resource block model is representative of the informing data, and that the data is of sufficient quality to support the 2021 Mineral Resource Estimate.

The 2021 Mineral Resource Estimate may be materially impacted by any future changes in the break-even cut-off grade, potentially resulting from changes in mining costs, processing recoveries, or metal prices or from changes in geological knowledge as a result of new exploration data.

1.12 MINERAL RESERVE ESTIMATES

This sub-section contains forward-looking information related to Mineral Reserve estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including Mineral Resource model tonnes and grade, modifying factors including mining and recovery factors, production rate and schedule, mining equipment productivity, commodity market and prices and projected operating and capital costs.

Table 1-4 below summarizes the El Pilar Mineral Reserve as estimated by Danny Tolmer, P. Eng., the Qualified Person responsible. At an economic block cutoff utilizing the metallurgical recovery curve, the Proven and Probable Mineral Reserves are 317 Mt at 0.25% total copper for 1.74 billion pounds of contained copper. The effective date of the Mineral Reserve estimate is December 31, 2021.

Table 1-4: El Pilar Mineral Reserves

Classification	ROM Ore Tonnes (Mt)	Total Cu (%)	Cu Recovery	Contained Cu (kt)	Recovered Cu (Mlbs)
Proven	63	0.266	60%	168	221
Probable	254	0.245	52%	623	720
Proven and Probable	317	0.249	54%	790	941

Note:
1. Mineral Reserves are mined tonnes and grade; the reference point is the leach pad and includes considerations for operational modifying factors such as loss (2%) and dilution (3%).
2. The recovered copper estimate utilizes the recovery discussed in Section 12.2.2 (Cu Rec % = 0.3349 x LN(Cu_Ratio) + 0.7949).
3. Numbers have been rounded to reflect appropriate accuracy.

1.13 MINING METHODS

This sub-section contains forward-looking information related to Mineral Reserve Estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including Mineral Resource model tonnes and grade, modifying factors including mining and recovery factors, production rate and schedule, mining equipment productivity, commodity market and prices and projected operating and capital costs.

The Mineral Reserve Estimate of 317 Mt was scheduled to develop an annual forecast of projected ore tonnes and copper grade. Conventional open cut mining methods deploying shovels, trucks, loaders and support operations were assumed for the Project. The mine schedule targeted the delivery of ROM to the leach pads to achieve an assumed maximum production limit of 70 Mlbs of copper per year.

The mine production schedule is shown in Table 1-5 below.

Table 1-5: Mine Production Schedule

Annual Mine Production Schedule								
Mining Period	ROM Ore (Ktonnes)	Total Copper (%)	Waste (Ktonnes)	Total (Ktonnes)	Waste: Ore Ratio	Copper Recovery (%)	Contained Copper (Ktonnes)	Recovered Copper (MPounds)
PP	12,556	0.29%	21,492	34,048	1.71	69.0%	36.6	55.8
Y1	14,736	0.32%	24,140	38,876	1.64	67.3%	46.6	69.2
Y2	20,843	0.23%	17,720	38,562	0.85	63.5%	47.5	66.5
Y3	21,469	0.23%	28,336	49,805	1.32	58.8%	49.5	64.1
Y4	20,280	0.26%	32,927	53,207	1.62	57.6%	53.1	67.5
Y5	22,725	0.24%	34,352	57,077	1.51	55.1%	55.0	66.8
Y6	22,303	0.24%	31,589	53,892	1.42	57.2%	52.7	66.5
Y7	18,891	0.27%	29,631	48,522	1.57	57.5%	51.8	65.7
Y8	16,894	0.29%	34,421	51,315	2.04	55.8%	48.8	60.0
Y9	20,008	0.24%	28,245	48,253	1.41	50.5%	47.7	53.1
Y10	20,851	0.19%	35,826	56,677	1.72	51.7%	39.7	45.2
Y11	23,791	0.22%	25,230	49,021	1.06	50.5%	51.9	57.7
Y12	19,809	0.26%	25,855	45,665	1.31	45.3%	50.5	50.5
Y13	26,814	0.23%	9,233	36,046	0.34	45.2%	61.4	61.2
Y14	24,534	0.28%	3,765	28,299	0.15	44.5%	69.2	68.0
Y15	10,960	0.26%	2,095	13,054	0.19	37.0%	28.2	23.0
Total/Avg.	**317,465**	**0.25%**	**384,857**	**702,321**	**1.21**	**54.0%**	**790**	**941**

Note: PP denotes pre-production year.

1.14 PROCESSING AND RECOVERY METHODS

The following items summarize the process operations required for copper extraction and recovery from El Pilar Project:

- ROM ore is loaded in haul trucks, transported to the pad and stacked in in 6 m high lifts.
- After the irrigation piping system is placed on top of each lift, ore is cured for 7 days using a high acid concentration raffinate solution.
- Acid bearing raffinate solution is used for leaching during the standard cycle; pregnant solution is collected by the perforated pipe system at the bottom of the leach pad and transferred into the PLS pond.
- PLS (aqueous) is transferred by gravity to the solvent extraction (SX) plant where it is mixed with an organic solution comprising a mix of solvent such as a kerosene or equivalent and an extractant reagent with an affinity to copper. Copper in solution is then transferred from the aqueous phase to the organic phase. The resulting aqueous solution low in copper (raffinate) is returned to the leach pad. The loaded organic is transferred to the stripping stage.
- Copper is stripped from the organic phase using a high acid concentration aqueous solution, lean electrolyte. Copper is transferred from the organic phase back to the aqueous phase. The resulting aqueous phase is called Rich Electrolyte.
- Raffinate solution from SX is transferred to the raffinate pond, acid content in the raffinate is adjusted to the desired set point and is pumped to the leach pad irrigation system.

- Rich electrolyte is transferred to the electrowinning cells (EW), where copper from the solution is deposited onto stainless steel sheets by means of a direct current system that includes transformer-rectifiers, non-soluble lead anodes, bus bar, etc. The rich electrolyte acts as the media required to allow the flow of current. Copper cathodes are harvested every 7 days. The resulting aqueous phase from EW is called lean electrolyte which is returned back to the stripping settlers for reuse.
- Copper cathodes are stripped using a cathode stripping machine, copper is washed, separated from permanent stainless steel cathode, sampled, corrugated using a press, and bundled as final product.
- Auxiliary process facilities include heat exchangers, water boiler for optimum temperature of rich electrolyte, a centrifuge for organic recovery from crud, electrolyte filters, reagents preparation and addition, etc.

Standard heap leaching technology, extensively used throughout the international mining community, is being proposed for copper recovery. The mine is expecting to process a total of 317 Mt of copper ore bearing material at the proposed heap leach facility located about 0.5 km north of the open pit. The leach pad and ponds will consist of the following:

- A heap leach pad, constructed in five phases to accommodate a total of 317 Mt of ROM ore. The leach pad will be lined using a composite liner system consisting of prepared subgrade overlain by a compacted clay soil-liner (CCL) or a geosynthetic clay liner (GCL) overlain by a 2 millimeters thick textured or smooth linear low density polyethylene (LLDPE) geomembrane liner.
- A solution collection system consisting of dual-walled perforated corrugated polyethylene (PCPE) pipes placed on top of the primary liner. The collection system is and covered with a 0.5 meter (m) thickness of liner cover fill with 1.0 m thick liner cover over the main solution collection pipes.
- A PLS Pond to collect and manage solution flows from the heap leach pad to the SX processing plant.
- A Raffinate Pond to collect and manage solution flows from the SX processing plant back to the heap leach pad.
- The Solution Ponds described above shall have primary and secondary high-density polyethylene (HDPE) geomembrane liners with an HDPE geonet in between the geomembranes. The geonet will collect any leaks through the primary HDPE geomembrane liner. Solution leaks will be conveyed by gravity flow to a leak detection sump with drain gravel, which shall be monitored on a regular basis.
- Two Contingency Ponds shall be constructed as emergency overflow ponds that have been designed for total containment volume for a maximum recorded design storm of 132 mm and a 24-hour drain down duration in the event of loss of pumping capacity. These Contingency Ponds will have a single synthetic 2.0-mm HDPE liner installed for primary containment and a GCL as the secondary containment. These ponds will not have a leak detection system as this pond is intended to be managed and be empty except in the case of emergency solution management.

1.15 INFRASTRUCTURE

Project infrastructure beyond the processing plant includes the following:

- Site Access Roads
- Water
- Power Lines
- A new Railroad Spur

The copper production process will require 700 to 1,700 metric tons per day (t/d) of sulfuric acid to support the leaching of copper from the ore.

The El Pilar property can be reached by road from Hermosillo, Sonora in Mexico and from Tucson, Arizona in the USA. The route from Hermosillo to Miguel Hidalgo takes about 3 1/2 hours of driving time. The route from Tucson to Miguel Hidalgo is currently a two hour drive and utilizes a paved road from Nogales, Sonora to Miguel Hidalgo (30 km).

A main access road to the plant site from the main Nogales access road on the west side of the Project will be by way of a 6.5 km long gravel road that will be constructed early in the project development schedule. The project access road includes a crossing over the Santa Cruz River bed by way of a concrete dip, with hydraulic/drainage structures as required.

Based on the hydrological study conducted by IDEAS and the process water balance, the Project includes three water wells to supply the necessary water volume for processing and services. Three wells have been drilled, cased and tested; these yield more than the 3.5 Mm^3/yr required. All wells are located within the property and are located a relatively short distance from the facilities (about 2.5 km).

Power will be supplied to the project area via a 115 kV transmission line from a substation located 21 km south of Nogales, Mexico. The substation is 30 km west of the project area. The substation is owned and operated by Comisión Federal de Electricidad (CFE), which has confirmed power availability and provided an area next to the substation for the installation of switchgear and instrumentation. The power line will be 31 km long and built with dip galvanized structural steel towers, except for along urban areas where steel tapered poles may be used. The line will have capacity to supply all of the power requirements of the Project estimated at a maximum of ~20 MW.

As part of the infrastructure, a railroad spur will be constructed to the plant site. The purpose of the spur is to provide a safe, economic and efficient access to sulfuric acid deliveries by rail. The rail spur will access the property from the Ferromex rail system located on west side of the property, about 3.8 km distance from the sulfuric acid unloading station. Rail facilities will allow for unloading and parking at least 18 railcars, with deliveries expected on a weekly basis.

1.16 MARKET STUDIES AND CONTRACTS

Cathode sales from El Pilar Project will be negotiated directly by SCC's corporate commercial area that currently has an important worldwide share in the copper market, commercializing 2.1% of the total copper cathodes in the world through a corporate strategy that obeys the presence in the market for several years due to long-term contracts with strategic business partners in the Asian and European markets, as well as annual contracts with other active market participants.

1.17 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The area encompasses modest hilly topography formed by erosion and weathering primarily of unconsolidated range-front sediments. The area is bounded to the west and south by the Santa Cruz River.

Landscape will be affected at first by clearing and grubbing, road construction and construction of mining facilities. Ultimately, impacts will be from the mine pit, waste dumps, and placement of ore on the heap leach pad. The effects of mining are irreversible, although some landscape effects are partially reversible in the long run through planned restoration and reforestation methods.

Surface preservation and mitigation measures planned are: impermeable retention areas where chemical substances or process solutions are handled, implementation of a hazardous and non-hazardous waste handling program, monitoring of surface water and creek sedimentation and water quality, and storm water diversion around disturbed areas where required.

Prevention and mitigation measures contemplated to protect groundwater quality include an impermeable layer in the leach pad, sumps and process areas, as well as installation of water monitoring wells below mining facilities with regular water quality monitoring.

Actions that are planned to mitigate vegetation impacts include compensation payments to the forest fund for land use rights, organic topsoil recovery during clearing and reuse of this material in the closure phase, and implementation of a flora and fauna species protection program during all stages of the Project, including soil scarification and planting native species to restore the affected areas.

Waste generated during development and mining operations will be handled according to the provisions of the General Law for Prevention and Integrated Waste Management. A landfill will be built in the western part of the site to manage non-hazardous solid waste that cannot be recycled or reused, in compliance with NOM-083-SEMARNAT-2003.

There are three SEMARNAT permits required prior to construction; Environmental Impact Assessment (MIA), Change of Land Use (CUS) and Risk Analysis (AR). A construction permit is required from the local municipality and an archaeological release letter from the National Institute of Anthropology and History (INAH). An explosives permit is required from the Ministry of Defense, SEDENA, before construction as well. The key permits and the stages at which they are required are summarized below in Table 1-6.

Table 1-6: Key Environmental Permits

Permit	Mining Stage	Agency
Environmental Impact Manifest - MIA	Construction/Operation/Abandonment	SEMARNAT
Land Use Change -CUS	Construction/Operation	SEMARNAT
Risk Analysis - AR	Construction/Operation	SEMARNAT
Construction Permit	Construction	Santa Cruz Municipality
Explosive & Storage Permits	Construction/Operation	SEDENA
Archaeological Release	Construction	INAH
Water Use Concession	Construction/Operation	CNA
Water Discharge Permit	Operation	CAN
Unique License	Operation	SEMARNAT
Accident Prevention Plan	Operation	SEMARNAT

In accordance with the general work schedule of the El Pilar Project, the abandonment phase will commence after year 16. As part of the permitting requirements, SCC will prepare a detailed Closure and Reclamation Plan, which will be concurrently executed from the operation phase of the Project and will be completed in the abandonment phase.

In order to mitigate and minimize the potential impacts to the environment, SCC will use the best available technology and will comply with all environmental norms and good practices applicable to the different development phases of the El Pilar Project.

Environmental impacts resulting from the development of the mine are generally positive with minor negative impacts outweighed by the overall social and economic benefits.

1.18 CAPITAL AND OPERATING COSTS

This section contains forward-looking information related to capital and operating cost estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this section including prevailing economic conditions continue such that unit costs are as estimated in constant (or real) dollar terms, projected labor and equipment productivity levels and that contingency is sufficient to account for changes in material factors or assumptions.

1.18.1 Operating Costs

The El Pilar Project Base Case operating and maintenance costs are shown in Table 1-7. These costs include mining and all other cost areas that include mine department, heap leach and the SX-EW Plant.

Table 1-7: Operating Cash Cost Summary

Description	Operating cost ($/lb Cu)
Mining	$1.04
Process Plant	$0.71
G&A	$0.09
Royalty	$0.03
Total Operating Cost	**$1.87**

1.18.2 Capital Cost Estimate

The total initial capital cost is $364.9 million. A summary of capital costs is shown in Table 1-8. The detailed capital cost estimate is found in Section 18.

Table 1-8: Overall Capital Cost Estimate

DIRECT FIELD COST	($000)
General Site	$22,233
Mine Truck Shop, Fuel, Lube, Truck Wash, etc.	$6,179
Water System	$2,959
Heap Leach	$60,416
Solvent Extraction	$24,151
Tank Farm	$11,686
Electrowinning	$39,789
Main Substation	$3,778
Internal Power Distribution Lines	$1,108
Power Line and Switch Substation	$9,915
Acid Storage	$3,927
Laboratory	$2,113
Offices and Warehouse Building	$2,318
Gate House	$230
Explosive Storage	$582
Freight	$5,832
TOTAL DIRECT FIELD COST	**$197,218**
INDIRECT FIELD COSTS & OTHER CONSTRUCTED COSTS	
Duties	$2,878
Contractor Mobilization Costs	$749
Construction Power, Construction & Utilities	$150
TOTAL CONSTRUCTED COST	**$200,994**
EPCM Cost	
EPCM & QC	$24,942
Site CM Facilities	$797
Vendor Commissioning and Spare Parts	$2,333
Leach Pad and Ponds, Studies and Engineering	$4,078
EPCM SUBTOTAL	**$32,150**
TOTAL DIRECT FIELD + EPCM COST	**$233,144**
Contingency (10%)	$23,314
First Fill	$5,370
FACILITIES INITIAL CAPITAL COST ESTIMATE	**$261,829**
MINE EQUIPMENT	**$95,435**
Contingency (8%)	**$7,635**
GRAND TOTAL INITIAL CAPITAL COST ESTIMATE	**$364,899**

1.19 ECONOMIC ANALYSIS

This section contains forward-looking information related to economic analysis for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including estimated capital and operating costs, project schedule and approvals timing, availability of funding, projected commodities markets and prices.

The capital and operating cost estimates for the El Pilar Project base case have been completed along with mine scheduling. There are no known or anticipated environmental or permitting issues that would ultimately affect SCC's ability to construct and operate the Project under the assumptions detailed in this report.

The work completed in this Technical Report Summary indicates that the El Pilar Project is economically viable for the production of copper from heap leaching. The reserves are sufficient for 16 years of production at an average leaching rate of 58,000 t/d. The Project is projected to average 59 Mlbs of copper production per year and to produce 940 Mlbs of copper cathode over the LOM.

The Base Case financial model, which incorporates capital and operating estimates along with copper price assumptions, demonstrates that the Project is economic with an after-tax net present value of $54.2 million at a discount rate of 8%. Capital pay-back of initial facilities capital is estimated in 7.0 years and the IRR of the Project is 9.88%.

1.20 ADJACENT PROPERTIES

El Pilar is an extensive property (9,571.3771 ha) consisting of nineteen (19) mining concessions as described in Section 3 of this report. The property extends approximately 8 km east-west and approximately 9 km north-south. The El Pilar mineral reserve is centrally located on the Southern Copper concessions. Other property owners have concessions surrounding Southern Copper's El Pilar property on the north, west and eastern sides.

There are no known mineral deposits on the concessions immediately adjacent to the El Pilar property.

1.21 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information to this report that has not already been covered in the other sections.

1.22 INTERPRETATION AND CONCLUSIONS

1.22.1 Mineral Resources and Mineral Reserves

The QP's are of the opinion that the following pose potential risks for the El Pilar Project for the Mineral Resource, Mineral Reserves and mining:

- Given the low percentage of the Measured mineral resources relative to Indicated, the predictability of the tonnage and grade at the local level may have some uncertainties.
- Failure to maintain design slope angles. If operational slope angles are slightly flatter than design angles over several benches, the result is significantly less ore available at the bottom of a mining phase than anticipated.
- The risk to slope stability within the Quaternary Alluvial Wash Deposits is primarily geological. While the degree of consolidation is known to be variable, the exact distribution of poorly consolidated zones cannot be determined in advance or reliably modeled in stability analyses. If extensive zones of poorly-consolidated Quaternary Alluvial Wash Deposits are encountered, such as in the upper portion of the Qwu, then flatter slope designs may be required.
- In the Intrusive, risks of rock mass failure appear low due to the competent character of this material, and the limited slope heights. The risks of encountering a well-developed structural set that could control large-scale stability is indicated to be low based on the geological model and the structural data developed from the oriented core and surface mapping. However, there is indicated to be a moderate risk of locally encountering structural conditions that could result in the development of bench-scale wedges that could control bench stability and require local modifications to the slope design; this risk is indicated to be limited to the ends of

the slope in Intrusive where the orientation of the slope swings away from east-west. This is limited to the northern pit limit where the intrusive is localized.

- Higher than anticipate mining dilution and mining loss during operations could negatively impact the Mineral Reserves.
- Fluctuations in exchange rates, copper selling price, and key consumable costs (i.e. fuel) could result in lower than expected economics for the Project.

The QP's believe the following are opportunities for the El Pilar Project for the Mineral Resource, Mineral Reserves and mining:

- Improve the resource model by including a differentiation of oxidized species, this will also help to improve the metallurgical response.
- Carry out a conditional simulation study to understand the variability, and therefore risks, of the behavior of grades and geometallurgical variables in order to concentrate drilling and testing efforts.
- There is the potential beyond the known resource/reserve to further expand the deposit by drilling to the south.
- There is potential to find the ultimate source of copper mineralization believed to be a higher grade breccia and/or porphyry copper deposit.
- Optimize the design of the mining phases, final pit and the mine production schedule to delay or reduce waste material. Also, optimize designs of the waste dumps to smooth truck requirements by time period and possibly reduce the number of trucks required.
- Assess the possibility of utilizing crusher/conveyor to minimize haulage.
- Assess the possibility of utilizing electric rope shovels to save on operating costs.
- Assess the possibility of utilizing autonomous haulage and drilling to increase reduce equipment numbers.
- Slope design optimization based on performance of slopes developed within the Quaternary Alluvial Wash Deposits will provide the opportunity to increase slope angles during pit development if greater shear strengths can be demonstrated by documentation of carefully excavated, over-steepened slopes in non-critical areas of the phase pits.

1.22.2 Economic Analysis

The results of the Financial Model show that, under current market conditions and following the assumptions and considerations noted in the body of the Study, the El Pilar Project is economically feasible.

The main parameters, before and after taxes are those shown in Table 1-9.

Table 1-9: Main Parameters Before and After Taxes

Parameter	After-Tax	Pre-Tax
Total Cash Operating Costs ($/lb Cu)	1.84	1.84
NPV@8% ($M)	54.2	139.5
EBITDA ($M)	2,089	2,089
IRR (%)	9.88%	13.5%
Capital Payback (Years)	7.0	5.5

1.23 RECOMMENDATIONS

1.23.1 Mineral Resources and Mineral Reserves

The QP's recommend the following for the El Pilar Project:

- Include in the geological modeling the differentiation of mineralogical species from oxidized ores.

- SG/Density sampling should be completed in lithologies that have limited data.
- Continue with infill drilling campaigns to improve Mineral Resource confidence and Mineral Resource categorization.
- An evaluation of utilizing contractor to pre-strip to save on capital expenditures in the pre-production phase.
- An evaluation of an in-pit crusher conveyor scenario to evaluate potential operating cost savings compared to haulage.
- Review of the heap leach pad design and in particular consideration of alternate access ramp and placement considerations
- Review of the ex-pit roads and overburden storage facility designs.
- Review of the haul road width depending on final selection of the trucks and which mine regulation SCC considers appropriate.
- Review of electric rope shovels as an alternative to diesel hydraulic that was assumed for the current base case scenario.
- Review of electric production drills as an alternative to diesel powered drills.
- Review of potential for autonomous haulage and drill fleets to increase utilization and productivity.
- Review of trolley assist haulage for potential fuel savings.

1.23.2 Land Tenure

- SCC should secure the surface land required for the Project.

1.23.3 Metallurgy

- SCC should implement an early assay and metallurgical data collection program, first directed to evaluate the performance of leaching of the first ore to be placed in the heap, during the preproduction period. This will serve as an early detection of possible problems, and afterwards should be directed to monitoring, reporting and control of Leaching and SX-EW plant operation. This could require contracting an outside laboratory to process samples, while the permanent facilities are built.

1.23.4 Heap Leach Facility

- Detail engineering of the HLF should be developed to evaluate potential improvements reducing earthwork quantities to improve initial CAPEX.
- Additional percolation testing should be performed to confirm maximum number of lifts recommended before an interlift liner is required.

1.24 REFERENCES

Referenced documents are listed in Section 24 of this document.

1.25 RELIANCE ON OTHER EXPERTS

Reports received from other experts have been reviewed for factual errors by SCC and M3. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. The statements and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports. Details on information provided from other experts is found in Section 25 of this report.

2 INTRODUCTION

2.1 ISSUER

Southern Copper Corporation (SCC) is the issuer of this Technical Report Summary (TRS).

2.2 EFFECTIVE DATE

The effective date of this TRS is February 28, 2022.

2.3 PURPOSE OF ISSUE

SCC plans to develop the 100% wholly owned El Pilar Oxide Copper Project, located in Sonora, Mexico, as a new open pit mining, heap leach project with an SX-EW processing plant. The El Pilar deposit contains 317 Mt of ore with a copper content of 0.25%, containing a total of 1.7 billion pounds of copper, of which 940 Mlbs are calculated as recoverable. The process plant will be designed to produce up to 70 Mlbs of copper cathode per year. SCC plans to begin engineering and then construction immediately upon completion of environmental permitting, surface acquisition and project financing.

SCC engaged M3 to prepare a TRS at a Feasibility Study (FS) level of the El Pilar Oxide Copper Project to assess the viability of commercial operation. Alberto Bennett, P.E., of M3 is the principal Qualified Person (QP) and author of the TRS. This TRS will be the cornerstone for project financing, permitting, planning and construction of a copper mining operation at El Pilar.

El Pilar Project is currently in good standing regarding environmental permits for the project area, access road and railroad spur; the Environmental Impact Manifest, or Manifiesto de Impacto Ambiental (MIA) and the Land Use Change, or Cambio de Uso de Suelo (CUS) studies have been approved by the Mexican government agency SEMARNAT. The acceptance by authorities of these studies allows SCC to proceed with the project construction. SCC has been granted two Water Use Concessions for a total of 3.5 million cubic meters (Mm^3) of water per year. Other permitting documents are pending submittal to the required agencies for power line construction.

2.4 SOURCES OF INFORMATION

The qualified persons (QPs) who have provided input to this TRS have extensive experience in the mining industry and are members in good standing of appropriate professional institutions. Table 2-1 provides a list of the QPs and sections for which they are responsible.

Table 2-1: List of Qualified Persons

Qualified Person	Company	Section Responsibility
Alberto Bennett, P.E.	M3 Engineering & Technology Corp.	Sections 1.1, 1.2, 1.3, 1.4, 1.5, 1.15, 1.16, 1.17, 1.18, 1.19, 1.20, 1.21, 1.22.2, 1.23.2, 2, 3, 4, 5, 7.3, 9.2.2, 9.2.3, 15, 16, 17, 18 (except 18.1.2, 18.2.2), 19, 20, 21, 22 (except 22.1.2, 22.1.3, 22.2.1, 22.2.2), 23.1, 23.5, 24, 25.4, and 25.6
Laurie Tahija, Q.P.	M3 Engineering & Technology Corp.	Sections 1.10, 1.14, 1.23.3, 9.2.4, 10, and 14 (except 14.2.1), 22.1.3, 22.2.2, 23.3, and 25.3
Armando Murrieta	Ingeniería Geomex, S.A. de C.V.	Sections 1.23.4, 14.2.1, and 23.4
Ronald Turner, MAusIMM, CP, Geo	Golder Associates Inc.	Sections 1.6, 1.7, 1.8, 1.9.1, 1.11, 1.22.1, 1.23.1, 6, 7.1, 7.2, 8, 9.1, 11, 22.1.2, 22.2.1, 23.2, 23.3, and 25.1
Danny Tolmer, P.Eng.	Golder Associates Inc.	Sections 1.9.2, 1.12, 1.13, 1.22.1, 1.23.1, 9.2.5 – 9.2.12, 12, 13, 18.1.2, 18.2.2, 22.1.2, 22.2.1, 23.2, and 25.2
Michael Pegnam, P.E.	Golder Associates Inc.	Sections 7.4, 9.2.1, and 25.5

2.5 PERSONAL INSPECTIONS

Mr. Alberto Bennett, P.E., of M3 visited El Pilar on December 20, 2020. During the visit, Mr. Bennett toured the property area, reviewed site conditions and the area where the process facilities will be located, gaining a better understanding on how the terrain and topography could affect the location of project facilities as well as understanding logistics of project access and infrastructure.

Mr. Armando Murrieta conducted a site visit of the El Pilar Project on June 05, 2017. During his site visit, Mr. Murrieta was accompanied by three SCC engineers (Mr. Ramon Bustamante, Mr. Enrique Sobrevilla, and Mr. Cesar Romero). The SCC engineers provided Mr. Murrieta with a detailed overview of the El Pilar Project. The mine area and proposed sites for the waste dump, processing facilities, heap leach pad, solution storage ponds and infrastructure were inspected. Additional time was spent inspecting core samples from previous drilling campaigns for overliner materials and discussing additional details for the heap leach pad and solution ponds with SCC personnel.

The independent QP, as defined in S-K 1300, responsible for the preparation of the Mineral Resources provided in this TRS is Mr. Ronald Turner (MAusIMM), (Senior Resource Geologist). Mr. Turner visited El Pilar on August 23, 2021. During the site visit, Mr. Turner visited and inspected the El Pilar site, data capture facilities and the current conditions for sample storage. He inspected representative core of the deposit, sample cutting and logging areas. Mr. Turner also conducted discussions with site personnel regarding the geology and mineralization and reviewed geological interpretations with staff.

The independent QP, as defined in S-K 1300, responsible for the preparation of the Mineral Reserves provided in this TRS is Mr. Danny Tolmer (P.Eng), (Principal Mining Engineer). Mr. Tolmer visited El Pilar on August 23, 2021. During the site visit, Mr. Tolmer visited the Project pit area that is currently being pre-stripped by contractor miners. Mr. Tolmer also visited the approximate area that the leach pads will be constructed. During the site visit an information package was provided that gave details on the location, geology, land concessions, general site layout, and drillhole data.

2.6 UNITS OF TERMS AND REFERENCE

The units of production in this TRS are metric unless otherwise noted. Production of copper is in tonnes (t). All dollars are US dollars ($) along with other variables such as copper price, unless otherwise noted.

The units and acronyms used in this TRS are listed in Table 2-2.

Table 2-2: List of Units and Acronyms

Term	Unit /Acronym
Above mean sea level	AMSL
Acid soluble copper	ASCu
Amperes	amp
Buenavista del Cobre	BVC
Centimeter	cm
Centipoises	cP
Cubic meters	m^3
Cubic meters per day	m^3/d
Cubic meters per hour	m^3/h
Current density	amp/m^2
Density	t/m^3
Dollar	$
Feasibility Study	FS
grams/liter	g/L
Golder Associates Inc.	Golder
Heap Leach Facility	HLF
Hectares	ha
Hertz	Hz
Inch	”
Inductively Coupled Plasma - Atomic Emission Spectroscopy	ICP-AES
Ingeniería Geomex, S.A. de C.V.	Geomex
Internal Rate of Return	IRR
Kilo (1000)	k
Kilogram	kg
Kilometer	km
Kilotonnes	kt or Ktonnes
Liters	L
liters per hour per square meter	$L/h/m^2$
Liters per second	L/s
Lux	Lx
M3 Engineering and Technology Corp.	M3
Manifiesto de Impacto Ambiental (Environmental Impact Manifest)	MIA
Mega (1,000,000)	M
Mercator Minerals Ltd.	ML
Meters	m
Meters above mean sea level	m AMSL
Metric tons per day	mtpd
Millimeters	mm
Million cubic meters	Mm^3
Million loose cubic meters	Mlcm
Million pounds	Mlbs
Million tonnes	Mt
Million tonnes per year	Mtpy
Minute	min
Net Present Value	NPV
Overburden storage facility	OSF
Parts per million	ppm

Term	Unit /Acronym
Pascal	Pa
Qualified Person	QP
Residual copper	ResCu
Run of mine	ROM
Soluble copper	SCu
Second	s
Southern Copper Corporation	SCC
Specific gravity	S.G.
Square meter	m^2
Technical Report Summary	TRS
Temperature Celsius	°C
Temperature Fahrenheit	°F
Tonnage factor or specific volume	m^3/t
Tonnes	t
Tonnes per day	t/d
Tonnes per year	t/y
Total copper	TCu
Volts	V
Watts	W
Year	y

3 PROPERTY DESCRIPTION

3.1 PROPERTY DESCRIPTION

The El Pilar property comprises 9,571.3771 hectares (ha) in nineteen mining concessions located in the state of Sonora, Mexico as shown in Figure 3-1. The concessions are owned by Recursos Stingray de Cobre S.A de C.V. (Stingray), the wholly owned Mexican subsidiary of Southern Copper Corporation (SCC).

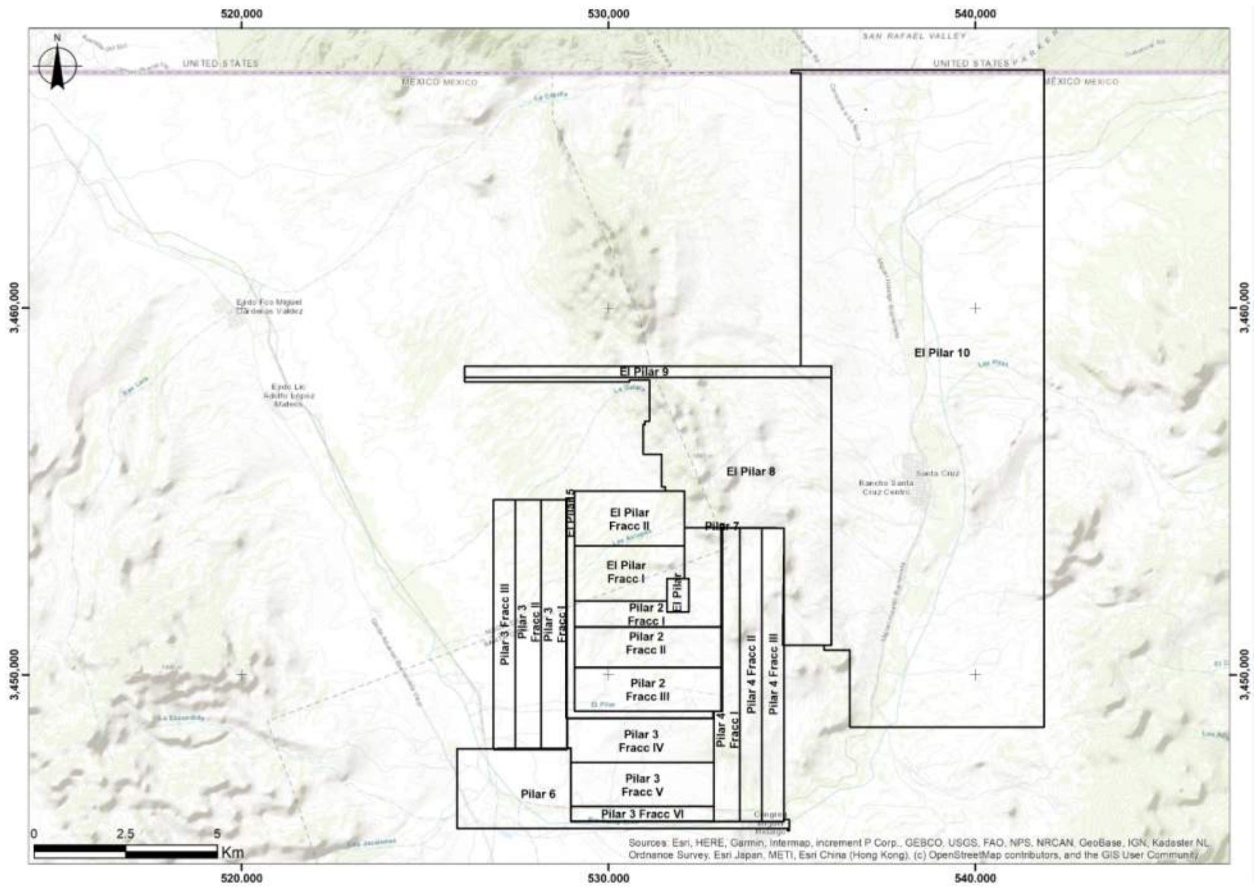


Figure 3-1: El Pilar Mining Concessions Map

3.2 MINERAL TENURE, ROYALTIES AND AGREEMENTS

3.2.1 Mineral Concessions

The status of the mining concessions is outlined in Table 3-1. Mining concessions in Mexico are granted by the Mexican Federal Government and have tenure of 50 years (renewable) subject to the payment of annual taxes. The El Pilar concession, the oldest of the mining concession that form the El Pilar property, was acquired by staking by Noranda (Normex) in 1999. Stingray acquired the El Pilar property in April 2007 by purchasing Noranda Mexico (Normex) from Xstrata Plc. Mercator Minerals LTD (ML) acquired Stingray in 2009 and SCC acquired Stingray and its eighteen concessions in 2015; SCC obtained another mining concession in 2017 (Pilar 8), totaling 9,571.3771 ha in nineteen concessions.

Table 3-1: Mining Concession Status

Mining Concession	Title	Area (Has)	Filing Date	Expiry Date
El Pilar	210725	54.000	26-Nov-99	25-Nov-49
El Pilar Fracción I	226352	420.419	13-Jan-06	09-April-2051
El Pilar Fracción II	226353	450.000	13-Jan-06	09-April-2051
Pilar 2 Fracción I	226357	455.581	13-Jan-06	27-July-2050
Pilar 2 Fracción II	226358	440.000	13-Jan-06	27-July-2050
Pilar 2 Fracción III	226359	480.000	13-Jan-06	27-July-2050
Pilar 3 Fracción I	226360	476.000	13-Jan-06	24-February-2055
Pilar 3 Fracción II	226361	476.000	13-Jan-06	24-February-2055
Pilar 3 Fracción III	226362	408.000	13-Jan-06	24-February-2055
Pilar 3 Fracción IV	226363	476.089	13-Jan-06	24-February-2055
Pilar 3 Fracción V	226364	468.000	13-Jan-06	24-February-2055
Pilar 3 Fracción VI	226365	156.000	13-Jan-06	24-February-2055
Pilar 4 Fracción I	226354	446.726	13-Jan-06	17-August-2054
Pilar 4 Fracción II	226355	480.000	13-Jan-06	17-August-2054
Pilar 4 Fracción III	226356	480.000	13-Jan-06	17-August-2054
Pilar 5	221639	208.0574	09-Mar-04	08-March-2054
Pilar 6	232447	794.0218	08-Aug-08	07-Aug-2058
Pilar 7	234984	1.0000	23-Sep-09	22-Sep-2059
Pilar 8	245548	2,401.4829	11-Aug-17	10-Aug-2067
Total		**9,571.3771**		

3.2.2 Surface Rights

In January 2010, the Ejido Miguel Hidalgo Assembly approved a purchase agreement with SCC for 1,632 hectares for permanent ownership of the land. The Ejido obtained the change of status of the land from common use to private parcels, allowing the involved Ejido members to obtain property titles and sell/transfer the property at their convenience. The individual parcel titles were granted to the Ejido members in September 2011 by the National Agrarian Registry (RAN), the Mexican agency in charge of regulating Ejido land issues. The purchase agreement was executed in October 2011 with the individual landowners following a 30-day waiting period as stipulated by RAN regulations.

Due to the relocation of the heap leach pad facility closer to the pit to the north, 294 hectares of additional land was needed to accommodate the new pad area. The additional land was negotiated with the Ejido in two agreements. The first agreement is a 261 ha long-term lease agreement for 30 years, which was approved by the Ejido assembly in October 2013. In November 2013, a second agreement was signed with Ejido Miguel Hidalgo to purchase the surface rights of 133 ha. The terms of this agreement were negotiated under two conditions, due to timing of the privatization process, as follows:

- The first agreement is for temporary occupation and surface rights, which allows for development of the Project under a long-term lease agreement. This agreement will be valid until the ejido members obtain titles to the parcels, which will then allow them to sell/transfer the land rights.
- The second portion of the agreement is for purchase of the land, once the Ejido members obtain the titles. This will allow for full land ownership and control of the 133 ha plus the 1,632 ha already in full ownership and 161 ha on 30 years lease agreements totaling 1,926 ha.

The three agreements already allow SCC to commence work in accordance with the environmental, State and Local permits. The final purchase contract of the 133 ha will be signed when the Agrarian Registry issues corresponding titles and ownership is transferred to SCC. At that time, the lease agreement will be terminated. The land purchase process of 133 ha is estimated to be completed by 2023.

In summary, a total of 1,926 ha of surface rights have been successfully negotiated with the Ejido Miguel Hidalgo, which allows for all required land ownership rights needed for project development.

Right of way for the 115 kV power line was secured through agreements with three Ejidos and one land proprietor along the route for a total of 28.9 km. The agreements were approved by the Ejido assemblies and executed as follows:

- Ejido Cibuta: signed on March 18, 2012, 11.6 km
- Ejido Peñalosa: signed on March 17, 2012, 6.4 km
- Ejido Miguel Hidalgo: signed on March 10, 2012, 4.3 km
- Joaquin Pompa (private property): signed on April 25, 2012, 6.6 km

The agreement with Joaquin Pompa also includes the right of way for the access road from the Nogales-San Antonio road to the property boundary on the west as well as the railroad spur from the existing railway to the property boundary on the west.

3.2.3 Royalties

Under the terms of the Xstrata agreement to purchase the property, SCC will pay a gross metal sales royalty of 1% to Xstrata. Xstrata has a right to buy back 50% of the property in the event that more than 3 billion pounds of copper ore are defined in an S-K 1300 Technical Report Summary. SCC will remain the operator of the property.

3.2.4 Corporate Ownership

On September, 2015, after a tender bid process initiated and conducted by Stingray for the purchase of the El Pilar Assets (as hereinafter defined) and the El Pilar Business, SCC completed a Share Purchase Agreement with Deloitte Restructuring Inc, in its capacity as trustee in bankruptcy of Mercator Minerals Ltd. and Stingray Copper Inc., as a result, upon completion of such purchase and sale, SCC became the direct and indirect holder of all of the issued and outstanding shares of Company 4394909, Company 4394895, Minera Stingray, and Recursos Stingray which directly and indirectly hold a 100% interest in the El Pilar Assets and the El Pilar Business.

3.3 LOCATION

The El Pilar Property is located in north central Sonora about 15 km south of the international border with United States of America (USA). The property is situated within lands of Ejido Miguel Hidalgo (also referred to as San Lazaro), in Santa Cruz Municipality. The property is situated between UTM coordinates 3,446,000N to 3,455,000N and 526,800 E to 534,700 E. Figure 3-2 shows the location of the El Pilar Property.

Figure 3-2: El Pilar Location Map

4 ACCESSIBILITY, CLIMATE LOCAL RESOURCES, INFRASTRUCTURE, PHYSIOGRAPHY

4.1 ACCESSIBILITY

The El Pilar property can be reached by road from Hermosillo, Sonora, Mexico and from Tucson, Arizona, USA (See Figure 3-2). From Hermosillo, the easiest access is via Hermosillo to Imuris (210 km) and Imuris to San Antonio (36 km), and from San Antonio to Miguel Hidalgo (San Lazaro) (35 km). The route from Hermosillo to Miguel Hidalgo takes about 3 1/2 hours of driving time. The route from Tucson to Miguel Hidalgo is currently a 2 hour drive and utilizes a newly paved road from Nogales, Sonora to Miguel Hidalgo (30 km).

4.2 CLIMATE

The project area climate is classified as semidry with the rainy season in summer and limited rain the rest of the year. The average annual temperature is 17.8°C which would allow for year round operations. The project site occurs at elevations between 1,250 m and 1,425 m above mean sea level (AMSL). The hottest months are June to September. Mean monthly temperatures vary between 11.2°C (January) and 23.4°C (July). Precipitation at the El Pilar site is bi-seasonal. Most of the rainfall occurs in the form of summer storms during the months of July, August and September with June and October marking the beginning and the end of the rainy season. There is a secondary, minor rainy season in the winter, with precipitation occurring primarily in December and January. The spring months, from February to May, typically have no rainfall. Mean annual precipitation, according to the records of the Santa Cruz Railroad Station is 543.6 mm, with July as the wettest month with 138.5 mm and May as the driest month with 7.8 mm. The climate is amenable to year around mining and processing operations. Vegetation is described in Section 17.

4.3 LOCAL RESOURCES AND INFRASTRUCTURE

The site is a green-field mining site with no existing infrastructure. Experienced mining personnel and related contractors are available in Cananea, located 45 km southeast of El Pilar, in Nogales, located 30 km northwest, as well as in several nearby villages. The nearby Mariquita and Milpillas mines attracted much of their labor force from these nearby villages, both mines are now in closure process, releasing most of their local workforce.

A power line is located three kilometers to the south, in the village of Miguel Hidalgo (San Lazaro) where SCC has an office and warehouse facilities. The Project will require the construction of a power line from the operation to connect with the high voltage power lines accessible in Nogales, 32 km west of the property. A project has been developed to the necessary level to define the final route, secure easement and begin discussions with CFE (Comisión Federal de Electricidad) and CENACE (Centro Nacional de Control de Energía).

Access road to the project area will be constructed from the Nogales-San Antonio road, accessing the property from the west and, crossing the Santa Cruz River by means of a bridge.

A railroad parallels the Santa Cruz River, and is located 3 km west of the property. Construction of a new railway spur approximately 4 km in length is planned for the delivery sulfuric acid. The railway is operated by Ferrocarril Mexicano S.A. de C.V.

Water rights have been secured by two water concessions totalling 3.5 million cubic meters per annum, three wells will supply all water requirements for the plant and are located within the property boundaries.

Cellular phone and internet service are available within the Miguel Hidalgo community through infrastructure operated by the main cell phone operator in the Country.

4.4 PHYSIOGRAPHY

El Pilar is situated in the Basin and Range physiographic province of North America, where topography is generally rugged at higher elevations and flat to gentle at lower elevations. The El Pilar deposit is located at the southwest margin of the Patagonia Mountains near the base of a mountain range. The topographic characteristics near the deposit are favorable and permit sufficient surface space for a mining operation with enough flat to gentle topography for building leaching pads, waste disposal areas, etc. The project site elevation ranges between 1,250 and 1,425 m AMSL.

5 HISTORY

5.1 HISTORY PRIOR TO RECURSOS STINGRAY

The history of exploration in the El Pilar area before 1992 is not well documented. However, it is known that in the 1970's, Cyprus Amax Minerals had claims in the area. According to verbal communications with local people and field evidence, it is clear that a geophysical survey and a few drill holes were completed in the northern part of the El Pilar Discovery area. Results of both the drilling and the geophysical survey are unknown.

Before Normex's direct involvement, there was no mineral resource known, only small old workings following narrow veins with erratic copper and molybdenum mineralization. Other small workings in the area were dug by local miners (gambusinos), searching for azurite and turquoise.

In 1992, Normex began acquiring ground at El Pilar. From 1992 to 1997, Normex carried out an exploration program that included regional mapping, sampling and limited geophysical surveying. From 1998 to 1999, Freeport Copper, under an agreement with Normex, carried out an exploration program that included regional mapping, rock and vegetation sampling, and some geophysical surveys. This exploration program concluded with a short drilling campaign consisting of a total of 1,561 m drilled in eight (8) reverse circulation holes. This program encountered copper mineralization in three holes, with one intercept in bedrock and two in gravels. After the exploration agreement with Freeport ended in 2000, Normex continued with a short CSAMT survey and soil sampling exploration program. In addition, Normex carefully assessed the data generated by Freeport, emphasizing lab checks for validating the copper assays from the gravels. From September 2000 to March 2001, Normex completed a drill program of 10,336 m in 52 diamond drill holes spaced at 200 m centers.

Following Normex's drilling campaign, resource calculations were undertaken in 2001, and again in 2003.

5.2 2001 POLYGONAL RESOURCES BY NORMEX

Geological interpretation and modeling of El Pilar was done using 100 m spaced vertical sections in both E-W and N-S directions, as well as horizontal plans every 10 m including composite geology and topography.

A data base of 4,047 samples was captured in GEMCOM format, including 3,318 TCu assays and 1,863 CuSol assays. The result of this calculation was an inferred mineral resource of 162,196,370 t @ 0.39% TCu at a 0.25% TCu cut-off grade.

5.3 2003 BLOCK MODEL RESOURCE ESTIMATE BY MAGRI

Geological interpretation was based on 11 north-south sections looking north, 19 east-west sections looking west and level plans at 10 m intervals from level 1080 m to level 1390 m. Sections were spaced every 200 m with a few at 100 m intervals and contained lithology interpreted as polygons. Two sets of three dimensional solids were generated for lithology: solids obtained from the north-south and east-west sections using the 3D ring and tie-lines technology, and solids obtained from the level plans by "extruding" each plan 5 m above and 5 m below the plan elevation. Once sections and level plan solids were validated on screen, their lithology was assigned to the corresponding drillhole intersections. Thus each drill hole ultimately contained three lithology fields: logged lithology, lithology obtained from the section solids and lithology obtained from the level plan solids. Comparative statistics for total and soluble copper were generated between the logged lithology and the lithology obtained from section and plan solids. Results show that solids generated from plans cover a larger area and are more reliable than solids generated from sections; therefore, they were used for the resource estimation. Any obvious erroneous lithological codes were found and corrected. Basic statistics were made for the different lithologies as part of the resource estimate. Densities varied from 2.17 for the overburden to 2.53 for host rock. The calculated average specific gravities for each lithology were used in the tonnage calculations.

The block model was estimated by ordinary kriging in three passes using appropriate search radii and parameters in each case. As a means of checking the model and declusterizing the composite data, a nearest neighbor estimation was done in a single pass. The result of the kriging at various cut-off grades is given in Table 5-1.

Table 5-1: Magri 2003 Kriging Results for Lithological Units

Lithological Units: CGI CGI+V F IBX - KRIGING					
Cut-Off	**Volume**	**Density**	**Tonnage**	**TCu Krig.**	**TCu Fine**
0.500	5,196,000	2.32	12,040,600	0.581	69,956
0.400	24,283,999	2.30	55,771,718	0.469	261,569
0.350	40,755,999	2.29	93,504,037	0.430	402,067
0.300	57,499,998	2.29	131,841,035	0.399	526,046
0.250	78,447,997	2.29	179,817,274	0.365	656,333
0.200	103,663,996	2.29	237,518,111	0.331	786,185
0.150	128,923,995	2.29	295,300,228	0.301	888,854
0.100	163,143,993	2.29	373,570,385	0.264	986,226
0.001	234,371,990	2.29	536,587,696	0.202	1,083,907

The results of this block model Inferred Mineral Resource estimate completed by Magri in 2003 resulted in a tonnage calculation of 237,518,111 tonnes at 0.331% total copper (TCu) at a cut-off of 0.2% TCu.

5.4 2007 INFERRED MINERAL RESOURCE BY WOODS

In an April, 2007 NI 43-101 Technical Report on the El Pilar Property, Gary Woods, P. Geo, reviewed the earlier Mineral Resource estimations and confirmed an estimate of 237,922,918 t at a grade of 0.331% TCu calculated at a 0.20% TCu cut-off as shown in Table 5-2.

Table 5-2: Woods 2007 Inferred Resource

2007 INFERRED MINERAL RESOURCE					
Grade Group	**Volume**	**Density**	**Tonnage**	**TCu%**	**CuS%**
Units	**m^3**	**t/m^3**	**1,000 t**	**Grade**	**Grade**
>0.50% TCu	5,196	2.32	12,041	0.58	0.51
>0.45% TCu	11,304	2.30	26,041	0.52	0.46
>0.40% TCu	24,312	2.30	55,836	0.47	0.41
>0.35% TCu	40,792	2.29	93,587	0.43	0.38
>0.30% TCu	57,556	2.29	131,970	0.40	0.35
>0.25% TCu	78,548	2.29	180,047	0.37	0.32
>0.20% TCu	103,840	2.29	237,923	0.33	0.29
>0.15% TCu	129,324	2.29	296,220	0.30	0.27
>0.10% TCu	165,296	2.29	378,520	0.26	0.24
>0.05% TCu	219,244	2.29	502,032	0.22	0.20
>0.00% TCu	480,724	2.29	1,102,550	0.10	0.11

Preliminary metallurgical test work was carried out at Falconbridge's Lomas Bayas mine in Chile in 2003 and 2004 and by METCON in 2001 and 2005. This preliminary work included bottle roll testing and column leach tests. A scoping level economic evaluation was prepared by AMEC E&C Services Inc. of Phoenix, Arizona in 2005-2006 for Normex.

The study concluded that the El Pilar Project demonstrated positive economics and was worthy of further assessment. Stingray acquired the property at that juncture.

5.5 2009 STINGRAY MINERAL RESERVE FEASIBILITY STUDY

In April 2009, a NI 43-101 compliant Feasibility Study was filed on behalf of Stingray by M3. Included within the NI 43-101 are Mineral Resource and Mineral Reserve estimates done independently by Mike Hester of IMC in Tucson. The following summarizes the 2009 Hester Mineral Resource and Mineral Reserve estimates.

5.5.1 2009 Hester Mineral Resources

Table 5-3 to Table 5-5 summarize the Mineral Resource for the El Pilar Project as calculated in 2009 by Mike Hester, FAus IMM of IMC, the Qualified Person responsible for the mineral resource/mineral reserve portion of the 2009 Technical Report. The mineral resources are inclusive of the mineral reserve and are based on the original Stingray 10-meter (bench height) model, with 5-meter composites.

Table 5-3: Hester 2009 Mineral Resource - 0.15% Copper Cut-off

El Pilar Mineral Resource – 0.15% Copper Cut-off			
Resource Class	**kt**	**Copper (%)**	**Copper (Mlbs)**
Measured Mineral Resource	103,819	0.311	711.8
Indicated Mineral Resource	241,088	0.273	1,451.0
Measured/Indicated Resource	**344,907**	**0.284**	**2,162.8**
Inferred Mineral Resource	72,848	0.240	385.4

Table 5-4: Hester 2009 Mineral Resource - 0.20% Copper Cut-off

El Pilar Mineral Resource – 0.20% Copper Cut-off			
Resource Class	**kt**	**Copper (%)**	**Copper (Mlbs)**
Measured Mineral Resource	87,690	0.335	646.0
Indicated Mineral Resource	188,694	0.300	1,245.0
Measured/Indicated Resource	**276,384**	**0.311**	**1,891.0**
Inferred Mineral Resource	42,556	0.287	268.0

Table 5-5: Hester 2009 Mineral Resource - 0.25% Copper Cut-off

El Pilar Mineral Resource – 0.25% Copper Cut-off			
Resource Class	**kt**	**Copper (%)**	**Copper (Mlbs)**
Measured Mineral Resource	68,777	0.366	553.0
Indicated Mineral Resource	129,787	0.335	956.0
Measured/Indicated Resource	**198,564**	**0.346**	**1,510.0**
Inferred Mineral Resource	22,806	0.341	171.0

5.5.2 2009 Hester Mineral Reserves

Table 5-6 summarizes the El Pilar Mineral Reserves as calculated by Mike Hester of IMC in the 2009 Stingray Feasibility Study. At a 0.15% TCu cut-off grade, the Proven and Probable Mineral Reserves determined by Hester total 229.7 Mt of ore containing 1.55 billion pounds of copper at an average TCu grade of 0.31%. Table 5-7 summarizes additional Mineral Resources which are exclusive of the Mineral Reserves. Measured and Indicated Mineral Resource adds 115.2 Mt grading 0.24% TCu or 606 million pounds of contained copper. The Inferred Mineral Resource adds an additional 72.8 Mt at 0.24% TCu or 385 million pounds of contained copper.

Table 5-6: Hester 2009 Mineral Reserves at 0.15% Copper Cut-off

El Pilar Mineral Reserve			
	0.15 % Total Copper Cut-off		
Mineral Reserve Class	**Ore (kt)**	**Copper (%)**	**Copper (Mlbs)**
Proven Mineral Reserve	88,434	0.323	629.7
Probable Mineral Reserve	141,290	0.298	927.3
Proven/Probable Mineral	**229,724**	**0.307**	**1,557**
Total Pit Material	599,455	Waste: Ore	1.6

Table 5-7: Hester 2009 Mineral Resources (Exclusive of Reserve)

El Pilar Mineral Resource (Exclusive Reserve)			
	0.15 % Total Copper Cut-off		
Mineral Resource Class	**kt**	**Total Cu (%)**	**Cu (Mlbs)**
Measured Mineral Resource	15,385	0.242	82.1
Indicated Mineral Resource	99,798	0.238	523.7
Measured/Indicated Resource	**115,183**	**0.239**	**605.8**
Inferred Mineral Resource	72,848	0.24	385.4

5.6 2011 STINGRAY MINERAL RESERVE FEASIBILITY STUDY UPDATE

In November 2011, a NI 43-101 compliant Feasibility Study Update was filed on behalf of Stingray by M3. Included within the NI 43-101 are Mineral Resources and Mineral Reserves estimates done independently by Mike Hester of IMC in Tucson. The following summarizes the 2011 Hester Mineral Resource and Mineral Reserve estimates.

5.6.1 2011 Hester Mineral Resources

Table 5-8 presents the Mineral Resource for the El Pilar Project as calculated in 2011 by Mike Hester, FAus IMM of IMC, the Qualified Person responsible for the Resource/Reserve portion of the 2011 Technical Report. The Mineral Resources are inclusive of the Mineral Reserves.

Table 5-8: Hester 2011 Mineral Resource – 0.15% Copper Cut-off

El Pilar Mineral Resource – 0.15% Copper Cut-off			
Resource Class	**kt**	**Copper (%)**	**Copper (Mlbs)**
Measured Mineral Resource	128,094	0.307	867.0
Indicated Mineral Resource	231,154	0.266	1,355.5
Measured/Indicated Resource	**359,248**	**0.281**	**2,222.5**
Inferred Mineral Resource	67,996	0.239	358.3

Table 5-9 presents the Mineral Resource as calculated by Mike Hester of IMC in the 2011 Stingray Feasibility Study, exclusive of the Mineral Reserve.

Table 5-9: Hester 2011 Mineral Resources (Exclusive of Reserve)

El Pilar Mineral Resource (Exclusive Reserve)			
	0.15 % Total Copper Cut-off		
Mineral Resource Class	**kt**	**Total Cu (%)**	**Cu (Mlbs)**
Measured Mineral Resource	28,823	0.221	140.4
Indicated Mineral Resource	101,677	0.221	496.4
Measured/Indicated Resource	**130,500**	**0.222**	**636.7**
Inferred Mineral Resource	67,996	0.239	358.3

Table 5-10 and Table 5-11 shows the tabulation of Measured, Indicated and Inferred resources, respectively, contained in the block model at various total copper cut-off grades. The cut-off grade results that most closely match the Mineral Reserve is highlighted in yellow.

Table 5-10: Hester 2011 Measured & Indicated Mineral Resources at Various Cut-off Grades

Measured and Indicated Mineral Resources @ Different Cut-off Grades									
TCu Cut-off (%)	Measured Mineral Resource			Indicated Mineral Resource			Measured/Indicated Mineral Resource		
	kt	Total Cu (%)	Soluble Cu (%)	kt	Total Cu (%)	Soluble Cu (%)	kt	Total Cu (%)	Soluble Cu (%)
0.30	64,377	0.385	0.193	67,814	0.366	0.172	132,191	0.375	0.182
0.25	86,144	0.357	0.168	119,222	0.326	0.132	205,366	0.339	0.151
0.20	109,304	0.329	0.145	178,495	0.292	0.113	287,799	0.306	0.125
0.15	128,094	0.307	0.128	231,154	0.266	0.095	359,248	0.281	0.107
0.10	137,301	0.295	0.121	255,218	0.253	0.087	392,519	0.268	0.099

Table 5-11: Hester 2011 Inferred Mineral Resources at Various Cut-off Grades

Inferred Resources @ Various Cut-offs			
	Inferred Mineral Resources		
TCu Cut-off (%)	kt	Total Cu (%)	Soluble Cu (%)
0.30	9,969	0.422	0.216
0.25	20,898	0.344	0.153
0.20	39,612	0.286	0.111
0.15	67,966	0.239	0.082
0.10	90,333	0.212	0.068

5.6.2 2011 Hester Mineral Reserves

Table 5-12 presents the El Pilar Mineral Reserves as calculated by Mike Hester of IMC in the 2011 Technical Report.

Table 5-12: Hester 2011 El Pilar Mineral Reserves

El Pilar Mineral Reserve				
Reserve Class	**Ore (kt)**	**Tot Cu (%)**	**Sol Cu (%)**	**Copper (Mlbs)**
Proven Mineral Reserve	99,572	0.332	0.153	728.8
Probable Mineral Reserve	130,583	0.299	0.131	860.8
Proven/Probable Mineral Reserve	**230,155**	**0.313**	**0.140**	**1,589.6**
Total Pit Material	678,686	Waste:Ore	1.95	

6 GEOLOGICAL SETTING, MINERALIZATION AND DEPOSIT

6.1 GEOLOGICAL SETTING

6.1.1 Regional Geology

The El Pilar copper (Cu) deposit is hosted within unconsolidated sedimentary units located along the southwest flank of the Patagonia Mountains, about 15 km south of the U.S.- Mexico border. The Patagonia Mountains are within the Basin and Range province of the western USA and Mexico. The mountains are similar to other nearby ranges in that they were formed in the late Miocene during an ongoing event of crustal extension. This extension resulted in northwest-trending uplifted blocks and intervening, sediment-covered down dropped blocks.

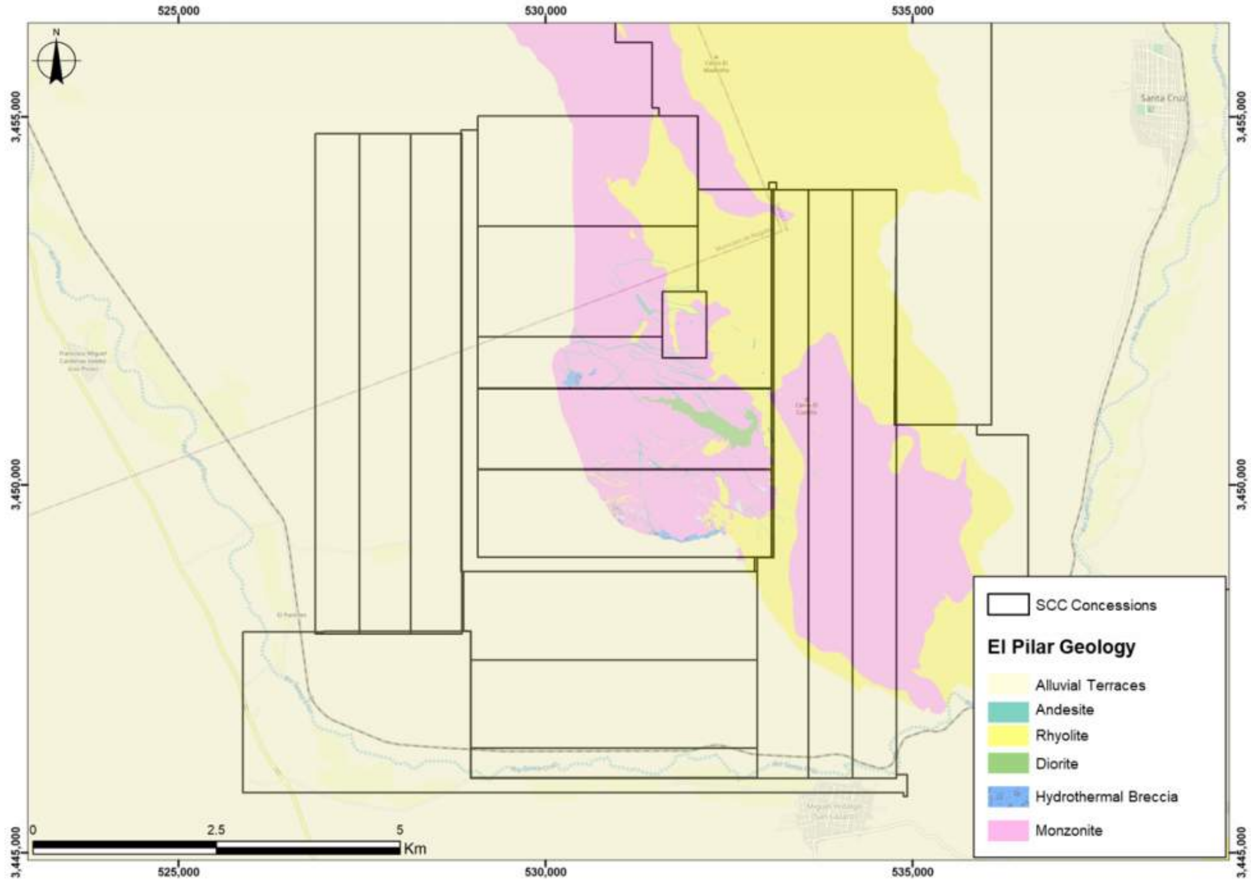
North of the US-Mexico border, extensive geologic work shows that the west side of the Patagonia Mountains is underlain by schist, gneiss and granitic rocks of mid Proterozoic age. The east side is underlain by folded Paleozoic sedimentary rocks, lower Mesozoic sedimentary and volcanic rocks, and Paleocene-Eocene volcanic rocks. All the above rocks are intruded by the composite Patagonia batholith, which is made up of Jurassic and Early Tertiary granodiorite-monzonite and by dikes and other small bodies that range in composition from andesite to granite.

North of the US-Mexico border the Patagonia Mountains host two well-defined porphyry Cu deposits, Red Mountain and Sunnyside (Corn, 1975 and Graybeal, 1996). These deposits are buried beneath approximately 2,000 meter (m) thick lithocaps of intensively altered volcanic rocks of Eocene age. Radiometric ages for the deposits range from 50-55 million-years-ago (mya). In addition to the porphyry Cu deposits, the Patagonia Mountains also contain many mineralized breccia pipes, including the 3-R, Four Metals, Santo Nino and Line Boy, and, skarn-type base metals deposits, including those at Washington Camp located about 25 km north of El Pilar (Schrader and Hill, 1911; Simons, 1974).

The geology of the El Pilar property (Source: SCC 2022
Figure 6-1) has been previously described by Woods (2007) and Sauceda (2010) and this description of geology and mineralization is based on those sources and on a field visit to the property by J. E. Dreier in 2006.

Immediately to the north and east of the El Pilar Cu deposit, exposed bedrock consists of a coarse-grained biotite hornblende, granite-quartz monzonite intrusive body about 5 km in diameter. This intrusive is similar in composition and texture to mid Proterozoic granites along the west side of the Patagonia Mountains in Arizona about 20 km to the north of the El Pilar property (Drewes, 1972 and Simons, 1974). The granitic rocks at El Pilar, which are herein interpreted to be Precambrian in age, are mostly unmineralized and are variably intruded by a northwest-trending, coarse-grained diorite body about 1.5 km long and 150 m to 200 m wide and by swarms of northwest-trending andesite dikes and northeast-trending felsic dikes. At the southern edge of its outcrop area, the Precambrian granite is intruded by an aplite. The area to the east and northeast of the granite is underlain by rhyolite and rhyodacite ash flow tuffs, which unconformably overly the granitic rocks and are juxtaposed against the granitic rocks by faults. Based on radiometric dating of similar rocks just to the north of El Pilar, in Arizona, the volcanics are interpreted to be Triassic or Jurassic in age.

The region to the south and west of the previously described bedrock area is comprised solely of alluvial deposits interpreted to be preserved within a down-dropped basin filled with unconsolidated sediments. In the area of the El Pilar Cu deposit, these basin-fill deposits are interpreted to be of Quaternary age and are juxtaposed against the Precambrian granitic rocks by an east-west to northwest-trending, south dipping zone of faulting and hydrothermal brecciation. The faulting is of unknown displacement. The breccia zone comprises a multi-stage, highly silicified, Cu mineralized hydrothermal breccia that is up to 100 m wide and 600 m long. This breccia is interpreted to be the source of the El Pilar Cu deposit.



Source: SCC 2022

Figure 6-1: Geologic Map of the El Pilar Project Area

Immediately to the south of the east-west fault and the mineralized breccia zone, exposed materials are comprised of range-front Quaternary alluvial fan and alluvial wash deposits. These basin-fill deposits are poorly sorted, poorly bedded, angular to sub-angular weakly cemented sediments with matrix supported clasts up to a maximum of 2 m in diameter. The matrix consists of clay to sand-sized material that is generally similar in mineralogy to the clasts but exhibits increasing clay content in the finer sieve sizes. The range front sediments are moderately dissected at the surface by modern drainage valleys up to 30 m - 40 m deep. In the El Pilar Cu deposit area, the alluvial deposits have been further subdivided as described in the following sections.

6.1.2 Local Geology

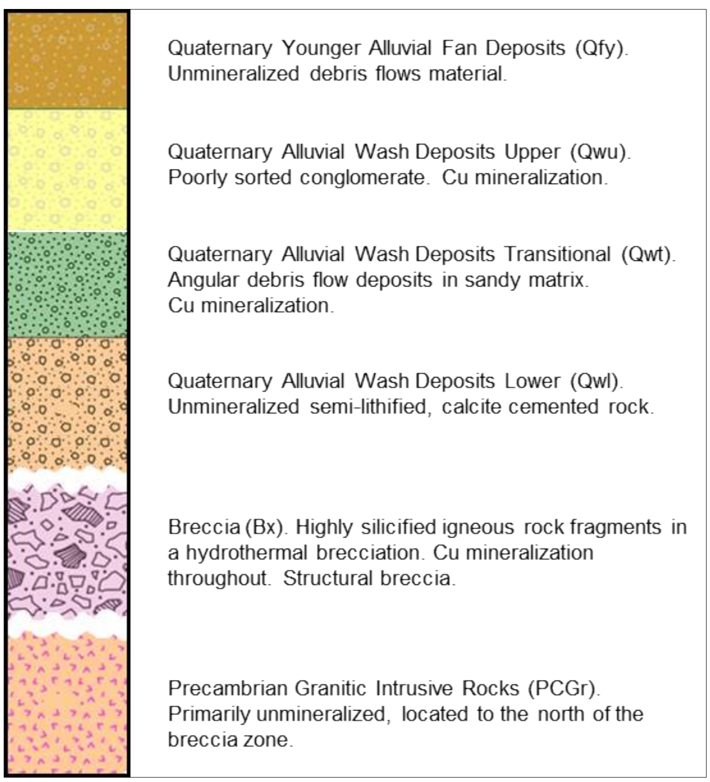
Noranda completed an interpretation of cross sections, photos of drill core, the examination of core from eight drill holes (98-05, EP-06, 00-18, 00-23, 00-28, 00-32, 00-59, and 01-01) and mineralized outcrops, the El Pilar Cu deposit occurs within unconsolidated, poorly sorted, poorly bedded, proximal facies alluvial wash deposits that are overlain by dissected younger alluvial fan deposits. According to criteria and discussions in Rust, B. R., 1979 and Davis, R. A., 1983, these proximal facies debris flow deposits were formed in a proximal range front alluvial setting.

The El Pilar Cu deposit is comprised of mineralized sediments that form a continuous southwest trending body up to 200 m thick. This body extends for 2,300 m S33°W and is approximately 750 m wide NW-SE. The mineralized horizon is hosted by unconsolidated to poorly consolidated alluvial wash deposits that are interpreted to have filled an older, range-front topographic depression located proximal to, and to the south of, the exposed hydrothermal breccia zone. The mineralized wash deposits are overlain by unconsolidated younger alluvial fan deposits that are generally similar in character to those in the mineralized horizon. The mineralized deposits overlie and rest on more consolidated proximal facies alluvial wash deposits that are cemented by calcite. In general, there appears to be a progressive change in the lithology of the fragments within the overall sedimentary sequence, such that rock fragments in the overlying unmineralized alluvial fan sediments are entirely granitic, those in the upper part of the mineralized wash deposits are mostly granitic with some felsic and andesitic volcanic fragments, while those in the lower half of the mineralized horizon tend to be more than half volcanic and those in the underlying cemented unit are largely volcanic.

Based on the lithological differences described above, Noranda and Stingray subdivided the alluvial deposits into four units: 1) Qo (Quaternary overburden); 2) Mineralized Conglomerate - Cgi (with granitic intrusive fragments); 3) Transitional Conglomerate - Cgi+v (with granitic intrusive and volcanic fragments); and 4) Lower Conglomerate Cgv (with volcanic fragments). However, because the units described by Noranda are not truly conglomerates, as they are comprised of generally unsorted and angular materials rather than being rounded. Since the 2011 NI 43-101, these units have been renamed as follows:

- Qo (Quaternary overburden) now Qfy (alluvial fan deposits, younger)
- Cgi (mineralized conglomerate) now Qwu (alluvial wash deposits, upper)
- Cgi+v (transitional conglomerate) now Qwt (alluvial wash deposits, transitional)
- Cgv (lower conglomerate) now Qwl (alluvial wash deposits, lower)

A stratigraphic column is presented in Figure 6-2 and a cross-section in Figure 6-3. The units are discussed from oldest to youngest in the following sections.



Source: SCC (Stingray 2021)

Figure 6-2: General Stratigraphic Column

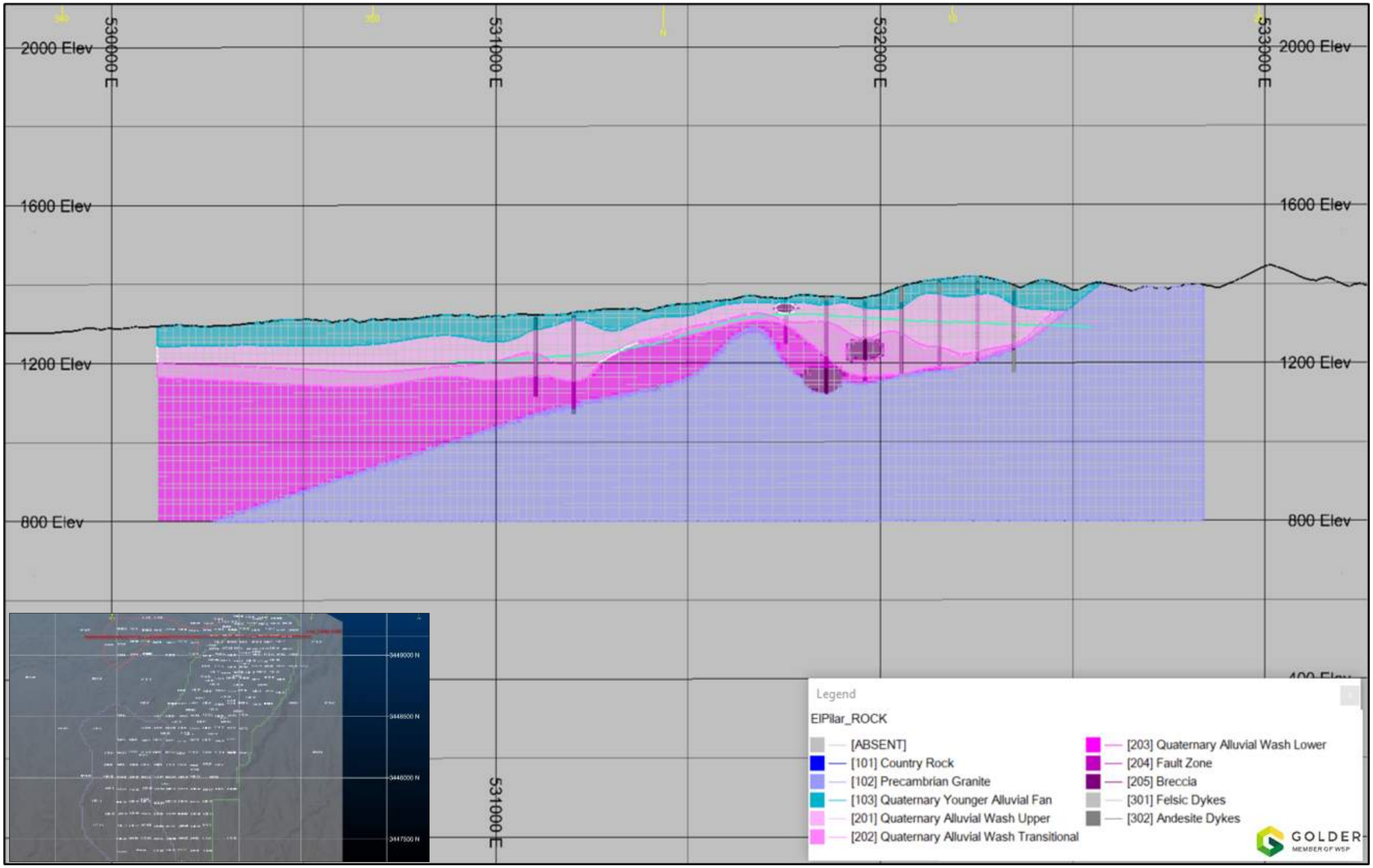


Figure 6-3: Geological Cross-Section – Line 3,449,150N

6.1.2.1 Precambrian Intrusive (PCGr)

A granitic intrusive interpreted to be Precambrian in age bounds the mineralized breccia zone along the northern side of the El Pilar Cu deposit. The intrusive is for the most part unmineralized (Figure 6-4).

Source: SCC (Mercator 2011)

Figure 6-4: Precambrian Granite

6.1.2.2 Breccia (Bx)

The east-west to northeast-trending zone that marks the boundary between Precambrian intrusive rocks to the north and the Quaternary alluvial deposits to the south, is characterized by a highly eroded and weathered remnant of a mineralized silicic breccia. The breccia(s) consists of highly silicified igneous rock fragments in a multigenerational matrix of repeated hydrothermal brecciation and silicification. The breccia is similar to other breccias that occur in breccia pipes around the world related to porphyry Cu deposits, such as the nearby Buenavista del Cobre deposit. However, because the breccia is highly elongate east-west, along a true structural zone rather than a pipe, the area of brecciation is more properly referred to as a breccia zone, rather than a breccia pipe.

The breccia is mineralized throughout its extent with oxide Cu and remnant iron oxide minerals. Figure 6-5 shows typical mineralized breccia in outcrop.

Source: SCC (Mercator 2011)

Figure 6-5: Mineralized Silicic Breccia in Outcrop

6.1.2.3 Quaternary Alluvial Wash Deposits Lower (Qwl)

Underlying Qwt and the El Pilar Cu deposit are unmineralized sediments designated herein as Qwl (Figure 6-6). Although these deposits are similar in overall appearance and origin to Qwu and Qwt, Qwl is a semi-lithified, calcite-cemented rock. The fragments in Qwl are dominantly felsic and andesitic volcanics. Qwl ranges in thickness from 10 m to greater than 100 m, and the drill holes that penetrate Qwl terminated within it and did not extend through it.

Source: SCC (Mercator 2011)

Figure 6-6: Typical Qwl Core Samples - Calcite Cemented Debris Flow Deposit with Volcanic Fragments

6.1.2.4 Quaternary Alluvial Wash Deposits Transitional (Qwt)

According to previous work, the percentage of volcanic rock fragments gradually increases downward in the mineral deposit. As a result, Noranda and Stingray further subdivided the mineralized horizon into an upper and lower unit based on the percentage of volcanic fragments present in the material. On this basis, the lower mineralized unit was designated Cgi+v (granite + volcanic), but it was also referred to as “Transitional Conglomerate”. In this report, the previous designation has been changed to Qwt or Quaternary alluvial wash deposits transitional (Figure 6-7). This material is similar in size distribution and poor cementation to Qwu and exhibits the same tendency to disaggregate into a “pre-crushed” size distribution when split in core.

The Qwt unit ranges in thickness from 10 m to greater than 150 m, but it does not crop out anywhere in the El Pilar deposit area. Based on observed variations in the elevation of the Qwu/Qwt contact and photos of core alleged to represent Qwu or Qwt, it is likely that the criteria used to differentiate the two units were not applied in a consistent manner during core logging. In core photos and in the drill holes examined in 2006, Qwt appears to be dominated by debris flow deposits composed of angular fragments of intrusive and volcanic rocks in a matrix of sand and clay-sized particles very similar in form to Qwu. Qwt is red brown in color due to the presence of hematite and limonite in the matrix and as coatings on some rock fragments. Similar to Qwu, Qwt is interpreted to consist of debris flows emplaced in the proximal portion of an alluvial outflow wash. XRD analysis of material believed to be from Qwt shows the principal minerals in the +10-mesh fraction as quartz, K-feldspar, plagioclase, mica/illite and smectite (Table 6-1). Whereas Qwu is similar in whole rock chemistry to granite, Qwt more closely resembles andesite.

Table 6-1: Mineralogy of Qwt by Sieve Size (%)

Head Column 14	-200	150x200	100x150	65x100	48x65	35x48	28x35	20x28	16x20	10x16
Quartz	17	22	26	24	26	27	26	29	29	29
K-feldspar	15	12	13	18	18	18	20	23	21	20
Plagioclase	8	10	11	11	12	13	14	15	16	14
Muscovite/Illite	6	12	9	9	10	11	9	9	14	11
Chlorite	0	0	0	0	0	0	0	0	0	0
Smectite	28	20	22	23	20	17	16	12	10	13
Kaolinite	0	2.5	0	1.5	1.5	1.5	2.5	1.5	0	2.5
Clinoptilolite	18	12	10	2.5	2.5	6	2.5	2.5	2.5	2.5
Hematite	1.5	3	3	3	1.5	1.5	1.5	1.5	1.5	1.5
Unidentified	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5

Source: SCC (Mercator 2011)

Similar to Qwu, Qwt also contains Cu mineralization that is primarily associated with the oxide minerals chrysocolla, conicalcite and ajoite. Very minor remnant chalcocite, chalcopyrite, bornite, and possibly tetrahedrite also exist. The Cu mineralization primarily occurs as coatings to veins within clasts and as fine disseminations free within the sediment matrix. Cu is also structurally bound within smectite, biotite, and iron oxides. Again, similar to the Qwu unit, the Cu mineralization within the Qwt is primarily related to erosion of the exposed mineralized breccia zone.

Source: SCC (Mercator 2011)

Figure 6-7: Typical Qwt Core Samples - (note red-brown color and angular fragments of volcanic rocks)

6.1.2.5 Quaternary Alluvial Wash Deposits Upper (Qwu)

The upper part of the mineral deposit, designated as Qwu, crops out in the bottom of an arroyo near coordinates 3,449,100 N, 532,100 E (Figure 6-8 and Figure 6-9). Qwu is a poorly sorted, clast supported conglomerate with clasts of granodiorite porphyry, aplite, granite porphyry, mineralized silicic breccia, and minor felsic and unaltered andesitic volcanics in a matrix that ranges from 10 mesh to -200 mesh. Qwu is red to tan in outcrop and drill core, due to the presence of hematite and limonite in the matrix and as coatings on some rock fragments. Many fragments, including all breccia, several porphyry types, aplite, and felsic volcanic rock fragments, show evidence of hypogene mineralization including quartz and quartz + chalcopyrite veins, quartz + sericite + pyrite (QSP) alteration, and quartz veins with potassic alteration envelopes. Based on the presence of the porphyritic intrusive rocks and associated hypogene alteration and mineralization, it is concluded that mineralization in the Qwu is in part derived from the erosion of a porphyry Cu deposit, but much of the Cu mineralization is believed to be derived from the proximal exposed mineralized breccia zone.

Source: SCC (Mercator 2011)

Figure 6-8: Typical Qwu Core Samples – (note large, mineralized breccia fragments)

As determined by petrographic and X-ray diffraction (XRD) studies, the matrix of the Qwu consists of quartz, K-feldspar, plagioclase, chlorite, biotite, smectite, fine-grained hematite, limonite and clinoptilolite. The mineralogical composition by sieve size is shown in Table 6-2). Cu mineralization within the Qwu is present as abundant chrysocolla, with lesser conichalcite ($CaCuAsO_4OH$) and ajoite $(Na,K)Cu_7AlSi_9O_{24}(OH)_{63}(H_2O)$ and very minor remnant chalcocite, chalcopyrite, and bornite, and possibly tetrahedrite. The Cu mineralization is primarily in the form of chrysocolla, which occurs as both coatings to, and veins within, clasts and as fine disseminations free within the matrix. Cu also occurs together with illite as replacements of plagioclase phenocrysts in granite, quartz monzonite and granodiorite and structurally bound within smectite, biotite, and iron oxides. According to cross sections, in the mineral resource area Qwu ranges in thickness from a few meters up to 80 m.

Source: SCC (Mercator 2011)

Figure 6-9: Qwu Out Crop Consisting of Poorly Sorted and Angular Wash Material

Table 6-2: Mineralogy of Qwu by Sieve Size (%)

Head Bulk Sample	-200	150x200	100x150	65x100	48x65	35x48	28x35	20x28	16x20	10x16	1/4x10m	3/8x1/4	1/2x3/8	3/4x1/2	Pulverized Head
Quartz	28	29	33	33	30	25	32	34	32	33	34	34	34	28	36
K-feldspar	16	18	23	25	25	23	27	28	26	27	29	30	32	25	29
Plagioclase	16	17	18	20	18	17	18	18	17	18	19	20	17	30	18
Muscovite/Illite	8	8	5	7	7	10	5	6	7	5	5	7	7	7	6
Chlorite	0	2.5	1.5	0	5	5	2.5	2.5	5	2.5	2.5	5	5	5	2.5
Smectite	13	11	5	2.5	2.5	2.5	2.5	2.5	5	2.5	0	0	0	0	0
Kaolinite	1.5	0	0	0	0	0	0	0	0	1.5	0	0	0	0	0
Clinoptilolite	11	8	2.5	2.5	2.5	10	5	5	7	5	0.3	0	0	0	1.5
Hematite	1.5	1.5	2.5	2.5	1.5	2.5	1.5	1.5	1.5	1.5	1.5	1.5	1.5	0	1.5
Unidentified	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.5

Source: SCC (Mercator 2011)

6.1.2.6 Quaternary Alluvial Fan Deposits Younger (Qfy)

The El Pilar Cu deposit is overlain by up to 120 m of unmineralized proximal alluvial fan material deposited as debris flows (Figure 6-10). These alluvial fan sediments are made up of angular to sub-round, clast-supported fragments derived from nearby intrusive rocks set in an unconsolidated coarse sandy matrix. Qfy is thickest in the southern and eastern portions of the El Pilar Deposit area and thins toward the north, particularly in the northernmost section of the proposed pit where it provides a thin (<10 m) cover over the underlying intrusive rocks.

Source: SCC (Mercator 2011)

Figure 6-10: Typical Qfy Core Samples

6.2 MINERALIZATION

6.2.1 Cu Mineralization

The El Pilar Cu deposit is about 2,300 m long N33°E, and approximately 750 m wide, NW-SE with a mean thickness of 110 m (range 5 m to 220 m) and plunges 28° to the southeast. Approximately 98% of the Cu mineralization at El Pilar is hosted by Quaternary alluvial flow wash deposits proximal to bedrock and the remainder is within the E-W-trending mineralized breccia.

Prior to erosion, the breccia is interpreted to have been much larger than its present extent. It appears that proximal erosion of this mineralized, but now oxidized breccia, into the nearby range front depression during weathering was the primary source of the El Pilar Cu deposit. Although this erosional/depositional event was largely dilutive of Cu mineralization, because many of the larger clasts in the sedimentary material are comprised of unmineralized older intrusive and possibly younger volcanic rocks, total Cu grades are very consistent throughout the sedimentary sequence.

Total Cu grades are very consistent throughout the core of the mineralized alluvial body at El Pilar. Once the 0.30% total Cu envelope is encountered in drilling, essentially all the material within that envelope grades above 0.30% total Cu and the grades consistently fall within a narrow range of approximately 0.30% to 0.50% total Cu. As a precursor to a discussion of other Cu assay grades variations at El Pilar, it is first important to understand some relative Cu grade terminology. Very importantly, whereas some of the Cu at El Pilar is tied up in minerals that are soluble under laboratory acid soluble assay tests, some of the Cu at El Pilar is not easily soluble under the same acid laboratory procedures. Cu that is assay soluble is referred to in this report as acid soluble Cu (soluble Cu). The less assay soluble Cu is referred to herein as residual Cu. The combination of both soluble Cu and residual Cu is referred to as total Cu. The other important Cu grade designation at El Pilar is the ratio of soluble Cu to total Cu, or the Soluble Cu Ratio. This ratio is referred to as the Ratio or percent soluble Cu. The formula for calculating the Ratio is soluble Cu percent divided by total Cu percent.

$$Soluble\ Cu\ Ratio = \frac{Soluble\ Cu\ (wt.\%)}{Total\ Cu\ (wt.\%)}$$

For the 2007 and 2008 drilling, approximately 60% of the drill hole database core samples were assayed by ALS Chemex using a four-acid digestion procedure and approximately 40% of the samples were assayed by METCON using a three-acid digestion. Comparison of the two sample sets showed that the results were equivalent at each lab. For the 2016 drilling, all samples were analyzed using the three acid-digestion. As discussed later in the metallurgical section, this resulted in some differences in soluble Cu and residual Cu assay results.

At El Pilar the relative amounts of soluble Cu and residual Cu, and as a consequence the Soluble Cu Ratio, vary noticeably from the surface of the deposit downwards. Mean soluble Cu grades decrease with depth, whereas residual Cu bench average grades increase. Due to the additive effects of changes in both the soluble Cu and residual Cu grades, the Ratio or percentage of soluble Cu declines from an average of about 64% soluble Cu near the top of the deposit to 26% soluble Cu at the bottom. These grade changes by bench translate to a similar drop in the percentage of soluble Cu encountered over the mine life, as progressively deeper benches are mined.

As mentioned previously, total Cu grade is very consistent throughout the El Pilar deposit and average bench total Cu grades do not change significantly with depth. This is because as soluble Cu grades decline downward residual Cu grades increase by about the same amount.

6.2.2 Mineralogy

In the upper portions of the El Pilar deposit, Cu mineralization is comprised principally of the visible blue Cu oxide mineral chrysocolla. Chrysocolla occurs as thin to thick coatings up to 1.0 centimeter (cm) thick on larger rock fragments. It also occurs in small microcrystalline patches between rock-forming minerals and as free chrysocolla grains in the finer size fractions. Chrysocolla is also intimately intergrown with chalcedonic quartz.

Further updated details on the mineralogy and metallurgy can be found in Section 10.0 of this TRS.

6.3 DEPOSIT TYPES

El Pilar lies within the Sonora-Arizona Porphyry Cu Province, about 45 km northwest of the Buenavista del Cobre (BVC) Cu mine, a Grupo Mexico deposit. BVC is the largest porphyry Cu deposit in Mexico and one of the largest in the world (Figure 6-11). Both El Pilar and BVC are situated in a highly prospective belt of Cu deposits that range from La Caridad in the south through to central Arizona. The main types of economic Cu deposits in this belt are related to porphyry Cu systems and mineralization typically occurs in hydrothermal breccia pipes and as disseminations and stockworks. Deposits like BVC also exhibit significant replacement mineralization at contacts with limestone, as well as vein type deposits and high-grade pegmatite zones. This belt hosts numerous very large, or what are considered to be “world class”, Cu deposits. These large deposits range from the BVC Cu mine (7.1 billion tonnes @ 0.42% Cu) in the south to the Morenci Cu mine of Phelps Dodge Corp. in the north (4.7 billion tonnes @ 0.52% Cu). The Sonora-Arizona porphyry Cu trend accounts for the second largest concentration of porphyry Cu deposits in the world and mining for Cu in this trend has been continuous for over 100 years.

The El Pilar property hosts an unusual exotic and/or transported Cu resource that is atypical for the area. Copper at El Pilar is hosted by range front alluvial wash deposits with clasts of intrusive, porphyry and highly silicified rock derived from a proximal, exposed hydrothermal breccia zone. The mineralization is interpreted to have been derived at least in part from that breccia. Reconstruction of events suggests that the breccia was mechanically weathered and eroded, transported and deposited in a channel and alluvial fan sequence that overlies a lower more indurated alluvial wash unit (Qwl).

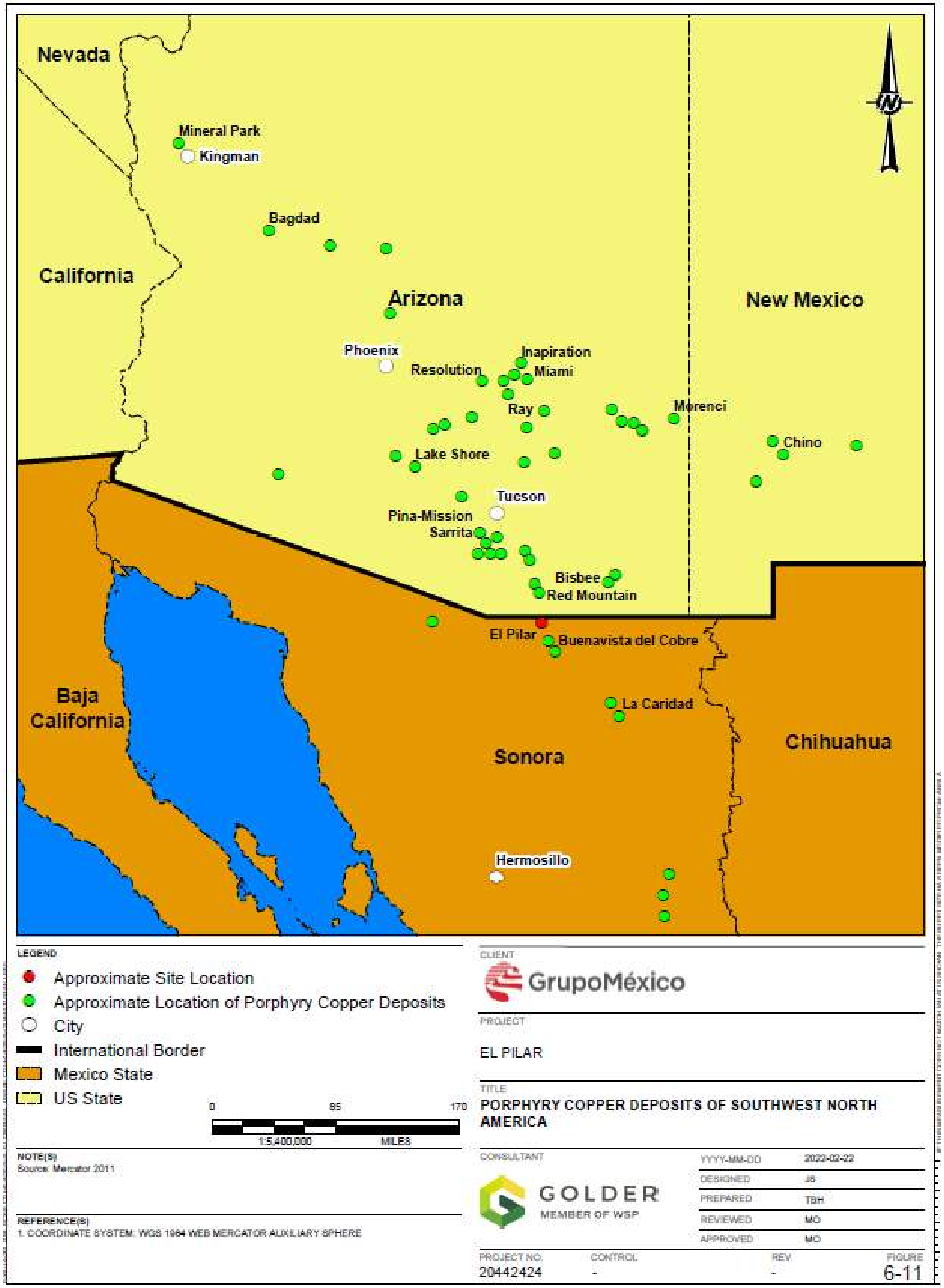


Figure 6-11: Porphyry Cu Deposits of Southwest North America

7 EXPLORATION

7.1 EXPLORATION WORK

As presented in Section 5.1 of this TRS, the Project area has been subject to several historical and recent exploration campaigns targeting Cu mineralization at the Project site. These exploration campaigns included a combination of surface exploration, surface geophysics, surface geological mapping, topographic surveys, exploration drilling, hydrogeological drilling, and geotechnical drilling. A high-level summary of the historical and recent exploration campaigns is presented in Table 7-1.

Table 7-1: Summary of Exploration Campaigns

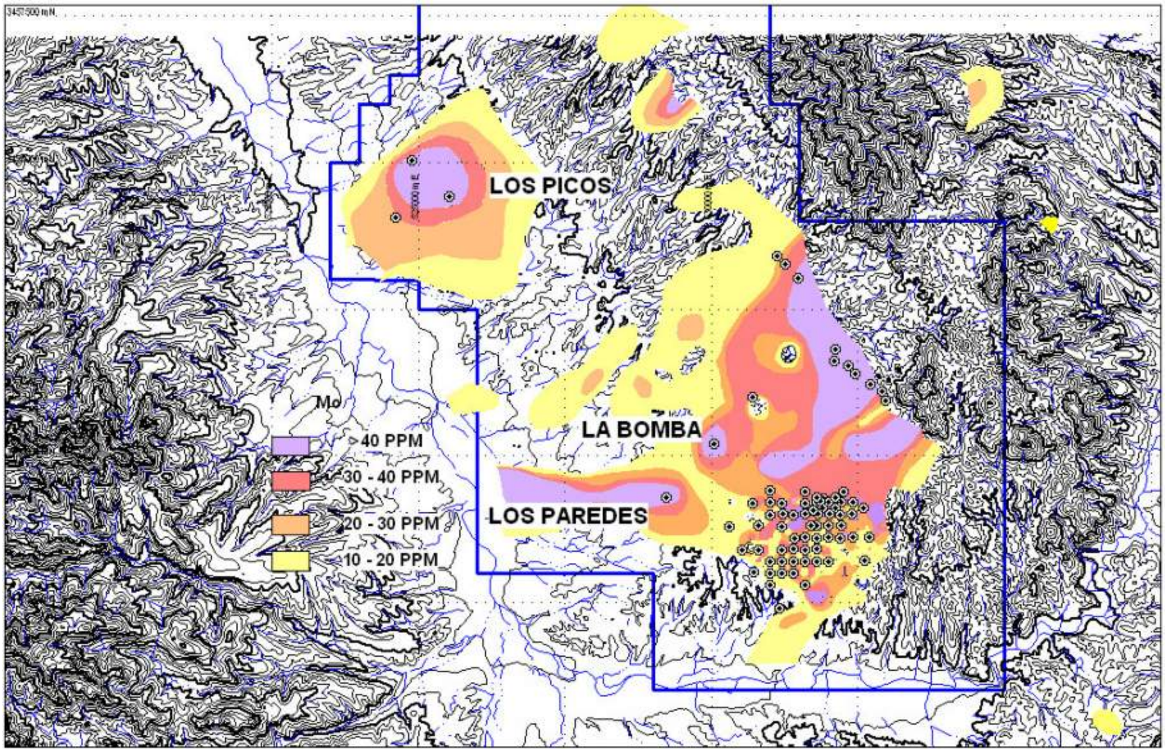
Year	Operator	Type of Exploration Work
1998-1999	Freeport	Regional Mapping, Rock & Vegetation Sampling, Geophysical (CSAMT) Surveying
1999		Exploration Drilling (RC)
1999		Semi-Reconnaissance Mesquite Survey
2000-2001	Noranda	Exploration Drilling
2001		Geophysical Survey
2001		Detailed Mesquite Survey
2002		IP Survey
2003		Magnetic Survey
2004		Exploration Drilling
2007	Stingray	Exploration Drilling
2008		Exploration Drilling
2010		Bulk Sample
2016		Exploration Drilling

7.1.1 Surface Exploration

From 1998 to 1999, Freeport, under agreement with Noranda, carried out an exploration program that included regional mapping, rock and vegetation sampling, and geophysical (CSAMT) surveying. This exploration program concluded with a short reverse circulation drilling campaign consisting of a total of 1,560 m drilled in eight (8) drill holes. This program encountered mineralization in three holes, with one in bedrock and two in conglomerate.

In 1999, a semi-reconnaissance mesquite survey was conducted at El Pilar by Freeport's staff. Approximately 117 samples were collected and analyzed by Actlabs-Skyline via hashing/ICPMS. The sample density for this survey was three to four mesquite samples per square km. The results of this survey clearly outlined the structurally controlled El Pilar bedrock mineralization, the oxide mineralization in the deposit area, and indicated the possibility of blind mineralization in alluvium covered areas.

Detailed and reconnaissance mesquite surveys were conducted in 2001 over the El Pilar oxide mineralization and along the western and eastern alluvial covered pediments of the Sierra San Antonio. A total of 340 mesquite samples were collected on a 100 m X 100 m grid over and around the El Pilar discovery area. A total of 218 mesquite samples were collected for the reconnaissance survey utilizing a density of approximately two (2) samples per square km. In the detailed mesquite survey anomalous Mo characterized the outcropping mineralization to the northeast. Cu in mesquite clearly outlines the drill indicated oxide mineralization in the northeast portion of the sampling grid. Concentrations range from 90 parts per million (ppm) to 193 ppm in a background of 60 ppm. Anomalous Cu extends to the west following an east-west ridge, which may reflect a major structure. This west trending anomaly is open to the north and west. Weaker Cu anomalies trend to the southwest and west creating a horseshoe pattern around the sub-cropping conglomerate high (Figure 7-1).



Source: SCC (Mercator 2011)

Figure 7-1: Regional Mesquite Sample

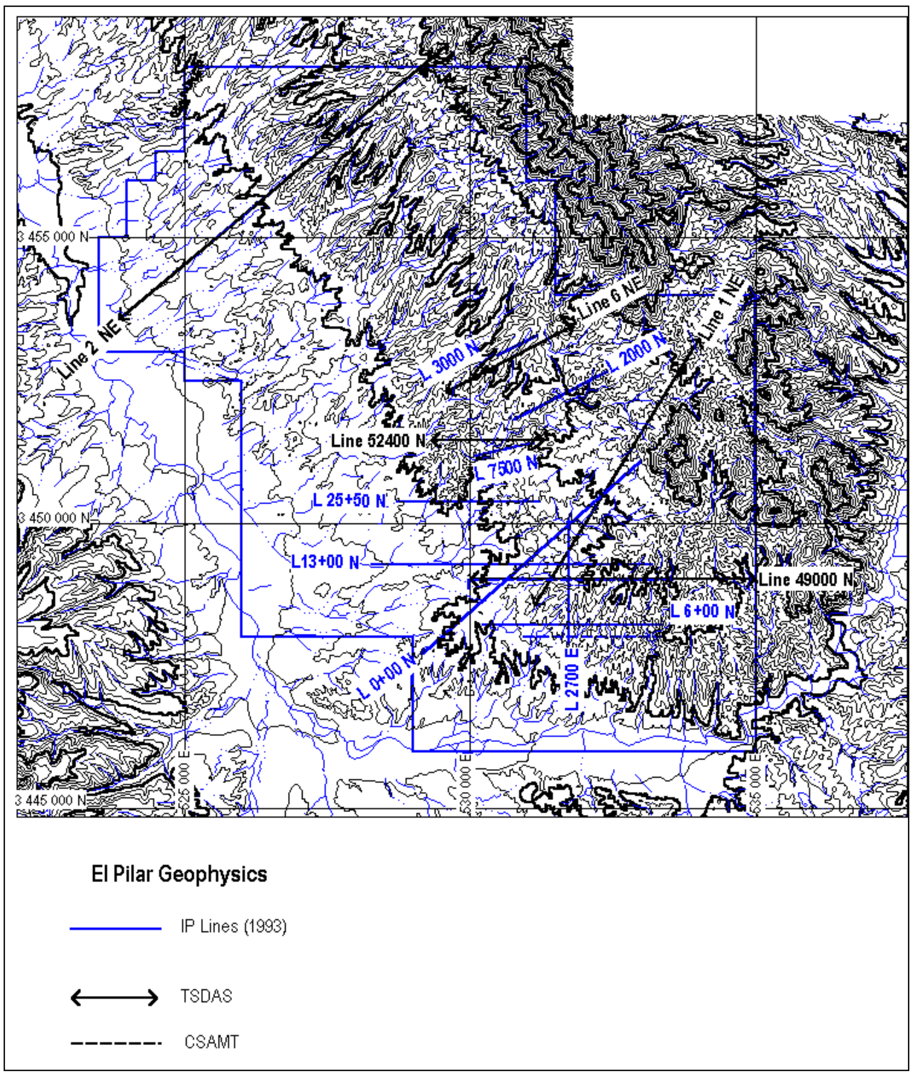
In general, the results of the reconnaissance survey indicated the following:

1. Moderate to strongly anomalous Mo and Cu concentrations in mesquite twigs characterized the concealed El Pilar oxide resource.
2. A strongly anomalous west trending Mo anomaly, Los Paredes, occurs west of the El Pilar discovery area.
3. Weak to moderately anomalous Mo and Cu values reflect outcropping mineralization in the area of the old El Pilar shaft as well as along the bedrock alluvium contact in the area.
4. A moderate Mo and weak Cu anomaly were outlined northwest of the El Pilar shaft in an area of past drilling by Cyprus. Another weak Cu anomaly is located west of this anomaly in pediment south of La Bomba Arroyo.
5. An extensive moderate to strong Mo anomaly was outlined in alluvium in the Los Picos zone. Mo concentrations ranged from 19 to 52 ppm. This anomaly appears to originate at a topographically projected northwest trending range front structure. Additionally, weak Cu in mesquite was outlined along the bedrock-alluvial contact to the northeast.
6. Two weak to moderate Cu and Mo anomalies were outlined along the north end of the reconnaissance survey at the Normex claim boundary (Arroyo El Tubo).
7. Strongly anomalous Cu and Mo characterize the outcropping mineralization in the shear zone of El Pilar de Arriba area. Mo values ranged up to 228 ppm and Cu ranged from 100 to 186 ppm.

The mesquite survey anomalies at Los Paredes, La Bomba and Los Picos were drill tested in 2004 with negative results by Normex.

7.1.2 Geophysical Surveys

From September to November 2001, six lines totaling 21.6 km were selected for surveying, employing the Titan 24 MT-DCIP Time Series Distributed Array System (TSDAS). The survey configuration utilized twenty-four 100 m spaced in-line dipoles along with twelve 100 m spaced cross line dipoles (see Figure 7-2). Cross-line (perpendicular to the line) 100 m spaced dipoles were placed every 200 m, or every second station. The QARA (centre-pole) array was used to collect the DCIP data with 100 m current injection points located at the midpoint of each in-line receiver dipole. Current injections were also placed off the ends of the array on lines that required more than a single spread (Line 1NE in this case) to provide some overlap information. Data acquisition and results obtained by Quantec were very high quality and the survey objectives were met. Interesting targets requiring follow-up were identified on Lines 1NE and L51400N. The target on line 1NE was field checked and found to be hosted by weakly altered volcanics. On line 51400N, interpretation of the MT and DCIP resistivity surveys over the center of the large biogeochemistry anomaly suggested that the depth of cover was beyond the limits of an economically viable conventional porphyry deposit. However, it did not rule out near surface mineralized conglomerates as the source of the biogeochemical anomalies situated over areas of deep overburden.

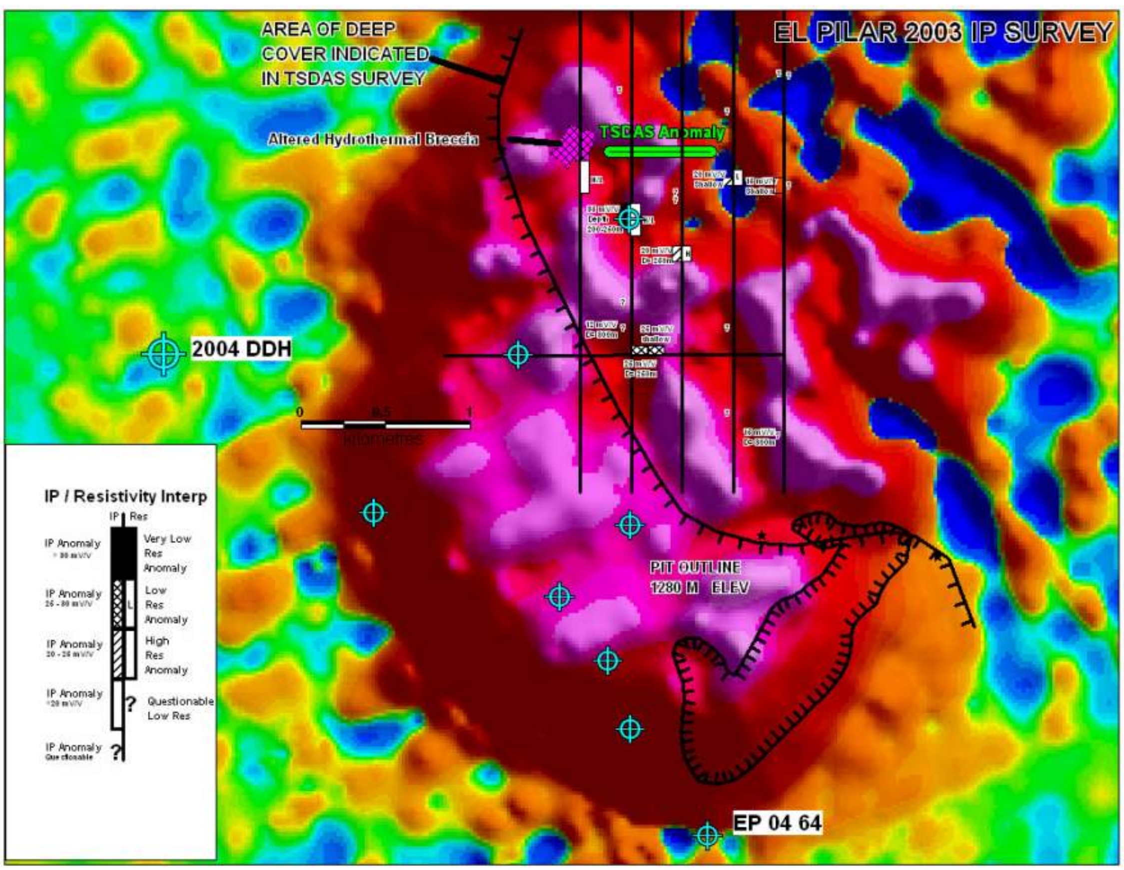


Source: SCC (Mercator 2011)

Figure 7-2: Titan 24 MT-DCIP Line Locations

A dipole - dipole IP survey with 200 m A-spacing was conducted in 2003 over the bedrock area north of the conglomerate resource on 300 m spaced lines (Figure 7-2). The survey was conducted by Pacific Geophysics of Vancouver, Canada using a six-channel time domain IP/resistivity receiver. A significant TSDAS anomaly had been detected over the bedrock area from the TSDAS survey. This anomaly occurred in the vicinity of an intensely altered hydrothermal breccia that is heavily oxidized and contains iron oxide veinlets and fracture coatings with 'live limonite', indicating the presence of Cu prior to the weathering. The IP survey indicated a reasonably strong chargeability source (35 mv) at a roughly 250 m depth southeast of the hydrothermal breccia. This feature was drilled during the 2004 drill campaign, and a pyritic fault zone was encountered with trace amounts of Cu in otherwise weakly altered intrusive.

In 2003, a helicopter borne Magnetic Survey was flown over the El Pilar property by McPhar Geosurveys Ltd. of Sunderland, Ontario Canada (Figure 7-3). The survey was flown perpendicular to the regional trend of the ranges with the towed magnetic sensor at an elevation of 30 to 40 m. Line spacing was 250 m and a total of 675-line km were flown, including tie lines. With the increased magnetic detail in the El Pilar deposit area, it was possible to outline the extent of the more magnetic volcanic clast unit (Qwl), which underlies the Cu bearing intrusive clast conglomerate. Since the TSDAS and Titan surveys both indicated that the depth to bedrock west of the edge of the outcrop was probably >500 m, the high contrast magnetic features were interpreted to be due to near surface sources. The magnetic high in the vicinity of the Cu resource coincided extremely well with the location of the volcanic clast unit (Qwl) paleohigh known from drilling and described earlier in the Mineralization section of this report. It was therefore thought that the entire magnetic feature west of the edge of the outcrop could be due to an apron of the more magnetic volcanic clast unit (Qwl). This would signify the probability that the mineralized intrusive clast unit (Qwu) existed in the local areas of lower magnetism surrounding the magnetic highs as was the case in the El Pilar deposit area.



Source: SCC (Mercator 2011)

Figure 7-3: Dipole-Dipole Time Domain IP

It was known from the 2001 drill program that low grade mineralization occurred west of the volcanic clast unit (Qwl) paleohigh. The 2004 drilling program utilized this data and the high contrast magnetic features to plan drill holes on the flanks of the magnetic highs west of the known paleohigh. Other drill holes in the 2004 drilling campaign targeted the mesquite anomalies from the regional survey.

Stingray's drill program of 2007 and 2008 proceeded to in fill drill the resource area and to test the southern limit to the Cu mineralization.

7.1.3 Bulk Sampling

In 2010, a 600-tonne bulk sample was collected at El Pilar for ROM metallurgical crib and column testing. The sample was collected from the only outcrop of the mineralized Qwu unit in the project area. The results of dry screening and assaying of the sieve sizes done at METCON Laboratories in Tucson are shown in Table 7-2. About 88% of the sample is comprised of material that is +100 mesh and only 3.5% of the sample is material +100 millimeter (mm) in size (6"). In essence, this size distribution of material is considered to be "pre-crushed" when considering the difference between run- of-mine (ROM) leaching versus crushing. It is important to note that the bulk sample pre-crushed material size distribution was the result of digging and transportation with no blasting; effectively these materials were so poorly cemented that they fell apart into segregated particles simply by being disturbed. Also, total Cu contents increase significantly in the finer the screen fractions, whereas the ratio of soluble Cu / total Cu was similar, irrespective of screen size.

Table 7-2: Qwu Bulk Sample Size Analysis & Assays

Screen Analysis			Sample: Head Bulk Composite Size: As received at METCON Research				Analytical Results			
Nominal openings			Sample Weight (kg)	Weight Distribution (%)	Cumulative (%)		Cu Assays			
mm	Inch	Tyler Mesh			Retained	Passing	Total Cu	Soluble Cu	Residual Cu	Ratio
	8		0.00			100.00				
152.40	6		80.00	3.47	3.47	96.53	0.177	0.141	0.036	0.797
100.00	4		93.00	4.03	7.50	92.50	0.452	0.371	0.081	0.821
75.00	3		12.62	0.55	8.05	91.95	0.297	0.211	0.086	0.710
50.00	2		61.90	2.69	10.74	89.26	0.287	0.163	0.124	0.568
31.50	1 1/4		231.50	10.04	20.78	79.22	0.363	0.238	0.125	0.656
25.00	0.984	1	53.88	2.34	23.12	76.88	0.346	0.224	0.122	0.647
19.00	0.748	3/4	240.50	10.43	33.55	66.45	0.391	0.226	0.165	0.578
12.50	0.492	1/2	119.00	5.16	38.71	61.29	0.438	0.297	0.141	0.678
9.50	0.374	3/8	136.00	5.90	44.61	55.39	0.466	0.317	0.149	0.680
6.30	0.248	1/4	211.00	9.15	53.76	46.24	0.465	0.331	0.134	0.712
1.70	0.067	10	495.00	21.47	75.23	24.77	0.551	0.388	0.163	0.704
1.00	0.039	16	92.86	4.03	79.26	20.74	0.583	0.453	0.130	0.777
0.84	0.033	20	26.11	1.13	80.39	19.61	0.637	0.530	0.107	0.832
0.60	0.023	28	50.60	2.19	82.59	17.41	0.701	0.560	0.141	0.799
0.42	0.017	35	43.10	1.87	84.46	15.54	0.758	0.630	0.128	0.831
0.30	0.012	48	38.71	1.68	86.14	13.86	0.881	0.700	0.181	0.795
0.21	0.008	65	31.84	1.38	87.52	12.48	0.915	0.780	0.135	0.852
0.15	0.006	100	28.48	1.24	88.75	11.25	0.925	0.770	0.155	0.832
0.11	0.004	150	34.51	1.50	90.25	9.75	0.821	0.680	0.141	0.828
0.07	0.003	200	29.03	1.26	91.51	8.49	0.724	0.600	0.124	0.829
Minus		-200	195.75	8.49	100.00					
Totals			**2,305.40**	100.00						
Calculated Assay							**0.537**	**0.382**	**0.128**	**0.746**

Source: SCC (Mercator 2011)

7.2 GEOLOGICAL EXPLORATION DRILLING

7.2.1 Exploration Drilling Methods and Results

Exploration drilling programs targeting Cu mineralization on the Project have been implemented by Freeport (1998-1999), Noranda (2000-2001, 2004) and Stingray (2007-2008, 2016). Primarily core drilling techniques have been used during each of the exploration drilling programs, with the exception being the reverse circulation (RC) drilling done by Freeport. A total of 316 drill holes totaling 71,825 m of drilling have been completed to date on the Project.

A summary of the RC and core drilling completed during the various drilling programs is presented in Table 7-3. A drill hole location map is illustrated in Figure 7-4.

Table 7-3: Exploration Drilling Summary – Geological

Year	Operator	Type	No. of Holes	Meters
1998	Freeport	RC	7	1,369
2000	Noranda	HQ Core	32	6,023
2001	Noranda	HQ Core	17	3,663
2004	Noranda	HQ Core	5	933
Subtotal Freeport/Noranda			**61**	**11,988**
2007	Stingray	HQ Core	98	19,408
2008	Stingray	HQ Core	96	21,414
2016	Stingray	PQ Core	61	19,015
Subtotal All Stingray			**255**	**59,837**
Grand Total			**316**	**71,825**

Pre-Stingray drilling amounts to 61 holes representing 11,988 m drilled by Freeport and Noranda between 1998 and 2004. In terms of total meters drilled, this represents 17% of the total drilling to date. Previous reports, such as the 2006 Woods Report, represent pre-Stingray drilling as 82 holes and 15,483 m of drilling, but this total includes drilling that does not pertain to the El Pilar deposit, such as the El Pilar de Arriba area, which is a few kilometers away.

The pre-Stingray drilling was summarized in the 2006 Woods Report. Freeport conducted an exploration program in 1998 and 1999 which concluded with a short drilling campaign consisting of a total of 1,369 m drilled in seven (7) RC holes. This program encountered mineralization in three holes, with one hosted in bedrock and two in conglomerate.

Noranda drilled 54 HQ (63.5-millimeter [mm] core diameter) core holes during three exploration programs in 2000, 2001 and 2004. All the 49 core holes drilled in 2000 and 2001 were within the target deposit area. Five (5) 2004 drill holes were outside of the main deposit area, testing an extension of the mineralized area. The report states that Noranda also drilled 19 RC holes, but they were not in the immediate El Pilar area. Details of the core and RC drilling procedures from the Pre-Stingray drilling were not available for review, nor was Golder provided with the Freeport or Noranda drilling logs. Golder is therefore relying on the information documented in the historical reports.

During 2007 and 2008, Stingray drilled 194 HQ core holes representing 40,822 m. In 2016, Stingray drilled an additional 61 PQ (85-mm core diameter) core holes, totaling 19,015 m. However, 14 of the 61 drill holes were used for condemnation drilling surrounding the main deposit, and 5 for regional exploration. None of these 19 holes were included in the model, or in the drill hole database provided to Golder. The Stingray drilling represents 83% of the drill hole database in terms of meters drilled. The purpose for the drilling was to: 1) validate the Freeport and Noranda drilling, 2) extend the resource base, and 3) collect samples to perform metallurgical testing over the entire deposit.

No additional drilling was conducted at El Pilar from 2017 through 2021.

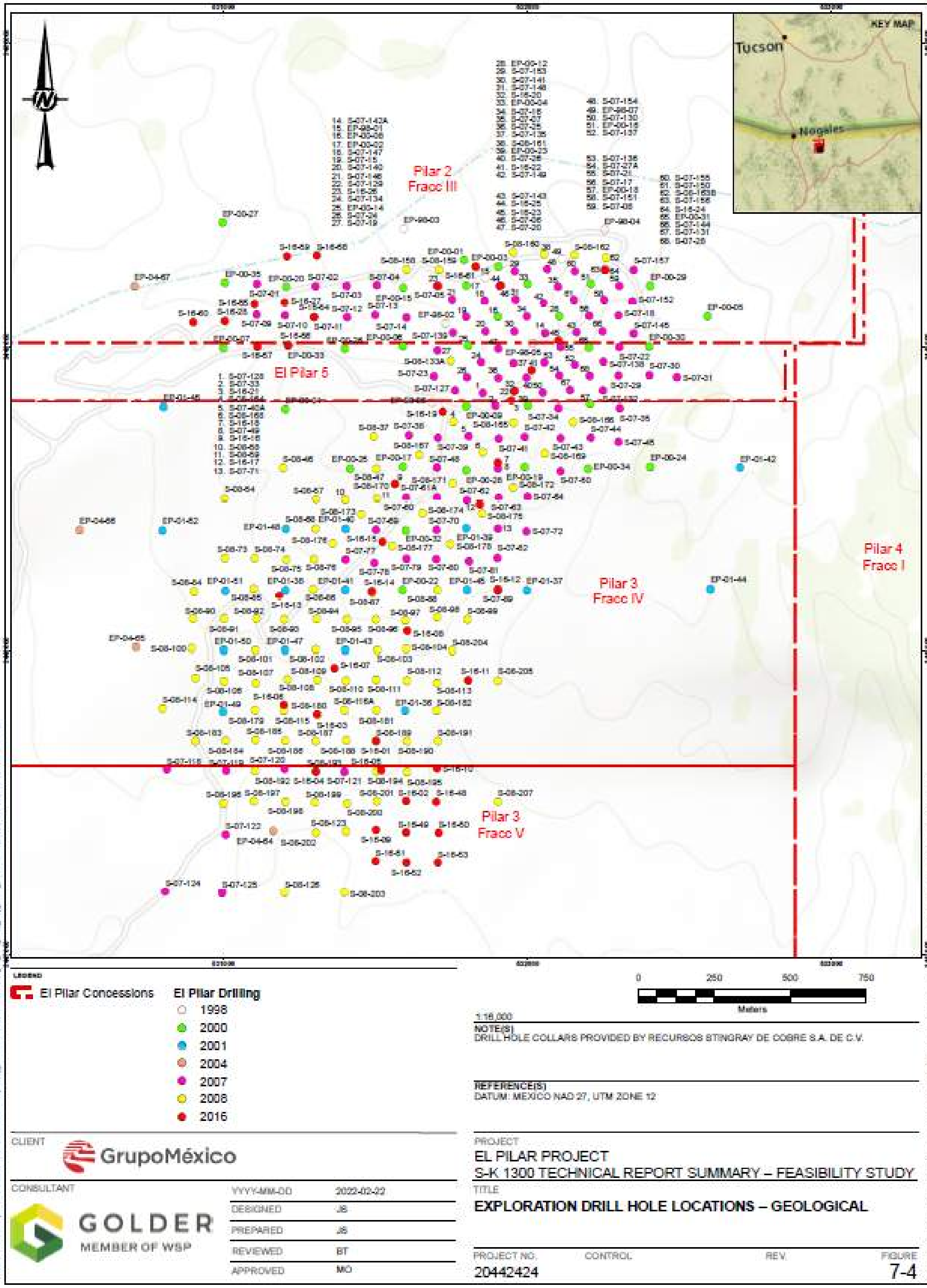


Figure 7-4: El Pilar Drill Hole Locations

7.2.2 Exploration Drill Sample Recovery

As summarized in the Woods report, for the pre-Stingray drilling, core recovery was estimated for all intervals. Golder was provided with the 2012 drill hole database from Stingray, which included core recovery for all pre-Stingray drilling and the 2007 and 2008 Stingray drilling. Upon review, Golder determined that approximately 5% of the drill holes were missing this information in the database.

For the 2016 Stingray core drilling programs, core recovery was recorded for each cored interval. Core recovery then was determined by measuring the recovered linear core length and calculating the recovered percentage using the sample weight divided by the length multiplied by the lithology density and volume of the core. The core recovery values were recorded by the logging geologist. Stingray used the densities determined by Noranda to calculate the core recovery. Rock quality index (RQD) was not recorded for any of the drilling programs.

Overall mean core recovery for all units was 87.3%. Core recovery was lowest in the Qfy overburden unit (62.0%); however, in the mineralized units (Qwu, Qwt, Qwl, and IBX) the mean core recovery was between 88.5% and 97.1%. Table 7-4 summarizes the mean core recoveries for by drilling program and by unit.

Table 7-4: Summary of Mean Core Recovery by Drilling Program and Unit

Year	Operator	Mean Core Recovery	Mean Core Recovery By Unit					
			Qfy	Qwu	Qwt	Qwl	Bx	PCGr
2000	Noranda	90.4%	73.3%	92.9%	98.4%	98.7%	92.5%	97.8%
2001		90.0%	64.1%	94.1%	98.7%	99.4%		
2004		91.4%	78.0%	96.9%	99.8%	100.0%		
2007	Stingray	75.6%	28.5%	74.0%	84.2%	92.8%	71.0%	80.8%
2008		83.4%	37.9%	80.3%	88.5%	96.5%	76.6%	86.9%
2016		93.1%	90.3%	93.0%	95.4%	95.0%	90.6%	91.5%
Mean		**87.3%**	**62.0%**	**88.5%**	**94.2%**	**97.1%**	**82.7%**	**89.3%**

The Golder QP considers the core recovery for the Noranda and Stingray core drilling programs to be acceptable based on statistical analysis, which identified no grade bias between sample intervals with high- versus low-core recoveries. On this basis, the Golder QP has made the reasonable assumption that the sample results are reliable for use in estimating mineral resources.

7.2.3 Exploration Drill Hole Logging

For the 2007, 2008 and 2016 drilling, drill hole logging was conducted by core logging geologists at the Stingray core logging and storage facility (Figure 7-5). All core samples have been geologically logged to a level of detail to support appropriate Mineral Resource estimation, such that there are lithological intervals for each drill hole, with a correlatable geological/lithological unit assigned to each interval. The core drill holes from all the core drilling programs were also geotechnically logged to a level of detail to support appropriate Mineral Resource estimation.

Additionally, all drill core boxes were photographed during logging and the photo stored electronically for reference. Examples of core photos from each of the Stingray drilling programs are shown in Figure 7-6 through Figure 7-8.

Source: Golder QP Site Visit, August 2021

Figure 7-5: Stingray Core Logging and Storage Facility

Source: SCC (Stingray 2021)

Figure 7-6: Example Core Drill Hole Photo (S-07-148, Qwu)

Source: SCC (Stingray 2021)

Figure 7-7: Example Core Drill Hole Photo (S-08-102, Qwt)

Source: SCC (Stingray 2021)

Figure 7-8: Example Core Drill Hole Photo (S-16-21, Qwl)

7.2.4 Exploration Drill Hole Location of Data Points

Drill hole collar locations were determined by the Stingray geologists. At the completion of drilling, the drill casing was removed, and the drill collars were marked with a permanent concrete monument with the drill hole name recorded on a metal tag on the monument. All drill holes were surveyed by a professional surveyor (Geo Ingeniería) in Mexico North American Datum 1927 (NAD 27) coordinates.

7.2.5 Exploration Drill Hole Data Spacing and Distribution

Drilling at El Pilar has been completed on essentially a north-south, east-west grid system, with drill holes situated at approximately 100 m x 100 m. Infill drilling in the north portion of the deposit has reduced the distance to approximately 65 m.

The QP considers the drill hole spacing sufficient to establish geological and grade continuity appropriate for a Mineral Resource estimation.

7.2.6 Relationship Between Mineralization Thickness and Intercept Lengths

All the drilling at El Pilar was done vertically. The Cu deposit is tabular, with dips ranging from flat to approximately 24 degrees, so thickness of mineralization from the samples is approximately 10% greater than the true width. Based on a 0.10% total Cu grade as the threshold of interesting mineralization, the median mineralized width is approximately 93 m. Of the 316 total drill holes, 286 include intervals of greater than 0.10% total Cu, and 173 include a mean mineralization over 0.10% total Cu for the entire drill hole. A summary of the approximate distribution of the mineralized width is as follows:

- About 5% of holes have less than 5 m or no assays over 0.1% Cu
- About 19% of holes indicate a mineralized thickness between 5 and 45 m
- About 24% between 45 m and 90 m
- About 18% between 90 m and 120 m
- About 34% of the holes indicate a mineralized width greater than 120 m

7.2.7 QP Statement on Exploration Drilling

The QP is not aware of any drilling, sampling, or recovery factors that could materially affect the accuracy and reliability of the results of the historical or recent exploration drilling. The data are well documented via original digital and hard copy records and were collected using industry standard practices in place at the time. All data has been organized into a current and secure spatial relational database. The data has undergone thorough internal data verification reviews, as described in Section 9 of this TRS.

7.3 HYDROGEOLOGICAL DRILLING AND SAMPLING

The core logging reports of all drill holes carried out in the different exploration campaigns were reviewed and none of the reports reported the presence of water, so it is expected that the bottom of the pit does not come in contact with groundwater. However, as the pit mining progresses, rainwater or groundwater flows could be captured during the rainy season.

For additional hydrogeological sampling and data verification details see Section 9.2.3.

7.4 GEOTECHNICAL DRILLING AND SAMPLING

Golder completed a geotechnical study in support of the Feasibility Study (FS) in 2008. The scope of work included completion of a site investigation, site characterization, and stability evaluations suitable to support FS level pit slope designs. Geotechnical units mentioned in this section correspond to the geological units presented in Section 6.1. The geotechnical exploration study results are summarized in Sections 7.4.1 through 7.4.6 of this TRS.

7.4.1 Geotechnical Surface Mapping

Intrusive (PCGr) outcrops in the north area of the pit do not generally exhibit well-developed or systematic structure. Representative structures found in outcrops along the ridge line to the north of the pit area were mapped and orientations were plotted, but these are not expected to provide a reliable representation of subsurface structural conditions.

Natural slopes developed in drainages within Overburden (Qfy) were characterized as lightly cemented sands, and slope heights and angles were documented to assist in characterizing the performance of slopes within this material.

7.4.2 Geotechnical Core Drilling Program

Rock mass and structure data was collected from six drill holes located along the South, East, and North boundaries of the pit (Figure 7-9 and Figure 7-10) that were drilled between April 28 and June 7, 2008. Core hole locations were selected in cooperation with Stingray personnel to provide representative samples from the principal geologic units that will be exposed in the ultimate pit slopes. Core holes were inclined to intersect the walls of the Year 12 pit design.

Landdrill S.A. de C.V. performed the drilling under contract to Stingray, using triple-tube coring equipment. Core orientation was undertaken in the PCGr unit in the north slope of the pit using the A.C.T. system provided by International Directional Services of Chandler, Arizona. All other lithological units are unconsolidated and without geologic structure, and are, therefore, not suitable for orientation.

Data collection tasks associated with the drilling program included geotechnical core logging, core orientation, and collection of samples for laboratory testing. Golder geologists Ian Thomsen and Jesse Laurie were onsite throughout the program to carry out these tasks with the assistance of Stingray geologists.

The seven geotechnical drill holes totaled 1,319 m. Locations, orientations, lengths, and the geological units intercepted are summarized in the Table 7-5 and illustrated in Figure 7-10.

Table 7-5: Summary of Geotechnical Drill Holes

Core Hole	Easting	Northing	Nominal Azimuth (°)	Nominal Dip (°)	Total Depth (m)	Primary Formations Encountered
GT-01-08	531,800	3,449,200	0	75	230	Qfy, Qwu, PCGr
GT-02-08	532,000	3,449,200	0	45	79.5	Qfy, Qwu
GT-02A- 08	532,500	3,449,300	0	75	201	PCGr
GT-03-08	532,500	3,449,100	270	45	273	Qfy, Qwu, Qwt
GT-04-08	531,950	3,448,200	0	90	273	Qfy, Qwu, Qwt
GT-05-08	531,400	3,447,500	0	90	262.5	Qfy, Qwu, Qwt
GT-05A- 08	531,400	3,447,500	0	90	99	Qfy, Qwu, Qwt

Core hole GT-02-08 was lost due to drill rods lodged in the hole at 79.5 m. Core hole GT-02A-08 was located approximately 100 m North of the lost hole and drilled to the target depth of 200 m. Core hole GT-05A-08 was drilled parallel to core hole GT-05-08 to a depth of 99 m in response to poor recovery over that interval of GT-05-08. The poor recovery was a function of incorrect drilling equipment and the start of the geotechnical drill program.

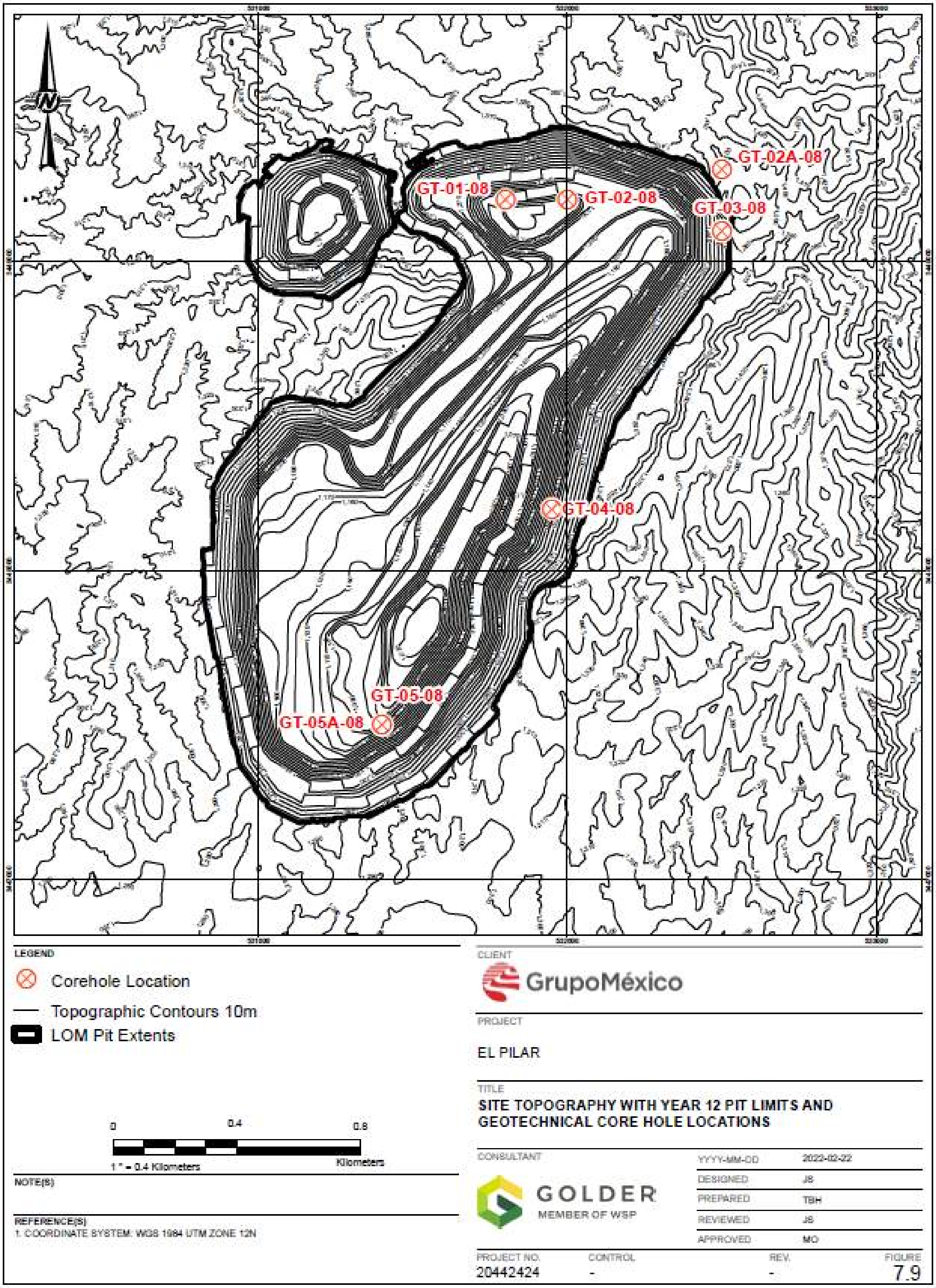


Figure 7-9: Site Topography with Year 12 Pit Limits and Geotechnical Core Hole Locations

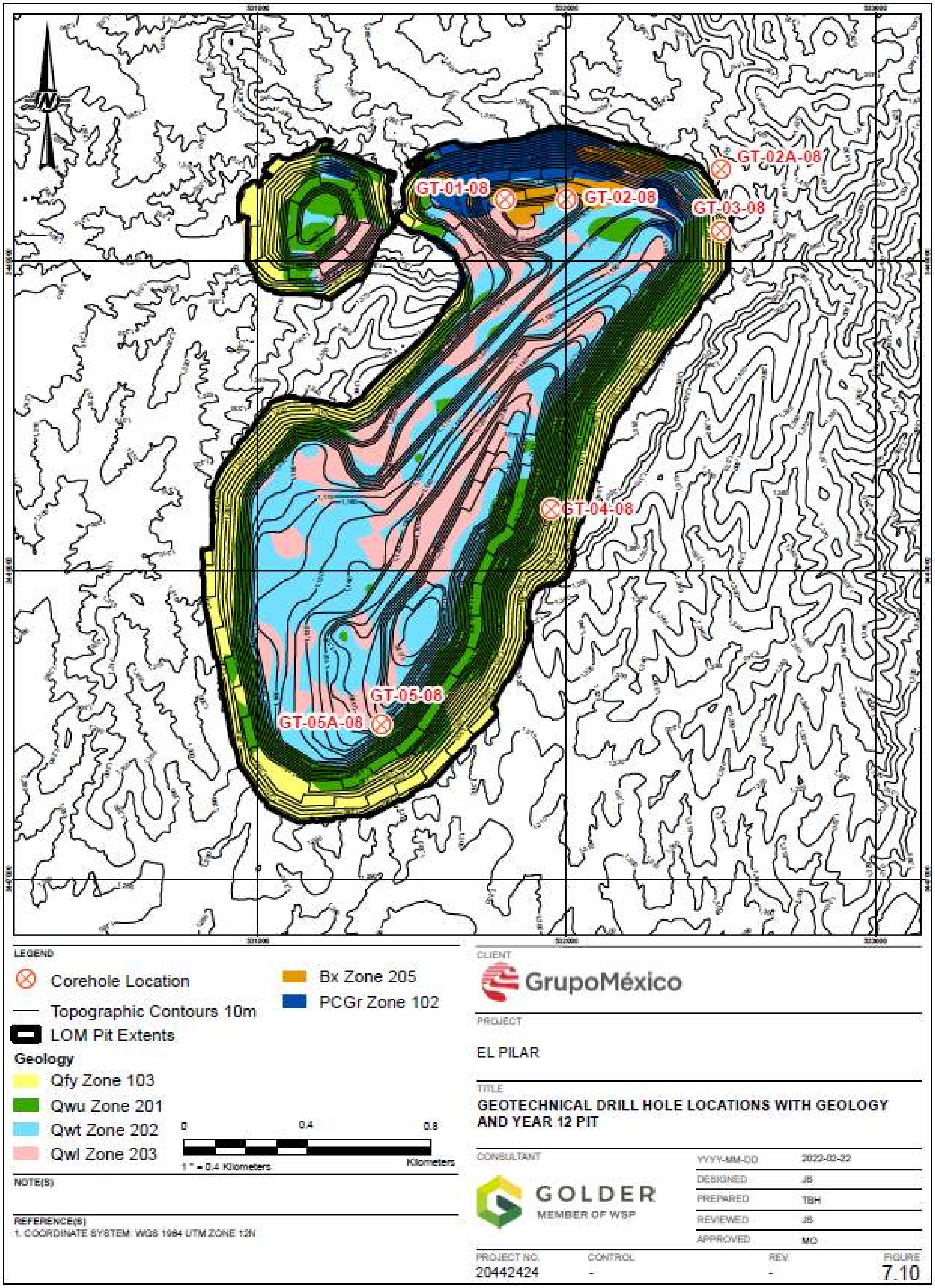


Figure 7-10: Geotechnical Drill Hole Locations with Geology and Year 12 Pit

Geotechnical core logging completed for each run in all drill holes included the following parameters:

- Core recovery
- Rock Quality Designation (RQD)
- Fracture frequency
- Rubble and gouge zones
- Joint Condition Rating (JCR)
- Detailed description of discontinuity characteristics
- ISRM strength index

Field strength tests were not performed on PCGr because in view of its generally high strength and the modest slope heights, limited laboratory testing of compressive strength was considered to provide sufficient information regarding PCGr strength to preclude the possibility of slope failures caused by low intact rock strength.

The following data was collected for each oriented discontinuity in areas of competent rock:

- Depth
- Discontinuity type
- Dip angle with respect to the core axis (α)
- Circumferential angle between top of core and the apparent "dip direction" of the discontinuity (β; measured clockwise from top of core to point on fracture surface furthest downhole)
- JCR
- Infill type
- Infill thickness
- Degree difference between orientation line of current run and previous run measured clockwise from the upper run orientation line to the lower run orientation line

The quantity of data collected from these core holes is summarized by core hole and by lithology in the Table 7-6.

Table 7-6: Types of Geotechnical Data Collected from Core by Unit

Unit	RQD (m of core)	JCR (m of core)	Fracture Frequency/ Joint Spacing (m of core)	Holes Encountered
Qfy	0	0	95.36	GT- 01,02,02A,03,04,05,05A
Qwu	3	0	74.74	GT-01,02,03,04,05,05A
Qwt	17	0	9.47	GT-03,04,05,05A
PCGr	175	11	95.36	GT-01,02A

Orientation intervals were assigned a "confidence level" based on the differences in orientation between successive runs as follows:

1. High reliability based on alignment of successive core orientations
2. Good reliability based on core alignment, but without alignment of successive orientation runs
3. Indeterminate reliability, based on single orientation runs that appear to be reliable
4. Low reliability, data from orientation runs with unknown reliability

7.4.2.1 Field Strength Classifications

ISRM field strength classifications assigned to transported (Ofy and Conglomerate (Qwu, Qwt)), PCGr, and Breccia (Bx) core are summarized in the Table 7-7.

Table 7-7: Summary of Field Strength Classifications

Rock Strength	Qwu		Qwt		Bx		PCGr	
	Recovered Length (m)	Relative Abundance (%)	Recovered Length (m)	Relative Abundance (%)	Recovered Length (m)	Relative Abundance (%)	Recovered Length (m)	Relative Abundance (%)
S0					1.1	1		
S1	2.8	1						
S2	67.6	18	15.4	5	4.6	5		
S3	178.8	48	132.7	41	20.8	22		
S4	75.7	20	114.1	35	14.7	15		
S5	43.5	12	38.9	12	10.2	11		
S6	2.7	1	23.2	7	9.1	10		
R0					2.3	2	12.4	5
R1					10.1	11	27.2	12
R2					13.7	14	40.6	17
R3					9.3	10	140.6	60
R4							13.2	6
R5								
R6								
Total Length	**371**		**324.2**		**96**		**234.1**	

Transported materials (Qfy, Qwu, Qwt, and Qwl) are characterized by soil strengths, while PCGr is generally medium strength rock. Conglomerates (Qwu and Qwt) generally classify as firm to stiff soils, while Qfy colluvium was loose and samples were generally recovered in a highly disturbed state that could not be assigned a strength. Bx strength classifications cover a broad range, with approximately 65% of the samples characterized by soil strengths where the strength of the material is controlled by weak matrix, and the remainder characterized by rock strengths where the clasts are variably silicified and are not contained within a soil-like matrix.

7.4.3 Laboratory Testing

Representative samples for possible laboratory testing were collected from the geotechnical core holes and were shipped to the Golder Associates' soils testing laboratory in Denver, Colorado. Ten were used for classification testing, and five were used for triaxial compressive strength testing. In addition, five rock samples were sent to Advanced Terra Testing, Inc. (ATT) in Denver, Colorado, for unconfined compressive strength (UCS) testing, and one was sent for point load testing.

The laboratory testing program included the following:

- Sieve Analysis (ASTM D421, D422) - Sieve analyses were completed on 10 samples to determine the particle size distribution. Atterberg Limits (ASTM D4318) - Plastic Limit (PL), Liquid Limit (LL), and Plasticity Index (PI) tests were completed on 10 samples to aid in soil classification, comparison of materials, and prediction of engineering characteristics. Atterberg Limits tests provide a useful predictor of engineering behavior by the determination of the Liquid Limit (the water content of a soil at the boundary between the semi-liquid and plastic states), the Plastic Limit (the water content at the boundary between the plastic and semi-solid state), and the Plasticity Index (the range of water content over which a soil behaves plastically, given as the difference between the Liquid Limit and Plastic Limit).

- Triaxial Shear Tests (ASTM D4767) – Five Consolidated-Undrained triaxial shear strength tests with pore pressure measurements (CU/pp) were completed on representative core samples of Qwu, Qwt and Qfy. The CU tests were used to develop Total Stress and Effective Stress failure envelopes. Confining pressures were applied in stages. The results of testing were reported as a Mohr-Coulomb failure envelope defined by a cohesion and an angle of internal friction. The test results were used to support selection of shear strength parameters of the Qwu, Qwt and Qfy for slope stability analysis.
- Unconfined Compressive Strength Tests (ASTM D7012) – Five core samples were tested to measure the intact strength of the PCGr.
- Axial Point Load Text (ASTM D5731) – One sample was tested to help characterize the intact rock strength.

7.4.3.1 Soils Classification Testing

Soils classification test results are summarized in Table 7-8.

Table 7-8: Summary of Soil Testing Data

					Atterberg Limits			Grain Size Distribution		
Core Hole	Depth (m)	Lithology	USCS Soil Classification	Delivered Moisture (%)	LL	PL	PI	% Finer ¾"	% Finer #4	% Finer #200
GT-01	22.88 - 23.06	Bx	GC	--	29	19	10	86	54	15
GT-01	50.45 - 50.65	Bx	--	--	--	--	--	96	90	61
GT-01	100.69 - 100.85	Bx	--	--	--	--	--	73	19	0.4
GT-01	156.95 - 157.13	PCGr Gouge	SC	--	34	17	17	100	91	32
GT-03	81.5 - 81.64	Qwu	--	--	--	--	--	58	43	14
GT-05	14.56 - 14.73	Qfy	GC	--	31	23	8	95	57	15
GT-05	99.5 - 99.7	Qwu	SW-SM	--	NP	NP	NP	93	68	9
GT-05	247.01 - 247 - 12	Qwt	SC	--	52	26	26	100	94	14
GT-05A	50.58 - 50.77	Qwu	GC-GM	--	27	22	5	80	54	14
GT-05A	82.74 - 82.95	Qwt	GW-GM	--	32	27	5	84	50	7

All samples classified as medium plasticity to non-plastic clays/silts, with between 0.4% and 61% of each sample comprised of silt and clay size particles (<200# mesh). The index tests indicate a wide range of plasticity from non-plastic up to a PI of 26. These results indicate a high degree of variability. The samples ranged from clayey sands and gravels (SC and GC) to silty sands and gravels (SM to GM) according to the Unified Soil Classification system.

7.4.3.2 Triaxial Shear Test Results

Four-point staged triaxial tests were performed to measure sample shear strengths. Stages were conducted in order of increasing confining pressure: 25 psi, 50 psi, 75 psi, and 100 psi. Failure was defined as the maximum principal stress ratio. Staged testing of core samples will generally underestimate peak strengths because of the effects of shearing during testing.

"Undisturbed" core samples of Qwu and Qwt were used for testing. Qfy lacked cohesive strength to enable a testable sample to be recovered from core drilling, so a composite sample was prepared by scalping over-size material and remolding the sample to a moderated density.

The undrained (Total Stress) parameters for the Qwu and Qwt generally indicated higher strengths than the effective stress parameters because the test samples demonstrate a negative pore pressure response during loading. This is characteristic of heavily over-consolidated clays and clay-sands or compacted clay-gravel. Effective stress test results represent drained strength, which is appropriate for stability analyses, and are summarized as follows in Table 7-9.

Table 7-9: Summary of Triaxial Test Results (Effective Stress)

Core Hole	Sample Depth (m)	Parent Lithology	Dry Unit Weight (pcf)	Saturated Unit Weight (pcf)	Field Strength Classification	Cohesion (psi)	Friction Angle (°)
GT-03	147.0-147.2	Qwu	118.1	131.6	S4	27.6	41.0
GT-05	46.5-46.8	Qwu	121.3	130.4	S5	34.7	38.6
	Mean	***Qwu***	***119.7***	***131.0***		***31.5***	***39.8***
GT-03	267.5-267.7	Qwt	123.7	132.3	S4	21.0	42.4
GT-05	230.4-230.6	Qwt	120.7	133.6	S6	27.8	42.7
	Mean	***Qwt***	***122.2***	***133.0***		***24.4***	***42.6***
GT-03 and GT-05	2.8-2.9 25.9-26	Qfy (Composite remolded sample)	94.2	101.3		0	38.1

In general, these test results indicate high shear strengths for both the Qwu and the Qwt units, with little strength difference evident between these two units. Although the sample descriptions range from sandy clay with gravel, to sand with gravel and almost no clay, the responses of all samples were similar with regard to pore pressure generation and strength parameters.

The remolded Qfy sample is characterized by a slightly lower friction angle and zero cohesion. All Qwu and Qwt samples had particle sizes exceeding ASTM recommendations for specimen diameter and were difficult to trim due to oversize particles. This could lead to an over-estimation of soil strength by an unknown amount. However, the consistency of the four tests at each loading stage suggests that the tests overall are reliable.

7.4.3.3 UCS Test Results

Results for UCS testing are summarized in Table 7-10.

Table 7-10: Summary of UCS Test Results

Core Hole ID	Depth (m)	Lithology	Field Strength Classification	Failure Type**	Unconfined Compressive Strength	ISRM Strength Based on UCS
GT-01	161.18 - 161.37	PCGr Gouge	R1	F/S	80.0	R0
GT-02A	56.15 - 56.32	PCGr	R1	F/S	170.0	R1
GT-02A	71.07 - 71.27	PCGr	R3	F/S	4,260.0	R3
GT-02A	113.1 - 113.4	PCGr	R3	F	10,510.0	R3
GT-02A	195.32 - 195.53	PCGr	R3	F	3,890.0	R3

** Note: F = failure along fracture; F/S = part failure along fracture and part shear failure

The UCS test results indicate that the PCGr unit classifies as a medium strong rock. All samples failed, at least in part, along a pre-existing fracture surface, so UCS strength likely underestimates true rock strength.

The test result from sample GT-02A 56.15-56.32 m is not considered representative because it is so much lower than the other test results for the same unit. A suitable design UCS value for the PCGr is 6,200 psi based on the average compressive strength of the other three UCS samples.

7.4.3.4 Point Load Test Results

Test sample GT-01 83.31 – 83.49 m resulted in a Point Load Index, Is50, of 436. This corresponds to an estimated Uniaxial Compressive Strength of 10,850 psi using the commonly recommended conversion factor of 25. The field strength classification of this zone was R2.

7.4.3.5 Calibration of Field Strength Classifications with Laboratory Strength Testing

Laboratory test results correspond with field strength classifications as follows in Table 7-11.

Table 7-11: Comparison of Field Strength Classification with Laboratory Test Results

Field Strength Classification	Corresponding Strength Range (psi)	Number of Laboratory Test Samples (UCS and PL)	Average Laboratory Strength (psi)
R1	150-725	2 UCS	125.0
R2	725-3,500	1 PL	10,850.0
R3	3,500-7,500	3 UCS	6,220.0

This comparison indicates that the field strength classifications are generally reasonable to slightly conservative.

7.4.4 Rock Mass Rating

Rock mass rating systems provide a quantitative method of characterizing rock masses. Bieniawski's (1976) RMR classification system incorporates UCS, RQD, joint spacing (the inverse of Fracture Frequency), joint condition (as defined by JCR), and groundwater condition to characterize rock masses. Each of these parameters is assigned a numerical rating, and the sum of the rating values yields an RMR rating from 0 (very poor quality rock mass) to 100 (very good quality rock mass). The RMR classification system is summarized in the Table 7-12.

Table 7-12: Summary of Rock Mass Rating System (Bieniawski, 1976)

Parameter	Range of Values						
UCS (psi) **(Rating)**	>30,000 **(15)**	15,000-30,000 **(12)**	7,500-15,000 **(7)**	3,500-75,00 **(4)**	1,500-3,500 **(2)**	<450-1,500 **(1)**	150-450 **(0)**
RQD **(Rating)**	90-100% **(20)**	75-90% **(17)**	50-75% **(13)**	25-50% **(8)**	<25% **(3)**		
Joint Spacing **(Rating)**	>10ft **(30)**	3-10ft **(25)**	1-3ft **(20)**	0.2-1ft **(10)**	<0.2ft **(5)**		
Joint Condition **(Rating)**	Very rough No separation Hard wall rock **(25)**	Slightly rough Separation<1mm Hard wall rock **(20)**	Slightly rough Separation<1mm Soft wall rock **(12)**	Slickensides, Separation or gouge <5mm **(6)**	Soft gouge or Separation >5mm **(0)**		
Groundwater **(Rating)**	Completely Dry **(10)**		Moist **(7)**	Mod. Pressure **(4)**	Severe **(0)**		
Total RMR Value (Sum of Ratings for 5 Items) =							
Rating	100 – 81	80 – 61	60 – 41	40 - 21	20 – 0		
Description	I – Very Good	II – Good	III – Fair	IV - Poor	V - Very Poor		

The geotechnical core logs for El Pilar directly provide the data required for calculation of RMR. Table 7-13 summarizes the data used in calculating the RMRs of the geologic formations.

Table 7-13: Summary of RMR Data of the Geological Formations

Core Hole	Cross Section	Core Hole Length (m)	PCGr (m)	Bx (m)
GT-01-08	531,800E	230.0	56.0	118.5
GT-02-08	531,950E	52.5	-	25.5
GT-02A-08	531,950E	201.0	187.5	-
Totals		**483.5**	**243.5**	**144.0**

Table 7-14 summarizes the weighted average geotechnical parameters calculated from geotechnical core hole logs except for UCS, which is estimated from field strength classifications for Bx and laboratory strength testing for PCGr. The corresponding RMR rating value is listed below the geotechnical parameters, and a total RMR is given for each unit.

Table 7-14: RMR Ratings

Unit	Parameter	Strength	RQD	Joint Spacing	Joint Condition	Groundwater	Total RMR
PCGr	Value	6,200 psi	0.7	0.17 m	12	Dry	51
	Rating	5.0	14.0	10.0	12	10	
Bx	Value	2,175 psi	0.4	0.08 m	7	Dry	34
	Rating	2.0	9.0	6.0	7	10	

RQD values are a weighted average of logged data. Joint spacing was calculated from Fracture Frequency data. JCR is collected directly during core logging. The groundwater condition at the El Pilar is classified as "dry" based on our understanding that groundwater levels are below the pit bottom elevation. This also corresponds to the recommended designation of groundwater condition when RMR is used for rock mass strength estimates, as discussed subsequently.

The RMR rating for PCGr is 51, which corresponds to a Fair quality rock mass. Bx has an RMR rating of 34 and classifies as a poor quality rock mass.

7.4.5 Rock Mass Strength Estimates

A method of estimating rock mass strength based on Rock Mass Rating (RMR), uniaxial compressive strength, and rock type was developed by Hoek and Brown (1980) and was subsequently revised (Hoek and Brown, 1988). Hoek and Brown's rock mass strength criterion is based on empirical results and utilizes the RMR classification system developed by Bieniawski (1976). Bieniawski's RMR system has been revised several times; Bieniawski's 1976 RMR system should be applied with the Hoek-Brown (1988) empirical rock mass strength procedure.

The Hoek-Brown criterion is the most widely accepted method of estimating rock mass shear strength in rock masses comprised of brittle, fractured rock. This criterion defines the relationship between major principal stress and minor principal stress at the time of failure based on the following equation:

$$\sigma'_1 = \sigma'_3 + \sigma ci(m_b(\sigma'_3/\ \sigma_{ci}) + s)^a$$

where:

- σ'_1 and σ'_3 = maximum and minimum effective stresses at failure
- m_b = the value of the Hoek-Brown constant, m, for the rock mass
- s and a = material constants that depend on the rock mass characteristics
- σ_{ci} = the uniaxial compressive strength of intact rock

This is an empirical method, originally derived by fitting curves (i.e., shear strength envelopes) to the Mohr circle results from large-dimension triaxial compression tests on core samples of fractured rock. The curves represented shear strength envelopes for the fractured rock described in terms of "Hoek- Brown" parameters m, s, and a, which could be applied directly to rock masses in the field. This type of sampling is difficult, and the tests are costly, so an alternative approach to developing Hoek-Brown parameters was devised that makes use of testing that is more easily performed and the RMR ratings described previously. This process was used to develop estimates of rock mass shear strength as detailed in the following points:

- Estimate intact rock strength – UCS was estimated from laboratory tests and field strength classifications.
- Determine values for the constants mi and s for intact rock – The material constant mi can be obtained from triaxial testing; however, no triaxial test results are available for geotechnical units at El Pilar. The m_i parameter for intact rock can also be estimated from published typical values for specific rock types. Values of mi were selected from Hoek and Karzulovic (2000), as tabulated below.
- Determine value for constant a – while this value has typically been taken as 0.5 for brittle rock masses, Hoek's most recent recommendations (Hoek et. al, 2002) are to use a value of:

$$a = 1/2 + 1/6(e^{-GSI/15} - e^{20/3})$$

- Relate the material constants for intact rock (m_i and s) to material constants for "broken rock", or the field-scale rock mass (m_b, s) – This process involves application of a rock mass classification and disturbance parameter to reflect field rock mass conditions. In earlier versions of the Hoek-Brown criteria (Hoek and Brown, 1988), the rock mass classification parameter was based on Bieniawski's (1976) RMR. More recently, a parameter called the Geological Strength Index (GSI) was introduced for this purpose (Hoek and Karzulovic, 2000), based in part on the difficulty involved in estimating RMR for very poor quality rock masses. For more competent rock masses (RMR > 25), the GSI value is equal to the RMR.
- Values for mb and s vary with the amount of disturbance to the rock mass, which reflects the extent to which the integrity of the rock mass has been affected by blasting and excavation. Extremes in this regard reflect "disturbed rock", which is typical of blast-excavated mine slopes; and "undisturbed rock" such as could be expected in machine-bored tunnels. With careful controlled blasting, an intermediate degree of disturbance could be expected, and we used a conservative value of 100% disturbance to generate shear strength parameters. The following equations are used to calculate mb and s based on disturbance:

$$m_b = m_i e^{\left(\frac{GSI - 100}{28 - 14D}\right)}$$

$$s = e^{\left(\frac{GSI - 100}{9 - 3D}\right)}$$

  where:

  m_i = value of m for intact rock

  GSI = Design Geological Strength Index value (equal to RMR if RMR > 25)

  m_b = value of m for rock mass

  s = value of s for rock mass (assume 0 for RMR ≤ 25)

  D = disturbance factor that ranges from 0.7 for pit slopes with good, controlled blasting, to 1.0 for pit slopes with poor blasting

Based on this procedure and the design values of UCS and RMR, the following parameters (Table 7-15), which conservatively assume a disturbance factor of D = 1, were used to determine the design rock mass shear strength:

Table 7-15: Design RMR Parameters

Formation	UCS (psi)	RMR	Unit weight (pcf)*	mi**	mb	s	a
PCGr	6,200	51.0	162.0	29.0	0.87572	0.00028396	0.505
BX	2,175	34.0	145.0	14.0	0.12553	0.0000167	0.517

Note: * Unit weight for PCGr from laboratory testing; unit weight for Bx estimated
** Published values of mi from Hoek and Karzulovic (2000)

7.4.6 Structural Data from Oriented Core Holes and Surface Mapping

The structural orientation data collected from surface mapping and oriented core drilling at El Pilar were plotted on stereonets using equal-area, lower hemisphere projection and contoured using the Schmidt method. Major concentrations of structural orientations were identified, and the results are presented and discussed below.

7.4.6.1 Oriented Core Hole Data

Oriented structural data were obtained from zones of competent PCGr in inclined core holes GT-01 and GT-02A. A summary of the quantity and distribution of the data is contained in Table 7-16.

Table 7-16: Oriented Core Data Type by Core Hole

Core Hole	Joint	Fault	Total
GT-01-08	56.0	12.0	68.0
GT-02A-08	87.0	4.0	91.0
Totals	**143.0**	**16.0**	**159.0**

7.4.6.2 Surface Mapping

Orientation data was collected from ten surface outcrops of monzonite along the northern edge of the Year 12 pit (Figure 7-9). Only six joints were identified for which orientations were measured, reflecting the poor exposure of structures in these weathered outcrops. Plots of the joint orientations show a wide scatter and no evident correlation with the subsurface structural data, indicating that the poorly developed structures in the weathered outcrops are not representative of sub-surface structural conditions.

7.4.6.3 Identification of Prominent Structural Sets

Rock structure includes both large-scale structures, and systematic discontinuity sets (“rock fabric”). The contact feature/structure that forms the contact between the PCGr and the Qwu and Qwt units is a significant large-scale structure. Discontinuities consist of joints and smaller faults. Rock fabric was characterized by combining and plotting measured structural orientations from the two oriented core holes. The stereonet of the pole orientations was contoured to provide density plots of preferred structural orientations (Figure 7-11).

Figure 7-11 shows three main concentrations of poles representing joints or joints and faults. These concentrations were evident in oriented core data from both GT-01-08 and GT-02-08, indicating that they are likely developed throughout the PCGr. Mean orientations calculated for these sets are listed in the Table 7-17.

Table 7-17: Structural Data from Oriented Coring

Set ID	Dip (°)	Dip Dir. (°)	Peak Conc., (%)	Comments
1	48.0	113.0	8.6	Joints and faults
2	67.0	257.0	6.0	Joints
3	66.0	176.0	5.0	Parallel to regional fault separating intrusive and conglomerates; joints and faults

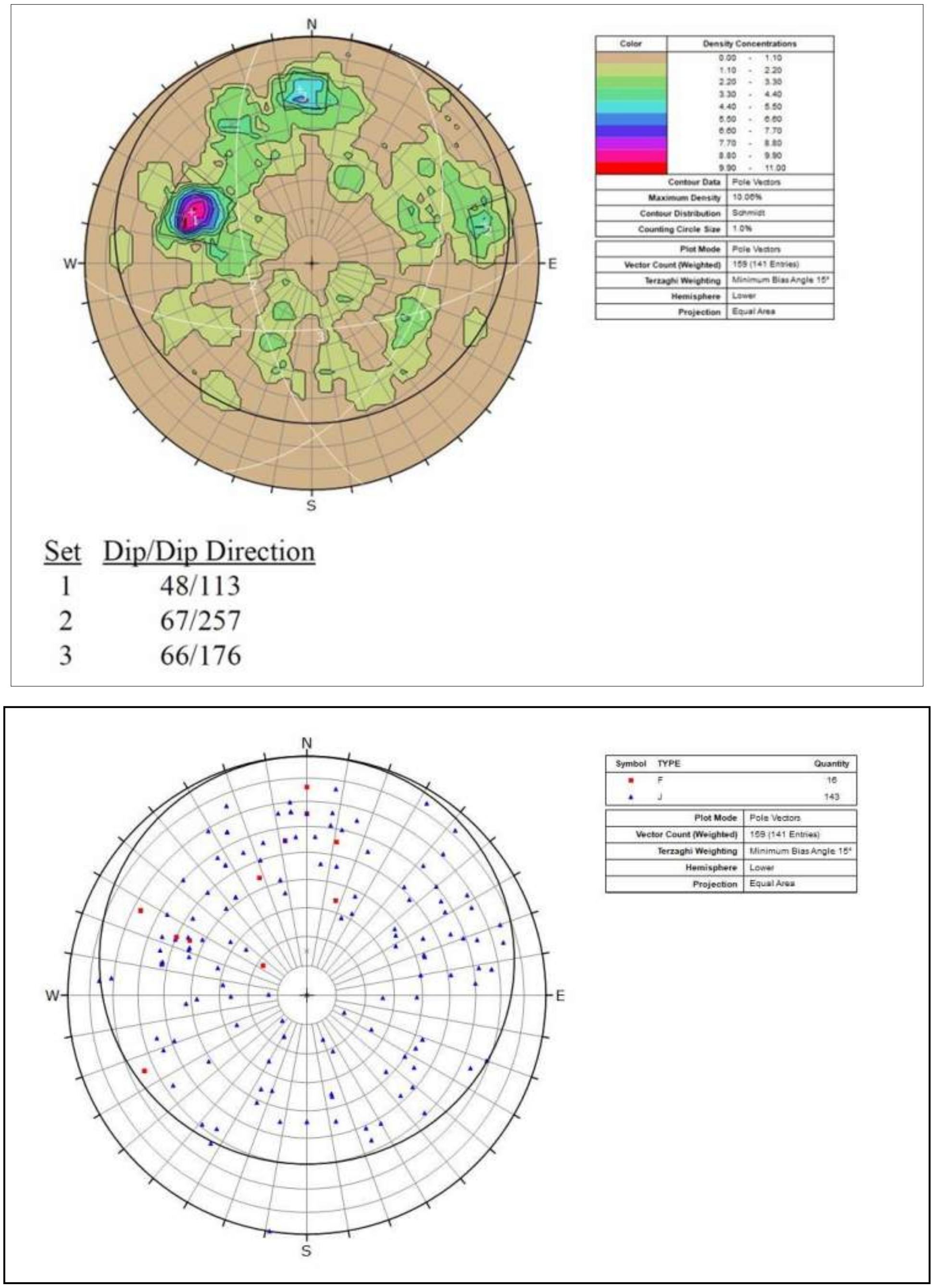


Figure 7-11: Structural Data from Oriented Core

Sets are labeled in Figure 7-11. Set 1 dips moderately southeast and is comprised primarily of joints that are caused through cooling and sub-tectonic settling of rocks after emplacement. These joints often mimic larger scale features that are controlling cooling features in PCGr rocks. Set 2 consists of joints only, and dips moderately to steeply west-southwest. Set 3 strikes east-west and dips moderately to steeply south. This set is parallel to the regional fault structure that places the PCGr against the Qwu and Qwt units.

The orientations collected from the surface mapping data do not correspond to any of the main structural sets identified in sub-surface structural data. This is a consequence of structural mapping on weathered surfaces where features are difficult to observe.

7.4.7 QP Statement on Geotechnical Drilling and Sampling

The QP is not aware of any drilling, sampling, or recovery factors that could materially affect the accuracy and reliability of the results of the historical geotechnical drilling and sampling. The data are well documented via original digital and hard copy records and were collected using industry standard practices in place at the time. All data has been organized into a current and secure spatial relational database. The data has undergone thorough internal data verification reviews, as described in Section 9 of this TRS.

8 SAMPLE PREPARATION, ANALYSES AND SECURITY

8.1 SITE SAMPLE PREPARATION METHODS AND SECURITY

All sampling was completed by geologists employed by Noranda or Stingray depending on the program; details of the exploration programs by operator and year are provided in Section 7.2 of this TRS. The QP was not directly involved during the exploration drilling programs or sample selection. Based on review of the procedures during the site visit and subsequent review of the data, it is the opinion of the QP that the measures taken to ensure sample representativeness were reasonable for the purpose of estimating Mineral Resources.

Several different sampling techniques have been used on the Project since 1998. The nature and quality of the sampling from the various sampling programs is summarized in the following sections.

8.1.1 Sampling Techniques and Preparation

8.1.1.1 RC Drilling

The QP could not verify the sampling techniques for the RC drilling completed by Freeport and Noranda during the 1998 drilling program. A limited number (seven) of these drill holes were completed on the Project in the project area, with only one (EP-98-05) intercepting significant mineralization. EP-98-05 is situated in an area of closely spaced (35 m to 55 m) infill drilling; and therefore, the QP is of the opinion that it has been verified by the subsequent drilling. During the 1998 drilling program, a total of 199 RC chip samples were collected.

8.1.1.2 Core Drilling

For the Pre-Stingray drilling, core samples were collected from HQ size drill core, on a mean 2.5 m interval. Core was split in half using a manual core splitter, and one half was sent in plastic numbered bags to Bondar Clegg's Lab for assaying. The remaining second half was kept in the core box and stored in a warehouse in the town of San Lazaro. Best efforts were made to avoid sampling bias but due to the nature of the mineralization (disseminated throughout the matrix but also concentrated in certain clasts) and the host rock (clasts from mm to 10s of cm) some degree of sampling bias was unavoidable.

The sampling protocol for the Pre-Stingray drilling at the El Pilar deposit was determined by geologic factors including rock type, visible Cu, alteration, and contacts between units. The length of the samples was 2 m when visible Cu mineralization was present and 2.5 m where Cu mineralization was not detected visually. In the intervals assumed to be barren a 2.5 m sample was taken every 12.5 m.

For the Stingray drilling, core samples were collected from HQ (2007 and 2008 drilling programs) and PQ (85 mm core diameter; 2016 drilling program) size drill core on a mean 2.5 m interval. Core was split in half using a manual core splitter, with one half placed in a numbered sample bag and the other retained in the core box. The samples were then transported to Hermosillo (Chemex) or Tucson (METCON) for the 2007 and 2008 drilling, and to Hermosillo (Bureau Veritas/Inspectorate) or Tucson (Skyline) primarily for the 2016 drilling, with an additional 1,264 samples sent to Copper State Analytical Lab Inc. (CSAL) in Prescott, Arizona.

8.1.2 Sample Results

To date there has been a total of 25,718 samples collected on the Project of which 25,519 samples are from the cored drill holes and 199 samples are from the RC drill holes. Included in this total are 3,907 Quality Assurance and Quality Control (QA/QC) samples. A summary of the sampling by year and operator is presented in Table 8-1.

Table 8-1: Summary of Sampling by Exploration Campaigns

Year	Operator	Laboratory	Total Samples	Mean Sample Thickness (m)
1998	Freeport	Bondar Clegg	199	2.36
2000	Noranda	Bondar Clegg	1,890	2.17
2001		Bondar Clegg	1,126	2.26
2004		ALS Chemex	184	2.41
2007	Stingray	ALS Chemex	4,565	2.23
		METCOM	1,933	2.94
2008		ALS Chemex	4,266	2.60
		METCOM	1,824	2.64
2016		Inspectorate	4,847	2.00
		Skyline	3,620	2.26
		CSAL	1,264	2.26
		Total	**25,718**	**2.35**

A summary of the assay samples by model unit, within the main deposit area, is included in Table 8-2. Table 8-2 excludes QA/QC samples and any outlier drill holes not included in the geological model.

Table 8-2: Summary of Assay Samples by Model Unit

Model Unit	Sample Count	Mean Sample Thickness (m)	Total Cu (wt. %)		Soluble Cu (wt. %)	
			Mean	Max	Mean	Max
Qfy	1,244	2.86	0.06	1.07	0.03	0.91
Qwu	8,721	2.37	0.22	2.43	0.08	2.19
Qwt	5,504	2.27	0.20	4.90	0.10	4.07
Qwl	1,527	2.05	0.03	0.55	0.01	0.46
Bx	1,103	2.30	0.27	6.44	0.19	5.67
PCGr	894	2.18	0.06	1.03	0.03	0.94
Total	**18,993**	**2.33**	**0.18**	**6.44**	**0.08**	**5.67**

International Mining Consultants (IMC), during their review in support of the 2011 NI 43-101, noted that the soluble Cu assays from the pre-Stingray versus Stingray drilling were different. The pre-Stingray soluble Cu assays used a stronger sulfuric acid concentration and higher temperatures for the digestion; they were significantly more aggressive than the Stingray soluble Cu assays. They indicated that the pre-Stingray and Stingray soluble Cu assays should be treated as two different assays and segregated into separate variables in the database. Golder reviewed the assay data set and chose to include both the Stingray and pre-Stingray soluble Cu data, as the QP felt the differences, would not significantly impact the Mineral Resource estimate. The pre-Stingray soluble Cu assays amounted to less than 10% of the total soluble Cu assays available. Several instances of absent values were infilled using regression.

8.1.3 Verification of Sampling and Assaying

To verify the sampling and assaying, both duplicate sampling and twinned drill holes were implemented for the Project.

During the 2004 Noranda drilling program and the 2007, 2008, and 2016 Stingray drilling programs, field duplicate/replicate samples were obtained. Two ¼ core samples were taken at the same time and were analyzed in sequence by the laboratory to assess the representativeness.

Twelve twin drill holes at the same site were drilled during the 2016 Stingray drilling program. The twin drill hole pairings comprised one pre-2016 drill hole and one 2016 drill hole, within 5 m of the original drill hole.

The QP reviewed the results of the duplicate/replicate sampling and twin drill holes. For the duplicate/replicate samples, the R^2 value is 0.954, which is very good. Visual observation of the lithological intervals and the assays for the twin drill holes show that they are very similar. The QP considers the samples to be representative of the in-situ material as they conform to lithological boundaries determined during core logging.

8.1.4 Sample Audits and Reviews

The QP reviewed the core and sampling techniques during a site visit in August 2021. The QP found that the sampling techniques were appropriate for collecting data for the purpose of preparing geological models and Mineral Resource estimates.

8.1.5 Sample Security

For the Pre-Stingray drilling, after splitting of the core, the samples were bagged by project geologists and driven to the sample preparation laboratory in Hermosillo by Noranda personnel. Sample preparation and shipment of the pulps to the Vancouver analytical laboratory was conducted by lab personnel. Noranda personnel were not involved in sample preparation.

For the Stingray drilling, core was collected at the drill rig by Stingray employees and transported to the core shed, a locked and fenced facility. The core was then logged, photographed, samples tagged with a water-resistant tag, split and put in sample bags and sealed. The Laboratories send their own transportation to pick up sample lots, each lot comprises all samples from one hole. Forms with the details of each lot and assays required is signed by the laboratory personnel receiving the samples. The samples are identified by sample number only, no hole ID is included. The remaining core is stored in the core warehouse in order of hole number and box number. Other than initial splitting of the core, Stingray personnel were not involved with sample preparation, or analysis.

8.2 LABORATORY SAMPLE PREPARATION METHODS AND ANALYTICAL PROCEDURES

8.2.1 Pre-Stingray Drilling

All analytical work was performed by the Bondar Clegg Lab in Vancouver, B.C., following sample preparation by Bondar's affiliate in Hermosillo.

The samples were crushed using a Rhino Crusher to a minimum of 75% passing -10 mesh. At least one sample in every order was sieved through a 10-mesh screen to verify that this standard was met. The crushed sample was then passed through a Jones Splitter, to provide a representative cut of 250 grams (g). The 250 g cut was packaged and labeled with a Bondar Clegg order number and sample number. The remaining crushed sample was re-bagged in a reject bag and also labeled with the order number and sample number. The 250 g sample was then pulverized in a ring and puck pulverizer to a minimum standard of 95% passing -150 mesh, re-bagged, and boxed for shipment to the Bondar-Clegg laboratory in Vancouver.

At the Vancouver laboratory, analysis was as follows:

- All samples were first analyzed for total Cu by Bondar-Clegg's code GA30 method. This is a four-acid digestion, hydrochloric, nitric, hydrofluoric, and perchloric, on a 0.25 g sample. Analysis is by atomic absorption spectrometry (AAS).

- For determining soluble Cu, samples that reported total Cu values over 1,000 ppm but less than 2,000 ppm were analyzed by Bondar-Clegg's code GA40 procedure. In this procedure 0.10 g of sample were digested in 30% hot sulfuric acid (80 degrees C), followed by AAS analysis.
- Soluble Cu analysis for samples that reported over 2,000 ppm total Cu was done by a sequential leach Cu procedure, code CUSL. A 0.25 g sample was first digested in 30% sulfuric acid at 80 degrees centigrade for 90 minutes with regular agitation. The solution was then extracted and analyzed by AAS. The residue was then rinsed and neutralized and leached in cyanide with final analysis on the solution by AAS.

The sequential analysis was discontinued early in the 2001 campaign. The method was somewhat expensive, and the incremental Cu extracted in the cyanide digestions step was small. Cyanide is not very effective for digesting chrysocolla, one of the main Cu minerals at El Pilar.

8.2.2 Stingray Core Drilling

Sample preparations and analysis for the 2007 and 2008 Stingray drilling was performed at two different laboratories. About 60% of the drilling analyses were done by ALS Chemex laboratories and about 40%, particularly the drilling used for metallurgical testing, were done by the METCON laboratory (part of SGS as of 2013) in Tucson, Arizona. Sample preparation for ALS Chemex was done at their Hermosillo, Mexico, facility after which the pulps were shipped to Vancouver for analysis.

Sample preparations and analysis for the 2016 Stingray drilling was performed at three different laboratories. About 50% of the drilling analyses were done by Inspectorate laboratories of Hermosillo, Mexico, a contract laboratory for Bureau Veritas. About 37% were completed by Skyline laboratory in Tucson, AZ, with an additional 13% at Cu State Analytical Laboratory in Prescott, AZ. Sample preparation for Bureau Veritas was done at their Hermosillo, facility, after which the pulps were shipped to Vancouver for analysis.

As part of the QA/QC work several hundred pulps were analyzed at both the ALS Chemex and METCON laboratories in 2007 and 2008 and at the Bureau Veritas and Skyline Laboratories in 2016.

The ALS Chemex sample preparation procedure consisted of logging the sample into the system, weighing and drying. The samples were then crushed to 70% passing 2 mm (Tyler 9 mesh or US 10 mesh). A split of 250 g was removed and then pulverized to 85% passing 75 microns (200 mesh). Periodic screen samples were done for the crushed and pulverized material to make sure that the standards were maintained.

Total Cu analysis was done by a four-acid digestion, nitric (HNO_3), perchloric ($HClO_4$), hydrofluoric (HF), and hydrochloric (HCl) acid, followed by analysis with Inductively Coupled Plasma - Atomic Emission Spectroscopy (ICP-AES). Soluble Cu analysis was done by leaching a 0.5 g sample with 3% sulfuric acid (H_2SO_4) for 15 minutes at room temperature. After leaching, analysis was by Atomic Absorption Spectroscopy (AAS).

The METCON sample preparation procedure consisted of drying the sample then crushing to a nominal 2 mm to 0.8 mm (-10 to -20 mesh). This was split and a portion pulverized to 80% passing 0.120 mm (-125 mesh). Total Cu analysis was done by a three-acid digestion of a 0.2 g pulverized sample with HCl, HNO_3, and $HClO_4$ acid. Final analysis was by atomic absorption (AA) or ICP. Soluble Cu analysis was done by leaching a 0.5 g pulverized sample in 5% H_2SO_4 for an hour, agitated and at room temperature. Final analysis was by AA.

The sample preparation procedures for the 2016 drilling were consistent across the three laboratories, following the procedures implemented at METCON (SGS), and included the following. Samples were logged into the system, weighed, and dried. The samples were then crossed to a 2 mm to 0.8 mm (-10 to -20 mesh), followed by pulverization to 80% passing 0.120 mm (-125 mesh) or better. Total Cu analysis was done by a three-acid digestion of 0.2 g pulverized sample with HCL, HNO_3, and $HClO_4$ acids. Final analysis was by AA or ICP. Soluble Cu analysis was done by leaching a 0.5 g pulverized sample in 5% H_2SO_4 for an hour, agitated and at room temperature. Final analysis was by AA.

Accreditation for the five laboratories used during the Stingray drilling are as follows:

- ALS Chemex laboratories are registered to ISO 9001:2015 standards and many are ISO/IEC 17025:2017 accredited.
- METCON (SGS) laboratories are registered to a minimum of ISO 9001:2000 standards and are ISO/IEC 17025:2017 accredited.
- Bureau Veritas/Inspectorate laboratories are registered to ISO 9001:2015 and are ISO/IEC 17025:2017 accredited.
- Skyline Laboratories are laboratories are registered to ISO 9001:2015 and are ISO/IEC 17025:2017 accredited.
- The QP could not confirm recent certification for Copper State Analytical Laboratory.

8.3 QUALITY CONTROL AND QUALITY ASSURANCE PROGRAMS

Several variations of QA/QC procedures were implemented on the Project for the various drilling programs. The QA/QC procedures for each program are as follows:

- Pre-Stingray drilling program information is limited, and it was determined that only a limited number of QA/QC samples, Standard Reference Material (SRM) samples and field blanks were inserted into the sample sequence.
- 2007 and 2008 Stingray drilling: SRMs, field blanks and field duplicates were inserted into the sample sequence every 10, 20 and 10 samples, respectively.
- 2016 Stingray drilling: Field blanks and duplicates were inserted into the sample sequence every 20 and 10 samples, respectively. SRMs were not used in 2016.

Table 8-3 summarizes the QA/QC sample counts by drilling program and type, as well as the percentage of the total assay samples submitted by program.

Table 8-3: Summary of QA/QC Samples by Drilling Program and Type

Year	Operator	Total Assay Samples	QA/QC Samples				
			SRM	Blank	Duplicate	Total QAQC Samples	Percentage of Total Samples
2000	Noranda	1,890					
2001		1,126					
2004		184	7	7		14	8%
2007	Stingray	6,498	336	250	499	1,085	17%
2008		6,090	653	328	653	1,634	27%
2016		9,731		362	812	1,174	12%
Total		**25,718**	**996**	**947**	**1,964**	**3,907**	**15%**

The following sections present findings relating to each of the types of QA/QC samples.

8.3.1 Standard Reference Material Samples

Stingray commissioned METCON to prepare two SRMs for use in the 2007 and 2008 QA/QC programs. A low-grade Standard Reference Material (SRM) and high-grade SRM were prepared at METCON utilizing 23 interval reject samples provided by Stingray. Low grade and high grade SRM samples were submitted for total Cu, soluble Cu, and Mo assays at METCON, SGS Lakefield, ALS Chemex, and Hazen for round robin testing. Table 8-4 summarizes the analytical data developed on the two SRM samples submitted for assays.

Table 8-4: Summary of Assays on SRM Samples

Sample Description	Laboratory	Total Cu (%)	Soluble Cu (%)	Mo (%)
Low Grade Copper Standard	METCON Research	0.182	0.127	0.007
	SGS Lakefield Research Ltd.	0.170	0.076	0.005
	ALS Chemex	0.186	0.116	0.005
	Hazen Research Inch	0.186	0.117	0.006
	Mean	***0.181***	***0.109***	***0.006***
	Standard Deviation	***0.008***	***0.109***	***0.006***
High Grade Copper Standard	METCON Research	0.346	0.288	0.012
	SGS Lakefield Research Ltd.	0.310	0.240	0.010
	ALS Chemex	0.341	0.284	0.010
	Hazen Research Inch	0.335	0.279	0.010
	Mean	***0.333***	***0.273***	***0.011***
	Standard Deviation	***0.016***	***0.273***	***0.011***

While the certified nature of commercially prepared Certified Reference Material (CRM) standards provides an added level of confidence to the evaluation of the laboratory analytical accuracy (against a known certified value), non-commercial SRM standards are commonly used in exploration projects and can be considered a reliable evaluation of laboratory analytical accuracy provided they have been prepared properly including efforts to homogenize the sample followed by round robin testing to establish the accepted value and inherent variability of the SRM material.

Review of the two SRMs used determined that there was a reasonable variability for total Cu and soluble Cu between the upper and lower control limits (±2 standard deviation [SD]) for the low grade SRM; however, the high grade SRM shows an overall bias toward higher-than-expected values (i.e., higher than the mean) for the 2007 and 2008 sample programs. For each of the two SRMs, there were some sample outliers (both low and high); however, the majority fell within the control limits. Stingray did not use the high or low grade SRMs during the 2016 drilling program.

8.3.2 Field Blanks

The field blank samples used by Stingray were prepared from an outcrop on El Pilar property. The material used consistently had little to no Cu in the rock. Review of the field blanks indicate that there is some variability in both the total Cu and soluble Cu results. There were several samples that returned higher than expected values.

The QP reviewed several of the higher anomalous outliers and determined the largest variability seemed to be in the samples submitted to the third laboratory, CSAL at the end of the 2016 program. Review of the surrounding sample sequences from CSAL indicate that there is a high degree of variability amongst all samples. Golder recommends monitoring of this laboratory closely if Stingray chooses to use CSAL again. The samples submitted to Bureau Veritas and Skyline all produced results within the acceptable limits.

The QP considers the assay samples to be reliable, despite these anomalous outliers. While several blank samples failed as outliers, the values for total Cu and soluble Cu are well below the mean grades in the sample sequence and have not adversely affected sample results.

8.3.3 Field Duplicates and Replicates

Field duplicates measure inherent variability and analytical precision of the laboratory while replicates measure analytical variability and precision of the laboratory. No field duplicates were submitted for the pre-Stingray drilling programs. For the 2007, 2008, and 2016 drilling, a duplicate sample was collected every 10th sample, where two ¼ core samples were taken at the same time and were analyzed in sequence by the laboratory to assess the representativeness.

Review of the 1,966 field duplicate sample pairs from the Stingray drilling programs determined that there was a strong correlation between each pair, as evidenced by an R^2 value of 0.953 for total Cu.

In addition to the field duplicates samples, Stingray also submitted several samples for replicate analysis at an alternate laboratory, ALS Chemex/METCON (2007 and 2008) and Bureau Veritas/Skyline (2016). Coarse rejects obtained during the sample preparatory stage were sent to the opposite laboratory for umpire laboratory analysis. Review of the 51 (2007 and 2008) and 85 (2016) umpire duplicate pairs found a strong correlation between each pair, with both total Cu and soluble Cu returning an R^2 value of 0.98 to 0.99.

The QP reviewed the control charts produced for each SRM, field blank and field duplicate, and determined that there was an acceptable level of accuracy and precision for each for the purpose of estimating Mineral Resources.

8.4 QP'S OPINION REGARDING SAMPLE PREPARATION, SECURITY AND ANALYTICAL PROCEDURES

It is the QP's opinion that the sample preparation, security, and analytical procedures applied by Stingray were appropriate and fit for the purpose of establishing an analytical database for use in grade modeling and preparation of Mineral Resource estimates, as summarized in this TRS. The QA/QC program completed by Noranda was very limited and does add some uncertainty as to the reliability of the 2000, 2001, and 2004 sampling programs.

The QP reviewed the core and sampling techniques during a site visit in August 2021. The QP found that the sampling techniques were appropriate for collecting data for the purpose of preparing geological models and Mineral Resource estimates.

9 DATA VERIFICATION

9.1 EXPLORATION AND MINERAL RESOURCE DATA VERIFICATION

9.1.1 Exploration Data Validation

The QP was provided with the compiled drilling database, in Excel file format, which included survey information, downhole geological units, sample intervals and analytical results. This database was compiled from data provided from SCC and loaded into an MS Access Database.

SCC provided the QP with a 2 m resolution topographic surface prepared from the 2008 survey, which was reviewed for consistency against the drill hole collars. The QP found no material issues with the topographic surface, and the drill hole collars were found to match closely with the topography.

Drill hole data for the El Pilar Project comprised 316 drill holes (7 RC and 309 core drill holes) totaling 71,825 m of drilling and containing 25,540 analytical samples. Supporting documentation for the SCC and Noranda drilling data included laboratory certificates, descriptive logs, collar survey reports, and internal report documents.

Collar survey and downhole geological unit intervals, sample intervals and analytical results were imported into Strater software and a graphic downhole log was prepared for each drill hole to facilitate visual inspection of each individual drill hole as well as to allow for a review of correlations of geological units and mineralized zones between adjacent drill holes during the data validation and interpretation processes.

9.1.2 Validated Drill Hole Information

All drill hole logs were recorded by logging geologists on formatted paper sheets, then transcribed into Microsoft (MS) Excel. Data and observations entered into the logging sheets were reviewed for transcription or keying errors or omissions by senior SCC geologists. The tabular data provided by SCC was evaluated for errors or omissions as part of the data validation procedures described in the following section.

The QP conducted validation spot-checks on 1,776 randomly selected samples, by comparing the database with the original laboratory certificate. Table 9-1 summarizes the findings from the validation. The QP notes that the 2007 and 2008 METCON assay certificates included the drill hole name, sample interval and type of QA/QC samples on the certificate. This is not recommended standard industry practice and should be avoided by excluding this information when submitting to the laboratory. The QP cautions against including any of this information on future drilling campaigns as it can affect security and anonymity of the samples when submitted to the laboratory. The QP would recommend reviewing sample submission procedures for any future drilling. There were some minor data rounding errors observed in the Noranda drilling that is likely due to the conversion from ppm to weight percent and the loss of a decimal point. Some minor data entry errors were noted in the 2007 and 2008 SCC samples, and no errors were found in the 2016 samples.

Table 9-1: Drill Hole Sample Data Validation

Year	No. of Drill Holes	No. of Samples	No. of Assays	No. of Errors	Type of Error
2000	3	81	162	72	Rounding
2001	2	159	318	144	Rounding
2007	4	272	544	4	Data entry
2008	5	214	428	2	Data entry
2016	6	1,050	2,100		
Total	**20**	**1,776**	**3,552**	**222**	

Data validation was performed on the drill hole database records using available underlying data and documentation including, but not limited to, original drill hole descriptive logs, and laboratory assay certificates. Drill hole data validation checks were performed in Access using a series of in-house data checks to evaluate for common drill hole data errors including data gaps and omissions, overlapping lithology, or sample intervals, miscorrelated units, drill hole deviation errors, and other indicators of data corruption including transcription and keying errors. Database assay values for every sample were visually compared to the laboratory assay certificates to ensure the tabular assay data was free of errors or omissions. Drill hole recovery data was also reviewed as described in Section 7.2.2, as well as QA/QC results, as described in Section 8.3.

The QP also reviewed the drill hole data during the August 23, 2021, site visit. The purpose of the site visit was to review the project site, geology, current, and previous exploration methods and data collection procedures, and results and identify any concerns and provide recommendations for consideration by SCC. The site visit was completed in fulfilment of the requirement that the Mineral Resource or Mineral Reserves QP(s) perform a current site visit to the Project in support of preparation of any S-K 1300 Mineral Resource and/or Mineral Reserve statements, or TRS.

During the site visit, the QP visited several completed drill hole locations. The drill holes visited were selected by the QP while in the field; selected drill holes spanned the spatial extent of the El Pilar deposit and included drill holes from the 2000 through 2016 drilling campaigns.

9.1.3 Limitations on Data Verification

The QP was not directly involved in the exploration drilling and sampling programs that formed the basis for collecting the data used in the geological modeling and Mineral Resource estimates for the Project. The QP has had to rely upon the intensive review of the pre-2016 exploration program data, documentation and standard database validation checks to ensure the resultant geological database is representative and reliable for use in geological modeling and Mineral Resource and Reserve estimation.

The QP is not aware of any other limitations on or failure to conduct appropriate data verification.

9.1.4 QP's Statement on Adequacy of Data Validation

The Golder QP has reviewed and confirmed the data disclosed, including collar survey, down hole geological data and observations, sampling, analytical, and other test data underlying the information or opinions contained in the written disclosure presented in this TRS. The QP, by way of the data verification process described in this Section of the TRS, has used only that data, which were deemed by the QP to have been generated with proper industry standard procedures, were accurately transcribed from the original source and were suitable to be used for the purpose of preparing geological models and Mineral Resource estimates.

9.2 MINING AND MINERAL RESERVE DATA VERIFICATION

9.2.1 Geotechnical

The geotechnical data contained in the October 2008 Slope Design Recommendations report includes surface mapping and oriented core structural data and laboratory testing of the strength of conglomerate and Intrusive bedrock geotechnical units. Additional engineering analysis of the shear strength of conglomerate is provided by a November 2010 memorandum prepared by The Mines Group, Inc. These data have been incorporated into a geotechnical model and slope stability analysis performed to determine stable pit wall configurations for each geological unit. The 2008 analysis includes results for all geological units, and 2010 analysis updates results only for conglomerate. These data and analysis have been reviewed and found to provide a reasonable basis for the slope designs presented for each geological unit.

9.2.2 Hydrology

As of the fourth quarter of 2011, a biannual surface water monitoring network developed for the El Pilar Project was implemented and located on the Santa Cruz River, which includes 3 sampling sites. Asup_01 is located upstream of the El Pilar Project, Asub_02 is located in the middle zone of interaction of the body of water and the Project, and the last site Asup_03 is located downstream of the El Pilar Project as shown in Figure 9-1.

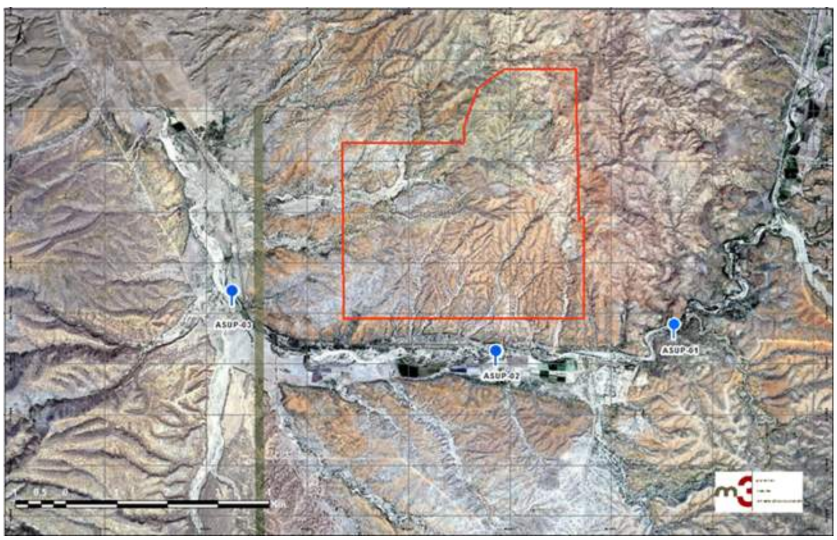


Figure 9-1: Sampling Sites Location

Sampling is carried out by M3 Mexicana S. de RL de CV and the water quality analysis is carried out by Analitica del Noroeste SA de CV, which is certified by Entidad Mexicana de Acreditación, A.C.

The results are compared with NOM-001-SEMARNAT-1996 that establishes the maximum permissible limits of pollutants from wastewater discharged into national waters and assets. The findings show that there is a high concentration of fecal coliforms within the Santa Cruz River inflow that is related to livestock activities and unauthorized discharges of wastewater (sewage) in the riverbed that, although they develop on the surface, they are more vulnerable and susceptible to be receptors of all these factors that trigger a synergistic and cumulative concentration in surface waters where the values recorded during the years sampled exceed 500 MPN/100mL.

For the other parameters considered for surface water quality characterization, the concentration levels are within the maximum permissible limits determined by the regulations, taking into account that these concentrations are represented naturally and their variations are external consequences to those derived from the development process of the El Pilar Project.

9.2.3 Hydrogeology

SCC has concession number: 02SON151018 dated July 21, 2020 to use national groundwater for a volume of 1,166,336 m^3 per year of water and expires on November 23, 2032. Concession number Nº 02SON150567/07FMGC11 was also issued for a volume of 2,333,664.00 m^3 per year and which is currently in the process of being renewed. These concessions guarantee the availability of water for the operation of the El Pilar Project.

As of the fourth quarter of 2011, a quarterly groundwater monitoring network developed for the El Pilar Project was implemented, which includes a total of 9 wells distributed inside and outside the Project as shown in Figure 9-2. Four monitoring wells are present within the limits of the El Pilar Project; Wells ASUB-01, ASUB-06, ASUB-07 and ASUB-09. The first well is represented by a pre-existing well (Noria) that was used by the community, the following three were enabled by the SCC, to closely monitor the behavior and quality of the resource before and during the useful life of the mine project. Wells ASUB-02, ASUB-03, ASUB-04, ASUB-05 and ASUB-08 are outside the limits of the El Pilar Project, but they are nonetheless equally important given that these wells are located in the vicinity of the Santa Cruz River bed and represent a very valuable indicator to determine abrupt changes in water quality. It should be noted that the monitoring well ASUB-05 supplies water to the San Lázaro ejido, a population that is adjacent to the south of the El Pilar Project.

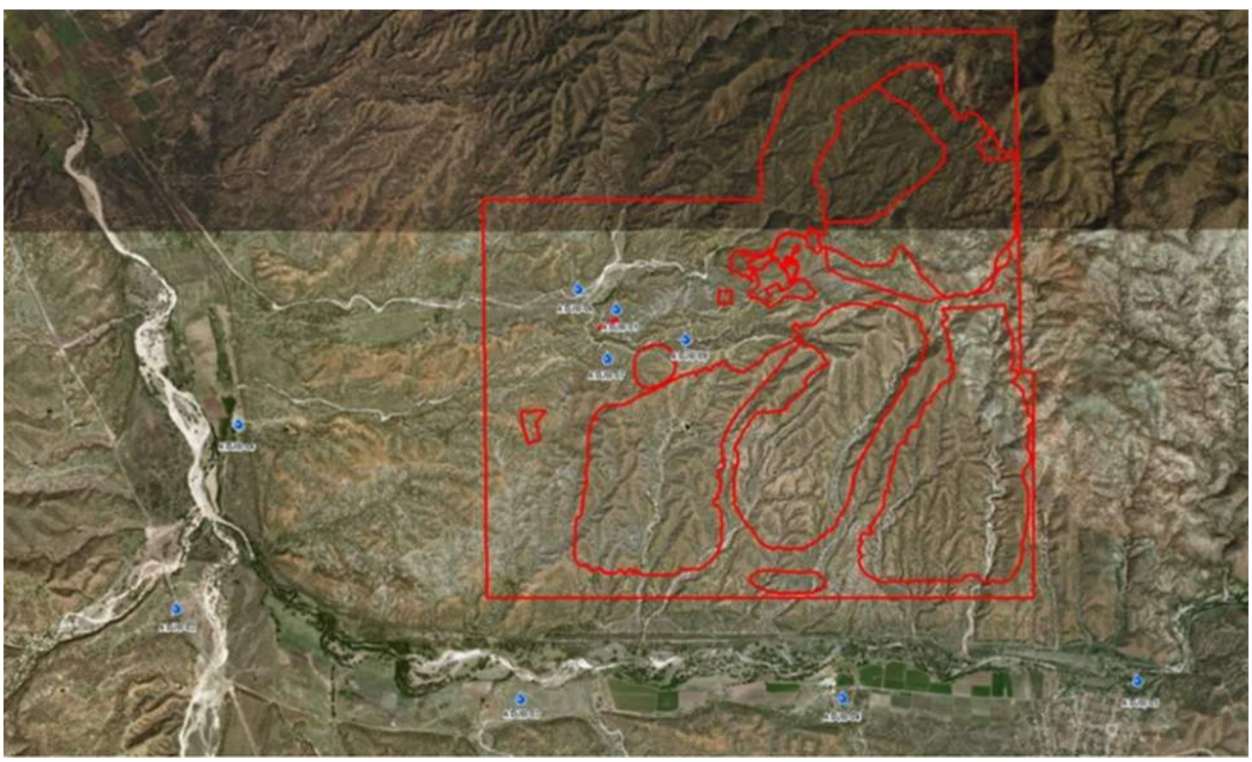


Figure 9-2: Monitoring Wells Locations

Sampling is carried out by M3 Mexicana S. de RL de CV and the water quality analysis is carried out by Analitica del Noroeste SA de CV, which is certified by Entidad Mexicana de Acreditación, A.C.

The database collected during this time was compiled to determine the level of compliance according to the environmental regulations applied to water resources, using as a reference the Official Mexican Standard NOM-127-

SSA1-1994, which determines the maximum permissible limits of water quality for human use and consumption, as well as treatments to purify water.

By comparing the processed results of the sampled wells with the maximum permissible values determined by the aforementioned standard, certain atypical values can be detected that are based on individual aspects and conditions or natural phenomena and/or effects derived from anthropogenic activities.

9.2.4 Metallurgy

The QP was not involved in the collection of samples for testing or the testwork completed on the samples. The QP has reviewed the sample preparation, analysis and security for collection of the metallurgical samples discussed in the 2011 Technical report and considers it reliable. Data verification work described in the 2011 Technical Report determined that metallurgical samples used for testing were representative of the material to be extracted. As described in the 2011 Technical Report there was a disparity between the composite average drillhole assays and the METCON head assays for the metallurgical samples used for testing. An independent metallurgical consultant supervised a QA/QC program for testwork completed in 2010/2011.

The QP is not aware of any other limitations on or failure to conduct appropriate data verification for the metallurgical testing.

9.2.5 Mining Methods

Mining of the El Pilar copper deposit will be accomplished by conventional open pit methods, with blasting of ore and waste along with shovel and truck (shovel/truck) loading and hauling. The proximity of the mineralized ore to the surface results in the use of surface mining methods to extract the material. The shape of the mineralized zone further defined the surface mining design as an open pit mine using excavators and trucks as the primary mining equipment. Waste material will be routed to one of two waste dumps and ore will be routed to the leach pad. In the development phase, drainage and water control will be established, and then the required infrastructure consisting of power, pipelines, and roadways is established.

Current plans are to engage an independent mining contractor for the preproduction period (Year 0) and Years 1 and 2 of ore production. Starting in Year 3 and extending throughout the LOM, owner mining is planned. The option of whether to use contract mining in the first two and one-half years, or to consider other options remains under review.

9.2.6 Cut-Off Grade and Modifying Factors

To comply with Regulation S-K Subpart 1300, a cut-off grade that defines the minimum grade of material that must be achieved to economically process material must be defined. Cut-off grades are defined by geometallurgical, processing, and economic criteria. For material to be sent to the ROM Leach Pad at El Pilar, it must be within mineralized Zones Qwu (201), Qwt (202), or Bx (205) and have sufficient recovery such that the Cu recovered from leaching generates enough revenue (at an assumed selling price of $3.30/lb Cu) to achieve a breakeven with the costs of mining and processing the ore and selling the resultant Cu Cathode generated from the SX-EW Plant. Additional details on the determination of cut-off grades can be found in Section 12.2.5.

Ore modifying factors have been applied to the geologic model to simulate the effects of extracting the Mineral Resource and help establish the economic viability of Mineral Reserves. Dilution in mining can be defined as the addition of waste material to the ore during the mining process and is due to a lack of selectivity, or in some cases, due to inadequate operational configuration. The process considers the neighborhood relationship between an ore block with the adjacent blocks, weighting the grades by a predetermined distance, and by the density of the blocks.

A dilution of 3% by mass was included in the Mineral Reserve estimate, by assuming dilution is the addition of waste to the ROM material at a grade of 0% copper. The mining recovery was estimated to be 98% (2% mining loss).

9.2.7 Pit Optimization

Standard pit optimization methodology using Whittle 4X software was used to determine the extent of economically mineable reserves at El Pilar. A nested pit shell analysis was performed on the geologic model. A 2% mining loss and 3% dilution adjustments were included in the pit optimization analysis. The NPV was estimated using an 8% discount rate. Based on this nested pit shell analysis, Pit 30, with a Revenue Factor of 0.88, was selected as the basis of the ultimate pit design.

Golder concludes that the ultimate pit shell and waste/ore quantities within the selected pit shell are reasonable given the pit optimization inputs and that this ultimate pit shell provides a positive economic value.

9.2.8 Mine Design

The ultimate pit shell selected from the pit optimization exercise was used as a guide to design the ultimate pit extents by integrating operational design characteristics, including ramp locations and grades, OSF locations, mining width and height, and other practical mining considerations, given pit geometry. The proposed mining equipment specifications, for instance operating width and tire height, were used to calculate a suitable ramp width. The ramp was designed to accommodate two-way traffic and assumed three times the width of the haul truck plus a drainage ditch and suitable berm. The mine design was split into 4 phases. Access ramps are designed with a maximum slope of 10%. Benches are designed to have either a 5 m or 8 m width, and a 10 m height with varying face angles depending on the lithology. The phase designs were checked to verify the geotechnical and operational design parameters were followed.

9.2.9 Production Schedule and Mineral Reserve Estimate

The mining strategy and production schedule employs the use of phases which have independent in-pit haul roads that specifically target the ore in that phase and connect to the as-built surface haul roads created by mining the previous phase(s). Production sequencing was carried out using the Deswik interactive scheduler which allows the user to visually plan multiple ongoing mining faces simultaneously.

Mining within the operational pit generally progresses from North to South away from the leach pad. The mine production schedule is based on providing sufficient ore to the heap leach pad, targeting an annual production of 19.8 million tonnes of ROM ore to the leach pad to produce a maximum of 70 million pounds of finished copper cathode per year over the LOM.

The total material in the production schedule was compared to the Mineral Reserve Estimate and checked in both Deswick, Vulcan, and Datamine software packages. The difference between software packages was found to be within an acceptable level of accuracy (<1% difference).

9.2.10 Manpower and Equipment

Mine major equipment requirements were estimated on a first principals basis based on the mine production schedule, the mine work schedule, and estimated equipment productivity rates. The mine is scheduled to operate three 8-hour shifts per day, 365 days per year, for 1,095 available shifts per year. Four mining crews are required, with an estimated 250 hourly employees and 40 staff. The mine equipment estimate assumes a well-managed mining operation with a well-trained labor pool, and that all equipment is new at the start of mining.

The largest haul truck planned will have a load capacity of approximately 180 tonnes. To develop the truck haulage requirements, the truck haulage profiles were measured for each material type by mining bench and mining phase per year

The truck fleet will initially consist of 11 trucks and will increase to a maximum of 25 in Year 8. The number of hydraulic shovels is maintained at 2 throughout the LOM, with one (1) wheel loader brought in to assist with production requirements, as needed.

9.2.11 Limitation of Data Verification

The Golder QP is not aware of any other limitations on nor failure to conduct appropriate data verification.

9.2.12 QP's Statement on Adequacy of Data

The Golder QP responsible for Mine Planning and Mineral Reserve estimates has verified the data used in the preparation of the mine design and resultant Mineral Reserve estimate, including geotechnical design criteria, cut-off grade calculations, mine modifying factors, production schedule, manpower and equipment estimates, and other test data underlying the information, or opinions, contained in the written disclosure presented in this TRS.

The QP has used only that data which was deemed appropriate by the QP to have been generated with proper industry standard procedures, was accurately transcribed from the original source and was suitable to be used for the purpose of preparing the mine design and Mineral Reserve estimates. Data that could not be verified to this standard was reviewed for information purposes only but was not used in the development of the mine design, or Mineral Reserve estimates, presented in this TRS.

10 MINERAL PROCESSING AND METALLURGICAL TESTING

10.1 GENERAL

The El Pilar mine is an open pit, oxide copper mine, whereby copper is recovered from the heap leach pad via the application of sulfuric acid-bearing raffinate pumped from a raffinate pond. The pregnant leach solution (PLS) is then collected and copper is recovered in the plant via the process of solvent extraction and electrowinning (SX-EW) in a two-stage process that first extracts and upgrades copper ions from low-grade leach solutions into a concentrated electrolyte, and then deposits pure copper onto cathodes using an electrolytic procedure. Extensive metallurgical work has been conducted historically to best determine how much copper will be recovered from the ore and how much acid will be consumed by the process.

10.2 HISTORIC WORK - 2004 NORANDA & 2009 STINGRAY METALLURGICAL TESTING

In 2004, Noranda, a subsidiary of Falconbridge Ltd., commissioned a preliminary leaching test program at Falconbridge's Lomas Bayas mine in Chile. That work was reported by WOODS in 2007.

For the purposes of the 2009 NI 43-101 Technical Report and Feasibility Study, Stingray developed an additional program of drilling and metallurgical testwork at El Pilar. METCON conducted all the leaching testwork, from sample preparation, bottle roll tests, mini-columns, and open-cycle columns to locked-cycle columns. The program is fully documented in Stingray's 2009 NI43-101 Technical Report entitled; "El Pilar Project Executive Summary - Sonora, Mexico", issued by M3.

The metallurgical testing program was on drill core composite samples that simulate the leach material for a simplified Year 1-9 mine schedule taking into account geology, material location and copper grade. METCON received 3,925 interval samples from 75 drill holes. A sample from each interval was split and submitted for total copper and acid soluble copper analysis. ALS Chemex received 9,237 interval samples from 141 drill holes and also conducted total copper and acid soluble copper assays on samples from each interval sample. Drill hole intercepts were selected to be representative of material in the mine schedule for Year 1, Year 2, Year 3, Years 4 to 6, and Years 7 to 9. The three single year composites and the two three-year composites resulted in the five separate drill core composite samples. Figure 10-1 shows the drill holes locations used for the metallurgical composites and illustrates that the five composites are well distributed throughout the deposit area.

The primary objective of the 2009 test program was to generate copper extraction and sulfuric acid consumption data for the composite samples. The leach tests were conducted utilizing bottle roll and column leach techniques in open cycle and locked cycle tests under variable irrigation rates and material crush sizes.

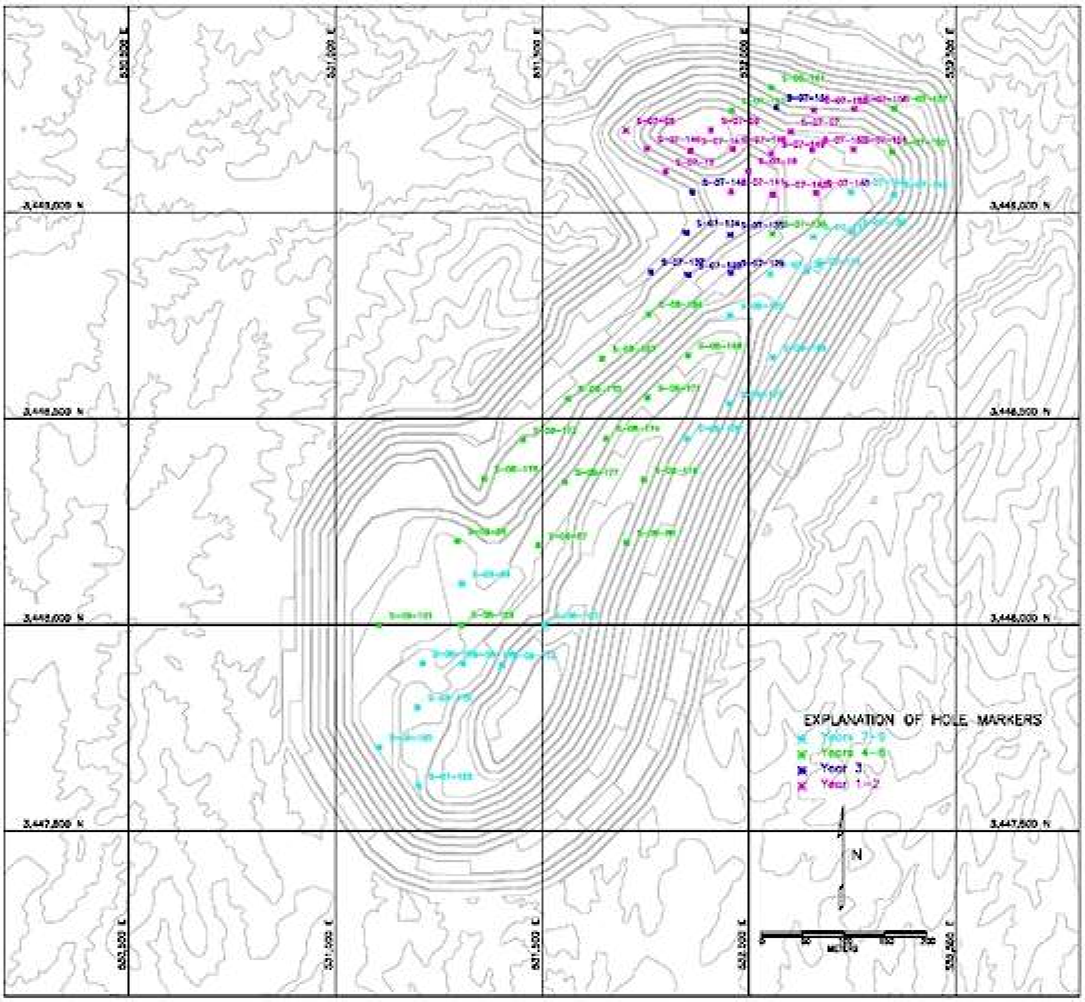


Figure 10-1: Distribution of Stingray Five Composite Metallurgical Samples

The result of the bottle roll tests that were done to assess maximum acid consumption are presented in Table 10-1.

Table 10-1: Yearly Composite Bottle Roll Results

EL Pilar Bottle Roll Testing Summary of Results						
Sample Id	**Extraction (%)**		**Gangue Acid Consumption**		**Total Acid Consumption**	
	Cu	**Fe**	**kg/t**	**kg/kg Cu**	**kg/t**	**kg/kg Cu**
Year 1	76.44	7.37	23.97	7.48	28.92	9.02
Year 2	75.98	6.49	24.36	7.72	29.23	9.26
Year 3	70.97	25.93	18.98	7.15	23.07	8.69
Year 4, 5 & 6	66.89	3.47	22.94	11.06	26.14	12.60
Year 7, 8 & 9	58.76	-2.19	25.46	15.04	28.07	16.59

A total of 20 columns were run in locked cycle with SX for a 120 day leach period, two for each yearly composite at a 37.5 mm and 19.0 mm crush size and two for each composite at a 7.8 and 6.11 L/h solution application rate. A detailed summary of the results of the locked cycle test for the five yearly composites is presented in Table 10-2.

Table 10-2: METCON Locked Cycle Test Results – Five Yearly Composites

Stingray - Five Yearly Composite METCON Locked Cycle Metallurgical Results																	
				Normalized Extraction		Gangue Acid Consumption			Sequential Copper Analysis (% Cu)						Calculated Copper Extraction (%)		
									Head Sample			Leach Residue Sample					
Sample ID	Crush Size P80 (mm)	Irrigation Flow Rate (lph/m2)	Leach Cycle (Days)	RCu (%)	RFe (%)	(kg/t)	(kg/ kg Cu)	Comp. Soluble Test ASCu	ASCu	ResCu (incl. CNsCu)	Calc TCu	ASCu	ResCu (incl. CNsCu)	Calc TCu	R-ASCu (%)	R-ResCu (incl. CNsCu)	Calc R-TCu (%)
Year 1	**37.5**	**6.11**	**128**	**66.0%**	**4.8%**	**19.32**	**7.12**	0.321	**0.295**	**0.134**	**0.429**	**0.050**	**0.080**	**0.130**	**83.1%**	**40.3%**	**69.7%**
	37.5	7.8	128	70.2%	4.5%	18.92	6.02	0.321	0.295	0.134	0.429	0.051	0.079	0.13	82.7%	41.0%	69.7%
	19.0	6.11	128	73.9%	2.3%	15.39	4.97	0.321	0.306	0.129	0.435	0.042	0.066	0.108	86.3%	48.8%	75.2%
	19.0	7.8	128	82.4%	3.0%	17.54	5.12	0.321	0.306	0.129	0.435	0.024	0.055	0.079	92.2%	57.4%	81.8%
Year 2	**37.5**	**6.11**	**128**	**70.1%**	**1.9%**	**17.83**	**5.92**	0.263	**0.267**	**0.125**	**0.392**	**0.045**	**0.080**	**0.125**	**83.1%**	**36.0%**	**68.1%**
	37.5	7.8	128	70.9%	0.0%	19.88	6.69	0.263	0.267	0.125	0.392	0.033	0.087	0.12	87.6%	30.4%	69.4%
	19.0	6.11	128	74.0%	1.3%	16.35	5.26	0.263	0.306	0.129	0.435	0.031	0.084	0.115	89.9%	34.9%	73.6%
	19.0	7.8	128	75.2%	2.4%	18.63	5.89	0.263	0.306	0.129	0.435	0.027	0.078	0.105	91.2%	39.5%	75.9%
Year 3	**37.5**	**6.11**	**128**	**63.7%**	**0.6%**	**17.33**	**7.58**	0.218	**0.237**	**0.137**	**0.374**	**0.033**	**0.089**	**0.122**	**86.1%**	**35.0%**	**67.4%**
	37.5	7.8	128	64.1%	2.4%	19.33	8.24	0.218	0.237	0.137	0.374	0.037	0.088	0.125	84.4%	35.8%	66.6%
	19.0	6.11	128	70.6%	2.9%	14.31	5.65	0.218	0.236	0.11	0.346	0.029	0.067	0.096	87.7%	39.1%	72.3%
	19.0	7.8	128	70.2%	0.2%	15.27	6.24	0.218	0.236	0.11	0.346	0.033	0.070	0.103	86.0%	36.4%	70.2%
Year 4-6	**37.5**	**6.11**	**128**	**59.6%**	**3.0%**	**15.04**	**7.97**	0.175	**0.188**	**0.108**	**0.296**	**0.030**	**0.095**	**0.125**	**84.0%**	**12.0%**	**57.8%**
	37.5	7.8	128	60.1%	0.6%	13.68	6.96	0.175	0.188	0.108	0.296	0.031	0.099	0.13	83.5%	8.3%	56.1%
	19.0	6.11	128	57.6%	2.1%	12.92	7.54	0.175	0.184	0.113	0.297	0.032	0.097	0.129	82.6%	14.2%	56.6%
	19.0	7.8	128	60.7%	2.0%	16.90	9.47	0.175	0.184	0.113	0.297	0.032	0.087	0.119	82.6%	23.0%	59.9%
Year 7-9	**37.5**	**6.11**	**128**	**55.9%**	**1.2%**	**14.37**	**8.82**	0.158	**0.16**	**0.135**	**0.295**	**0.025**	**0.105**	**0.13**	**84.4%**	**22.2%**	**55.9%**
	37.5	7.8	128	61.1%	2.5%	16.28	9.46	0.158	0.16	0.135	0.295	0.027	0.108	0.135	83.1%	20.0%	54.2%
	19.0	6.11	128	55.0%	-1.0%	14.10	8.58	0.158	0.154	0.133	0.287	0.021	0.116	0.137	86.4%	12.8%	52.3%
	19.0	7.8	128	61.3%	3.2%	13.48	6.87	0.158	0.154	0.133	0.287	0.021	0.112	0.133	86.4%	15.8%	53.7%

10.3 EVALUATION OF 2009 STINGRAY METALLURGICAL TESTING RESULTS

In preparation for the 2011 updated El Pilar Feasibility Study presented in this report, Mercator Minerals Ltd. (ML) conducted a detailed review all technical studies with particular attention to project metallurgical conclusions. An important discovery of the review, and something not recognized in the 2009 study, is the positive relationship between ASCu (Acid Soluble Copper) values and percent copper recovery in the column leach tests.

Specifically, the METCON column leach tests recovered an average of 84.3% ASCu with a small variance, but only 27.1% of ResCu (Residual Copper, i.e. copper that is not acid soluble) with a significant variance of 8% to 41% recovery (Figure 10-2). The high variability in ResCu recovery appears to be unrelated to TCu grade, due to the low variability of TCu (Total Copper) for the yearly composites. As a result, the high variability in ResCu recovery indicates that acid solubility must be taken into account in the evaluation of copper recovery at El Pilar.

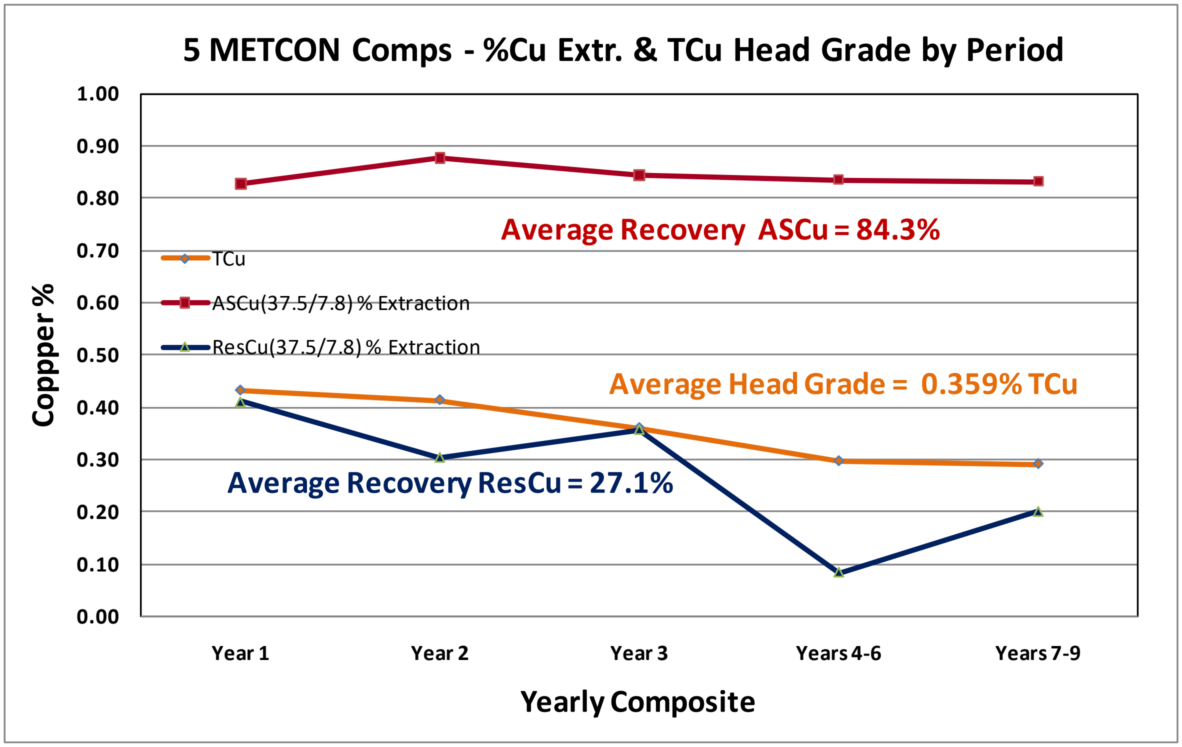


Figure 10-2: Copper Solubility & Recovery – Stingray Yearly Composites

Also, the review noted that the 2009 El Pilar reserve was based in part on a cut-off grade of 0.15% TCu, whereas the five yearly composites ranged from 0.428 to 0.289% TCu, well above the TCu grade range of many ore blocks.

10.4 ADDITIONAL METALLURGICAL TESTING DONE IN 2010 AND 2011 BY MERCATOR

In 2010 and 2011, ML conducted additional metallurgical testing on mineralized material from El Pilar for the following reasons:

1. To evaluate the metallurgical viability of using ROM leaching, as compared to the higher cost crushing program proposed by Stingray in 2009;
2. To develop additional metallurgical copper recovery and acid consumption information from mineralization with a wide range of copper acid solubility assays;
3. To investigate the metallurgical recovery of copper and consumption of acid from and by lower grade ore near or at the lower end of Hester's 2009 0.15 TCu% copper cut-off grade; and;
4. To assess metallurgically and economically the possibility of attaining better overall copper recoveries using a longer (than 120 day), more optimized leach period.

The primary objective of points #2 and #3 above was to attempt to develop a grade recovery relationship at El Pilar. Such a relationship would then allow for the development of one or more recovery algorithms, such that the recovery of copper from both variable grade and variable copper solubility ore could be predicted on a more accurate basis than was done previously.

The additional metallurgical testing performed in 2010 and 2011 was done on a 600 tonne bulk sample collected onsite and on 13 new drill core composites. These programs are discussed separately in the following sections.

10.5 BULK SAMPLE CRIB AND COLUMN TESTS 2010-2011

10.5.1 Bulk Sample – Objective and Sample Collection Details

Starting in Q2 2010, an approximately 600 tonne bulk sample of mineralized Qwu material was collected at El Pilar from surface outcrops. The primary purpose of collecting this sample was to evaluate the metallurgical viability of using ROM leaching at El Pilar. The tests were conducted on two cribs and two columns under locked cycle conditions, with the main objective of the locked cycle testing being to determine copper extraction and gangue acid consumption with an irrigation flow rate of 5.2 L/h/m^2 at four meter lift height in the cribs and an equivalent 7.8 L/h/m^2 for the seven meter columns.

The bulk sample was collected at El Pilar at coordinates 531,690 E and 3,449,130 N (see Figure 10-3) at a location just within the northern boundary of proposed open pit. This location is the only outcrop area of copper mineralized alluvial wash material (Qwu) on the El Pilar property.

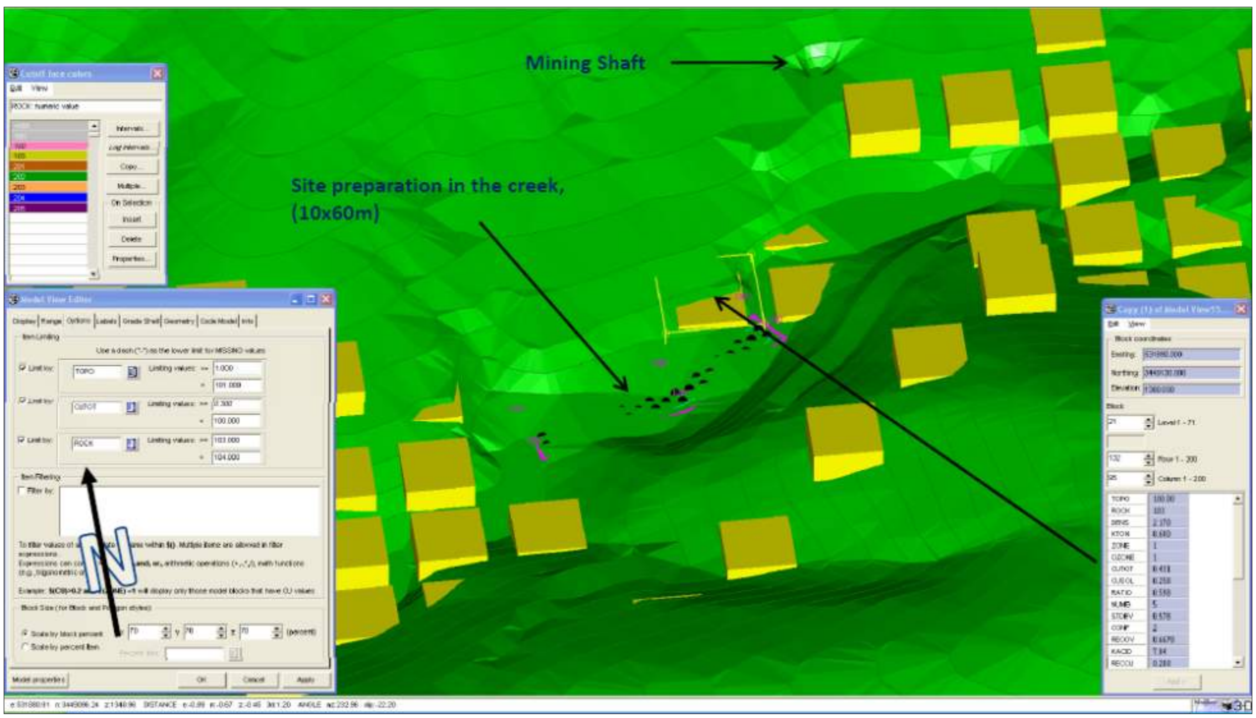


Figure 10-3: MineSight Screen Capture Showing Bulk Sample Location

Details of collecting the bulk sample are included in a separate report entitled QAQC Program – Bulk Sample (McNulty, 2011), but the information in that report is summarized in the following section. Briefly, the bulk sample was collected from an elongate approximately 10 X 60 m surface trench. The sample area was first outlined in the field, then the upper 1 meter of top soil was removed and the sample collected from a depth of from 1 to 4 m by means of a CAT 325 excavator and directly loaded into 10-yard dump trucks. The sample was then hauled to a yard in Naco, Sonora for its final transport to the Mineral Park mine (MP) in Kingman, AZ.

10.5.2 Bulk Sample – Screening and Results

After the bulk sample was received onsite at Mineral Park, it was weighed, offloaded and dry screened at a screening plant available onsite. The results of the screening program are tabulated in Table 10-3.

Table 10-3: Results of Bulk Sample Dry Screening

SCREEN ANALYSIS			Sample: Size:	Bulk Sample Screened		
NOMINAL OPENINGS			Sample Weights (kg)	Weight Distribution (%)	Cumulative (%)	
mm	Inch	Tyler Mesh			Retained	Passing
152.4	6		11,249	1.95	1.95	98.05
100.00	4		13,699	2.38	4.33	95.67
31.50	1 1/4		100,979	17.53	21.86	78.14
19.00	3/4		81,565	14.16	36.02	63.98
M I N U S		3/4	368,581	63.98	100.00	
	TOTALS		**576,072**	**100.00**		

Drums containing fraction samples were identified and sealed, and the samples that were assigned to METCON (Table 10-4) for head assay and column testing were shipped in compliance with a sample collection QA/QC program. Also, duplicate samples were sealed and saved for future use.

Table 10-4: Bulk Sample Fractions to METCON & Duplicates

Fraction	Metcon Sample (kg)	Duplicate (kg)
+6"	69.5	54.5
-6" to +4"	90.5	97
-4" to +1 1/2"	307.5	293
-1 1/2" to +3/4"	281.5	303.5
-3/4"	1537.5	1541
TOTAL (kg)	2287	2289

After the bulk sample was screened and weighed and samples from each fraction had been collected, the bulk sample was reconstituted at Mineral Park by gradually forming a pile with portions from each fraction. Once one pile was formed with all the material from all fractions, homogenization was done by coning and quartering up to five times to achieve a final homogeneous pile in compliance with the program QA/QC implementation schedule.

In order to independently determine the head grade of the bulk sample and to have material for five additional backup columns (discussed separately), 2,287 kg of screen fraction duplicate material were shipped to METCON in Tucson. This sample was recombined by METCON and then dry screened into 22 screen fractions. Each of the individual screen fractions was then assayed by METCON for TCu and ASCu and a bulk sample head grade was computed from the fraction assay results (see Table 10-5). The head grade of the bulk sample average 0.537% TCu and had an acid soluble/total copper ratio of 74.6%. Tail screen size analysis (Table 10-6) of the same head sample after leaching shows that very little degradation of the ore can be expected during the leaching process. This is highlighted by the comparison graph of the head vs. tail results in Figure 10-4.

Table 10-5: Bulk Sample (Qwu) Head Size Analysis & Assays

SCREEN ANALYSIS			Sample: Head Bulk Composite Size: As Received at METCON Research				Analytical Results			
NOMINAL OPENINGS			Sample Weights (kg)	Weight Distribution (%)	Cumulative (%)		Copper Assays			
mm	Inch	Tyler Mesh			Retained	Passing	TCu	ASCu	ResCu	Ratio
	8		0.00			100.00	0.177	0.141	0.036	0.797
152.4	6		80.00	3.47	3.47	96.53	0.452	0.371	0.081	0.821
100.00	4		93.00	4.03	7.50	92.50	0.297	0.211	0.086	0.710
75.00	3		12.62	0.55	8.05	91.95	0.287	0.163	0.124	0.568
50.00	2		61.90	2.69	10.74	89.26	0.363	0.238	0.125	0.656
31.50	1 1/4		231.50	10.04	20.78	79.22	0.346	0.224	0.122	0.647
25.00	0.984	1	53.88	2.34	23.12	76.88	0.391	0.226	0.165	0.578
19.00	0.748	3/4	240.50	10.43	33.55	66.45	0.438	0.297	0.141	0.678
12.50	0.492	1/2	119.00	5.16	38.71	61.29	0.466	0.317	0.149	0.680
9.50	0.374	3/8	136.00	5.90	44.61	55.39	0.465	0.331	0.134	0.712
6.30	0.248	1/4	211.00	9.15	53.76	46.24	0.551	0.388	0.163	0.704
1.70	0.067	10	495.00	21.47	75.23	24.77	0.583	0.453	0.130	0.777
1.00	0.039	16	92.86	4.03	79.26	20.74	0.637	0.530	0.107	0.832
0.84	0.033	20	26.11	1.13	80.39	19.61	0.701	0.560	0.141	0.799
0.60	0.023	28	50.60	2.19	82.59	17.41	0.758	0.630	0.128	0.831
0.42	0.017	35	43.10	1.87	84.46	15.54	0.881	0.700	0.181	0.795
0.30	0.012	48	38.71	1.68	86.14	13.86	0.915	0.780	0.135	0.852
0.21	0.008	65	31.84	1.38	87.52	12.48	0.925	0.770	0.155	0.832
0.15	0.006	100	28.48	1.24	88.75	11.25	0.821	0.680	0.141	0.828
0.11	0.004	150	34.51	1.50	90.25	9.75	0.724	0.600	0.124	0.829
0.07	0.003	200	29.03	1.26	91.51	8.49	1.020	0.690	0.330	0.676
MINUS		-200	195.75	8.49	100.00					
	TOTALS		**2305.40**	**100.00**						
CALCULATED ASSAY							**0.537**	**0.382**	**0.155**	**0.712**

Table 10-6: Bulk Sample (Qwu) Tail Size Analysis & Assays

SCREEN ANALYSIS			Sample: Tail Bulk Composite Size: As Received at METCON Research				Analytical Results			
NOMINAL OPENINGS			Sample Weights (kg)	Weight Distribution (%)	Cumulative (%)		Copper Assays			
mm	Inch	Tyler Mesh			Retained	Passing	TCu	ASCu	ResCu	Ratio
	8		0.00			100.00	0.115	0.086	0.029	0.748
152.4	6		32.06	1.04	1.04	98.96	0.205	0.135	0.070	0.659
100.00	4		29.34	0.95	2.00	98.00	0.190	0.106	0.084	0.558
75.00	3		54.50	1.77	3.77	96.23	0.183	0.111	0.072	0.607
50.00	2		103.04	3.35	7.12	92.88	0.221	0.142	0.079	0.643
31.50	1 1/4		236.64	7.69	14.81	85.19	0.238	0.144	0.094	0.605
25.00	0.984	1	158.98	5.17	19.98	80.02	0.220	0.126	0.094	0.573
19.00	0.748	3/4	195.92	6.37	26.34	73.66	0.205	0.111	0.094	0.541
12.50	0.492	1/2	311.50	10.13	36.47	63.53	0.198	0.107	0.091	0.540
9.50	0.374	3/8	204.00	6.63	43.10	56.90	0.225	0.126	0.099	0.560
6.30	0.248	1/4	279.00	9.07	52.17	47.83	0.258	0.139	0.119	0.539
1.70	0.067	10	703.50	22.87	75.04	24.96	0.201	0.108	0.093	0.537
1.00	0.039	16	108.23	3.52	78.55	21.45	0.217	0.106	0.111	0.488
0.84	0.033	20	34.90	1.13	79.69	20.31	0.240	0.109	0.131	0.454
0.60	0.023	28	67.14	2.18	81.87	18.13	0.257	0.115	0.142	0.447
0.42	0.017	35	61.90	2.01	83.88	16.12	0.279	0.122	0.157	0.437
0.30	0.012	48	53.40	1.74	85.62	14.38	0.298	0.132	0.166	0.443
0.21	0.008	65	42.85	1.39	87.01	12.99	0.294	0.118	0.176	0.401
0.15	0.006	100	41.04	1.33	88.35	11.65	0.297	0.126	0.171	0.424
0.11	0.004	150	45.76	1.49	89.83	10.17	0.278	0.108	0.170	0.388
0.07	0.003	200	38.63	1.26	91.09	8.91	0.486	0.104	0.382	0.214
MINUS		-200	274.15	8.49	100.00					
	TOTALS		3076.48	100.00						
CALCULATED ASSAY							0.273	0.134	0.139	0.491

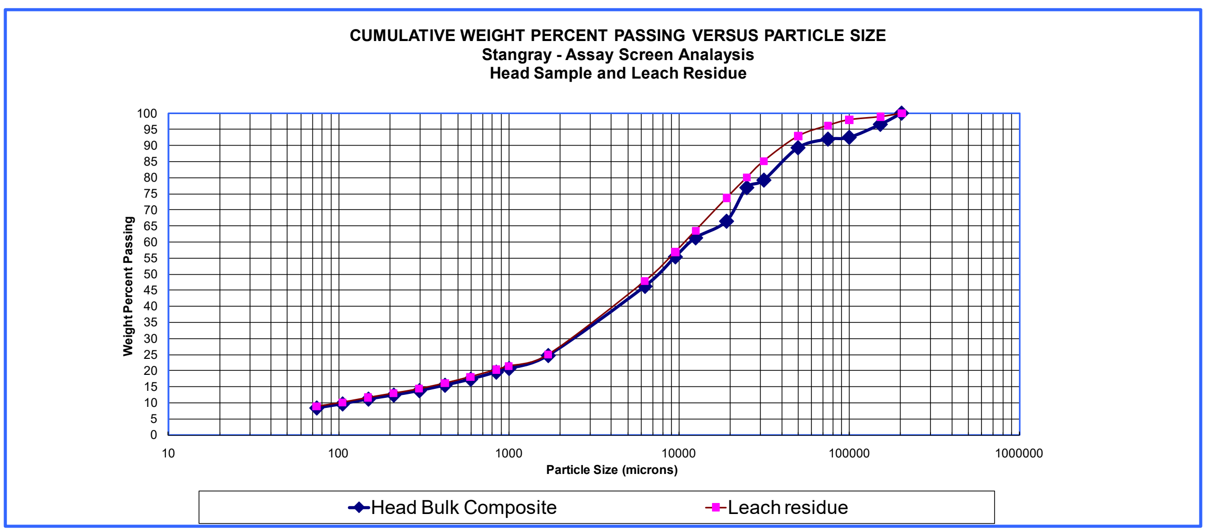


Figure 10-4: Chart Comparison Bulk Sample Screen Fractions Head vs. Tails

10.5.3 Bulk Sample – Crib & Column Loading

After the sample was recombined, two ~250 tonne concrete bins or cribs at Mineral Park were loaded with the bulk sample using dump trucks. The cribs measure 7.73 m in length by 4.5 m in width by 4.45 m deep and were loaded to a height of 4.0 m. The cribs were first lined with 80 mil plastic liners and the return piping installed. Representative loading of the dump trucks was accomplished by collecting front-end loader bucket loads from around the pile until each truck was fully loaded. Then the trucks were weighed and unloaded into the designated crib, first with the aid of an excavator to form a bed in the bottom of each crib and then by direct end dumping until each crib was filled to the desired level with the bulk sample material. Table 10-7 is a tabulation of bulk sample truck loads and weights for each crib.

Table 10-7: Crib Truck Loads & Weights

CRIB C-01 LOADED ORE SAMPLE				CRIB C-02 LOADED ORE SAMPLE			
Gross (lb)	Tare (lb)	Ore Weight (lb)	Ore weight (kg)	Gross (lb)	Tare (lb)	Ore Weight (lb)	Ore weight (kg)
50,440	22,660	27,780	12,627	50,580	22,360	28,220	12,827
50,480	22,420	28,060	12,755	42,920	22,660	20,260	9,209
53,860	22,520	31,340	14,245	49,660	22,720	26,940	12,245
50,920	22,480	28,440	12,927	49,340	23,140	26,200	11,909
51,080	22,600	28,480	12,945	51,680	22,700	28,980	13,173
51,620	22,520	29,100	13,227	52,020	22,700	29,320	13,327
52,580	22,480	30,100	13,682	52,040	22,760	29,280	13,309
48,960	22,440	26,520	12,055	49,980	22,900	27,080	12,309
49,760	22,500	27,260	12,391	51,520	22,660	28,860	13,118
45,420	22,340	23,080	10,491	50,580	22,620	27,960	12,709
49,640	22,060	27,580	12,536	54,640	21,700	32,940	14,973
51,540	22,040	29,500	13,409	51,020	21,860	29,160	13,255
47,680	21,900	25,780	11,718	50,020	21,800	28,220	12,827
50,500	21,900	28,600	13,000	51,960	21,940	30,020	13,645
50,380	21,960	28,420	12,918	49,140	21,980	27,160	12,345
52,780	22,040	30,740	13,973	53,140	22,020	31,120	14,145
49,220	22,120	27,100	12,318	51,840	22,080	29,760	13,527
49,320	21,980	27,340	12,427	50,980	22,100	28,880	13,127
49,080	21,980	27,100	12,318	49,240	22,280	26,960	12,255
49,880	21,880	28,000	12,727	50,700	22,000	28,700	13,045
1,005,140	**444,820**	**560,320**	**254,691**	**1,013,000**	**446,980**	**566,020**	**257,282**
		+3/4 filter	279			+3/4 filter	281
		+1 1/4 filter	665			+1 1/4 filter	635
		TOTAL Kg	**255,635**			**TOTAL Kg**	**258,197**

In addition to the cribs, two columns measuring ~6 meter high (20 ft) by 27” I.D. were manually loaded with buckets of the bulk sample material, first by weighing an approximate amount of material to fill each column (~3 tonnes) and then by loading each column independently by filling, weighing and dumping individual buckets after net weights were recorded.

10.5.4 Bulk Sample – Moisture Content

Material moisture was determined by using a 12.29 kg split from a sample that was collected when the sample was being mixed. The sample was dried in an oven overnight at a temperature of 105 °C and weighed. The result was a dry weight of 11.82 kg and a moisture content of 3.82%.

10.5.5 Bulk Sample – Crib & Column Acid Precure

Both the cribs and the columns were precured with acid. For the cribs, two batches of curing solution were prepared by adding 1,070 kg of sulfuric acid to 5,600 liters of liquid for each batch. The batches in two tanks were mixed by recirculation while the acid was added then and thereafter for two hours to ensure solution homogeneity. Assays for free acid were done using Mineral Park assay procedures. The two bins were left to rest for seven days before leaching (see Table 10-8 for crib acid cure details).

Table 10-8: Final Acid Curing Data for MP Cribs

Mineral Park Crib Acid Cure Details			
	kg Acid	Assayed Free Acid	kg/ton acid
Crib 1	1001.24	100	4.08
Crib 2	1103.92	130	4.45

For the columns, a 100 g/L free acid concentration solution was prepared and then pumped to each column at an average rate of 55 L/day. A total of 120 L of solution were pumped onto each column during the 48 hour precure period. Columns were left to rest for 7 days before leaching (see Table 10-9 for column acid cure details).

Table 10-9: Final Acid Curing Data for MP Columns

Mineral Park Column Acid Cure Details			
	kg Acid	Assayed Free Acid (g/L)	kg/ton acid
Column 1	11.095	96.2	3.59
Column 2	11.095	94.3	3.52

10.5.6 Bulk Sample – Leach Parameters & Assay QAQC

After the precure process was completed, initially the feed solution to the cribs was prepared at 14 g/L of free acid, which was pumped at an average rate of 3.24 L/min. This initial PLS began to flow out of the bins after 3 days. PLS volume was recorded using a flow meter. After three days the cribs were irrigated initially for 120 days at a rate of 5.3 L/h/m^2 of raffinate solution maintained at an acid concentration of 10 g/L free acid.

For the columns, feed solution was initially pumped using 10 g/L of free acid at a rate of 2.3 L/h/m^2 using a mixture of water and concentrated sulfuric acid. To adjust the acid concentration on the raffinate tank two assays were conducted each time. The raffinate tank was sampled as is, without additional acid added. Tank volume was estimated from the markings on the tank. Acid to be added was then calculated using the data just mentioned. An Avery Weight-Tronix W1-125 analytical scale was used to weight the acid added to the raffinate tank based on calculations. After three days the columns were irrigated initially for 120 days at a rate of 7.8 L/h/m^2 of raffinate solution maintained at an acid concentration of 10 g/L free acid.

Originally, one crib (C-01) and one backup column (CL-01) were intended to represent ROM material and the other crib (C-02) and backup column (CL-02) were intended to represent minus 6” crush material. However, the ROM bulk sample had only 1.95% of the weight at size greater than 6” in diameter (see Table 10-7) and thus both crib and column samples were essentially treated as ROM duplicates.

PLS and organic (ACORGA extractant and diluent) from each of the closed cycle cribs and columns was fed to a small onsite SX-EW plant at equal flow rates where they were mixed to extract the recovered copper from the PLS. Copper extraction and acid consumption, as well as other factors, were then computed daily for each of the cribs and columns.

Sampling was done on the PLS solutions for copper, total iron, free acid, pH, ORP and ferrous iron. A strict QA-QC program for sample collection and analysis was implemented and supervised independently by metallurgical consultant Terry McNulty. Samples were collected and split into three portions. One portion was assayed by ML at Mineral Park. One out of 10 samples was sent to METCON and a third sample was stored for future use if necessary. Additionally, a weekly composite using 3 mL from daily samples was assembled to be analyzed by ICP for multiple elements by METCON. A total of 68 samples were sent to METCON, 17 samples from each PLS obtained in the test: Crib 1, Crib 2, Column 1, and Column 2. Samples were also assayed for copper and iron.

Standards for both copper and iron were prepared onsite at Mineral Park using a certified 1000 mg/L solution. The standards were prepared twice during the duration of the test. The first set of standards was prepared on August 13, 2010 with the copper concentrations in ppm of 5, 10, 20 and 50. As PLS grade decreased, a second set of standards was required to measure lower concentrations of copper. The second set of standards was prepared on October 19 2010 with the copper concentrations in ppm of 1, 3 and 5.

Samples were analyzed at Mineral Park using a Perkin Elmer Analyst 200 Spectrometer. The wavelength for copper was set at 324.75 nm with a slit opening of 2.7/0.8 mm using a Copper Lumina Hollow Cathode lamp. Air-acetylene flow was 5:1 for copper. A read delay of three seconds was used sample to sample, with an integration time of four seconds. A linear calibration was used as defined from three standards. Samples were diluted according to the standards to be used so that the sample concentration was in the middle of the standard range.

A comparison of the analytical results from METCON and ML exhibits a good correlation between both laboratories for all check assays done at Mineral Park. The correlation coefficient R2 for copper is 0.9946 and the slope is 0.9966, as shown in Figure 10-5.

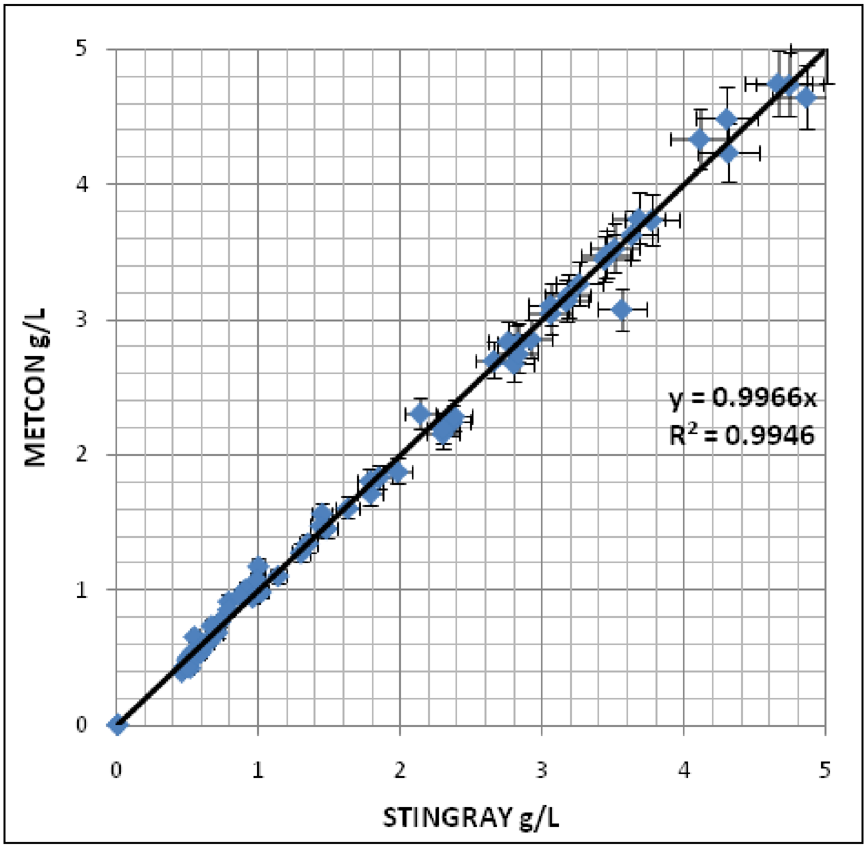


Figure 10-5: Correlation between METCON and Mercator Copper Assays

10.5.7 Bulk Sample – 14 Day Bottle Roll Leach Results

Bottle roll testing was done on the bulk sample for different size screens and for two different crush sizes from the head sample. Table 10-10 and Table 10-11 shows the results of the 72-hour bottle roll tests on the screen size fractions. The screens have a simple average acid consumption of 12.31 kg per tonne.

Table 10-10: Bulk Sample Bottle Roll Results – Part 1

SUMMARY OF METALLURGICAL RESULTS - PART 1
Ambient 72 hr. Bottle Roll Leach Tests
Bulk Sample-All Fractions- Stingray Copper
METCON Research Project No. M-760-02

Test Number	Sample/ Fraction	Total Acid Consumption		Gangue Acid Consumption		Leach	CUMULATIVE EXTRACTION(%)	
						Time	Cu	Fe
		kg/t	kg/kgCu	kg/t	kg/kg Cu	(hr)	%	%
BR-02	6 Inch	9.83	7.13	7.71	5.59	0	0.00	0.00
						2	52.25	1.01
						4	52.46	1.08
						6	54.58	1.14
						24	61.01	1.43
						48	63.82	1.80
						72	70.14	1.92
BR-03	4 Inch	11.88	4.20	7.51	2.66	0	0.00	0.00
						2	41.91	0.59
						4	45.09	0.69
						6	46.91	0.67
						24	54.83	0.87
						48	59.24	1.02
						72	62.95	1.27
BR-04	3 Inch	9.86	5.32	7.00	3.78	0	0.00	0.00
						2	34.63	0.53
						4	40.69	0.64
						6	43.76	0.75
						24	50.76	1.13
						48	55.46	1.46
						72	59.67	1.63
BR-05	2 Inch	10.23	6.83	7.92	5.29	0	0.00	0.00
						2	31.02	0.67
						4	34.48	0.77
						6	36.82	0.96
						24	43.98	1.22
						48	48.05	1.63
						72	51.34	1.89
BR-06	11/4 Inch	12.39	6.17	9.29	4.62	0	0.00	0.00
						2	35.73	0.63
						4	40.26	0.74
						6	41.99	0.78
						24	48.61	1.04
						48	50.70	1.59
						72	54.81	1.52
BR-07	1 Inch	11.20	5.82	8.23	4.28	0	0.00	0.00
						2	36.81	0.71
						4	38.88	0.72
						6	40.55	0.83
						24	48.11	1.10
						48	51.94	1.42
						72	54.32	1.59

Table 10-11: Bulk Sample Bottle Roll Results – Part 2

SUMMARY OF METALLURGICAL RESULTS - PART 2
Ambient 72 hr. Bottle Roll Leach Tests
Bulk Sample-All Fractions- Stingray Copper
METCON Research Project No. M-760-02

Test Number	Sample/ Fraction	Total Acid		Gangue Acid		Leach Time	CUMULATIVE	
		kg/t	kg/kgCu	kg/t	kg/kg Cu	(hr)	Cu %	Fe %
BR-08	3/4 Inch	13.13	5.20	9.23	4.22	0	0.00	0.00
						2	38.80	0.58
						4	43.93	0.61
						6	47.90	0.72
						24	54.39	0.91
						48	58.31	1.07
						72	57.72	1.39
BR-09	1/2 Inch	11.82	5.76	8.65	4.22	0	0.00	0.00
						2	37.92	0.59
						4	42.50	0.62
						6	47.03	0.79
						24	52.99	0.98
						48	55.76	1.16
						72	54.14	1.46
BR-10	3/8 Inch	13.37	5.22	9.41	3.67	0	0.00	0.00
						2	29.86	0.39
						4	45.82	0.60
						6	51.13	0.70
						24	56.45	0.89
						48	57.69	0.86
						72	56.43	1.09
BR-11	1/4 inch	12.84	5.14	8.98	3.59	0	0.00	0.00
						2	38.40	0.39
						4	43.38	0.48
						6	47.31	0.57
						24	54.62	0.90
						48	58.31	0.79
						72	56.10	0.81
BR-12	10 Mesh	13.56	4.39	8.80	2.85	0	0.00	0.00
						2	33.25	0.23
						4	38.93	0.30
						6	43.98	0.32
						24	51.23	0.44
						48	53.99	0.42
						72	55.85	0.59
BR-13	-10 Mesh	17.60	4.04	10.88	2.50	0	0.00	0.00
						2	27.12	0.09
						4	33.00	0.05
						6	35.60	0.05
						24	41.20	0.15
						48	42.83	0.16
						72	48.33	0.35

10.5.8 Bulk Sample – Crib & Column 120 Day Leach Results

The metallurgical results from the Mineral Park locked cycle crib and column leaching program are summarized below. After 120 days of leaching, the two cribs averaged 63% extraction of TCu at an average gangue acid consumption rate of 18.2 kg/tonne (see Table 10-12).

Table 10-12: MP Crib & Column Metallurgical Test Results

Bulk Sample MP Crib & Column 120 Day Leaching Results						
	Crib C-01	Crib C-02	Column CL-01	Column CL-02	*Average Cribs*	*Average Columns*
Copper Extraction %	64.2	61.8	55.0	53.1	***63.0***	***54.1***
Gangue Acid Consumption Kg/Tonne	18.8	17.6	17.4	17.3	***18.2***	***17.3***
Acid Consumption Kg Acid/Kg Cu	5.4	5.3	5.9	6.0	***5.4***	***6.0***
PLS pH	1.5	1.5	1.6	1.6	***1.5***	***1.6***
ORP (Mv)	613	609	589	525	***611***	***557***

The rate recovery curves for copper for the two cribs and two columns are shown in Figure 10-6.

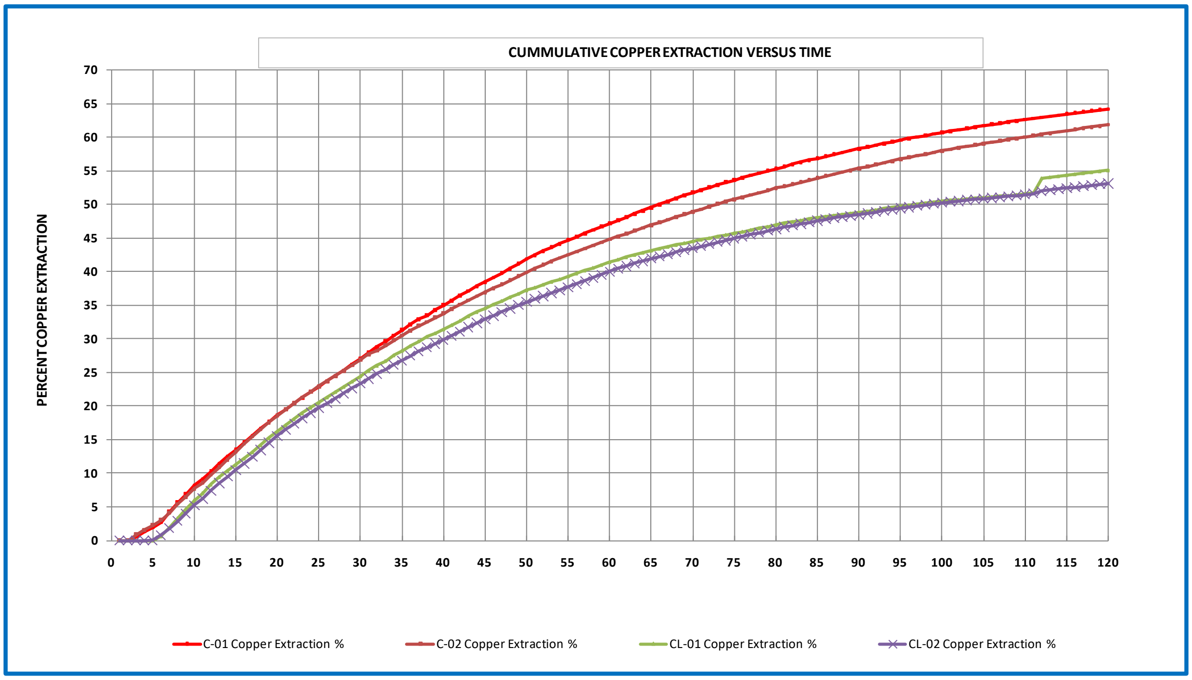


Figure 10-6: Copper Recovery Curves for Mineral Park Cribs and Columns

Acid consumption curves for the two cribs and two columns are shown in Figure 10-7.

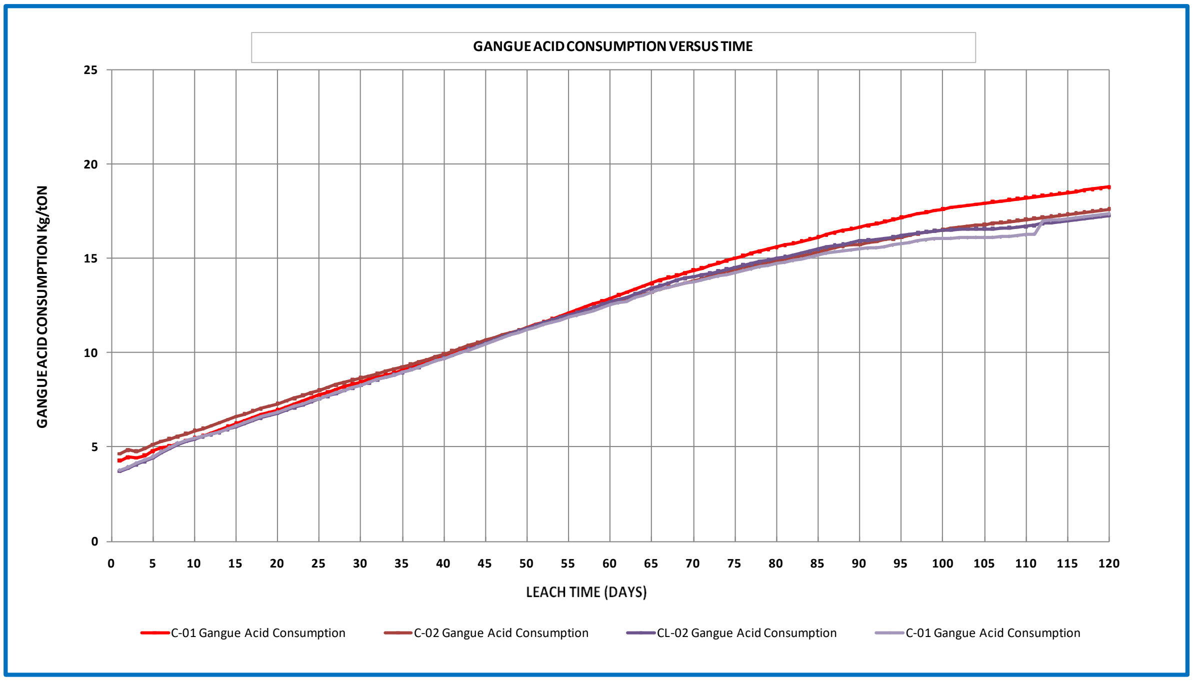


Figure 10-7: Acid Consumption Curves for Mineral Park Cribs and Columns

10.5.9 Bulk Sample – Crib & Column Leaching Results from 120 to 150 Days

After 120 days, the MP cribs were allowed to rest for 28 days and then leaching was continued for another 30 days (150 days total) under modified conditions. The modified conditions were to leach Crib 1 the same as before, by adding 10 g/L free acid and irrigating at 5.8 L/h/m^2, but Crib 2 was treated as a second lift, with the same irrigation rate but the raffinate acid content was controlled at pH 1.6. This was done to assess whether, under pH controlled conditions for C-02, the same rate of continued copper extraction could be attained but at lower acid consumption. The concept of free acid vs. pH-controlled acid management will be discussed in a separate section.

After 150 days, both cribs C-01 and C-02 and column CL-01 were rinsed according to a set of tear-down procedures provided by McNulty and then torn down meter by meter. Both cribs were screened at Mineral Park using the same screening plant as the head sample, and a residue sample sent to METCON for tail assaying (see Table 10-13 and Table 10-14 below and compare to head screen analysis in Table 10-5).

Table 10-13: C-01 Crib Tail Analysis

C-01 Tail Screen & Residue Sample Summary Table			
Fraction	**Sample Weight (kg)**	**% Fraction**	**Metcon Residue Sample (kg)**
+6"	3,547	1.38	80
-6" to +4"	7,212	2.82	93
-4" to +1 1/4"	53,914	21.04	316
-1 1/4" to +3/4"	31,198	12.18	302
-3/4"	160,327	62.58	1,633
TOTAL	**256,198**	**100**	**2,425**

Table 10-14: C-02 Crib Tail Analysis

C-02 Tail Screen & Residue Sample Summary Table			
Fraction	**Sample Weight (kg)**	**% Fraction**	**METCON Residue Sample (kg)**
+6"	4,137	1.56	71
-6" to +4"	6,305	2.38	91
-4" to +1 1/4"	49,587	18.74	311
-1 1/4" to +3/4"	32,568	12.31	281
-3/4"	172,039	65.01	1,587
TOTAL	**264,635**	**100**	**2,341**

The results obtained on the bulk sample composite sample for the additional 30 days of leaching are summarized in Table 10-15. Crib C-01 attained an additional 3.2 percentage points of copper recovery for an additional 1.9 kg/tonne acid consumption. Crib C-02 attained an additional 1.9 percentage points of copper recovery for an additional 0.1 kg/tonne acid consumption.

Table 10-15: Summary of Bulk Sample 150 Day Leach Crib & Column Results

	C-01	C-02	CL-01	CL-02
Copper Extraction	**67.8**	**65.4**	55.5	53.6
Iron Extraction (%)	**3.1**	**1.5**	1.1	1.3
Gangue Acid Consumption (kg/tonne)	**21.1**	**17.7**	17.4	17.3
Acid Consumption (kg Acid/kg Cu)	**5.8**	**5.0**	5.9	6.0
pH	**1.24**	**1.71**	1.55	1.55
ORP (Mv)	**581.00**	**613.00**	589.00	525.00
Humidity (%)	3.9	3.9	3.9	3.9
Wet Weight (kg)	255,635	258,197	3,217	3,278
Dry Weight (kg)	245,665	248,127	3,092	3,150

10.5.10 Bulk Sample Metallurgical Test Conclusions

The following can be concluded from results of the locked cycle tests conducted on the bulk sample composite.

- ROM particle size distribution at 80 percent passing 1 ¼" did not negatively impact copper extraction and gangue acid consumption compared to 80 percent passing 37.5 mm, 19 mm and irrigation flow rates of 6.1 and 7.8 L/h/m^2, respectively, using a sulfuric acid cured dosage of 4 kilogram per tonne of material.
- The copper extractions on the two composites, C-01 and C-02 ranged from 67.8% to 65.4% and gangue acid consumption ranged from 5.8 to 5.0 kilogram per kilogram of copper extracted (kg/kg Cu), and 21.1 to 17.7 kilogram acid per tonne of material (kg/ tonne) after a total leach cycle of 166 days (including 7 days of cure, 150 days of leach, 7 days of wash and 2 days of drain cycles).
- The highest copper extraction of approximately 71 percent was achieved at the size distribution of minus ¾" on the crib C-01.
- The lowest copper extraction of approximately of 43.5 percent was achieved at the size distribution of plus ¾" on the crib C-02.
- Percolation problems were not observed in the cribs during the leach cycle.
- There is a good correlation between the calculated head and assay head for copper and iron.

During tear down of the cribs and especially column CL-01, it was noted that the upper approximately three meters of the cribs and column CL-01 had very little visible oxide copper remaining after leaching. However, below three meters, and especially in the six meter tall Column CL-01 very significant amounts of unleached oxide copper were noted on fragment surfaces. To test whether or not this oxide copper was indeed leachable, samples were leached in beakers at Hazen Laboratories in Denver for three days at a pH of 1.6. Within three days all of the visible oxide copper was dissolved.

Sequential copper analyses on the head and leach residue samples were used to calculate the acid soluble copper, residual copper, and calculated total copper extractions for layers 1 meter thick. The results are summarized in Table 10-16.

Table 10-16: Summary of Crib Results Sequential Copper Extraction per Layer

MERCATOR MINERALS INC. RECURSOS STINGRAY DE COBRE S.A. DE C.V.					SUMMARY OF RESULTS CRIBS SEQUENTIAL CU EXTRACTION PER LAYER		
TEST No.: C-01 and C-02 SAMPLE : ROM "Bulk Sample"					CRUSH SIZE : ROM		
Copper Extraction vs Depth Locked Cycle Crib Evaluation							
	Sample	Sequential Copper			Calculated Extraction		
Head sequential Copper	ID	**ASCu**	**ResCu**	**Total Copper**	**ASCu**	**ResCu**	**Calculated Cu**
	Bulk Sample	**0.382**	**0.155**	**0.537**			
Residue sequential Copper	**C-01**						
Layers	L-01	0.033	0.072	0.12	91.3%	53.1%	77.75
	L-02	0.041	0.101	0.16	89.2%	34.6%	70.60
	L-03	0.063	0.114	0.20	83.5%	26.5%	63.42
	L-04	0.069	0.130	0.22	82.1%	15.9%	59.03
Residue sequential Copper	**C-02**						
Layers	L-01	0.035	0.089	0.14	91.0%	42.4%	74.73
	L-02	0.054	0.115	0.18	85.8%	25.8%	65.57
	L-03	0.106	0.128	0.26	72.2%	17.3%	51.12
	L-04	0.122	0.120	0.278	68.0%	22.4%	48.187

The following comments relate to the calculated acid soluble copper, cyanide soluble copper and total residual copper extractions on the bulk sample composite samples C-01 and C-02.

- Acid soluble copper extraction ranged from the top (1 m) to the bottom (4 m), from 91% to 82% on C-01 and 91% to 68% on C-02, respectively.
- Residual copper extraction ranged from the top (1 m) to the bottom (4 m), from 53% to 16% on C-01 and 42% to 22% on C-02, respectively.
- Calculated copper extraction ranged from the top (1 m) to the bottom (4 m), from 78% to 59% on C-01 and 75% to 48% on C-02, respectively.
- The metallurgical data developed from this analysis indicated that the distance below surface did impact the acid soluble copper and total copper extractions, with the best recoveries being obtained in the upper three meters of the cribs and columns.

In conclusion, copper recoveries from the bulk sample metallurgical tests at Mineral Park, and in particular the large sample crib copper recovery results, are interpreted to indicate that ROM leaching of the El Pilar should attain comparable recoveries to those predicted by the Stingray 2009 columns.

The results suggest that, if thin lifts on the order of three meters high are used at El Pilar, significantly better copper recoveries than those attained in the crib and column tests may be realized. Alternative interpretations are that (1) a longer leach time may have dissolved more copper from the lowest layer, and/or (2) more thorough precuring with better contact and a larger volume of curing solution may have improved the copper extraction from the lowest layer.

10.6 FIVE METCON COLUMNS TO BACKUP THE MP BULK SAMPLE CRIB AND COLUMN TESTS

Five additional columns were run at the METCON Tucson facility using material from the Bulk Sample employed in the two columns and two crib tests at MP. The objectives of the five METCON column tests were as follows:

1. Provide backup data for the Mineral Park crib and column tests.
2. Evaluate the effect of crush size on copper recovery and acid consumption.
3. Evaluate the effect of raffinate type on copper recovery and acid consumption.
4. Evaluate the effect of column aeration on copper recovery and acid consumption.

The five METCON columns were 203.20 mm in diameter by 6.0 m in height. The columns were run under locked cycle conditions at an acid strength of 10 g/L and at an irrigation rate of 5.3 L/h/m^2. The results of leaching the five backup columns for 120 days are summarized in Table 10-17.

Table 10-17: METCON 5 Column Bulk Sample 120 Day Metallurgical Test Results

METCON 5 Column 120 Day Leach Results on MP Bulk Sample						
	Column CL-01	**Column CL-02**	**Column CL-03**	**Column CL-04**	**Column CL-05**	***Average***
Column Parameters	**P80 0.75" Artificial Raffinate**	**P80 0.75" Mature Raffinate**	**P100 3" Mature Raffinate & Aeration**	**P100 3" Mature Raffinate**	**P100 3" Artificial Raffinate**	
Copper Extraction %	60.5	59.3	63.9	61.5	62.7	***61.6***
Gangue Acid Consumption Kg/Tonne	15.6	15.4	17.2	16.7	16.8	***16.3***
Acid Consumption Kg Acid/Kg Cu	4.7	4.8	4.7	4.8	4.7	***4.7***
PLS pH	2.1	2.0	1.5	1.4	1.5	***1.7***
ORP (Mv)	468	480	521	508	520	***499***

As backup tests, the five METCON column leach tests resulted in copper recoveries and acid consumptions that essentially duplicated those of the MP cribs, such that the average 120 day extraction of 61.6% copper from the five backup columns is very close to the MP crib average of 63% copper recovery and the gangue acid consumption of the five METCON columns averaged 16.3 kg/tonne compared to 18.2 kg/tonne for the MP cribs, respectively.

As a matter of history regarding some of the duplicate column strategy, the original five yearly composite columns run by METCON/Stingray in 2009 were run using “mature raffinate”. In this case, the mature raffinate used by METCON was from the nearby Silver Bell mine chalcocite copper heap leach operation in Arizona. Because this raffinate was from a mature sulfide bioleach pad, it contains ferric iron in solution. The concern was that using a mature raffinate on the El Pilar samples could conceivably result in enhanced recoveries not otherwise obtainable if natural El Pilar raffinate was allowed to mature. Accordingly, duplicate columns were run at METCON using “mature” raffinate from Silver Bell and “artificial” raffinate that was composed only of acid and water, similar to new mine startup conditions.

Two crush sizes were run in the columns at minus ¾” and minus 3” to evaluate the effect of crush size on copper recovery and acid consumption. One column was run using aeration to see if better copper recoveries might be attainable under more oxygenated column (pad) conditions.

The results of the five column leach tests show that crush size, aeration and raffinate type have little or no effect on copper recovery or acid consumption. In fact, the best overall recoveries were on the coarsest material, again supporting the plan to use ROM leaching at El Pilar.

Leaching of the five backup columns was continued at METCON beyond 120 days. This was done in an effort to look at what might be a real economic leach cycle for the El Pilar ore. The 180 day results are tabulated below in Table

10-18. On the average, the additional 60 days of leaching netted almost an extra 10 percentage points of additional copper recovery, or a very significant 15.9% average increase in copper recovery over the 120 day results. The gangue acid consumption during the same period added 7 kg/tonne of additional acid consumption for that ~15% increase in copper recovery.

Table 10-18: METCON 5 Column Bulk Sample 180 Day Metallurgical Test Results

METCON 5 Column 180 Day Leach Results on MP Bulk Sample						
	Column CL-01	**Column CL-02**	**Column CL-03**	**Column CL-04**	**Column CL-05**	***Average***
Column Parameters	P80 0.75" Artificial Raffinate	P80 0.75" Mature Raffinate	P100 3" Mature Raffinate & Aeration	P100 3" Mature Raffinate	P100 3" Artificial Raffinate	
Copper Extraction %	73.5	73.2	70.8	69.0	70.3	***71.4***
Gangue Acid Consumption Kg/Tonne	23.7	22.9	22.3	23.6	23.3	***23.2***
Acid Consumption Kg Acid/Kg Cu	5.8	5.6	5.4	5.9	5.7	***5.7***
PLS pH	1.7	1.7	1.6	1.6	1.6	***1.7***
ORP (Mv)	480	491	642	525	640	***556***

10.7 THIRTEEN COLUMNS – ADDITIONAL METALLURGICAL WORK 2010-2011

10.7.1 Objective

In order to develop the best estimate of copper recovery for the El Pilar deposit, 13 additional metallurgical samples were composited from drill core for column testing at METCON. The objective of the columns was to determine if there is a deposit-wide relationship between soluble copper grades and copper recovery and to define a mathematical grade-recovery relationship for copper recovery modeling purposes.

10.7.2 Thirteen Columns - Composite Selection

To develop the data needed to define the El Pilar copper grade-recovery relationship, the 13 columns were composited on the basis of targeted copper grade "bins" (see Table 10-19). A majority of the new column composites (Columns #'s 2, 3, 5, 6, 8, 9, 10 and 11) where selected on the basis of Ratio (%ASCu/%TCu) grade bins that were not represented/tested by the previous metallurgical work (this work included the original 5 Stingray composites and the MP/METCON bulk sample).

Three columns were selected on the basis of TCu (total copper) grade bins around the 0.15% TCu cut-off grade used for the 2009 reserve calculation. Column #1 was selected on the basis of rock type, in the breccia, to assess the copper recovery and acid consumption of this unit. Except for the breccia, the column composites were targeted from throughout the deposit to be as representative as possible by splitting the deposit vertically into thirds by bench elevation (1375-1275 m, 1265-1165 m & 1155-1055 m). To ensure that samples were representative, the theoretical target copper grade for each bin was determined using MineSight to average all of the blocks within those bench elevations for the selected grade bins.

The 13 samples were composited irrespective of host sediment lithology, due to the work discussed previously that shows that acid soluble copper grades have no apparent relationship to host lithology. Originally, 14 columns were identified to be composited but there was not enough sample material to composite Column numbers 4 & 7. Columns 8 and 8a were constructed from the same sample to determine the effect of raffinate temperature on copper recovery.

Table 10-19: Composite Selection Criteria for Thirteen New Columns

Grade Bins - For 2011 Metallurgical Columns Composite Selection								
					Target - from Average Bench Grades			
Category	Grade Bin	Data	New Column #	Composite Selection Bench Range	AVG TCu	AVG ASCu	AVG ResCu	AVG Ratio
Rock Type	NA - IBX	No	1	Column #1 - Equally Distributed in IBX	0.396	0.210	0.186	52.8%
% ASCu (Ratio)	.65 - .70	Yes		Not needed	-	-	-	-
% ASCu (Ratio)	.60 - .65	Yes		Not needed	-	-	-	-
% ASCu (Ratio)	.55 - .65	Yes		Not needed	-	-	-	-
% ASCu (Ratio)	.50 - .55	Yes		Not needed	-	-	-	-
% ASCu (Ratio)	.45 - .50	No	2	Column #2 from 1265-1165	0.307	0.147	0.160	47.8%
% ASCu (Ratio)	.45 - .50	No	3	Column #3 from 1375-1275	0.307	0.147	0.160	47.8%
% ASCu (Ratio)	.40 - .45	No	4 (IS*)	Column #4 from 1265-1165	0.291	0.132	0.159	45.4%
% ASCu (Ratio)	.40 - .45	No	5	Column #5 from 1375-1275	0.291	0.132	0.159	45.4%
% ASCu (Ratio)	.35 - .40	No	6	Column #6 from 1265-1165	0.273	0.110	0.163	40.2%
% ASCu (Ratio)	.35 - .40	No	7 (IS*)	Column #7 from 1375-1275	0.273	0.110	0.163	40.2%
% ASCu (Ratio)	.30 - .35	No	8 & 8A	Column #8 & 8A from 1265-1055	0.259	0.090	0.169	34.9%
% ASCu (Ratio)	.25 - .30	No	9	Column #9 from 1155-1055	0.263	0.082	0.181	31.1%
% ASCu (Ratio)	.20 -.25	No	10	Column #10 from 1155-1055	0.254	0.063	0.191	24.9%
% ASCu (Ratio)	.15 -.20	No	11	Column #11 from 1155-1055	0.257	0.055	0.202	21.5%
%TCu	.20 - .25	No	12	Column#12 from 1375-1275	0.223	0.093	0.130	41.9%
%TCu	.15 - .20	No	13	Column#13 from 1375-1275	0.174	0.049	0.126	27.9%
%TCu	.10 - .15	No	14	Column #14 from throughout deposit.	0.132	0.022	0.110	16.7%

**Note - IS denotes Insufficient Sample*

10.7.3 Thirteen Columns - Sample Collection QAQC

A majority of the material for the new composites was comprised of HQ core. The selection, bagging and shipping of these materials was supervised onsite by Rodolfo Sauceda, an independent geologist who was contracted from Resource Geosciences Hermosillo to oversee compositing QAQC. A complete QAQC report of the program was written by Sauceda and that report is summarized briefly herein.

It should be noted that, because some of the targeted composite intervals had been used previously for the original five Stingray composites, some of the material used for the composites were -10 mesh rejects from the previous metallurgical testing. Sauceda in his report summarizes the following sample collection procedures:

1. The Mineral Park Mine in Kingman, Arizona sent 204 barrels with drill core rejects from previous column tests to the core shack yard at El Pilar.
2. The drill core splits were also taken from the same core shack located at San Lazaro ejido outside of Nogales.
3. A clearly identified space was allocated for each column in the core shack yard in order to put together the samples for each composite.
4. Local staff, under the onsite supervision of the author (Sauceda), identified each sample in the barrels of crushed material or from drill hole cores based on the sample list provided by ML and placed them in their respective site columns (pallets) in the core shack.
5. At the end of this process, complete selections of the samples for each column were organized in the respective areas properly identified in the yard.
6. Each sample was then weighed and the weights recorded to calculate the combination of weights and copper grade parameters required for each composite. This compilation process was overseen on site by ML.
7. After obtaining the proper combination for each composite according to the targets requested by ML, the selected samples were packed in large properly identified plastic bags.
8. Finally, the plastic sacks were packed on pallets and then properly secured with plastic strips.
9. ML staff subsequently sent the pallets with the selected samples by column to METCON Research Inc. for sample preparation and further metallurgical testing in Tucson.
10. The work program resulted in the collection of 688 samples packed in individual plastic bags to make 13 sample composites with a total weight of 5,002.8 kg packed in 13 pallets containing 154 plastic sacks.

10.7.4 Thirteen Columns – Sample Locations & Final Composite Assays

The pit locations and distribution of samples that make up each of the 13 composites are shown in Figure 10-8 to Figure 10-19.

Figure 10-8: Sample Composite Locations – Column #1

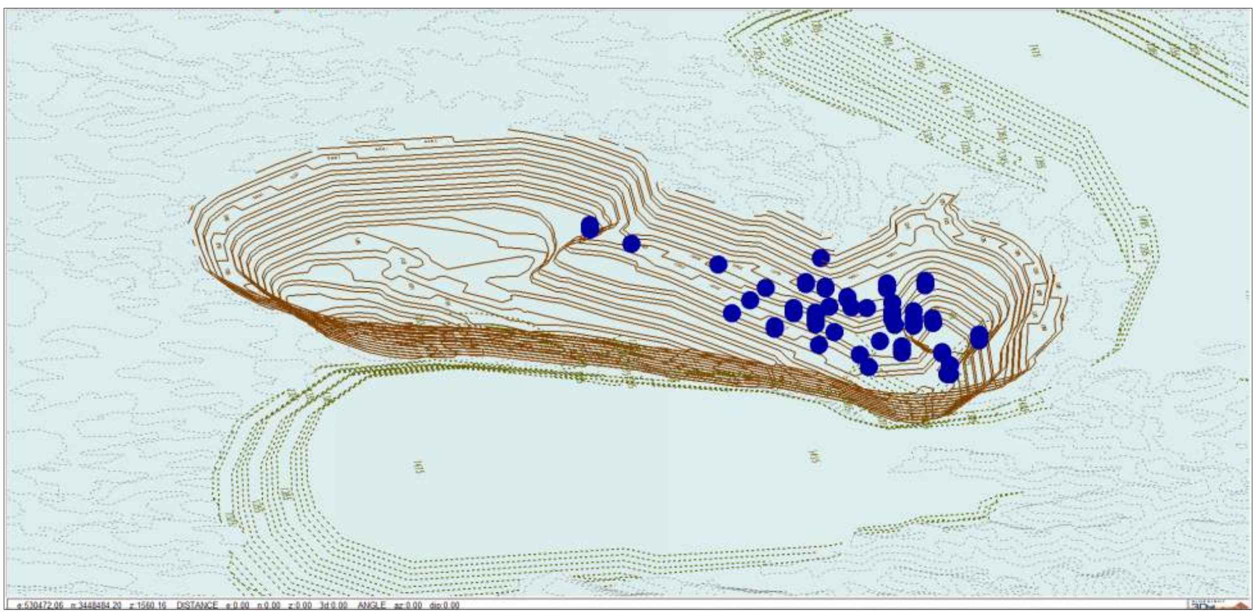


Figure 10-9: Sample Composite Locations – Column #2

Figure 10-10: Sample Composite Locations – Column #3

Figure 10-11: Sample Composite Locations – Column #5

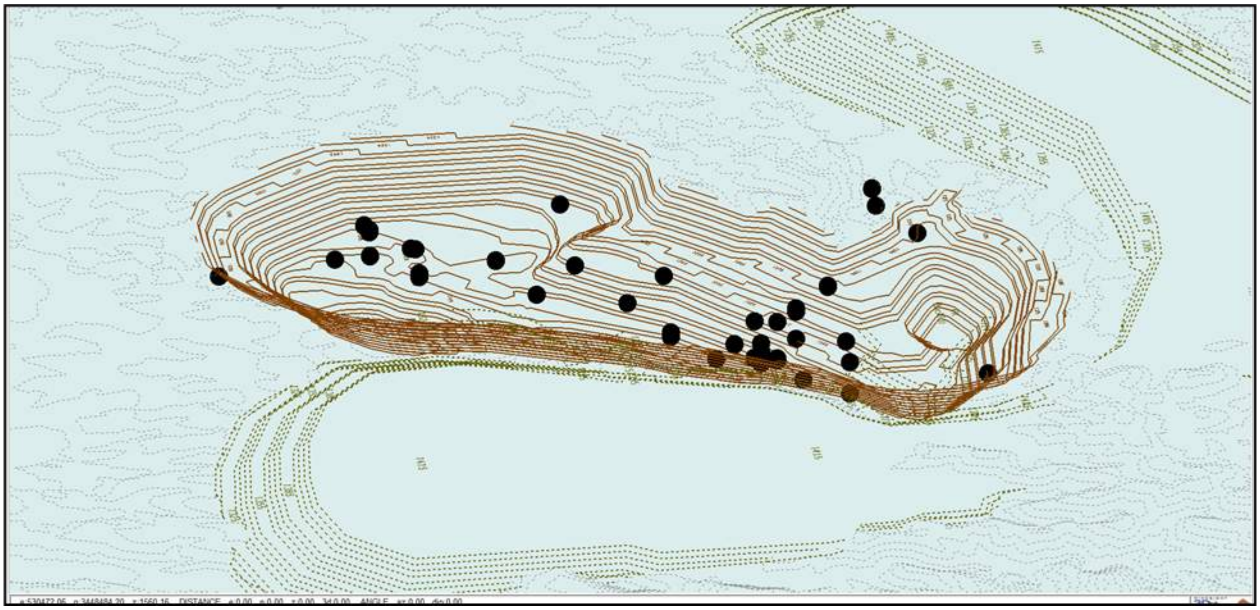


Figure 10-12: Sample Composite Locations – Column #6

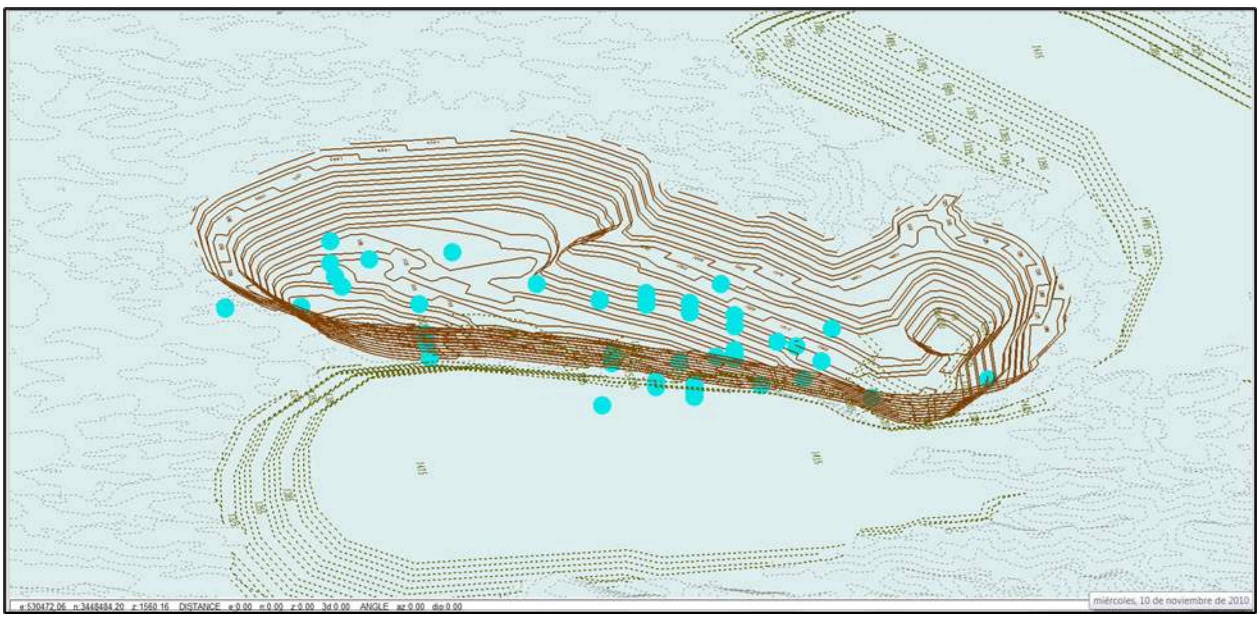


Figure 10-13: Sample Composite Locations – Column #8

Figure 10-14: Sample Composite Locations – Column #9

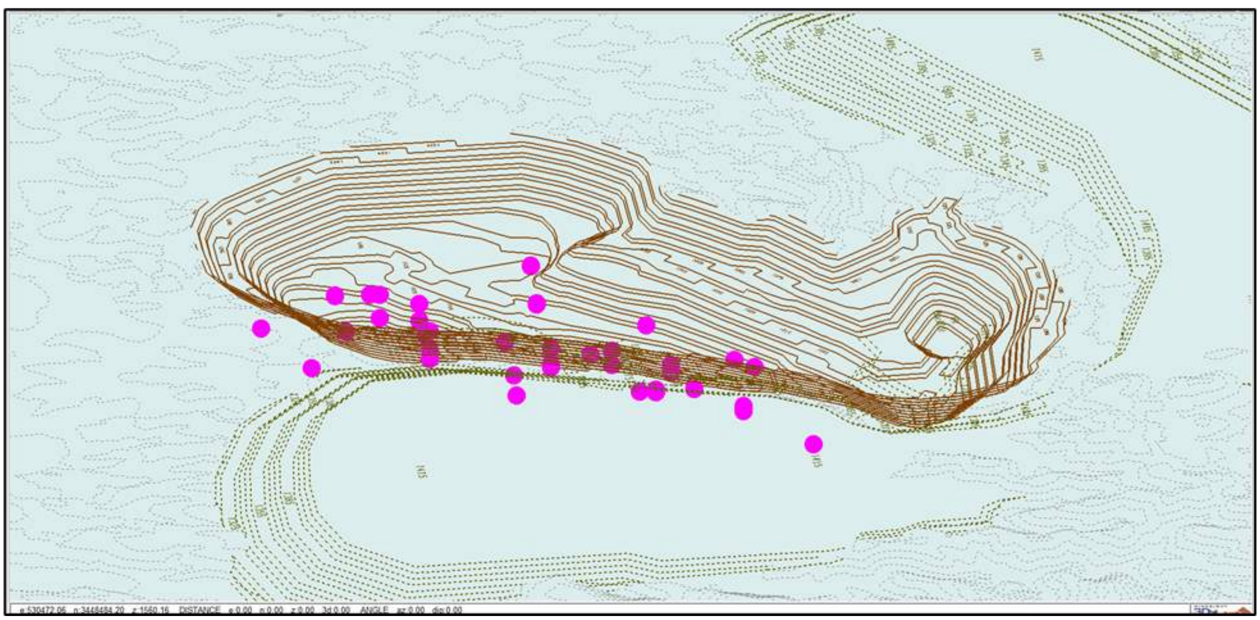


Figure 10-15: Sample Composite Locations – Column #10

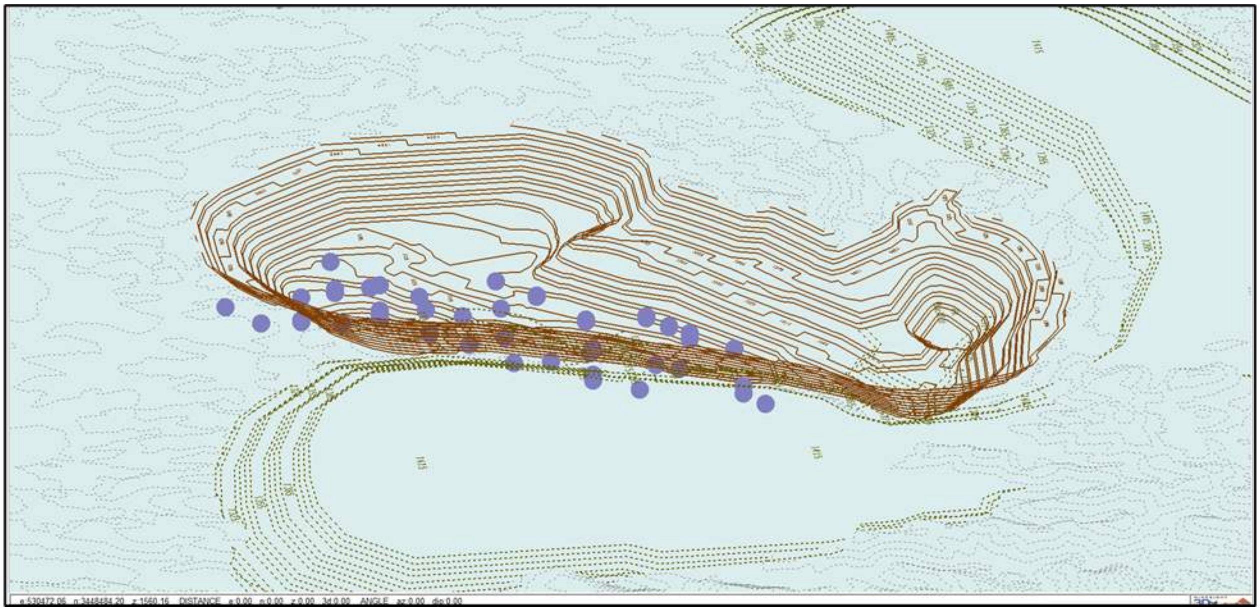


Figure 10-16: Sample Composite Locations – Column #11

Figure 10-17: Sample Composite Locations – Column #12

Figure 10-18: Sample Composite Locations – Column #13

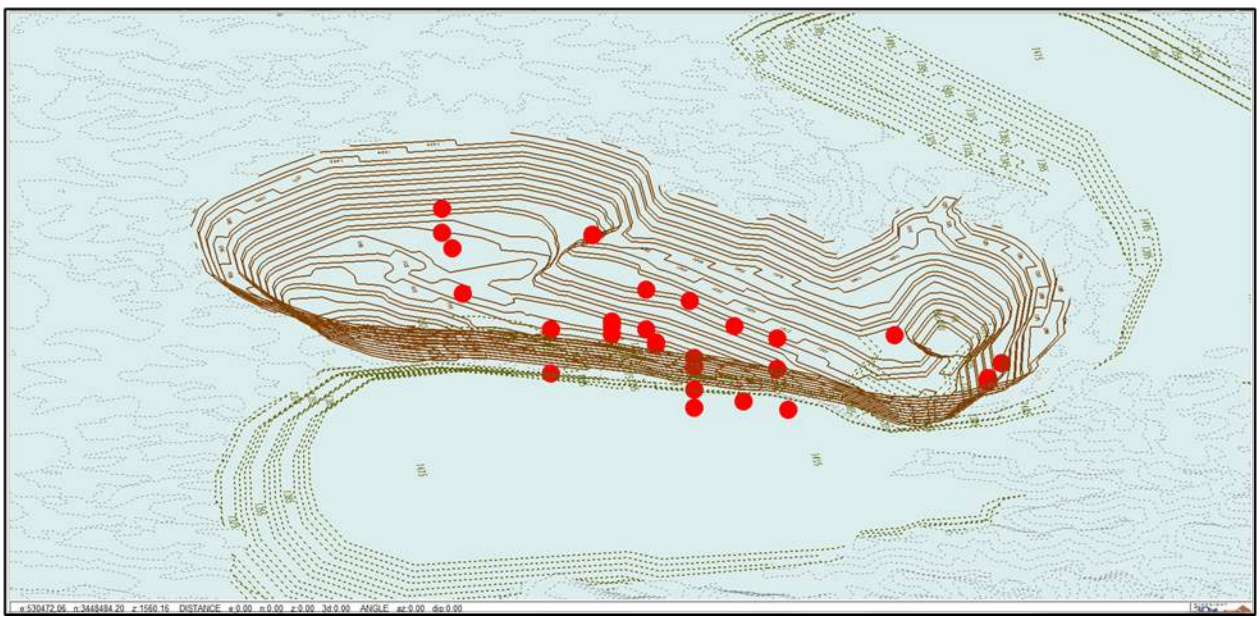


Figure 10-19: Sample Composite Locations – Column #14

The final composites from drill core/pulps for the 13 columns are tabulated in Table 10-20.

Table 10-20: 13 Column Final Composite Assays

Final Composite Assays - 13 Columns					
	Weight (kg)	**TCu**	**ASCu**	**Ratio**	**ResCu**
COLUMN 1	380.40	0.307	0.189	0.616	0.118
COLUMN 2	388.20	0.314	0.145	0.461	0.169
COLUMN 3	388.00	0.325	0.147	0.451	0.179
COLUMN 5	385.00	0.308	0.128	0.415	0.180
COLUMN 6	383.90	0.275	0.105	0.380	0.171
COLUMN 8	385.10	0.257	0.085	0.329	0.173
COLUMN 8-A	390.20	0.280	0.088	0.315	0.192
COLUMN 9	383.20	0.278	0.072	0.258	0.206
COLUMN 10	381.50	0.260	0.063	0.244	0.197
COLUMN 11	380.20	0.252	0.048	0.192	0.203
COLUMN 12	385.70	0.231	0.102	0.441	0.129
COLUMN 13	379.80	0.169	0.052	0.306	0.117
COLUMN 14	391.60	0.139	0.022	0.157	0.117
Total/Average	**5002.80**	**0.261**	**0.096**	**0.351**	**0.165**

10.7.5 Thirteen Columns - Leach Parameters

The 13 columns were leached at METCON according to the following leaching conditions:

- Irrigation rate: 7.8 liters per hour per square meter ($L/h/m^2$).
- Column height: 6 m.
- Curing: 4 kg of sulfuric acid per tonne, trickle down curing,
- Feed acid concentration: 10 g/L,
- Leach solution: fresh (water + acid)
- Leach cycle: locked, 180 days
- Column #8-A was run with heated solution, at a temperature of 65°F.
- Because the columns started out for the first ~20 days at high pH levels, due to an insufficient cure dosage, concentrated sulfuric acid was added to the feed by METCON, until the pregnant solution pH was at 2 or below.

10.7.6 Thirteen Columns - Bottle Roll Results

Bottle rolls were conducted on each of the column head samples using material ground to minus 10 mesh. The results of the bottle roll analysis are tabulated in Table 10-21 and Table 10-22.

Table 10-21: 13 Column Bottle Rolls Columns 1 – 7

SUMMARY OF METALLURGICAL RESULTS (1)
Ambient 72 hr. Bottle Roll Leach Tests (Pulverized)
Stingray Copper
METCON Research Project No. M-760-03

Test Number	Sample	Total Acid kg/t	Total Acid kg/kgCu	Gangue Acid kg/t	Gangue Acid kg/kg Cu	Leach Time (hr)	CUMULATIVE Cu %	CUMULATIVE Fe %
BR-01	Column 1	42.01	16.48	38.07	14.94	0	0.00	0.00
						2	24.88	12.49
						4	49.04	16.68
						6	64.72	17.10
						24	66.90	17.82
						48	69.46	18.93
						72	**75.39**	16.88
BR-02	Column 2	33.02	20.47	30.53	18.93	0	0.00	0.00
						2	26.96	13.32
						4	42.47	15.32
						6	47.61	16.84
						24	49.53	16.45
						48	51.21	18.16
						72	**52.20**	19.63
BR-03	Column 3	25.95	17.97	23.72	16.43	0	0.00	0.00
						2	23.15	10.28
						4	38.98	11.86
						6	42.62	12.46
						24	44.07	12.15
						48	44.56	13.91
						72	**44.79**	15.83
BR-04	Column 5	30.31	22.72	28.25	21.18	0	0.00	0.00
						2	21.60	10.07
						4	38.62	12.08
						6	41.73	12.53
						24	43.01	12.43
						48	43.10	13.50
						72	**45.62**	14.99
BR-05	Column 6	29.93	27.44	28.25	25.90	0	0.00	0.00
						2	21.35	7.84
						4	34.16	9.12
						6	37.87	9.58
						24	40.61	9.59
						48	40.54	10.28
						72	**41.35**	11.08
BR-06	Column 8-8A	28.65	33.63	27.33	32.09	0	0.00	0.00
						2	11.73	8.63
						4	25.18	10.13
						6	29.39	10.76
						24	31.01	10.60
						48	32.01	11.26
						72	**33.87**	12.33
BR-07	Column 9	24.29	31.54	23.10	30.00	0	0.00	0.00
						2	10.63	7.69
						4	22.76	8.66
						6	25.86	9.18
						24	27.70	9.05
						48	28.64	10.90
						72	**29.00**	10.78

Table 10-22: 13 Column Bottle Rolls Columns 8 – 12

SUMMARY OF METALLURGICAL RESULTS (2) Ambient 72 hr. Bottle Roll Leach Tests (Pulverized) Stingray Copper METCON Research Project No. M-760-03								
BR-08	Column 10	26.65	43.96	25.71	42.42	0	0.00	0.00
						2	8.28	7.33
						4	19.31	8.22
						6	22.32	8.66
						24	23.85	8.70
						48	25.81	9.76
						72	**25.53**	10.10
BR-09	Column 11	29.72	65.46	29.02	63.92	0	0.00	0.00
						2	6.38	7.90
						4	16.40	8.99
						6	18.41	9.43
						24	20.98	9.22
						48	23.30	10.58
						72	**21.43**	10.57
BR-10	Column 12	40.43	28.88	38.27	27.34	0	0.00	0.00
						2	10.99	14.20
						4	37.24	16.96
						6	37.41	18.75
						24	50.72	18.20
						48	51.08	19.42
						72	**48.54**	20.02
BR-11	Column 13	37.27	77.46	36.53	75.91	0	0.00	0.00
						2	3.90	11.07
						4	20.40	14.52
						6	27.76	15.73
						24	30.51	15.91
						48	34.41	17.44
						72	**30.21**	18.35
BR-12	Column 14	42.29	141.05	41.82	139.51	0	0.00	0.00
						2	2.31	46.19
						4	11.57	53.70
						6	13.31	57.21
						24	19.77	58.75
						48	20.14	65.32
						72	**18.37**	67.00

10.7.7 Thirteen Columns – Laboratory QAQC

The 13 column leach tests and assaying were conducted at METCON Laboratories in Tucson using in-house QAQC procedures.

10.7.8 Thirteen Columns - Results

The results of leaching the 13 columns are shown below for up to 120 days of leaching (Table 10-23). The following tabulations are based on daily column leach data for the 13 columns and not on tail results.

Table 10-23: 13 Columns 120 Day Leach Data

Cure Type	COLUMN TEST	As received Crush Size P80 mm	Sequential Head Assays as Cu	CNs Cu	Res. Cu	Cal Cu	% Extraction Cu (30 day cycles) 30	60	90	120	Solubility Index as Cu
Trickle Cured	1	38	0.228	0.009	0.066	0.303	38.40	57.50	**64.34**	68.40	**75%**
Trickle Cured	2	38	0.151	0.01	0.121	0.282	45.64	51.68	**55.38**	58.50	**54%**
Trickle Cured	3	25	0.152	0.014	0.151	0.317	41.81	46.74	**49.93**	52.75	**48%**
Trickle Cured	5	25	0.152	0.014	0.151	0.317	45.30	52.52	**56.05**	59.08	**48%**
Trickle Cured	6_7	38	0.111	0.01	0.123	0.244	38.79	46.26	**50.57**	53.93	**45%**
Trickle Cured	8	25	0.091	0.011	0.147	0.249	33.60	41.37	**45.51**	48.63	**37%**
Trickle Cured	8a	25	0.091	0.011	0.147	0.249	32.30	39.13	**43.71**	47.22	**37%**
Trickle Cured	9	25	0.086	0.008	0.164	0.258	31.31	37.81	**42.23**	45.80	**33%**
Trickle Cured	10	38	0.067	0.007	0.153	0.227	27.22	34.30	**38.50**	42.20	**30%**
Trickle Cured	11	38	0.054	0.006	0.156	0.216	22.70	29.07	**32.96**	36.38	**25%**
Trickle Cured	12	25	0.141	0.011	0.115	0.267	39.97	48.15	**52.02**	55.24	**53%**
Trickle Cured	13	25	0.054	0.005	0.096	0.155	25.04	32.97	**36.65**	39.84	**35%**
Trickle Cured	14	25	0.035	0.018	0.098	0.151	9.66	15.67	**19.05**	22.25	**23%**

Figure 10-20, Figure 10-21 and Figure 10-22 show cumulative copper extraction, gangue acid consumption and PLS pH, respectively.

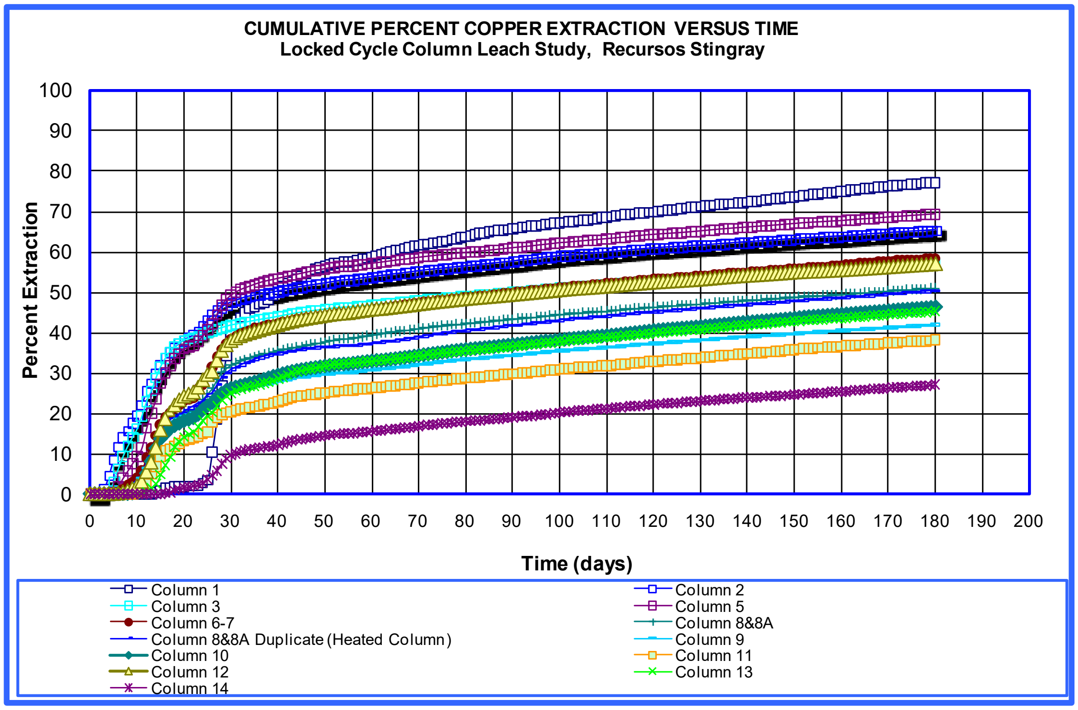


Figure 10-20: 13 Columns Cumulative Copper Extraction

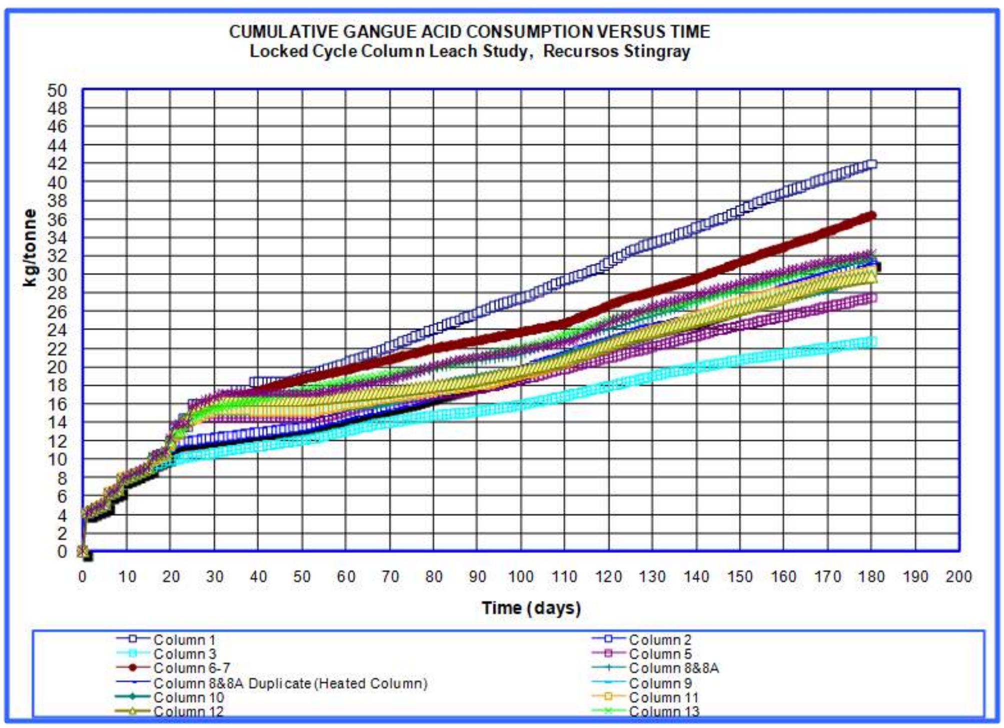


Figure 10-21: 13 Columns Gangue Acid Consumption

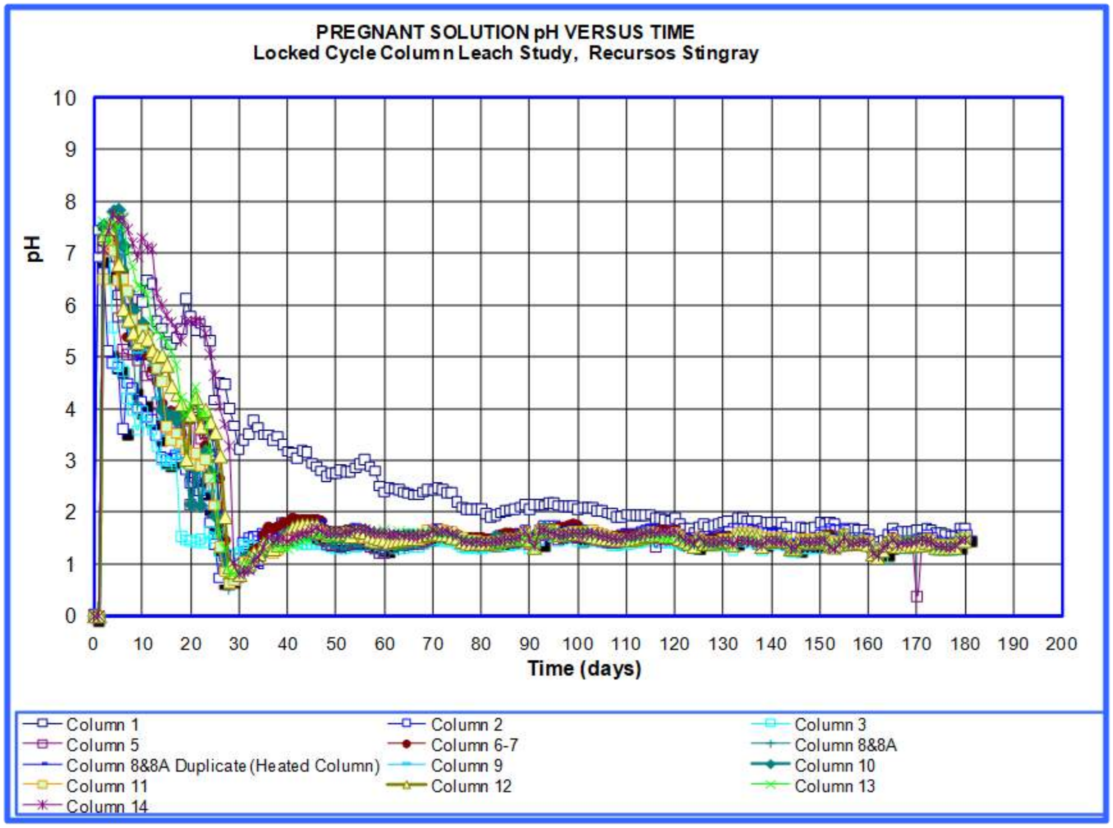


Figure 10-22: 13 Columns PLS pH

Results of the 13 column leach tests are as follows:

1. There is a clear correlation between the ASCu/TCu ratio and recovery.
2. ASCu/TCu ratio is progressively reduced as the production pit goes deeper, which will have an impact on recoveries of the later years.
3. Acid consumption averaged 23.4 kg/t at 120 days and 31.3 kg/t at 180 days. There is an increase of gangue acid consumption after 120 days, which may indicate this is the optimum cycle.

These samples selected for the 13 columns has the highest range in ASCu/TCu ratio and are representative of the ore throughout the LOM.

10.8 EL PILAR LOM CU RECOVERY PROJECTION

Data from the 2009 METCON composites and the 2011 variability composites (13 columns) were used to determine a recovery equation to be used in the economic model for this study. A recovery profile for leaching at 120 days was used from these data sets. Recovery has been normalized to the Solubility Index Ratio (SCu/Tcu), which can be used to infer the amount of copper oxides present in the ore. It also provides an indicator of the maximum amount of Cu that can be recovered.

A linear regression model was used to determine a recovery equation, using a logarithmic fit. Results are plotted in Figure 10-23.

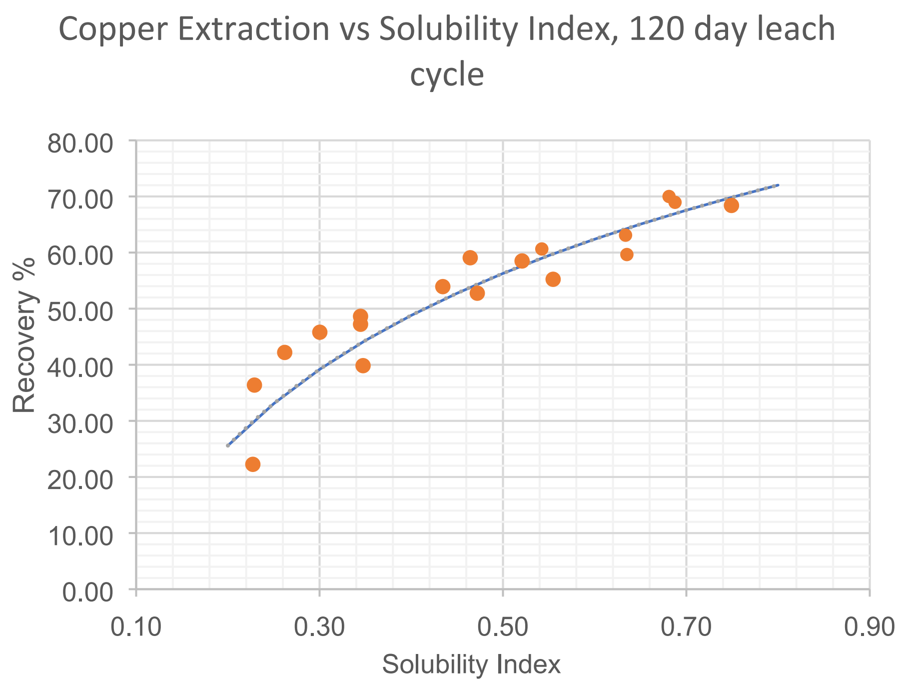


Figure 10-23 Recovery vs Solubility Index Recovery Equation

The recovery equation fits the data with a correlation coefficient of 0.92, over a wide range of solubility indexes representative of the deposit.

10.9 EL PILAR LOM ACID CONSUMPTION PROJECTIONS

Long-term acid consumption in this study is based on a value of 90% of the acid consumed in bottle roll tests performed on material employed in the original five Stingray yearly composites and the 13 columns. The rationale in using the bottle roll acid consumption data is that all of the reactive gangue material at a high fluid-solid ratio will react with the leach solution during the life of the test. The bottle roll tests used to determine the LOM acid consumption employed 500 gram samples at both -10 mesh and pulverized sizes, under conditions of 45% solids, iso-pH (1.4-1.8), and 72-hour test duration. Results used to calculate the 90% of bottle roll LOM acid consumption are shown in Table 10-24. The very low bottle roll acid consumption of 12.31 kg per tonne for the bulk sample was not included to avoid biasing results on the low acid consumption side. The ultimate LOM average acid consumption of 90% of the bottle roll average is 21.95 kg/t.

In addition, the bottle rolls, as opposed to the column test data, were used because most of the columns were run under inappropriate conditions wherein pH descended to levels as low as 1.0. Under such conditions, the kinetics of gangue dissolution may be increased by a factor of ten or more and most acid production precipitation reactions, such as that for ferrihydrite wherein the precipitation of 56 grams of iron produce 147 grams of sulfuric acid as shown below, are prevented.

$$Fe^{+3} + 3H_20 = Fe(OH)^3 + 3H^+ \quad \textit{(the precipitation of ferrihydrite, a goethite precursor).}$$

The acid consumption based on bottle roll data is in line with estimates of acid consumption based on gangue mineralogy. The principal gangue minerals are quartz, which is inert to leach solutions, and K and Na-feldspars which have very low rates of dissolution as shown by various laboratory studies. Based on quantitative X-ray diffraction (XRD) studies of material from heads and column leach tails, the principal acid consuming minerals are located in the -10 mesh fraction and consist of biotite, smectite, and clinoptilolite, which together make up less than about 10% of the total gangue. As shown by the XRD work, smectite and biotite are converted to illite, a mineral that is relatively stable in the leach solution. By contrast, clinoptilolite probably acts as an ion exchange medium in which hydrogen ions are taken up and copper and other ions are expelled. As shown by core logging and quantitative XRD study, the percentage of reactive gangue minerals increases with depth in the deposit. Based on the 90% average acid consumption, a flat 22 kg/t value is used in the economic model.

Table 10-24: Bottle Roll Acid Consumptions Used for 90% Total

EL Pilar Bottle Roll Testing Summary of Results						
Sample Id	**Extraction (%)**		**Gangue Acid Consumption**		**Total Acid Consumption**	
	Cu	**Fe**	**kg/t**	**Kg/kg Cu**	**kg/t**	**Kg/kg Cu**
Year 1	76.44	7.37	23.97	7.48	28.92	9.02
Year 2	75.98	6.49	24.36	7.72	29.23	9.26
Year 3	70.97	25.93	18.98	7.15	23.07	8.69
Year 4, 5 & 6	66.89	3.47	22.94	11.06	26.14	12.6
Year 7, 8 & 9	58.76	-2.19	25.46	15.04	28.07	16.59
Average	**69.81**	**8.21**	**23.14**	**9.69**	**27.09**	**11.23**
13 Column Average			30.61			
Average All			**24.39**			
90% of Average			***21.95***			

10.10 DISCUSSION OF OPTIMUM ACID PRECURE AMOUNTS

It is important to note that all of the El Pilar metallurgical column and crib tests were run using a relatively low acid precure amount of 4 kgs acid per tonne of mineralized material. The effect of this was twofold. First, essentially all of the columns run in 2010 and 2011 spent significant periods of time (up to ~20 days) under Eh-pH conditions outside of the copper solubility field. And second, acid precure can have a significant impact on copper rate recovery kinetics and as well may also have an impact on ultimate copper recovery.

To show how important initial Eh-pH conditions are to copper recovery at El Pilar, below is a Copper Species Eh-pH diagram (Figure 10-24) for 120 days of leach results for column MCL-01 (the first column in the 13 column series). For the first 25 days, the PLS solutions from the column stayed under high Eh-pH conditions well within the stability fields of both chrysocolla (yellow) and antlerite-brochantite (green). That means that the solutions were not in the field of copper aqueous solubility (blue) and that copper was not being dissolved, for the most part. To highlight this point, during that 25 day initial period the column experienced only 3.48% cumulative copper recovery. For this column, the high Eh-pH conditions were the result of high pH (the solutions after 25 days were still at pH 4.16). While the column actually attained 77% recovery of copper after 180 days of leaching, final recoveries might have been higher if an optimal precure was used.

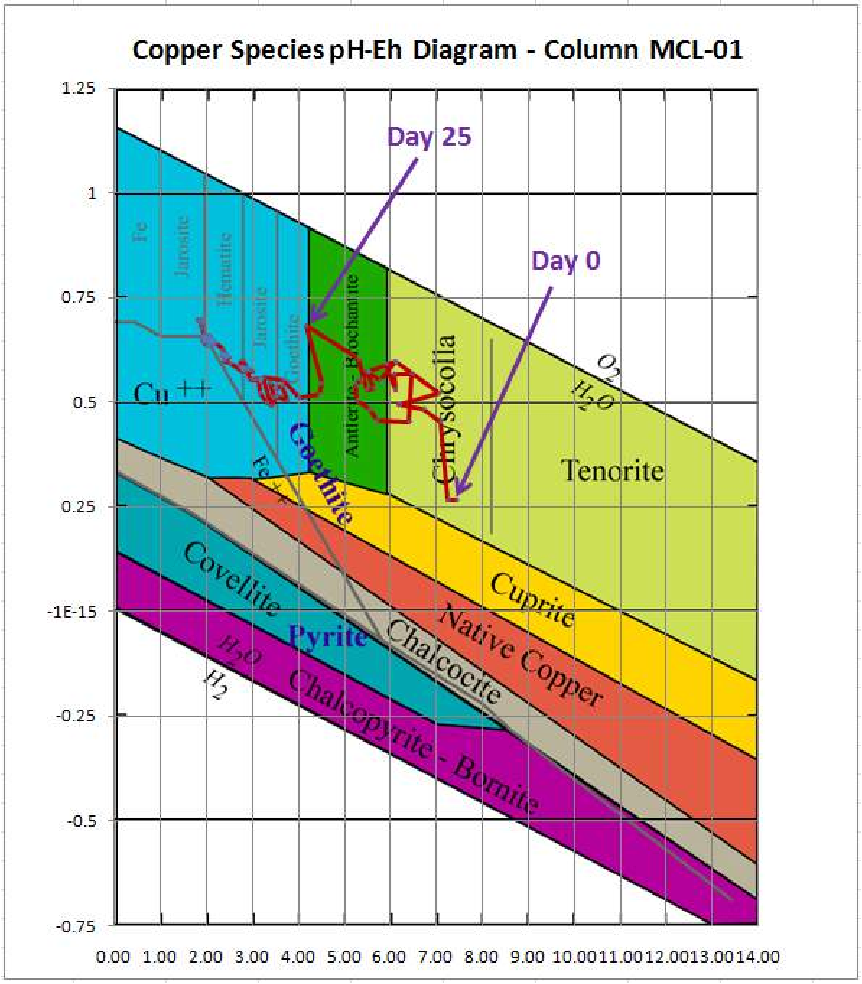


Figure 10-24: Copper Species Eh-pH Diagram for Column MCL-01

The diagram above illustrates why acid precures are used and why the right amount of acid precure should be selected for the El Pilar ore. The concept behind a precure is to move the material under leach as quickly as possible into the right pH conditions for copper dissolution. If this is not done correctly, copper can be dissolved at the top of the pile, where new acid being applied is concentrated, and then reprecipitated lower in the pile, where pH conditions are high. When this occurs, some of the copper can be reprecipitated in mineral/rock sites from which further dissolution may not easily occur. Figure 10-25 shows the results of 144-hour, sequential bottle roll tests run at Hazen Research on the Mineral Park bulk sample. These tests were run using different acid precures starting at 5 kg/tonne. The graph illustrates that twice as much copper was recovered using a 10 kg/tonne acid precure (40%) compared to using 5 kg/tonne (20%).

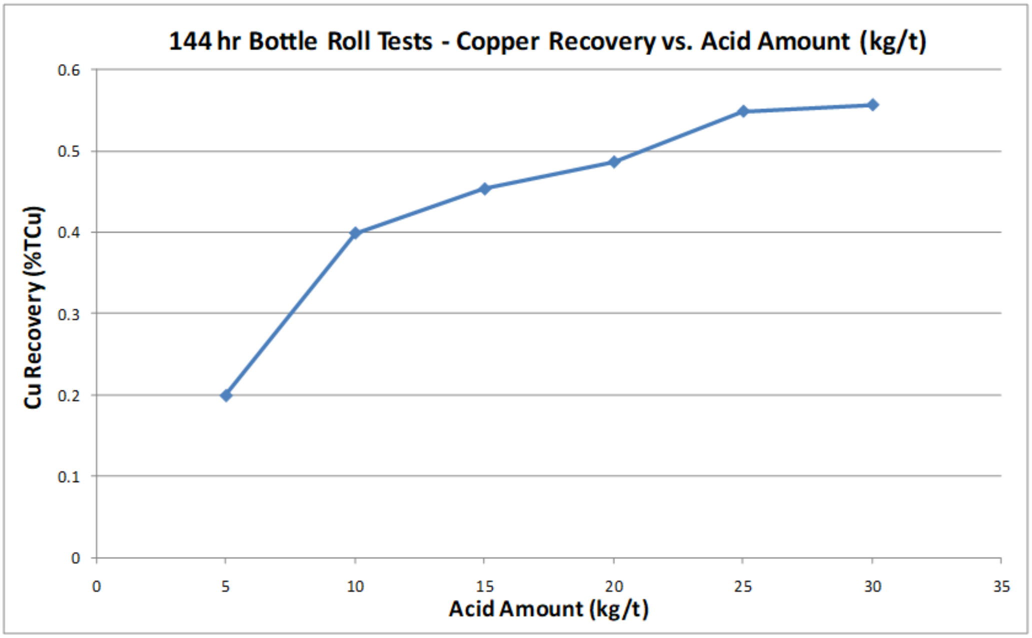


Figure 10-25: 144 Hour Bottle Rolls Copper Recovery vs. Acid Precure

Figure 10-26 also shows the results of bottle roll tests that were run at Hazen Research to assess the amount of acid needed to bring the bulk sample material to a pH of 1.5.

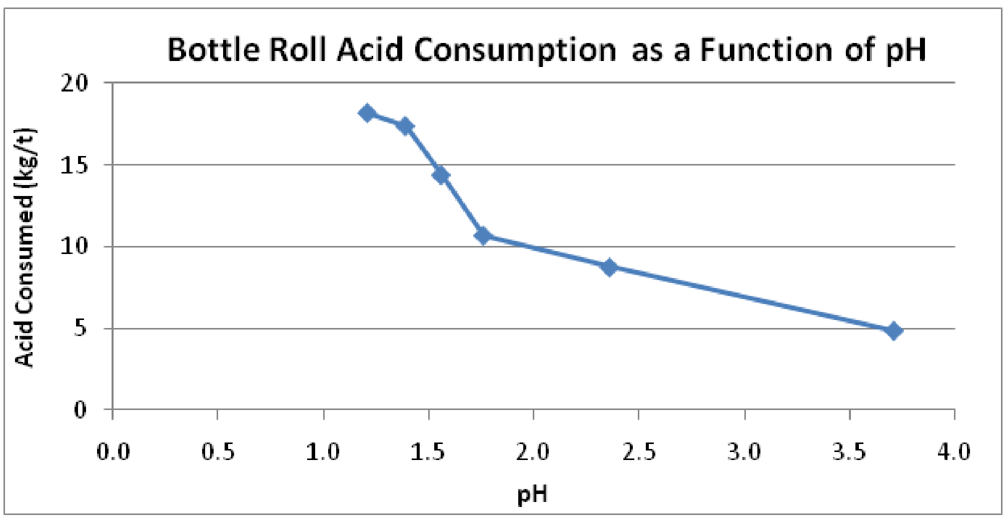


Figure 10-26: 144 Hour Bottle Roll Acid Consumption & pH

Within the test range, the above referenced bottle roll results indicate that acid consumption and period copper recovery are linear functions of the amount of acid added to the ore and that the optimal cure strength is between 10 and 15 kg/t of ore. For the purposes of this report, an initial acid cure rate of 10 kg/tonne is planned preliminarily for El Pilar operations. The El Pilar heap leach pad will undergo a top-down trickle down precure.

10.11 SUMMARY OF EL PILAR METALLURGICAL RESULTS AND PROJECT UPSIDES

Combined metallurgical tests from the Stingray 2009 program and from the bulk sample and additional 13 column composites tested in 2010 and 2011 result in the following conclusions:

1. El Pilar copper deposit consists of gravels that are poorly cemented and disaggregate almost completely into a "pre-crushed" size distribution on mining.
2. As a result of the above and based on the crib results, ROM leaching should attain recoveries comparable to the column test averages.
3. A 120 day leach cycle should be assumed initially, although real operating conditions may show that a different leach cycle is viable. Stacking plan must be adapted accordingly to accommodate these cycles.
4. Copper recoveries at El Pilar are at least initially a function of copper solubility, although mineralogical studies suggest that over longer periods of time a considerable amount of the residual copper may be recovered.
5. There is a grade recovery relationship for 120 days of leaching as defined by the formula, Recovery % (of TCu) = $33.49\ln(X) + 79.49$, where X is the Ratio (%ASCu/%TCu).
6. An initial precure rate of about 10 kg per tonne acid is recommended.
7. LOM acid consumption should average approximately 22 kg acid per tonne of ore.

Several potential project upsides are suggested by the metallurgical testing program and results. While these potential project enhancements are not fully quantified in this study they are significant and include the following:

1. Using a ~10 kg per tonne precure, rather than the 4 kg per tonne precure used in the metallurgical tests, will likely result in faster copper recovery rates that could positively impact project economics and allow for a shortened leach cycle, as well as potentially better copper recoveries over the life of mine.
2. Curing method will also be important, since ROM ore will be used. Trickle down curing will cause a lag in recovery, a thorough wetting of the ore will be crucial to achieve the projected recovery curve.

11 MINERAL RESOURCE ESTIMATES

11.1 KEY ASSUMPTIONS, PARAMETERS, AND METHODS

This sub-section contains forward-looking information related to density and grade for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including actual in-situ characteristics that are different from the samples collected and tested to date, equipment and operational performance that yield different results from current test work results.

11.1.1 Introduction

This Mineral Resource estimate was determined using a block model methodology based on the Inverse Distance squared (ID^2) and Ordinary Kriging (OK) interpolation methods. Drill hole sample data was capped to control outlier values and composited for equal sample weighting. Mineral Resource categories were assigned to the model based on drill hole spacing relative to the spatial continuity of the deposit and the search pass that block grades were estimated in. Mineral Resource estimates were constrained by an open pit shell based on economic criteria outlined in Section 11.6.

11.1.2 Available Data

A March 2008 topographic surface was provided by SCC for the EL Pilar deposit area. The surface was found to reasonably match the drill hole collar locations and no material issues were identified.

The drill hole database provided by SCC consisted of 297 drill holes, including 20,929 sample intervals, 3,289 specific gravity (SG) measurements for a combined total of 65,218 m of drill hole data. The sample data consisted of 18,992 total Cu and 16,844 soluble Cu assays. The drill holes data was collected between 1998 and 2016, as described in Section 7 of this TRS.

Lithological units included in the model are summarized in Table 11-1.

Table 11-1: Description of El Pilar Rock Types

Rock	Code	Description
Qfy	103	Quaternary Younger Alluvial Fan Deposits
Qwu	201	Quaternary Alluvial Wash Deposits Upper
Qwt	202	Quaternary Alluvial Wash Deposits Transitional
Qwl	203	Quaternary Alluvial Wash Deposits Lower
PCGr	102	Precambrian Granitic Intrusive Rocks
Bx	205	Breccia

The data was reviewed for interval errors and out of range assays values prior to import into Datamine RM software. Additional data validation is discussed in Section 9.1. No material issues were identified during this process.

The assay data was modified to account for zero grade values and unsampled intervals. All zero-grade total Cu and soluble Cu data were set to a minimum value of 0.0005% and all unsampled total Cu intervals were assumed to be waste and also set to a value of 0.0005%. Intervals with an absent soluble Cu assay that have a valid total Cu assay were assigned a calculated soluble Cu value based on the following linear regression formulas for each lithological unit outlined in Table 11-2. The use of regression formulas was supported by strong correlations identified between soluble Cu and total Cu.

Table 11-2: Regression Formulas for Soluble Cu

Lithological Unit	Regression Formula
Qfy	CuSoL= Cu x 0.72108 - 0.01231
Qwu	CuSoL = Cu x 0.814 - 0.06697
Qwt	CuSoL = Cu x 0.60637 - 0.05142
Qwl	CuSoL = Cu x 0.63067 - 0.00941
PCGr	CuSoL = Cu x 0.72061 - 0.01348
Bx	CuSoL = Cu x 0.87772 - 0.04348

A total of 2,023 soluble Cu values were calculated by regression formula representing 8.9% of the soluble Cu assay data.

The SG data was modified to remove invalid measurement values that were entered as 0 and to limit the dataset to within expected values. All zero values were removed and set to absent data and SG values greater than 2.8 g/cm^3 were capped at a value of 2.8 g/cm^3 based on the analysis of data using descriptive statistics including histograms, XY scatter plots and box plots.

11.1.3 Geological Domains and Modelling

A Lithological domain model was provided by SCC for the El Pilar deposit, as shown in Figure 11-1. The model was imported into Datamine RM software and compared against the drill hole data. On review, the model was found to be representative of the data and was accepted for the purpose of mineral domain constraints for use in the Mineral Resource estimate.

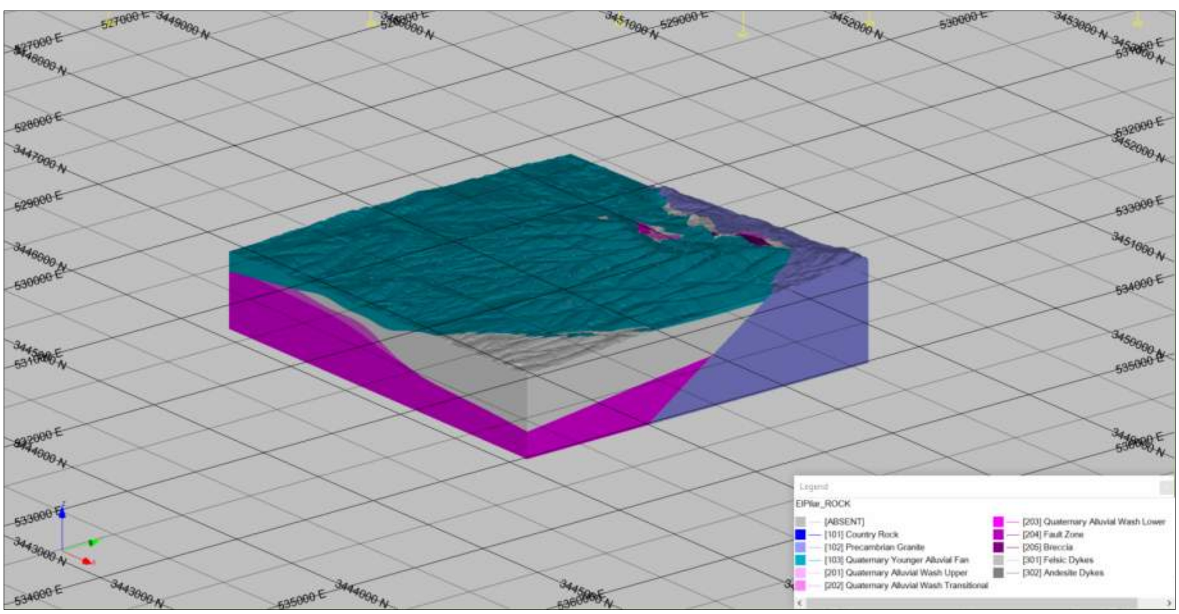


Figure 11-1: El Pilar Lithological Model (Oblique View Facing North-West)

The Cu mineralization is hosted mainly within the Quaternary alluvial deposits and the breccia unit, as outlined in Figure 11-2 and Figure 11-3. Figure 11-2 illustrates the lithological domains, and Figure 11-3 illustrates the lithological domains and the distribution of Cu mineralization in the drill holes.

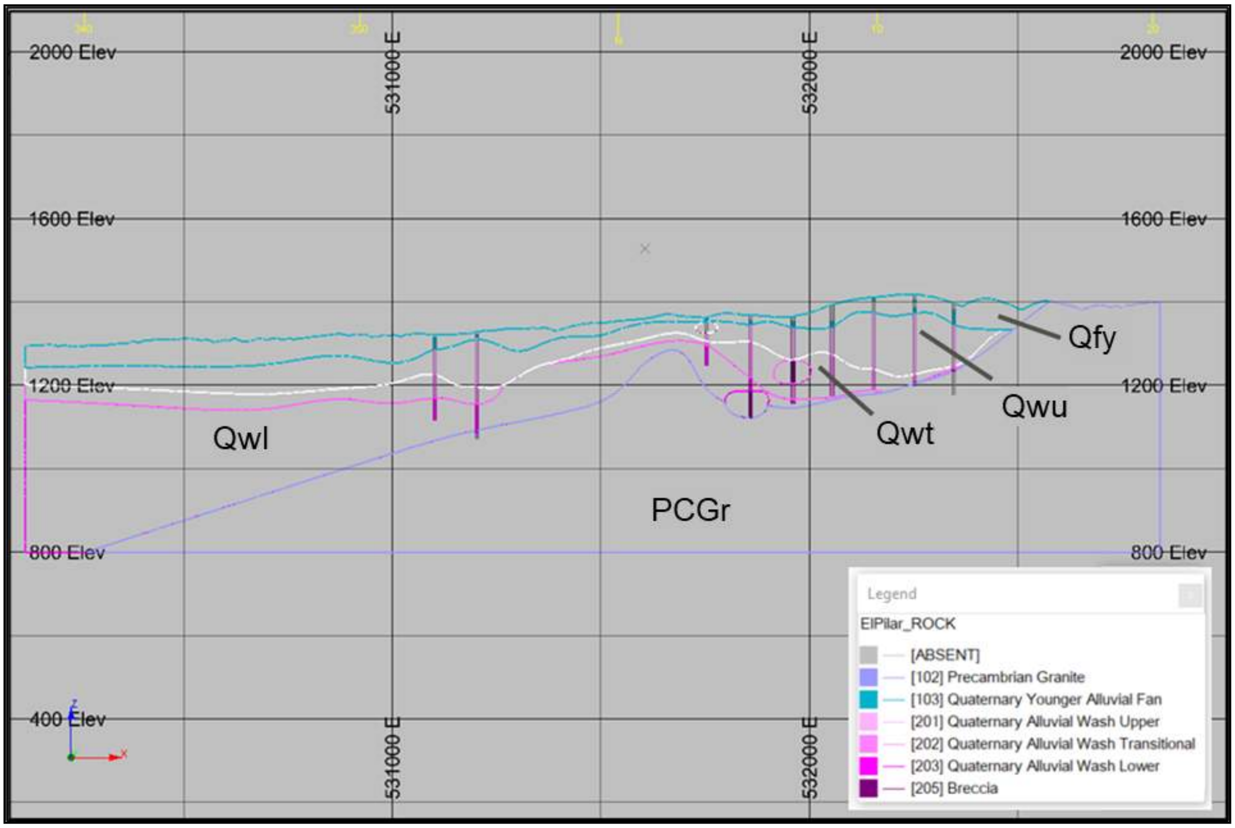


Figure 11-2: East-West Cross-Section of Lithological Domains

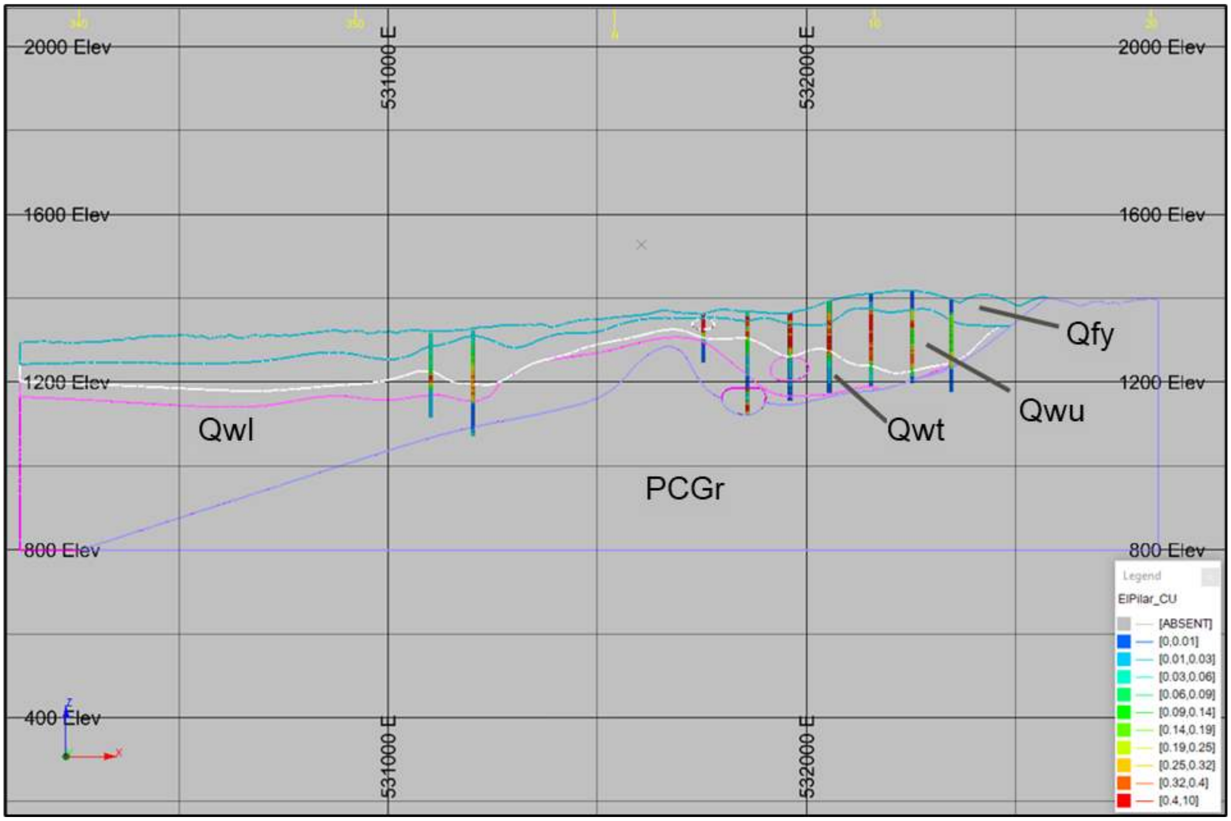


Figure 11-3: East-West Cross-Section Showing the Distribution of Cu Mineralization

The majority of the mineralization is hosted within the Qwu and Qwt alluvial units as well as the Bx unit as indicated in the statistical analysis presented in Section 11.1.3 of this TRS.

11.1.4 Exploratory Data Analysis

The sample data, selected within the limits of the lithological model, was analyzed for total Cu and soluble Cu within each lithological domain using descriptive statistics as well as a series of graphs including histograms, probability plots, X-Y scatter plots and box plots for the purpose of describing the sample population and identifying outlier assay values. Total and soluble Cu populations were found to have a positively skewed distribution with the presence of some outlier grade values. Box plots for total and soluble Cu demonstrate that the majority of the mineralization is hosted within units Qwu, Qwt, and Qwl, as shown in Figure 11-4 and Figure 11-5. Table 11-3 and Table 11-4 summarize the descriptive statistics for these sample populations.

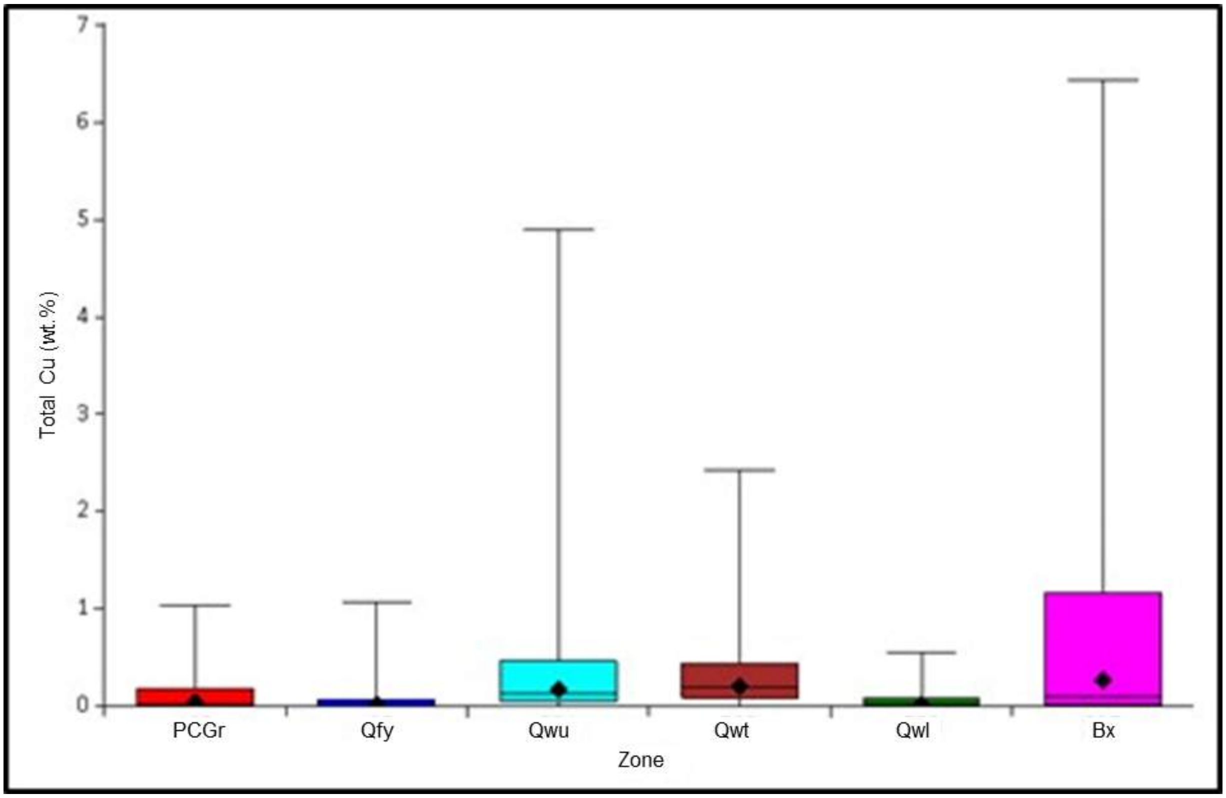


Figure 11-4: Box Plot of Total Cu

Table 11-3: Summary Total Cu Statistics by Zone

Metal	Unit	Count	Min	Max	Mean	Variance	StDev	CV
Total Cu	All	22,754	0.00	6.44	0.12	0.03	0.17	1.44
Total Cu	Qfy	3,101	0.00	1.07	0.02	0.00	0.07	4.14
Total Cu	Qwu	9,541	0.00	4.90	0.17	0.03	0.16	1.00
Total Cu	Qwt	5,902	0.00	2.43	0.20	0.02	0.14	0.70
Total Cu	Qwl	2,128	0.00	0.55	0.02	0.00	0.05	2.74
Total Cu	PCGr	1,066	0.00	1.03	0.03	0.01	0.09	2.59
Total Cu	Bx	1,016	0.00	6.44	0.27	0.21	0.46	1.70

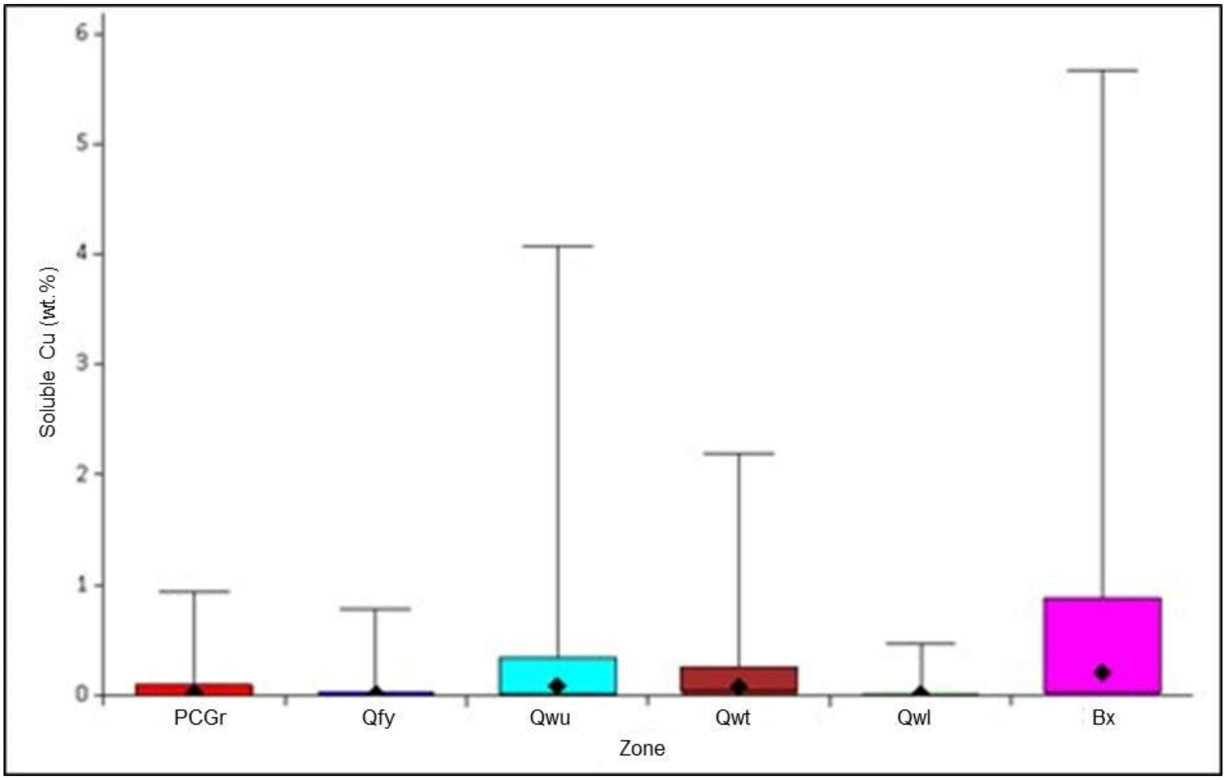


Figure 11-5: Box Plot of Soluble Cu

Table 11-4: Summary Soluble Cu Statistics by Zone

Metal	Unit	Count	Min	Max	Mean	Variance	StDev	CV
Soluble Cu	All	22,754	-	5.67	0.06	0.02	0.13	2.26
Soluble Cu	Qfy	3,101	-	0.78	0.01	0.00	0.05	5.20
Soluble Cu	Qwu	9,541	-	4.07	0.08	0.02	0.13	1.63
Soluble Cu	Qwt	5,902	-	2.19	0.08	0.01	0.09	1.23
Soluble Cu	Qwl	2,128	-	0.46	0.01	0.00	0.03	4.85
Soluble Cu	PCGr	1,066	-	0.94	0.02	0.00	0.06	3.68
Soluble Cu	Bx	1,016	-	5.67	0.20	0.15	0.39	2.00

Assay grade data was evaluated for outlier values using probability plots and scatter plots for each unit. Outlier values were identified and capped (top-cut) for the purposes of grade estimation. Capping values were defined for each unit as presented in Table 11-5.

Table 11-5: Capping Limits for Total and Soluble Cu

Unit	Total Cu (%)	Soluble Cu (%)
Qfy	0.60	0.60
Qwu	1.00	1.00
Qwt	0.80	0.80
Qwl	0.40	0.40
PCGr	0.60	0.60
Bx	2.50	2.50

Capped sample statistics were generated and indicate a minor reduction in mean grades and CV values for some units, but overall reductions were found to be insignificant as outlined in Table 11-6.

Table 11-6: Summary Comparison of Capped vs Uncapped Statistics

Metal	Unit	No. of Samples Capped	Uncapped Mean	Capped Mean	Uncapped CV	Capped CV
Total Cu	Qfy	8	0.02	0.02	4.14	3.86
Total Cu	Qwu	5	0.17	0.16	1.00	0.95
Total Cu	Qwt	5	0.20	0.20	0.70	0.68
Total Cu	Qwl	16	0.02	0.02	2.74	2.62
Total Cu	PCGr	7	0.03	0.03	2.59	2.49
Total Cu	Bx	6	0.27	0.26	1.70	1.58
Soluble Cu	Qfy	0	0.01	0.01	5.20	5.13
Soluble Cu	Qwu	5	0.08	0.08	1.63	1.54
Soluble Cu	Qwt	3	0.08	0.08	1.23	1.18
Soluble Cu	Qwl	3	0.01	0.01	4.85	4.80
Soluble Cu	PCGr	0	0.02	0.02	3.68	3.57
Soluble Cu	Bx	6	0.20	0.19	2.00	1.89

Raw sample interval lengths were analyzed for the purpose of selecting an average composite length for block model grade estimation. The modal sample length was found to be 2 m, which was therefore selected as the length used for compositing the sample data into relatively equal lengths.

11.1.5 Spatial Continuity

The spatial continuity of total and soluble Cu grades was evaluated for the mineralized units Qwu, Qwt and Bx through the use of variogram analysis. Experimental variogram data was generated using the parameters presented in Table 11-7.

Table 11-7: Experimental Variogram Parameters

Variogram Parameter	Value
Lag	20 m
# of lags	25
Horizontal Angle	22 degrees
Vertical Angle	22 degrees
Cylinder Radius	30 m
Z-axis Rotation (Strike Direction)	40 degrees
Y-axis Rotation (Dip Direction)	-25 degrees

Two-structure spherical variograms were then modeled based on the experimental variogram data, as shown in the total Cu example for unit Qwu in Figure 11-6 and Figure 11-7. These models were then used to define the sample search ellipse dimensions and for the assignment of Kriging weight values to the samples for the purpose of grade estimation using OK, as further discussed in Section 11.2.

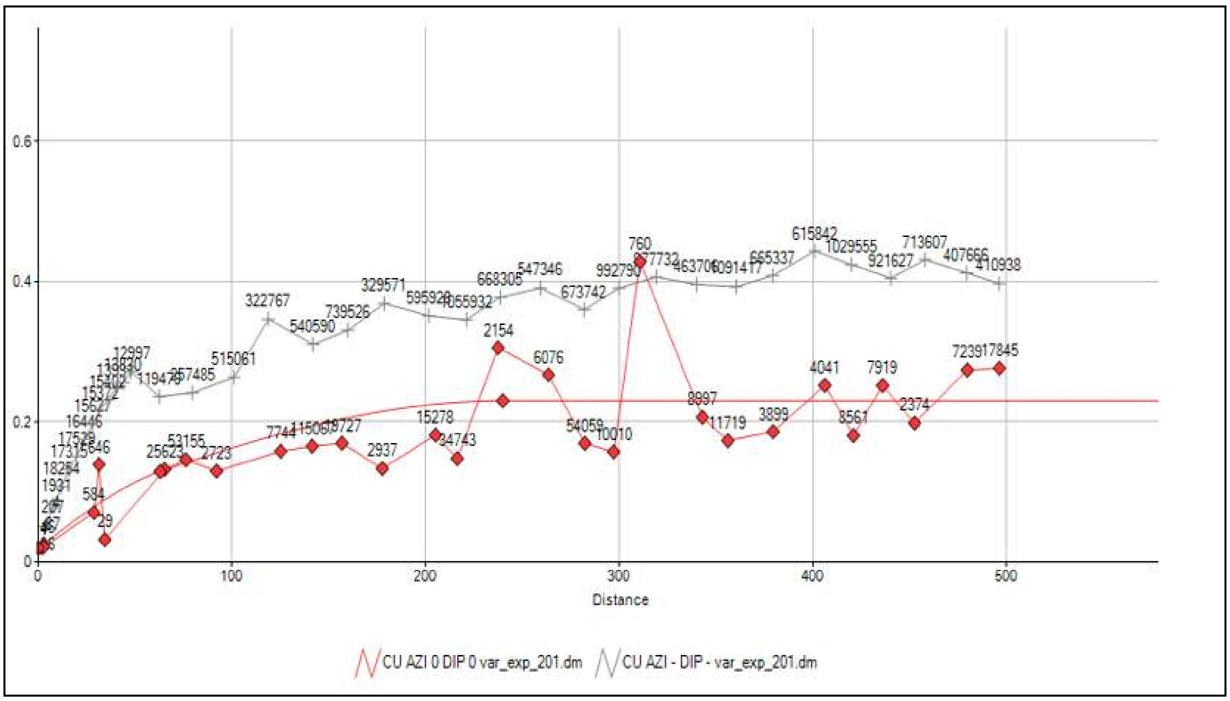


Figure 11-6: Along-Strike Variogram Model for Unit Qwu Total Cu

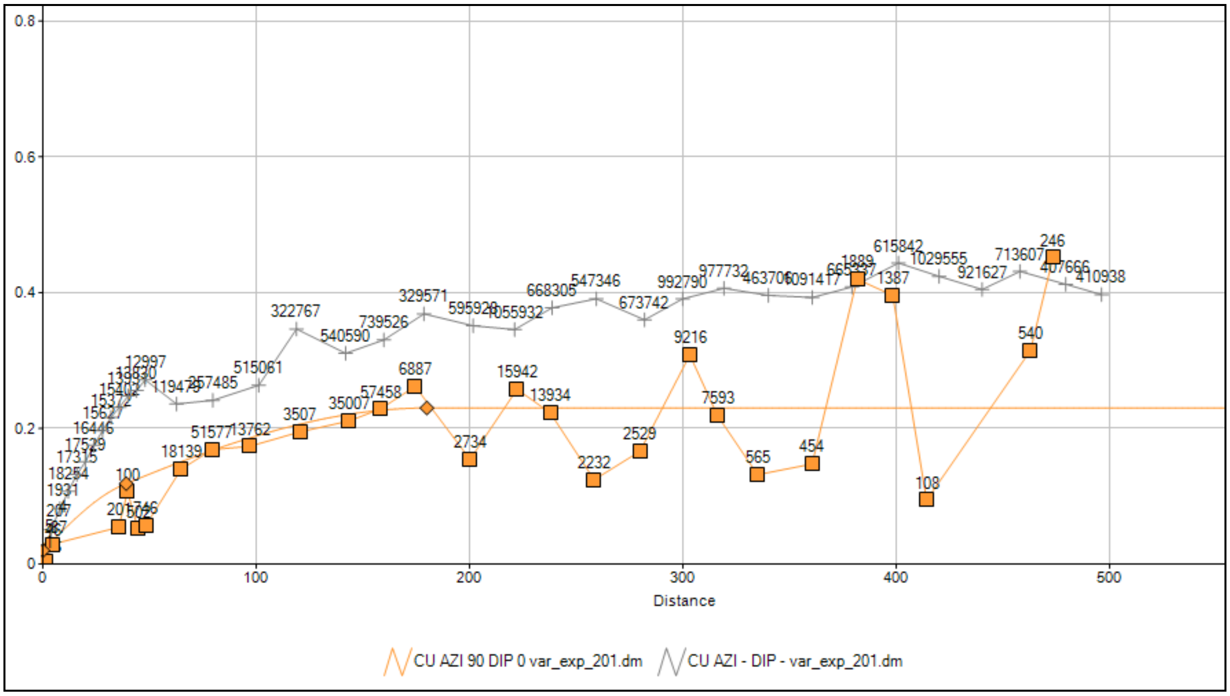


Figure 11-7: Down-Dip Variogram Model for Unit Qwu Total Cu

A summary of all variogram model parameters is presented in Table 11-8.

Table 11-8: Summary of Variogram Model Parameters

Unit	Variable	Vangle1	Vangle2	Nugget	Range1 X	Range1 Y	Range1 Z	Sill 1	Range2 X	Range2 Y	Range2 Z	Sill 2
Qwu	Total Cu	40	-25	0.02	37.50	63.00	35.60	0.05	179.60	239.70	79.90	0.16
Qwu	Soluble Cu	40	-25	0.03	49.20	70.10	28.30	0.12	175.50	239.70	79.60	0.59
Qwt	Total Cu	40	-25	0.04	43.20	69.00	35.60	0.17	178.70	239.40	79.90	0.15
Qwt	Soluble Cu	40	-25	0.04	42.00	36.40	35.60	0.47	190.90	183.60	79.90	0.13
Bx	Total Cu	0	0	0.08	22.10	37.90	34.80	0.37	130.90	68.30	58.80	0.17
Bx	Soluble Cu	0	0	0.08	7.20	25.20	26.70	0.49	94.50	51.20	58.80	0.15

11.1.6 Grade Modelling

This sub-section contains forward-looking information related to grade for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including actual in situ characteristics that are different from the samples collected and tested to date, equipment and operational performance that yield different results from current test work results.

A 3D grade block model was generated using Datamine RM software. Block model parameters, including origin and parent block size, are summarized in Table 11-9. Block splitting was used to a maximum of one split in each direction resulting in blocks no smaller than 10 m x 10 m x 5 m.

Table 11-9: Summary of Block Model Details

Direction (Axis)	Model Origin	No. of Blocks	Block Size (m)
Easting (X)	529,800	170	20
Northing (Y)	3,446,700	165	20
Elevation (Z)	665	87	10

Total and soluble Cu grade interpolation methods included Nearest Neighbor (NN), Inverse Distance Squared (ID^2) and Ordinary Kriging (OK), as summarized in Table 11-10. OK was only used for the units where there was sufficient data to support the completion of a variogram model, including Qwu, Qwt and Bx. Datamine RM dynamic search controls (Dynamic Anisotropy) were used to dynamically control the orientation of the search ellipse for each block based on the orientation of the contact between units Qwu and Qwt. The orientation of this contact surface was found to be representative of the mineralization in the alluvial units.

Table 11-10: Summary of Interpolation Methodology

Unit	Interpolation Methods	Dynamic Search Controls Used
Qfy	NN, ID^2	No
Qwu	NN, ID^2, OK	Yes
Qwt	NN, ID^2, OK	Yes
Qwl	NN, ID^2	Yes
PCGr	NN, ID^2	No
Bx	NN, ID^2, OK	No

The sample search strategy consisted of a 3-pass elliptical search where the 1st pass search radius was equal to approximately half the second structure variogram range, the 2nd pass search equal to the full variogram range and the 3rd pass search distance equal to twice the full variogram range. Estimates required a minimum of 8 samples in the 1st and 2nd passes and 4 in the 3rd pass with maximums of 12 samples for each pass having a maximum of 4 samples per drill hole. Search strategy parameters are summarized in Table 11-11.

Table 11-11: Summary of Search Strategy Parameters

Unit	1st Search					2nd Search			3rd Search			All
	X-Range	Y-Range	Z-Range	Min. Samples	Max. Samples	SVOL Factor 2	Min. Samples	Max. Samples	SVOL Factor 3	Min. Samples	Max. Samples	Max. per hole
Qfy	120	90	20	8	12	2	8	12	4	4	12	4
Qwu	120	90	20	8	12	2	8	12	4	4	12	4
Qwt	120	90	20	8	12	2	8	12	4	4	12	4
Qwl	120	90	20	8	12	2	8	12	4	4	12	4
PCGr	70	35	20	8	12	2	8	12	4	4	12	4
Bx	70	35	20	8	12	2	8	12	4	4	12	4

ID^2 estimates were chosen as the final estimated grades for units PCGr, Qfy, and Qwl, and OK estimates were chosen for Qwu, Qwt, and Bx as the final grades for total and soluble Cu.

11.1.7 Specific Gravity

This sub-section contains forward-looking information related to density for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including actual in situ characteristics that are different from the samples collected and tested to date, equipment and operational performance that yield different results from current test work results.

Specific Gravity (SG) data was analyzed using a box plot (Figure 11-8) and descriptive statistics (Table 11-12).

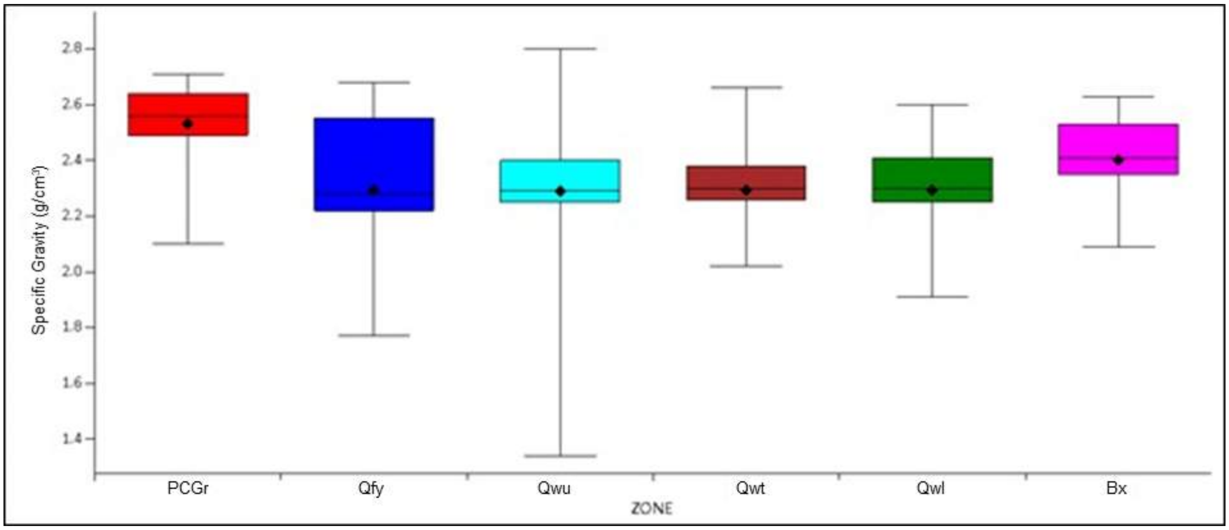


Figure 11-8: Box Plot of SG by Unit

Table 11-12: Summary Statistics of Raw SG Data by Unit

Variable	Unit	Count	Min	Max	Mean	Variance	StDev	CV
Specific Gravity		3,252	1.34	2.80	2.30	0.01	0.11	0.05
Specific Gravity	Qfy	775	1.77	2.68	2.29	0.02	0.12	0.05
Specific Gravity	Qwu	1,474	1.34	2.80	2.29	0.01	0.09	0.04
Specific Gravity	Qwt	467	2.02	2.66	2.29	0.00	0.06	0.03
Specific Gravity	Qwl	255	1.91	2.60	2.29	0.01	0.09	0.04
Specific Gravity	PCGr	92	2.10	2.71	2.53	0.01	0.11	0.04
Specific Gravity	Bx	189	2.09	2.63	2.40	0.01	0.09	0.04

SG was estimated using NN and ID^2 interpolation methods; however, due to an insufficient number of measurements in most units, the estimation of SG was not sufficient and therefore the declustered NN mean values for each unit were assigned to the models, as outlined in Table 11-13.

Table 11-13: Summary of Mean SG Values

Unit	Mean Value
Qfy	2.30
Qwu	2.28
Qwt	2.29
Qwl	2.30
PCGr	2.56
Bx	2.37

11.1.8 Model Validation

The model validation process included a visual comparison of block model and composite grades in plan and section, along with a global comparison of mean grades, evaluation of smoothing ratios and swath plots. Block grades were visually compared to the drill hole composite data in all domains, to ensure agreement and no material grade bias issues were identified, as demonstrated in Figure 11-9 and Figure 11-10.

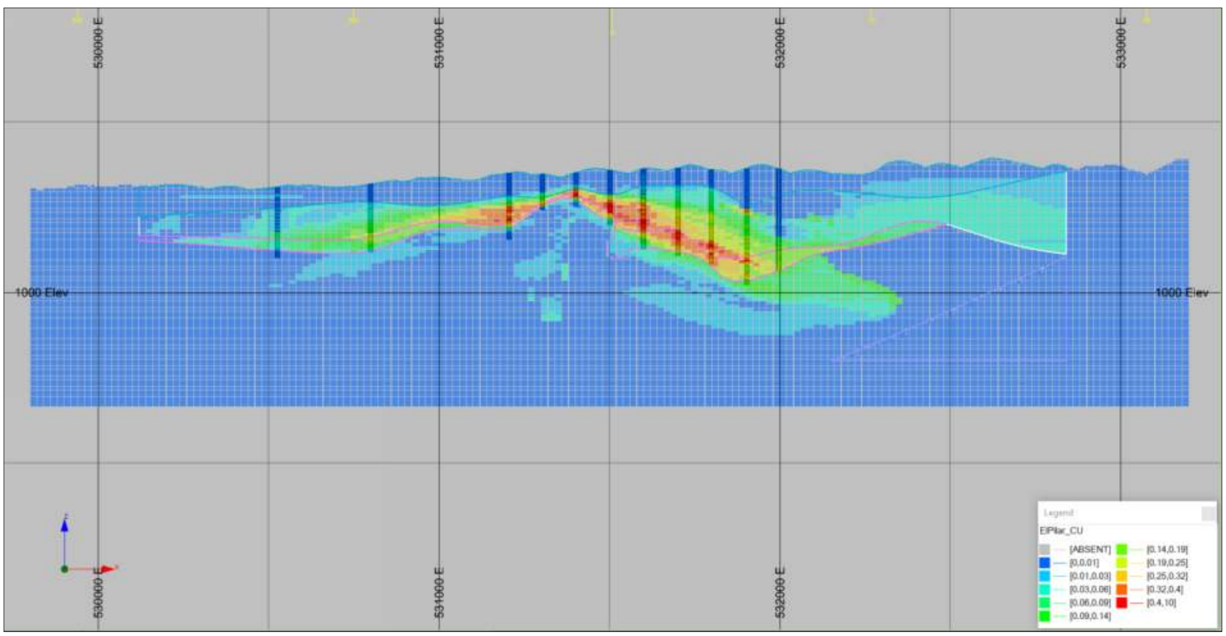


Figure 11-9: East-West Section Comparison of Composite Samples and Block Grades (3,448,400N)

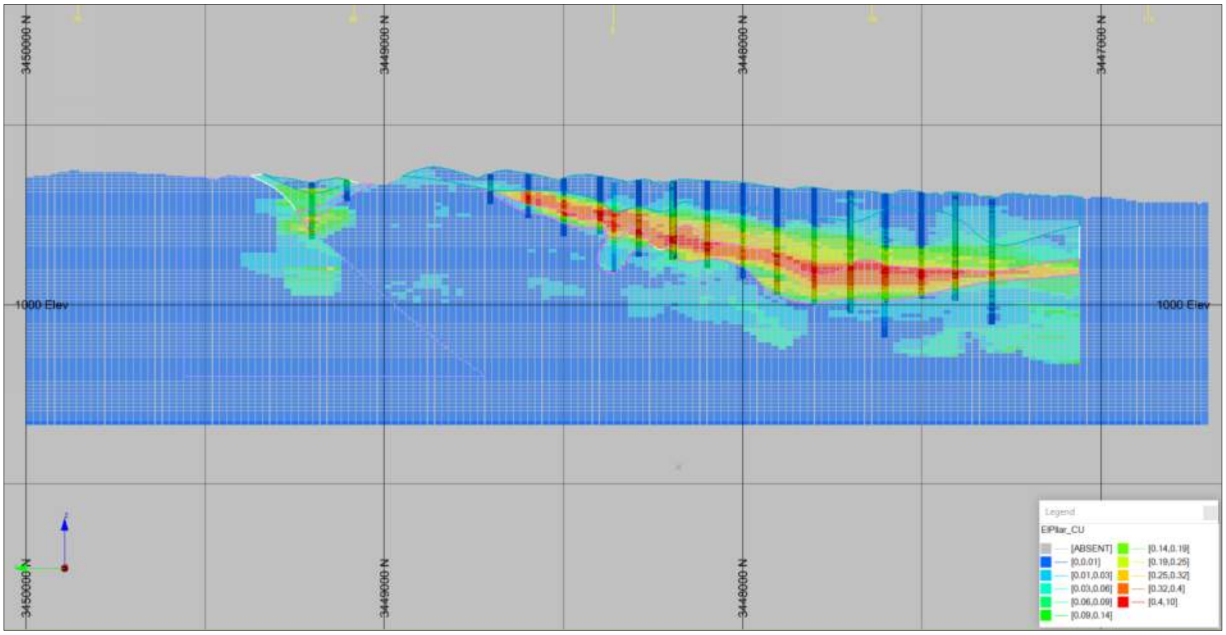


Figure 11-10: North-South Section Comparison of Composite Samples and Block Grades (531,500E)

Global mean grades for total Cu were compared between declustered composite grades from the NN estimates and the ID^2 and OK estimates to determine if there was and significant global bias. No significant global bias was identified in the grade estimates, as shown in Table 11-14.

Table 11-14: Comparison of Global Mean Total Cu Estimates

Zone	Variable	NN Mean (%)	ID2 Mean (%)	OK Mean (%)
Qfy	Total Cu	0.01	0.01	-
Qwu	Total Cu	0.08	0.08	0.08
Qwt	Total Cu	0.16	0.16	0.16
Qwl	Total Cu	0.02	0.01	0.01
PCGr	Total Cu	0.02	0.02	-
Bx	Total Cu	0.21	0.23	0.23

Grade estimates for units Qwu, Qwt, and Bx were evaluated for smoothing by calculating smoothing ratios which are based on the ratio between the theoretical model variance and actual model variance, where the theoretical variance is calculated based on the sum of the variance inside the block and variance between blocks using parameters, including the variogram model sill, block size, and F Function. A certain degree of smoothing is expected due to the change of support size from core sized samples to large mining blocks, ex., 20 m^3. It is common when using OK to see higher smoothing than expected, which can be an issue when reporting resources above a mining cut-off, as the overly smoothed distribution results in resource tonnages being overestimated and grades being underestimated. The smoothing ratios were reviewed and found to be better (i.e., Closer to 1) in the ID^2 estimates, as summarized in Table 11-15.

Table 11-15: Summary of Smoothing Ratios

Unit	Grade Variable	ID^2 Smoothing Ratio	OK Smoothing Ratio
Qwu	Total Cu	0.92	0.94
Qwu	Soluble Cu	1.10	1.12
Qwt	Total Cu	1.01	1.07
Qwt	Soluble Cu	1.25	1.37
Bx	Total Cu	1.23	1.47
Bx	Soluble Cu	1.32	1.71

Swath plots were generated for Total Cu within units Qwu, Qwt, Bx to evaluate local grade comparisons between NN, ID^2 and OK estimates. Example swath plots for the Qwt unit are presented in Figure 11-11, Figure 11-12, and Figure 11-13. In general, there was reasonable correlation between all estimates, with ID^2 estimates being closer the declustered composite values represented by the NN estimates. Some divergence between swath lines were observed around the margins of the model but in general no material issues were identified.

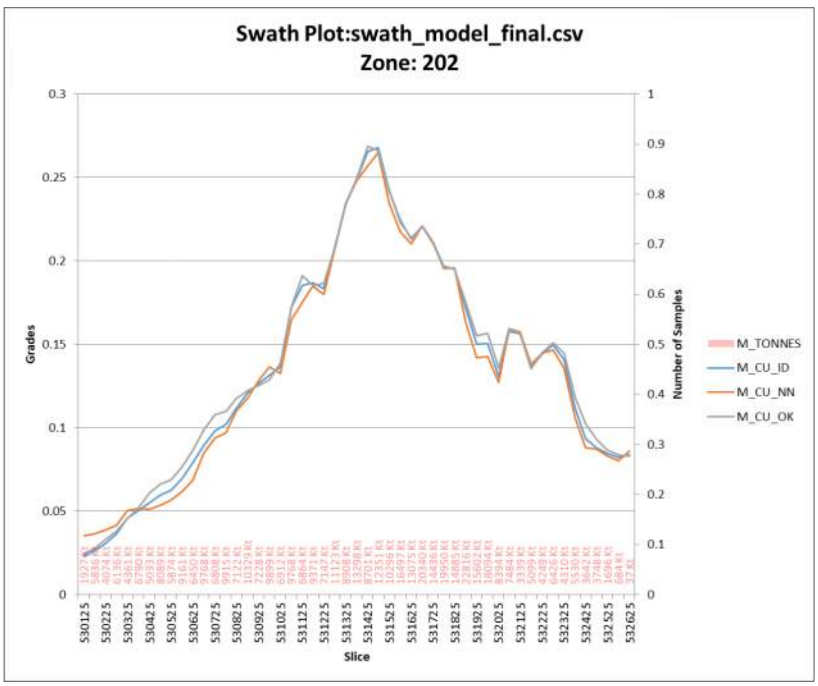


Figure 11-11: North-South Total Cu Swath Plot for Unit Qwt

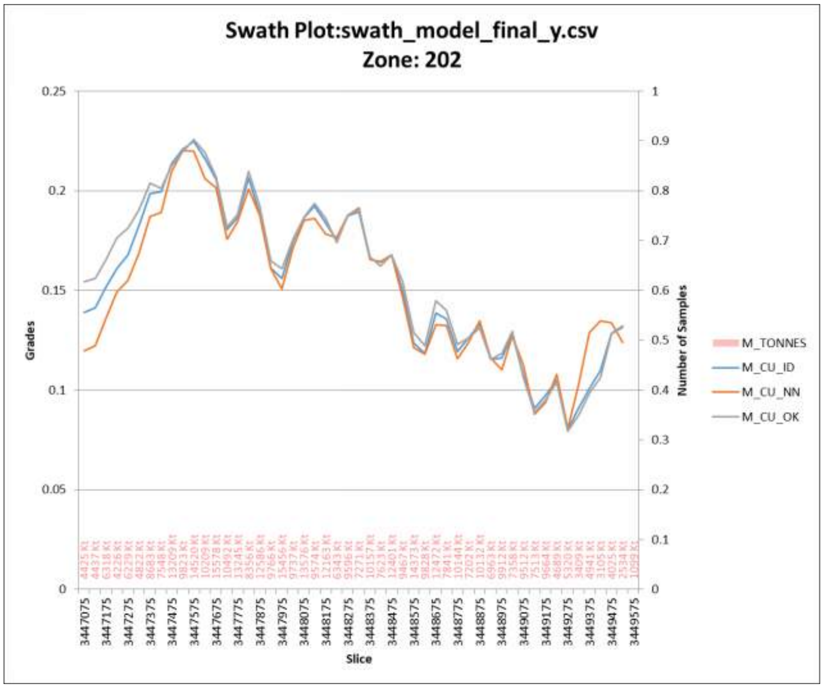


Figure 11-12: East-West Total Cu Swath Plot for Unit Qwt

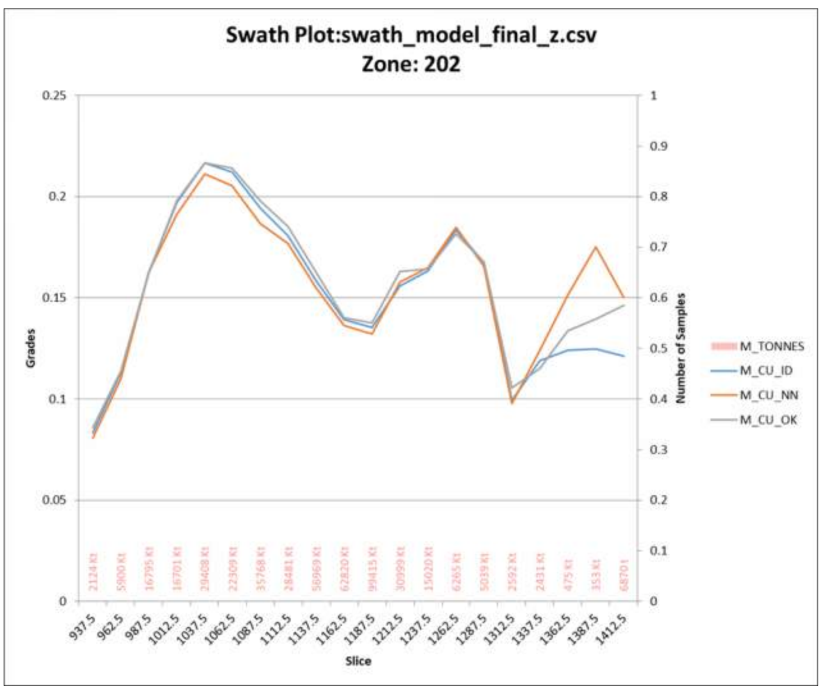


Figure 11-13: Plan View Total Cu Swath Plot for Unit Qwt

11.2 MINERAL RESOURCE ESTIMATE

This sub-section contains forward-looking information related to Mineral Resource estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including geological and grade interpretations and controls and assumptions and forecasts associated with establishing the prospects for economic extraction.

The Mineral Resource estimate for the project is reported here in accordance with the SEC S-K 1300 regulations. For estimating the Mineral Resources of El Pilar, the following definition as set forth in the S-K 1300 Definition Standards adopted December 26, 2018 was applied.

Under S-K 1300, a Mineral Resource is defined as:

> "…is a concentration or occurrence of material of economic interest in or on the Earth's crust in such form, grade or quality, and quantity that there are reasonable prospects for economic extraction. A mineral resource is a reasonable estimate of mineralization, taking into account relevant factors such as cut-off grade, likely mining dimensions, location or continuity, that, with the assumed and justifiable technical and economic conditions, is likely to, in whole or in part, become economically extractable. It is not merely an inventory of all mineralization drilled or sampled."

Note to readers: The Mineral Resources presented in this section are not Mineral Reserves and do not reflect demonstrated economic viability. The reported 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. There is no certainty that all or any part of this Mineral Resource will be converted into Mineral Reserve. All figures are rounded to reflect the relative accuracy of the estimates and totals may not add correctly.

Mineral Resource estimates exclusive of Mineral Reserves are summarized in Table 11-16. Mineral Resources presented in the table are in accordance with the definitions presented in S-K 1300. The effective date of the Mineral Resource estimate is December 31, 2021.

Table 11-16: Mineral Resource Estimates Exclusive of Mineral Reserves

Classification	Tonnes (Mt)	Total Cu (%)	Soluble Cu (%)
Measured	2.2	0.20	0.10
Indicated	81.3	0.18	0.08
Total Measured and Indicated	**83.4**	**0.18**	**0.08**
Inferred	88.6	0.12	0.06

Notes:

- The Mineral Resource estimates were prepared by Ronald Turner, P.Geo . (who is the independent Qualified Person for these Mineral Resource estimates), reported using the S-K 1300 Definition Standards adopted December 26, 2018
- Tonnages are rounded to the nearest 100,000 tonnes.
- Resources are reported on a break-even profit basis and constrained within a pit shell outlined using a Cu price assumption of $3.795 / lb.

11.3 BASIS FOR ESTABLISHING THE PROSPECTS OF ECONOMIC EXTRACTION FOR MINERAL RESOURCES

This sub-section contains forward-looking information related to establishing the prospects of economic extraction for Mineral Resources for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including COG assumptions, costing forecasts and product pricing forecasts. Heap leach recoveries for the El Pilar Project were determined on a block-by-block basis using a regression equation established by M3 and clarified in a document issued to the registrant. The regression equations is stated as follows:

$$Cu\ Recovery\ = 0.3349 \times LN\left(\frac{Soluble\ Cu}{Total\ Cu}\right) + 0.7949$$

Additional description of the lab testing, and analyses can be found in Section 8, 10, 12 and 14.

A Cu price of US$3.30 per pound (/lb) was used for estimating Mineral Reserves while a 15% higher price of US$3.795/lb was used when estimating Mineral Resources. These price assumptions were provided by SCC.

The Mineral Resource estimate was reported based on a block profit calculation where blocks having a profit greater than or equal to zero (break-even) were reported from within a constrained pit shell developed using the following criteria to establish reasonable prospects for economic extraction.

- Cu price assumption = US$ 3.795 / lb
- Mining cost = US$ 1.34 / t
- Processing cost = US$ 0.57 / t
- Selling Cost = $0.68 / lb
- Recovery was computed from the regression equation developed by M3 Engineering and described earlier in this section

Mining and selling costs were estimated from previous studies conducted on the El Pilar projected and were deemed to be reasonable based on general experience with other operations. The selling cost includes estimates for the solvent extraction and electrowinning (SX-EW), cathode transport, general administration, and royalty costs. Processing costs were provided by M3.

11.4 MINERAL RESOURCE CLASSIFICATION

This sub-section contains forward-looking information related to Mineral Resource classification for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including geological and grade continuity analysis and assumptions.

According to the S-K 1300 regulations, to reflect geological confidence, Mineral Resources are subdivided into the following categories based on increased geological confidence: Inferred, Indicated, and Measured, which are defined under S-K 1300 as:

> "Inferred Mineral Resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. The level of geological uncertainty associated with an inferred mineral resource is too high to apply relevant technical and economic factors likely to influence the prospects of economic extraction in a manner useful for evaluation of economic viability. Because an inferred mineral resource has the lowest level of geological confidence of all mineral resources, which prevents the application of the modifying factors in a manner useful for evaluation of economic viability, an inferred mineral resource may not be considered when assessing the economic viability of a mining project and may not be converted to a mineral reserve."

> "Indicated Mineral Resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of adequate geological evidence and sampling. The level of geological certainty associated with an indicated mineral resource is sufficient to allow a QP to apply modifying

factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Because an indicated mineral resource has a lower level of confidence than the level of confidence of a measured mineral resource, an indicated mineral resource may only be converted to a probable mineral reserve."

"Measured Mineral Resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of conclusive geological evidence and sampling. The level of geological certainty associated with a measured mineral resource is sufficient to allow a QP to apply modifying factors, as defined in this section, in sufficient detail to support detailed mine planning and final evaluation of the economic viability of the deposit. Because a measured mineral resource has a higher level of confidence than the level of confidence of either an indicated mineral resource or an inferred mineral resource, a measured mineral resource may be converted to a proven mineral reserve or to a probable mineral reserve."

Mineral Resource categories were assigned to broad regions of the block model based on the confidence related to drill hole density, geological understanding, continuity of mineralization relative to the style of mineralization, and data quality. A combination of drill hole density and the estimation pass used to estimate the grade of the block were used as a guide for outlining classification regions. Areas where the average drill hole spacing was 60 m and blocks were estimated in the first pass were classified as "Measured Mineral Resource." Areas where the average drill hole spacing was 100 m and most blocks were estimated in the first or second pass were classified as "Indicated Mineral Resource" and areas where the drill hole spacing was greater than 100 m and the blocks were estimated in the second or third pass were classified as "Inferred Mineral Resource." Figure 11-14 outlines the Mineral Resource categories assigned to the block model within the mineralized units. Areas outside of the Measured, Indicated, and Inferred categories, including all of the Qwl unit were coded as Potential and were not considered as Mineral Resources. Since the majority of mineralization occurs in the upper and transitional zones of Qwu and Qwt, the drillholes did not consistently penetrate into the lower Qwl and were, in the QP's opinion, too sparse to support categorization as a Mineral Resource.

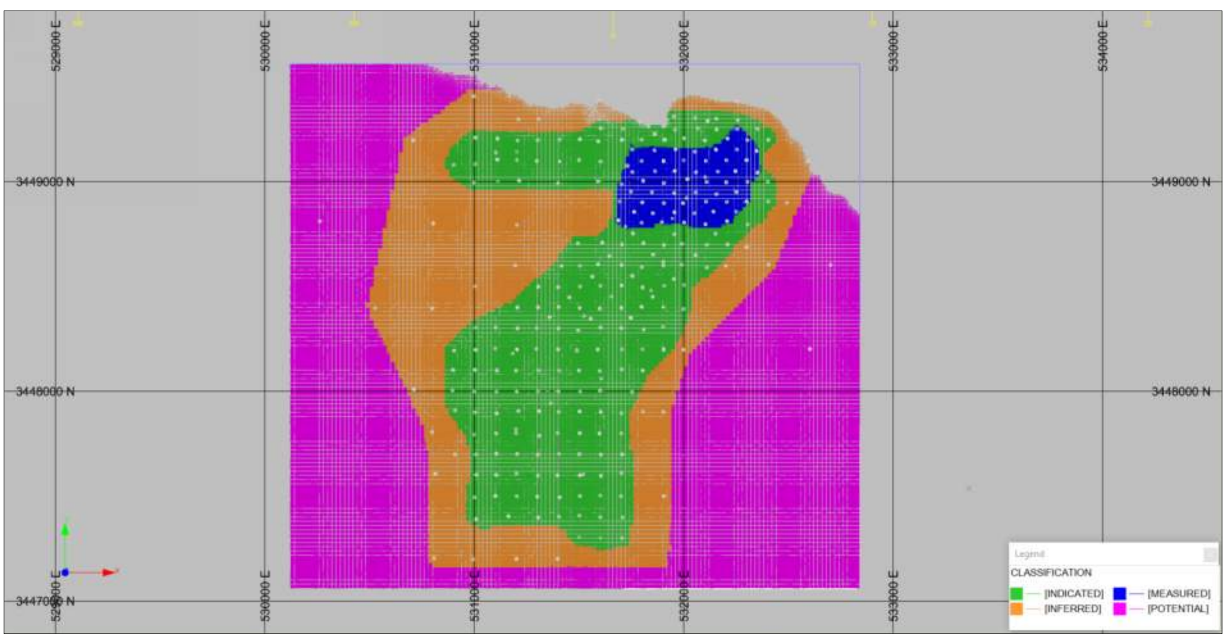


Figure 11-14: Plan View of Mineral Resource Classification Assigned to the Block Model

11.5 MINERAL RESOURCE UNCERTAINTY DISCUSSION

Mineral Resources are not Mineral Reserves and do not necessarily demonstrate economic viability. There is no certainty that all or any part of this Mineral Resource will be converted into Mineral Reserve.

Inferred Mineral Resources are too speculative geologically to have economic considerations applied to them to enable them to be categorized as Mineral Reserves.

Mineral Resource estimates may be materially affected by the quality of data, natural geological variability of mineralization and / or metallurgical recovery and the accuracy of the economic assumptions supporting reasonable prospects for economic extraction including metal prices, and mining and processing costs.

The quality of the data was found to be generally acceptable for the purpose of resource estimation but some of the older data was found to be not as strongly supported using current best practice guidelines for Quality Assurance and Quality Control (QA/QC). This doesn't necessarily mean that the data is inaccurate but does increase the level of uncertainty regarding the quality of the data. The potential uncertainty surrounding this data is mitigated by the fact that Stingray completed a large infill drill program resulting in a much larger data set that mitigates the risk of any potential impact from the older and much smaller data set.

Much of the Indicated Mineral Resources are defined based on 100-m drill spacing and subsequent infill drilling could potentially identify areas of lower grade in between existing holes due to natural geological variability or previously unknown geological structures, such as offsetting faults or intrusive dykes. It is the QP's opinion this has a relatively low likelihood of happening due to the geological grade continuity observed in the drill hole data. These risks can be reasonably mitigated by further infill drilling to a drill spacing of approximately 50 m, which is consistent with the drill density defined for Measured resources.

Mineral Resources may also be affected by the estimation methodology and parameters and assumptions used in the grade estimation process including top-cutting (capping) of data or search and estimation strategies although it is the QP's opinion that there is a low likelihood of this having a material impact on the Mineral Resource estimate.

11.6 ASSUMPTIONS FOR MULTIPLE COMMODITY MINERAL RESOURCE ESTIMATE

Not applicable to this TRS as no metal/mineral equivalents are being used or reported.

11.7 QUALIFIED PERSON'S OPINION ON FACTORS THAT ARE LIKELY TO INFLUENCE THE PROSPECT OF ECONOMIC EXTRACTION

It is the QP's opinion that the Mineral Resource block model is representative of the informing data and that the data is of sufficient quality to support the 2021 Mineral Resource Estimate.

The 2021 Mineral Resource Estimate may be materially impacted by any future changes in the break-even cut-off grade, potentially resulting from changes in mining costs, processing recoveries, or metal prices or from changes in geological knowledge as a result of new exploration data.

12 MINERAL RESERVE ESTIMATES

12.1 KEY ASSUMPTIONS, PARAMETERS, AND METHODS

This sub-section contains forward-looking information related to the key assumptions, parameters and methods for the Mineral Reserve estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including Mineral Resource model tonnes and grade and mine design parameters.

12.1.1 Geological Resource Model

The geological block model previously described in Section 11 and used to estimate Mineral Resources was the basis for the estimate of Mineral Reserves. The geological block model is based on Reverse Circulation (RC) and core drilling from 1998 to 2016. A Mineral Resource pit was developed to define and limit the estimation of Mineral Resources to the "reasonable prospects for economic extraction."

The dimensions and principal variables of the geological block model are shown in Table 12-1 and Table 12-2, respectively. The lithological units of the El Pilar block model include 7 rock types with 6 rock types, as shown in Table 12-3.

Table 12-1: Dimensions of the Block Model

Block Dimension	Block Size (m)	Origin (m)	Block Model Extents		No. of Blocks	Sub-block Size (m)
			Minimum (m)	Maximum (m)		
X	20	0	529,800	533,200	170	10
Y	20	0	3,446,700	3,450,000	165	10
Z	10	0	665	1535	87	5

Table 12-2: Principal Variables of the Block Model

Variable	Description
Cu	Total Cu
Cu_sol	Soluble Cu
Cu_ns	Insoluble Cu
Cu_svol	Soluble Volume
Cu_ratio	Solubility Index
Mineral	Mineralized Unit Identifier
Class	Resource Classification
Density	Specific Gravity
Tonnes	Tonnage

Table 12-3: Block Model Estimation Domains

Rock	Code	Description
Qfy	103	Quaternary Younger Alluvial Fan Deposits
Qwu	201	Quaternary Alluvial Wash Deposits Upper
Qwt	202	Quaternary Alluvial Wash Deposits Transitional
Qwl	203	Quaternary Alluvial Wash Deposits Lower
PCGr	102	Precambrian Granitic Intrusive Rocks
Bx	205	Breccia

12.1.2 Mine Design Criteria

The general mine design criteria used to estimate the Mineral Reserves are listed below:

1. Surface open-pit mining approach
2. Typical bench height of 10 m
3. Typical bench width 8 m in overburden, 5 m elsewhere
4. Overall pit slope criteria: 35° in the Qfy (Zone 103) and 47° in all other zones
5. Haul road design width of 30 m
6. Maximum ramp grade of 10%

The potential mining area was limited to the permitted area.

A pit shell optimization exercise, described in Section 12.2.6, was completed and used as the basis for the pit phase designs. The pit phases were designed within the permit boundary using the following pit parameters:

- Loss and dilution based on historical information from similar deposits (see Section 12.2.1)
- General mine design criteria listed above.
- Geotechnical parameters as described in Section 13.2.1.
- Process recovery methodology and factors described in Section 12.2.2.

Using the designs and the parameters mentioned above, an ultimate mining pit design was developed, the potential reserves were estimated within the ultimate pit, and an economic analysis was performed (see Section 19).

The point of reference of the Mineral Reserves estimate is Run of Mine (ROM) ore delivered to the heap leach pad. All Mineral Reserves are as of December 31, 2021.

12.2 MODIFYING FACTORS

This sub-section contains forward-looking information related to the modifying factors for the Mineral Reserve estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including modifying factors including dilution and mining and recovery factors, beneficiation assumptions, property limits, commodity price, cut off grades, pit optimization assumptions and the ultimate pit design. Modifying factors are applied to mineralized material within the Measured and Indicated resource classifications to establish the economic viability of Mineral Reserves. A summary of modifying factors applied to the El Pilar mine Mineral Reserve estimate is provided below.

12.2.1 Dilution, Loss, and Mine Recovery

Modifying factors have been applied to the geologic model to simulate the effects of extracting the Mineral Resource and help establish the economic viability of Mineral Reserves. Dilution in mining can be defined as the addition of waste material to the ore during the mining process and is due to a lack of selectivity, or in some cases, due to inadequate operational configuration. The dilution effects result in a reduction of the in-situ Cu grade for the mining model as well as a reduction in mass recovery. The factors that cause dilution are diverse and include:

- Nature of ore contacts and boundaries
- Pit boundary zones
- Mining block size and position
- Density estimate
- Geological complexity
- Selectivity of mining and equipment size
- Mining method

Dilution can be internal (caused by intrinsic deposit factors) or external (caused by operational factors). Dilution cannot be fully eliminated as it is impossible to define the exact accuracy of the mining limits; however, it can be estimated and considered, thus minimizing the differences between the mine plan and the actual operations. A dilution of 3% by mass with 0% copper grade was included in the Mineral Reserve estimate. The mining recovery was estimated to be 98%.

12.2.2 Processing

All ROM material mined is assumed to be leached. Leach recovery was incorporated into the model on a block-by-block basis, using the equation:

$$Cu\ Recovery = 0.3349 \times LN(Cu\_Ratio) + 0.7949$$

This equation was provided by M3 Engineering. Details of the recovery assumptions are provided in Section 10 and Section 14.

12.2.3 Property Limits

The ultimate pit design was not impacted by any property limits. The land required for stockpiles and infrastructure is sufficient for the ultimate pit design.

12.2.4 Commodity Price Used

A Cu price of US$3.30/lb was used for estimating Mineral Reserves while a 15% higher price (US$3.795/lb) was used when estimating Mineral Resources. These price assumptions were provided by SCC.

12.2.5 Cut-off Grade Estimate

Per the definitions is S-K Subpart 1300, "For the purposes of establishing 'prospects of economic extraction', the COG is the grade that distinguishes material deemed to have no economic value from material deemed to have economic value." In simpler terms, the COG is the grade at which revenue generated by a block is equal to its total cost resulting in a net value of $0.

To comply with Regulation S-K Subpart 1300, a COG that defines the minimum grade of material that must be achieved to economically process material must be defined. COGs are defined by geometallurgical, processing, and economic criteria. For material to be sent to the ROM Leach Pad at El Pilar, it must be within mineralized Zones Qwu, Qwt, or Bx (Zone 103, 201, 202, or 205) and have a high enough recovery such that the Cu recovered from leaching generates enough revenue (at an assumed selling price of $3.30/lb Cu) to breakeven with the costs of mining and processing the ore and selling the resultant Cu Cathode generated from the SX-EW Plant.

The economic assumptions used to determine the estimated COGs are shown in Table 12-4.

Table 12-4: Operating Costs Used for Reserves and Economics

Description	Units	Price
Mining Cost		
Ore	$/t	1.34
Waste	$/t	1.34
Processing Cost[1]	$/t	0.57
Selling Costs[2]		
SX-EW	$/lb Cu	0.54
Selling & Transport	$/lb Cu	0.03
G&A Mexico	$/lb Cu	0.08
Royalty	$/lb Cu	0.03
Total	$/lb Cu	0.68
Selling Price	$/lb Cu	3.30

1. The Processing Cost includes G&A Mexico costs
2. Total Selling Costs include SX-EW, Cathode Transport, G&A, and Royalty Costs

Measured and Indicated blocks within the model with a value greater than or equal to zero were considered ore. The value equation incorporates the copper recovery as well as copper grade. The formula is shown below:

$$Value = Revenue \; - Selling\ Cost - Processing\ Cost$$

$$Revenue = \frac{\$3.3}{lb\ Cu} * \frac{2204.6\ lb}{tonne} * \frac{Cu\ Recvoery\ (\%)}{100} * \frac{Cu\ grade\ (\%)}{100}$$

$$Revenue = \frac{\$0.73 * Cu\ Recovery * Cu\ Grade}{tonne}$$

$$Selling\ Cost = \frac{\$0.68}{lb\ CU} * \frac{2204.6\ lb}{tonne} * \frac{Cu\ recovery\ (\%)}{100} * \frac{Cu\ Grade\ (\%)}{100}$$

$$Selling\ Cost = \frac{\$0.15 * Cu\ Recovery * Cu\ Grade}{tonne}$$

$$Processing\ Cost = \frac{\$0.57}{tonne}$$

$$Value = \frac{\$0.73 * Cu\ Recovery * Cu\ Grade}{tonne} - \frac{\$0.15 * Cu\ Recovery * Cu\ Grade}{tonne} - \$0.57/tonne$$

12.2.6 Pit Optimization Methodology and Ultimate Pit Limits

Standard pit optimization methodology using Whittle 4X software was completed to determine the extent of economically mineable reserves at El Pilar. A nested pit analysis was performed on the geologic model. Mining loss and dilution adjustments were included in the pit optimization analysis. The discounted cashflow was estimated using an 8% discount rate. Based on this nested pit analysis, Pit 30 was chosen with a Revenue Factor of 0.88 (Cu selling price of $2.90/lb) as the basis of the ultimate pit design described in Section 12.2.7. The results of the nested pit analysis are shown in Figure 12-1.

The discounted cashflows shown in Figure 12-1 represent:

- Best case - assumes pit shells can be mined entirely without consideration of vertical advance or other operational constraints.
- Worst case - assumes each pit shell is mined bench by bench, with a yearly mining constraint of 55 Mt and a yearly leach constraint of 19 Mt. These mining rates were assumed based on an analysis of previous studies completed on the Project.
- Specified case - assumes selected pushbacks and an eight-bench maximum vertical advance prior to advancing to the next selected phase. This case is a first pass production schedule using pit shells as phases

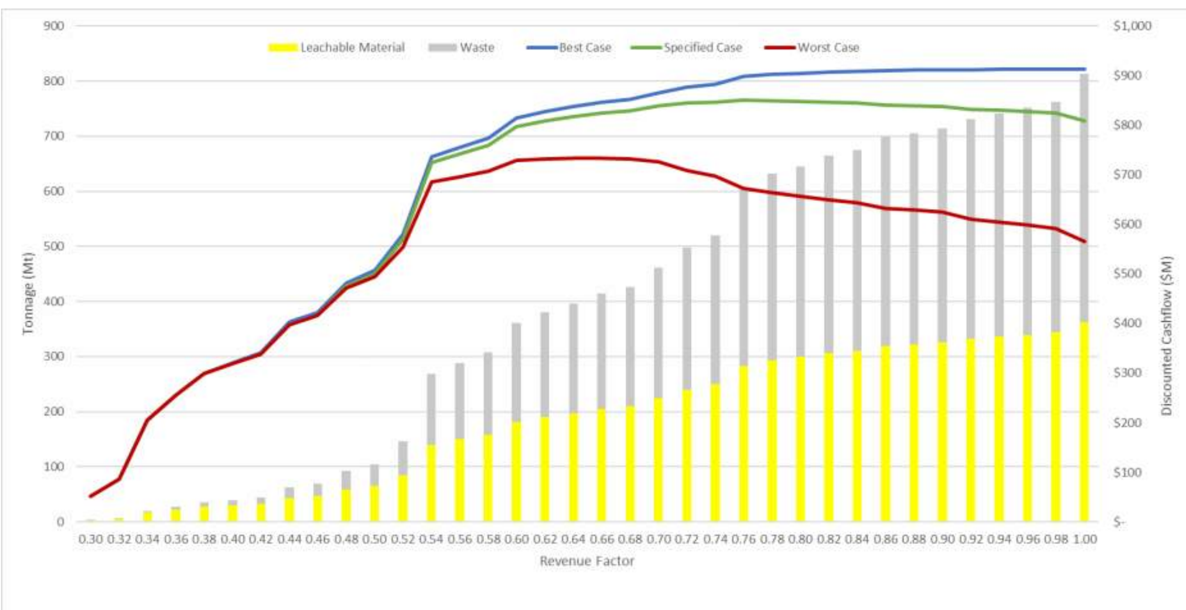


Figure 12-1: Pit Optimization Results

12.2.7 Ultimate Pit Design

The ultimate pit design considers geotechnical and hydrological factors that are described in Section 13. A map showing the design and extents of the ultimate pit and the topographic intercept of the pit shell (Revenue Factor 0.88) is provided as Figure 12-2.The ultimate pit, overburden storage facilities (OSFs), and leach pad is shown in Figure 12-3.

The ultimate design has approximately 1% less leach material and approximately 0.5% more total material compared to the Revenue Factor 0.88 pit shell. This additional material in the design is the result of adding access ramps and considering minimum mining widths.

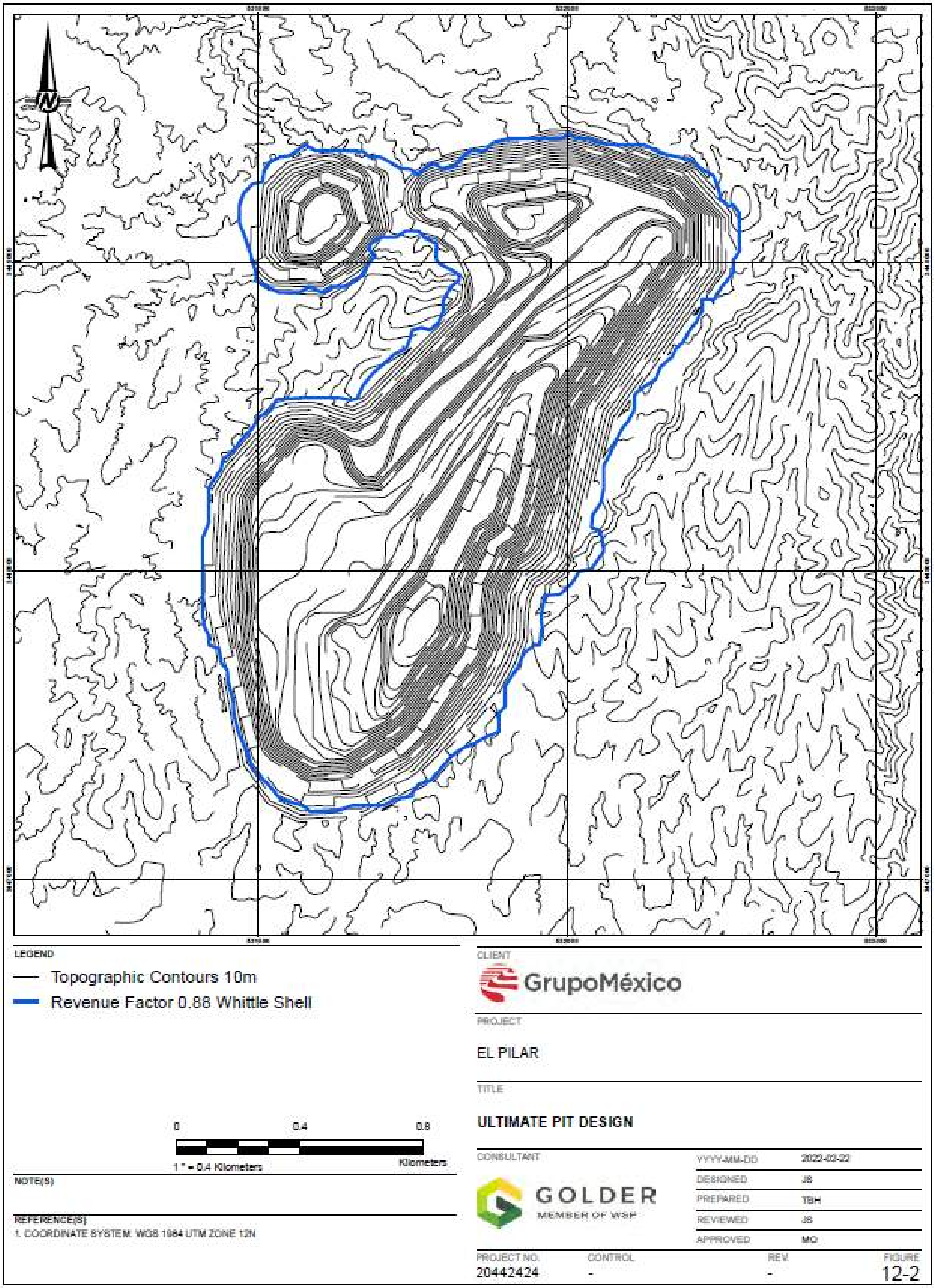


Figure 12-2: Ultimate Pit Design

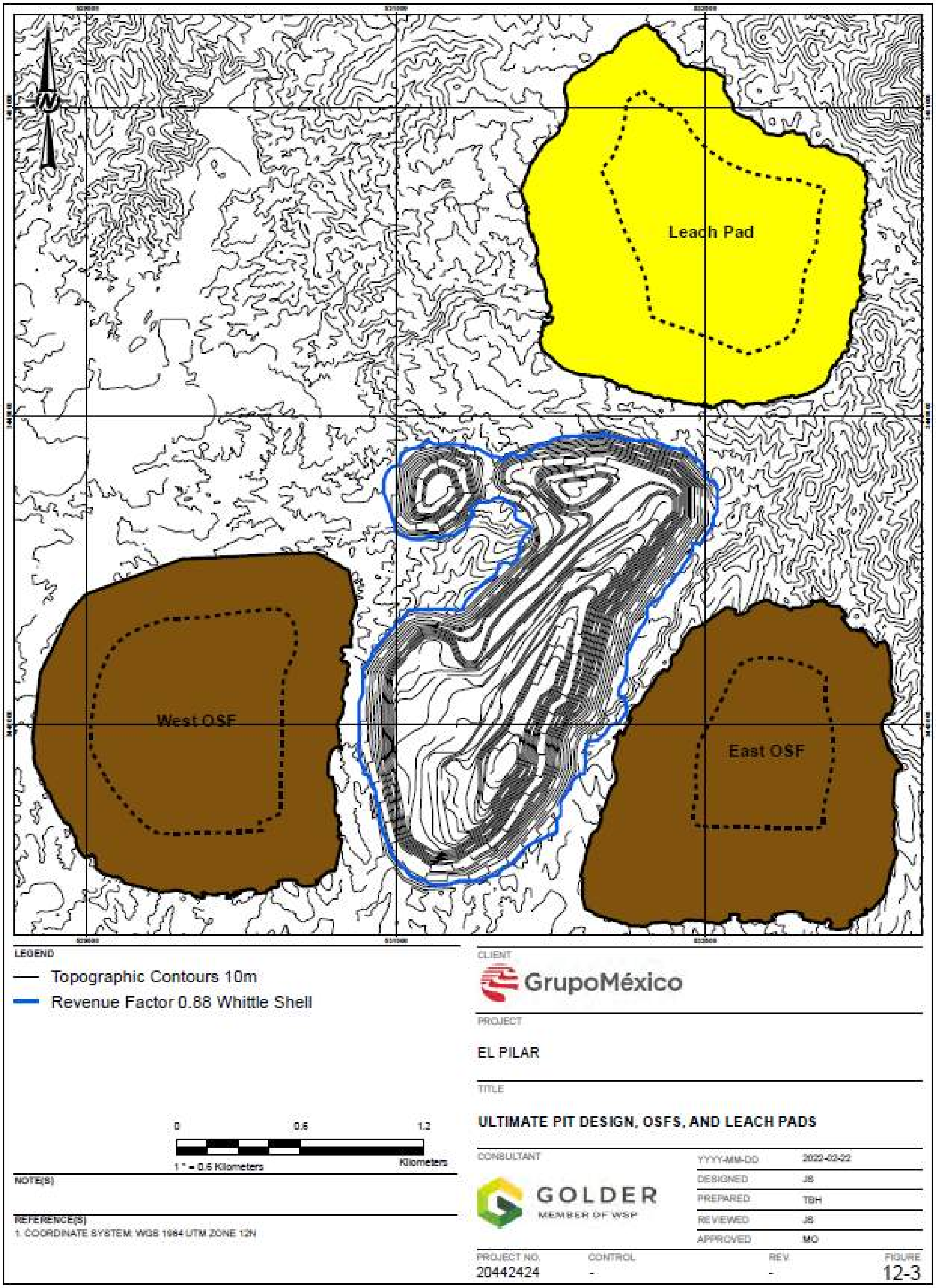


Figure 12-3: Ultimate Pit Design, OSFs, and Leach Pad

12.3 MINERAL RESERVE CLASSIFICATION

This sub-section contains forward-looking information related to the Mineral Reserve classification for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including Mineral Resource model tonnes, grade, and classification. For estimating the Mineral Reserves for El Pilar, the following definition as set forth in the S-K 1300 Definition Standards adopted December 26, 2018, was applied.

Under S-K 1300, a Mineral Reserve is defined as:

> "… an estimate of tonnage and grade or quality of indicated and measured mineral resources that, in the opinion of the qualified person, can be the basis of an economically viable project. More specifically, it is the economically mineable part of a measured or indicated mineral resource, which includes diluting materials and allowances for losses that may occur when the material is mined or extracted."

Mineral Reserves are subdivided into classes of Proven Mineral Reserves and Probable Mineral Reserves, which correspond to Measured and Indicated Mineral Resources, respectively, with the level of confidence reducing with each class. Mineral Reserves are always reported as the economically mineable portion of a Measured and/or Indicated Mineral Resource, and take into consideration the mining, processing, metallurgical, economic, marketing, legal, environmental, infrastructure, social, and governmental factors (the "Modifying Factors") that may be applicable to the deposit.

12.4 MINERAL RESERVE ESTIMATE

This sub-section contains forward-looking information related to Mineral Reserve estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including Mineral Resource model tonnes and grade, modifying factors including mining and recovery factors, production rate and schedule, mining equipment productivity, commodity market and prices and projected operating and capital costs.

Based on the mining boundaries and modifying factors discussed above, the recovery factors discussed in Section 12.2.1 and the Economic Assessment discussed in Section 19, the El Pilar Project contains the economically mineable Mineral Reserves listed in Table 12-5. The Mineral Reserves include approximately 317 Mt of ROM ore with a Cu grade of 0.249% total Cu for 940 million pounds (Mlbs) of recovered Cu. The point of reference for Mineral Reserves is as delivered to the leach pad. Total material in the pit is 702 Mt, resulting in a waste to ore ratio of 1.21. The schedule is based on routing all blocks that pay for full processing and selling costs to the leach pad.

For this Mineral Reserve estimate, Measured Mineral Resources inside the ultimate pit were converted to Proven Mineral Reserve and Indicated Mineral Resources inside the ultimate pit were converted to Probable Mineral Reserves. The Mineral Reserves are estimated at a constant copper price of $3.30 per pound (provided by SCC). Mineral Reserves are reported effective December 31, 2021.

Table 12-5: El Pilar Mineral Reserve Estimate as of December 31, 2021

Classification	ROM Ore Tonnes (Mt)	Total Cu (%)	Cu Recovery	Contained Cu (kt)	Recovered Cu (Mlbs)
Proven	63	0.266	60%	168	221
Probable	254	0.245	52%	623	720
Proven and Probable	317	0.249	54%	790	941

Note:

1. Mineral Reserves are mined tonnes and grade; the reference point is the leach pad and includes considerations for operational modifying factors such as loss (2%) and dilution (3%).
2. The recovered copper estimate utilizes the recovery discussed in Section 12.2.2 (Cu Rec % = 0.3349 x LN(Cu_Ratio) + 0.7949).
3. Numbers have been rounded to reflect appropriate accuracy

12.5 QUALIFIED PERSON'S OPINION ON RISK FACTORS THAT COULD MATERIALLY AFFECT THE RESERVE ESTIMATES

The QP is unaware of any mining, metallurgical, infrastructure, or other factors that might materially affect the Mineral Reserve. Based on the scenarios Golder evaluated the Project is sensitive to price, costs, and leach recovery assumptions.

13 MINING METHODS

The El Pilar mine is in the pre-production phase of mining and has not yet begun full operation. The geological and mining knowledge is based on the collective experience of personnel from Southern Copper site operations, geology, mining, metallurgy, and other technical disciplines gained during years of metals mining in Mexico. This knowledge is supported by production data and observations from other Southern Copper mining operations in Mexico.

Mining of the El Pilar copper deposit will be accomplished by conventional open pit methods, with blasting of ore and waste along with shovel and truck (shovel/truck) loading and hauling. Waste material will be routed to one of two overburden storage facilities (OSF) and ore will be routed to the leach pad. In the development phase, drainage and water control will be established, and then the required infrastructure consisting of power, pipelines, and roadways will be established.

13.1 PRODUCTION TASKS

Mining operations progress in a multi-step process, which includes on-going top-soil removal and pre-production work, drilling and blasting, overburden removal and OSF maintenance, ore production, and reclamation.

13.1.1 Pre-Production

Surface areas to be disturbed during the mining process are progressively cleared of vegetation using dozers, as necessary. Topsoil is removed and stored for future reclamation work. The pre-production phase of the Project was in progress at the time the QP visited the site. The main pit area was being pre-stripped of overburden and some of the earthworks were commencing

13.1.2 Drilling & Blasting

The primary drilling fleet consists of diesel-powered drills with a pull-down force of about 60,000 pounds or about 27,300 kg drill. Material will be drilled with 229 mm diameter holes on 10 m mining benches.

Shift productivities are estimated at 27,500 tonnes for ore and waste, and 25,900 tonnes for overburden. Annual production per drill is estimated at 20.3 million tonnes (Mt) for ore and waste, and 19.1 Mt for overburden.

The main explosive planned for use is ammonium nitrate/fuel oil (ANFO). The blast design for both ore and waste includes an approximately 8 m burden and 8 m spacing. A 9” hole diameter will be used, with a 10 m bench height.

The productivity calculations are also based on a powder factor of 0.3 kilograms per cubic meter (kg/m^3) for ore and the waste. Drill penetration was assumed as 35 m/hr for ore and waste. The powder factor and penetration assumptions are preliminary estimates based on the Uniaxial Compressive Strength (UCS) for the expected site conditions.

A maximum of 3 drills will be required at peak production.

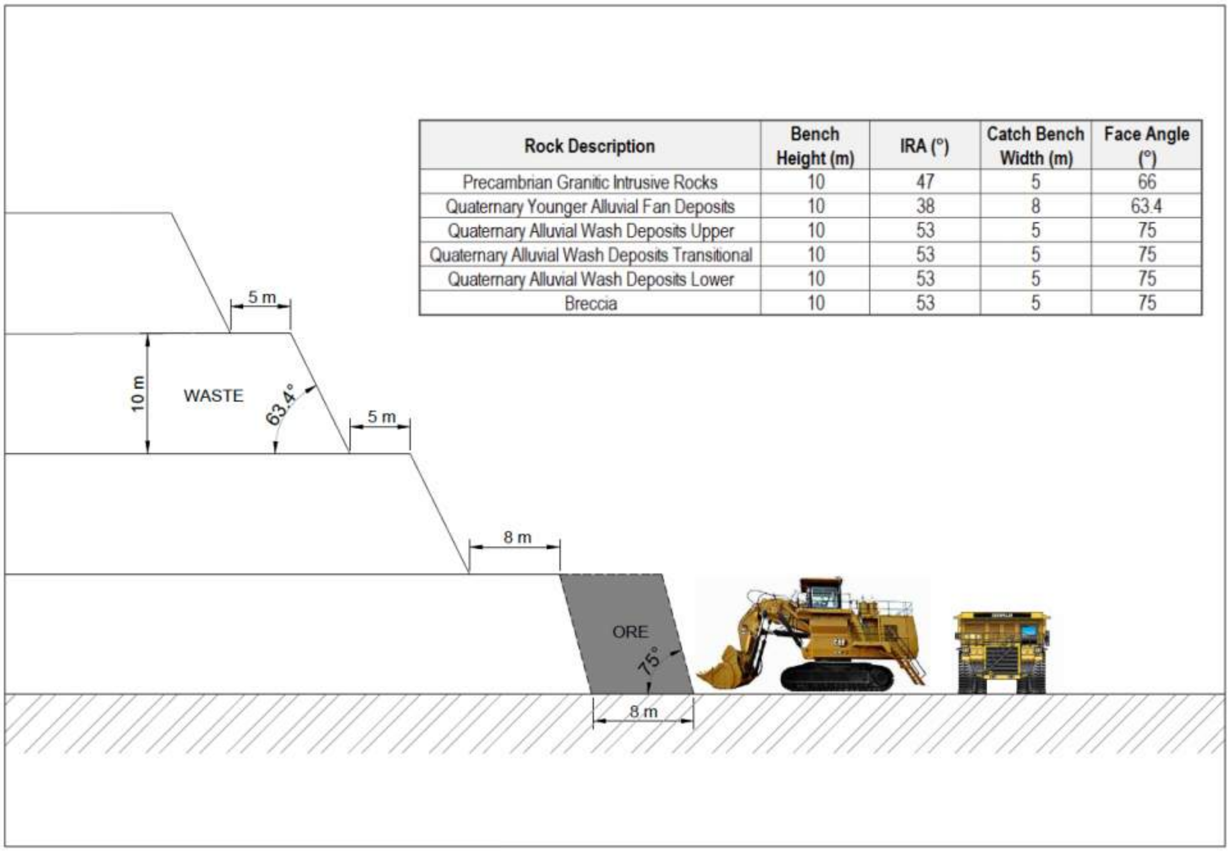
13.1.3 Overburden Removal and Waste Dump Maintenance

Overburden and waste rock will be loaded by either hydraulic shovels with a 34.0 m^3 bucket or Front-End Loaders with a 20.0 m^3 bucket into trucks with a nominal capacity of approximately 181 metric tonnes. Dozers will assist the loading fleet with general clean-up and material removal, as necessary. Overburden and waste material is hauled to one of two ex-pit OSF and dozers are used to push overburden down the sides of the dump on an as-needed basis. Total waste haulage routes using mine access ramps will vary over the life of the operation, but generally range from about 2 km to 12 km.

13.1.4 Ore Production

Ore loading operations will use similar equipment, with 34 m^3 bucket class shovels or 20 m^3 front-end loaders loading end-dump haul trucks of approximately 180 metric tonne capacity. The excavators will be supported by dozers. Ore material is hauled up the active mining face and ex-pit to the leach pad. Total ore haulage routes using mine access ramps will vary over the life-of-mine (LOM) but range from about 2 km to 17 km.

Figure 13-1 demonstrates a typical open-pit operation utilizing excavators in shovel configuration and haul trucks to remove both ore and overburden. The general sizing and depth of the mine at most stages of the operation requires multiple working benches on the advancing faces. This will allow consistent mine development with a continued pushback and assist with continuous ore deliveries to the leach pad. The geotechnical parameters shown in Figure 13-1 are described in detail in Section 13.2.1.



Rock Description	Bench Height (m)	IRA (°)	Catch Bench Width (m)	Face Angle (°)
Precambrian Granitic Intrusive Rocks	10	47	5	66
Quaternary Younger Alluvial Fan Deposits	10	38	8	63.4
Quaternary Alluvial Wash Deposits Upper	10	53	5	75
Quaternary Alluvial Wash Deposits Transitional	10	53	5	75
Quaternary Alluvial Wash Deposits Lower	10	53	5	75
Breccia	10	53	5	75

Figure 13-1: El Pilar Typical Mining Configuration

13.1.5 Reclamation

Due to the phased mining approach and construction of the OSF, there will be limited opportunity to perform ongoing reclamation. It is assumed that majority of reclamation will occur towards the end of the mine life. The OSFs, for instance, will be re-sloped (with dozers) to a suitable slope for revegetation.

13.2 PARAMETERS RELATIVE TO THE MINE DESIGN AND PLANS

This sub-section contains forward-looking information related to mine design for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including geotechnical and hydrogeological.

13.2.1 Geotechnical

A geotechnical unit is a unit within which slope performance is expected to be similar throughout. Geotechnical units may be subdivisions of geological units, where alteration or structure varies within the geological units to a degree that it would affect the mechanical properties and potential stability of slopes within the units. Geotechnical units may combine geological units where the mechanical characteristics of different geological units are similar.

At El Pilar, the geological units generally represent units within which the mechanical characteristics and structure are consistent, but which differ from each other. In the transported materials, which lack structure, the principal difference between units is the degree of consolidation and the resulting shear strength. The intrusive and breccia material units are distinctive units characterized by different fracture intensities and strength.

Golder Associates Inc. completed a geotechnical study in support of a Feasibility Study (FS) in 2008. Geotechnical analysis of surface mapping data, drill hole logging data, and laboratory test data are described in Section 7.4 of this TRS. These data were used to develop slope design recommendations for each of the geotechnical units. In the transported materials, recommended slope designs are based on adequate overall stability based on reasonable estimates of available mass shear strength. The slope design in intrusive rocks is based on the average orientations of systematic discontinuity sets as a basis for evaluating structural control of bench stability. The Mines Group, Inc. provided a pit slope stability review for the El Pilar Project in 2010, which updated the assessment of available shear strength in the cemented alluvial deposits, which includes the cemented portions of the overburden (Quaternary Younger Alluvial Fan Deposits) and the upper, transitional, and lower Alluvial Wash Deposits.

The pit slope parameters used in the final El Pilar pit design are shown in Table 13-1.

Table 13-1: Final El Pilar Pit Slope Parameters

Rock Description	Bench Height (m)	IRA (°)	Catch Bench Width (m)	Face Angle (°)
Precambrian Granitic Intrusive Rocks	10	47	5	66
Quaternary Younger Alluvial Fan Deposits	10	38	8	63.4
Quaternary Alluvial Wash Deposits Upper	10	53	5	75
Quaternary Alluvial Wash Deposits Transitional	10	53	5	75
Quaternary Alluvial Wash Deposits Lower	10	53	5	75
Breccia	10	53	5	75

The results of the stability analysis indicate that at an inter-ramp and global scale, the slopes are stable, with factors of safety exceeding the acceptance criteria. There is a portion of the north wall (in Phase 1) that may require further investigation as the transition from granitic intrusive rock to breccia is not well defined. This area exhibits a poorly defined distribution of silicified zones and sand/clay matrix. However, the extent zones with a soft clay matrix may be limited, so it is appropriate to adopt the slope design for cemented/silicified materials since it should be possible to remediate any slope instability that develops in limited zones without reducing overall slope angles.

13.2.2 Hydrological and Hydrogeological Conditions

Exploration holes drilled at El Pilar were typically dry. The groundwater table is understood to be below the pit bottom elevation and there are no known hydrogeological concerns that affect the mine design.

The Santa Cruz River is the main hydrologic resource in the area; during the rainy season, wash runoff drains into the Santa Cruz River. Surface preservation and mitigation measures planned include impermeable retention areas where chemical substances or process solutions are handled, implementation of a hazardous and non-hazardous waste handling program, monitoring of surface water and creek sedimentation, and storm water diversion around disturbed area.

In a hydrological study carried out by Rangel (2008) a total of 196 groundwater sources are considered to be available to the Santa Cruz River aquifer. An availability of 12.907 Mm^3 of groundwater per year was provided as a baseline for permitting purposes. The groundwater static level depth was determined to range between 1.99 m to 48.93 m with a 7.46 m, average. The shallowest depths of 10 m occur near the Santa Cruz River.

Groundwater sampling results show that water quality in the area is good, although one of the area wells sampled has iron concentrations above norm and three of them have total coli forms above stated limits.

Prevention and mitigation measures contemplated to protect groundwater quality include a waterproof layer in the leach pad, sumps, and process areas, as well as installation of water monitoring wells down gradient from mining facilities with regular water quality monitoring.

SCC has concession number: 02SON151018 dated July 21, 2020 to use national groundwater for a volume of 1,166,336 m^3 per year of water and expires on November 23, 2032. Concession number Nº 02SON150567/07FMGC11 was also issued for a volume of 2,333,664.00 m^3 per year and which is currently in the process of being renewed. With these concessions and three wells built to date, the availability of water for the operation of the El Pilar Project is guaranteed.

13.3 MINE DESIGN FACTORS

This sub-section contains forward-looking information related to mine design and production plans for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including mining strategy and production rates, expected mine life and mining unit dimensions.

Mining at El Pilar follows the typical standards for open-pit mining. The processes involved include:

1. Review and modification of Resource Model, described in Section 11 and 12
2. Application of dilution and recovery factors
3. Estimation of cut-off grade and applicable constraints
4. Pit optimization and selection of optimal pit shell to be used for the basis of the ultimate pit design
5. Development of phase designs
6. Estimation of mine planning targets and operational constraints

The selected pit shells derived from the pit optimization process were used as guides to develop detailed open pit mine phase designs including ramps and geotechnical requirements. The phase designs were also limited by mine permit limits.

The design road width of 30 m includes approximately 3 times the width of the largest truck for a running surface, ditch, and appropriate berm, Figure 13-2. This allows for two-way traffic with an adequate separation distance along main haulage routes. Access ramps are designed with a maximum slope of 10%. The pit was designed for 10 m benches. Bench widths and face angles are dependent on lithology, as shown in Table 13-1 above. Given the ultimate pit limits, annual waste and ore tonnages were generated for the mine plan periods with corresponding mining production sequences. The mine design was split into 4 phases. Table 13-2 shows the corresponding tonnages produced over the LOM by phase. Figure 13-3 shows the phases locations within the ultimate pit boundaries as well as the dump and leach pad locations.

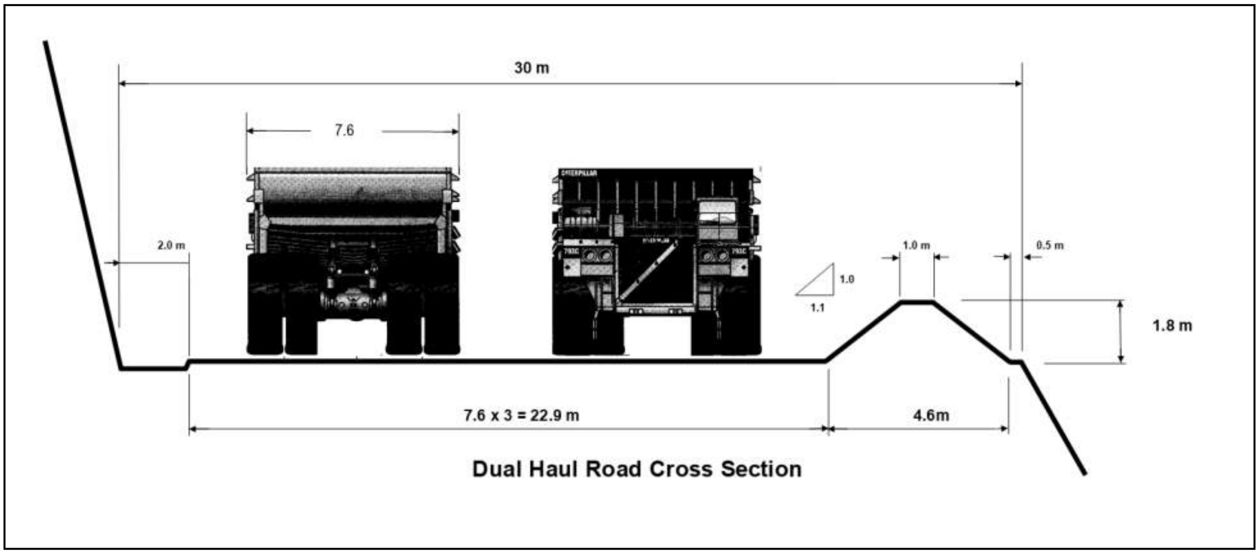


Figure 13-2: Ramp Design Cross Section

Mining within the phases generally progress from North to South away from the leach pad. Phase 1 includes establishing the final wall on the North and West edges of the pit. Pushbacks will be to the East and the South, and access to the dumps and leach pad is to the North. Phase 2 is to the South of Phase 1 and shares the same North, West, and East highwalls. Haulage access to the dumps and leach pad is out of the South. The Phase 3 pushback is to the South and East of Phase 2. This Phase also includes the Northwest satellite pit located west of Phase 1. Haulage access for Phase 3 is out of the East and West. The Phase 4 pushback is to the South and East of Phase 3, with haulage access out of the East and West.

Table 13-2: LOM Mining Quantities by Phase

Descriptions	Units	Total	Phase 1	Phase 2	Phase 3	Phase 4
Incremental Values						
Total Rock	Mt	702	190	165	234	114
Total Waste	Mt	385	83	93	139	70
Total Ore	Mt	317	107	72	95	44
Strip Ratio	t/t	1.21	0.78	1.30	1.46	1.60
Cumulative Values						
Total Rock	Mt	702	190	355	589	702
Total Waste	Mt	385	83	176	315	385
Total Ore	Mt	317	107	179	274	317
Strip Ratio	t/t	1.21	0.78	0.98	1.15	1.21

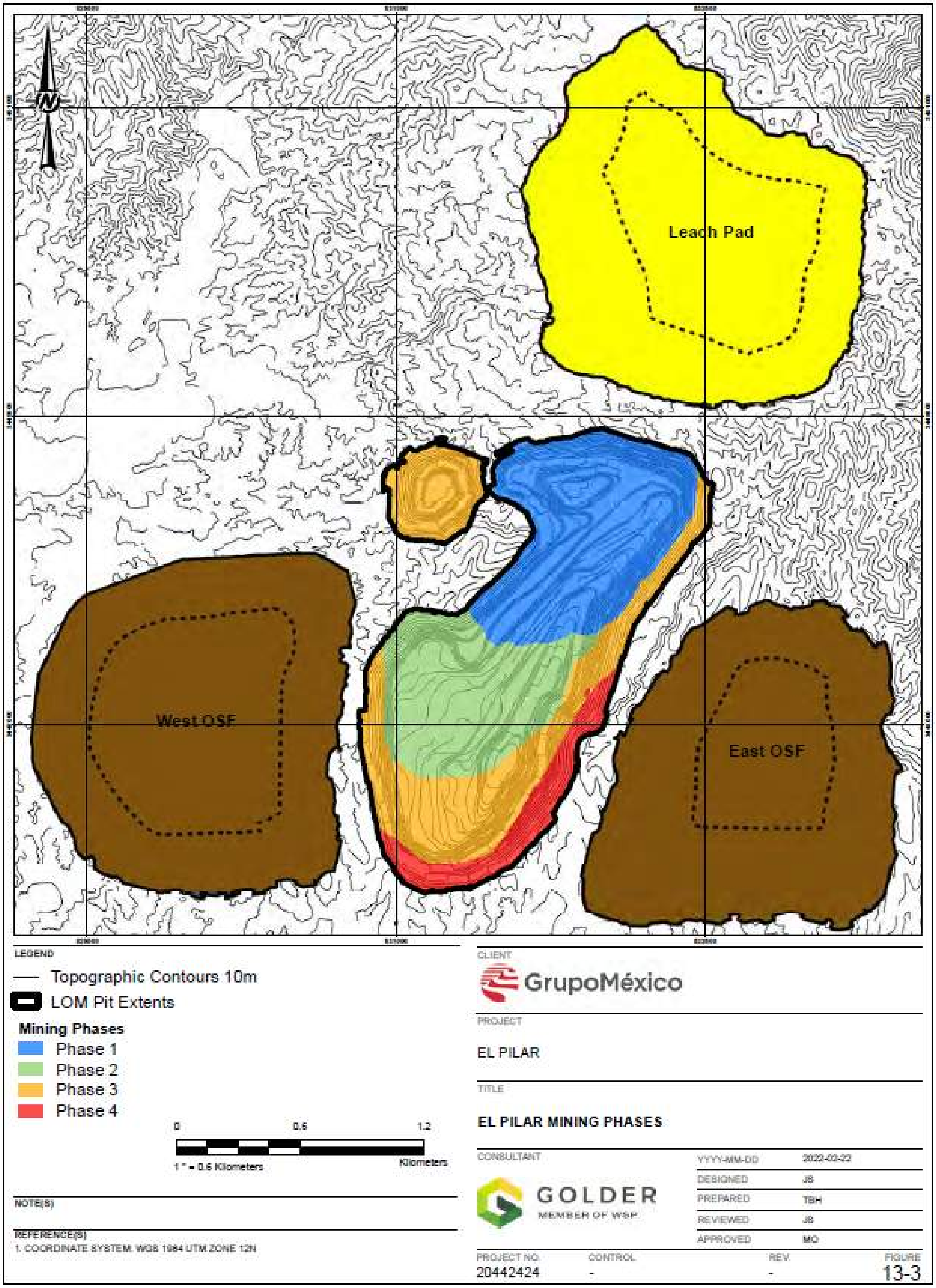


Figure 13-3: El Pilar Mining Phases

13.3.1 Mining Strategy and Production Rates

The mining strategy employs the use of phases which have independent in-pit haul roads that specifically target the ore in that phase and connect to the as-built surface haul roads created by mining the previous phase(s). Production sequencing was carried out using the Deswik interactive scheduler which targets highest value blocks while allowing the user to visually plan multiple ongoing mining faces simultaneously.

The mine production schedule is based on providing an average of 20 Mt ROM to the heap leach pad to produce a maximum of 70 million pounds (Mlbs) of copper cathode per year over the LOM. The maximum total tonnage moved in any given year was limited to 60 Mt.

Table 13-3 and Table 13-4 show the details of the annual production schedule and quarterly with annual production, respectively. Total ROM Ore production for the LOMP is approximately 317 Mt, with a total recovered copper quantity of approximately 427 kt. The LOM stripping ratio is 1.21 tonnes of waste per tonne of ore.

The schedule ramps up the total material movement over the first couple years of mining to reach an average of 50 million tonnes per year (Mtpy) for Year 4 through Year 13 after which total material movement is decreased to about 21 Mt in the last two years of the mine life as waste movement requirements decrease. The commercial life of the pit is about 15 years, in addition to one year of preproduction. The average copper recovery is also shown by period based on the schedule. The estimated average LOM copper recovery is 54%, discussed in Section 12.

Table 13-3: Mine Production Schedule by Year

Annual Mine Production Schedule								
Mining Period	ROM Ore (Ktonnes)	Total Copper (%)	Waste (Ktonnes)	Total (Ktonnes)	Waste: Ore Ratio	Copper Recovery (%)	Contained Copper (Ktonnes)	Recovered Copper (MPounds)
PP	12,556	0.29%	21,492	34,048	1.71	69.0%	36.6	55.8
Y1	14,736	0.32%	24,140	38,876	1.64	67.3%	46.6	69.2
Y2	20,843	0.23%	17,720	38,562	0.85	63.5%	47.5	66.5
Y3	21,469	0.23%	28,336	49,805	1.32	58.8%	49.5	64.1
Y4	20,280	0.26%	32,927	53,207	1.62	57.6%	53.1	67.5
Y5	22,725	0.24%	34,352	57,077	1.51	55.1%	55.0	66.8
Y6	22,303	0.24%	31,589	53,892	1.42	57.2%	52.7	66.5
Y7	18,891	0.27%	29,631	48,522	1.57	57.5%	51.8	65.7
Y8	16,894	0.29%	34,421	51,315	2.04	55.8%	48.8	60.0
Y9	20,008	0.24%	28,245	48,253	1.41	50.5%	47.7	53.1
Y10	20,851	0.19%	35,826	56,677	1.72	51.7%	39.7	45.2
Y11	23,791	0.22%	25,230	49,021	1.06	50.5%	51.9	57.7
Y12	19,809	0.26%	25,855	45,665	1.31	45.3%	50.5	50.5
Y13	26,814	0.23%	9,233	36,046	0.34	45.2%	61.4	61.2
Y14	24,534	0.28%	3,765	28,299	0.15	44.5%	69.2	68.0
Y15	10,960	0.26%	2,095	13,054	0.19	37.0%	28.2	23.0
Total/Avg.	**317,465**	**0.25%**	**384,857**	**702,321**	**1.21**	**54.0%**	**790**	**941**

Note: Figures may not sum due to rounding. Copper Recovery is discussed in Section 12.

Table 13-4: Mine Production Schedule by Quarters and Year

Mine Production Schedule								
Mining Period	ROM Ore (Ktonnes)	Total Copper (%)	Waste (Ktonnes)	Total (Ktonnes)	Waste: Ore Ratio	Copper Recovery (%)	Contained Copper (Ktonnes)	Recovered Copper (Mpounds)
PP Q1	2,594	0.17%	5,414	8,008	2.09	69.2%	4.3	6.6
PP Q2	3,682	0.26%	5,110	8,792	1.39	69.4%	9.5	14.5
PP Q3	3,386	0.34%	5,280	8,666	1.56	69.0%	11.4	17.4
PP Q4	2,895	0.39%	5,687	8,582	1.96	68.7%	11.4	17.3
Y1 Q1	3,083	0.36%	7,087	10,171	2.30	69.8%	11.0	16.9
Y1 Q2	3,458	0.34%	6,679	10,137	1.93	68.3%	11.6	17.5
Y1 Q3	3,953	0.30%	5,878	9,830	1.49	66.2%	11.9	17.3
Y1 Q4	4,243	0.29%	4,481	8,723	1.06	65.0%	12.2	17.5
Y2	20,843	0.23%	17,735	38,577	0.85	63.5%	47.5	66.5
Y3	21,469	0.23%	28,336	49,805	1.32	58.8%	49.5	64.1
Y4	20,280	0.26%	32,927	53,207	1.62	57.6%	53.1	67.5
Y5	22,725	0.24%	34,352	57,077	1.51	55.1%	55.0	66.8
Y6	22,303	0.24%	31,589	53,892	1.42	57.2%	52.7	66.5
Y7	18,891	0.27%	29,631	48,522	1.57	57.5%	51.8	65.7
Y8	16,894	0.29%	34,421	51,315	2.04	55.8%	48.8	60.0
Y9	20,008	0.24%	28,245	48,253	1.41	50.5%	47.7	53.1
Y10	20,851	0.19%	35,826	56,677	1.72	51.7%	39.7	45.2
Y11	23,791	0.22%	25,230	49,021	1.06	50.5%	51.9	57.7
Y12	19,809	0.26%	25,855	45,665	1.31	45.3%	50.5	50.5
Y13	26,814	0.23%	9,233	36,046	0.34	45.2%	61.4	61.2
Y14	24,534	0.28%	3,765	28,299	0.15	44.5%	69.2	68.0
Y15	10,960	0.26%	2,095	13,054	0.19	37.0%	28.2	23.0
Total/Avg.	**317,465**	**0.25%**	**384,857**	**702,321**	**1.21**	**54.0%**	**790**	**941**

Note: Figures may not sum due to rounding. Copper Recovery is discussed in Section 12.

The total material movement and total copper cathode production is shown by year in Figure 13-4.

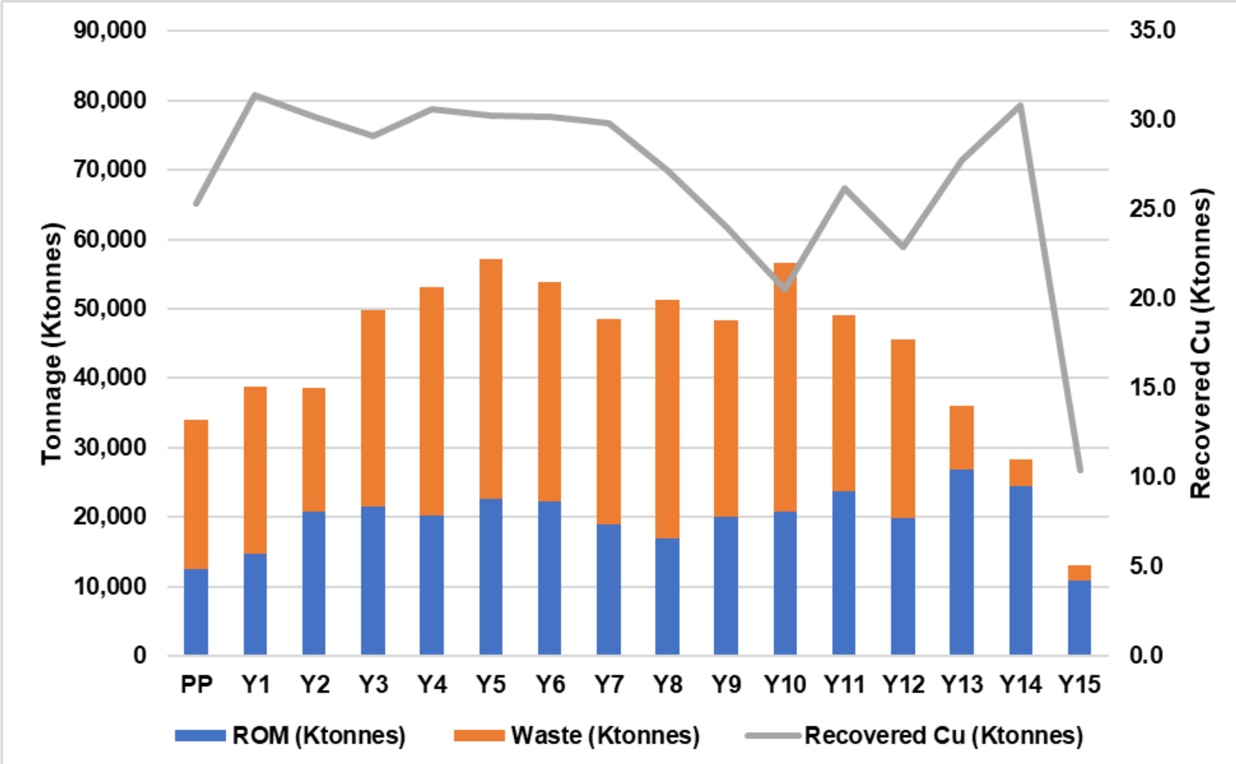


Figure 13-4: Annual ROM and Waste Production and Resultant Copper Cathode Production

13.3.2 Expected Mine Life

The ROM ore production rate is estimated at 12.6 Mtpy in the Pre-Production Year increasing to an average of 21 Mtpy during the peak production years of Year 4 through 13 and averaging to 20 Mtpy over the mine life of approximately 15 years after the pre-production period. The average annual contained copper production is approximately 50,000 tonnes, with an average estimated recovered copper quantity of 27,000 tonnes. The peak total material movement is 57.1 Mtpy.

13.3.3 Mining Unit Dimensions

The operational pit will have catch benches that are 5 to 8 m wide by 10 m high to match the digging profiles of the selected excavators, discussed in Section 13.2.1. The face angles and overall angles vary depending on lithology. Haul roads are designed for two-way traffic and will have a width of 30 m and a maximum ramp grade of 10%, Section 13.3.

13.4 STRIPPING AND BACKFILLING REQUIREMENTS

ROM is hauled to the heap leach pad while waste is hauled to one of two ex-pit overburden storage facilities, or to the in-pit overburden storage area when it becomes available. As the mine progresses, the main haul roads are planned to be moved over time to stay near the edge of the ultimate pit.

The waste storage facility embankments are built in 20 m lifts at the angle of repose of 36° (1.38H:1V) with a 22 m setback between every other lift, every 20 m vertically. This results in an overall angle of about 22° (2.5H:1V) which is conducive to long-term stability and re-vegetation. The waste storage facility design specifications were provided by

SCC. The designs will need to be reviewed by a geotechnical engineer prior to construction. The volume calculations also assumed a material swell factor of 40% for the waste storage facilities.

The design information for the East and West overburden storage facility (OSF) is shown in Table 13-5. The East OSF has a current bench elevation of 1,405 m above mean sea level (AMSL), a maximum design elevation of 1,485 m AMSL and a capacity of 162.4 million loose cubic meters (Mlcm). The West OSF has a current bench elevation of 1,395 m AMSL, a maximum bench elevation of 1,455 m AMSL and a capacity of 183.9 Mlcm. In addition to these, 8.1 Mt of waste is placed in the North end of the pit during Year 11. Figure 13-3 shows the final configuration of the pit and OSFs.

Table 13-5: OSF Design Information

Overburden Storage Facility	Current Bench Elevation	Maximum Design Bench Elevation	Capacity
	meters AMSL	meters AMSL	Mlcm
East OSF	1,405	1,485	162.4
West OSF	1,395	1,455	183.9

13.5 MINING FLEET, MACHINERY, AND PERSONNEL REQUIREMENTS

This sub-section contains forward-looking information related to equipment selection for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this sub-section including labor and equipment availability and productivity.

Mine major equipment requirements were sized and estimated on a first principles basis based on the mine production schedule, the mine work schedule, and estimated equipment productivity rates. The mine is scheduled to operate three 8-hour shifts per day, 365 days per year, for 1,095 available shifts per year. Four mining crews are required. The mine equipment estimate assumes a well-managed mining operation with a well-trained labor pool, and that all equipment is new at the start of mining.

The haul truck planned for use at the site has a capacity of approximately 181 tonnes. The mechanical availability for the shovels and wheel loader is estimated to be 85%, with an operational usage of 67.4% for the shovels and 69.9% for the wheel loaders. The truck haulage profile analyses was completed in Deswik. LHS and yielded information on total distance, cycle time, elevation difference, and fuel consumption for each haulage profile.

The fleet sizing on an annual basis for major production equipment is shown in Table 13-6. The truck fleet will initially consist of 11 trucks and will increase to a maximum of 25 in Year 8. The number of hydraulic shovels is maintained at 2 throughout the LOM, with one (1) wheel loader brought in to assist with production requirements, as needed.

Table 13-6: Major Equipment Fleet

Year	Hydraulic Shovel: CAT 6060 FS	Wheel Loader: CAT 994K	Haulage Truck: CAT 789D	Drill: CAT MD6250	Track Dozer: CAT D10T2	Motor Grader: CAT 16M3	Water Truck: CAT 777	Wheeled Dozer: CAT 834K
PP Q1	2	-	11	2	5	2	2	2
PP Q2	2	1	11	2	5	2	2	2
PP Q3	2	1	11	2	5	2	2	2
PP Q4	2	1	11	2	5	2	2	2
Y1 Q1	2	1	12	3	6	2	2	2
Y1 Q2	2	1	12	3	6	2	2	2
Y1 Q3	2	1	12	3	6	2	2	2
Y1 Q4	2	1	12	3	6	2	2	2
Y2	2	1	13	3	6	2	2	2
Y3	2	1	18	3	6	2	2	2
Y4	2	1	18	3	6	2	2	2
Y5	2	1	22	3	6	2	2	2
Y6	2	1	24	3	6	2	2	2
Y7	2	1	24	3	6	2	2	2
Y8	2	1	25	3	6	2	2	2
Y9	2	1	25	3	6	2	2	2
Y10	2	1	25	3	6	2	2	2
Y11	2	1	25	3	6	2	2	2
Y12	2	1	25	3	6	2	2	2
Y13	2	1	25	3	6	2	2	2
Y14	2	1	25	3	6	2	2	2

The mine support equipment includes the following equipment types and number of units:

- Track Dozer, 600 HP (5-6 units)
- Wheel Dozer, 485 HP (2 units)
- Motor Grader, 4.9 m (2 units)
- Water Truck, 50,000 liter (2 units)

The support equipment is required to perform the following duties:

- Construct roads to the initial mining areas as well as to the crusher and waste storage facilities
- Preproduction development required to expose ore for initial production.
- Mine and transport ore to the crusher. Mine and transport waste material to the appropriate waste storage areas.
- Maintain all the mine work areas, in-pit haul roads, and external haul roads. Also, maintain the waste storage areas.

The equipment list does not include equipment required for construction of the leach pad or plant area, which is included in the capital requirements. It also does not include any equipment required to maintain the leach pad during commercial operations. Starting in Year 0 (pre-production) and extending throughout the LOM, owner mining is planned.

The production schedule of the El Pilar Mine assumes the use of 4 hourly employee crews working three 8-hour shifts, operating 24 hours per day, 365 days per year. The annual hourly personnel ranges from about 200 employees in the pre-production years, up to a maximum of about 290 employees in the latter years, as shown in Table 13-8.

Table 13-7: Derivation of Working Shifts and Paid Labor Hours

Parameter	3 x 8-hour Shift Schedule
Calendar Days Per Year	365
Annual Paid Time Off /Sick	25
Sched. Working Days Per Year	340
Days Per Scheduling Week	7
# Of Scheduled Weeks Per Year	52.1
Hours Paid Per Shift	8.0
S.T. Hours Per Week	40.0
O.T. Hours Per Week	2.0
Equivalent S.T. Hours Per Year	***2,242.1***
S.T. Shifts Paid Per Year	***260.7***
O.T. Shifts Paid Per Year	13.0
Total Working Shifts/Man/Year	***248.8***
O.T. Rate	5%

To calculate the required personnel, the required equipment hours are divided by the hours per shift and the number of shifts per year. The resultant value is then multiplied by the number of crews to yield the required number of operators for each piece of equipment. The annual estimate of the required workforce including maintenance personnel and general staff is shown in Table 13-8.

Table 13-8: Annual Estimate of Required Workforce

Description	PP Q1	PP Q2	PP Q3	PP Q4	Y1 Q1	Y1 Q2	Y1 Q3	Y1 Q4	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14
Operations																					
Shovel Operator	3	4	5	6	6	6	7	7	7	7	8	8	8	8	8	8	8	8	8	8	8
Loader Operator	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Dozer Operator	18	19	20	20	20	20	20	20	21	21	21	21	21	21	21	21	21	21	21	21	21
Drill Operator	5	6	8	9	9	10	10	10	11	11	12	12	12	12	12	12	12	12	12	12	12
Grader Operator	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Haul Truck Operator	61	61	72	72	72	72	72	72	77	77	79	94	94	102	103	103	103	103	103	103	103
Subtotal - Operations	**97**	**100**	**114**	**115**	**115**	**117**	**118**	**118**	**125**	**125**	**128**	**144**	**144**	**152**	**153**	**153**	**153**	**153**	**153**	**153**	**153**
Maintenance																					
Service Truck Driver	10	10	11	11	11	11	11	11	13	13	13	15	15	16	16	16	16	16	16	16	16
Diesel Mechanic	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32	32
Welder	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Journeyman	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
Light Duty Mechanic	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Electrician	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16	16
Subtotal - Maintenance	**90**	**90**	**91**	**91**	**91**	**91**	**91**	**91**	**93**	**93**	**93**	**95**	**95**	**96**	**96**	**96**	**96**	**96**	**96**	**96**	**96**
Staff																					
General Mine																					
Administrative Assistant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Dispatcher	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Maintenance Asst./Clerk	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance Trainer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Equipment Maintenance																					
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Shift Foreman	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Maintenance General Foreman	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Shop Foreman	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Maintenance Planner	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Opearations																					
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine General Foreman	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Foreman	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine Engineering																					
Chief Mining Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mining Engineer	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Chief Surveyor	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Technicians	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Technician/Blasting	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Geology																					
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Geolgist	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Subtotal - Staff	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**
Maintenance to Operations Ratio	48%	47%	44%	44%	44%	44%	44%	44%	43%	43%	42%	40%	40%	39%	39%	39%	39%	39%	39%	39%	39%
Total Hourly Personnel	**187**	**189**	**206**	**207**	**207**	**209**	**209**	**209**	**217**	**217**	**222**	**238**	**238**	**247**	**249**	**249**	**249**	**249**	**249**	**249**	**249**
Total Staff	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**	**37**
Grand Total of Personnel	**224**	**226**	**243**	**244**	**244**	**246**	**246**	**246**	**254**	**254**	**259**	**275**	**275**	**284**	**286**	**286**	**286**	**286**	**286**	**286**	**286**

14 PROCESSING AND RECOVERY METHODS

14.1 PROCESS OVERVIEW

The El Pilar mine is an open pit, oxide copper mine, whereby copper is recovered using conventional copper heap leach technology. ROM ore is stacked on the leach pad and irrigated with a weak sulfuric acid solution to extract copper. The pregnant leach solution (PLS) is collected in a pond and then sent to the solvent extraction-electrowinning (SX-EW) facility to recover copper. Barren solution (raffinate) from the SX facility will be supplemented with acid and then recirculated to the heap leach pad to continue the leach cycle. The SX plant will process at an average rate of 3,800 m^3/h.

Copper is extracted from the PLS by a selective organic reagent solution in the SX extraction circuit. Copper is stripped from the loaded organic in the stripping section using aqueous spent electrolyte from EW. Stripped organic returns to the extraction section to extract more copper from the PLS. Copper is deposited electrolytically in plate form from strong electrolyte. Spent electrolyte returns to strip more copper from loaded organic.

The El Pilar Project is expected to leach a LOM average of 58,000 tonnes per day (t/d) of ore and produce 59 Mlbs of copper cathodes per year.

The following items summarize the process operations required for copper extraction and recovery from El Pilar Project:

- ROM ore is loaded in haul trucks, transported to the pad and stacked in the leach pad in 6 m high lifts.
- After the irrigation piping system is placed on top of each lift, ore is cured for 7 days using a high acid concentration raffinate solution.
- Acid bearing raffinate solution is used for leaching during the standard cycle; pregnant solution is collected by the perforated pipe system at the bottom of the leach pad and transferred into the PLS pond.
- PLS (aqueous) is transferred by gravity to the solvent extraction (SX) plant where it is mixed with an organic solution comprising a mix of solvent such as a kerosene or equivalent and an extractant reagent with an affinity to copper. Copper in solution is then transferred from the aqueous phase to the organic phase. The resulting aqueous solution low in copper (raffinate) is returned to the leach pad. The loaded organic is transferred to the stripping stage.
- Copper is stripped from the organic phase using a high acid concentration aqueous solution, lean electrolyte. Copper is transferred from the organic phase back to the aqueous phase. The resulting aqueous phase is called Rich Electrolyte.
- Raffinate solution from SX is transferred to the raffinate pond, acid content in the raffinate is adjusted to the desired set point and is pumped to the leach pad irrigation system.
- Rich electrolyte is transferred to the electrowinning cells (EW), where copper from the solution is deposited onto stainless steel sheets by means of a direct current system that includes transformer-rectifiers, non-soluble lead anodes, bus bar, etc. The rich electrolyte acts as the media required to allow the flow of current. Copper cathodes are harvested every 7 days. The resulting aqueous phase from EW is called lean electrolyte which is returned back to the stripping settlers for reuse.
- Copper cathodes are stripped using a cathode stripping machine, copper is washed, separated from the permanent stainless steel cathodes, sampled, corrugated using a press, and bundled as final product.
- Auxiliary process facilities include heat exchangers, water boiler for optimum temperature of rich electrolyte, a centrifuge for organic recovery from crud, electrolyte filters, reagents preparation and addition, etc.

14.2 PROCESS FACILITIES

The process facilities include the following:

- Heap Leach Facility, (HLF)
- Solvent Extraction (SX)
- Tank Farm (TF)
- Electrowinning (EW)
- Ancillary Facilities and Services

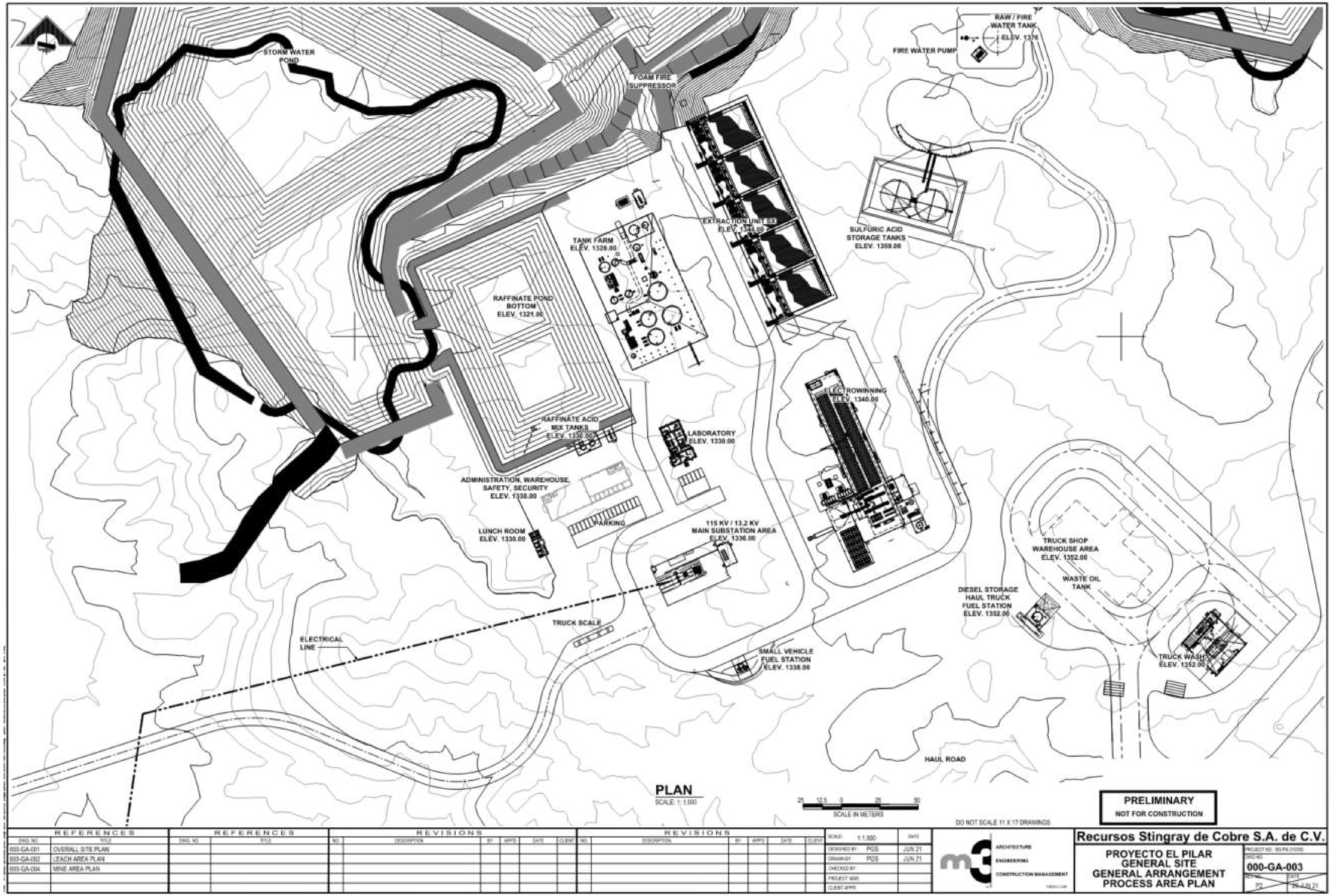


Figure 14-1: General Site Plan

14.2.1 Leaching Facility

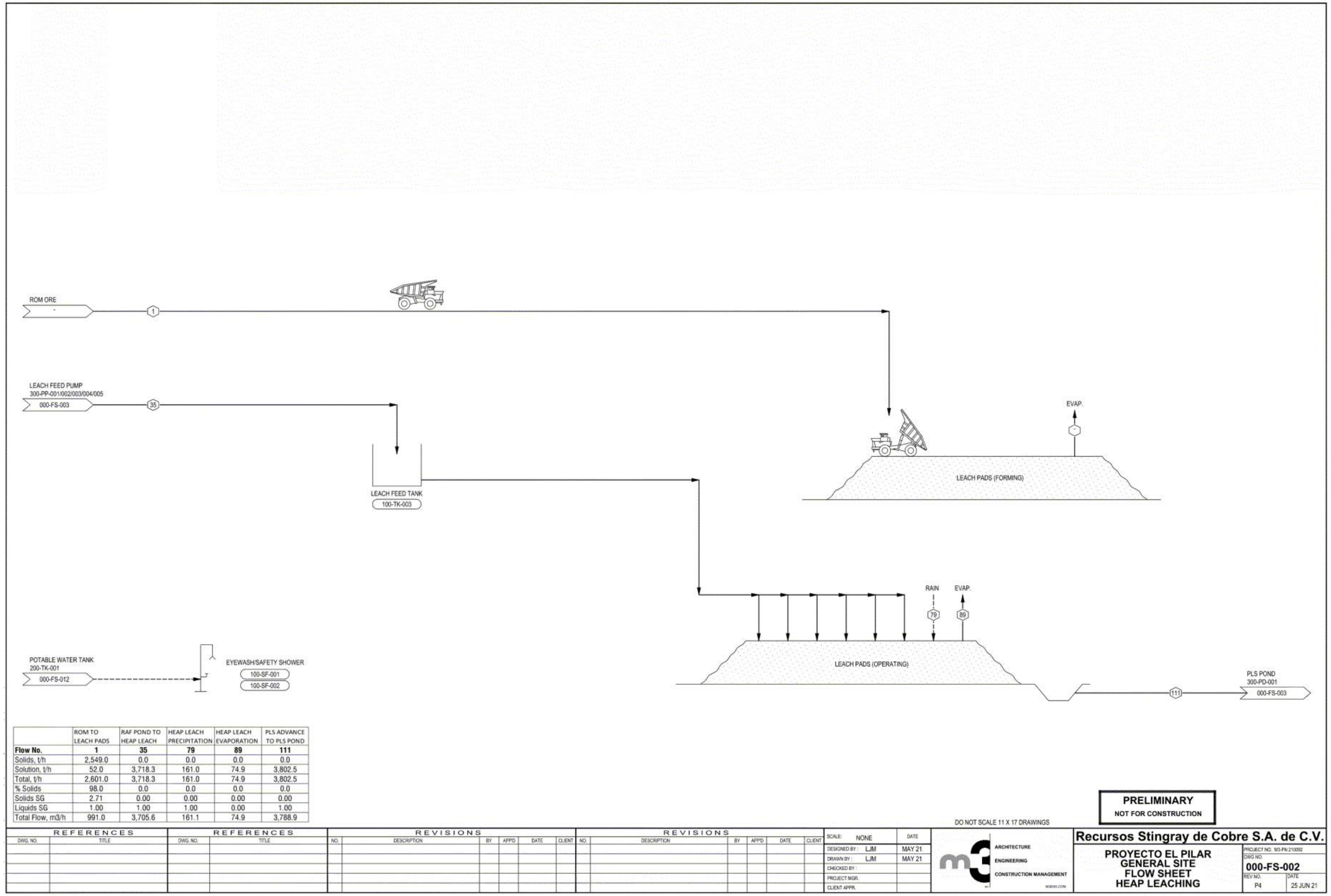
14.2.1.1 Introduction

Standard heap leaching technology, extensively used throughout the international mining community, is being proposed for the recovery of copper. The mine is expecting to process a total of 317 Mt of copper ore bearing material at the proposed heap leach facility located about 0.5 km north of the open pit. Run of mine ore removed from the open pit will be transported via mine haul trucks and stacked on the lined leach pad facility in 6-meter high lifts and irrigated with a weak acid solution to extract the copper metal. The acidic solutions will be contained on, and within, a geomembrane lined pad with an underlying geosynthetic clay liner, and double lined solution ponds with leak collection and recovery systems. After metal extraction, the solutions will be recycled to the heap after supplementing the barren solution with acidified raffinate as necessary. The leach pad and ponds will consist of the following:

- A heap leach pad, constructed in five phases to accommodate a total of approximately 317 Mt of run of mine (ROM) ore. The leach pad will be lined using a composite liner system consisting of prepared subgrade overlain by a compacted clay soil-liner (CCL) or a geosynthetic clay liner (GCL) overlain by a 2 mm thick textured or smooth linear low density polyethylene (LLDPE) geomembrane liner.
- A solution collection system consisting of dual-walled perforated corrugated polyethylene (PCPE) pipes placed on top of the primary liner. The collection system is covered with a 0.5 m thickness of liner cover fill with 1.0 m thick liner cover over the main solution collection pipes.
- A PLS Pond to collect and manage solution flows from the heap leach pad to the SX processing plant.
- A Raffinate Pond to collect and manage solution flows from the SX processing plant back to the heap leach pad.
- The Solution Ponds described above shall have primary and secondary high-density polyethylene (HDPE) geomembrane liners with an HDPE geonet in between the geomembranes. The geonet will collect any leaks through the primary HDPE geomembrane liner. Solution leaks will be conveyed by gravity flow to a leak detection sump with drain gravel, which shall be monitored on a regular basis.
- Two Contingency Ponds shall be constructed as emergency overflow ponds that have been designed for total containment volume for a maximum recorded design storm of 132 mm and a 24-hour drain down duration in the event of loss of pumping capacity.
- These Contingency Ponds will have a single synthetic 2.0-mm (80-mil) HDPE liner installed for primary containment and a GCL as the secondary containment. These ponds will not have a leak detection system as this pond is intended to be managed and be empty except in the case of emergency solution management.

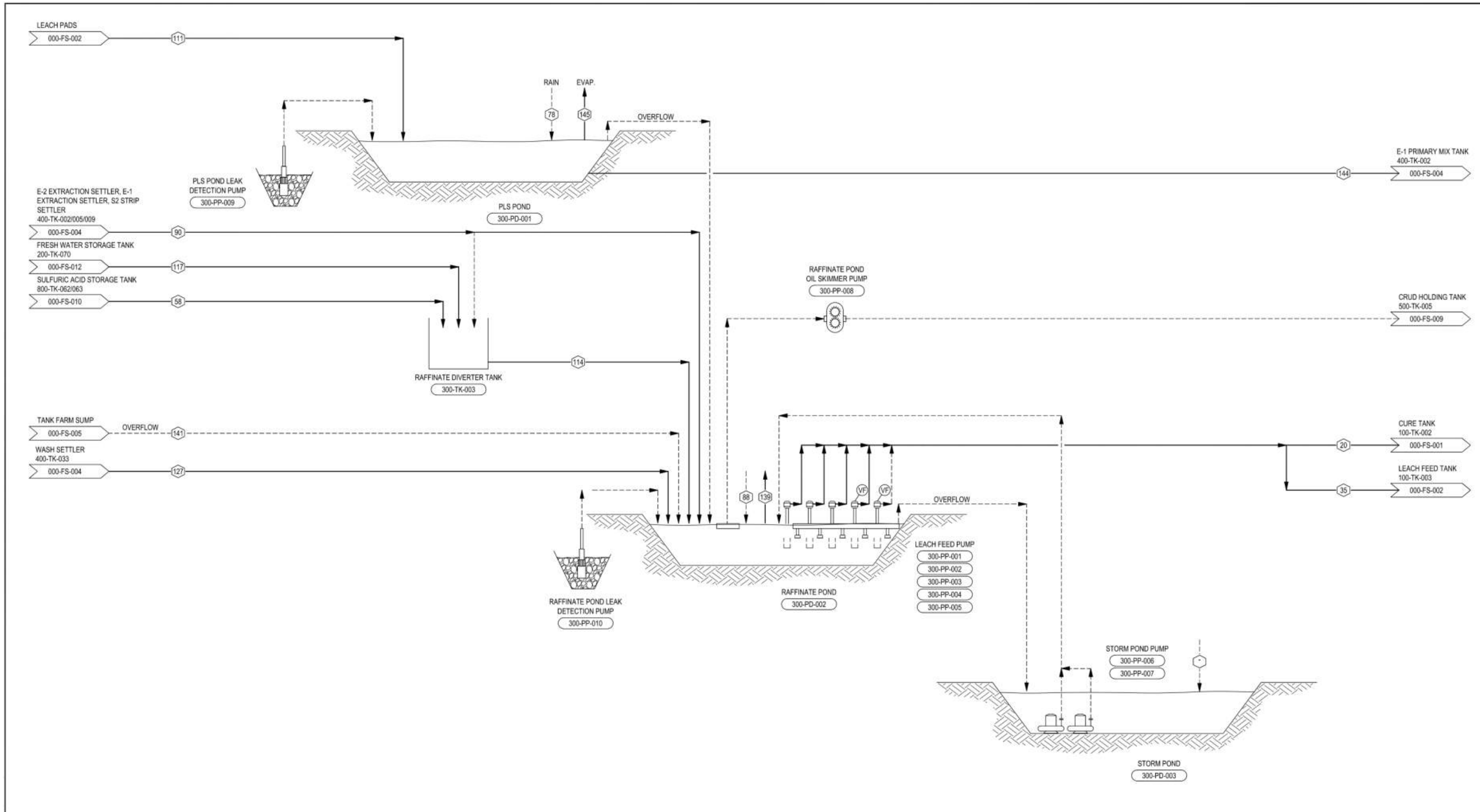
El Pilar Heap Leach Pad and Solution Ponds are expected to have an operation life of 15 years at an average ore production rate of 58,000 t/d. Both PLS and Raffinate have a planned flow rate of 3,800 cubic meters per hour (m^3/hr), respectively.

To minimize acid consumption and have an optimal recovery with respect to the percolation exhibited by the ore, the use of interlift liners between ore lifts is being proposed. The Heap Leach Pad is located approximately 0.5 km north of the proposed open pit in foothill type terrain. The pad is approximately 2.0 km long in the north-south direction and approximately 1.6 km in the east-west direction.



	ROM TO LEACH PADS	RAF POND TO HEAP LEACH	HEAP LEACH PRECIPITATION	HEAP LEACH EVAPORATION	PLS ADVANCE TO PLS POND
Flow No.	**1**	**35**	**79**	**89**	**111**
Solids, t/h	2,549.0	0.0	0.0	0.0	0.0
Solution, t/h	52.0	3,718.3	161.0	74.9	3,802.5
Total, t/h	2,601.0	3,718.3	161.0	74.9	3,802.5
% Solids	98.0	0.0	0.0	0.0	0.0
Solids SG	2.71	0.00	0.00	0.00	0.00
Liquids SG	1.00	1.00	1.00	0.00	1.00
Total Flow, m3/h	991.0	3,705.6	161.1	74.9	3,788.9

Figure 14-2: Heap Leach Flowsheet



	RAFF POND TO CURE TANK	RAFF POND TO HEAP LEACH	SULFURIC ACID TO MIX TANK	RAIN WATER TO PLS POND	RAIN WATER TO RAFF POND	E1/E2 RAFF TO RAFF POND	PLS ADVANCE TO PLS POND	RAFF ADVANCE TO RAFF POND	FRESH WATER TO RAFF TANK	W SETTLER TO RAFF POND	RAFF POND EVAPORATION	T. FARM SUMP TO RAFF POND	PLS TO SX PLANT	PLS POND EVAPORATION
Flow No.	**20**	**35**	**58**	**78**	**88**	**90**	**111**	**114**	**117**	**127**	**139**	**141**	**144**	**145**
Aqueous, t/h	188.2	3,718.3	14.7	0.90	0.55	3,795.1	3,802.5	3,809.9	64.1	21.8	2.4	12.6	3,799.3	4.1
Total, t/h	188.2	3,718.5	14.7	0.90	0.55	3,795.4	3,802.5	3,810.2	64.1	21.8	2.4	12.6	3,799.3	4.1
Aqueous SG	1.00	1.00	1.86	1.00	1.00	1.00	1.00	1.00	1.00	1.09	1.00	1.00	1.00	0.00
H_2SO_4, g/L	1.39	27.41	14.01	0.00	0.00	13.01	6.16	27.02	0.00	1.79	0.00	0.00	6.16	0.00
Cu, g/L	0.21	0.21	0.00	0.00	0.00	0.13	1.31	0.12	0.00	17.77	0.00	0.00	1.31	0.00
Total Flow, m3/h	187.6	3,705.6	7.9	0.9	0.6	3,789.1	3,788.9	3,795.0	64.2	19.9	2.4	12.6	3,789.1	4.1

PRELIMINARY
NOT FOR CONSTRUCTION

DO NOT SCALE 11 X 17 DRAWINGS

REFERENCES — DWG. NO. | TITLE

REFERENCES — DWG. NO. | TITLE

REVISIONS — NO. | DESCRIPTION | BY | APP'D | DATE | CLIENT

REVISIONS — NO. | DESCRIPTION | BY | APP'D | DATE | CLIENT

SCALE: NONE
DESIGNED BY: LJM — MAY 21
DRAWN BY: LJM — MAY 21
CHECKED BY:
PROJECT MGR:
CLIENT APPR.

ARCHITECTURE
ENGINEERING
CONSTRUCTION MANAGEMENT

Recursos Stingray de Cobre S.A. de C.V.

PROYECTO EL PILAR
GENERAL SITE
FLOW SHEET
SOLUTION PONDS

PROJECT NO. M3-PN 210092
DWG NO. 000-FS-003
REV NO. P4
DATE 25 JUN 21

Figure 14-3: Heap Leach Flowsheet - Solution Ponds

14.2.1.2 Project Description

Heap Leach Pad Design Criteria

The following design criteria were used for the conceptual design of the El Pilar Heap Leach Pad, Solution and Contingency Ponds:

Geometry

- The ultimate heap leach pad footprint is 2,424,042 m^2
- The design El Pilar ROM ore capacity requirement: 317 million tonnes Mt
- Number of ore phases: five (5)
- Average ore heap height over the liner: 90 m
- Maximum ore height over liner: 102 m above low center pad area
- Nominal ROM ore lift height: 6 m
- Dry density of stacked ore: 1.72 tonnes per m^3
- Routing of all pad drainage by gravity flow to the PLS Solution Pond;
- Nominal ore angle of repose slope: 1.4 horizontal (H):1 vertical (V) (assumed)
- Overall heap side slopes: 2.5H:1V

Underdrain design

- Underdrain consisting of dual-walled PCPE pipes placed within a geotextile lined trench backfilled with drain gravel.

Stormwater Management

- Design storm event for erosion protection: 100-year, 24-hour event
- Design storm event for flow conveyance: 100-year, 24-hour event;
- Precipitation-Surface water runoff relationship: SCS method
- Maximum channel velocity w/o riprap (design): 1.5 meter per second (m/s)
- The stormwater diversion open channels around the Heap Leach Pad and Solution Ponds

Solution Collection

- A solution collection system consisting of dual-walled PCPE pipes placed on top of the primary liner and covered with a 0.5 m thickness of drain cover fill on the entire heap leach pad and 1.0 m over main solution collection pipes
- Maximum hydraulic head on ore storage liner: 0.3 m
- Primary solution collection pipe: Dual-wall PCPE pipe
- Secondary solution collection pipe: Dual-wall PCPE pipe
- Tertiary solution collection pipe: Dual-wall PCPE pipe
- Solution collection: Gravity flow to PLS Pond.

Solution Balance

- Design flow: 3,800 m³ per hour
- Typical solution application rate: Nominal 7.8 liters per hour per m² (L/h/m²)
- Ore moisture content loss to ore wetting: 5 %
- Average ore production rate range: 55,000 t/d
- Design precipitation event: 132 mm (from Santa Cruz FFCC weather station)
- Design draindown period for water balance: 24 hours
- Contingency Pond: Total containment volume for a maximum recorded design storm and a 24-hour drain design duration in the event of loss of pumping capacity

Interlift liner

- Nominal ROM ore lift height: 6 m
- Interlift liner installation every 2 or 3 lifts
- Solution Collection System included for each interlift liner
- Interlift solution collection pipe: Dual-wall perforated PCPE pipe
- A minimum layer thickness of 50 cm of drain gravel Fill
- HDPE pipe with HDPE drop boxes

Heap Leach Pad Liner System Design

HLF Foundation Preparation and Grading

The HLF will be constructed in five Phases. The phased expansion of the facility has been developed to consider deferral of upfront capital expenditures, while accelerating the development of the initial footprint to accommodate the required initial ore loading for initial Phases. Phase 1 will require the most significant HLF earthworks compared to later phases since it requires full development of the following:

- Excavation of geotechnically unsuitable materials
- Construction of the HLF´s underdrain system
- Surface rough and finish grading to establish positive drainage for the HLF
- Construction of the main access road and ore haul roads
- Construction of the temporary surface water diversion system for Phase 1
- Construction of the PLS Solution Pond
- Construction of the Raffinate Solution Pond
- Construction of the Contingency Ponds No. 1 and No. 2
- Production of the drain cover fill from crushed ore and/or waste rock
- Construction of the permanent surface water diversion system
- Construction of the ore haul road from the Pit to HLF

Construction of the HLF will consist of stripping organic and unsuitable soils from the foundation, construction of underdrain system, developing foundation drainage, development of suitable borrow soils (internal and external to the HLF), general foundation preparation (moisture conditioning and compaction), placement of the CCL in toe area, and deployment and placement of the GCL and geomembranes.

Site grading of the Heap Leach Pad will involve local cuts and fill of native alluvial materials. The limit area where the natural slopes within the leach pad footprint area are steeper than 2.5H:1.0V will be flattened to facilitate the construction of the liner system and ore placement. Site grading in the leach pad area will be done primarily with engineered placement compacted fill to create a foundation that will accommodate the placement of ROM ore.

The sideslopes of the Solution Ponds will be graded in preparation for placement of the engineered double liner system. The maximum sideslope within the Solution Ponds will be 3.0H:1V.

Heap Leach Pad Liner System

The liner system for the proposed El Pilar Heap Leach Pad will be a composite system consisting of a single composite liner (i.e., a geomembrane in direct contact with a clay soil liner) to provide solution containment. The liner system consists of the following components (from the foundation upward) for the central toe area:

- A compacted subgrade or engineered fill
- A secondary liner composed of a minimum layer thickness of 30 cm of low-permeability soil (Liner Bedding Fill) compacted to achieve an in-place permeability of 1 x 10^{-6} centimeters per second (cm/sec) or less, this condition is only applicable for the central toe area estimated in Phase 1 for stability conditions
- The primary liner consisting of a 2 mm thick LLDPE textured geomembrane
- A solution collection system consisting of dual-walled PCPE pipes placed on top of the primary liner and covered with a 0.5 m thickness of drain cover fill on the entire heap leach pad and 1.0 m over main solution collection pipes.

The rest of the Heap Leach Pad will be a composite system consisting of the following (from the foundation upward):

- A compacted subgrade or engineered fill
- A secondary liner consisting of a geosynthetic clay liner (GCL)
- A primary liner consisting of a 2 mm thick LLDPE smooth geomembrane
- A solution collection system consisting of dual-walled PCPE pipes placed on top of the primary liner and covered with a 0.5 m thickness of drain cover fill on the entire heap leach pad and 1.0 m over main solution collection pipes.

LLDPE is proposed for the Heap Leach Pad liner system because it has the following benefits (Lupo and Morrison, 2005):

- Generally higher interface friction values, compared to other geomembrane materials.
- Ease of installation in cold climates due to added flexibility.
- Good performance under high confining stresses (high ore heap heights).
- Higher allowable strain for projects where moderate settlement may become an issue.

Solution Collection System Design

The design of the solution collection system for the heap leach pad utilizes a high permeability granular drainage fill layer processed from crushed rock materials, supplemented with perforated corrugated polyethylene (PCPE) pipes.

The Solution Collection System has the following functions:

- Collect and convey the leach solutions from the base of the Heap Leach Pad to the PLS Pond.

- Minimize hydraulic head from the leach solution on the liner system and reduce the risk of solution leakage to the subsoil environment.
- Protect the composite liner system from any damage during the placement of the ROM ore.

The Drain Cover Fill and PCPE pipes will be placed over the geomembrane liner. The flow of the main solution in the solution collection layer will be captured by a network of smooth inner wall PCPE pipes 6” diameter placed under the ore leach pile, in parallel with a separation of 12 m from center to center in the leveling zone with a slope greater than 2% in the Leaching Pad. These 6” tertiary pipes will direct the solution flow to PCPE Secondary Pipes of 12”, 18”, 24” and 30" diameter, with smooth inner wall.

The solution catchment area on the Leaching Pad is divided into several areas by the configuration of the secondary solution collection pipes. The captured solution will pass through the solution collection pipes crossing the different Phases and finally connect to the solution conduction pipes that carry the flow to the PLS Pond.

The heap leach pad utilizes PCPE pipe as solution collection piping. Header pipe in size from 12”, 18”, 24” and 30” diameter are used within the heap Leach Pad. Lateral pipes feeding the header pipes are 6” diameter PCPE. All PCPE pipes will be of double-walled construction, with an outside corrugated wall and a smooth interior wall that will provide a Manning´s Roughness coefficient of 0.012.

Solution Pipe Capacity Design

The solution collection system is designed to minimize the head on the geomembrane liner system. Therefore, the PCPE collection pipes are designed considering the maximum area under leach that each pipe would be required to drain with a maximum design solution application rate of 10 L/h/m² and a design solution flow rate of 3,800 m³/hr. The solution collection pipe are designed to increase in diameter as they progress toward the collection ponds to accommodate the increased flows from large tributary solution application areas.

The length of the lateral solution collection pipes was sized based on a typical spacing of 12 m, a slope of 2%, and the area under leach, assuming an application rate of 10 L/h/m². The maximum length of the tertiary pipe is 254 linear m; each pipe must transmit a flow rate of approximately 43.2 m³/h.

All 12”, 18”, 24” and 30” PCPE solution collection pipes area designed to carry at least 200% of the production flows collected from their upgradient tributary area. The capacity of all pipes was calculated assuming a Manning´s roughness coefficient of 0.012 and a slope of 2.0% within the majority of the heap. The 18” diameter PCPE pipes will be required to convey solution flows of up to 828 m³/hr. The 24” diameter pipes will be required to convey solution flows of up to 1,785 m³/hr. The maximum flow capacity for each main pipe will be 30" is 6,404.4 m³/h.

All 36” HDPE solution collection pipes are designed to carry at least 200% of the production flows collected from the upgradient PCPE pipeline. The capacity of all pipes was calculated assuming a Manning´s coefficient of 0.012 and a slope of 2% within the solution collection spillway. The 36” diameter pipes will be required to convey solution flows of up to 8,708.4 m³/hr.

PLS Raffinate Pond Liner System

The liner system for the PLS Pond will be constructed with a double HDPE geomembrane liner system separated by an HDPE geonet layer. The HDPE geonet is the component of the leak collection and recovery system (LCRS). Solution that migrates through the upper geomembrane liner is conveyed through the HDPE geonet to the gravel filled LCRS sump located in the lowest corner of the pond.

The liners system for the ponds will consist of, from the bottom upwards:

- A compacted subgrade
- A geosynthetic clay liner (GCL) secondary liner
- A 1.5-mm thick HDPE geomembrane secondary liner
- An HDPE geonet for the LCRS layer
- A 2.0-mm thick HDPE geomembrane primary liner.

HDPE geomembrane is proposed for the PLS Pond, because it has a higher ultraviolet resistance than the LLDPE material. The PLS Pond is designed to overflow into Contingency Pond No. 1. The Raffinate Solution Pond is designed to overflow into the Contingency Pond No. 1 during contingency conditions.

PLS and Raffinate Solution Pond Leak Detection System

The ponds have been designed with an HDPE geonet LCRS between the upper and lower geomembranes. Solution will be collected in the LCRS and transported by gravity to a sump in the event of a leak in the upper geomembrane. The sump will be constructed in the lowest corner of the bottom of the pond. The sump will contain a 0.6-meter thick layer of free draining gravel. A 6" diameter pipe will extend along the entire sump, from the base of the sump to the crest of the pond, where the presence of solutions may be checked, sampled, or measured on a regular basis. The solution in the sumps may be removed with a small diameter submersible pump.

Contingency Pond Liner System

The PLS and Raffinate Solution Ponds are designed to overflow into the Contingency Pond No. 1 and No. 2 during contingency conditions. Since the Contingency Ponds will only be used during contingency conditions the solution reporting to the pond will be dilute.

The liner system for the Contingency Ponds will consist, from the bottom upwards, of:

- A prepared and compacted subgrade
- A GCL secondary liner
- A 2.0-mm thick HDPE geomembrane primary liner.

Interlift Liner System

Geomex was retained by SCC early 2021 to perform percolation field tests on superficial material excavated from the open pit area. Two test pads were constructed to perform field percolation tests. The percolation field tests were performed according to the Porchet method (Drainage Manual, USBR 1993).

Laboratory permeability and percolation tests were also performed on the same ore materials in Golder´s laboratory in Denver, Colorado. The results of the field and laboratory indicate a low percolation rate for the fresh ore and leached ore samples less than 7.8 L/h/m². Based on these results, interlift liners are recommended during operations to maintain adequate percolation of the ore with low ore loads.

The proposed liner system for each interlift layer on the Heap Leach will be constructed with a single LLDPE geomembrane liner, placed on top of a geotextile overlying the surface of the ROM ore. Site preparation activities include preparing the surface of the ROM ore, before placement of the liner system. It will be necessary to construct perimeter berms with structural fill to anchor the geosynthetics and to contain the solutions.

The ROM ore surface should be graded with positive slopes to efficiently conduct the solution flows between ore lifts. The stacking plan considered in this conceptual study does not consider the sloping ore surfaces.

The Interlift Liner Design on the ore heap surfaces consists of the following (from the bottom towards the top):

- A sloping prepared ROM ore surface
- A geotextile for geomembrane protection
- A 1.5 mm thick LLDPE geomembrane liner
- A minimum layer thickness of 0.50 m of Drain Cover Fill.

The conceptual configuration of interlift phases solution collection piping system will consist high permeability granular Drain Cover Fill layer of 0.5 m thick in general, and 1.0 m thick where the secondary pipes of the solution collection system are located. The PCPE pipes and Drain Cover fill will be laid over the geomembrane liner.

The solution flow on each interlift will be collected by a network of 6" diameter dual wall PCPE pipes within the drain cover fill. The PCPE pipes will be spaced at a distance of 12 m and the prepared or surface will have a minimum slope of 2% towards the low point on each interlift ore surface. These 6" tertiary pipes will direct the solution flow to the secondary 24" and 30" diameter PCPE pipes.

The solution catchment on the interlift ore will be conveyed through two 36" HDPE pipes and will be connected to the main network of solution conduit pipes that carry the flow to the PLS Pond.

14.2.2 Process Plant

14.2.2.1 Introduction

The locations of the heap leach, process plant, and other process facilities are shown on Figure 14-1.

The PLS coming from the leaching area will be processed to recover copper using conventional solvent extraction and electrowinning (SX-EW) technology. The SX-EW plant is designed to process 3,800 m^3/h of PLS and produce an average of 59 Mlbs of copper cathode per year over the LOM.

The copper will be recovered from the PLS in a SX facility consisting of two parallel extraction stages, two stripping stages and a wash stage as shown in Figure 14-4, below. In the extraction stages, the PLS loaded with copper (or aqueous solution) is contacted with a reagent diluted in a solvent, together called the "organic solution". The aqueous/organic mixture is left to separate by density difference in the settlers. The reagent extracts the copper from the PLS, then the solution depleted of copper, or Raffinate, is pumped back to the leach pad as leaching solution. The organic solution carries the copper to the Stripping Stage, where the organic solution is contacted with a highly acidic solution, called Lean Electrolyte returning from the EW tankhouse. The high acid concentration in the Lean Electrolyte causes the reagent to release the copper, increasing the copper concentration to produce a rich electrolyte, which is pumped to the EW tankhouse. The wash stage uses water as aqueous solution at a high organic to aqueous ratio, which causes iron and manganese to be transferred into the aqueous phase, reducing their concentrations. Aqueous solution leaving the wash settler will go into the Raffinate pond for dilution to acceptable concentration. Manganese causes deterioration of the organic reagent, which is the main objective of the wash settler in this project.

The rich electrolyte pumped from the SX plant to the EW tankhouse will be distributed to 92 electrowinning cells each of which contains 67 lead anodes paired with 66 stainless steel cathodes. A direct electrical current will be applied to the cells to plate the copper on the stainless steel blanks to form the copper cathodes on both sides of the SS blanks. The copper cathodes will be removed in a 7 day cycle. Copper cathodes will be the final product of the EW circuit. At the planned throughput rate, the plant will produce 59 Mlbs of copper cathode per year.

Copper cathode will be weighed, sampled, corrugated, identified, strapped and securely stored on site pending delivery to market. The El Pilar copper cathode product should meet ASTM B115, latest revision, Grade 1 specifications.

M3 used the METSIM™ process simulation system to perform the mass balance for the El Pilar leaching and SX-EW plant. METSIM™ is a metallurgical process simulation computer program written to perform mass balances around process unit operations.

SX process design criteria is shown in Table 14-1.

Table 14-1: SX Process Design Criteria

Parameter	Value	Unit
SX Configuration	2 EP, 2S, 1W	
Availability	98	%
PLS Design Flow	3,800	m^3/h
Copper Concentration	1.2	g/L
Iron Concentration	7.0	g/L
Manganese Concentration	2,000	ppm
Wash Solution Design Flow	10 to 30	m^3/h
Stripping Solution Design Flow	345	m^3/h
Organic Design Flow	1,900	m^3/h
O/A External/Internal ratio, extraction	1:1	
O/A External/Internal ratio, washing	38-95:1	
O/A External/Internal ratio, stripping	6.5:1	
Total Mixing Flowrate	3,800	m^3/h
Total Mixing Time, Extraction Settlers	3	min
Total Mixing Time, Stripping Settlers	2	min
Settler Flux @ 3,800 m^3/h	5.2	$m^3/h/m^2$

14.2.2.2 Solvent Extraction Plant

The Solvent Extraction (SX) Plant is located immediately southwest of the leach pad. The prevailing wind direction in the project area is from the southwest. The general location of the SX Plant with respect to the other process facilities is such that no major facilities are downwind from or completely adjacent to the SX Plant. Although there have been only four plant fires reported since the commissioning of the first SX-EW plant in 1968, this is a precaution taken in recent plant designs to help protect the rest of the plant.

The plant consists of one train with five individual mixer-settler stages. There are two parallel stages of extraction (E1 and E2), two stages of stripping in series (S1 and S2) and a wash settler (W). One of the stripping settlers can be used as another extraction stage (E3) to increase flow in the later years of operation, operating in parallel with E1 and E2. Each extraction stage includes a primary mixer/tank, a secondary mixer/tank, a tertiary mixer/tank and a settler. The stripping and wash stages have primary and secondary mixers and a settler. Settlers are reverse flow and are 26 meter wide x 28 m length and 1,120 mm deep, with a flux rate of 5.2 $m^3/h/m^2$.

The PLS pond is located in the southwest corner of the leach pad, at an elevation higher than the SX plant. PLS will flow by gravity to the plant. This arrangement was dictated by the need to optimize earthworks in the area available for construction of the leach pad and the process plant.

The solvent extraction flowsheet in Figure 14-4 below shows the flow distribution and Figure 14-5 shows the proposed arrangement.

14.2.2.3 Tank Farm

The purpose of the tank farm (TF) is to collect loaded organic and return it to SX for stripping, filter rich electrolyte, heat rich electrolyte to feed the EW plant and recover organics.

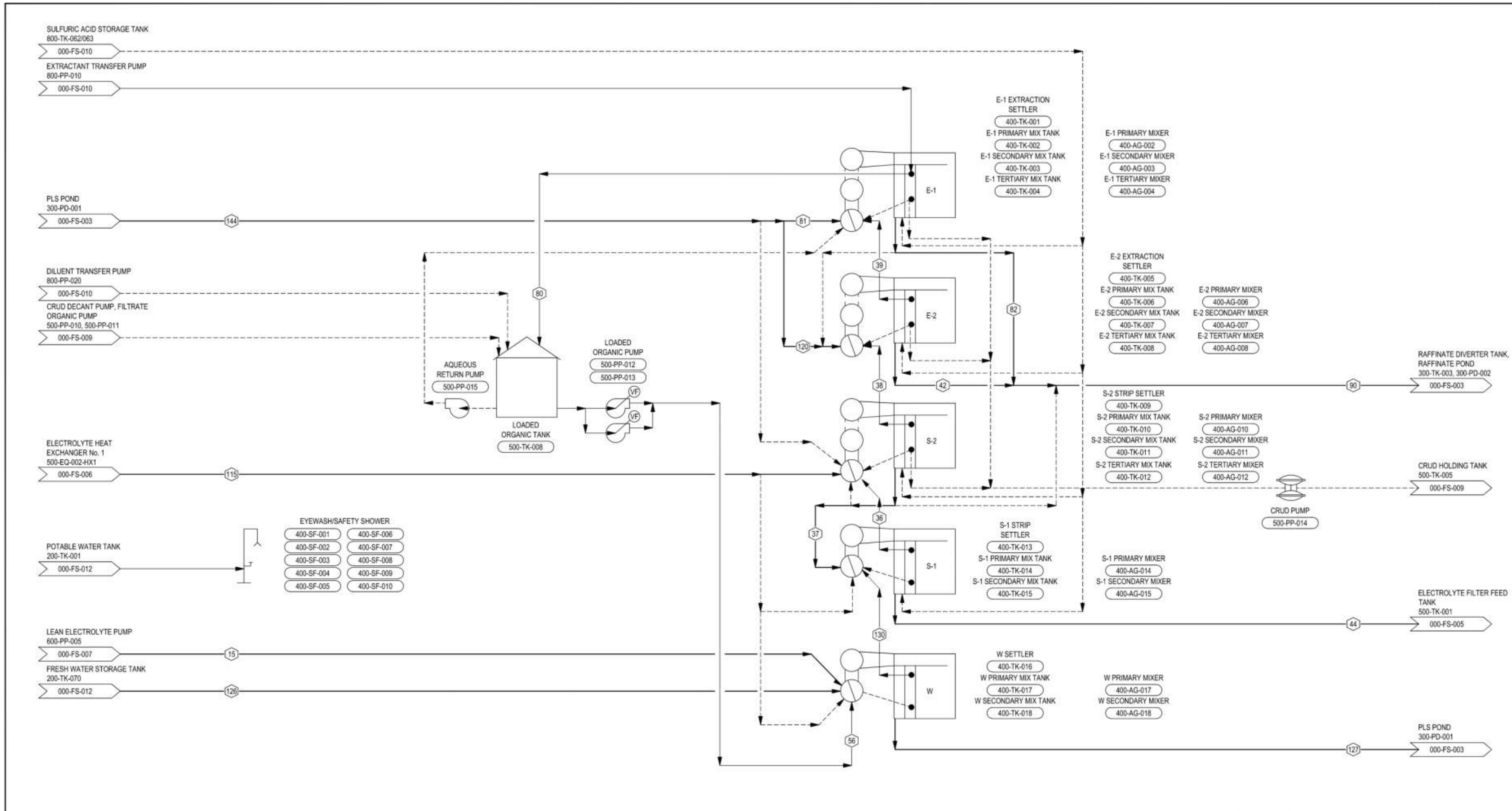
Loaded Organic is collected in the loaded organic tank (11.7 m diameter x 10.9 m height, 900 m^3 effective volume). This tank has an aqueous return pump which will send aqueous back into the E1 settler. Diluent will be replenished in this tank. Loaded organic will be pumped into the W settler, and will flow into subsequent stripping stages downstream.

Rich electrolyte is passed through an electrolyte filter to remove solids and organics. The rich electrolyte flows by gravity from the S1 stripper to the electrolyte filter feed tank, which has a capacity for holding 30 minutes of rich electrolyte flow (4.3 m diameter x 10.9 m height, 142 m^3 effective volume) at a design flow of 284 m^3/h. The filtered electrolyte is collected in the electrolyte feed tank, which has a capacity for holding 1 hour of filtered electrolyte flow (8.2 m diameter x 6.6 m diameter, 284 m^3 effective volume).

The filtered electrolyte solution is pumped to the EW tankhouse through 2 heat exchangers operating in series. The first heat exchanger warms the rich electrolyte using lean electrolyte returning from the EW tankhouse. Hot water from a hot water boiler system is used to heat the solution, the system consists of three 1,850 kW boilers and a 15 m^3 capacity hot water tank with heating coil.

A system to process solvent extraction crud is provided. Crud is material which accumulates at the organic/aqueous interface in the SX settlers. This material is treated to recover the valuable organics. The crud is removed from the settlers via air operated pumps and transferred to a crud decant tank designed with sufficient capacity to hold the volume of a complete settler (11.5 m diameter x 8.7 m height, 750 m^3 effective volume).

The crud is diluted with diluents and allowed to settle (decant) in the decant tank. The upper organic in the decant tank is recovered and sent to the loaded organic system. Underflow from the crud decant tank is pumped using an air operated diaphragm pump into another tank, where clay is added and mixed with the crud. The resulting crud/clay mixture is pumped into a three-phase decanter centrifuge, where a separation of solids, organic and aqueous phases takes place. Crud is removed in the solids phase into a box, aqueous is recovered through the tank farm sump, and the treated organic is recovered into a clarified organic tank, which is returned to process onto the loaded organic tank.



	L ELECTROLYTE BLEED TO W SETTLER	S1 ELECTROLYTE ADVANCE TO S2	S2 ELECTROLYTE ADVANCE TO S1	S2 ORGANIC ADVANCE TO E2	E2 ORGANIC ADVANCE TO E1	E2 RAFF TO RAFF POND	LOADED ELECTROLYTE TO EW	LOADED ORGANIC TO W SETTLER	E1 ORGANIC ADVANCE TO LO TANK	PLS ADVANCE TO E1	E1 RAFF TO RAFF POND	E1/E2 RAFF TO RAFF POND	LOADED ELECTROLYTE TO S2	PLS ADVANCE TO E2	FRESH WATER TO W SETTLER	W SETTLER TO RAFF POND	W ORGANIC ADVANCE TO S1	PLS TO SX PLANT
Flow No.	**15**	**36**	**37**	**38**	**39**	**42**	**44**	**56**	**80**	**81**	**82**	**90**	**115**	**120**	**126**	**127**	**130**	**144**
Aqueous, t/h	11.8	0.1	348.3	0.1	0.2	1,897.5	350.6	0.0	0.1	1,899.7	1,897.7	3,795.1	346.4	1,899.7	10.0	21.8	0.0	3,799.3
Organic, t/h	0.0	1,948.0	0.0	1,946.4	1,948.2	0.1	0.0	1,950.0	1,949.7	0.0	0.2	0.3	0.0	0.0	0.0	0.0	1,950.0	0.0
Total, t/h	11.8	1,948.1	348.3	1,946.5	1,948.3	1,897.6	350.6	1,950.0	1,949.7	1,899.7	1,897.9	3,795.5	346.4	1,899.7	10.0	21.8	1,950.0	3,799.3
Aqueous SG	1.18	1.19	1.18	1.18	1.00	1.00	1.19	0.00	1.00	1.00	1.00	1.00	1.18	1.00	1.00	1.09	0.00	1.00
Organic SG	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.84	0.00	0.84	0.84	0.84	0.00	0.00	0.00	0.84	0.00
Aqueous Cu, g/L	34.81	50.31	41.70	41.70	0.09	0.09	50.31	0.00	0.17	1.31	0.17	0.13	34.81	1.31	0.00	17.77	0.00	1.31
Organic Cu, g/L	1.43	1.43	0.42	0.42	1.59	1.59	1.43	2.69	2.69	0.00	2.69	2.33	1.43	0.00	0.00	0.00	2.69	0.00
H_2SO_4, g/L	1.80	0.01	50.00	0.01	0.00	6.62	46.21	0.00	0.00	3.08	6.39	13.01	53.05	3.08	0.00	1.79	0.00	6.16
Total Flow, m3/h	10.0	1,948.1	295.0	1,946.5	1,948.3	1,896.2	295.1	1,950.0	1,949.7	1,894.6	1,894.6	3,789.1	294.7	1,894.6	10.0	19.9	1,950.0	3,789.1

PRELIMINARY
NOT FOR CONSTRUCTION

DO NOT SCALE 11 X 17 DRAWINGS

REFERENCES		REFERENCES		REVISIONS						REVISIONS					
DWG. NO.	TITLE	DWG. NO.	TITLE	NO.	DESCRIPTION	BY	APP'D	DATE	CLIENT	NO.	DESCRIPTION	BY	APP'D	DATE	CLIENT

		DATE
SCALE:	NONE	
DESIGNED BY:	LJM	MAY 21
DRAWN BY:	LJM	MAY 21
CHECKED BY:		
PROJECT MGR.		
CLIENT APPR.		

ARCHITECTURE
ENGINEERING
CONSTRUCTION MANAGEMENT

Recursos Stingray de Cobre S.A. de C.V.

PROYECTO EL PILAR
GENERAL SITE
FLOW SHEET
SOLVENT EXTRACTION

PROJECT NO. M3-PN 210092
DWG NO. **000-FS-004**
REV NO. P4
DATE 25 JUN 21

Figure 14-4: Solvent Extraction Flowsheet

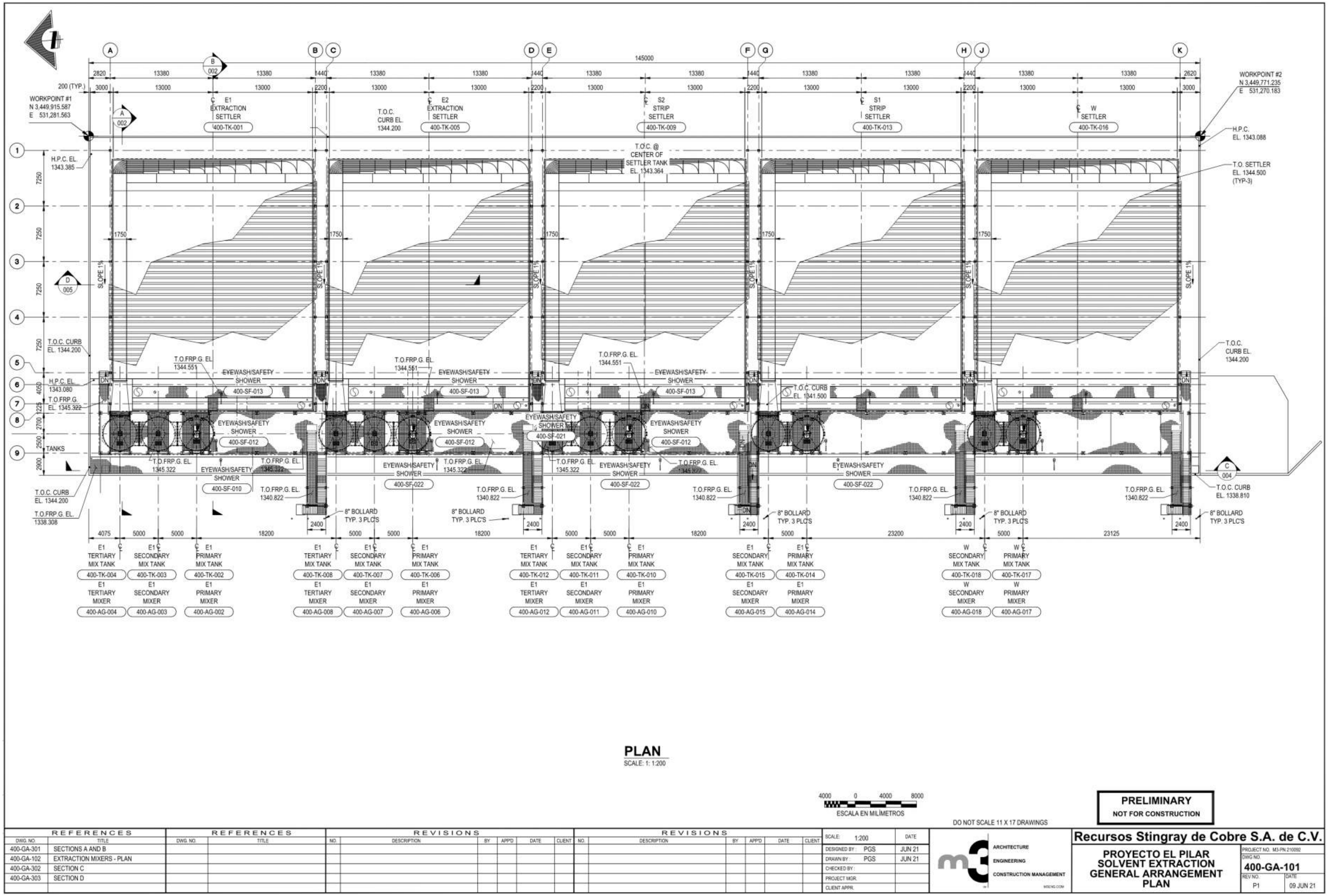


Figure 14-5: Solvent Extraction Plan

14.2.2.4 Electrowinning Plant

The electrowinning facility (EW) is located south of the tank farm and the SX plant. The plant will utilize the permanent cathode technology, with 92 cells, each containing 67 lead anodes and 66 stainless steel cathodes, with space provided for an additional 4 cells, two in each row. Located on the south end of the tankhouse building is the cathode washing and stripping machine. The tankhouse orientation allows the long axis of the building to be almost parallel to the prevailing winds.

The EW tankhouse cells are arranged in two parallel banks of 46 cells each, and are monolithic built, using reinforced polymer concrete. In the hydraulic circuit, all 92 cells are arranged in parallel allowing each cell to have the same feed solution and lean electrolyte discharge solution. In the electrical circuit, all 92 cells are arranged in series. The electrical current is supplied by 2 rectifiers connected in series, electrical current flows from the rectifiers through a bus bar to the bank of cells. Each cell is equipped with intercell bus bars, 66 cathode plates and 67 anode plates arranged in parallel. The cathodes are made of stainless steel and the anodes of rolled lead. Within each bank, direct electrical current flows from an intercell bus bar to the anode and then through the electrolyte to the cathode plates, to an intercell bus bar and onto the next cell successively and finally returns to the rectifiers.

Heated filtered rich electrolyte flows from the tank farm heat exchangers into the electrolyte recirculation tank where it mixes with overflow from the lean electrolyte tank. The solution from this tank is pumped to the tankhouse cells where copper in solution is plated onto the cathode plates.

As a result of the electrochemical reaction at the anode, oxygen evolves from the EW cells creating an acid mist. To control the mist generated, polypropylene balls and a surfactant are utilized. Cobalt sulfate is also added to passivate the anode, and guar (a bean powder) is added as a surface modifier for the cathode.

The major components of the electrowinning process are listed below in Table 14-2 and a graphical description of the process is shown in Figure 14-6:

- Electrolyte circulation tank
- Rectifiers
- Electrowinning cells
- Anodes and cathodes
- Cathode washing and stripping machine
- Reagents
- Overhead bridge crane
- EW cell ventilation system
- Utilities
- Shorting frame
- Anode/cathode refurbishment area.

EW process design criteria is shown in Table 14-2.

Table 14-2: EW Process Design Criteria

Parameter	Value	Unit
Annual copper cathode production	31,752	tonnes
EW plant availability	99	%
Cathode quality	ASTM B115-10 Grade 1	
Number of EW cells	92	
Cell voltage, design	2.3	V
Current density, design	364	A/m^2
Current efficiency	92	
Cell growth period	7	Days
Cathode bundle size	2-3	tonnes
Permanent cathodes, per cell	66	
Cathode plating area	1	m^2
Rectifier voltage, design	222	V
Rectifier output, design	44,000	Amps
Cathode stripping capacity	165	Cathodes per hour

14.2.3 Ancillary Facilities and Services

14.2.3.1 Fresh Water

Fresh water will be supplied to the Project from three wells located within the property. Fresh water from wells will be pumped into a freshwater booster tank, located near the wells. Water will then be pumped into a freshwater tank located at high ground. Water for services will flow by gravity to plant and process areas. Water for domestic services will be filtered for sediments and organic matter. Water for domestic services will not be potable. An extraction permit is in place to extract up to 3.5 Mm^3 of water per year. The total freshwater requirement for the Project is 1.75 Mm^3 of water per year. The water balance summary is provided in Table 14-3, and the water balance flowsheet is shown in Figure 14-7.

Table 14-3: Water Balance Summary

IN			OUT		
Stream	Description	t/h	Stream	Description	t/h
1	Rom Ore Moisture	52.0	109	Heap Retained Humidity	224.1
119	Fresh Water	199.5	89	Heap Evaporation	150.4
79	Precipitation Heap Leach	161.0	136	Domestic Water	5.0
78	Precipitation PLS Pond	0.9	145	PLS Pond Evaporation	4.1
88	Precipitation Raffinate Pond	0.6	139	Raffinate Pond Evaporation	2.4
			93	Road Dust Control	21.0
			-	Water Losses By Chemical RX	7.0
	Total	**414.0**		**Total**	**414.0**

14.2.3.2 Sulfuric Acid

Acid consumption for the El Pilar Project is estimated at 1,276 t/d (22 kg H_2SO_4/tonne ore), design capacity for sulfuric acid storage will be 1,350 t/d. Sulfuric acid with a purity of 98% will be supplied to Project via rail cars. An acid unloading station with two sulfuric acid tanks capable of each storing 2,363 tonnes of sulfuric acid and will be installed beside the rails. Sulfuric acid will be transferred pneumatically from the rail car into the tanks. Two additional sulfuric acid tanks of the same capacity will be located upstream of the process plant, acid will be transported using tanker trucks and unloaded by gravity into these two tanks.

14.2.3.3 Extractant

Extractant consumption for El Pilar is estimated at 0.56 m^3/d from entrainments and degradation. Design capacity for extractant storage tank will be 25 m^3 operating capacity, enough to hold a tanker truck's contents. Capacity for this tank storage is approximately 37 days of operation. Extractant will be pumped from the tanker truck into the tank.

14.2.3.4 Diluent

Kerosene will be used as diluent at El Pilar, consumption is estimated at 6.2 m^3/d. Design capacity for diluent will be 114 m^3 operating capacity, enough to hold 4-tanker truck's contents for 15 days of operation. Kerosene will be pumped from the tanker truck into the tank.

14.3 POWER CONSUMPTION

The process plant power consumption in a typical year (Year 6) is shown in Table 14-4, with a total consumption of 120.1 million kWh. This translates to about $0.13 per pound cathode produced or $0.39 per tonne ore.

Table 14-4: Process Plant Power Consumption

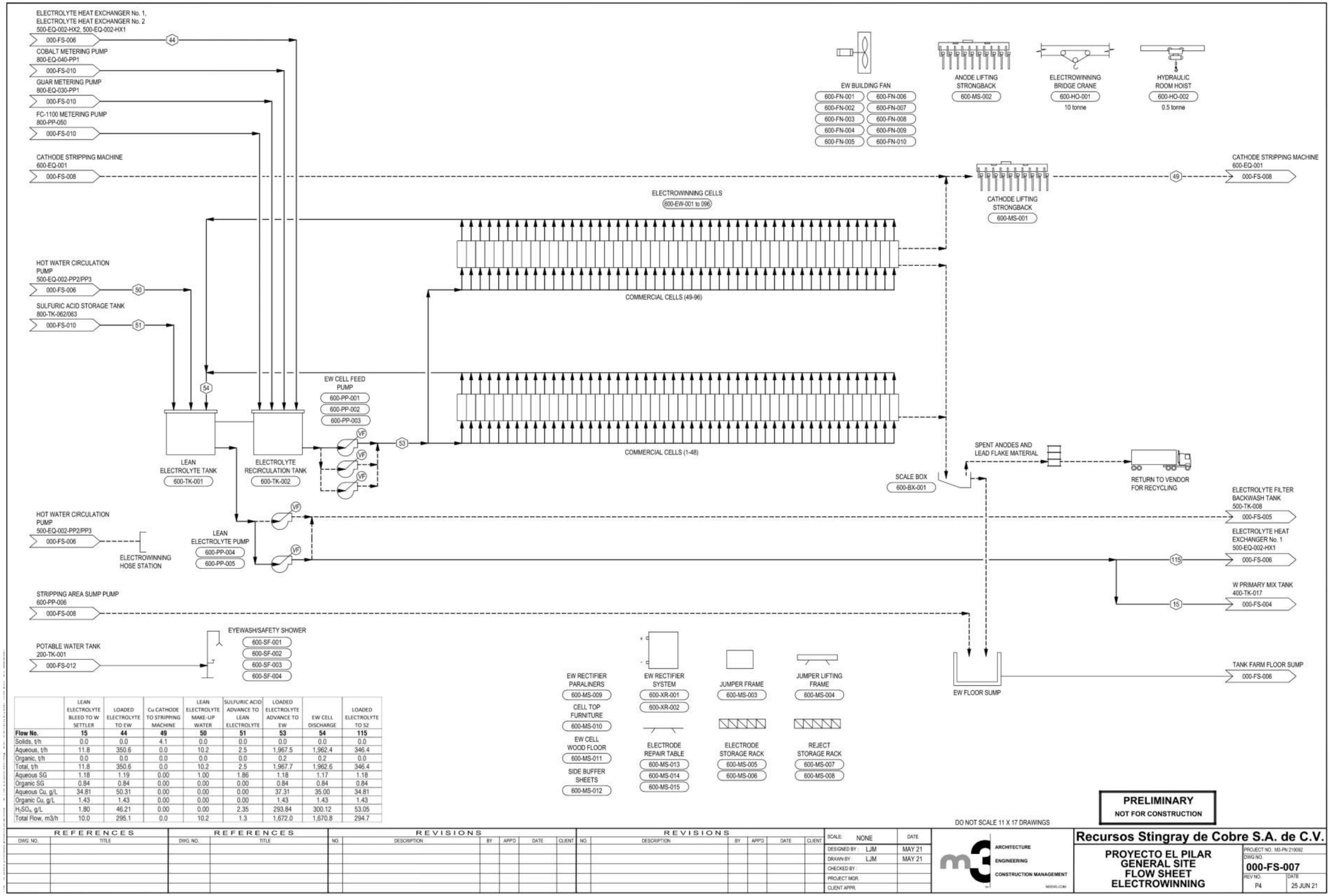
Area	Annual kWh	Annual Cost ($)
Heap Leaching	17,372,153	1,271,653
Solvent Extraction Plant	2,633,953	192,807
Tank Farm	2,403,667	175,950
Electrowinning	85,384,514	6,250,200
Water Systems	2,241,089	164,049
Sulfuric Acid Storage and Unloading	1,111,786	81,383
Ancillaries	8,930,820	653,742
Total	**120,077,983**	**8,789,784**

14.4 PERSONNEL

A total of 73 personnel will be required for operations, working three 8 hour shifts. A summary is shown in Table 14-5.

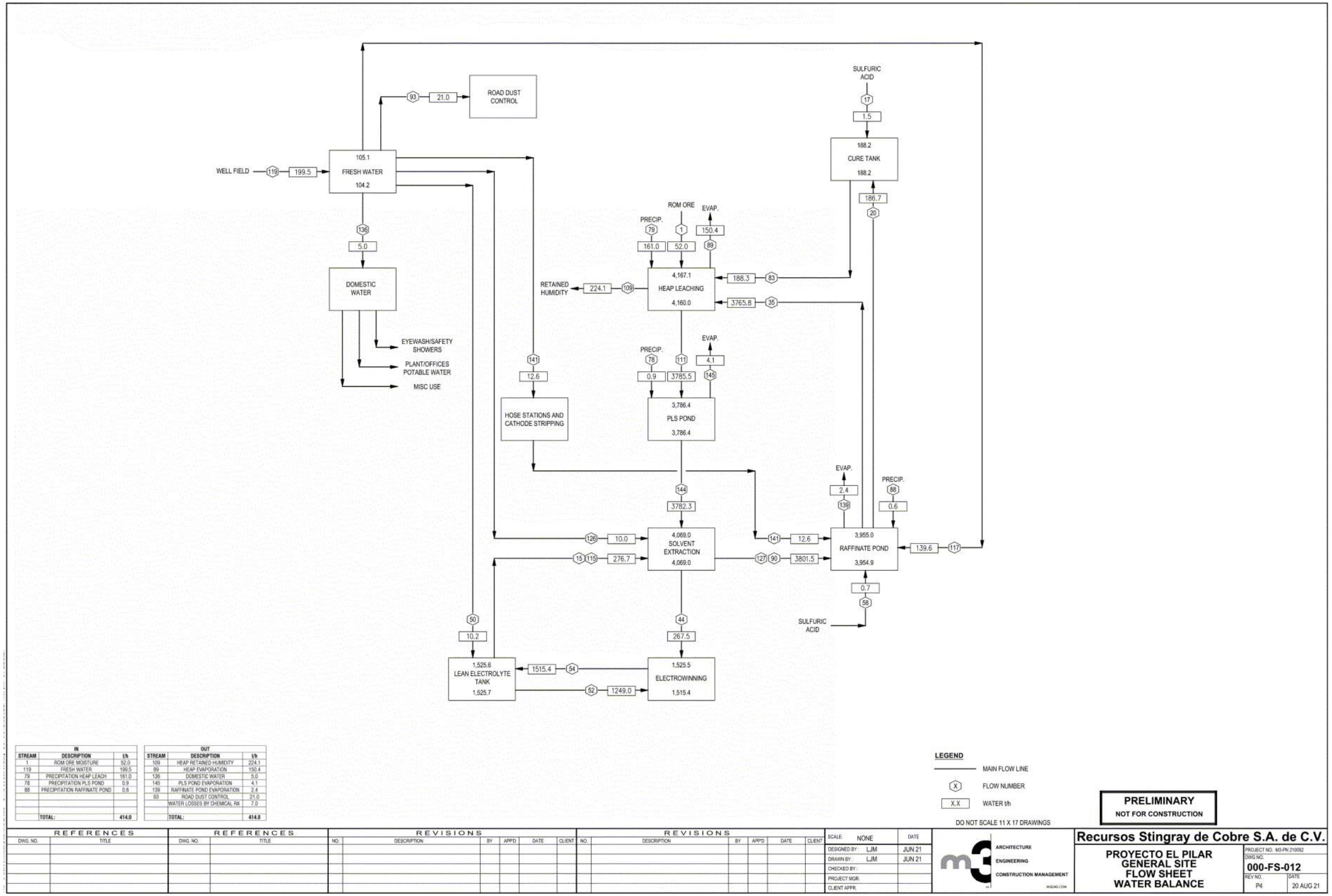
Table 14-5: Operating Personnel Summary

Area	Staff	Annual Cost ($)
Heap Leach and SX-EW Administration	5	341,880
Heap Leach	18	317,312
SX-EW	17	398,416
Maintenance	33	838,864
Total	**73**	**1,896,472**



	LEAN ELECTROLYTE BLEED TO W SETTLER	LOADED ELECTROLYTE TO EW	Cu CATHODE TO STRIPPING MACHINE	LEAN ELECTROLYTE MAKE-UP WATER	SULFURIC ACID ADVANCE TO LEAN ELECTROLYTE	LOADED ELECTROLYTE ADVANCE TO EW	EW CELL DISCHARGE	LOADED ELECTROLYTE TO S2
Flow No.	**15**	**44**	**49**	**50**	**51**	**53**	**54**	**115**
Solids, t/h	0.0	0.0	4.1	0.0	0.0	0.0	0.0	0.0
Aqueous, t/h	11.8	350.6	0.0	10.2	2.5	1,967.5	1,962.4	346.4
Organic, t/h	0.0	0.0	0.0	0.0	0.0	0.2	0.2	0.0
Total, t/h	11.8	350.6	0.0	10.2	2.5	1,967.7	1,962.6	346.4
Aqueous SG	1.18	1.19	0.00	1.00	1.86	1.18	1.17	1.18
Organic SG	0.84	0.84	0.00	0.00	0.00	0.84	0.84	0.84
Aqueous Cu, g/L	34.81	50.31	0.00	0.00	0.00	37.31	35.00	34.81
Organic Cu, g/L	1.43	1.43	0.00	0.00	0.00	1.43	1.43	1.43
H_2SO_4, g/L	1.80	46.21	0.00	0.00	2.35	293.84	300.12	53.05
Total Flow, m3/h	10.0	295.1	0.0	10.2	1.3	1,672.0	1,670.8	294.7

Figure 14-6: Electrowinning Flowsheet



IN		
STREAM	DESCRIPTION	t/h
1	ROM ORE MOISTURE	52.0
119	FRESH WATER	199.5
79	PRECIPITATION HEAP LEACH	161.0
78	PRECIPITATION PLS POND	0.9
88	PRECIPITATION RAFFINATE POND	0.6
	TOTAL:	414.0

OUT		
STREAM	DESCRIPTION	t/h
109	HEAP RETAINED HUMIDITY	224.1
89	HEAP EVAPORATION	150.4
136	DOMESTIC WATER	5.0
145	PLS POND EVAPORATION	4.1
139	RAFFINATE POND EVAPORATION	2.4
93	ROAD DUST CONTROL	21.0
	WATER LOSSES BY CHEMICAL RX	7.0
	TOTAL:	414.0

Figure 14-7: Water Balance Flowsheet

15 INFRASTRUCTURE

15.1 ACID UNLOADING STORAGE

The 1,350 t/d sulfuric acid requirement for the process plant will be delivered to the site by rail. Approximately 95 tank cars (100-tonne capacity) per week will be required. Sulfuric acid deliveries will occur in batches of 10-15 railcars.

The unloading system will be designed to accommodate 18 railcars. The railcars will be weighed full as they are received on the property and then positioned at the unloading station, where sulfuric acid will be pneumatically unloaded into one of two sulfuric acid tanks, each with a 2,300-tonne capacity.

Two additional sulfuric acid tanks will be located at plant level. Acid will be transported into these tanks by road, using tanker cars. Total sulfuric acid storage at plant site is around 9,400 tonnes, which is approximately 7 days of plant operation.

15.2 ACCESS

The El Pilar property can be reached by road from Hermosillo, Sonora, Mexico and from Tucson, Arizona, USA. From Hermosillo, the easiest access is via Hermosillo to Imuris (210 km) and Imuris to San Antonio (36 km), and from San Antonio to Miguel Hidalgo (San Lazaro) (35 km). Road access is by paved highway except for the final section of an all-season gravel road. The route from Hermosillo to Miguel Hidalgo takes about 3 1/2 hours of driving time. The route from Tucson to Miguel Hidalgo is currently a two hour drive and utilizes a paved road from Nogales, Sonora to Miguel Hidalgo (30 km). Access road is shown in Figure 15-1.

A main access road to the plant site from the main Nogales access road on the west side of the Project will be by way of a 6.5 Km long gravel road that will be constructed early in the project development schedule. The Project access road includes a crossing over the Santa Cruz River bed by way of a concrete dip, with hydraulic/drainage structures as required.

15.3 WATER

Based on the hydrological study conducted by IDEAS and the process water balance, the Project includes three water wells to supply necessary water for processing and services. The three wells have been drilled, cased and tested. This yields more than the 3.5 Mm^3/yr required. All wells are located within the property and are located a relatively short distance from the facilities (about 2.5 km). The location of the wells is shown in Figure 15-1.

15.4 POWER LINES

Power will be supplied to the Project area via a 115 KV transmission line from a substation located at 21 km south of Nogales, Mexico. The substation is 30 km west of the Project area. The substation is owned and operated by Comisión Federal de Electricidad (CFE), which has confirmed power availability and provided an area next to the substation for the installation of a switchyard and instrumentation. The power line will be 31 km long and built by dip galvanized structural steel towers, except for along urban areas where steel tapered poles will be used. The line will have the capacity to supply all of the power requirements for the Project. Power line trajectory into site is shown in Figure 15-1.

15.5 RAILROAD SPUR

As part of the infrastructure, a railroad spur will be constructed to the plant site. The purpose of the spur is to provide a safe, economic and efficient access to sulfuric acid deliveries by rail. The rail spur will access the property from the Ferromex rail system located on the west side of the property, about 3.8 km distance from the acid unloading station.

Rail facilities will allow for unloading and parking at least 18 railcars, with deliveries expected on a weekly basis. Railroad access is shown in Figure 15-1.

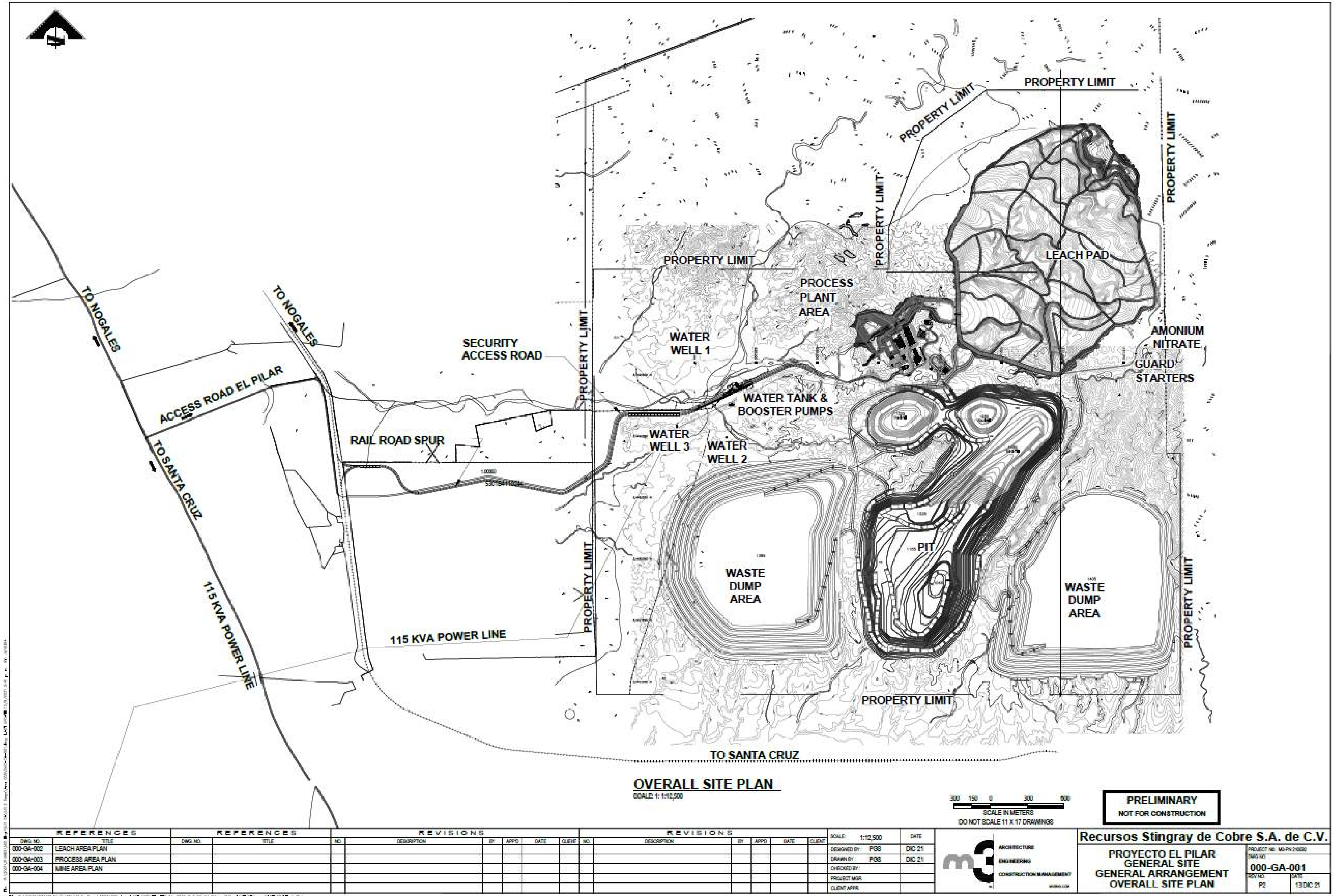


Figure 15-1: General Site Plan, 115 kVA Power Line, Railroad Spur and Access Road

16 MARKET STUDIES AND CONTRACTS

16.1 INTRODUCTION

El Pilar's copper cathodes production will meet the American Society for Testing and Materials (ASTM) standard specifications with designation B-115-10, COMEX Grade 1 and L.M.E. and SHFE Grade "A" copper cathodes. Therefore, the copper cathodes will sell with a premium cost per metric tonne.

Premiums have a high volatility on the spot market as well as each region sometimes has different premiums. It is in the best interest of the mining companies to settle an annual premium, since it provides support for internal production and sales costs that make the operation viable.

SCC maintains the corporate sales policy that premiums for Grade "A" copper cathodes are a key part for the settlement of this type of product businesses, which is negotiated directly by the corporate commercial department.

Over the last 20 years, shortage and surplus periods of copper cathodes have been observed in the worldwide market that have led to different scenarios for suppliers and consumers. In view of the international policies for sustainability of the planet and a greener production and lifestyle for all markets and users, it is safe to assume that the increase in demand, in the short- and long-term, will continue due to the long-term plans of companies and countries that will contribute to the steady growth of copper cathodes demand.

Table 16-1: Historical Prices for the Preceding Ten Years

Data Set	Unit	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020
LME Cash	$/t	7539.324	8810.902	7949.745	7322	6862.002	5497.968	4862.272	6165.601	6523.038	5999.733	6180.626
LME Cash	USDc/lb	341.9779	399.6557	360.5942	332.1203	311.2551	249.3836	220.5489	279.6669	295.88	272.1433	280.3484

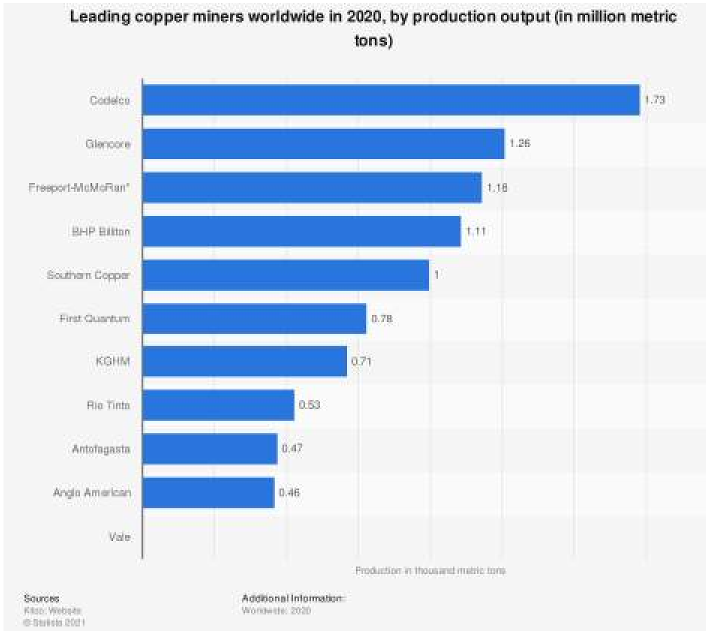
As demand for the copper cathodes will continue its growth in the short-, mid- and long-term, most copper market participants seek to increase their supply starting now. This guarantees not only a firm demand from enterprises but also a pressure from countries to achieve international agreements and a better lifestyle for their citizens in the future. One of the main supports for the growth of global copper cathode demand of 1.8% per year in the upcoming 25-year period will be the energy transition market.

Table 16-2: Copper Price Projections

	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Base Case - Global												
Copper Stock Days	69	63	66	72	79	81	75	73	71	69	67	65
Nominal $/t	6181	9338	8575	7450	6557	6449	6944	7331	7731	8145	8571	8877
Real $/t	6335	9338	8399	7154	6173	5952	6283	6504	6724	6945	7165	7275
Low Price Scenario - Global												
Copper Stock Days	69	66	69	78	85	87	81	77	74	70	68	65
Nominal $/t	6181	8818	8103	7117	6323	5972	6335	6834	7351	7886	8439	8877
Real $/t	6335	8818	7937	6834	5952	5512	5732	6063	6393	6724	7055	7275
High Price Scenario - Global												
Copper Stock Days	69	63	60	62	65	71	70	69	68	67	66	65
Nominal $/t	6181	9338	9904	9643	9367	9077	9015	8947	8872	8791	8835	8877
Real $/t	6335	9338	9700	9259	8818	8378	8157	7937	7716	7496	7385	7275

16.2 CATHODE SALES TERMS AND NET REALIZATIONS

Cathode sales from El Pilar Project will be negotiated directly by SCC´s corporate commercial area that currently has an important worldwide share in the copper market, commercializing 2.1% of the total copper cathodes in the world through a corporate strategy that obeys the presence in the market for several years due to long-term contracts with strategic business partners in the Asian and European markets, as well as annual contracts with other active market participants.



Note: Southern Copper is the fifth world copper producer

Figure 16-1: Worldwide Top Copper Producers

Based on SCC's current commercial strategies, sales terms will be similar to the following:

- Delivery: FCA Refinery. Buyer arranges and pays for cathode transportation.
- Price: COMEX average price during the Quotational Period plus a premium of 4.0 cents /lb. for rod mill quality ASTM B-115 Grade 1 quality material, with negotiated discounts for lesser quality material. This estimate is based on a rough weighted average freight and marketing costs (including merchant profit margins) serving the Asian, European, Brazilian and / or North American markets.
- Quotational Period: the previous month of shipment or month following month of shipment declarable by Buyer once or twice per year, depending on market conditions.
- Payment: 2 days after shipment date and upon confirmation of border crossing.
- Merchants provide useful services to the market. In recent years, consumers have shifted a larger share of their cathode requirements to them for three main reasons:
    1. There are still only a few consumers that are equipped and willing to buy imported cathode on a CIF Port basis because of insufficient or inadequately trained staff and/or a desire to avoid lengthy pipeline financing. Nor are they willing to take on the performance risk in dealing with relatively small independent cathode producers.
    2. Consumers use merchants as a way of managing inventory at relatively low cost – buying unanticipated requirements from them and selling them back excess stocks as required to minimize these costs.
    3. Some merchants are more flexible on payment terms than producers who now all sell on a net cash basis.

17 ENVIRONMENTAL STUDIES, PERMITTING, AND PLANS, NEGOTIATIONS, OR AGREEMENTS WITH LOCAL INDIVIDUALS OR GROUPS

17.1 GENERAL DESCRIPTION OF ENVIRONMENTAL CONDITIONS

17.1.1 Geomorphologic Conditions

Morphologically, the area encompasses modest hilly topography formed by erosion and weathering primarily of unconsolidated range-front sediments. The area is bounded to the west and south by the Santa Cruz River.

Landscape will be affected at first by clearing and grubbing, road construction and construction of mining facilities. Ultimately, impacts will be from the mine pit, waste dumps, and placement of ore on the heap leach pad. The effects of mining are irreversible, although some landscape effects are partially reversible in the long run through planned restoration and reforestation methods.

17.1.2 Soils

Most soils in the area are poorly developed C-horizon gravels. The project area shows moderate surface erosional degradation with some topsoil erosion due to over grazing activities.

In order to avoid adverse effects to the surface during mining, a series of prevention and mitigation measures will be taken, including proper waste collection and disposal, and machinery and equipment maintenance. In the closure stage, restoration activities will be carried out as defined in the Environmental Impact Manifest (MIA) permit. Ground quality will be monitored and a Ground Preservation Plan will be implemented.

In the El Pilar Project, sampling of soil and sediments strategically distributed within the limits of the mining project has been conducted, and the different sampling points are located downstream from where the Project's main infrastructure work will be located. In total, six sample collection sites are considered which are analyzed annually since 2012.

The processed data analysis of the sediment sampling results is in favorable condition as it does not record values above the maximum permissible limits established by the Norma Oficial Mexicana NOM-147-SEMARNAT/SSA1-2004, therefore, the El Pilar Project complies 100% with the parameters considered for the environmental quality characterization of the soil and sediment factor.

17.1.3 Surface Hydrology

The Santa Cruz River is the main hydrologic resource in the area, with permanent flow in the study area and intermittent ephemeral flow downstream of the project area. During the rainy season, wash runoff drains into the Santa Cruz River.

Three locations alongside the Santa Cruz River were sampled in April 2011 for water quality purposes; one location is upstream NW of the town of San Lázaro and two are downstream of the town. Results from one location show elevated values slightly above permissible limits (NOM-001-SEMARNAT-1996) in grease and oil content but all other locations and parameters are within permissible limits.

The planned surface preservation and mitigation measures are: impermeable retention areas where chemical substances or process solutions are handled, implementation of a hazardous and non-hazardous waste handling program, monitoring of surface water and creek sedimentation and water quality, and storm water diversion around disturbed areas where required.

As of the fourth quarter of 2011, the water quality of the Santa Cruz River continued to be monitored in three single and instantaneous sampling sites. These selected sites represent the three regulatory monitoring phases, upstream, midstream (northwest of the town of San Lázaro) and downstream, respectively. Although the sampling in these sites is dependant on the season in which the Santa Cruz River inflow presents a greater flow (rainy season), the Santa Cruz River tends to maintain a constant flow most of the time, being in few occasions the times that a null sample has been recorded when the river bed is dry. The sampling is carried out semi-annually, and the results are compared with the NOM-001-SEMARNAT-1996 that establishes the maximum permissible limits of pollutants of wastewater discharged in national waters and assets. Findings show that there is a high concentration of fecal coliforms within the Santa Cruz River inflow which is related to livestock activities and unauthorized discharges of wastewater (sewage) on the riverbed that, although they develop on the surface, are more vulnerable and susceptible to be receptors of all these factors that trigger a synergistic and cumulative concentration in surface waters where the values recorded during the years sampled exceed 500 MPN/100mL.

For the other parameters considered for surface water quality characterization, the concentration levels are within the maximum permissible limits determined by the regulations, taking into account that these concentrations are represented naturally and their variations are external consequences to those derived from the development process of the El Pilar Project. The Project is in a planning phase, and no industrialized process has been developed that may contribute possible adverse effects to the quality of the river.

The Santa Cruz River conditions regarding to the environmental quality can be determined as stable and in good condition, and that in turn, has as a history of the preservation of the indicators determined by each of the parameters of NOM-001-SEMARNAT-1996.

17.1.4 Subsurface Hydrology

In a hydrological study carried out by Rangel (2008), a total of 196 groundwater sources are considered to be available to the Santa Cruz River aquifer, providing an average annual extraction rate of 26.38 hm^3, mainly for public-urban and agriculture uses. An availability of 12.907 Mm^3 of groundwater per year was provided as a baseline for permitting purposes. The groundwater static level depth was determined to range between 1.99 m to 48.93 m with a 7.46 m, average. The shallowest depths of 10 m occur near the Santa Cruz River.

Groundwater sampling results show that water quality in the area is good, although one of the area wells sampled has iron concentrations above norm and three of them have total coliforms above the limits stated by NOM-127-SSA1-1994.

By December 2020, the Comisión Nacional del Agua (CNA) updated the average annual availability of water in the Santa Cruz River Aquifer, determining that there is an average annual extraction rate of 33.87 hm^3, mainly for public-urban and agricultural uses. An availability of 2.227 Mm^3 of groundwater per year was provided as a baseline for future permitting purposes.

According to the Federal Law on Water Rights 2020, the aquifer is classified as availability zone 3, in which the capacity of the aquifer allows limited withdrawals for domestic, industrial, irrigation and other uses. The main water user is the urban and agricultural public. The aquifer is not in an irrigation district, and the Groundwater Technical Committee has not been established to date.

Prevention and mitigation measures contemplated to protect groundwater quality include a waterproof layer in the leach pad, sumps and process areas, as well as installation of water monitoring wells below mining facilities with regular water quality monitoring.

As of the fourth quarter of 2011, a quarterly groundwater monitoring of nine wells located in and around the El Pilar mining complex began. Of all the sampled wells, there is a record of three wells owned by SCC, five private wells that are used for livestock production purposes (agriculture and livestock) and one well that is owned by Miguel Hidalgo Ejido located in the town of San Lázaro, which is used for supply and consumption by the ejido population.

The database collected during this time was compiled to determine the level of compliance per the environmental regulations applied to the water resource, using the Norma Oficial Mexicana NOM-127-SSA1-1994 as a reference which determines the maximum permissible limits of water quality for human use and consumption, as well as the treatments to purify the water.

Comparing the processed results of the sampled wells to the maximum permissible values determined by the aforementioned standard, certain outliers can be detected that are based on individual aspects and conditions or natural phenomena and/or effects derived from anthropogenic activities.

In general terms, the physical-chemical parameters considered in this analysis show a compliance of 88.46% with the standard, taking into account that of the 26 parameters analyzed, only three frequently present high values in each of the sampling campaigns (fecal coliforms, total coliforms and iron); which allows to determine that the behavior of these 3 parameters will continue to yield values above the maximum permissible.

17.1.5 Vegetation

The Project's surface area is covered mainly by grazing land and to a lesser degree by scrub oak and bushes comprised mostly of mesquite. Currently, the land is used for cattle grazing. Because the area is semi arid and not easily suitable for large scale agriculture, mining is a preferred development activity. An independent review of the area vegetation done for permitting purposes found that no protected species are present.

Actions that are planned to mitigate vegetation impacts include compensation payments to the forest fund for land use rights, organic topsoil recovery during clearing and reuse of this material in the closure phase, and implementation of a flora and fauna species protection program during all stages of the Project, including soil scarification and planting native species to restore the affected areas.

17.1.6 Fauna

An independent regional survey found 305 species of fauna, none of which have an impact on planned operations in the area.

17.2 WASTE MANAGEMENT, SITE MONITORING AND WATER

17.2.1 Waste Management

Waste generated during development and mining operations will be handled according to the provisions of the General Law for Prevention and Integrated Waste Management. A landfill will be built in the western part of the site to manage non-hazardous solid waste that cannot be recycled or reused, in compliance with NOM-083-SEMARNAT-2003.

Hazardous waste will be disposed of offsite in full compliance with NOM-052-SEMARNAT-2005, NOM-053-SEMARNAT-1995 and NOM-054-SEMARNAT-1995. Wastewater from sanitation services will be collected by a septic tank system constructed in compliance with NOM-006-CNA-1997.

Testing of waste rock by the Analítica del Noroeste lab analysis under NOM-141-SEMARNAT-2003 shows that the material is not an acid generator and thus requires no special provisions for handling and disposal.

17.2.2 Monitoring Requirements

Mexican laws require that mandatory monitoring programs to be implemented under the Environmental Protection Agency SEMARNAT. For El Pilar Project, the following monitoring programs have been established by SEMERNAT for the life of the mine (Table 17-1).

Table 17-1: Environmental Monitoring Program

Action	Criteria/ Variables to Consider	Applicable Norms	Monitoring Point	Frequency
Groundwater Quality Monitoring	Parameters stated by applicable standard	NOM-127-SSA1-1994 Compared with baseline	9 monitoring wells	Quarterly
Surface Water Quality Monitoring	In accordance with water quality criteria which depend on the use of the receiving body of water	NOM-001-SEMARNAT-1996 (consider quality criteria for aquatic life use on Santa Cruz River)	3 monitoring sites along Santa Cruz River	Biannual
Creek Sediment Quality Monitoring	Total metals (As, Cu, Ni, Cd, Pb, Au, Ag, Se, Hg, Cr)	Baseline conditions	6 sediment sampling sites	Annual
Air Quality Monitoring	SO_2, SO_3, H_2SO_4 fog particles	NOM-039-SEMARNAT-1993 NOM-043-SEMARNAT-1993	Smokestack emissions	Annual
Perimetral Noise	Decibels	NOM-081-SEMARNAT-1994	Project boundary towards Miguel Hidalgo ejido	Annual
Fauna Registry	Species and amount	Compensation commitment	All project areas	Biannual
Flora Species Rescue Records and Nursery Plant Production	% of survival, Amount and type of plants produced	Compensation/restoration commitment	Replanting and safeguard areas	Biannual
Soil	Collection and safeguard of organic soils. Application of Remediation techniques on polluted soils. Erosion control works.	Compensation commitment	Soil safeguard areas. Remediation sites. Roads and banks	Biannual
Cleared Surface and Restored/Reforested Registry	Surface (hectares)	Compensation/restoration commitment		Biannual or when needed

17.3 PROJECT PERMITTING REQUIREMENTS AND STATUS

17.3.1 Permitting Requirements

There are a number of environmental permits required for operation of the Project. Most of the mining regulations are at a federal level, but there are also a number regulated and approved at state and local level.

There are three SEMARNAT permits required prior to construction; MIA, CUS and RA. A construction permit is required from the local municipality and an archaeological release letter from the National Institute of Anthropology and History (INAH). An explosives permit is required from the Ministry of Defense before construction. The key permits and the stage at which they are required are summarized below in Table 17-2.

Table 17-2: Key Environmental Permits

Permit	Mining Stage	Agency
Environmental Impact Manifest - MIA	Construction/Operation/Abandonment	SEMARNAT
Land Use Change -CUS	Construction/Operation/Abandonment	SEMARNAT
Risk Analysis - RA	Construction/Operation	SEMARNAT
Construction Permit	Construction	Santa Cruz Municipality
Explosive & Storage Permits	Construction/Operation	SEDENA
Archaeological Release	Construction	INAH

Permit	Mining Stage	Agency
Water Use Concession	Construction/Operation/Abandonment	CNA
Water Discharge Permit	Operation	CNA
Unique License	Operation	SEMARNAT
Accident Prevention Plan	Operation	SEMARNAT

Environmental Impact Manifest (MIA) – Regulations within Mexico require that an Environmental Impact Manifest (*Manifiesto de Impacto Ambiental [MIA]*) be prepared by a third-party contractor for submittal to SEMARNAT. The MIA must include a detailed analysis of climate, air quality, water, soil, vegetation, wildlife, cultural resources and socio-economic impacts.

Risk Analysis (RA) – A second required permit is a Risk Analysis (RA) (*Análisis de Riesgo [AR]*). A study needs to be developed to obtain this permit. This study identifies potential environmental releases of hazardous substances and evaluates the risks in order to establish methods to prevent, respond to, and control environmental emergencies. In El Pilar Project, no hazardous substances will be used or processed, thus SEMARNAT will not need a Risk Analysis to be done for current project conditions.

Land Use Change (CUS) – The third permit is a Land Use Change Study *(Cambio de Uso de Suelo [CUS])*. In Mexico, all land has a designated use. The various areas comprising the project site are designated as forest land, cattle grazing, and agriculture. The CUS is a formal instrument for changing the designation to allow mining on these areas. The CUS study is based on the Forestry Law and its regulations. It requires that an evaluation be made of the existing conditions of the land, including a plant and wildlife study, an evaluation of the current and proposed use of the land and impacts on natural resources and an evaluation of the reclamation and revegetation plans. The establishment of agreements with all affected surface landowners is also required.

17.3.2 Permitting Status

Other approvals, licenses and permits required for various aspects of a mine development and their status are shown in Table 17-3.

Table 17-3: Permits Matrix

Permit	Agency	Date Required	Description / Comments	Status Completion Est.	Applied by
Environmental Impact Manifest (Mining) Particular modality	SEMARNAT Environmental and Risk Ministry SEMARNAT State Office	Prior to construction	Similar to an EIA in US NEPA terms. Requires an evaluation of baseline conditions and predicted effects with regards to air, water, soils, wildlife, plants, cultural resources and socioeconomic factors. Requires a discussion and evaluation of mitigation measures such as preventive strategies, control equipment, monitoring plans, and reclamation plans.	September 26, 2011 by agency	M3
Land Use Change (Mine)	SEMARNAT Forestry Resources SEMARNAT State Office	Prior to construction	This permit from SEMARNAT is required to change the use of land where such a change might have a serious adverse impact on soil or ecology. For example, stripping of vegetation in preparation for mine construction would require such a permit. The permit application process requires submittal of a justification for the change that takes into account not only the predicted effects on soil and ecology, but also the economic benefits that would arise should the change be permitted	January 30, 2012 by agency	M3
Archaeological release letter (Mining)	INAH (State offices)	Prior to construction	INAH will review project plans and inspect project area for historic and archaeological resources. Following inspection they will issue a clearance letter or advise on requirements for protection or recovery of resource.	Release delivered on April 29, 2011	M3
Environmental Impact Manifest (Access Road and Railroad)	SEMARNAT Environmental and Risk Ministry SEMARNAT State Office	Prior to construction	Similar to an EIA in US NEPA terms. Requires an evaluation of baseline conditions and predicted effects with regards to air, water, soils, wildlife, plants, cultural resources and socioeconomic factors. Requires a discussion and evaluation of mitigation measures such as preventive strategies, control equipment, monitoring plans, and reclamation plans.	September 21, 2012 by agency	M3
Land Use Change (Access Road and Railroad)	SEMARNAT Forestry Resources SEMARNAT State Office	Prior to construction	This permit from SEMARNAT is required to change the use of land where such a change might have a serious adverse impact on soil or ecology. For example, stripping of vegetation in preparation for mine construction would require such a permit. The permit application process requires submittal of a justification for the change that takes into account not only the predicted effects on soil and ecology, but also the economic benefits that would arise should the change be permitted.	October 10, 2012 by agency	M3

Permit	Agency	Date Required	Description / Comments	Status Completion Est.	Applied by
Environmental Impact Manifest (Power Line)	SEMARNAT Environmental and Risk Ministry SEMARNAT State Office Municipality FERROMEX SCT	Prior to construction	Similar to an EIA in US NEPA terms. Requires an evaluation of baseline conditions and predicted effects with regards to air, water, soils, wildlife, plants, cultural resources and socioeconomic factors. Requires a discussion and evaluation of mitigation measures. Also requires an evaluation of project alternatives. Ferromex, SCT and municipality approved crossings and routes within city limits. Power lines do not require a Risk Analysis.	May 19, 2015 by agency	M3
Land Use Change (Power Line)	SEMARNAT Forestry Resources SEMARNAT State Office FERROMEX Municipality SCT	Prior to construction	This permit from SEMARNAT is required to change the use of land where such a change might have a serious adverse impact on soil or ecology. The permit application process requires submittal of a justification for the change that takes into account not only the predicted effects on soil and ecology, but also the economic benefits that would arise should the change be permitted.	In process	
Archaeological release letter (Power Line)	INAH (State offices)	Prior to construction	INAH will review project plans and inspect project area for historic and archaeological resources. Following inspection they will issue a clearance letter or advise on requirements for protection or recovery of resource.	September 9, 2011	M3
Concession of Ground Water	SEMARNAT CNA	Prior to Construction	Allocation of surface water for beneficial use.	2.3 Mm^3: in final approval, Title No. 02SON150567 / 07FMGC13, date December 16, 2013 1.2 Mm^3 in final approval, Title No. 02SON151018 / date July 21, 2020	El Pilar
Use of Explosives (presented for evaluation)	Secretaria de la Defensa Nacional (SEDENA)	In order to buy, transport, store, or use explosives	Transactions are made in Mexico City and must comply with the following format: Letter of notification from the Governor of the State. Certificate of Security. Location map of powder magazines and accessories, with reference to the places where the explosives are used and stored in relation to human occupation. Relation of the type of explosives and amount to be used monthly. Legal documentation of the company.	November 17, 2020 by agency	El Pilar
Concession for the Extraction of Building Materials	SEMARNAT CNA	When it is required	For extraction of sand and gravel or other industrial minerals from federal zone adjacent to waterways.	Not yet required	El Pilar, M3 & Consultant
Permission to carry out Hydraulic Construction	SEMARNAT CNA	when it is required	For construction of structures such as bridges, canals, dams, etc. in a federal zone adjacent to waterways.	Not yet required	El Pilar, M3 & Consultant

17.4 RECLAMATION AND CLOSURE

17.4.1 Overview

In accordance with the general work schedule of the El Pilar Project, the abandonment phase will commence after Year 12 from the start of operations. As part of the permitting requirements, SCC will prepare a detailed Closure and

Reclamation Plan, which will be concurrently executed from the operation phase of the Project and will be completed in the abandonment phase.

Conditions of the final closure and reclamation plan will depend on land use after the mining operations. It is anticipated that designated uses will be one or a combination of the following:

- Natural habitat for wild flora and fauna.
- Land with potential for livestock activities.
- Sites with touristic-recreational potential (includes the banks of the Santa Cruz River).

General guidelines and criteria for closure and reclamation are described below.

17.4.2 Pit

Pit banks must be structurally stable and a barrier will be built around the pit for safety. If necessary, there will be designated viewpoints. Also, signs will be placed around the pit perimeter to warn about the area risks. Other important pit restoration activities will include diversion or channeling of rainwater towards pit, closing of access roads and scarification and cultivation of native seeds and plants. The northeastern portion of the pit will be filled partially with waste rock during the final mine phase.

17.4.3 Waste Rock Dumps

Waste rock dumps will remain onsite, and piles and banks will be smoothed to prevent ponding. Earthworks will be constructed, if necessary, to control and divert drainage routes towards natural creeks.

The waste dumps will be covered with recovered topsoil from the initial construction phase. Where needed and when operationally possible, land will be scarified and native species will be planted.

Acid-base accountability (ABA) testing results show that waste rock will not generate acid mine drainage and, therefore, is not classified as hazardous waste. This means that no isolation or special treatment during the closure stage will be required.

17.4.4 Leach Pad

Characteristics of the exhausted ore will be evaluated at the end of the mine life to define requirements of a leach pad restoration program. For baseline purposes, several acid-base accountability and toxicity tests were carried out with leached material subject to metallurgic tests. A composite of the material to be mined during the first years of operation and results show that the exhausted ore is not acid generating and thus is not characterized as hazardous waste. Nevertheless, annual composites will be extracted from the leach pads during the LOM to monitor exhausted ore behavior.

The following is the preliminary plan for leach pad reclamation:

- Once leaching is complete, the pad will be rinsed. The rinsing process will conclude when metal and metalloid concentrations at the ponds are below maximum limits defined in NOM-001-SEMARNAT-1996. Sampling must be done in accordance with this standard. Hydrogen ion potential (pH) in the rinse solution must be between pH 5 and 10. Equilibrium is reached when values remain the same over a sufficient period of time.
- Pond washing will take place commensurately with pad rinsing. Once this process is complete, prevention and control measures will be implemented to guarantee long term pond physical stability.
- In order to reduce wind and water erosion, reclamation work will be performed at the base and banks of the system, including:

   a. Bank reconfiguration will occur based on a bank stability analysis supported by geo-technical studies erosion.
   b. Construction of bank berms to reduce and channel wind and water erosion.
   c. Construction of waterworks within the perimeter of the pond in order to avoid inclusion of water from washes.
- The following safety measures will be taken for surface restoration:
   a. Cover pond surfaces with recovered soil, if applicable, or with materials that allow the proliferation of flora.
   b. Promote reforestation or re-vegetation with regional native species in order to guarantee succession and permanence with minimum preservation efforts.
   c. Create a geometric area with reduced visual effects.
- Pregnant and raffinate ponds will be reclaimed as follows:
   a. Filled with non-hazardous material where applicable.
   b. Surface water drainage capacity handled compliance with pre-mining levels.

17.4.5 Process Ponds

Remaining process pond solutions will be removed and unloaded in compliance with NOM-001-SEMARNAT-1996. Remnant sludge will be left at the bottom of ponds and then covered and buried with a plastic membrane. All ponds will be filled with waste from different mining phases. The land will then be scarified and prepared for seeding or planting with native species. Surface water drainage will be in compliance with pre-mining levels.

17.4.6 Process Plant and Service Facilities

Recovery plant cleaning, dismantling and closure will be carried out, as well as the recovery of equipment and material useful for the company or third parties.

All facilities will be dismantled or demolished. Major production equipment will be dismantled and sold at closure. Foundations will be removed and holes filled to restore natural topography where possible. The land will be scarified and native species planted and cultivated.

17.4.7 Roads

Some roads will be left to access main areas of the site to facilitate closure and monitoring. Internal roads will be leveled and scarified to promote local plant growth via the addition of topsoil and native species planting or cultivation with seeds.

17.4.8 Environmental Monitoring

Groundwater monitoring will be conducted for two years post closure using operation wells and surface water monitoring will be conducted on the main creeks and the Santa Cruz River.

17.4.9 Community Impact

The El Pilar Project is located within the town of San Lázaro (Miguel Hidalgo Ejido), in the municipality of Santa Cruz, in the State of Sonora. During the MIA permitting process, mine socioeconomic impacts were considered over a 20 km radius in which 16 communities are located consisting of 3,314 people. The independent review process included interviewing the entire population of the town of San Lázaro and a representative population from nearby towns of Santa Cruz and Mascarenas, and as well smaller communities within a 10 km radius. Most of the local population is concerned by the lack of employment, education and services in the area and 95% of the people interviewed indicated support for development of the El Pilar Project.

The El Pilar Project will offer over 220 potential temporary and permanent jobs to the region. Special effort will be taken during the Project to provide support to the local communities, including assisting with introducing and improving basic services and educational institutions.

It is of vital importance for SCC, El Pilar Project owner, to have a relationship with the population and society in the area.

SCC reached out to the community to express the intention of developing a mining project within private and ejido property land, that would provide a socio-economic well-being for the local and foreign population that boosts local economy. A lease and purchase scenario was proposed for the land that comprises the project's mineral reserve and surface for supplementary and support works. A land occupation agreement was achieved by the company. Consequently, a mutual benefit was generated as the surface owners would economically benefit from the sale of their lands, and in turn SCC benefits from having the property to develop the mining project.

Over the years, the company has fulfilled its social commitment with the community of the Miguel Hidalgo ejido, developing a series of activities and programs that allow maintaining a healthy and pleasant relationship with the residents of the ejido. The company maintains constant contact with the needs of the population and community, which in turn has the connection with those responsible for the Project to request economic and other support for community needs.

The main activities developed by SCC are focused on improvements in the supply of drinking water, modification and maintenance of educational centers and health centers, paving and maintenance of streets and main roads, as well as improvements in public lighting and drainage.

In addition, social activities and recreation for the Miguel Hidalgo ejido population is a main part of the contributions that SCC has been maintaining over the years, by mainly providing necessary financial resources per request of the people, and needed for the festivities and recreational activities that as a society are performed locally, such as; Mother's Day, Children's Day, Christmas festivities and posadas, national commemorations parades and festivities, national holidays, and sport activities such as volleyball, soccer, basketball and baseball tournaments.

The outreach from the people of the Miguel Hidalgo ejido to the company has been focused on the request for donations, whether economic or in kind, which range from monetary resources to remedy health situations and support for expensive medical treatments, as well as the request for donation of various items ranging from: sports equipment for schools, uniforms and modifications to sports facilities, equipment for the telesecundaria marching band, basic tools for preventive maintenance and cleaning, such as shovels, picks, rakes, dustpan, brooms and mops.

SCC's joint collaboration and contributions to improvements for the general well-being of the community has allowed positive growth in relationships and social ties in the area, generating a state of trust and reciprocity that is maintained to date. It is expected that these relationships will continue to strengthen with the development of all phases of the El Pilar Project, which will be reflected with the employment of a large portion of the population that resides at the Miguel Hidalgo ejido as well as at the surrounding towns and communities.

18 CAPITAL AND OPERATING COSTS

This section contains forward-looking information related to capital and operating cost estimates for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were set forth in this section including prevailing economic conditions continue such that unit costs are as estimated in constant (or real) dollar terms, projected labor and equipment productivity levels and that contingency is sufficient to account for changes in material factors or assumptions.

18.1 CAPITAL COSTS

18.1.1 Facilities Capital Cost

18.1.1.1 Facilities Initial Capital Costs

Facility capital cost estimates were developed independently by M3 Engineering. These costs include construction of the heap leach pad, building of the SX-EW, and costs for infrastructure requirements, including ancillary buildings, access road, rail spur, power line and water source and distribution system.

A contingency of 10% of the total contracted cost (Total Direct Field Cost + Indirects) was used. A first fill of reagents includes the filling of organic tanks, settlers, sulfuric acid tanks, diesel and reagents.

Table 18-1 is a tabulation of the facilities initial capital costs with a total of $261.8 M.

Table 18-1: Facilities Initial Capital Costs

DIRECT FIELD COST	
General Site	$22,233,382
Mine Truck Shop, Fuel, Lube, Truck Wash, etc.	$6,178,683
Water System	$2,958,917
Heap Leach	$60,415,692
Solvent Extraction	$24,150,722
Tank Farm	$11,686,103
Electrowinning	$39,789,392
Main Substation	$3,778,195
Internal Power Distribution Lines	$1,108,334
Power Line and Switch Substation	$9,914,903
Acid Storage	$3,927,478
Laboratory	$2,113,483
Offices and Warehouse Building	$2,317,695
Gate House	$230,079
Explosive Storage	$582,379
Freight	$5,832,403
TOTAL DIRECT FIELD COST	**$197,217,841**
INDIRECT FIELD COSTS & OTHER CONSTRUCTED COSTS	
Duties	$2,877,900
Contractor Mobilization Costs	$748,554
Construction Power, Construction & Utilities	$149,711
TOTAL CONSTRUCTED COST	**$200,994,007**
EPCM Cost	
EPCM & QC	$24,941,647
Site CM Facilities	$797,178
Vendor Commissioning and Spare Parts	$2,332,961
Leach Pad and Ponds, Studies and Engineering	$4,078,429
EPCM SUBTOTAL	**$32,150,216**
TOTAL DIRECT FIELD + EPCM COST	**$233,144,223**
Contingency (10%)	$23,314,422
First Fill	$5,370,000
GRAND TOTAL INITIAL CAPITAL COST ESTIMATE	***$261,828,645***

18.1.1.2 Facilities Mine Development/Prestripping Capital Costs

Earthworks for stockpile pads, ex-pit roads and laydowns have been included in general site costs and summarized in Table 18-1.

18.1.1.3 Facilities Sustaining Capital Costs

Facilities sustaining capital includes heap leach expansion and interlift liners. Earthworks, subdrains, geosynthetics, collection piping and collection drains are included in the estimates. A summary of the sustaining capital projection is shown in Table 18-2.

Table 18-2: Facilities Sustaining Capital Costs

Year	Sustaining Capital ($000)
1	$5,861
2	$12,440
3	$22,681
4	$15,467
5	$40,635
6	$27,484
7	
8	$25,830
9	
10	$58,936
11	
12	$15,272
13	
14	$5,596
15	
16	

18.1.1.4 Working Capital Costs

Working capital requirements are calculated on the basis of inventories and accounts receivable and payable by year. Working capital costs in Year 0 total $10.5 M.

18.1.2 Mine Capital Costs Summary

The estimated mine capital costs include the following items:

- Major mine equipment.
- Mine support equipment.

18.1.2.1 Major Mine Equipment Capital Costs

The initial cost of mine equipment is estimated to be $103.1 million in Year 0, as a pre-production year, which is apportioned into $96.2 million in Q1 and $6.9 million in Q2, which includes an 8% contingency. Sustaining capital costs, which includes acquisition of new equipment, and rebuild costs, are being considered after the pre-production year is $144.6 million which includes an 8% contingency.

The estimate is based on vendor quotes for all equipment as provided by Southern Copper Corporation (SCC). Assembly and delivery costs are included in the estimates. Table 18-3 shows the vendor quotes for the major equipment. All costs are subjected to Value Added Tax (VAT) of 16%, which are not included.

Table 18-3: Unit Costs for Major Mine Equipment

Equipment	Price ($,000)	Rebuild hours
Hydraulic Shovel: 34 m^3	$12,400	40,000
Wheel Loader: 20 m^3	$6,200	40,000
Haulage Truck: 180 tonnes	$3,200	40,000
Production Drill: 152-250 mm	$1,800	60,000
Track Dozer: 600 HP	$1,800	25,000
Motor Grader: 16 m	$1,100	30,000
Water Truck	$1,700	30,000
Wheeled Dozer: 560 Gross HP	$1,300	30,000

Note: Above Capital is inclusive of freight, assembly, and excludes required consumables (i.e., tires, ground engaging and other consumables).

18.1.2.2 Support Equipment Capital Costs

The list below shows the support equipment being considered in the cost model:

- Field Service Truck
- Tire Service Truck
- Fuel & Lube Trucks
- Bus
- Light Pickup Trucks
- Rough Terrain Crane
- Light Plants

The initial cost of mine support equipment is $6.4 million in Q1 Year 0. Sustaining capital cost, which accounts for acquisition of new equipment costs, is being considered after the pre-production year, which for support equipment is $15.8 million. All support equipment was assumed to be replaced at the end of their useful lives, not considering a rebuild cost.

18.1.2.3 Contingency

A contingency of 8% is included in the above capital cost estimates for major and support mine equipment in the cashflow.

18.1.2.4 Total Mine Equipment Capital Costs

Capital costs includes initial and sustaining costs and were estimated using current vendor quotes for major equipment. Estimates from recent projects were utilized for the support equipment. Total mine equipment cost are summarized in Table 18-4, which includes replacement and rebuild costs. The costs are exclusive of contingency.

Table 18-4: Major and Support Mine Equipment Cost Summary (US$000's)

		Total Major Equipment Costs	Total Support Equipment Costs	Total
Initial Capital	PP Q1	82,693	6,370	89,063
	PP Q2	6,372	-	6,372
	PP Q3	-	-	-
	PP Q4	-	-	-
Sustaining Capital	Y1 Q1	6,982	20	7,002
	Y1 Q2	-	-	-
	Y1 Q3	-	-	-
	Y1 Q4	-	-	-
	Y2	3,362	-	3,362
	Y3	16,810	1,680	18,490
	Y4	3,157	-	3,157
	Y5	16,114	2,100	18,214
	Y6	19,649	-	19,649
	Y7	3,821	2,180	6,001
	Y8	26,762	-	26,762
	Y9	3,896	500	4,396
	Y10	2,091	1,680	3,771
	Y11	1,029	-	1,029
	Y12	454	-	454
	Y13	15,631	6,000	21,631
	Y14	-	-	-
	Y15	-	-	-
	Total	208,823	20,530	229,353

The capital costs were based on vendor quotes and hence assumed to be to a Class 4 estimate as defined by AACE International Recommended Practice No. 47R-11 "Cost Estimation Classification System – as Applied in the Mining and Mineral Processing Industry." Under this classification, capital cost have been developed to a Prefeasibility Level of accuracy suitable for the development of the Mineral Reserves Estimate.

It should be noted that, at the time of this report, Mexico has been experiencing varying rates of inflation, which introduces uncertainty in forecasting the capital costs. This uncertainty is considered a risk to the capital cost estimates. There may also be a risk associated to the time require to secure delivery of the major mining equipment.

18.2 OPERATING COSTS

18.2.1 Introduction

Operating costs used for the economic analysis are based on a robust engineering cost model developed specifically for the El Pilar Project. This model calculates and projects all project costs using equipment and personnel requirements and costs, reagent and power costs and consumptions, haul profiles for mining, ancillary costs for laboratory assaying and administrative costs and requirements. Details of the cost analyses results are summarized in the flowing subsections.

Mine Operating costs are calculated on the basis of all costs required for drilling and blasting, loading and hauling, dumps and roads and mine supervision. Fuel costs are based on using a constant diesel price of $0.87 per liter.

Personnel costs are based on a 24-7 schedule, with three eight hour shifts per day. Labor rates are based on estimates provided by SCC and include a 37.6% burden for benefits. Equipment requirements and other details that pertain to the development of the mining cost model are summarized in Section 13.5.

Plant operation costs are calculated on basis of operating and maintenance labor, reagents required for operation, power consumption, maintenance parts and supplies and services charges, including water consumption.

18.2.2 Mine Operating Costs

Mine operating costs are estimated from a zero-based model. Costs for each major equipment was developed based on the assumed material movement, equipment productivities and required support equipment. The unit equipment costs were estimated using estimated hourly consumption of fuel, lube, tires, undercarriage, repair and replacement parts, and wear parts, among others. Costs were estimated for each mining unit operation including drilling and blasting, loading and hauling, and maintenance and support, and general mine and engineering. Major mining equipment included 34-m^3 hydraulic shovels, 20-m^3 loader and 180-tonne haul trucks. Table 18-5 summarizes some of the general mine equipment operating cost assumptions.

Table 18-5: Equipment Operating Cost Assumptions

Parameter	Value
Number of Crews	4 crews
Shifts per Day	3
Fuel Price	$0.87/liter

Labor rates are based on estimates provided by SCC and include a 37.6% burden for benefits.

The sections below refer to the unit mining operations and estimated costs.

18.2.2.1 Drill and Blast

A summary of the drilling and blasting assumptions used in the cost estimate are summarized in Table 18-6.

Table 18-6: Drilling and Blasting Assumptions

Parameter	Unit	Value
Drill Productvity	m/hr	28.1
Time for Moving	min	6
Booster Unit Price	US$	4.30
Powder Factor	kg/bcm	0.23
Boosters per Hole	#	1
ANFO Price	US$/kg	0.66

The average LOM drill and blast unit cost was $0.14/tonne.

18.2.2.2 Loading

Shovels were assumed to have a mechanical availability of 85% and an operational usage of 67.4% were determined to have an estimated annual production capacity of 20.2 Mtpy, when working in overburden material. When in rock or mineralized material, the production rate was determined to be 23.1 Mtpy.

Loaders were assumed to have a mechanical availability of 85% and an operational usage of 69.9% resulting in an estimated annual production capacity of 9.1 Mtpy when in overburden material. In rock or mineralized material, the production rate was estimated to be 10.1 Mtpy.

Table 18-7 shows key items used in determining shovel and loader productivities. The average LOM loading unit cost was $0.26/tonne.

Table 18-7: Loading Assumptions

Parameter	Shovel	Loader
Bucket Capacity (bcm)	34	20
No. of Passes	4	6
Cycle Time secs (Overburden)	40	60
Cycle Time secs (Waste/Mineralized Material)	35	55
Truck Capacity ROM and OSF (Tonnes)	179.5	178.2

18.2.2.3 Haulage

Table 18-8 summarizes the haulage assumptions.

Table 18-8: Haulage Assumptions

Parameter	Value
Mechanical Availability	85.0%
Utilization	87.5%
Delays	10.0%
Maximum Total Hours per Year per Truck	5,870

Deswik LHS was used to estimate the required haulage hours. Deswik uses the centerlines of haul routes and creates a distance from the center of a block in the pit to the center of a block on the leach pad or OSF to calculate a distance. Speed limits were assumed from the previous report to be, as shown in Table 18-9.

Table 18-9: Truck Speed Limits

Truck Speed					
Loaded			Empty		
Up	Flat	Down	Up	Flat	Down
12	44	29	26	54	47

Cycle times were then calculated using the distances and the speeds. The fuel was calculated in Deswik LHS using fuel consumption rates from manufacturer specifications based on speeds and grades. The required engine load for each segment of each haul route is determined using the rimpull and retard graphs.

The total fuel consumption per year was developed in Deswik scheduler and applied to the cost model, with an average of 187 liter/hr. The maximum number of trucks was 25, and it was reached in Year 8. As mentioned in Table 18-5, a fuel price of $0.87/liter was assumed. The average LOM haulage cost was estimated to be $0.75/tonne.

18.2.2.4 Support Equipment

Hours per shift for the support equipment were derived from major equipment usage. For example, track dozer hours were estimated based on a percentage of the shovel and loader hours. The average support cost was estimated to be $0.18/tonne.

18.2.2.5 Total LOM Operating Costs

LOM operating costs are summarized in Table 18-11. A detailed bottom-up unit cost model was constructed to estimate the operating costs for the LOM. The production schedule was imported into the cost model production, hauling, drilling and support equipment hours were considered. Annual operating costs were then estimated by applying the unit hourly operating costs for each equipment in addition to supplies such as ANFO, fuel and electricity. The LOM summary operating cost is shown in Table 18-11. The average LOM mining cost is $1.39 per tonne total material, with projected costs per year, as shown in Table 18-10. The overall average LOM cost is based on the total LOM rock moved and it does not include the 10-cm pre-stripping, throughout approximately 850,000 m^3, since part of it is outside the pit.

Table 18-10: Mining Operating Cost per Unit (Total Material)

Total Cost	Unit	Value
General Mine & Engineering	US$/t	$0.02
Drilling	US$/t	$0.05
Loading	US$/t	$0.26
Haulage	US$/t	$0.75
Maintenance	US$/t	$0.04
Blasting	US$/t	$0.09
Support	US$/t	$0.18
Total Cost	**US$/t**	**$1.39**

Table 18-11: Average LOM Unit Mining Operating Cost LOM

Period	Total Tonnes Moved	Avg Unit Cost (US$/t)
PP Q1	8,008	1.31
PP Q2	8,792	1.24
PP Q3	8,666	1.25
PP Q4	8,582	1.25
Y1 Q1	10,171	1.18
Y1 Q2	10,137	1.19
Y1 Q3	9,830	1.21
Y1 Q4	8,738	1.27
Y2	38,562	1.22
Y3	49,805	1.20
Y4	53,957	1.15
Y5	56,327	1.23
Y6	53,892	1.32
Y7	48,522	1.40
Y8	51,315	1.40
Y9	48,603	1.46
Y10	56,327	1.31
Y11	49,021	1.42
Y12	45,665	1.53
Y13	36,046	1.80
Y14	28,299	2.05
Y15	13,054	2.17
Total/Average	**702,321**	**1.39**

The operating costs were based on operating costs from SCC affiliate operations and an internal library of projects. It is, therefore, assumed to be to a Class 4 estimate as defined by AACE International Recommended Practice No. 47R-11 "Cost Estimation Classification System – as Applied in the Mining and Mineral Processing Industry." Under this classification operating costs have been developed to a Prefeasibility Level of accuracy suitable for the development of the Mineral Reserves Estimate.

18.2.3 Processing Operating Costs

A power line will be built to provide power to the Project. The price for grid power from CFE is $0.073 per KWh. This grid power cost is based on power costs currently being assessed by CFE in northern Mexico.

Personnel required for heap leaching and SX-EW process average 73 over the LOM. Labor rates are based on best estimates of labor rates in Mexico, which includes burden of benefits.

The reagent costs and consumptions used for leaching and for the SX-EW plant are detailed in the Processing section of this report. Acid consumption for the economic model assumes the 23.7 kg/tonne average acid consumption over the LOM, as discussed in the metallurgical section. Sulfuric acid will be delivered to site via railroad. SCC purchases sulfuric acid at a rate of $55 per kg.

An allowance for process maintenance of 5% of capital equipment cost was used in estimate.

The average processing cost per tonne of ore over the LOM is $0.71/tonne, with projected costs per year tabulated below in Table 18-12.

Table 18-12: Average LOM Processing Operating Cost

Period	Unit Cost ($/lb Cu)
Y1	$0.78
Y2	$0.48
Y3	$0.67
Y4	$0.70
Y5	$0.66
Y6	$0.71
Y7	$0.70
Y8	$0.64
Y9	$0.64
Y10	$0.78
Y11	$0.90
Y12	$0.85
Y13	$0.81
Y14	$0.87
Y15	$0.75
Y16	$0.66
Average LOM	**$0.71**

A breakdown of main operating costs is shown in Table 18-13.

Table 18-13: Average LOM operating cost breakdown

Description	% Cost	$/lb Cu
Operating & Maintenance Labor	4.5%	$0.03
Sulfuric Acid	61.6%	$0.44
Reagents	6.9%	$0.05
Power	19.0%	$0.14
Maintenance Parts	5.1%	$0.04
Water Charges	1.2%	$0.01
Supplies and Services	1.7%	$0.01
Total/Average	**100.0%**	**$0.71**

18.2.4 Administrative (G&A) Operating Costs

Administrative costs assume an average of 33 personnel over the LOM and labor rates are based on best estimates of labor rates in Mexico, including burden of benefits. Other administrative costs include the costs associated with a laboratory, including sample preparation and analysis, and other costs for insurance, license, fees and permits and for property taxes, including all other costs related to mine administration.

The average administrative cost projected for the LOM is $0.09 per ore tonne.

18.2.5 Cathode Transportation and Royalty Operating Costs

Costs for the transportation of copper cathode is by buyer, FCA (Free Carrier) incoterms will be used.

As part of a previous agreement between Xstrata and SCC, a 1% of gross revenues royalty is owed to Xstrata from El Pilar production. This royalty amounts to an additional cost of $0.03 per pound of copper, or $0.09 per tonne ore average.

18.2.6 Cost per Pound of Copper Production

Total cost per pound of copper production is shown in Table 18-14.

Table 18-14: Cost per lb of Copper Production

Description	Operating cost ($/lb Cu)
Mining	$1.04
Process Plant	$0.71
G&A	$0.09
Royalty	$0.03
Total	**$1.87**

19 ECONOMIC ANALYSIS

19.1 INTRODUCTION

The financial evaluation presents the determination of the Net Present Value (NPV), payback period (time in years to recapture the initial capital investment), and the Internal Rate of Return (IRR) for the Project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures and production cost and sales revenue. The sales revenue is based on the production of copper cathode. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report. All amounts are in US dollars ($).

The economic analysis is done on an unlevered or full equity basis. Actual project financing, may include a single financing option or a combination of different financing instruments. Any acquisition cost or expenditures prior to the full project production decision have been treated as “sunk” cost and have not been included in the analysis.

19.2 MINE PRODUCTION STATISTICS

Mine production is reported as ore and waste material from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report.

The life of mine ore, waste quantities, and ore grade are presented in the Table 19-1.

Table 19-1: Life of Mine Ore, Waste Quantities, and Ore Grade

	Quantities (kt)		Ore grades (%)		
	Ore	Waste	Total	Soluble	Recoverable
Year 1	12,556	21,492	0.292%	0.217%	69.1%
Year 2	14,736	24,140	0.317%	0.225%	67.1%
Year 3	20,843	17,720	0.228%	0.147%	63.5%
Year 4	21,469	28,336	0.230%	0.133%	58.8%
Year 5	20,280	33,677	0.262%	0.144%	57.6%
Year 6	22,725	33,602	0.242%	0.126%	55.1%
Year 7	22,303	31,589	0.236%	0.130%	57.2%
Year 8	18,891	29,631	0.274%	0.152%	57.5%
Year 9	16,894	34,421	0.289%	0.152%	55.8%
Year 10	20,008	28,595	0.238%	0.109%	50.5%
Year 11	20,851	35,476	0.190%	0.093%	51.7%
Year 12	23,791	25,230	0.218%	0.102%	50.5%
Year 13	19,809	25,855	0.255%	0.101%	45.3%
Year 14	26,814	9,233	0.229%	0.089%	45.2%
Year 15	24,534	3,765	0.282%	0.106%	44.5%
Year 16	10,960	2,095	0.258%	0.078%	37.0%

19.3 PLANT PRODUCTION STATISTICS

The oxide ore will be processed using heap leaching and SX-EW plant recovery technology to produce copper cathode. In the current plan, the oxide ore is being leached starting one month after it is mined and stacked and the recovery of copper in the economic model is scheduled over a 120 day leach cycle according to the recoveries shown in Table 19-2. The average overall recoveries of copper are expected to be 53.8% of TCu.

Table 19-2: 120-Day Leach Copper Recovery by Period

Leach Recovery by Period	
Days Leached	Adjusted Recovery (to TCu)
30	65%
60	20%
90	10%
120	5%

The estimated cathode production for the life of the mine is 940 million pounds of copper cathode.

19.4 CAPITAL EXPENDITURE

19.4.1 Initial Facility and Mining Equipment Capital

The cash flow for the new construction is shown being expended in the pre-production year (PP). Mining initial capital includes the purchase of shovels, haul trucks, loaders, dozers, graders and drills, as well as service trucks. Initial capital for process plant includes process plant, heap leach facility, site preparation, EPCM contracts, QA/QC, access and internal roads and ancillary facilities. The total capital in the financial model for mining equipment is $103.1 million, whereas the initial capital cost for process facilities is $261.8 million. An 8% contingency is included in mining equipment cost and a 10% contingency is included in the process facilities initial capital cost.

In the financial model, 100% of initial capital cost for mine equipment is applied at PP. While initial capital expenditures for process plant is deferred 40% two years prior to production, 50% at PP and 10% in Year 1.

The financial indicators have been determined on the basis of 100% equity financing of the initial capital.

19.4.2 Sustaining Capital

Sustaining capital is considered to be any facility capital required after build-out in PP, the preproduction period. The total life of mine sustaining capital is estimated to be $144.6 million for mine operation and $230.2 million mainly for heap leach future expansions and interlif liners for the heap leach. Initial and sustaining capital is shown in Section 18.1.

19.4.3 Working Capital

Operating cost expenditures accrued for working capital purposes include the first six months of operations. Expenditures for working capital are defined as commencing when the first ore is placed on the heap leach pad. The total life of mine working capital is estimated on the basis of inventory and accounts receivable and payable and is adjusted on this basis by year. Initial working capital required in PP totals $10.5 M, based on an allowance of 5% of capital equipment cost used for purchase of spare parts.

19.5 REVENUE AND COPPER PRICES

Annual revenue is determined by applying a copper price to the annual payable metal for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The copper price used for the Base Case evaluation is a fixed price of $3.30/lb copper, provided by internal projections from Southern Copper Corporation (SCC).

19.6 TOTAL CASH OPERATING COST

The average Total Cash Operating Cost over the life of the mine is estimated to be $1.84 per copper pound. The Total Cash Operating Cost includes mine operations, process operations, and general administrative cost.

19.6.1 Mine

Mine operating cost was based on a detailed estimate previously discussed in Section 18.2.2.

19.6.2 Process

The SX-EW operating costs were based on a detailed estimate previously discussed in Section 18.2.3.

19.6.3 Cathode Shipping and Royalty

Cathode shipping terms will be arranged as FCA, buyer will pay for shipping. As part of a previous agreement between Xstrata and SCC, a 1% of gross revenues royalty is owed to Xstrata from El Pilar production. This royalty amounts to an additional cost of $0.03 per pound of copper.

19.6.4 Property Tax

An allowance of $300,000 per year is included in the Total Cash Operating Cost as an administrative (G&A) cost.

19.7 TOTAL CASH COST

The average Total Cash Cost over the life of the mine is estimated to be $1.87 per copper pound. Total Cash Cost is the Total Cash Operating Cost, plus reclamation and closure expense, royalty charges, and salvage sales.

19.7.1 Reclamation & Closure

An allowance of $5.1 million for the cost of final reclamation and closure of the property has been included in the cash flow projection; continual early reclamation is done throughout the life of the mine and cost have included for such, e.g. borrow pits.

19.7.2 Salvage Value

An allowance was made for the salvage value of assets of $10.9 million. This represents 3% of the total LOM constructed cost and mine capital equipment cost.

19.8 DEPRECIATION

Depreciation is calculated using the Straight Line method starting with first year of production. All assets are depreciated by the end of the last year of production. Other forms of depreciation are permissible in Mexico.

19.9 TAXES

Taxes are calculated based on Mexico tax law, which assumes a 30% income tax rate (after depreciation) and a 7.5% mining royalty tax applied to operating income (before depreciation).

19.10 FINANCIAL MODEL

19.10.1 Financial Model

The financial model for the El Pilar Project is shown in Table 19-3 and Table 19-4.

Table 19-3: El Pilar Financial Model

		PP																			
	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Oxide Ore																					
Beginning Inventory(kt)	317,465	317,465	317,465	317,465	304,908	290,172	269,329	247,860	227,580	204,855	182,552	163,661	146,766	126,759	105,907	82,116	62,307	35,493	10,960	(0)	(0)
Mined (kt)	317,465	-	-	12,556	14,736	20,843	21,469	20,280	22,725	22,303	18,891	16,894	20,008	20,851	23,791	19,809	26,814	24,534	10,960	-	-
Ending Inventory (kt)	-	317,465	317,465	304,908	290,172	269,329	247,860	227,580	204,855	182,552	163,661	146,766	126,759	105,907	82,116	62,307	35,493	10,960	(0)	(0)	(0)
Copper Grade (%)	0.249%	0.000%	0.000%	0.292%	0.317%	0.228%	0.230%	0.262%	0.242%	0.236%	0.274%	0.289%	0.238%	0.190%	0.218%	0.255%	0.229%	0.282%	0.258%	0.000%	0.000%
Contained Copper (klbs)	1,742,463	-	-	80,793	102,843	104,825	109,098	117,128	121,256	116,224	114,308	107,540	105,088	87,466	114,408	111,382	135,342	152,533	62,229	-	-
Waste																					
Beginning Inventory(kt)	384,857	384,857	384,857	384,857	363,365	339,225	321,506	293,170	259,492	225,890	194,301	164,670	130,249	101,654	66,178	40,948	15,093	5,860	2,095	(0)	(0)
Mined (kt)	384,857	-	-	21,492	24,140	17,720	28,336	33,677	33,602	31,589	29,631	34,421	28,595	35,476	25,230	25,855	9,233	3,765	2,095	-	-
Ending Inventory (kt)	-	384,857	384,857	363,365	339,225	321,506	293,170	259,492	225,890	194,301	164,670	130,249	101,654	66,178	40,948	15,093	5,860	2,095	(0)	(0)	(0)
Total Material Mined (kt)	**702,321**	**-**	**-**	**34,048**	**38,876**	**38,562**	**49,805**	**53,957**	**56,327**	**53,892**	**48,522**	**51,315**	**48,603**	**56,327**	**49,021**	**45,665**	**36,046**	**28,299**	**13,054**	**-**	**-**
Waste to Ore Ratio	1.21	-	-	1.71	1.64	0.85	1.32	1.66	1.48	1.42	1.57	2.04	1.43	1.70	1.06	1.31	0.34	0.15	0.19	-	-
PROCESS PLANT OPERATIONS																					
Leach Pad Stacking																					
Mined Ore to Heap Leach (kt)	317,465	-	-	12,556	14,736	20,843	21,469	20,280	22,725	22,303	18,891	16,894	20,008	20,851	23,791	19,809	26,814	24,534	10,960	-	-
Copper Grade Processed (%)	0.249%	0.000%	0.000%	0.292%	0.317%	0.228%	0.230%	0.262%	0.242%	0.236%	0.274%	0.289%	0.238%	0.190%	0.218%	0.255%	0.229%	0.282%	0.258%	0.000%	0.000%
Contained Copper (klbs)	1,742,463	-	-	80,793	102,843	104,825	109,098	117,128	121,256	116,224	114,308	107,540	105,088	87,466	114,408	111,382	135,342	152,533	62,229	-	-
Recovery Copper (%)	54.0%	0.0%	0.0%	69.0%	67.3%	63.5%	58.8%	57.6%	55.1%	57.2%	57.5%	55.8%	50.5%	51.7%	50.5%	45.3%	45.2%	44.5%	37.0%	0.0%	0.0%
Recovered Copper Cathode (klbs)	**940,777**	**-**		**55,777**	**69,170**	**66,512**	**64,126**	**67,479**	**66,767**	**66,528**	**65,729**	**59,970**	**53,073**	**45,226**	**57,728**	**50,511**	**61,236**	**67,952**	**22,995**	**-**	**-**
Cathode Production Schedule	**940,777**		**-**	**35,303**	**80,606**	**66,959**	**64,434**	**67,046**	**66,859**	**66,559**	**65,832**	**60,714**	**53,963**	**46,239**	**56,113**	**51,443**	**59,851**	**67,085**	**31,772**	**-**	**-**
Payable Metals																					
Copper Payable Metal (klbs)	940,777		-	35,303	80,606	66,959	64,434	67,046	66,859	66,559	65,832	60,714	53,963	46,239	56,113	51,443	59,851	67,085	31,772	-	-
Income Statement ($000)																					
Copper ($/lb.)	$3.30		$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30	$3.30
Revenues																					
Copper Cathode	$3,104,566		$0	$116,498	$265,999	$220,965	$212,634	$221,252	$220,636	$219,644	$217,245	$200,356	$178,079	$152,589	$185,172	$169,762	$197,507	$221,379	$104,847	$0	$0
Total Revenues	**$3,104,566**			**$116,498**	**$265,999**	**$220,965**	**$212,634**	**$221,252**	**$220,636**	**$219,644**	**$217,245**	**$200,356**	**$178,079**	**$152,589**	**$185,172**	**$169,762**	**$197,507**	**$221,379**	**$104,847**	**$0**	**$0**
	$0.03																				
OPERATING COST																					
Mining	$974,426		$0	$42,931	$47,009	$47,062	$59,806	$61,967	$69,300	$71,363	$67,767	$71,974	$70,793	$73,839	$69,618	$69,649	$64,964	$58,046	$28,336	$0	$0
SXEW Plant	$672,260		$0	$27,695	$38,781	$44,924	$45,145	$44,090	$47,368	$46,820	$42,225	$38,737	$41,828	$41,805	$47,488	$41,418	$52,220	$50,602	$21,113	$0	$0
General Administration	$86,001		$0	$5,580	$5,654	$5,680	$5,684	$5,686	$5,707	$5,713	$5,704	$5,718	$5,583	$5,594	$5,577	$5,577	$5,577	$5,577	$1,394	$0	$0
Treatment & Refining Charges	$1.04																				
Copper Cathode																					
Selling & Transportation	$0		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Operating Cost	**$1,732,687**		**$0**	**$76,206**	**$91,444**	**$97,665**	**$110,635**	**$111,743**	**$122,375**	**$123,896**	**$115,696**	**$116,429**	**$118,204**	**$121,238**	**$122,684**	**$116,644**	**$122,761**	**$114,224**	**$50,843**	**$0**	**$0**
Total Operating Cost per lb	**$1.84**			**$2.16**	**$1.13**	**$1.46**	**$1.72**	**$1.67**	**$1.83**	**$1.86**	**$1.76**	**$1.92**	**$2.19**	**$2.62**	**$2.19**	**$2.27**	**$2.05**	**$1.70**	**$1.60**		

Table 19-4: El Pilar Financial Model (cont.)

		PP																			
	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Royalty (1% of Gross Revenue)	$28,022		$0	$1,165	$2,660	$2,210	$2,126	$2,213	$2,206	$2,196	$2,172	$2,004	$1,781	$1,526	$1,852	$1,698	$0	$2,214	$0	$0	$0
Property Tax (Included in G&A)	$0		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Salvage Value	-$10,947		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	-$10,947	$0
Reclamation & Closure	$5,100		$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$5,100
Total Production Cost	**$1,754,862**		**$0**	**$77,371**	**$94,104**	**$99,875**	**$112,762**	**$113,956**	**$124,582**	**$126,092**	**$117,869**	**$118,433**	**$119,984**	**$122,764**	**$124,535**	**$118,341**	**$122,761**	**$116,438**	**$50,843**	**-$10,947**	**$5,100**
Total Production Cost per lb	**$1.87**			**$2.19**	**$1.17**	**$1.49**	**$1.75**	**$1.70**	**$1.86**	**$1.89**	**$1.79**	**$1.95**	**$2.22**	**$2.65**	**$2.22**	**$2.30**	**$2.05**	**$1.74**	**$1.60**	**$0.00**	**$0.00**
Operating Income	**$1,349,703**		**$0**	**$39,127**	**$171,895**	**$121,091**	**$99,872**	**$107,296**	**$96,054**	**$93,552**	**$99,376**	**$81,923**	**$58,095**	**$29,826**	**$60,637**	**$51,421**	**$74,746**	**$104,941**	**$54,004**	**$10,947**	**-$5,100**
Mining Royalty (7.5% of Operating Income)	$104,840			$0	$2,935	$12,892	$9,082	$7,490	$8,047	$7,204	$7,016	$7,453	$6,144	$4,357	$2,237	$4,548	$3,857	$5,606	$7,871	$4,050	$4,050
Initial Capital Depreciation	$364,899			$40,544	$40,544	$40,544	$40,544	$40,544	$40,544	$40,544	$40,544	$40,544	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital Depreciation	$374,832			$1,491	$3,277	$8,016	$10,113	$16,814	$22,226	$22,946	$29,027	$29,555	$35,064	$33,402	$30,415	$30,913	$101,573	$30,913	$0	-$30,913	$0
Total Depreciation	$739,731		$0	$42,036	$43,821	$48,560	$50,658	$57,358	$62,770	$63,490	$69,571	$70,099	$35,064	$33,402	$30,415	$30,913	$101,573	$30,913	$0	-$30,913	$0
Net Income After Depreciation	**$505,133**		**$0**	**-$2,909**	**$125,139**	**$59,638**	**$40,133**	**$42,448**	**$25,237**	**$22,857**	**$22,789**	**$4,371**	**$16,886**	**-$7,933**	**$27,985**	**$15,960**	**-$30,684**	**$68,423**	**$46,134**	**$37,810**	**-$9,150**
Taxable Income	$505,133		$0	-$2,909	$125,139	$59,638	$40,133	$42,448	$25,237	$22,857	$22,789	$4,371	$16,886	-$7,933	$27,985	$15,960	-$30,684	$68,423	$46,134	$37,810	-$9,150
Taxes at 30%	$122,277		$0	$0	$37,542	$17,892	$12,040	$12,734	$7,571	$6,857	$6,837	$1,311	$4,193	$0	$8,396	$4,788	$0	$20,527	-$9,205	-$9,205	$0
Net Income After Taxes	**$382,856**		**$0**	**-$2,909**	**$87,597**	**$41,747**	**$28,093**	**$29,713**	**$17,666**	**$16,000**	**$15,952**	**$3,060**	**$12,693**	**-$7,933**	**$19,590**	**$11,172**	**-$30,684**	**$47,896**	**$55,339**	**$47,015**	**-$9,150**
CASH FLOW																					
Operating Income	**$1,244,864**	**$0**	**$0**	**$39,127**	**$168,960**	**$108,199**	**$90,790**	**$99,806**	**$88,007**	**$86,347**	**$92,360**	**$74,470**	**$51,951**	**$25,469**	**$58,400**	**$46,873**	**$70,890**	**$99,336**	**$46,134**	**$6,897**	**-$9,150**
Working Capital																					
Account Recievable (10 days)	$0	$0	$0	-$3,192	-$4,096	$1,234	$228	-$236	$17	$27	$66	$463	$610	$698	-$893	$422	-$760	-$654	$3,193	$2,873	$0
Accounts Payable (30 days)	$0	$0	$0	$6,264	$1,252	$511	$1,066	$91	$874	$125	-$674	$60	$146	$249	$119	-$496	$503	-$702	-$5,209	-$4,179	$0
Inventory - Parts, Supplies	$0	$0	-$10,500	$0	$0	-$5,250	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$15,750	$0
Total Working Capital	**$0**	**$0**	**-$10,500**	**$3,072**	**-$2,844**	**-$3,505**	**$1,294**	**-$145**	**$891**	**$152**	**-$608**	**$523**	**$756**	**$948**	**-$774**	**-$74**	**-$257**	**-$1,356**	**-$2,017**	**$14,444**	**$0**
CAPITAL EXPENDITURES																					
Initial Capital																					
Mine	$103,070	$0	$103,070	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
SXEW Plant	$261,829	$104,731	$130,914	$26,183	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Owners Cost	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital																					
Mine	$144,631	$0	$0	$7,562	$3,631	$19,969	$3,410	$19,671	$21,221	$6,481	$28,903	$4,747	$4,073	$1,111	$490	$23,361	$0	$0	$0	$0	$0
Process Plant	$230,201	$0	$0	$5,861	$12,440	$22,681	$15,467	$40,635	$27,484	$0	$25,830	$0	$58,936	$0	$15,272	$0	$5,596	$0	$0	$0	$0
Total Capital Expenditures	**$739,731**	**$104,731**	**$233,985**	**$39,606**	**$16,070**	**$42,650**	**$18,877**	**$60,306**	**$48,704**	**$6,481**	**$54,733**	**$4,747**	**$63,009**	**$1,111**	**$15,763**	**$23,361**	**$5,596**	**$0**	**$0**	**$0**	**$0**
Cash Flow before Taxes	**$505,133**	**-$104,731**	**-$244,485**	**$2,593**	**$150,046**	**$62,044**	**$73,208**	**$39,355**	**$40,193**	**$80,019**	**$37,019**	**$70,246**	**-$10,302**	**$25,305**	**$41,863**	**$23,437**	**$65,037**	**$97,980**	**$44,117**	**$21,340**	**-$9,150**
Cummulative Cash Flow before Taxes		-$104,731	-$349,216	-$346,623	-$196,576	-$134,532	-$61,324	-$21,970	$18,224	$98,242	$135,261	$205,507	$195,204	$220,509	$262,372	$285,810	$350,846	$448,826	$492,943	$514,283	$505,133

19.10.2 Sensitivity Analysis

Figure 19-1 through Figure 19-4 show the sensitivity analyses for the after-tax base case option for copper prices, total operating costs, initial capital expenditures and total copper recovery.

Project is most sensitive to copper price, followed by copper recovery and operating cost. It is least sensitive to initial capital expenditure. Impacts to sensitivity is shown in Figure 19-5.

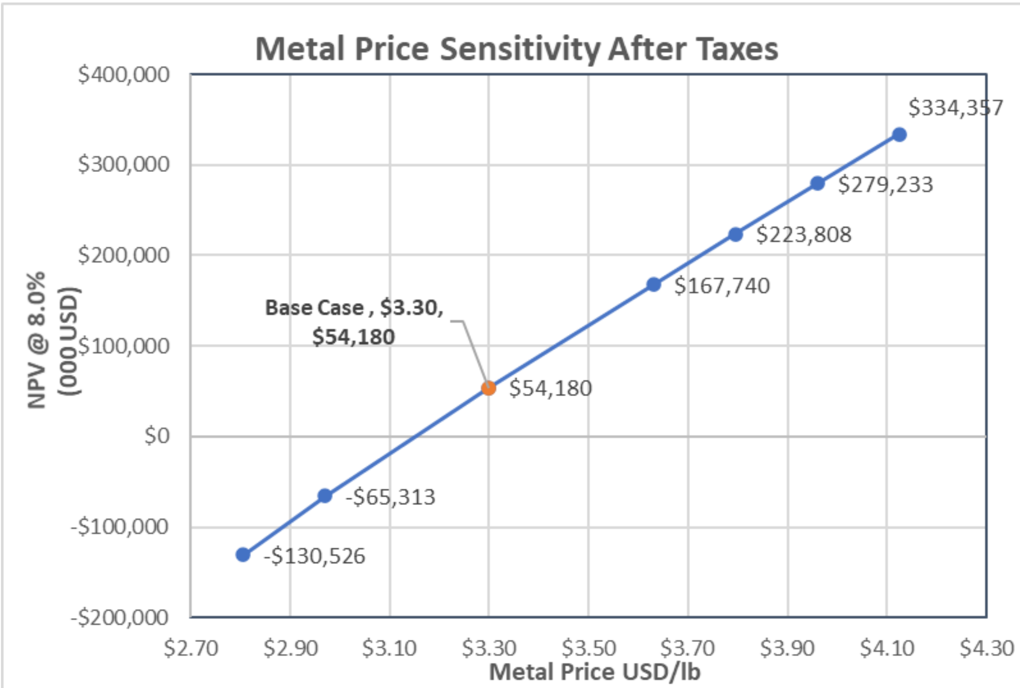


Figure 19-1: Copper Price Sensitivity, After Tax NPV @8%

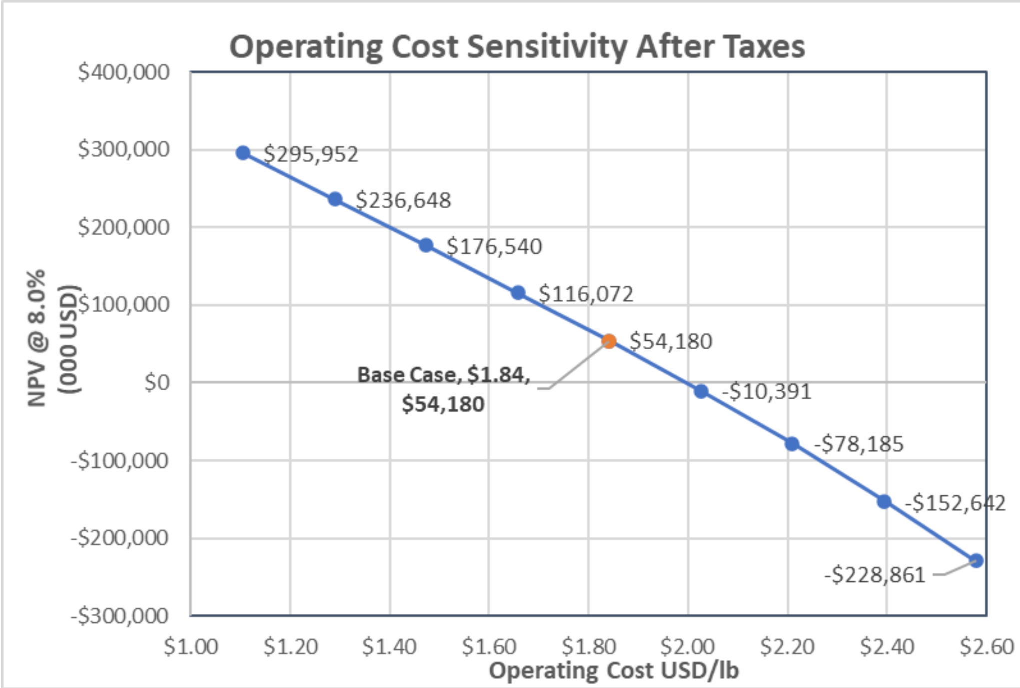


Figure 19-2: Operating Cost Sensitivity, After Tax NPV @8%

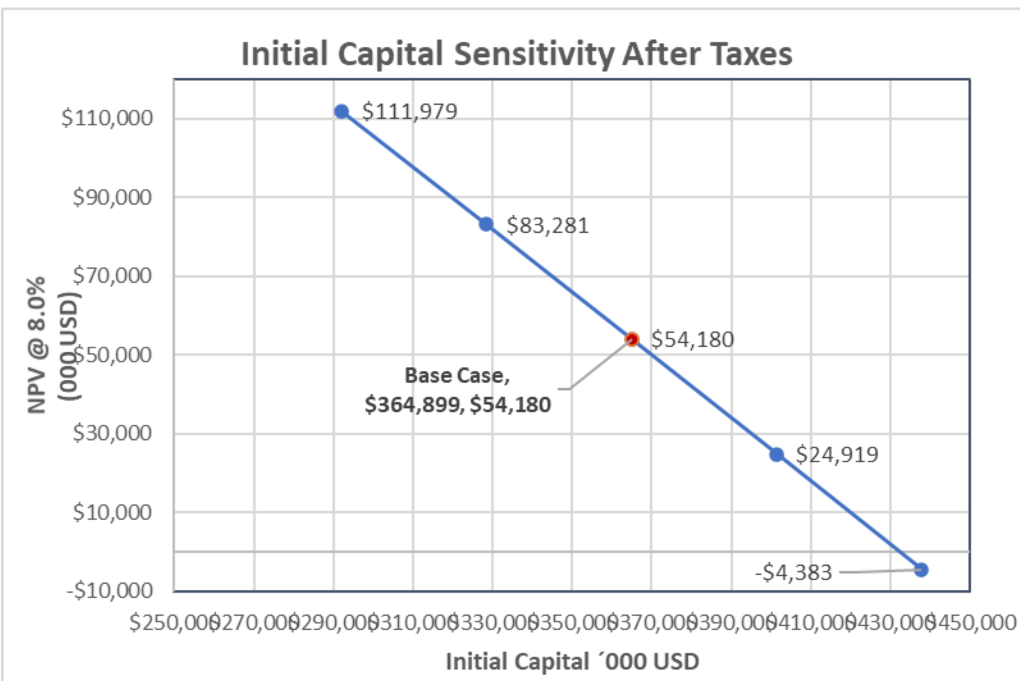


Figure 19-3: Initial Capital Sensitivity, After Tax NPV @8%

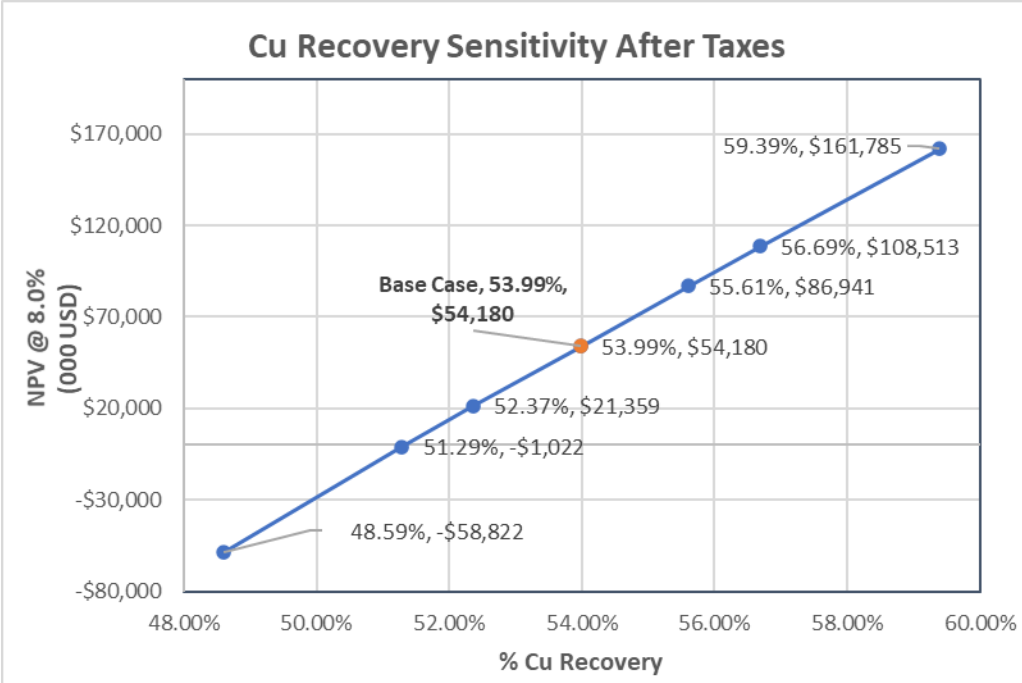


Figure 19-4: Copper Recovery Sensitivity, After Tax NPV @8%

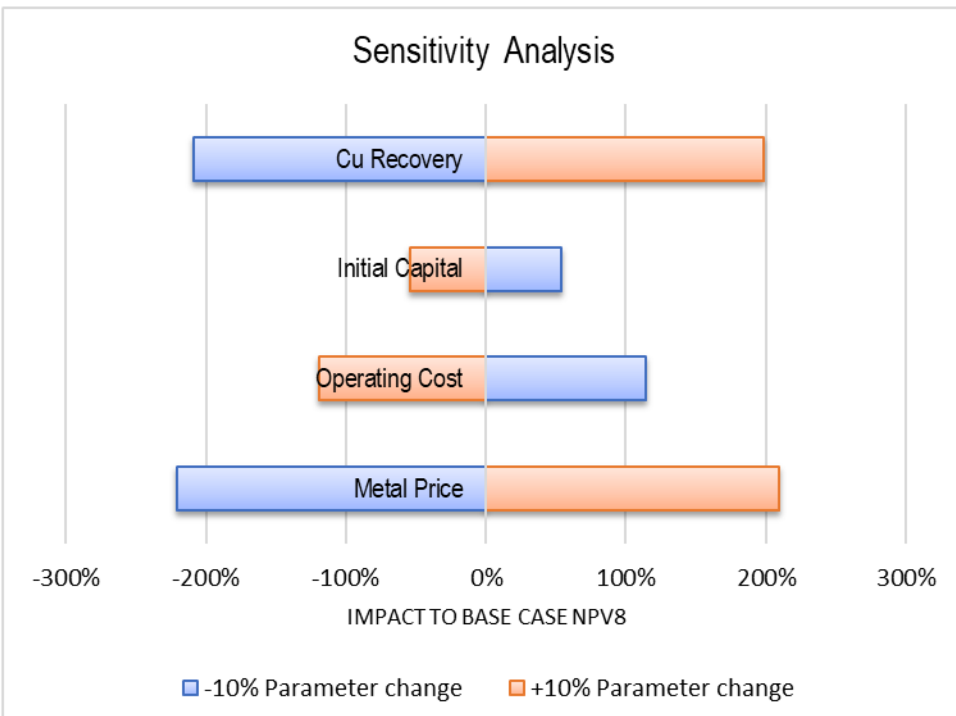


Figure 19-5: Sensitivity Analysis

19.10.3 Financial Analysis Summary

The capital and operating cost estimates for the El Pilar Project have been completed along with mine scheduling. There are no known or anticipated environmental or permitting issues that would ultimately affect SCC to construct and operate the Project under the assumptions detailed in this report.

The work completed in this Technical Report indicates that the El Pilar Project is economically viable for the production of copper from heap leaching. The reserves are sufficient for 16 years of production at an average leaching rate of 58,000 t/d. The Project is projected to average 59 Mlbs of copper production per year and to produce 940 Mlbs of copper cathode over the LOM.

The financial model, which incorporates capital and operating estimates along with copper price assumptions, demonstrates that the Project is economic with an after-tax net present value of $54.2 million at a discount rate of 8%. Capital pay-back of initial facilities capital is estimated in 7.0 years and the IRR of the Project is 9.88%. Summary of main economic parameters and indicators is shown in Table 19-5.

Table 19-5: Financial Analysis Summary

Financial Analysis Summary	
CAPEX, Processing Facility ($000)	$261,829
CAPEX, Mine ($000)	$103,070
Total CAPEX ($000)	**$364,899**
Sustaining Capital, Processing Facility ($000)	$230,201
Sustaining Capital, Mine ($000)	$144,631
Total Sustaining Capital ($000)	**$374,832**
Production Cost ($/lb Cu)	$1.84
NPV @ 8% ($000)	**$54,180**
IRR %	**9.88%**
Payback (Years)	**7.0**
Cu Reserves (kt)	**317,465**
Cu Cathode Produced (klbs)	**940,777**
Gross Revenue ($000)	**$3,104,566**

20 ADJACENT PROPERTIES

El Pilar is an extensive property (9,571.3771 hectares) consisting of nineteen (19) mining concessions as described in Section 3. The property extends approximately 8 km east-west and approximately 9 km north-south. The El Pilar mineral reserve is centrally located on the SCC concessions. Other property owners have concessions surrounding SCC's El Pilar property on the north, west and eastern sides.

There are no known mineral deposits on the concessions immediately adjacent to the El Pilar property.

21 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information to this report that has not already been covered in the other sections.

22 INTERPRETATION AND CONCLUSIONS

This section contains forward-looking information related to Mineral Resources and the LOM plan for the Project. The material factors that could cause actual results to differ materially from the conclusions, estimates, designs, forecasts or projections in the forward-looking information include any significant differences from one or more of the material factors or assumptions that were forth in this sub-section including geological and grade interpretations and controls and assumptions and forecasts associated with establishing the prospects for economic extraction, grade continuity analysis and assumptions, Mineral Resource model tonnes and grade and mine design parameters, actual plant feed characteristics that are different from the historical operations or from samples tested to date, equipment and operational performance that yield different results from the historical operations and historical and current test work results, mining strategy and production rates, expected mine life and mining unit dimensions, prevailing economic conditions, commodity demand and prices are as forecast over the LOM period, waste disposal volumes increase from historical values and predicted values, that regulatory framework is unchanged during the Study period, and no unforeseen environmental, social or community events disrupt timely approvals, regulatory framework is unchanged for Study period and no unforeseen environmental, social or community events disrupt timely approvals, and estimated capital and operating costs, project schedule and approvals timing, availability of funding, projected commodities markets and prices.

The results of the Financial Model, which is presented in Section 19, shows that under current market conditions and following the assumptions and considerations noted in the body of the Study, the El Pilar Project is economically feasible.

The main parameters, before taxes and after taxes are shown in Table 22-1:

Table 22-1: Main Parameters Before and After Taxes

Parameter	After-Tax	Pre-Tax
Total Cash Operating Costs ($/lb Cu)	1.84	1.84
NPV@8% ($M)	54.2	139.5
EBITDA ($M)	2,089	2,089
IRR (%)	9.88%	13.5%
Capital Payback (Years)	7.0	5.5

22.1 RISKS

22.1.1 Land Tenure

- SCC should secure the surface land required for the project.

22.1.2 Mineral Resource, Reserves and Mining

- Given the low percentage of the Measured resources relative to Indicated, the predictability of the tonnage and grade at the local level may have some uncertainties.
- Failure to maintain design slope angles. If operational slope angles are slightly flatter than design angles over several benches the result is significantly less ore available at the bottom of a mining phase than anticipated.
- The risk to slope stability within the Quaternary Alluvial Wash Deposits is primarily geological. While the degree of consolidation is known to be variable, the exact distribution of poorly consolidated zones cannot be determined in advance or reliably modeled in stability analyses. If extensive zones of poorly-consolidated

Quaternary Alluvial Wash Deposits are encountered, such as in the upper portion of the Qwu, then flatter slope designs may be required.

- In the Intrusive, risks of rock mass failure appear low due to the competent character of this material, and the limited slope heights. The risks of encountering a well-developed structural set that could control large-scale stability is indicated to be low based on the geological model and the structural data developed from the oriented core and surface mapping. However, there is indicated to be a moderate risk of locally encountering structural conditions that could result in the development of bench-scale wedges that could control bench stability and require local modifications to the slope design; this risk is indicated to be limited to the ends of the slope in Intrusive where the orientation of the slope swings away from east-west. This is limited to the northern pit limit where the intrusive is localized.
- Higher than anticipate mining dilution and mining loss during operations could negatively impact the Mineral Reserves.
- Fluctuations in exchange rates, copper selling price, and key consumable costs (i.e. fuel) could result in lower than expected economics for the Project.

22.1.3 Metallurgy

- The main potential risks are associated with copper recovery and acid consumption. There is a risk with respect to acid pricing.
- Given the nature of the ore, there is a potential for percolation problems which should be manageable with interliner installation every number of lifts.
- There is a risk that leaching of ore during the pre-production period could not reach the expected concentration for the start-up of the process plant due to delays in placing the ore on the heap and possible lower copper recovery if the ore is placed as ROM.

22.1.4 Environmental

- M3 did not find risks associated with the climate data. A project weather station installation could help optimize engineering and design of drainage of plant areas.
- The development of the Environmental Impact Manifest and water studies assumes a seamless process for the permits for construction. If the process is slowed, delays can occur.

22.1.5 Operating Cost

- The mining industry is very active in Mexico and the market for trained personnel is getting very competitive and there is a potential for the local market to see higher competition for employers in order to retain employees. This could potentially drive up the operating cost.

22.2 OPPORTUNITIES

22.2.1 Mineral Resource, Reserves and Mining

- Improve the resource model by including a differentiation of oxidized species, this will also help to improve the metallurgical response.
- Carry out a conditional simulation study to understand the variability, and therefore risks, of the behavior of grades and geometallurgical variables in order to concentrate drilling and testing efforts.
- There is the potential beyond the known resource/reserve to further expand the deposit by drilling to the south.
- There is potential to find the ultimate source of copper mineralization believed to be a higher grade breccia and/or porphyry copper deposit.

- Optimize the design of the mining phases, final pit and the mine production schedule to delay or reduce waste material. Also, optimize designs of the waste dumps to smooth truck requirements by time period and possibly reduce the number of trucks required.
- Assess the possibility of utilizing crusher/conveyor to minimize haulage.
- Assess the possibility of utilizing electric rope shovels to save on operating costs.
- Assess the possibility of utilizing autonomous haulage and drilling to increase reduce equipment numbers.
- Slope design optimization based on performance of slopes developed within the Quaternary Alluvial Wash Deposits will provide the opportunity to increase slope angles during pit development if greater shear strengths can be demonstrated by documentation of carefully excavated, over-steepened slopes in non-critical areas of the phase pits.

22.2.2 Metallurgy

In general, an increase in copper recovery in Leaching could lead to an optimization of the production schedule, eventually helping to smooth out the lower SCu expected after Year 5 of the project. Additionally, an increase in recovery has a direct effect on the unit cost of operation because more copper is produced from proportionally less ore. Copper recovery and other potential metallurgical upsides include the following:

- Copper recoveries at El Pilar are at least initially a function of copper solubility, although mineralogical studies suggest that over longer periods of time a considerable amount of the residual copper may be recovered.
- Recovery equation has been developed using a wide range of copper solubility ratios (SCu/TCu) from representative samples of the ore body and column testing. A 120 day leach cycle was considered in the development of the recovery equation. Recovery follows the following equation: Recovery % (of TCu) = $33.49\ln(X) + 79.49$, where X is the Ratio (%ASCu/%TCu).
- A 120 day leach cycle should be assumed initially, although real operating conditions may show that a different leach cycle is viable. Stacking plan must be adapted accordingly to accommodate these cycles.
- LOM acid consumption should average approximately 22 kg acid per tonne of ore.
- Curing method will also be important, since ROM ore will be used. Trickle down curing will cause a lag in recovery, a thorough wetting of the ore will be crucial to achieve the projected recovery curve.

22.2.3 Capital Cost

- The capital cost estimate is based on actual current equipment and material pricing, the current international environment and freight restrictions has affected pricing significantly and if these situation changes and materials become more available prices could get reduced affecting on a positive way the total project initial Capex.
- Optimization of heap leach design could reduce sustaining capital costs. Objective should be optimizing the earthworks and reducing the amount of interliner used when stacking the ore.
- The development of detail engineering, with a possible optimization of the arrangement of facilities, could show savings in the expected quantities for excavation and fill, especially on the heap leach facility.

22.2.4 Operating Costs

- As mentioned before, the possible optimization of copper recovery and acid consumption based on operating experience could result in a reduction of unit operating cost.
- Mine operating costs represent a significant portion (65%) of the total operating cost according to the financial model. There is an opportunity for improvement during detail development of the detail mine plan and schedule.

23 RECOMMENDATIONS

23.1 LAND TENURE AND WATER

- SCC should secure the surface land required for the project.

23.2 MINING AND MINERAL RESERVES

- Include in the geological modeling the differentiation of mineralogical species from oxidized ores
- SG/Density sampling should be completed in lithologies that have limited data
- Continue with infill drilling campaigns to improve Resource confidence and Mineral Resource categorization.
- An evaluation of utilizing contractor to pre-strip to save on capital expenditures in the pre-production phase.
- An evaluation of an in-pit crusher conveyor scenario to evaluate potential operating cost savings compared to haulage.
- Review of the heap leach pad design and in particular consideration of alternate access ramp and placement considerations
- Review of the ex-pit roads and overburden storage facility designs.
- Review of the haul road width depending on final selection of the trucks and which mine regulation SCC considers appropriate.
- Review of electric rope shovels as an alternative to diesel hydraulic that was assumed for the current base case scenario.
- Review of electric production drills as an alternative to diesel powered drills.
- Review of potential for autonomous haulage and drill fleets to increase utilization and productivity.
- Review of trolley assist haulage for potential fuel savings.

23.3 METALLURGY

- SCC should implement an early assay and metallurgical data collection program, first directed to evaluate the performance of leaching of the first ore to be placed in the heap, during the preproduction period. This will serve as an early detection of possible problems, and afterwards should be directed to monitoring, reporting and control of leaching and SX-EW plant operation. This could require contracting an outside laboratory to process samples, while the permanent facilities are built.

23.4 HEAP LEACH FACILITY

- Detail engineering of the HLF should be developed to evaluate potential improvements reducing earthwork quantities to improve initial CAPEX.
- Additional percolation testing should be performed to confirm maximum number of lifts recommended before an interlift liner is required.

23.5 ENVIRONMENTAL

- There are no recommendations for environmental concerns.

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25 RELIANCE ON INFORMATION SUPPLIED BY REGISTRANT

The results and opinions outlined in this TRS that are dependent on information provided by Qualified Persons outside of M3 are assumed to be current, accurate and complete as of the date of this report.

Reports received from other experts have been reviewed for factual errors by SCC and M3. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

M3 has not reviewed the drilling results, the pit design, or the land and incorporation documents.

Metallurgical testing done by SCC's consultants (METCON, and others) depends on the samples' accuracy that represents mineralization within the copper deposit. This may or may not be the case. The generation of fines is uncertain.

The metal prices utilized for the Base Case financial analysis are set based on SCC information. Metal prices change rapidly. There is no guarantee that the costs and financials presented herein will be accurate.

The acid costs is based on cost of acid produced by SCC in their own plants. They could vary depending on market conditions.

Mining is a risky business. The risk must be borne by the Owner.

25.1 GEOLOGY AND MINERAL RESOURCE

The Mineral Resource estimate was prepared by Golder based on information provided by SCC. All work has been reviewed by Ronald Turner (CP) of Golder, who is the Qualified Person responsible for the Mineral Resource estimate.

25.2 MINERAL RESERVE

The Mineral Reserve estimate was prepared under the direction of SCC, in consultation with M3. All work has been reviewed by Danny Tolmer, P.Eng of Golder who is the Qualified Person responsible for the mine planning and Mineral Reserve Estimate. The mine production schedule, operating and capital cost estimates for the mining portion of this Study was provided to M3 for final compilation of the cashflow and economic assessment.

25.3 METALLURGY AND PROCESS ENGINEERING

The metallurgical testing program for the El Pilar deposit was developed under the direction of SCC, in consultation with METCON. All work has been reviewed by Laurie Tahija, Q.P., of M3, who is the Qualified Person responsible for the metallurgy, flow sheets, and process plant sections of this TRS.

25.4 ENVIRONMENTAL AND PERMITTING

The Environmental Impact Manifest (MIA) and Change of Land Use (CUS) were prepared by M3 Mexicana S. de R.L. de C.V. (M3M). M3M also developed related permitting for this TRS. The MIA and CUS have been submitted and approved by the authorities.

25.5 GEOTECHNICAL

The geotechnical work associated with pit slope design was prepared by Golder and updated by the Mines Group. This work has been reviewed by Michael Pegnam of Golder, who is the Qualified Person responsible for the geotechnical aspects of the Mining and Mineral Reserve Data Verification.

25.6 HYDROLOGY

Investigación y Desarrollo de Acuiferos y Ambiente (IDEAS) provided the hydrological study and assisted SCC in obtaining the necessary water permit. Dr. Miguel Rangel Medina is the official contact for IDEAS.