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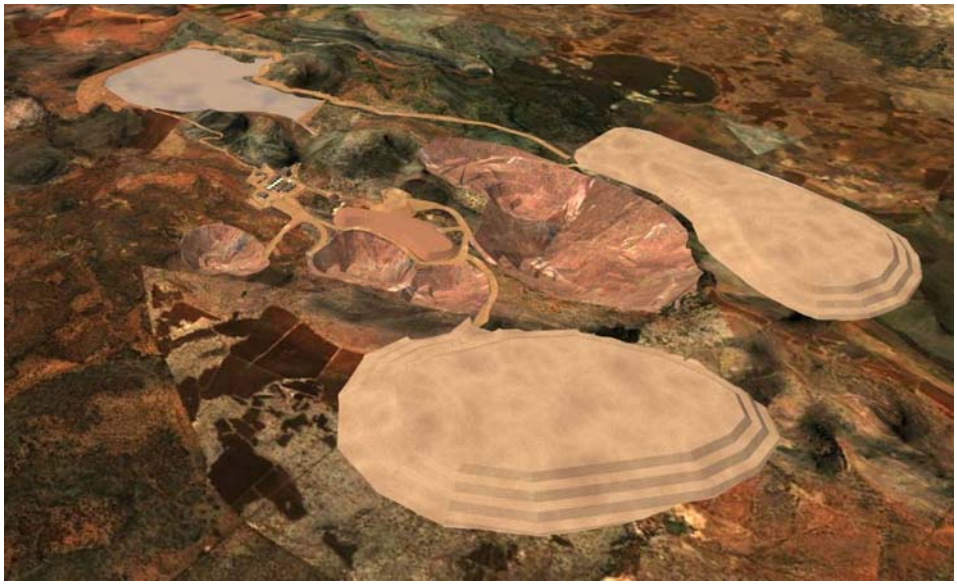
La Preciosa Silver-Gold Project



NI 43-101 Technical Report
Feasibility Study
Durango, Mexico

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DATE AND SIGNATURES PAGE

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LA PRECIOSA SILVER-GOLD PROJECT
FORM 43-101F1 TECHNICAL REPORT

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A	Feasibility Study Contributors and Professional Qualifications
	• Certificate of Qualified Person (QP)

1 SUMMARY

M3 Engineering & Technology Corporation of Tucson, Arizona was contracted by Coeur Mining, Inc. (Coeur) to prepare a Feasibility Study (the FS) and an Independent Technical Report (the Report), compliant with National Instrument 43-101 (NI 43-101) on the La Preciosa Silver-Gold Project (the Property or the Project). This section briefly summarizes the findings of the Feasibility Study.

The Project is situated on the eastern flank of the Sierra Madre Occidental mountain range located approximately 85 kilometers (km) northeast of the city of Durango, in the municipalities of Pánuco de Coronado and Canatlán, within Durango State, México. The property is centered on coordinates 24°25'42.4200"N Latitude and 104°27'27.2380"W Longitude (554,987.8815mE, 2,701,771.0046mN) in the Universal Transverse Mercator (WGS 84 datum), Zone 13R (Northern Hemisphere).

The proposed Project is an open pit silver-gold mine. The mill throughput rate is estimated at 3,650 ktonnes/yr or 10,000 dry tonnes per day (dtpd) for an 11-year life, but is reliant on other factors such as equipment productivity. Over the life of the Project, 106 million troy ounces of silver and 128 thousand ounces of gold are projected to be recovered.

Coeur selected M3 Engineering & Technology Corporation (M3) and other respected third-party consultants to prepare mine plans, resource and reserve estimates, process plant designs, and to complete environmental studies and cost estimates used for this Report. All consultants have the capability to support the Project, as required and within the confines of expertise, from feasibility study to full operation. The costs are based on end of first quarter 2014 US dollars.

1.1 KEY DATA

Key project parameters are presented in Table 1-1 including a summary of the project size, production, operating costs, metal prices, and financial indicators. This Project is sensitive to changes in metals pricing. A metals pricing scenario of US$22/oz for silver and US$1,350/oz for gold resulted in an economic project with an approximately 11-year mine life.

Table 1-1: Key Project Data

Mine Life (years)	11 years		
Mine Type:	Open Pit		
Process Description:	SAG/Ball mill, CCD, Merrill-Crowe		
Total Material Mined (Tonnes per day)	166,700		
Design Mill Throughput (Tonnes per day)	10,000 metric tonnes		
LOM Silver Ore Grade	105.8 g/t		
LOM Gold Ore Grade	0.174 g/t		
Initial Capital Costs ($US Millions)	$327		
Sustaining Capital Costs ($US Millions)	$127		
Payable Metals			
Gold (Thousand troy ounces)	128		
Silver (Thousand troy ounces)	105,790		
Unit Operating Cost: ($/tonne ore treated)	**LOM**		
Mining Cost	22.52		
Equipment Lease Costs	3.08		
Processing Cost	14.70		
G&A Costs	2.33		
Shipping, Smelting and Refining Costs	0.69		
Total cost	43.32		
Financial Indicators	**Base Case**	**High Metal Price (+20%)**	**Low Metal Price (-20%)**
Gold Price (per troy ounce)	$ 1,350	1,620	1,080
Silver Price (per troy ounce)	$22.00	26.40	17.60
After Tax Project Internal Rate of Return (IRR)	9.5%	20.3%	-5.5%
After Tax NPV at 5% Discount Rate ($ thousands)	$87,605	$326,665	-$179.304
After Tax Payback (years)	6.9	3.0	n/a

1.2 PROPERTY DESCRIPTION

1.2.1 Location and Access

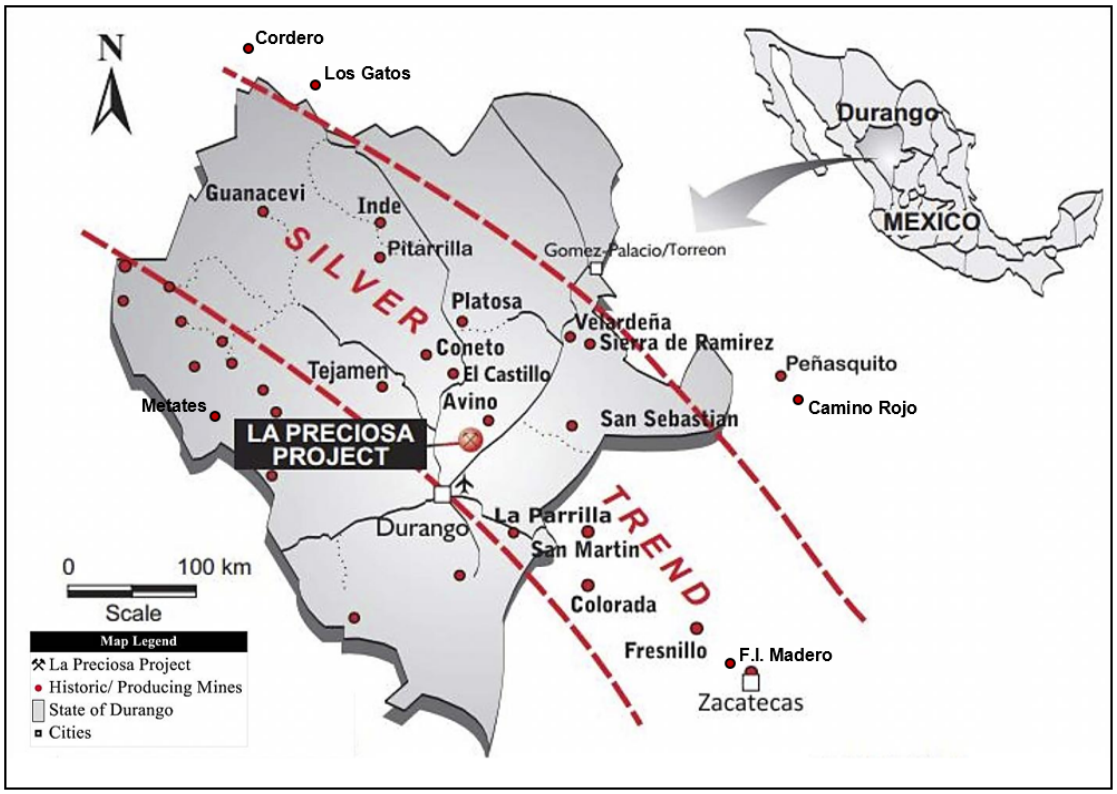
The Property is located approximately 85 km by existing roads, northeast of the city of Durango and can be accessed by vehicle from Durango in approximately 90 minutes. A new access road is anticipated to be constructed which would reduce this to a distance of 50 km and a time of 45 minutes. Durango is the capital city of Durango State and has a population of over 600,000. One of the major industries in Durango is mining, particularly for silver, and the region is expected to be a good source of skilled personnel, support services, and mining equipment. An international airport serves the city of Durango with daily flights connecting to destinations within México and connections to the United States. Figure 1-1 shows the Project's regional location.

Durango is situated along Mexican Federal Highway 40, which connects Durango to Mazatlán, approximately 310 km. to the southwest on the Pacific coast, and to Torreón, approximately 245 km to the northeast. A railway line runs between Durango and Torreón and connects to other cities in Mexico and the United States.

To get to the site from Durango, travel northeast toward Torreón by sealed Federal Highway 40, to the town of Francisco I. Madero. At this point, a secondary paved road is followed northwest to the village of Lázaro Cárdenas, then by newly paved road to the village of Francisco R. Serrano. After 9 km, turn and traverse a newly paved road in a southwest direction to the village of Francisco Javier Mina, then travel in a southerly direction for 5.5 km by gravel road to the access road to the Project site. The access road is a 3.5 km gravel road heading southeast and leads to the portal of

the historic workings and the main camp of the Project. It is to be noted that the road between Francisco Javier Mina and the town of Ricardo Flores Magón is in the process of being paved and should be completed in late 2014.

A much shorter site access road is planned to go from Highway 40 directly to the site and avoid the small communities. This road is also anticipated to be aligned with the anticipated water pipeline route. The pursuit and acquisition of rights in and to necessary or convenient surface estates is ongoing.



Source: Coeur, 2013

Figure 1-1: Project Location Map

The Property lies on the western edge of the high plains of northern México, an extensive volcanic plateau characterized by narrow, northwest trending ranges separated by wide, flat-floored filled basins. In the Durango area, the basins have elevations of between 1,900 m to 2,100 m above mean sea level (amsl) and the higher peaks rise to 3,000 m. The Property elevation in the area of the mineralized zones at the Property is between 1,990 m and 2,265 m. The highest elevations on the Property are at the northwest trending La Preciosa Ridge, which overlies the La Gloria and Abundancia veins. A broad valley forms to the east of the ridge and extends approximately 1 km toward another lower lying ridge to the northeast. Grasses, small shrubs, and cactuses comprise the typical vegetation on the steep hillsides with larger bushes and mesquite trees in the lower lying areas near springs and streams. Nearby farmers produce beans and maize with groundwater sourced from thick gravel beds in the surrounding plains or via dry farming during the rainy season. Local cattle graze on land dominated by rocky soils supporting nopal (prickly pear) and huizache (acacia) scrubland.

The Property area has a semi-arid climate, with an annual average temperature of about 25°C and an average annual precipitation of about 600 millimeters (mm), usually occurring between May and October. Temperatures can fall below freezing on winter nights but snow is rare. Mining activities can take place year round. The dominant wind direction is southeast.

1.2.2 History

In April 2013, Coeur completed the acquisition of Orko Silver Corp. (Orko), which owned the property. The Project now operates as a Mexican-registered company under the name of Proyectos Mineros La Preciosa, S.A. de C. V. (PMLP) a wholly owned subsidiary of Coeur. Since acquisition, activities have included land and water resource acquisition plus additional efforts on geological and technical studies. All involved property owners were identified and their titles verified to be in good standing prior to acquisition of surface rights.

1.3 GEOLOGY

The Project is located on the eastern flank of the Cretaceous to mid-Tertiary age Sierra Madre Occidental (SMO) a north to southeast trending mountain range in northwestern Mexico that parallels the Pacific coast of Mexico and extends south from the Arizona-Sonora border to Guanajuato. This mountain range is part of an island arc assemblage of early Mesozoic age consisting of metamorphosed, deep-water sedimentary rocks, and island arc volcanic rocks. These volcanic rocks are characterized by a thick sequence of lower Tertiary andesite units (Lower Volcanic Series) overlain by younger pyroclastic dominated rhyolites (Upper Volcanic Series).

The oldest rocks in the property are Jurassic-Cretaceous metasedimentary graphitic schist, chlorite schist, and layers of quartzite. Overlying the metasedimentary sequence is a thick package of unmetamorphosed polylithic conglomerate containing lenses of arkosic sandstone. The sedimentary package is overlain by intermediate tuff and agglomerate of the regional Tertiary age Lower Volcanic Series. Locally, the flows are porphyritic and the tuffs are partly welded. Post-mineralization basalt flows that erupted from several Pleistocene-age volcanic vents fill the lower valleys, and are the youngest rocks within the property.

The region is transected by a regional northwest-striking San Luis-Tepehuanes fault system, which roughly coincides with the eastern margin of the SMO. This fault system is a complex network of northwest- to north-striking, west-dipping fault segments that are associated with east to northeast tilting of Tertiary stratigraphy.

The Project area contains a series of Tertiary-age silver-bearing (±gold) low and intermediate epithermal quartz veins associated with adularia, barite, calcite, rhodochrosite, as well as acanthite, freibergite, silver sulfosalts and minor electrum, plus variable amounts of pyrite, honey-colored sphalerite, tennantite/tetrahedrite, chalcopyrite and galena, as well as supergene iron and manganese oxides. Mineralization variably occurs throughout all lithologic units, with the exception of post-mineralization basalts and alluvium, and mineralization is preferentially concentrated in the Lower Volcanic Sequence and Sedimentary Sequence.

Mineralization is present as three main types that occur as syn-mineralization veins and faults:

- South to southeast– and south–striking, shallow west-dipping structures;
- South to southeast– to south to southwest–striking, moderate to subvertical west-dipping structures; and
- East-southeast–striking, moderate to steep south-dipping structures.

In addition to mineralized structures, post-mineralization faults were observed that are typically steeply dipping and southwest- or northeast-striking.

Main visible alteration observed in the deposit consist of patchy albite-epidote ± chlorite, pseudomorphic clay or calcite, silica-sericite-pyrite, pervasive silica, and fracture-fill clay. Alteration is defined on the basis of alteration mineral assemblages, texture, and distribution.

1.4 SAMPLE COLLECTION AND DATA VERIFICATION

Sample collection, analysis, and security were conducted through industry-accepted practices throughout the life of the Project. The Project contains 874 exploration and development drillholes. Drilling methods consisted of both diamond core and reverse circulation. Analytical assay procedures were conducted at ISO-certified laboratories, with commercial standards, blanks, and duplicates inserted for quality control at variable rates dictated by the drill campaign and project owner.

Coeur completed multiple assay and data collection campaigns during 2013 and 2014. A category 3 development-drilling program resulted in the completion of 103 drillholes, adding 11,587 primary samples to the Project database. The Coeur exploration team conducted a visual inspection of legacy drill core; targeting previously un-sampled zones of altered rock and geologic structures which added 3,520 primary samples to the Project database. A comprehensive assay campaign was conducted to reanalyze 6,884 legacy sample pulps in the range of 25-100 ppm silver using a more precise 4-acid ICP digestion method. Results from two participating commercial laboratories indicated an increase in the median silver grade for the sample population tested. These assay campaigns adhered to Coeur Quality Assurance/ Quality Control (QA/QC) procedures and available results indicate a low failure rate of 1.8% for inserted standards and blanks. The QA/QC program is in progress, with continual analyses of failed samples and the initiation of a check assay program at a certified umpire laboratory.

Data collection and verification commissioned by Coeur included a comprehensive review of the original geologic logs from Orko, Pan American Silver (PAS), and Coeur; followed by the subsequent recoding and data entry of the information. Coeur continued to verify Orko era data by obtaining original assay certificates from commercial laboratories. Evaluation of the data and performance of the quality control samples were acceptable. Coeur provides recommendations for further QA/QC and Data Verification work and acknowledges that the current and former data collection methods, QA/QC procedures, and data performance are adequate for use in resource evaluation.

1.5 MINERAL RESOURCES AND MINERAL RESERVES

A new mineral resource model was built for this feasibility study, using updated drillhole data and multiple indicator kriging (MIK) was used to estimate both silver and gold grades. MIK was used to check for any potential biases in previous resource models and because MIK required less time to implement. Using the new resource model for the feasibility study, the resulting Mineral Reserves and Mineral Resources (MRMR) are shown in Table 1-2 and Table 1-3.

Table 1-2: La Preciosa Mineral Reserves as of 29 July 2014

Category	Tonnes	Average Grade (g/tonne)		Contained Ounces	
		Au	Ag	Au	Ag
Proven	18,365,000	0.200	113.3	118,100	66,920,000
Probable	18,959,000	0.148	97.7	90,400	59,523,000
Total P+P	**37,324,000**	**0.174**	**105.4**	**208,500**	**126,443,000**

Table 1-3: La Preciosa Mineral Resources, exclusive of Mineral Reserves, as of 29 July 2014

Category	Tonnes	Average Grade (g/tonne)		Contained Ounces	
		Au	Ag	Au	Ag
Measured	6,839,000	0.186	84.1	40,900	18,485,000
Indicated	10,540,000	0.160	88.3	54,100	29,920,000
Total M&I	**17,379,000**	**0.170**	**86.6**	**95,000**	**48,405,000**
Inferred	1,889,000	0.126	77.5	7,700	4,705,000

1. Metal prices used for estimation of Mineral Reserves were $22 per troy ounce of silver and $1,350 per troy ounce of gold. Metal prices used for the estimation of Mineral Resources were $25 per troy ounce of silver and $1,400 per troy ounce of gold.
2. A Net Smelter Return (NSR) cutoff of $21.93/tonne ($24.17/short tonne) was used, based on the following parameters:

NSR = [(Ag price per ounce -refining charge) × plant recovery × payable recovery] + [(Au price per ounce -refining charge) × plant recovery × payable recovery]

3. Rounding of short tonnes, grades and troy ounces, as required by reporting guidelines, may result in apparent differences between tonnes, grades and contained metal contents.
4. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be considered for estimation of Mineral Reserves.
5. U.S. Investors are cautioned that the term "mineral resource" is not defined or recognized by the U.S. Securities and Exchange Commission.

1.5.1 Methodology

The methodology used for this resource model was multiple indicator kriging (MIK). This was used for both the silver and gold estimation. The result of the modeling is a probabilistic model that represents the probability of finding ore as a model block rather than a deterministic model that will tag each block as ore or waste. There are two major advantages to probabilistic methods. First, they are quicker and less prone to biases that can be inadvertently introduced to the model when using deterministic methods. Secondly, probabilistic models allow for a rigorous implementation of estimating mining dilution. It is much more difficult to do this with deterministic models without the introduction of somewhat arbitrary factors.

1.5.2 Geological Database

Following Coeur's QA/QC protocols, the database was thoroughly validated by Coeur prior to being used to develop the resource model. One significant issue was found with the pre-Coeur drillhole data is that only 37 percent of the meters drilled were sampled and assayed because of sampling decisions made by previous operators. Many of these un-assayed intervals are away from the known mineralization, but a significant number of un-assayed intervals are intermixed with assayed intervals in the mineralized zones. Ignoring these un-assayed intervals and treating them as missing data would lead to a significant selection bias in the estimate. To address the issue of un-assayed intervals all of the un-assayed intervals were filled with zero values except those that were logged as having no core recovery. It is anticipated that many of the intervals actually would have silver and gold values above zero, but the actual grade would likely be significantly less than the ore cutoff, so the impact of using a zero value rather than a non-zero value is likely to have minimal impact on the resource estimate.

Coeur recognizes that the presence of null values in the data is problematic and has identified the un-assayed intervals within the first few years of mining and has instituted a program of sampling and assaying the missing intervals. As a standard practice, all intervals in Coeur's drill campaign are sampled.

The assay and geological database consists of data from 866 holes representing a total of 267,921 m of drilling. Silver and gold assays are present in 827 holes for a total of 100,446 m assayed. The average interval length is 0.84 m. It can be seen that only 37% of the drilled meters have been assayed. The data available, along with the assays, was mineralization, alteration and lithology logging. The exploratory data analysis focused primarily on analysis of ore controls associated with lithology and the relationship between the silver and gold.

Potential "ore grade" intervals were examined by looking at all intervals where the silver grade was greater than 31.1 ppm (Table 1-4). The veining appears to be the source of the mineralization, but it can be seen that significant mineralization occurs away from the veins within the adjacent volcanic and sedimentary rocks that host the veining.

Table 1-4: Occurrence of High-grade Intercepts.

Unit	Occurrence (m)	Relative Percent	Silver (ppm)	Gold (ppm)
Vein	3616	36	198.5	0.348
Volcanic	4067	41	99.6	0.143
Sedimentary	2248	23	119.5	0.208
Total	9931	100	140.1	0.232

The correlation between the silver and gold is important to the resource evaluation as both contribute to the economic value. The distributions of the silver and gold were estimated independently then evaluated jointly on a block by block basis. The correlation was examined by stratigraphic unit including host rocks, mineralized units, and the post-mineral rock and soil formations. Correlation is relatively poor between silver and gold, but it is not unreasonable within the ore-bearing units such as the Quartz Vein unit and the Upper and Lower Volcanic Series.

Silver variograms were fitted for the grade data and for the indicators for each domain. The orientation of the structures remained constant for each domain with the exception of North Martha, which changed slightly with the upper indicator variables (Table 1-5).

Table 1-5: Structural Orientation of Estimation Domains

Domain	Strike (Azimuth)	Dip (degrees)
North Martha (Ind. 0 to 4)	N15E	30 to West
North Martha (Ind. 5 to 11)	N10E	40 to West
South Martha	N15W	25 to West
Gloria & Abundancia	NS	30 to West

A general kriging plan was identical for all variables, only the search orientation and variograms used were changed by domain. Search ellipsoids with a radius of 500 m along strike, 200 m down dip and 40 m perpendicular to the structure were used and the orientation of the search ellipsoid was changed for each domain to be identical to the orientation of the variograms. Because of the limits on the number of composites used, the search limits were rarely extended to their limits.

The dimensions of the resource model are shown in Table 1-6. The model origin and dimensions are slightly different from previous models but were adjusted to be compatible with the geologic model. The block height of 5 m was chosen to be compatible with the anticipated mining bench height.

Table 1-6: Model Schema

Dimension	Minimum	Maximum	Block Size (m)	No. of Blocks
East	554,500	557,000	10	250
North	2,700,050	2,703,200	25	126
Elevation	1,550	2,270	5	144

A metallurgical model was developed and is primarily dependent on the lithological composition of the mineralized material. Metallurgy recoveries were grouped into three distinct rock types: vein quartz, volcanic, and sediments. It was necessary to attempt to predict the relative proportions of the various rock types that the mineralized material would be composed of, but the geological model was not sufficiently detailed to enable it to be used directly to estimate the mineralized material composition. An indicator model was used to predict the rock type proportions associated with the mineralized material by using the estimated proportions. A linear weighted average of the metallurgical recovery parameters were calculated for each block in the model. A separate silver and gold recovery were calculated based on the grade of the ore portion of the block and stored back into the block model.

The originally proposed metallurgical recovery parameters consisted of a simple constant tail fraction with steps at various grades. Post-processing of the grade distributions requires that the metallurgical recovery be a continuous function rather than a step function; to accomplish this, a linear regression function was fitted to the tail grade versus the calculated head grade of the bottle roll tests. Separate regressions were done for each of the three rock types for silver and gold and then the regression curves were used to predict the recovered grade as a function of the head grade.

There is a relatively small amount of Inferred material estimated into the model. This is in large part because of a function of the kriging plan. The kriging plan requires data to be present in at least three octants and from at least three drillholes. These limitations set in the kriging plan prevent excessive extrapolation of the model away from available data. Allowing further extrapolation away from the data would result in estimates that would be considered basically unsupported.

Three levels of model validation were done: first the model was checked to ensure that it was globally unbiased, second it was checked to ensure that it was spatially unbiased, and finally the resource model was visually checked in cross-section and plan to ensure that the model compared reasonably to the drilling.

The indicator kriged model was compared to a nearest neighbor model to check for unbiasedness. The nearest neighbor (NN) model will provide poor local estimates and should never be used for local estimates, but the NN model will provide a superior unbiased global estimate of the grade distribution within the model volume. Therefore, over large volumes, the NN model estimate can be used to provide an excellent check against unbiasedness in the model. The NN estimate honored the fault boundaries, as did the kriged model.

The e-type estimate is the estimated average block grade (at a zero cutoff) reconstructed from the indicator kriged distribution in each block. Looking at the average block grade at a zero cutoff allows it to be directly compared to the nearest-neighbor estimate, also at a zero cutoff. Global unbiasedness was checked by averaging all of the estimated Measured and Indicated blocks in the model and the E-type estimate for silver is slightly higher than the NN estimate while the gold estimate is slightly lower. These differences are within acceptable limits.

1.6 MINING METHODS

The Project mine plan was developed using conventional hard rock open pit mining methods. No underground resource was envisioned in this mine plan. While a smaller, higher grade portion of the resource could be recovered in an underground mine, the scale and percent recovery of the resource would be substantially diminished. In addition,

the complicated structure of the low angle Martha vein which makes up 60% of the resource would make underground mining complicated and costly.

Scoping studies indicated that a production schedule filling the mill at 10,000 tpd maximized the Project return on investment. The total material rate is tied to equipment productivity. The total material moved ramps up to 214,000 tonnes per day in year 6, and averages 60,400 ktonnes/yr or 167,700 tpd for the first 10 years. The mine is scheduled to operate 365 days/yr with two, 12-hour shifts/day.

The mine plan was developed with a phased approach to facilitate sufficient waste mining to provide the mill feed desired. In addition to the phases, mining was envisioned on 5 m in ore bencehs and 10 m benches in waste to increase productivity and mill feed. The approach to phase designs, mine schedule, and mine equipment requirements are summarized below.

A Lerchs Grossman algorithm using a 10% discount rate was used as a guide to the design of the phases or pushbacks. Multiple economic pits were developed using the costs, slope angles and recoveries outlined in the Section below. The slope angles used include an allowance for roads and recommended catch benches in the overall slopes. Metal prices were varied from low to high in order to establish a series of multiple nested pit geometries and resulting Net Present Values (NPVs). The results of this work indicated the starting point, final pit and the extraction sequence that maximized the NPV throughout the mine life.

The pit evaluation was based on a $22 per ounce silver and $1,350 per ounce gold price. Recoveries were based on the metallurgical model. Refining costs and payabilities were based on quotes received for refining doré. For pit design purposes, prices from the 2013 Preliminary Economic Assessment (PEA) (M3, 2013) were used to establish the pit limits. This pit was designed on only Measured and Indicated resources. Inferred resources were not included.

The economic pit was discounted at an annual discount rate of 10%. The benches were discounted according to a twelve 10-m bench per year annual advance rate.

Eight phases were designed for Project with a minimum of 100 m of operating width on each bench within a phase, in general. Phase 1 was sized to contain roughly one year of mineralized material. The following summarizes the basic parameters used for mine design. General design criteria are listed in Table 1-7.

Table 1-7: Phase Design Parameters

Haul Road Width	30 meters
Haul Road Grade	10% Maximum
Interramp Slope Angles	by sector (see)
Operating width between pushbacks	100 meters nominal

Preproduction stripping will occur in Phases 1, 2, and 3. In many cases there is less than 1 month of mineralized material per 5m bench within a phase at the Project. Consequently, the mine plan maintains 2 phases in mineralized material in order to assure that mineralized material is released and available to assure plant feed.

The mineralized material that is encountered during pre-production (509 ktonnes) will be stockpiled near the crusher (or in the pit) and delivered to the mill during Year 1. The mineralized material in the upper benches of Phase 1 and Phase 2 is limited in quantity. This material is accumulated and stockpiled to ensure there is sufficient mineralized material release in Phase 1, Phase 2, and the stockpile combined to assure mill production.

Low grade material that is above the internal cutoff of $16.11/tonne NSR and less than mill feed cutoff in the early years will be stored in the low grade stockpile that is located northeast of the crusher. That material is planned to be

re-mined and delivered to the mill in later years. The bottom of Table 20-4 illustrates the mineralized material planned for processing inclusive of stockpile re-handling.

The NSR calculation for the mine plan and mineral resource is:

NSR = [(Ag price per ounce -refining charge) x plant recovery x payable recovery] +

[(Au price per ounce -refining charge) x plant recovery x payable recovery]

NSR = [($22.00 / 31.1035) – 0.02] x Plant Recovery x 0.9975] + Silver Contribution

[($1350 / 31.1035) – 0.16] x Plant Recovery x 0.9950] Gold Contribution

The mine plan for the Project includes only Measured, and Indicated category mineralization. The economics do not include a credit from Inferred mineralization. The mine plan material in Table 20-4 contains 37.3 million tonnes of mill feed.

Mine equipment is expected to be comprised of standard commercially available units. Principal mining equipment is a fleet of hydraulic mining shovels and 184 tonne class mechanical drive rock trucks.

Ore will be mined on 5 m bench heights and most of the waste will be mined on 10 m bench heights. The amount of waste mined on 5 m benches is equivalent to 150% of the ore-grade material on each bench.

Mine total manpower is based on having a three-panel roster of crews to man the mine on a 2 by 12-hour shift basis. The rotations are anticipated to be 5 days of day shift, 5 days of night shift and 5 days off, with 24 hours between the end of day shift and the start of night shift. This allows for one crew to be off at all times.

Operating manpower numbers were set by the numbers of equipment, multiplied by the three crews. Maintenance manpower numbers were set by using experience and a ratio of about 60% maintenance to operations staff. These numbers do not include the contracted services for diesel supply, tire maintenance and explosives delivery. A blasting crew was put on to design and load the explosives.

Labor rates are modeled after Coeur's Palmarejo operations, and include fringe benefits. An allowance of 10 percent of the overall manning schedule was added to allow for vacation, sickness and absenteeism.

Table 1-8: Project Preliminary Mining Capital Cost Estimates

Item	Value ($M)
Pre-Production Mining	24.70
Misc. Mining Capital	7.92
Equipment Leasing during pre-production	8.35
TOTAL	**$40.97**

The mining equipment capital costs are based on a leased fleet. The lease costs were calculated from the equipment list. Escalation and contingency are not included in capital costs.

The costs for all the major equipment was based on vendor supplied quotes while costs for most of the minor equipment is based on historical quotes and estimates. Much of the cost information was current during 2013 and 2014. Older quotes for minor capital equipment were inflated as required.

Mine operating costs estimates are based on first principals for the scheduled production, equipment lease requirements, operating hours, hourly equipment operating costs, and manpower requirements. Neither escalation nor contingency are included in operating costs. Life-of-mine operating costs are expected to average $1.395 per tonne drilled and blasted, and $1.383 per tonne overall, including rehandled stockpile material.

It is expected that the mining fleet will be leased for a 5-year period with an option to purchase after the lease has expired. Sustaining capital includes the purchase of the leased mining fleet in every fifth year. Fuel costs were set at $0.77 USD per liter based on publically available price trends, excluding VAT.

1.6.1 Tailing Design

Knight Piésold Ltd. of North Bay, Ontario, has completed a feasibility level design for the tailings storage facility (TSF). The design has been carried out based on the preferred TSF option for tailings storage and water management as provided by Coeur. The selected site for the TSF lies to the southwest of the proposed open pit and process plant.

The current feasibility level design of the TSF is based on a projected 17-year mine life at a processing rate of approximately 10,000 dtpd. The design was performed concurrently to the mine design and planning, and used the PEA resources as a design basis. The actual mine life may be shorter or longer under certain economic conditions. Tailings at 50% solids content (approximately, by weight) will be delivered via pump and pipeline from the plant site for storage at the TSF. The TSF is broadly triangular in shape with the long axis trending slightly northwest-southeast. The ground slopes gently downwards towards the south, at an average gradient of about 4%. The TSF has been sized to store on a permanent basis approximately 45.5 million m^3 or 59.1 million tonnes of tailings at an estimated average in situ dry density of 1.3 tonnes/m^3.

The feasibility design will provide permanent and secure tailings storage, in addition to water storage and controls to ensure protection of the environment during operations and in the long-term (after closure). The TSF has been designed to manage stormwater and run off water inflows. The stormwater management design includes selection of an Inflow Design Flood and an Environmental Design Storm.

Depending on mine life, two embankments will be constructed to establish the TSF, including a main embankment along the south side of the basin and a smaller embankment constructed later at the north side of the basin. Natural topographical containment will form the northeast and west sides of the TSF. The TSF design section will include an initial starter embankment (Stage 1) for the south embankment with ongoing raises completed for the south and north embankments using centerline construction methods throughout the life of the facility. The initial starter embankment at the south will be constructed of zoned rockfill with a composite liner on the upstream slope. The centerline raises will consist of zoned rockfill with a low permeability core zone. Transition zones will be established between the core zone and the embankment rockfill to ensure internal stability. The Stage 1 TSF will provide storage for two years of tailings deposition. Construction of the north embankment will not be required until Year 9 of operations. The embankments will be extended and raised using rockfill from the mine over the life of the facility.

Tailings will be deposited sub-aerially using spigot offtakes starting in the southwest corner and cycling around the south, east and north sides of the facility. Discharge from the embankments and basin perimeter will maintain the supernatant pond near the central west side of the tailings basin away from the embankments. A water reclaim system will maintain the pond with limited size to minimize evaporation. Under these conditions, sufficient storage capacity will be provided within the TSF to store any tailings supernatant water, run off and the design storm events, in addition to an allowance for freeboard. Embankment construction will be scheduled to ensure that there is always sufficient storage capacity available in the facility to avoid overtopping and negate the need for interim spillways for each embankment stage.

The Project is located where water resources are scarce and water supply can be expensive. In addition to minimizing the downstream impacts of the facility on the environment, the primary water management objective for the TSF is to collect all available water for re-use in the process in order to minimize make-up water requirements.

Supernatant water will be removed from the TSF via a decant tower system. Water recovered from the decant system will be pumped back to the plant for re-use in the process circuits. The operating pond will be maintained at the minimum volume required to provide adequate clarification prior to reclaim. A monthly operational water balance was completed for the TSF and the results indicate that the TSF will operate at a net water deficit during all years of operations. Only a portion of the process water requirements can be satisfied by water reclaimed from the TSF. Additional make-up water, from other sources, is required in all years of operation. Overall, the average make-up water requirement over the life of the facility is estimated to be around 0.34 m^3/t of ore processed.

A number of seepage reduction strategies have been incorporated into the design and operation of the facility, including the installation of a liner system and an underdrain collection system to collect any available free water above the liner. The design includes a composite liner consisting of a 60 mil HDPE geomembrane overlying a low permeability soil layer placed on the upstream slope of the starter embankment and over the entire basin footprint. The centerline embankment raises will be constructed with a low permeability core zone. The underdrain system will comprise a series of perforated drainage pipes above the HDPE liner on a herringbone pattern over the TSF basin floor. The underdrain system will terminate in one of two reinforced concrete collection sump towers. The towers will be surrounded with clean, free-draining rockfill and will be raised with the embankments. Submersible pumps, located within the sumps, will return water to the supernatant pond.

Instrumentation will be installed in the TSF embankments and foundations during construction and over the life of the project. The instrumentation will be monitored, as part of the detailed monitoring plan to be developed for the facility during the construction and operation to assess embankment performance and to identify any conditions which differ from those assumed during design and analysis. Amendments to the ongoing designs and/or remediation work can be implemented to respond to changing conditions, should the need arise. Instrumentation will include vibrating wire piezometers, survey monuments and vibrating wire settlement gauges and groundwater monitoring wells.

The primary objective of reclamation and closure activities will be to ensure physical and chemical stability of the TSF. Closure and reclamation will focus on removal of surface infrastructure and stabilizing and covering the exposed tailings surface to ensure that acceptable downstream water quality is maintained. Additional closure work will involve progressive reclamation of the downstream face of the embankments and construction of permanent spillways.

1.6.2 Surface Geotechnical

1.6.2.1 Surface Infrastructure

Geotechnical investigations were completed by Knight Piésold in 2014 to support the feasibility designs for the Project Surface Infrastructure.

The focus of the 2014 geotechnical site investigation program was to characterize the general soil and bedrock conditions within the vicinity of the proposed Plant Site and TSF. The completed site investigations consisted of geotechnical drilling (soils and bedrock), test pit excavation, collection of representative samples, geotechnical logging, geomechanical logging, in situ testing and laboratory index testing.

A total of 18 drillholes and 24 test pits were completed to characterize the overburden and near surface bedrock conditions at the proposed Plant Site and TSF locations. Since the site investigation was initiated, the configuration of the Plant Site has been modified. Additional modifications to the layouts may occur during the next level of design.

The following paragraphs provide a brief summary of the findings of geotechnical site investigations.

Plant Site - A total of five drillholes and eight test pits were completed in the Plant Site area in order to evaluate the foundation conditions and to characterize the overburden and near surface bedrock. Main observations included:

- The depth to bedrock was relatively deep and typically ranged from 15 m to 20 m. The depth to bedrock at one drillhole location was shallow at 4 m.
- The overburden encountered in the area was generally found to comprise of nominal surficial layer of organic silt underlain by sand, with varying amounts of clay or gravel overlying bedrock. All of the soil samples collected from the Plant Site area, except for one, exhibited plasticity with plasticity index values ranging from 9.4% to 26.2%.
- Bedrock underlying the overburden at the Plant Site mainly consisted of andesite which ranged in quality from POOR to FAIR. Basalt was encountered at one drillhole location.

Tailings Storage Facility - A total of 12 drillholes and 16 test pits were completed in the TSF area in order to characterize the overburden and near-surface bedrock and evaluate the shallow bedrock hydraulic conductivity. Main observations included:

- The depth to bedrock was shallow (i.e. less than 5 m) in all the drillholes except on the west side of the TSF, where up to 15.5 m of overburden was locally encountered.
- The overburden encountered in the TSF area generally consisted of a nominal surficial layer of organic silt underlain by sand, with varying amounts of clay or gravel overlying bedrock. All of the soil samples collected from the TSF area exhibited plasticity, with plasticity index values ranging from 8.4% to 31%
- The bedrock consisted of basaltic flows with a well-developed vesicular structure, dense basalt with a weak to moderately developed vesicular structure, scoria and andesite. The encountered rock mass qualities typically ranged from POOR to FAIR.
- A total of 23 packer tests were performed at 12 drillhole locations. Hydraulic conductivity values were generally high especially within the shallow bedrock. This likely is attributed to the well-developed vesicular structure in the basalt, broken and rubble zones. A small area in the southwest corner of the TSF that is underlain by andesite exhibited relatively low hydraulic conductivities. Intervals with no recovered bedrock core were encountered in drillholes at the north end of the facility potentially indicating the presence of a lava tube or similar structure (void).

Geologic Structure between TSF and Open Pit - One deeper, inclined drillhole was advanced into bedrock within the northern area of the TSF to evaluate if the hydrogeological conditions and/or geologic structures were present that could convey seepage from the TSF to the Open Pit. Main observations included:

- Five possible faults were identified in this drillhole with consistent north-south or northeast-southwest striking fault orientations similar to the fault orientations identified in the area of the Open Pits. Faults with these orientations could represent a potential conduit for seepage between the TSF and the Open Pit.
- The measured hydraulic conductivity of the encountered faults was generally one to two orders of magnitude higher than the deep host rock, but several orders of magnitude lower than the shallow bedrock. Given the expected hydraulic conductivities, it is likely any seepage from the TSF will flow through the highly permeable shallow vesicular basalt and not be conveyed to the Open Pit via any specific geologic structure.

1.6.3 Geomechanical - Open Pit

PMLP is currently evaluating the feasibility of mining the deposit using several open pits:

- Abundancia Pit (the main pit, 460 m deep)
- North Satellite Pit (105 m deep)
- Central Satellite Pit (130 m deep)

- South Satellite Pit (115 m deep)

The geomechanical and hydrogeological site investigation program was designed to support the pit slope design of the proposed open pits. The site investigation program was completed between December 2013 and March 2014, and included the following:

- Six oriented and triple-tubed geomechanical drillholes with associated detailed geomechanical logging and hydraulic conductivity (packer) testing
- Installation of multi-point vibrating wire piezometers in three (3) geomechanical drillholes
- Detailed geomechanical logging of intervals of full core from seven (7) exploration drillholes
- Underground line mapping at two (2) locations, and surface line and spot mapping at one (1) and two (2) locations, respectively
- Collection of 123 samples for laboratory strength testing. Unconfined Compressive Strength (UCS), Triaxial and Direct-Shear tests were completed
- On-site evaluation of the susceptibility of the rock mass to time-dependent degradation
- Borehole televiewer surveys of eight (8) exploration drillholes and one (1) geomechanical drillhole conducted by Groupe Qualitas Inc.
- Training Bufete Minero y Servicios de Ingenería, S.A. de C.V. geologists in drill supervision and Expromin S.C. geologists in detailed geomechanical logging

Based on the location and characteristics of the geomechanical domains and the pit design provided by Coeur, 11 design sectors were identified. Slope stability analyses were undertaken to define achievable slope configurations for each sector.

A slope design summary for the Abundancia Pit is included below:

- Bench Face Angle: 55 to 70°
- Bench Width: 6.5 to 10 m
- Bench Height: 10 to 15 m (double or triple benching 5 m benches)
- Interramp Angle: 36 to 49° (for heights up to 150 m)
- Overall Slope Angle: 31 to 45°

1.7 MINERAL PROCESSING AND METALLURGICAL TESTING

Metallurgical testing prior to 2013 was summarized and reported in the PEA. An extensive drilling program and metallurgical investigation was initiated in support of this feasibility study. The data summarized here are from this recent investigation, and although the PEA data analysis supports this recent investigation, the older data are not included in this report summary.

The analysis for feasibility level metallurgical studies for the recovery of silver and gold is reported here along with conclusions. The deposit is amenable to conventional process technology and will be processed in a conventional crushing, grinding and Merrill-Crowe silver recovery circuit with detoxified tailings reporting to a tailings storage facility.

The metallurgical and mineralogy investigations were developed by the following techniques: XRD, QEMSCAN, clay and near infrared analyses, flotation, comminution parameters, variability and development sodium cyanide leaching, bulk sodium cyanide leaching, sodium cyanide detoxification, Merrill-Crowe simulation, flocculent screening, conventional, high rate and paste thickening, vacuum and pressure filtration, pressure clarification and slurry rheology.

Mineralogy Highlights:

- The gold grains are contained in iron hydroxide that is enclosed in quartz-rich particles.
- The majority of the located gold occurrences in the samples would be considered to be liberated.
- Gold particles precipitated in direct association with pyrite and non-sulfide gangue minerals were generally low in gold content and were of smaller average diameter than the liberated gold particles. Some of this gold was found as inclusions within the pyrite or gangue particle and may not be amenable to cyanide leaching. Gold was detected in association with silver sulfide minerals.
- The major portion of the silver in this sample occurs as silver sulfide (acanthite, Ag_2S). Traces of silver–selenium sulfide (possibly aguilarite) and complex silver–antimony–zinc sulfide were also observed. Silver-bearing grains range from 15 -100 µm in size. The silver-bearing grains observed and described here are locked in quartz or are associated with pyrite, galena and lead carbonate, and iron oxide or iron hydroxide. Acanthite also occurred as very fine veins within quartz.
- Minerals detected by short-wave infrared and visible light include: copiapite, cerussite, epidote, goethite, gypsum, hematite, illite, illite-NH4, jarosite, kaolinite, montmorillonite, nontronite, quartz, saponite. Oxidation state did not correlate with silver dissolution.

Bond crusher work index values were measured from 65 whole core samples. The Bond crusher work index averaged 9.7 kWh/t ± 3.4 and varied from 21.3 to 5.2 kWh/t for all samples tested.

Bond abrasion index (Ai) tests were completed on 47 samples. Ai results ranged from 0.37-1.27 g, and averaged 0.74 ± 0.25g.

Bond ball mill work index (BWi) tests were completed on 47 samples. BWi results ranged from 14.7–18.2 kWh/t and averaged 16.1 ± 1.0 kW/t.

Bond rod mill work index (RWi) tests were completed on 36 samples. RWi results ranged from 12.7–18.1 kWh/t and averaged 15.3 ± 1.4 kW/t.

Sag mill comminution tests were completed.

Silver dissolution testing in the variability study consisted of 141 bottle roll tests with samples ground to a particle size P_{80} of ~45 µm and 23 bottle roll tests with the samples ground to a particle size P_{80} of ~74 µm. The ground material was leached for 72 hours at a pulp density of 40% solids in 1.5 gpl NaCN at a pH of 11.5. Lead nitrate was not added.

The exploratory cyanide leach series investigated various alternative leaching schemes to determine silver and gold recoveries or improve metal leaching kinetics. The exploratory leaches included feed and residue size classification, baseline cyanide leaching, two-stage leaching, hot leaching, leaching of sand and slime size fractions, O_2 enriched leaching, and pressure conditioning and pressure leaching.

Thirteen leach tests were completed as standard NaCN bottle roll tests conducted for 72 or 96 hours at pH 11.5, pulp density of 40 wt% and ambient temperature. The test series evaluated three particle sizes P_{80}; 30, 45, and 75 *um*, and two sodium cyanide concentrations 1.5 and 4.0 gpl.

Feasibility study silver and gold recovery was calculated based on head grade versus tail grade regression of the Variability Average Grade, Lithology Elevation bench, and selected Exploratory bottle roll tests.

The weighted silver and gold recovery was determined to be 84% and 61%, respectively, based on life of mine head grades.

The average cyanide consumption was determined to be 1.0 kg/t. Lime consumption was determined to be 1.4 kg/t. The flocculant consumption was determined to be 0.085 kg/tonne.

Sodium cyanide detoxification reagent consumptions were determined to be 0.8 kg sodium metabisulfite/t and 0.07 kg copper sulfate pentahydrate/t.

1.8 RECOVERY METHODS

The mill feed will be delivered from the mine at a rate of 10,000 dtpd. A primary jaw crusher will crush the ore. Crushed ore will be ground in a closed SAG and ball mill grinding circuit. Ground material will be leached with sodium cyanide in agitated leach tanks. Soluble precious metals are washed from the leached slurry then recovered in the Merrill-Crowe process. The precious metal precipitate will be smelted to produce doré bars. Tailings will be thickened then routed to either the cyanide destruction circuit or to the tailing storage facility. A basic flow sheet of the process can be seen in Figure 1-2.

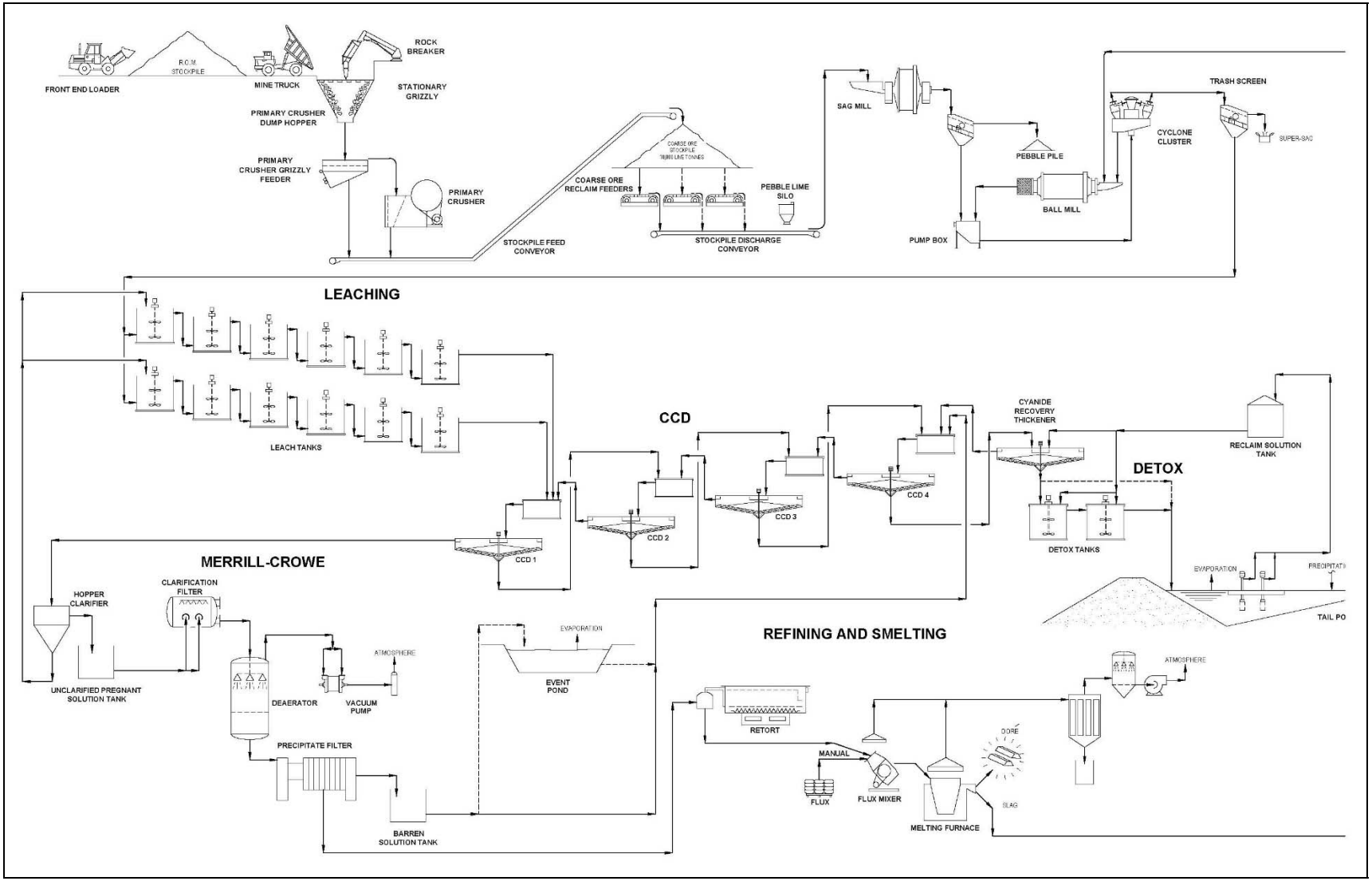


Figure 1-2: Project Process Flow Sheet

1.9 PROJECT INFRASTRUCTURE

It is contemplated that there will be several surface buildings constructed to support the mining and process operations. These facilities include administration, guardhouse, truck shop, warehouse, change house, explosives storage buildings, and truck wash and mill maintenance facilities.

A new access road and utility corridor has been anticipated for the site, and is shown on Figure 5-1. This road will connect to the site from an existing exit off of Mexican Federal Highway Route 40 at the exit to the community of Vicente Suarez. This mine access road will use a portion of the existing community road and then deviate from that road to take a more direct route to the mine. Total road to be constructed is ~14 km of which ~12.5 km will be new construction and 1.1 km of the existing road will be improved. The final road will be about 50 km from Durango to the mine.

Due to heavy transport to the site during construction, the road is anticipated to be 10 meters wide of driving surface, consisting of improved and compacted gravel, with drainage ditches. With exception of replacement or by-pass of one existing bridge on the portion of the existing road to Vicente Suarez, the remaining arroyo crossings are assumed to be “low water” crossings with concrete base.

The site is currently equipped with a radio communications tower that belongs to the site and has radio capacity to communicate back to the town of Durango. This radio tower was constructed by prior owners and consists of a dedicated 34.5 m fenced tower, solar power, with multi frequency and repeater capacity and has a second tower located on the roof of the Durango office building to assure 2-way communications to the site.

The Project’s location limits the ability to secure firm pipeline natural gas supply due to the unavailability of firm capacity in the nearby pipeline. Other options for natural gas delivery, including Compressed Natural Gas (CNG) and Liquefied Natural Gas (LNG), could provide firm gas supply, but would result in an all-in power cost that exceed published Comisión Federal de Electricidad (CFE) tariffs for electricity.

Three options are being explored for supply of electrical connections to the Project:

1. CFE has proposed to provide power from the future Canatlán Potencia substation located about 42 km west-northwest from the site. Canatlán Potencia substation is planned to have a capacity of 100 MW, and will step-down the voltage level from 230 kV to 115 kV. From the substation a 42 km long, 115 kV transmission line will feed the Project. A 115 kV power line could be constructed using wood poles or steel towers as favored by CFE with a 477 kCMIL size, ACSR type conductor. CFE offered to build the transmission line in 34 months, including the new Canatlán Potencia substation. It is typical for a mining project client, or their contractor, to build the transmission lines and related substations and then turn them over to CFE for their operation and maintenance. The construction of any infrastructure that needs to be turned over to CFE, switching substation and transmission line will need to be inspected and accepted by CFE technical personnel during and after construction and commissioning of the facilities.
2. There is an existing 115 kV power line at a distance of about 9 km from the mine site. It is currently providing power to the city of Guadalupe Victoria. This option would require construction of a new branch, running to the mine site at a length of about 11 km. Connecting to this power line is not feasible since it does not have sufficient power capacity.
3. The 230 kV power line connected to the national grid is located at a distance of about 9 km from the mine site. Construction of a branch line with a length of about 11 km will be required. Positive discussions are continuing with CFE. This option will also require construction of a new switching substation nearby route 40. The new 230 kV transmission line to the Project will follow the new access road to the north. The line will

need to be designed and constructed on steel tower with a minimum ACSR conductor size of 1113 circular mils (kcmil).

Option 3 is preferred due to shortened construction, potentially lower capital cost, and lessened right-of-way acquisition issues. Engineering could start once CFE approves final connection point and the route is defined. Design specifications and requirements will be based on CFE normalized standards based on substations and transmission line voltages.

The Project Substation would consist of two 30 MVA 230 kV to 13.2kV power transformers. Each transformer would be sufficient to power the entire plant load allowing one transformer to be taken down for maintenance at any given time without affecting the plant operations. Stepdown transformers from 13.2 kV to 4160 V or 480 V will be located at the individual process areas.

Site power distribution will be at 13.2 kV and will consist of both underground distribution and overhead lines. Underground distribution will be used to supply power to the main process areas near the substation while overhead lines will be used to provide power to the outlying areas such as primary crushing, stockpile, water tanks, and tailing. Wells and well water booster pumps will be fed by a 34.5 kV overhead line from a 13.2 kV to 34.5 kV step up transformer in the main substation.

A water source for the Project has been located in the Valle de Guadiana aquifer, and two wells were drilled to supply the anticipated approximately 80 L/s of plant and mine demand. Those two wells were drilled on farm land and all agreements are in place with the land-owners for the use of the land. Water rights have been obtained for approximately one half of the required water. The remaining water rights applications are being reviewed by the government agency CONAGUA.

Power for the two wells is anticipated to cost approximately US$180,000 and consist of extension of existing power lines that go to Vicente Suarez and also service other agricultural wells in the area. This will include posts, wire, transformers and control for the primary well pumps and the booster pumps.

A booster tank will be located so as to serve both the existing wells with an installation of two in-line booster pumps such that water can be pumped directly to the site.

Water pipeline design from the well booster tank to the site will cross easements with local landowners and the Vicente Suarez Ejido and will be constructed of steel in the lower portion and HDPE in the upper portion as pressure requirements drop.

1.9.1 Water Balance

A steady state water balance was prepared for inclusion in the environmental permitting application by SRK Consulting (SRK) of Tucson. The water balance indicates that the Project will be a net water user and that the site will rely on make-up water pumped from the water supply wellfield, stormwater run off ponds, and from groundwater inflows from pit dewatering. Water is primarily used in the plant process circuit and the tailings disposal system, but will also be used for dust suppression of haulage and access roads and of drilling rig operations. Reliance on well water will decline after Year 4 as the main pit (Abundancia) deepens and greater inflow water becomes available. Reclaiming the maximum water possible from the TSF pond, reducing water usage across the property, and recycling water, wherever feasible, are key components of the design of the plant and TSF facilities.

1.10 PROJECT EXECUTION PLAN

The proposed project execution plan incorporates an integrated strategy for engineering, procurement and construction management (EPCM). The primary objective of the execution methodology is to deliver the Project at the lowest capital

cost, on schedule, and consistent with the Project standards for quality, safety, and environmental compliance. Coeur would provide a Project Management Office for oversight and project controls through project turnover to operations.

Execution of the Project is based on several steps, some occurring simultaneously. It is anticipated that there will be 18 months of field construction to commencement of commissioning. Project location is sufficiently close to a major city that no camp is required for housing of employees.

- The detailed engineering schedule is based on interim approval to be granted in early Basic Engineering starting in the first quarter (Q1) of Y-2, and full EPCM release in Q3 of Y-2. The design is scheduled to be 60% complete by the end of Q4 Y-2. Engineering will be completed upon receipt of all pertinent vendor data.
- Procurement of long lead items (i.e. SAG mill, ball mill, leach agitators) would begin in Q2 Y-2, ensuring vendor data available for detailed engineering in a timely manner.
- Contractor mobilization for earth moving begins as soon as the environmental permitting is released at the beginning of Q3 Y-2.
- Coeur training of mine operators, mine pre-stripping activity, and access road works would begin as soon as permits allow. Water well preparation would begin immediately to provide construction water requirements.
- The construction program is scheduled to start in Q3 Y-2. Contract bid calls will start in Q2 Y-2. The work includes clearing and grubbing of the plant site, mass earthwork for site development, project access road and in-plant roads. Concrete foundations and underground utilities for the process buildings and other support structures will be constructed beginning in early Q4 Y-2.
- Construction work is scheduled for 18 months from ground breaking to the commencement of commissioning. Earthworks for process facilities will commence as soon as the contractor can be mobilized to the field, after all required permits have been obtained. This work will include surface water control diversions, process building foundations and tailings facilities.

The EPCM integrated approach includes project controls and services brought together at the appropriate times to ensure a successful project.

- Contracting – A combination of vertical (e.g., all trades for laboratory), horizontal (e.g., electrical trade for process area), and design construct (e.g., fire protection) contracts may be employed as best suits the work to be performed, degree of engineering and scope definition available at the time of award. Concrete batch plants are expected to be based on-site.
- Expediting, Logistics, and Inspections – The EPCM contractor will be responsible to expedite the receipt of vendor drawings to support the engineering effort as well as expediting the fabrication and delivery of major equipment to the site. An expediting report will be issued at regular intervals outlining the status of each purchase order in order to alert the Project of any delays in the expected shipping date or issue of critical vendor drawings. Corrective action can then be taken to mitigate any delay.
  The EPCM contractor will also ensure that qualified personnel inspect critical equipment before shipments are made.
- Quality Plans, Health and Safety Plans, and Commissioning plans will be developed before construction activity starts.

1.11 CAPITAL COST ESTIMATE

Initial capital costs have been estimated for Project in compliance with feasibility level design based on the associated material quantities, labor cost estimates and equipment quotations. Unit rates have been based on historic data, published sources and inputs from Coeur's operations in Mexico. The estimate includes all evaluated sections of the Project such as process, tailing, mining, and maintenance facilities. The costs also include pre-production mining, owner's costs and contingency. A more detailed initial capital breakdown is located in Section 21 of this report as well as in the economic model.

Table 1-9: Initial Capital Cost

Area	Description	($ Millions)
Process Plant and General Infrastructure	General Site, Mine support infrastructure, tailings disposal, primary crushing, grinding, water systems, water treatment, on site power distribution, ancillary facilities, EPCM, freight, import duties, capital and commissioning spares.	$234.3
Mine	Mining operating costs before the start of production including lease costs and royalty purchase.	$53.0
Owners Costs	Land Acquisition, Construction/ Operating Camps, Environmental Permits, Initial Fills, Coeur's Project Management, Security, Early Staffing, Community Relations.	$12.2
Contingency	Contingency on all process components of the project.	$28.1
Escalation	Not included in this estimate.	$0
Total Capital Cost	IVA (Valued added tax (VAT)) not included	$327.6

1.12 OPERATING AND MAINTENANCE COST ESTIMATE

The operating cost was estimated in compliance with feasibility level requirements based on bottom-up methodology. The operating cost summary is provided in Table 1-10, which includes the total cost and cost per tonne processed over the Life of the Mine (LOM).

Table 1-10: Process Operating Cost Summary

LOM Production		
Process Tonnes	37,326,000	
	LOM Cost	**Unit Cost per Tonne**
Primary Crushing	$13,791,511	$0.37
Grinding	$249,351,177	$6.68
Leach & CCD	$140,342,316	$3.76
Merrill Crowe	$34,808,681	$0.93
Refining & Smelting	$18,773,563	$0.50
Tailings Disposal	$51,357,659	$1.38
Ancillary	$40,105,825	$1.07
Total	**$548,530,732**	**$14.70**

1.13 ECONOMIC ANALYSIS

The economic analysis for the Project assumes an 18-month construction period, followed by a production period of 11 years, and reclamation and closure in Year 12. Total initial capital totals $327.6 million and sustaining capital equals $126.6 over the life of the Project.

Annual revenue in the economic analysis is calculated by applying forecast metal prices to the estimated annual payable metal produced in each operating year. No escalation or hedging is assumed in the analysis.

The Project is assumed to be subject to a Mexican corporate income tax rate of 30%, consistent with current income tax laws enacted in Mexico. Total income tax payments over the life of the Project equal $102.1 million. The Project is also assumed to be subject to two newly enacted mining royalty taxes: a 0.5% royalty tax on revenue from precious

metal, and a 7.5% royalty tax calculated on earnings before interest, depreciation and amortization. These taxes represent additional $12.5 million and $67.7 million payments, respectively, to the government over the life of the Project.

The average Total Cash Cost over the LOM is estimated to be $15.16 per ounce of silver processed. Total Cash Costs includes mine operations, process plant operations, general and administration cost, smelting and refining charges, and transportation costs. The estimated cash cost by area per ounce of silver produced is summarized in Table 1-11.

Table 1-11: Production Costs

		Life of Mine Average
Cash Operating Costs		
Direct Mining Cost	$ per oz. produced	7.88
Direct Mine Equipment Leases	$ per oz. produced	1.08
Direct Process Plant	$ per oz. produced	5.14
G&A	$ per oz. produced	0.82
Subtotal	$ per oz. produced	14.92
Cash Offsite Costs		
Shipping and Selling	$ per oz. produced	0.07
Treatment and Refining	$ per oz. produced	0.17
Subtotal	$ per oz. produced	$0.24
Total Cash Costs	$ per oz. produced	$15.16

A sensitivity analysis was completed for a range of factors including metal prices, grade, foreign exchange fluctuations, operating costs capital costs and other assumptions. The Project was determined to be most sensitive to metal prices. Table 1-12 provides a tabular summary of the NPV and IRR under the base case, and also under the metal price scenarios utilized in the economic analysis.

Table 1-12: Sensitivity Analysis

		NPV @ 5% (thousands)	IRR	Payback (years)
Base Case		$87,605	9.5%	6.9
Metal Prices	+20%	$326,665	20.3%	3.0
	-20%	-$179,304	-5.5%	N/A
Operating Cost	+20%	-$54,409	2.1%	9.8
	-20%	$225,335	15.9%	4.3
Capital Cost	+20%	$23,443	6.0%	8.8
	-20%	$143,607	14.0%	6.5
Recovery	+20%	$323,192	20.2%	3.0
	-20%	-$174,362	-5.2%	N/A

1.14 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

A number of site characterization and engineering studies have been completed and/or initiated as part of the feasibility studies and in support of the environmental permit applications and approvals. The inventories, site characterization studies, monitoring plans, and management plans were prepared for the Project area and adjacent land in compliance with required components of the main environmental documents that must be submitted to the federal government and

approved prior to mine development. The primary environmental documents include the Environmental Impact Statement (Manifestación de Impacto Ambiental or MIA), the Forest Land Use Modification (ETJ), and Risk Assessment (ER). Two MIA application documents are needed – one for the road access and utility corridor to the mine, and the other for the mine site itself. Further environmental work will continue at the site in the form of pre-mining baseline monitoring programs.

Baseline characterization has been initiated to establish pre-mine ambient conditions and to prepare monitoring and management plans to mitigate the disturbance and potential impacts to the environment in the Project area. PMLP has constructed a weather station to monitor the site climate conditions. Biological and wildlife inventories were completed over 14,261 hectares including scrub forest, natural and planted grassland, and the dry-land and irrigated agricultural areas to identify the nature of the existing flora and fauna and to assess the presence of protected species. No historic or archeological resources were found during these surveys. A census of local wells was completed to establish the local potentiometric surface and hydrogeologic gradient. The wells were sampled to provide baseline water quality characteristics. A preliminary assessment of the site aquifer properties was established by completing packer tests in select geotechnical boreholes. Surface water quality samples were taken in the Project area from a spring, several man-made reservoirs, and from water accumulating in a nearby volcanic crater. Preliminary geochemical characteristics of the mined waste materials were established by analyzing 50 waste rock samples compiled from drill core and from two tailings samples created from metallurgical test residues. Neither the waste rock nor tailings is considered to be toxic per Mexican regulations. Several samples of the volcanic waste rock were measured to be potentially acid generating; the tailings samples are acid neutralizing. Additional test work is recommended to confirm the geochemical characteristics for operations management and for closure planning.

A *Conceptual Closure and Reclamation Plan* has been prepared (SRK, 2014c) that outlines key activities prior to closure and during the closure and post-closure period. At present it is anticipated that the post-closure land use will be wildlife habitat, grazing, and potential future mining of underground resources. Undisturbed portions may also be used for dry farming uses. The areas of the open pits, tailings, and former plant site would be restricted from public access.

The closure process will include decommissioning, demolition, and rehabilitation. The decommissioning process would begin at the early stages of closure and would include the decommissioning of all cyanide-containing materials and equipment. The general reclamation process would be to break and backfill foundations and sumps, recontour for positive drainage, cover the regraded areas with growth medium, re-establish natural drainages, and revegetate with native species. A closure plan for the post-closure pit lake will be developed once sufficient hydrogeological and geochemical site data are available.

A reclamation and closure cost was estimated for third-party implementation. Rehabilitation of the major facilities was estimated in the cost estimate for the process plant, tailing impoundments, waste rock areas, infrastructure (buildings, roads, and landfill), contaminated areas, and utilities. The reclamation and closure process is estimated to cost approximately US$11.1 million including a 20% contingency that applies to the closure activities but not to the post-closure care and maintenance.

Health and safety issues are being monitored by PMLP's Safety Department. Standardized training is being developed to train employees and contractors and for tracking and reporting safety incidents. By 2013, employees, consultants, and contractors had worked 51 months and 201,600 man hours without a lost-time injury or a medical-treatment case.

PMLP has developed a community relations program in alignment with Coeur's corporate mission, vision, and values. Project personnel are in consultation with government agency representatives, local property owners, local community leaders, and the ejidos. Key stakeholders who are interested in being benefitted or impacted by the Project have been identified; interest to date in the Project has been neutral to strongly supportive. The community relations program has worked to communicate consistently about the Project Plans with the communities and interested parties, to provide

information and training about care for the environment, to participate in local social programs (medical support and educational activities), and to assist with land acquisition, and to provide assistance in the environmental permitting process.

1.15 LAND ACQUISITION AND STATUS

1.15.1 Land and Surface Property Regulations

Provided all the necessary rights in and to applicable surface estates and governmental permits are acquired, a Mining Concession grants the holder (Holder), the right to conduct all phases of exploration, exploitation, and processing (beneficiation) of a mineral deposit.

The Mining Law provides that exploration, exploitation, and processing of the minerals and substances allowed under the Ley Minera are preferred over any other types of land use. The rights granted by a Mining Concession do not include rights in and to the surface estate overlying and corresponding to the dimensions of the respective Mining Concession, which underlies said surface estate. Proper control of the surface estate is a key issue in terms of environmental permitting, exploration, exploitation, and beneficiation activities. Generally, the surface estate has to be either purchased or leased from the respective surface estate owner. Alternatively, pursuant to Article 19, Section IV of the Ley Minera, a Holder may also acquire rights in and to the surface estate by obtaining the legal expropriation, temporary occupancy or creation of a land easement needed to carry out the exploration, exploitation, and beneficiation works, as well as for the deposit of waste rock dumps, tailings, slag's and slag dumps, and underground rights-of-way through adjacent mining concessions.

1.15.2 Location and Mineral Tenure

The Project consists of eight (8) Mining Concessions that encompass approximately 1,120 hectares (ha). PMLP holds 100% of the registered, legal, and beneficial interest in and to these Mining Concessions. PMLP also holds 100% of the registered, legal, and beneficial interest in and to three (3) additional Mining Concessions that are adjacent to, and contiguous with, the Project. These three additional Mining Concessions encompass approximately 31,300 hectares (ha). Combined, the eleven (11) Mining Concessions of PMLP comprise approximately 32,400 ha and together with approximately 1,800 ha of surface estates controlled, represent the Consolidated Property Package (Property Package).

Table 1-13 provides notable information about the Mining Concessions. Figure 1-3 depicts the location of the Mining Concessions, which comprise the Property Package.

Table 1-13: La Preciosa Mining Concessions

№	Name	Expedient #	Title #	Issue Date	Area (Has.)	Semiannual Mining Duties[1]	Expiry Date
1.	La Preciosa	002/00398	182517	1988-07-15	143.6119	$18,560.40	2038-07-14
2.	Lupita	009/00303	182584	1988-08-12	27.1878	$3,513.75	2038-08-11
3.	Fracción La Preciosa	321.1/2-399	185128	1989-12-14	2.5249	$327.00	2039-12-13
4.	San Patricio	321.42/919	189616	1990-12-05	29.4740	$3,810.00	2053-07-01
5.	La B	2/1.3/01962	214232	2001-09-06	28.2006	$3,646.00	2051-09-05
6.	El Choque Tres	2/1.3/02251	218953	2003-01-28	10.0000	$1,293.00	2053-01-27
7.	El Choque Cuatro	25/30812	220251	2003-07-02	629.7778[2]	$81393.00	2053-07-01
8.	El Choque Seis	25/31144	220583	2003-09-02	249.0000	$32,182.00	2053-09-01
9.	Santa Monica	025/31208	221288	2004-01-20	16,385.4570	$2,117,657.00	2054-01-19
10.	Santa Monica Sur	025/31411	223097	2004-10-14	900.0000	$66,097.00	2054-10-24
11.	San Juan	025/31434	226663	2006-02-17	14,003.4737	$1,028,416.00	2056-02-16

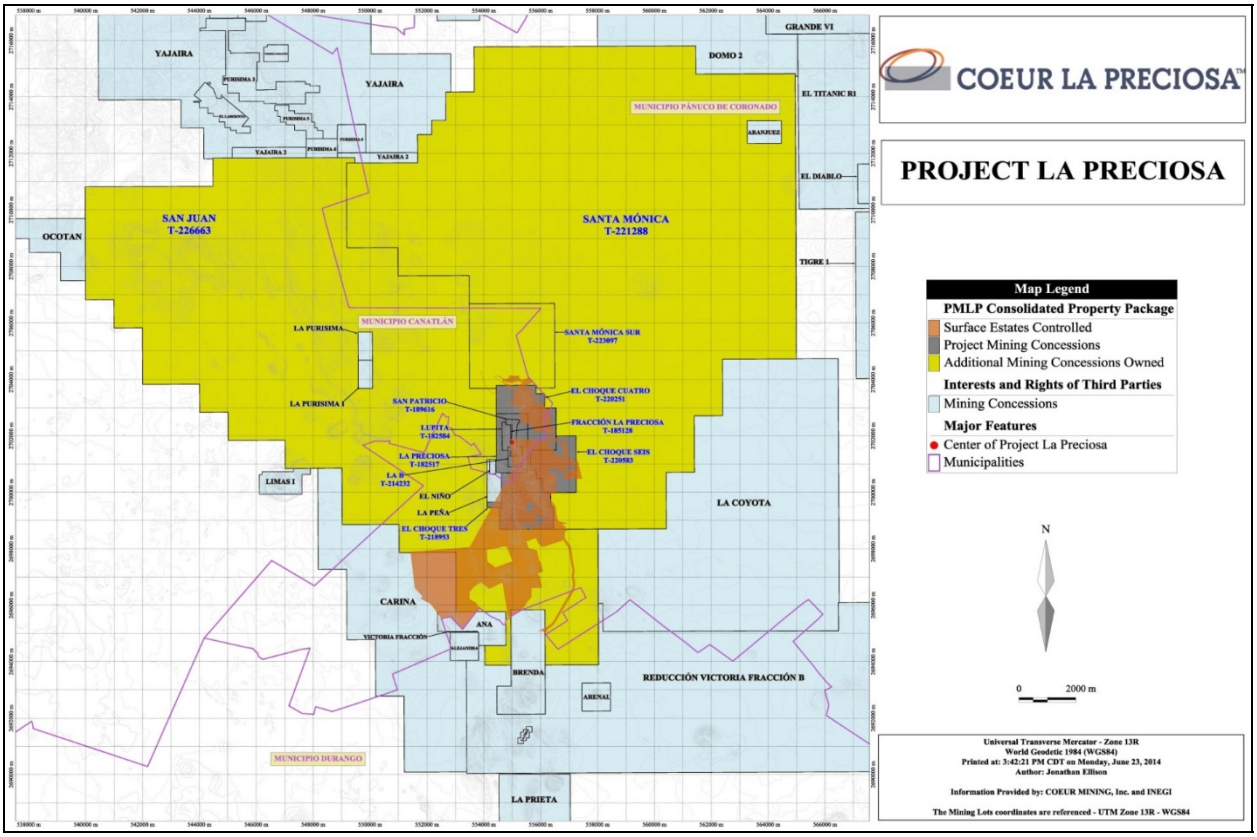


Figure 1-3: Map of the Project Consolidated Property Package

[1] 2014 Mining Duties assessed semiannually. All amounts are in Mexican Pesos.
[2] Pursuant to a subsequent "Corrección Administrativa de Título" the size of El Choque Cuatro is 629.7778 hectares. The Mining Concession's Tarjeta erroneously reflects 644.1296 hectares. The Dirección General de Minas is being petitioned to administratively correct the Tarjeta error.

The Dirección General de Minas (General Bureau of Mines) administers Mining Concessions in Mexico. A legal survey (Trabajos Periciales) of each Mining Concession was completed as a requirement of, and condition precedent to, the General Bureau of Mines granting such Concession.

Payment of Mining Duties are required for each Mining Concession and, each year, are payable semiannually in January and July to the Secretaría de Economía (Secretariat of Economy). The Mining Duties are calculated by determining the correct Cuota[3] (Fee), which vary based upon the age of the Mining Concession. The Fee is then multiplied by the number of hectares encompassed by the Mining Concession, the product of which equals the semiannual Mining Duty payable for the respective Mining Concession. A copy of the payment receipt of the Mining Duty must be filed with the Dirección General de Regulación Minera, a sub-directorate of the General Bureau of Mines, semiannually, each February and August.

Annually, in May, owners of Mining Concessions must file with the Dirección de Revisión de Obligaciones, a sub-directorate of the General Bureau of Mines, Informes Para Comprobar La Ejecución de Las Obras y Trabajos (Work Assessment Reports), disclosing the works to, and investments made in, each Mining Concession or sanctioned Agrupamiento (Group) of Mining Concessions, for the immediately preceding calendar year. The Regulations promulgated under the Mining Law establish tables containing the minimum investment amounts for each Mining Concession or Group(s) of Mining Concessions. These amounts are updated annually in accordance with the changes to the Mexican Consumer Price Index (CPI) and published in the Diario Oficial De La Federación in December, providing the Cuotas Fija (flat fee) and Cuotas Adicional (additional flat fee), which determine the total investment the owner is obligated to make in each Mining Concession or Agrupamiento thereof, for the succeeding year. In 2014, PMLP must invest a minimum of MXN$70,590,535.53 in Agrupamiento La Preciosa, which consist of all eleven (11) Mining Concessions.

Informes Estadístico Sobre La Producción, Beneficio y Destino de Minerales o Sustancias Concesibles (Production Reports), detailing the production, beneficiation, and destination of concessionable minerals, must be submitted by annually by January 30. These reports must be submitted for each Mining Concession bearing production and all Mining Concessions with over six (6) years of age, whether bearing production or not.

The surface estates overlying the Project are owned by a mixture of ejidos[4] and private parties.

1.15.3 Issuer's Interest

On February 20, 2013, Coeur announced it was entering into a definitive agreement pursuant to which Coeur would agree to acquire all of the issued and outstanding common shares of Orko, the parent company of PMLP, in a transaction with a total value of approximately CAD$350 million.

On April 16, 2013, Coeur announced the completion of its acquisition of Orko pursuant to its previously announced plan of arrangement, detailed in the news release on February 20, 2013. As a result of the completion of the arrangement, Coeur now owns all of the issued and outstanding shares of Orko.

PMLP remains a wholly owned subsidiary of Orko, which was renamed Coeur La Preciosa Silver Corp.

1.15.4 Royalties, Back-in Rights, Payments, Agreements, and Encumbrances

A Consulting and Finder's Fee Agreement dated May 01, 2002, between Silver Standard Resources, Inc., the predecessor in interest of PMLP and La Cuesta International, Inc. (LCI). In accordance with the terms thereof, PMLP

[3] The Base Rate or "Fee" is adjusted annually and published the Diario Oficial De La Federación each December for use in calculating Mining Duties payable due the following year.
[4] An ejido is one of two types of social property in Mexico, granted by the government that combines communal ownership with individual use. The ejido consists of common use land, community development land, and individual parcels, which may be assigned to ejido members.

pays a Finder's Fee to LCI, comprised of: (I.) an advance royalty, every six (6) months, equal to the greater of (i.) USD$5,000 or (ii.) 2% of direct exploration costs in and to the Mining Concession San Juan, and (II.) a one-quarter of one percent (0.25%) NSR royalty on production derived from the Mining Concession San Juan, which is located adjacent to the Property Package. The maximum amount payable under the terms of the Agreement is USD$2,000,000. PMLP has the right, at any time, to acquire from LCI all of the NSR payable in respect to LCI, including any amount remaining payable under the NSR.

Numerous *Contratos de Ocupación Temporal Para La Extracción, Explotación, Uso y Aprovechamiento de Las Fuentes de Agua del Subsuelo* (Temporary Occupancy Agreements) with local ejidos and land owners have been executed. All agreements executed to date have terms of occupancy and use of between 25 to 30 years.

1.15.5 Adjacent Properties

Arcelia Gold Corp., through its wholly owned subsidiary, Arcelia Gold, S.A. de C.V. owns two (2) Mining Concessions (La Peña, and El Niño) that are adjacent to and contiguous with the El Choque Tres, El Choque Cuatro, La Preciosa, and San Juan Mining Concessions of PMLP.

Canasil Resources, Inc., through its wholly owned subsidiary, Minera Canasil, S.A. de C.V. owns two (2) Mining Concessions (Carina, and Reducción Victoria Fracción B) that are adjacent to and contiguous with the San Juan Mining Concession of PMLP.

1.16 MARKET CONSIDERATIONS

The Project will produce silver and gold doré, which will be trucked from the mine site to the refinery.

Coeur has received quotes for the secure transportation and refining of the silver-gold doré bars produced at the Project from parties with which it has existing business relationships. Given the Project's proximity to Coeur's Palmarejo mine, the rates for secure transportation are similar to what Coeur is currently paying its secure transportation provider in the region. The quote included in the financial analysis includes a fixed charge of $14,500 per shipment (up to 7,500 kg) plus a liability charge of $0.30 per thousand of amount declared.

Refining terms include a treatment charge of $0.17/oz based on the net weight of the doré bars received by the refinery; (ii) a metal return of 99.7% of recoverable gold; and (iii) 99.92% of recoverable silver. Refining losses attributable to melt losses and differences in assays with the refiner is assumed to be 0.5%. This is based on Coeur's refiner loss experience at Coeur's San Bartolomé mine. No penalties are expected to be assessed against the refining. The refined silver and gold is expected to be available for Coeur to sell eight business days after receipt of the doré bars by the refinery.

Coeur plans to sell the refined silver and gold from the Project primarily to financial institutions, including multi-national banks and bullion trading houses. The markets for both refined silver and gold are highly liquid.

1.17 PROJECT RISKS AND OPPORTUNITIES

1.17.1 Risks

The Project is anticipated to be a large scale (150,000 mtpd) open pit operation feeding about 10,000 mtpd to a processing plant. The majority of the ore mineralization is contained within veins that will be mined and separated from the waste rock. There is risk in the continuity and grade of those veins and risk in the high strip ratio if mining costs are not well controlled.

Pursuit of the purchase or control of the necessary and convenient surface estates that overlie the Project is ongoing. There are risks that some of these surface estates, or portions thereof, may not be acquired due to unrealistic expectations of the parties, uncured or incurable defects in the legal land title, and/or survey and legal description inaccuracies. Many of the surface estates overlying the Project have been secured by long-term leasehold agreements. The application for the MIAs (environmental documents) can be pursued when some minimal issues regarding water and power are resolved.

Coeur has four water wells on the Project site, and an additional two water wells approximately 10 km south of the Project which have been drilled and water rights applied for. Conagua, the Mexican water agency, has received Coeur's application for water rights and has acknowledged receipt in writing. The two wells south of the Project have demonstrated capacity to produce on the order of 70 liters per second, which, in conjunction with capture of precipitation at the site, and dewatering of the pit, may represent sufficient water for the Project. Coeur is also investigating opportunities to treat and use municipal and industrial waste water available near the Project to futher supplement supplies. As of the date of this technical report Conagua is processing Coeur's application for industrial use of water from its wells.

Electrical Power is available from the local grid, either from an existing 230 kV power line located approximately 12 km south of the Project via a connection to a 115 kV power line located approximately 41 km to the west of the Project. However, the local governmental agency responsible for power supply and connections (CFE) has not responded to requests for connections that have been proposed by the company. Power can be obtained at lower rates than is being offered by CFE via wheeling of power from other sources through the CFE grid.

Environmental permit activities need to be completed and the anticipated approval time would be about 6-9 months.

Mining Claim maintenance fees and activities are required to maintain the status of mining concessions. This involves exploration or other activities on the property as well as annual reports. In the event of project delay, those activities would have to be included in the on-going activities and expenses at the site.

Mexican mining tax and royalty rates increased in recent years but regulations governing those new laws are still not available and may change interpretations of how those laws will be enforced or put into effect. Those tax and royalty laws were recently enacted and at a time of low metal prices. There is about two more years of the current government before the next major elections. Those elections may result in a relaxation, a continuation or other changes in those tax and royalty laws.

1.17.2 Opportunities

Due to the current lower metals price environment, the availability of mining and processing equipment and associated terms and conditions for acquiring that equipment are very advantageous.

The start-up of production will take approximately 2 years from date of approval. This is a quick, but attainable, timetable for bringing the production on-line. The plant and mine design are state-of-the-art but also uses well-established technologies.

Extended mine life is likely with improved metals prices due to the nature of the deposit. The deposit is not closed at depth and still has some inferred resources that could be added to the ultimate mine production. There are multiple veins within the deposit, and a quality Ore Control program could bring additional resources into the production matrix. There is additional exploration potential in the region and Coeur holds a large mining concession position.

The property is close to population centers without being too close, is close to power and transportation infrastructure, and has a ready workforce available locally and in the region.

Reduced water needs are likely due to a combination of water in-flow into the pit at depth and from the capture of rain water during the rainy season.

The opportunity to stockpile waste rock within mined out portions of the pits can reduce the haulage profile and thereby reduce the mining costs and equipment acquisition costs for the Project.

Power costs are available at better rates than are being offered by CFE. The quoted reduced rates are included in the current evaluation but there is a highly competitive market developing in the energy sector that could result in further reductions in those costs.

1.18 CONCLUSIONS AND RECOMMENDATIONS

On the basis of the results of this feasibility study, it is recommended to consider completing basic engineering. The Project has an estimated net present value of $87.6 million at a 5% discount rate, with an estimated internal rate of return of 9.5%, and a projected Project payback period of 6.9 years after taxes at the metal prices used for this study. Having basic engineering completed will allow the rapid start-up of construction, when the company decides to proceed with the Project.

Key up-front activities during the basic engineering period include:

Environmental

- Continue Baseline environmental monitoring, including the monitoring of the piezometers and weather station. Additional geochemical testing of tailings residues by humidity cells is recommended to confirm the long-term geochemical behavior. Initiate the air monitoring with contractors to have baseline information.
- Complete Archeological Survey.
- Reinitiate MIA applications for road construction and Project.

Land

- Continue with land acquisitions and right of ways (ROWs) for road and ancillary activities (power/water).
- Maintain mining claim rights through exploration programs.

Power

- Pursue 230 kVA connection point approval with CFE.

Metallurgy

It is recommended metallurgical developments continue to support optimization of process parameters:

- Optimization of leach parameters with Lithology and Variability composites; commercial tank pressure leach, temperature, and particle size.
- Evaluation of alternate technology: CELP, or equivalent, provided the development is stepwise, and without commercial obligations prior to proving the technology.
- Continue development of the geometallurgy model with additional sampling, bottle roll, and QEMSCAN analysis.
- An allowance for additional metallurgical test work is approximately $600,000.

Plant Design

- Initiate Basic Plant Design.

Water Rights

- Continue process to obtain water rights or confirmation of water rights dependent upon operation/approval of the Project.

Tailing Design

- Collection of site specific meteorological and hydrology data. These data will be used to refine seasonal run off values and design storms to be used in future detailed design work.
- Optimization of the water balance to incorporate updated run off and process flow estimates, in order to maximize the available reclaim back to the process. The TSF water balance should be combined with the site wide water balance for the Project.
- Additional site investigations should be completed as part of detailed design. The investigations should focus on construction materials and foundation conditions, with particular focus on the potential for large voids to be present in the near surface bedrock within the basin and below the embankments.
- The tailings materials and properties should be reviewed during detailed design to be sure they are representative, especially if any changes to the process occur or as more representative tailings samples become available.
- Development of a full closure plan for the TSF based on the final design configuration.

2 INTRODUCTION

This technical report (this Report) has been prepared by M3 Engineering & Technology Corporation (M3) for Coeur Mining, Inc. (Company, Coeur, or Coeur Mining) in compliance with the disclosure requirements of Canadian National Instrument 43-101 (NI 43-101), to disclose relevant information about the Property in Durango, Mexico. This information has resulted from an updated mineral resource estimate, and a Feasibility Study (FS) engineering detail. This Report is intended to disclose the recently updated mineral resources and declaration of mineral reserves at the Project and the results of the FS.

2.1 AUTHORS

Coeur contracted a team of qualified consultants to assemble this technical report. Table 2-1 lists the Qualified Persons (QP's) involved with this Report.

Table 2-1: Qualified Persons for this Report

Qualified Person	Registration	Company	Sections of Responsibility	Site Visit
Daniel H. Neff	P.E.	M3 Engineering & Technology Corporation	Section 1*	June 8, 2010
Conrad E. Huss	P.E., Ph.D.	M3 Engineering & Technology Corporation	Sections 2, 3, 4, 5, 18, 19, 21, 22, 24, 25*, 26* and 27.	None
David Tyler	SME-RM	Coeur Mining, Inc.	Sections 15 and 16, and corresponding items in sections 1*, 25* and 26*.	August 20-22, 2013 November 5-7, 2013 February 13, 2014
Robert Bruce Kennedy	P.E.	Coeur Mining, Inc.	Sections 6 and 23, and corresponding items in sections 1*, 25* and 26*.	July 22, 2014
Erin Patterson	P.E.	M3 Engineering & Technology Corporation	Section 17, and corresponding items in section 1*.	None
Tracy Barnes	P.E.	Barnes Engineering Services, Inc.	Section 14, and corresponding items in sections 1*, 25* and 26*.	None
Christopher L. Easton	MMSA-Q.P. Metallurgy	Easton Process Consulting, Inc.	Sections 13 and 24*, and corresponding items in sections 1*, 25* and 26*.	November 4-8, 2013 December 2-6, 2013
Dana Willis	SME-RM	Coeur Mining, Inc.	Sections 7, 8, 9, 10, 11, 12, corresponding items in sections 1*, 25* and 26*.	November 5-7, 2013 December 3-5, 2013 January 22-23, 2014
Corolla Hoag	C.P.G., SME-RM	SRK Consulting (U.S), Inc.	Section 20*, and corresponding items in sections 1*, 25* and 26*.	October 29, 2013 - November 1, 2013

*Indicates that some pieces of this section were developed by another.

2.2 SOURCES OF INFORMATION

This Report is based in part on internal company technical reports, the PEA, feasibility studies, maps, published government reports, company letters and memoranda, and public information as listed in the references section in the conclusion of this Report.

2.3 UNITS AND TERMS OF REFERENCE

This Report uses metric units expressed in metric tonnes, meters, and liters consistent with metric standards. The monetary units are expressed in US Dollars. The important terms used in this Report are presented in Table 2-2.

Table 2-2: Units, Terms and Abbreviations

Above mean sea Level	amsl
Acid-base accounting	ABA
Acid-generation potential	AP
Acid rock generating	ARD
Acidity	pH
Ampere	A
Annum (year)	a
Atomic absorption	AA
Atomic absorption spectroscopy	AAS
Below ground surface	bgs
Billion pounds	Glb
Billion years ago	Ga
Billion	G
Bond abrasion index	Ai
Bond ball mill work index	BWi
Bond rod mill work index	RWi
Cédula de Operación Annual (Annual Operations Report)	COA
Centimeter	cm
Centimeters per second	cm/s
Certificate of Ancestral Domain Title	CADT
Coeur Mining, Inc.	Coeur
Comisión Federal de Electricidad	CFE
Comisión Nacional del Agua (National Water Commission)	CONAGUA
Compressed natural gas	CNG
Concentrados Industriales, S.A. de C.V.	CI
Copper	Cu
Counter current decantation	CCD
Crushed Ore Stockpile	COS
Cubic centimeter	cm^3
Cubic meter	m^3
Cubic meters per day	m^3/d
Cubic meters per hour	m^3/hr
Dacite porphyry	DAP
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel	dB
Degree	°
Degrees Celsius	°C
Development Rock Stockpile	DRS
Dry metric tonne	dmt
Dry tonnes per day	dtpd
Electromotive Force	emf
Engineering, Procurement, and Construction Management	EPCM
Equivalent per tonne	/t
Estudio de Riesgo (Risk Assessment)	ER
Foot/feet	ft
Forest Land Use Modification	ETJ
Gallon	gal
Gallons per minute	gpm
General & Administration	G&A
G-force (seismic)	g
Giga (billion)	G
Gigajoule	GJ
Gold	Au
Grams	g
Grams per litre	g/L
Grams per tonne	g/t
Greater than	>
Hard Rock Consulting	HRC
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Independent Mining Consultants	IMC
Independent Power Producer	IPP
Inductively Coupled Plasma	ICP
Inductively Coupled Plasma – Atomic Emission Spectrometry	ICP-AES
Inductively Coupled Plasma – Mass Spectrometry	ICP-MS
Inductively Coupled Plasma – Optical Emission Spectrometry	ICP-OES

Internal Rate of Return	IRR
International Finance Corporation	IFC
Intra-mineral dacite porphyry	IDAP
Intra-mineral hornblende diorite porphyry	IHDP
Joule	J
Joules per kilowatt-hour	J/kWh
Kelvin	K
Kilo (thousand)	k
Kilobyte	kB
Kilogram	kg
Kilograms per cubic meter	kg/m^3
Kilograms per hour	kg/h
Kilograms per year	kg/a
Kilojoule	kJ
Kilometer	km
Kilometers per hour	km/h
Kilonewton	kN
Kilopascal gauge	kPa(g)
Kilopascal	kPa
Kiloton	kt
Kilovolt	kV
Kilovolt ampere	kVA
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne	kWh/t
Kilowatt hours per year	kWh/a
Knight Piésold	KP
Less than	<
Ley Federal de Responsabilidad Ambiental (Federal Law of Environmental Responsibility)	LFRA
Ley General del Equilibrio Ecológico y la Protección al Ambiente	LGEEPA
Life of Mine	LOM
Liquefied natural gas	LNG
Lithium metaborate fusion	LMF
Litre	L
Litres per hour	L/hr
Litres per minute	L/min
Litres per second	L/s
Load-Haul-Dump	LHD
Lower Volcanic Complex	LVC
M3 Engineering & Technology Corporation	M3
Major Drilling Group International Inc	Major
Manifestación de Impacto Ambiental (Environmental Impact Statement)	MIA
Material Take-off	MTO
Mega (million)	M
Megabyte	MB
Megabytes per second	MB/s
Megapascal	MPa
Megavolt ampere	MVA
Megawatt	MW
Megawatt hours	MWh
Metal leaching	ML
Meter	m
Meters above sea level	masl
Meters per minute	m/min
Meters per second	m/s
Metric tonne	mt
Micrometer (micron)	?m
Microsiemen (electrical)	?s
Milliamperes	mA
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimeter	mm
Millimeters per hour	mm/h
Million	M
Million cubic meters	Mm^3
Million litres	ML
Million tonnes	Mt
Million years before present	Ma
Minute (plane angle)	'
Minute (time)	min
Month	mo
Movement Magnitude (of an earthquake)	Mw
National Instrument 43-101	NI 43-101
Net Present Value	NPV
Net Smelter Prices	NSP
Net Smelter Return	NSR
Neutralization Potential	NP
Neutralization Potential Ratio	NPR
Newton	N
Newtons per meter	N/m
NOM-001-SEMARNAT-1996	NOM-001
NOM-021-SEMARNAT-2000	NOM-021
NOM-052-SEMARNAT-2005	NOM-052
NOM-053-SEMARNAT-1993	NOM-053
NOM-141-SEMARNAT-2003	NOM-141
NOM-147-SEMARNAT-2007	NOM-147
NOM-157-SEMARNAT-2009	NOM-157
Norma Oficial Mexicana	NOM
Ounce	oz
Oxidation-Reduction Potential	ORP
Pan American Silver	PAS
Parts per billion	ppb
Parts per million	ppm
Pascal (newtons per square meter)	Pa
Pascals per second	Pa/s
Percent	%
Potential Acid Generating	PAG
Pound(s)	lb
Preliminary Economic Assessment	PEA
Probable Maximum Flood	PMF

Probable Maximum Precipitation ..... PMP
procedimiento de extracción de constituyentes tóxicos (toxic constituents extraction procedure) ..... PECT
Procuraduría Federal de Protección al Ambiente (Federal Environmental Protection Agency) .....PROFEPA
Project management office.....PMO
Proyectos Mineros La Preciosa S.A de C.V. ..... PMLP
Qualified Persons .....QPs
Quality Assurance/Quality Control..... QA/QC
Quartz-Sericite-Chlorite..... QSC
Reverse Circulation .....RC
Right of way..... ROW
Rock Quality Designation .....RQD
Run of mine ..... ROM
Second (plane angle) ..... "
Second (time) ..... s
Secretaría de Medio Ambiente y Recursos Naturales .....SEMARNAT
semi-autogenous grinding ..... SAG
Shake Flask Extraction.....SFE
Sierra Madre Occidental.....SMO
Silver..... Ag
Specific gravity .....SG
Square centimeter cm2
Square kilometer km2
Square meter m2
SRK Consulting (U.S.), Inc. ..... SRK
Tailing Storage Facility ..... TSF
Thousand tonnes..... kt
Thousands of circular mils.....kcmil
Tonne (metric, 1,000 kg = 2205 lb) .....t
Tonnes per cubic meter .....t/m^3
Tonnes per day.....tpd
Tonnes per hour .....t/h
Tonnes per year .....t/a
Toronto Stock Exchange .....TSX
Total dissolved solids .....TDS
Total suspended solids.....TSS
Troy ounce (31.1035 g) ..... oz
Unified Soil Classification System .....USCS
Value added tax..... (IVA or VAT)
vibrating wire piezometer.....VWP
Volt V
watt..... W
weak acid dissociable..... WAD
Week ..... wk
Weight percent ..... wt%
Weight/weight .....w/w
Wet metric tonne ..... wmt
Yard..... yd
Year (annum) .....a
Year (U.S.) .....yr

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons responsible for this report as outlined in Section 2 did not rely on the reports, opinions, or statements of any experts that are not QPs.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION AND MINERAL TENURE

The Project is located in the state of Durango, Mexico, within the municipalities of Pánuco de Coronado and Canatlán, and is approximately 85 km by existing road, northeast of the city of Victoria de Durango, the state capital. A new access road is anticipated to be constructed which would reduce this to a distance of 50 km and a time of 45 minutes.

The Project is situated on the eastern flank of the Sierra Madre Occidental mountain range and can be found on the Instituto Nacional de Estadística, Geografía e Informática (INEGI) General Carlos Real Topographic Map G13D72 and is centered on coordinates 24°25'42.4200"N Latitude and 104°27'27.2380"W Longitude (554,987.8815mE, 2,701,771.0046mN) in the Universal Transverse Mercator (WGS 84), Zone 13R (Northern Hemisphere). A General Location Map of the Project is provided in Figure 4-1.

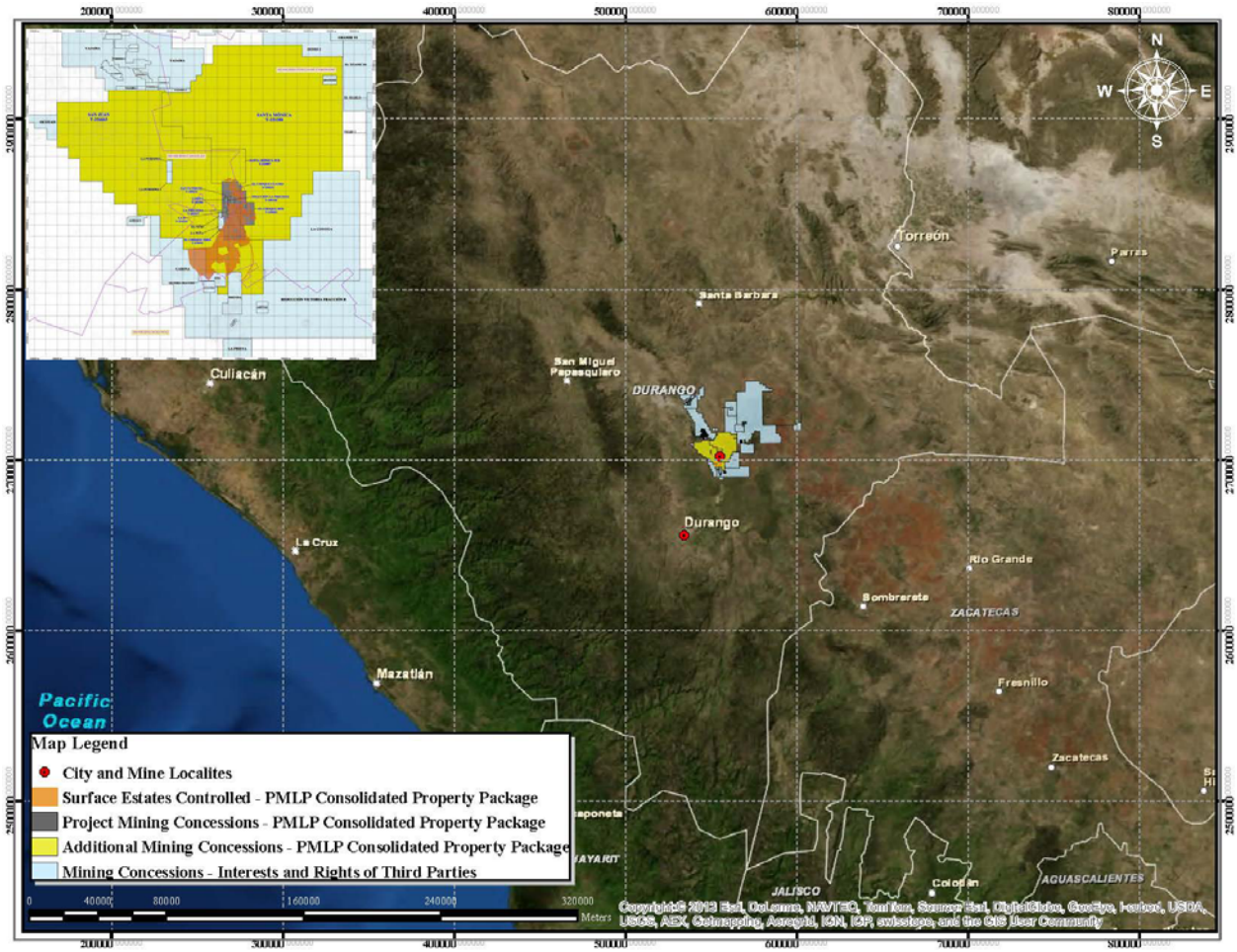


Figure 4-1: General Location Map

The Project consists of eight Mining Concessions that encompass approximately 1,120 hectares (Has.). PMLP holds 100% of the registered, legal, and beneficial interest in and to these Mining Concessions. PMLP also holds 100% of the registered, legal, and beneficial interest in and to three (3) additional Mining Concessions that are adjacent to, and contiguous with, the Project. These three additional Mining Concessions encompass approximately 31,300 hectares (Has.). Combined, the eleven (11) Mining Concessions of PMLP comprise approximately 32,400 hectares (Has.) and,

together with approximately 1,800 hectares (Has.) of surface estates controlled, represent the Project consolidated property package (Property Package).

Table 4-1 provides notable information about the Mining Concessions. Figure 4-2 depicts the location of the Mining Concessions and surface estates controlled, which comprise the Property Package.

Table 4-1: Project Property Package

№	Name	Expedient #	Title #	Issue Date	Area (Has.)	Semiannual Mining Duties[5]	Expiry Date
1.	La Preciosa	002/00398	182517	1988-07-15	143.6119	$18,560.40	2038-07-14
2.	Lupita	009/00303	182584	1988-08-12	27.1878	$3,513.75	2038-08-11
3.	Fracción La	321.1/2-399	185128	1989-12-14	2.5249	$327.00	2039-12-13
4.	San Patricio	321.42/919	189616	1990-12-05	29.4740	$3,810.00	2053-07-01
5.	La B	2/1.3/01962	214232	2001-09-06	28.2006	$3,646.00	2051-09-05
6.	El Choque Tres	2/1.3/02251	218953	2003-01-28	10.0000	$1,293.00	2053-01-27
7.	El Choque Cuatro	25/30812	220251	2003-07-02	629.7778[6]	$81393.00	2053-07-01
8.	El Choque Seis	25/31144	220583	2003-09-02	249.0000	$32,182.00	2053-09-01
9.	Santa Monica	025/31208	221288	2004-01-20	16,385.4570	$2,117,657.00	2054-01-19
10.	Santa Monica Sur	025/31411	223097	2004-10-14	900.0000	$66,097.00	2054-10-24
11.	San Juan	025/31434	226663	2006-02-17	14,003.4737	$1,028,416.00	2056-02-16

[5] 2014 Mining Duties assessed semiannually. All amounts are in Mexican Pesos.
[6] Pursuant to a subsequent "Corrección Administrativa de Título" the size of El Choque Cuatro is 629.7778 hectares. The Mining Concession's Tarjeta erroneously reflects 644.1296 hectares. The Dirección General de Minas is being petitioned to administratively correct the Tarjeta error.

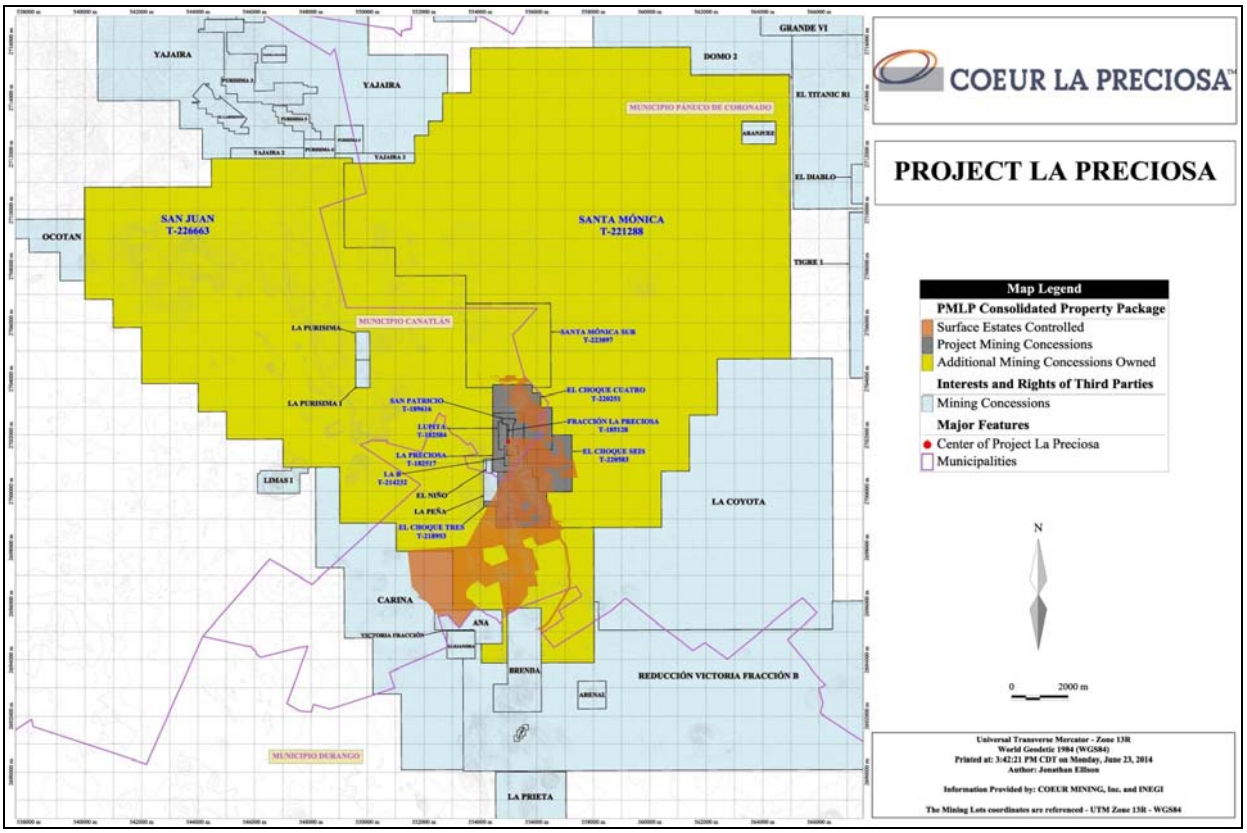


Figure 4-2: Map of the Project Consolidated Property Package

The Dirección General de Minas (General Bureau of Mines) administers Mining Concessions in Mexico. A legal survey (Trabajos Periciales) of each Mining Concession was completed as a requirement of, and condition precedent to, the General Bureau of Mines granting such Concession.

Pursuant to an amendment of the Mexican Mining Law (Law), by Congressional Decree of February 22, 2005, which was published in the Diaro Oficial de la Federación April 28, 2005, there is no longer any distinction between an Exploration Concession and an Exploitation Concession. Consequently, all Concessions are "Mining Concessions" (Exploration and Exploitation), and as a result, all Exploration and Exploitation Concessions have been converted into Mining Concessions, expiring fifty years (50) from the date they were originally granted.

Payment of Mining Duties are required for each Mining Concession and, each year, are payable semiannually in January and July to the Secretaría de Economía (Secretariat of Economy). The Mining Duties are calculated by determining the correct Cuota[7] (Fee), which vary based upon the age of the Mining Concession. The Fee is then multiplied by the number of hectares encompassed by the Mining Concession, the product of which equals the semiannual Mining Duty payable for the respective Mining Concession. A copy of the payment receipt of the Mining Duty must be filed with the Dirección General de Regulación Minera, a sub-directorate of the General Bureau of Mines, semiannually, each February and August.

Annually, in May, owners of Mining Concessions must file with the Dirección de Revisión de Obligaciones, a sub-directorate of the General Bureau of Mines, Informes Para Comprobar La Ejecucion de Las Obras y Trabajos (Work Assessment Reports), disclosing the works to, and investments made in, each Mining Concession or sanctioned

[7] The Base Rate or "Fee" is adjusted annually and published the Diario Oficial De La Federación each December for use in calculating Mining Duties payable due the following year.

Agrupamiento (Group) of Mining Concessions, for the immediately preceding calendar year. The Regulations of the Mining Law establish tables containing the minimum investment amounts for each Mining Concession or Group(s) of Mining Concessions. These amounts are updated annually in accordance with the changes to the Mexican Consumer Price Index (CPI) and published in the Diario Oficial De La Federación in December, providing the Cuotas Fija (flat fee) and Cuotas Adicional (additional flat fee), which determine the total investment the owner is obligated to make in each Mining Concession or Group thereof, for the succeeding year. In 2014, PMLP must invest a minimum of MXN$70,590,535.53 in Agrupamiento La Preciosa, which consist of all eleven (11) Mining Concessions.

Informes Estadístico Sobre La Producción, Beneficio y Destino de Minerales o Sustancias Concesibles (Production Reports), detailing the production, beneficiation, and destination of concessionable minerals, must be submitted by annually by January 30. These reports must be submitted for each Mining Concession bearing production and all Mining Concessions with over six (6) years of age, whether bearing production or not.

The surface estates overlying the Project are owned by a mixture of ejidos[8] and private parties.

4.2 ISSUER'S INTEREST

On February 20, 2013, Coeur Mining, Inc. (Coeur) announced it was entering into a definitive agreement pursuant to which Coeur would agree to acquire all of the issued and outstanding common shares of Orko, the parent company of PMLP, in a transaction with a total value of approximately CAD$350 million.

On April 16, 2013, Coeur announced the completion of its acquisition of Orko pursuant to its previously announced plan of arrangement, detailed in the news release on February 20, 2013. As a result of the completion of the arrangement, Coeur now owns all of the issued and outstanding shares of Orko.

Proyectos Mineros La Preciosa, S.A. de C.V. remains a wholly owned subsidiary of Orko.

4.3 ROYALTIES, BACK-IN RIGHTS, PAYMENTS, AGREEMENTS, AND ENCUMBRANCES

- A Consulting and Finder's Fee Agreement dated May 01, 2002, as amended ((Agreement) by and between Silver Standard Resources, Inc., the predecessor in interest of and to PMLP and La Cuesta International, Inc. (LCI). In accordance with the terms thereof, PMLP pays a Finder's Fee to LCI, comprised of: (I.) an advance royalty, every six (6) months, equal to the greater of (i.) USD$5,000 or (ii.) 2% of direct exploration costs in and to the Mining Concession San Juan (Título #226663) and (II.) a one-quarter of one percent (0.25%) NSR royalty on production derived from the Mining Concession San Juan (Título #226663), which is located adjacent to the Project, if any. The maximum amount payable under the terms of the Agreement is USD$2,000,000. PMLP has the right, at any time, to acquire from LCI all of the NSR payable in respect to LCI, including any amount remaining payable under the NSR. LCI shall not sell, transfer or otherwise assign all or any portion of its interest in the NSR ((NSR Interest) to any other party without first offering the NSR Interest to PMLP. Reciprocally, PMLP shall not sell, transfer or otherwise assign all or any portion of its interest in and to the Mining Concession to any other party without first offering the Mining Concession to LCI;
- A Net Smelter Return Royalty Agreement dated June 19, 2002 ((Sanluis Agreement #1) by and among Minas Luismin S.A. de C.V., Minas Sanluis, S.A. de C.V., collectively, the owner and the predecessor in interest of and to PMLP and Corporación Turística Sanluis, S.A. de C.V., the holder and predecessor in interest of and to SANLUIS Corporación, S.A.B. de C.V. In accordance with the terms thereof, the Owner conveyed a three percent (3%) NSR royalty ((Sanluis Royalty #1) on production to the holder, derived from Mining Concessions El Choque Tres (Título #218953) and La B (Título #214232), if any. Sanluis Royalty #1is a covenant that runs with, and binds, these two (2) Mining Concessions and the legal title thereto, the owners thereof, and their

[8] An ejido is one of two types of social property in Mexico, granted by the government that combines communal ownership with individual use. The ejido consists of common use land, community development land, and individual parcels, which may be assigned to ejido members.

successors and/or assigns. The Sanluis Agreement #1 provides that owner has a right of first refusal to acquire the Sanluis Royalty #1 if the holder receives a bona fide proposal to acquire the Sanluis Royalty #1 from a third party;

- On June 12, 2014 SANLUIS Corporación, S.A.B. de C.V. extended to Coeur its right of first refusal pursuant to the terms, covenants, and obligations of the Sanluis Agreement #1. Coeur exercised its right of first refusal and on July 2, 2014 repurchased the Sanluis Royalty #1 encumbering the two (2) Mining Concessions El Choque Tres (Título #21895) and La B (Título #214232) for USD$12,000,000.00. The repurchase price also reflects the concurrent extension and exercise of the right of first refusal of the Sanluis Royalty #2 described immediately hereinbelow.
- A Net Smelter Return Royalty Agreement dated June 19, 2002, ((Sanluis Agreement #2) by and among Minas Luismin S.A. de C.V., Minera Thesalia, S.A. de C.V., collectively, the owner and the predecessor in interest of and to PMLP and Corporación Turística Sanluis, S.A. de C.V., the holder and predecessor in interest of and to SANLUIS Corporación, S.A.B. de C.V. In accordance with the terms thereof, the Owner conveyed a three percent (3%) NSR royalty ((Sanluis Royalty #2) on production to the holder, derived from Mining Concessions La Preciosa (Título #182517), Lupita (Título #182584), Fracción La Preciosa (Título #185128), and San Patricio (Título #189616), if any. Sanluis Royalty #1 is a covenant that runs with, and binds, these four (4) Mining Concessions and the title thereto, the owners thereof, and their successors and/or assigns. The Sanluis Agreement #2 provides that owner has a right of first refusal to acquire the Sanluis Royalty #2 if the holder receives a bona fide proposal to acquire the Sanluis Royalty #2 from a third party;
- On June 12, 2014 SANLUIS Corporación, S.A.B. de C.V. extended to Coeur its right of first refusal pursuant to the terms, covenants, and obligations of the Sanluis Agreement #2. Coeur exercised its right of first refusal and on July 2, 2014 repurchased the Sanluis Royalty #2 encumbering the four (4) Mining Concessions La Preciosa (Título #182517), Lupita (Título #182584), Fracción La Preciosa (Título #185128), and San Patricio (Título #189616) for USD$12,000,000.00. The repurchase price also reflects the concurrent extension and exercise of the right of first refusal of the Sanluis Royalty #2 described immediately hereinabove.
- On June 12, 2013 PMLP executed a Contrato de Ocupación Temporal Para La Extracción, Explotación, Uso y Aprovechamiento de Las Fuentes de Agua del Subsuelo (Temporary Occupancy Agreement) with Fernando Rivas Cossío, an ejidatario of the ejido Vicente Suarez ((Posesionario). This Temporary Occupancy Agreement covers approx. five and nine-tenths (5.9) hectares, and has a term of thirty (30) years from June 12, 2013. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Posesionario is USD$75,000. PMLP has prepaid the annual rent through June 12, 2018;
- On June 12, 2013 PMLP executed a Temporary Occupancy Agreement with Fernando Rivas Cossío, an ejidatario of the ejido Vicente Suarez ((Posesionario). This Temporary Occupancy Agreement covers approx. eight and four-tenths (8.4) hectares, and has a term of thirty (30) years from June 12, 2013. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Posesionario is USD$75,000. PMLP has prepaid the annual rent through June 12, 2018;
- On July 23, 2013 PMLP executed a Temporary Occupancy Agreement with Alejandro Hernández Jarquín ((Propietario). This Temporary Occupancy Agreement covers approx. one thousand two hundred eighteen and five-tenths (1,218.5) hectares, has a term of twenty-five (25) years from July 23, 2013, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, a lump sum payment was tendered upon the execution thereof, equaling MXN$16,463,966.40, and represents the sum total consideration due from the Compañía thereunder. The land encumbered by this Temporary Occupancy Agreement overlies a portion of the Mining Concession El Choque Tres (Título #218953), El Choque Cuatro (Título #220251), La B (Título #214232), La Preciosa (Título #182517), and San Juan (Título #226663);

- On February 07, 2014 PMLP executed a Temporary Occupancy Agreement with Unidád Comercial Agricola y Ganadera Don Joaquin S. de R.L. de C.V. ((Propietario). This Temporary Occupancy Agreement covers approx. twenty (20) hectares, has a term of twenty-five (25) years from February 07, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Propietario is USD$18,000. PMLP has prepaid the annual rent through February 09, 2019. On February 10, 2019, PMLP must prepay the ensuing five (5) year's annual rent, through February 09, 2024. Thereafter, there is no longer an obligation to prepay the annual rent. The land encumbered by this Temporary Occupancy Agreement overlies a portion of the Mining Concession San Juan (Título #226663);
- On February 07, 2014 PMLP executed a Temporary Occupancy Agreement with Jorge Soto Enríquez ((Propietario). This Temporary Occupancy Agreement covers approx. six (06) hectares, has a term of twenty-five (25) years from February 07, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement the annual rent payable to the Propietario is USD$5,400. PMLP has prepaid the annual rent through February 09, 2019. On February 10, 2019, PMLP must prepay the ensuing five (5) year's annual rent, through February 09, 2024. Thereafter, there is no longer an obligation to prepay the annual rent. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), San Juan (Título #226663), and Santa Monica (Título #221288);
- On February 07, 2014 PMLP executed a Temporary Occupancy Agreement with Jorge Soto Enríquez ((Propietario). This Temporary Occupancy Agreement covers approx. ninety-four (94) hectares, has a term of twenty-five (25) years from February 07, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Propietario is USD$84,600. PMLP has prepaid the annual rent through February 09, 2019. On February 10, 2019, PMLP must prepay the ensuing five (5) year's annual rent, through February 09, 2024. Thereafter, there is no longer an obligation to prepay the annual rent. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), San Juan (Título #226663), and Santa Monica (Título #221288);
- On February 13, 2014 PMLP executed a Temporary Occupancy Agreement with Petra Higareda Briceño Viuda de García ((Propietario). This Temporary Occupancy Agreement covers approx. fifty-four and nine-tenths (54.9) hectares, has a term of twenty-five (25) years from February 13, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Propietario is USD$49,422.93. PMLP has prepaid the annual rent through February 13, 2019. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251) and La Preciosa (Título #182517);
- On February 18, 2014 PMLP executed a Temporary Occupancy Agreement with ejido Lázaro Cárdenas ((Ejido). This Temporary Occupancy Agreement covers approx. one hundred fifty-seven and two-tenths (157.2) hectares, has a term of thirty (30) years from February 18, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Ejido is MXN$785,944.95. The annual rent shall be adjusted annually in accordance with the changes to the Mexican Consumer Price Index (CPI). The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), El Choque Seis (Título #220583), and Santa Monica (Título #221288);
- On February 19, 2014 PMLP executed a Temporary Occupancy Agreement with ejido Francisco Javier Mina ((Ejido). This Temporary Occupancy Agreement covers approx. eighty-nine and two-tenths (89.2) hectares,

has a term of thirty (30) years from February 19, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Ejido is MXN$445,846.69. The annual rent shall be adjusted annually in accordance with the changes to the Mexican Consumer Price Index (CPI). The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), El Choque Seis (Título #220583), and Santa Monica (Título #221288);

- On April 11, 2014 PMLP executed a Temporary Occupancy Agreement with Candelaria Uves Solórzano ((Propietario). This Temporary Occupancy Agreement covers approx. two hundred eighteen and eight-tenths (218.8) hectares, has a term of twenty-five (25) years from April 11, 2014, and expressly allows for the exploration, exploitation, and beneficiation of concessionable minerals. In accordance with the terms of the Temporary Occupancy Agreement, the annual rent payable to the Propietario is USD$196,911.89. PMLP has prepaid the annual rent through March 28, 2019. However, beginning March 28, 2015 and continuing until March 28, 2019, PMLP must prepay, in each of those years, for the future lease periods from March 29, 2019 through March 28, 2024. On March 28, 2024, PMLP must prepay the annual rent for the ensuing five (5) years or until March 28, 2029. On March 28, 2029, PMLP must prepay the annual rent for the ensuing five (5) years or until March 28, 2034. The last annual rent payment under the terms of the Temporary Occupancy Agreement is scheduled to be made March 28, 2034, a prepayment of the annual rent for the last year of the term of the Temporary Occupancy Agreement, 2039. The land encumbered by this Temporary Occupancy Agreement overlies, to varying degrees, portions of the Mining Concessions El Choque Cuatro (Título #220251), El Choque Seis (Título #220583), Fracción La Preciosa (Título #185128), La B (Título #214232), La Preciosa (Título #182517), San Patricio (Título #189616), and Santa Monica (Título #221288).

There are no other known royalties, back-in rights, payments, agreements, or encumbrances.

4.4 ENVIRONMENTAL LIABILITIES AND PERMITS

Please refer to Section 20 for a discussion regarding environmental and permitting factors related to the Project.

4.5 SIGNIFICANT FACTORS AND RISKS

Pursuit of the purchase or control of the necessary and convenient surface estates that overlie the Project is ongoing. There are risks that some of these surface estates, or portions thereof, may not be acquired due to unrealistic expectations of the parties, uncured or incurable defects in the legal land title, and/or survey and legal description inaccuracies.

The accuracy and completeness of ownership records maintained by the several Registros Públicos de la Propiedad y del Commercio (RPPyC) and Direcciones de Catastro within the state of Durango varies greatly. Prior to commencing negotiations for the purchase or control of a surface estate, legal land titles are thoroughly abstracted to determine legal ownership and the defects affecting validity of said ownership. Many Certificados and Constancias, issued by the several RPPyC, Direcciones de Recaudación, and Registros Agrario Nacional (RAN), are requested and obtained, in order to cross reference our own research with that of these government entities. Any disparities between the two are flagged for curing or ameliorating the title risk(s).

Well before consummating the purchase or leasehold transaction, each surface estate parcel is surveyed in the field using high-precision equipment manufactured by Trimble Navigation, LTD. Any discrepancies between the survey results, legal descriptions within the chain of title, and/or previous surveys are analyzed and curative actions are taken to formally reconcile and/or correct the legal dimensions of said surface estate.

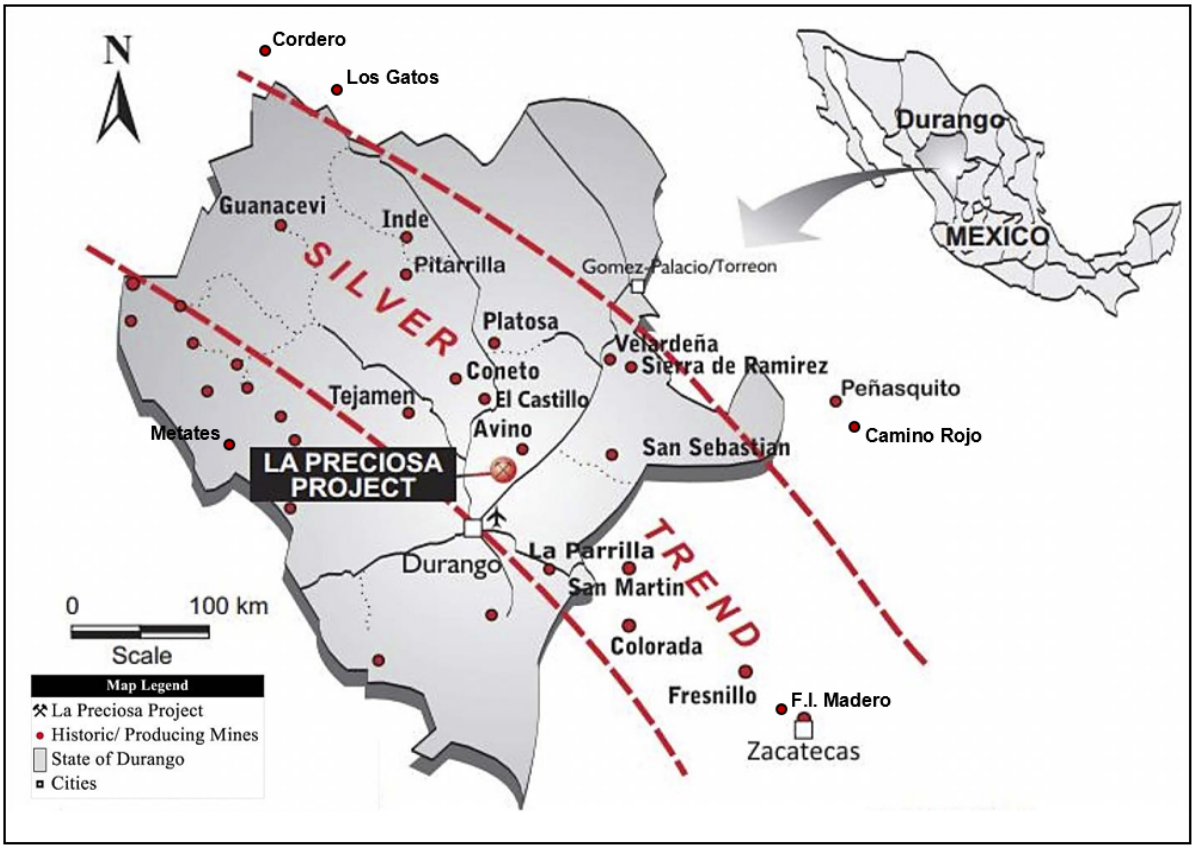
Many of the surface estates overlying the Project have been secured by long-term leasehold agreements.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 LOCATION ACCESS

The Property is located approximately 85 km by existing roads, northeast of the city of Durango and can be accessed by vehicle from Durango in approximately 90 minutes. Figure 5-1 shows the Project's regional location. A Google satellite map showing the location of the Property relative to Durango and the access roads is shown in Figure 5-2. From Durango, travel northeast toward Torreón by sealed Federal Highway 40, to the town of Francisco I. Madero. From this point, a secondary paved road is followed northwest to the village of Lázaro Cárdenas, then by newly paved road to the village of Francisco R. Serrano. After 9 km, turn and traverse a newly paved road in a southwest direction to the village of Francisco Javier Mina, then travel in a southerly direction for 5.5 km by gravel road to the access road to the Project site. The access road is a 3.5 km gravel road heading southeast and leads to the portal of the historic workings and the main camp of the Project.

A much shorter site access road is planned to go from Highway 40 directly to the site and avoid the small communities. This road is also anticipated to be aligned with the anticipated water pipeline route. Land acquisitions and road right of way with local landowners are in progress.



Source: Coeur, 2013

Figure 5-1: Project Location Map

(Source: Google maps, 2010)

Figure 5-2: Property location and access map (Satellite)

5.2 CLIMATE AND LENGTH OF OPERATING SEASON

The Property area has a semi-arid climate, with an annual average temperature of about 25°C and an average annual precipitation of about 600 millimeters (mm), usually occurring between May and October. Temperatures can fall below freezing on winter nights but snow is rare. Mining activities can take place year round. The dominant wind direction is southeast.

5.3 PROXIMITY TO POPULATION CENTER AND TRANSPORT

Durango is the capital city of Durango State and has a population of over 600,000. One of the major industries in Durango is mining, particularly for silver, and the region is expected to be a good source of skilled personnel, support services, and mining equipment. The city of Durango is served by an international airport with daily flights connecting to destinations in México and connections to the United States. Durango is situated along Mexican Federal Highway 40, which connects Durango to Mazatlán, approximately 310 km to the southwest on the Pacific coast, and to Torreón, approximately 245 km to the northeast. A railway line runs between Durango and Torreón and connects to other cities in México and the United States.

5.4 SURFACE RIGHTS, LAND AVAILABILITY, INFRASTRUCTURE, AND LOCAL RESOURCES

5.4.1 Surface Rights, Land Availability, and Mining Areas

The Project area includes suitable sites for construction of the plant and infrastructure for the mine. A plan of the proposed pits, dumps, TSF, plant, and infrastructure is shown in Figure 5-3.

Surface rights in the Project area are held by a combination of private landowners, ejidos, and ejidatarios, which are ejido members with rights to use specific tracts of land within the ejido. Ejidos are areas of communal land used for

agriculture where community members jointly control rights to access and use the land. Ejidos are registered with Mexico's National Agrarian Registry.

Figure 5-3: Plan of Proposed Pits, Dumps, Tailings Dam, Plant, and Infrastructure

5.4.2 Ownership

Coeur has verified who the registered land owners are who hold surface rights within the boundaries of the proposed mining activities. Contacts have been established with all registered surface holders and negotiations have been completed or are in progress with all involved individuals and/or groups.

5.4.3 Power, Infrastructure, and Water

The quality of infrastructure and the population density increases towards the city of Durango. The Property is not currently connected to the commercial electrical grid, but the nearby village of Francisco Javier Mina (population around 920) and the town of Francisco I. Madero (population around 4,550) are serviced by the commercial electrical grid. The Property presently is supplied electrical power by one 65-kilowatt (kW) diesel generator and two smaller 5.5 kW diesel generators. The main power grid for Durango follows a paved federal highway and a power connection. There is currently an application in to CFE to connect a trunk line to the existing 230kv power line located adjacent to the Ruta 40, about 12 km from the site.

The town of Francisco I. Madero has a Pemex gas station and the services of metal fabricators and mechanic shops. A railway line is present near the south boundary of the Property and the railway has a direct line to Torreón, the site of the nearest metal smelter. The smelter is owned by Met-Mex Peñoles S.A. de C.V. and is a primary producer of lead and zinc. The facility also has a silver refinery, which has the capacity to refine 120,000 kilos of silver per month.

Presently the Property has six core storage sheds, an office, lunch room, washrooms, small warehouse, flammable substances storage area, drilling company workshop, night watchman's accommodation, and a generator/core cutting shed.

The water for drilling and services is obtained from a water reservoir in Francisco Javier Mina, charged at a rate of $500 Mexican Pesos per 1.75 cubic meters (m^3), including the cost to haul water to the Project by tanker trucks to water tanks located adjacent to the drilling areas. Water for mining production is proposed to be supplied from aquifers to the south and east of the Project. Two large diameter water wells have been drilled on property 9.7 km to the south of the Project. Permits for these wells and the water rights have been submitted to CONAGUA and are being processed and permitted. Water reuse will be maximized to minimize the water cost to the Project.

5.4.4 Local Resources and Mining Personnel

There is expected to be sufficient qualified local work force available in Durango and the surrounding region for Project construction and operation. Durango is approximately 45 minutes away from the Project site. An inventory of the local education levels, employment levels and skill sets of the two nearest population centers, Ricardo Flores Magón and Francisco Javier Mina, have been conducted. Current employment levels and the skills and abilities found in the local communities are shown in Table 5-1 and Table 5-2, respectively. Additional detail on distance of various communities to the site, population, and the number of potentially active workers is found in Table 5-3.

Table 5-1: Current Employment in Nearby Population Centers

	Ricardo Flores Magón	Francisco Javier Mina
Unemployment	62.75%	44.73%
Field	27.58%	25.43%
Mine	0.69%	12.28%
Masonry	6.20%	4.38%
Manufacturing Plant	2.06%	5.26%
Gas Station	0.69%	0%
Public Work	0%	6.14%

Table 5-2: Skills and Abilities in Nearby Population Centers

	Ricardo Flores Magón	Francisco Javier Mina
Agriculture Tractor	51.54%	41.24%
Truck	48.45%	52.54%
Masonry	32.47%	22.03%
Office	14.94%	20.33%
Forklift	14.43%	5.08%
Motorcycle	12.37%	0%
Truck or Water Truck	10.30%	10.16%
Welder	4.63%	8.47%
Bobcat	3.60%	0%
Backhoe Loader	3.09%	3.96%
Drilling	0%	10.16%
Geological Technician	0%	6.21%
Bulldozer	0%	3.38%

Table 5-3: Project Work Force Availability

Federal Code	State	Municipality Code	Municipality	Local Code	Locality	Total Population	Active Population	Distance to Project (km)
10	Durango	1	Canatlán	1	Canatlán	11,495	4,285	43
10	Durango	1	Canatlán	110	Ricardo Flores Magón	1,467	479	13
10	Durango	5	Durango	1	Victoria de Durango	518,709	204,350	84
10	Durango	5	Durango	295	Vicente Suárez	92	31	48
10	Durango	20	Pánuco de Coronado	1	Francisco I. Madero	4,550	1,601	32
10	Durango	20	Pánuco de Coronado	8	Francisco Javier Mina (Corralejo)	919	201	9
10	Durango	20	Pánuco de Coronado	14	General Lázaro Cárdenas	389	121	27
10	Durango	20	Pánuco de Coronado	9	Francisco Rueda Serrano	541	160	23

5.5 TOPOGRAPHY, ELEVATION, AND VEGETATION

The Property lies on the western edge of the high plains of northern México, an extensive volcanic plateau characterized by narrow, northwest-trending ranges separated by wide, flat-floored filled basins. In the Durango area, the basins have elevations of between 1,900 m to 2,100 m above sea level and the higher peaks rise to 3,000 m. The Property elevation in the area of the mineralized zones at the Property is between 1,990 m and 2,265 m. The highest

elevations on the Property are at the northwest trending La Preciosa Ridge, which overlies the La Gloria and Abundancia veins. A broad valley forms to the east of the ridge and extends approximately 1 km toward another lower lying ridge to the northeast. Grasses, small shrubs, and cactuses comprise the typical vegetation on the steep hillsides with larger bushes and mesquite trees in the lower lying areas near springs and streams. Nearby farmers produce beans and maize with groundwater sourced from thick gravel beds in the surrounding plains or via dry farming during the rainy season. Local cattle graze on land dominated by rocky soils supporting nopal (prickly pear) and huizache (acacia) scrubland.

6 HISTORY

In the late 19th century, the Property was known as Mina La Preciosa. Early work was focused on the north end of La Preciosa Ridge where the Gloria and Abundancia veins outcropped to surface. Mining ceased at the onset of the Mexican Revolution, in 1910, and further mining did not occur until the 1970s. It has been estimated by Orko personnel that a total of ~30,000 tonnes was extracted during that time (Mining Plus, 2012).

Luismin (Compania Mineras Minas San Luis) operated under the name Minera Thesalia as a joint venture with Tormex S.A. and conducted exploration on the property in 1981, 1982, 1988 and 1994. This work consisted of a surface and underground channel sampling program, a single east-west line of induced polarization (IP) resistivity across the property, and drilled seven diamond drill core holes totaling 1319 meters. This included five surface drillholes targeting the Gloria and Abundancia veins 50-75 meters below the primary underground workings, and two holes drilled from within those older workings.

Luismin expanded the historic underground workings to a size of approximately 3 meters by 3 meters. This allowed for the underground drilling and also for a program of channel sampling. A reported 11,739 tonnes of material was removed from the sides of the historic underground workings at reported grades of 0.43 g/t Au and 157 g/t Ag. That material was stockpiled outside the portal of those underground workings and is still in-place. While Luismin staff did calculate several Mineral Resource estimates during that time, based on limited information, the channel sampling and shallow drilling were not used for the calculation of the current Mineral Resources.

Orko Gold Corp. (subsequently Orko Silver Corp.) entered into a joint venture (JV) agreement with Luismin in 2003 and subsequently acquired the control of the property with Luismin maintaining a royalty.

Orko performed a series of exploration programs beginning in 2005 and lasting until 2008, and drilled 388 core holes for a total of 152,368 meters on targets at Orito, San Juan and La Preciosa. Additional surface sampling and mapping was also performed during that time.

Orko signed a JV agreement with Pan-American Silver (PAS) in 2009. PAS drilled 363 drillholes for a total of 91,096 meters during 2009-2010. The desired result was a Measured Resource to support a feasibility study issued in 2012 by Quantitative Geoscience Pty. Ltd. and included a Technical Report by Snowden done in 2011 (Snowden, 2011a). PAS work included the use of some drillholes for geotechnical purposes, and four metallurgical test programs performed by SGS Mineral Services in Durango, Mexico. Problems with those metallurgical test programs were noted by Snowden and future work was recommended. However, a table of Mineral Resources was produced as a result of the Mining Plus Updated Mineral Resource Estimate prepared for Orko in 2012 (Mining Plus, 2012). A QP has not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves; and (ii) Coeur is not treating the historical estimate as current mineral resources or mineral reserves. The resources are presented for the disclosure for historical purposes only.

Table 6-1: La Preciosa Historical Mineral Resource Estimate – Effective October 25, 2012

La Preciosa Mineral Resources Estimate – effective October 25, 2012*									
Mining Method	Classification	Cut-off grade	Tonnes (millions)	Silver (g/t)	Silver (million ounces)	Gold (g/t)	Gold (000's ounces)	Silver equivalent (g/t)	Silver equivalent
Open Pit	Indicated	25	29.6	104	99	0.20	190	115	110
Open Pit	Inferred	25	47.7	86	132	0.16	245	95	146
Underground	Indicated	60	0.1	99	0	0.16	0	108	0.2
Underground	Inferred	60	1.9	124	8	0.21	13	136	8
Total	Indicated		29.7	104	99	0.20	191	115	110
	Inferred		49.6	87	140	0.16	259	97	154

Notes:

1. Open pit resources stated are contained within a potentially economically mineable pit shell. Pit optimization is based on assumed silver and gold prices of US$25.90/oz and US$1,465/oz respectively and mill recoveries of 88% and 78% respectively, mining costs of US$1.45/t, processing costs of US$17.25/t and G&A costs of US$4.35/t. Break-even cut-off grades used were 25 g/t Ag for open pit mill material and 60 g/t Ag for underground material.
2. Silver equivalency is based on unit values calculated from the above metal prices, and assumes 100% recovery of all metals.
3. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and sums may not total due to rounding
4. The Underground Indicated and Inferred Resources do not account for mining method and the probability that mine plan design could significantly change the these Resources.

In April 2013, the acquisition of Orko was completed by Coeur. Since completion of the acquisition, activities have included land and water resources acquisition plus additional efforts on geological and technical studies. All involved property owners were identified and their titles verified to be in good standing prior to acquisition of surface rights.

6.1 WATER

Two wells were drilled by Coeur in an aquifer located about 10 km due south of the proposed plant site. The wells were drilled on advice from CONAGUA, the water resource monitoring agency, and were drilled and completed with guidance from a respected hydrologist. This resulted in two wells with demonstrated large capacity, anticipated to meet or exceed the required 80 L/s needed for plant and mine operations.

In addition, four existing small diameter wells, previously drilled to provide water for grazing, were logged, cased and had pumps installed in them. These wells were located in the vicinity of the proposed TSF and are anticipated to be plugged and abandoned prior to construction of the tailing dam. These wells were not suitable for plant operational needs due to low capacity; however, a water right application was submitted for those wells at the same time as the application was submitted for the two larger wells. The water rights for those four wells have been granted and are being registered in the Regional and National offices of CONAGUA. Those water rights can be transferred within the aquifer and used for other uses such as monitor wells, dewatering and other uses.

6.2 GEOLOGIC MODELING PROGRAMS

In addition to the new data generated by the drilling programs, differences were noted in prior ore-body model data quality. As a result, all prior core logs were reviewed and interpreted via a common core logging system and re-entered into the model. As a result, finer details that had been previously logged but not entered into the model were able to be captured for future interpretation. This has resulted in common naming conventions, notations on mineralization type and oxidation states, alteration codes and host rock naming conventions. This greatly improved the quality of the ore-body model.

Over 1,400 new mineral density measurements were taken across a broad spectrum of both mineralized and non-mineralized sections of the model. This included both new and older core measurements and greatly improved the density data within the model.

A review of the prior core also resulted in the discovery that not all mineralized sections of the core had been reviewed by Orko. A re-sampling effort was undertaken to identify additional mineralized zones and also to sample zones between and adjacent to known veins to define if there was mineralization in those rock masses. This included 252 new assays of existing core. This resulted in defining some additional mineralized material that can be mined via bulk mining methods as well as providing guidance on the actual grade of material that will be assumed as dilution grade when the veins are mined.

During a review of the prior Orko data, it was also discovered that they had used a less expensive laboratory procedure of a 3-acid digestion rather than the ISO standard 4-acid digestion for AA (atomic adsorption) chemical analysis. Pulps for these samples were removed from storage and submitted for re-analysis. This included approximately 6,400 pulps located at the site and some ~800 located at a remote storage location in Canada. These were re-analyzed, with new standards and blanks, and compared to the prior data set. This resulted in a ~15% increase in grade that would be applied to a portion of the deposit, for a likely 5% increase in anticipated plant feed grade.

New structural models were defined for the deposit and regional areas. Clay alteration testing was performed to define age/temperature/chemical relationships within the veins so that future testing could be compared to those results.

6.3 METALLURGICAL TESTING

Metallurgical testing performed by Orko showed recoveries that had considerable variation. A review of that data showed that there was no testing performed to optimize recovery and that the testing was done with a variety of conditions. New programs of metallurgical testing were performed to define the optimum conditions for the testing and then re-perform that testing within those defined optimums.

This includes flotation and leaching testing at optimized pH, reagent addition levels and retention times to provide circuit design and operating characteristics.

Flotation testing did not prove to be a viable alternative due to low quality concentrate grades and low recoveries. Whole ore leaching, at a grind of 75 microns, was found to provide optimized recovery in agitated leaching applications. This metallurgical process is designed to feed a Merrill-Crowe plant to produce doré at site.

Additional testing was initiated to define leach recovery potential for heap leaching of low-grade ores. This testing was initiated with bottle roll tests based on previously-defined pH and reagent levels.

6.4 ENVIRONMENTAL STUDIES

The Project was divided into two study areas for environmental applications: one for the mine area and one for the access, power and waterlines. All plant and animal studies and clearances, surface and groundwater baseline data, drainage basin studies and storm water drainage volumes and flows were defined. Local and site studies have been completed for groundwater characterization (water quality, water level, pit-inflow rates), surface water quality, and geochemical characterization of mining wastes (waste rock and tailings). Monitoring and management plans have been developed for groundwater monitoring, waste rock, tailings, prevention and control of potential petroleum and chemical spills, sediment control plans, and tailing designs were completed based on those studies. A mine closure and reclamation plan and closure cost estimate has been prepared.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The Project is situated on the eastern flank of the Cretaceous to mid-Tertiary Sierra Madre Occidental (Figure 7-1). The SMO is the largest silicic igneous province in North America and it stretches from the USA-Mexico border to the latitude of Guadalajara, where the SMO is covered by the late Miocene to Quaternary Trans-Mexican Volcanic Belt.

The SMO is part of the Basin and Range physiographic province where magmatism and tectonism were related to the subduction of the Farallon Plate beneath North America. Physiographically, the core of the SMO forms the boundary between the Mexican Basin and Range Province to the east and the Gulf Extensional Province to the west.

Figure 7-1 shows a simplified geological map of Northern Mexico showing the main assemblages of the Sierra Madre Occidental (from Ferrari et al., 2007). The Lower Volcanic Complex is shown in blue and the Upper Volcanic Supergroup is shown in pink and orange.

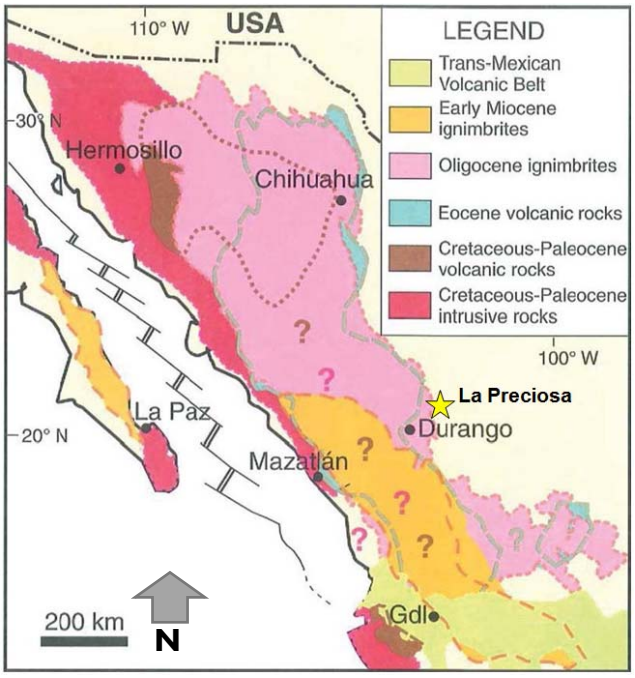


Figure 7-1: Simplified Geological Map of Northern Mexico (from Ferrari et al., 2007)

The stratigraphy of the SMO comprises the following main sequences:

- Late Cretaceous to Paleocene plutonic rocks;
- Paleocene-Eocene (ca. 67-55 Ma) andesites and lesser rhyolites, traditionally grouped into the Lower Volcanic Complex (LVC; McDowell and Keizer, 1977);
- Silicic ignimbrites mainly deposited during two pulses; e.g., Oligocene (ca. 32-28 Ma) and Early Miocene (ca. 24-20 Ma), and grouped into the Upper Volcanic Supergroup (UVS; McDowell and Keizer, 1977);
- Transitional basaltic-andesitic lavas that erupted toward the end of, and after, each ignimbrite pulse; and
- Post-subduction volcanism consisting of alkaline basalts and ignimbrites deposited in the Late Miocene, Pliocene, and Pleistocene.

In the area, deformed metasedimentary rocks of Cretaceous age are exposed in small windows through the Tertiary volcanic rocks of the SMO. These consist of folded and foliated clastic metasedimentary rocks that are unconformably overlain by undeformed Early Tertiary conglomerate and sandstone of the Ahuichila Formation (Aguirre-Diaz and McDowell, 1993), which are in turn overlain by a sequence of intermediate tuffs, flows and agglomerate of the Paleocene-Eocene age LVC. The LVC sequence is overlain by a thick sequence of rhyolite and intermediate to felsic ignimbrite, tuff, and volcanic breccia of Oligocene-age that are exposed along cliffs to the west of the Project.

The region is transected by the regional northwest-striking San Luis-Tepehuanes fault system (Nieto-Samaniego et al., 1999), which roughly coincides with the eastern margin of the SMO. This fault system comprises a complex network of northwest- to north-striking, west-dipping fault segments that are associated with east to northeast tilting of Tertiary stratigraphy. In the Durango region, the fault system is made up of north-northwest trending normal faults and associated (half) grabens that were active during two stages of extension between ca. 32 and 24 Ma (Nieto-Samaniego et al., 1999). The basins and parts of the lower hills in the region are covered with varying thicknesses of Pliocene to Pleistocene basalt that erupted from numerous vents now marked by small volcanic cinder cones and domes.

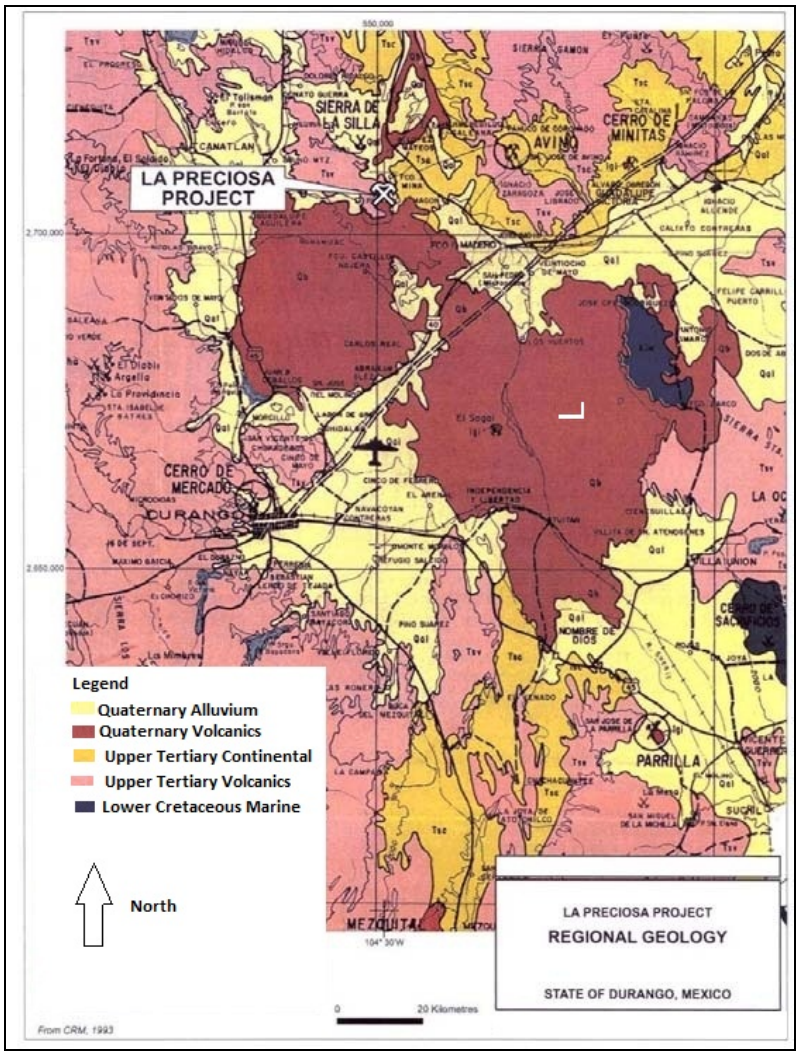


Figure 7-2: Regional Geology Map

7.2 REGIONAL MINERAL DEPOSITS

Mineral deposits of the Sierra Madre Occidental plateau and the La Preciosa District consist mainly of silver and gold mineralization with or without significant base metal components. The known deposits form a northwest-trending belt from the state of Zacatecas and the large Fresnillo silver deposit on the south east to the Guanaceví silver deposit near the border with the state of Chihuahua.

Adjacent to the Project, approximately 18 km to the northeast, is the historic Avino property, and in particular the Avino and San Gonzalo veins and stockwork system, which is considered a low- to intermediate-sulphidation Ag-Au-copper (Cu) epithermal deposit similar to other deposits in the Mexican silver belt.

Approximately 50 km north of the Project, is the El Castillo Mine (Argonaut Gold is the operator), which is thought to be a porphyry-style gold system related to Oligocene granodiorite-diorite porphyries that intrude Cretaceous clastic and carbonate sediments in an extensional tectonic setting. Gold mineralization occurs throughout the magmatic-hydrothermal system.

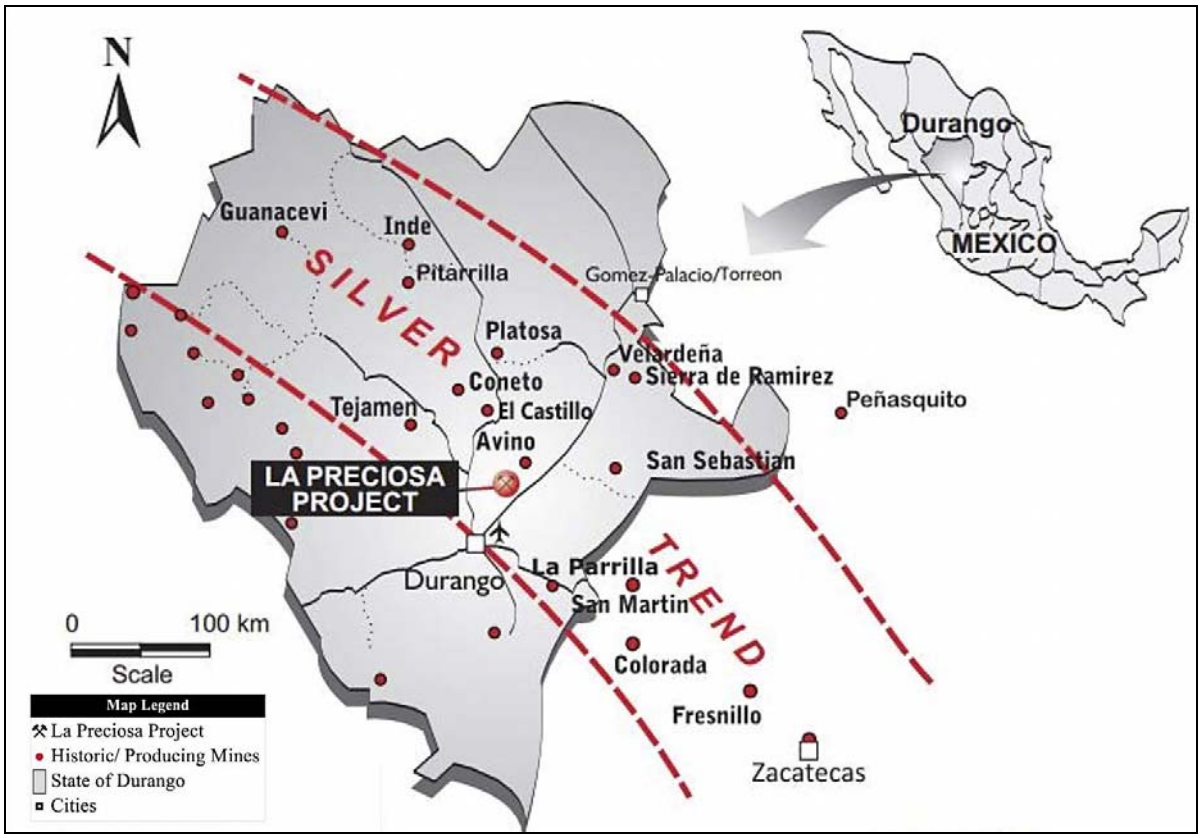


Figure 7-3: Mineral Deposits in the Project Area (from M3, 2013)

The nearest large mine is the Mina Avino of Avino Silver & Gold Mines Ltd., 20 km to the northeast of the Project, near the town of Pánuco de Coronado. The current mine operated from 1974 to 2001. Mill throughput reached 1,000 tonnes per day (tpd) in the 1990s, but mining was suspended in 2001 due to low metal prices. The Avino veins were reported to contain approximately 140 g/t Ag, 1.4 g/t Au, and 0.5 % Cu. A drilling program is currently underway at Avino.

San Sebastian, located 60 km to the east of the Project, contains a number of productive vein systems including Francine, Don Sergio and Andrea. Production by Hecla from the Francine vein was high-grade silver, with significant

gold values. Mineralization occurs in poly-phase chalcedonic quartz veins with an average width of 1.6 m. Production from the Don Sergio vein was high-grade gold, with some silver values. Several epithermal veins exist within the San Sebastian valley. The Francine, Professor, Middle and North vein systems are hosted within a series of shale units, with interbedded fine-grained sandstones. The Don Sergio, Jessica, Andrea, and Antonella veins located in the Cerro Pedernalito area, about 6 km from the Francine vein, are hosted in the same formation, with the addition of diorite intrusion. Mining ceased in 2005, however, Hecla is continuing with an active exploration program in the area, in particular on the Hugh Zone.

Directly adjacent to the Project on the west is the San Juan project of Silver Standard. Orko conducted prospecting, geological mapping and some surface sampling. Vein targets, La Plomosa, La Plomosa Sur, El Vaquero, San Juan, Nancy Sur and the down-dip projection of the Nancy ein, are known on the San Juan property. La Plomosa vein has approximately 80 meters of historical drifting and one drillhole.

Immediately south of La Plomosa and San Juan are the large Victoria and Salamandra concessions of Canasil Resources Inc. under joint venture with Blackcomb Minerals Inc. Salamandra is a skarn silver-zinc-copper prospect.

La Parrilla mine of First Majestic Silver Corporation, is located near the Durango – Zacatecas border, approximately 65 km southeast of the city of Durango and 80 km south of the Project. La Parrilla is currently in production at a rate of 800 tpd. First Majestic is focusing on the La Rosa/Los Rosarios, San Marcos, San José, San Nicolás, Vacas, Quebradilla, La Luz and Recuerdo structures. The silver-lead-zinc mineralization is hosted in vein-fault zones, breccias and replacement bodies. These occur within the porphyritic diorite intrusive rocks and in the adjacent limestone, skarn, and hornfels rocks. While the geology is different than that at the Project, it does illustrate another example of precious metal mineral endowment in the region.

There are numerous precious metal exploration and expansion projects underway in Durango State and adjacent areas, including Metates, La Cienega, La Parrilla, Pitarrilla, Guanacevi, San Agustin, Peñasquito, Santa Cruz, San Sebastian and Topia, as well as an expansion at the Tayoltita (San Dimas) operations. Neighboring Zacatecas state is also very active.

7.3 LOCAL GEOLOGY AND MINERAL DEPOSITS

7.3.1 Local Geology

The oldest rocks on the property are Jurassic-Cretaceous metasedimentary graphitic schist, chlorite schist, and layers of quartzite (Figure 7-4 and Figure 7-5). These metasedimentary rocks do not outcrop at surface but are intersected in drill core. Overlying the metasedimentary sequence is a thick package of unmetamorphosed polylithic conglomerate containing lenses of arkosic sandstone of unknown age.

The sedimentary package is overlain by intermediate tuff and agglomerate of the regional Tertiary age Lower Volcanic Complex. In places, the flows are porphyritic, or glomeroporphyritic and the tuffs are partly welded. The youngest rocks within the property are basalt flows that erupted from several Pleistocene-age volcanic vents and which now fill the lower valleys. Cerro Prieto, Cerro Blanco, and Cerro La Chicharronera are prominent examples of the volcanic vents. Other nearby (9 km west) volcanic vents is the Holocene age, La Breña-El Jagüey maar complex, which is part of the Durango Volcanic Field. Sporadic mafic to felsic dikes and sills of unknown age are found in the deeper parts of the area and rarely at surface.

The area contains a series of Tertiary-age silver-bearing (±gold) epithermal quartz veins associated with barite, fluorite, and sporadic base metals, primarily zinc and lead. There are two major vein and vein-breccia systems exposed on a series of hills and ridges, which are separated by a flat-floored valley roughly 800 m wide. The conglomerate and Tertiary Lower Volcanic andesitic rocks are the main host rocks for quartz veins, although vein mineralization does extend into the basement metasedimentary rocks.

The main veins system on the Abundancia Ridge consists of dominantly south-striking and west-dipping veins plus east-southeast-striking, south dipping crosscutting veins. For example, the Abundancia Ridge vein system has been traced on surface for more than 1.5 km, and drilling has revealed that the veins continue to the north, beneath basalt cover.

Along the eastern side of the Project, a series of hills expose a north- to northwest-striking, shallow west-dipping vein system with associated hanging wall veining and alteration. This vein system is referred to as the Martha vein or fault zone and has been traced by drilling for over 2.5 km along strike.

Mineralization at the Project is hosted within multiple discrete poly-phase quartz veins, often displaying banded, smoky, drusy, and chalcedony textures. Also in each stage there is variably crustiform banded fracture fill/breccia cement mineralogy. Fluorite, amethyst, a substantial number of barite laths, calcite, and rhodochrosite may also be present, and sulfide mineralization in the form of sphalerite, galena, pyrite, chalcopyrite, acanthite, sparse native silver and free gold, as well as iron and manganese oxides have been noted in drill core. The principal silver bearing mineral at the Project is acanthite-pseudomorphic after argentite or as microcrystalline to amorphous grains.

Vein mineralization does extend into the basement metasedimentary rocks, but its extent and distribution is not well understood. The main vein system on the Abundancia ridge consists of dominantly southward-striking and westward-dipping veins plus east-southeast–striking, south-dipping crosscutting veins. The Abundancia ridge vein system has been traced on surface for over 1.5 km. Along the eastern part of the Project, a series of hillocks expose a north- to northwest-striking, shallow west-dipping vein system with associated hanging wall veining and alteration. This vein system is referred to as the Martha vein or fault zone and has been traced by drilling for over 2.5 km along strike.

Examination of mineralized samples identified mainly argentite, tennantite/tetrahedrite, and Ag sulphosalts in samples. The majority of gold/electrum is inter-grown with or occupying the same paragenetic position as argentite, silver sulphosalts, sphalerite and galena, mostly transitional between quartz and carbonate/iron carbonate in formation.

Wall rocks hosting mineralization are variably silicified, with proximal patchy illite-smectite alteration and distal chlorite alteration. The presence of manganocalcite has been noted in several drillholes, but it is not uniformly distributed. In shallower drillholes, pyrolusite and limonite often appear on fracture surfaces.

The host rocks and veins have undergone intense weathering. The base of oxidation is erratically distributed as weathering is controlled by the presence of post mineralization faults which allowed the percolation of oxidized meteoric groundwater to vertical depths of 350 m below surface. Weathering minerals include iron oxides, iron carbonates, manganese oxides, and unidentified clays.

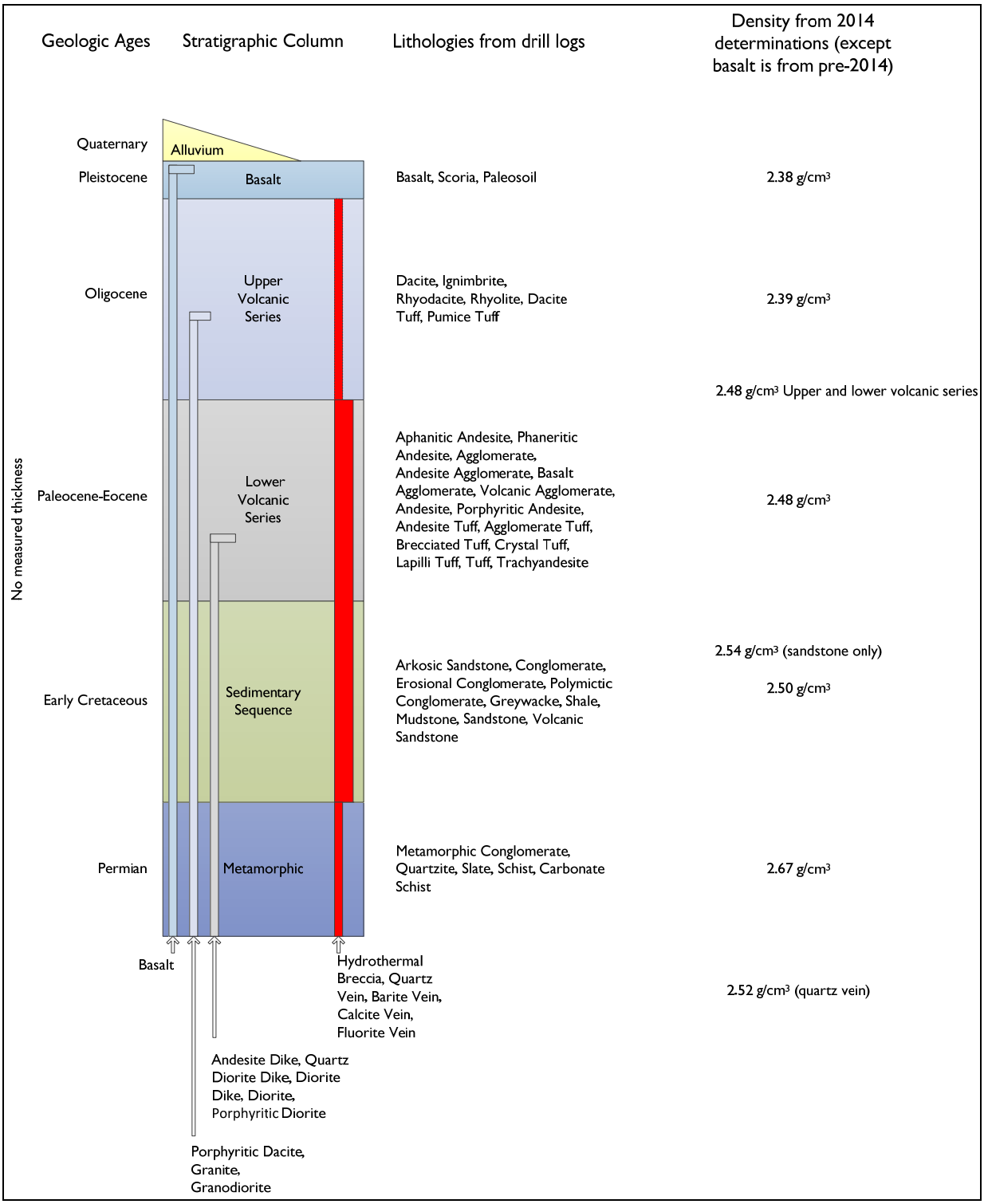


Figure 7-4: Project Local Stratigraphic Column

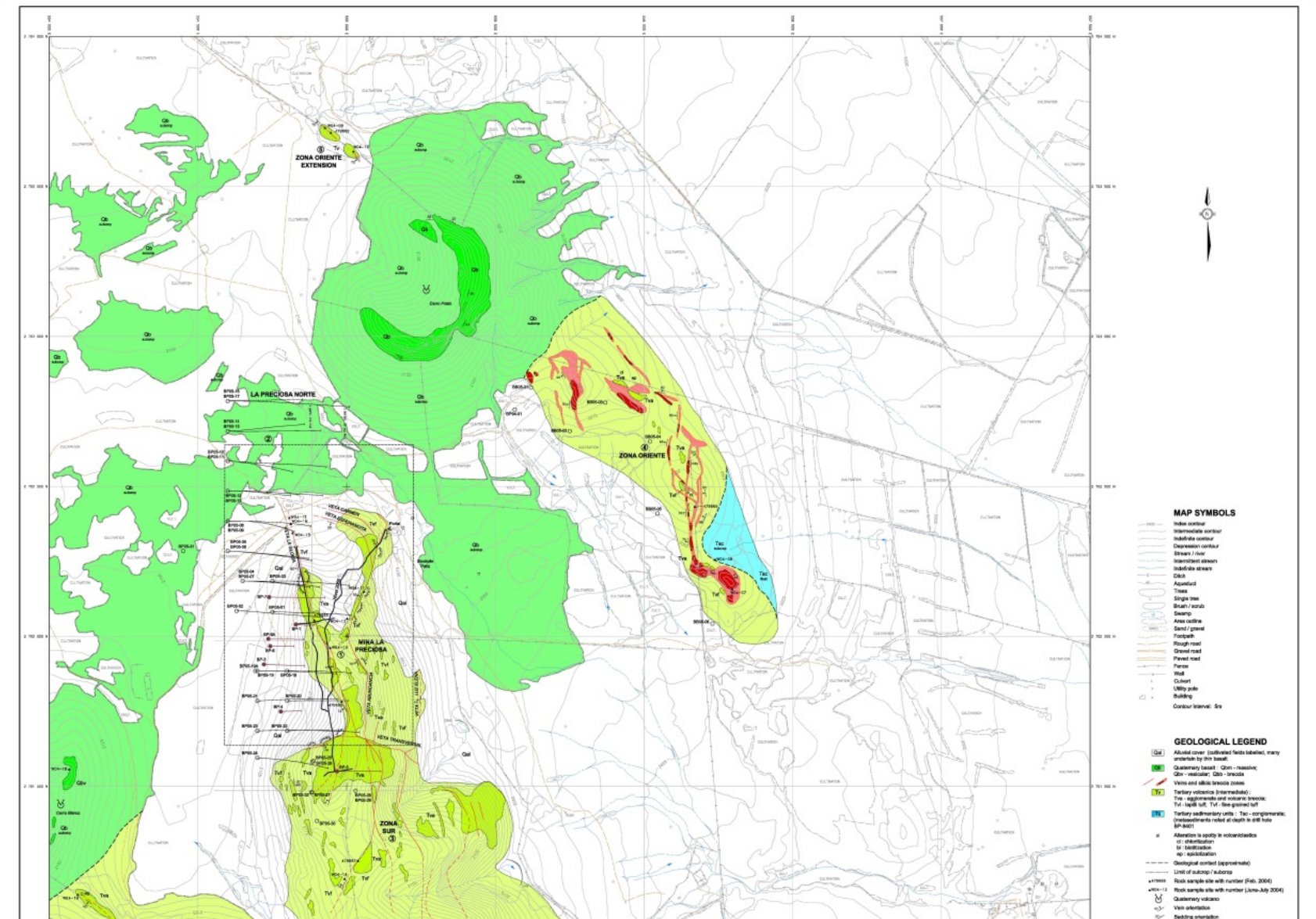


Figure 7-5: Local Geologic Map (Orko, 2006)

7.3.2 Lithological Units

The main rock types occurring in the Project area include volcanic flows, pyroclastic rocks, sedimentary rocks (clastic and epiclastic rocks), metamorphic rocks, and subvolcanic dikes. The lithological descriptions below include the alteration products that were most commonly observed.

Volcanic Flows: Distinctive volcanic flows include vesicular basalt, basaltic andesite, and latite to andesite. The lithology logs also include rhyolite. Rhyolite flows were not observed during the alteration study. Quartz-phaneritic felsic crystal-lithic tuff is abundant in the northeastern part of the property, and it is possible that rhyolite flows occur within that unit.

Vesicular basalt: Vesicular basalt is the youngest lithology in the property. It forms sub-horizontal flows that cover the tilted volcanic, pyroclastic, and volcaniclastic sequence beneath. The vesicular basalt unit is black, fine-grained, and has abundant vesicles. It commonly overlays a paleosol horizon that varies in thickness from tens of centimeters to meters. The vesicular basalt unit is not affected by visible hydrothermal alteration.

Basaltic andesite: Basaltic andesite is dark purple-brown, sparsely porphyritic, with phenocrysts that average less than 1 mm in length. Phenocrysts and to a lesser degree groundmass are commonly altered to carbonate.

Latite to andesite: Latite to andesite is beige colored, porphyritic, with few feldspar phenocrysts and few mafic phenocrysts. Feldspar phenocrysts are variably altered to clay, mafic phenocrysts are variably altered to chlorite, clay, pyrite, or hematite after pyrite, and groundmass is variably altered to clay.

Pyroclastic Rocks: Distinctive volcanic pyroclastic lithologies include lapilli-tuff breccia, crystal-lapilli tuff, felsic crystal-lapilli tuff-breccia, and lesser mafic lapilli tuff and mafic crystal tuff.

Lapilli-tuff-breccia: Lapilli-tuff-breccia is the most widespread pyroclastic lithology observed in the deposit. It has a dark green to brown green magmatic, porphyritic matrix enclosing angular lithic clasts and juvenile clasts of various sizes. Minor agglomerate and clastic horizons occur within the lapilli-tuff-breccia. Lapilli-tuff-breccia is consistently classified as agglomerate in lithology logs. However, according to the pyroclastic rock classification, the term agglomerate should only be applied for pyroclastic rocks that contain over 75% pyroclasts greater than 64 mm, whereas the lapilli-tuff-breccia commonly contains over 40% magmatic matrix. Intervals of autobreccia were also observed locally.

Significant differences in composition between magmatic matrix and lithic clasts makes this unit favorable for patchy alteration facies. Smectite clay is a common secondary mineral in the lapilli-tuff-breccia, and may be a product of devitrification of the magmatic matrix and of juvenile clasts, or of hydrothermal activity.

Crystal-lapilli-tuff: The lapilli-tuff-breccia (described above) grades into intervals of crystal-lapilli-tuff and lapilli-tuff that is up to several meters thick. Flow banding is observed locally in the form of aligned lapilli. Feldspar crystals are variably altered to clay, which are most commonly kaolinite or illite.

Felsic crystal-lapilli-tuff-breccia: Felsic crystal-lapilli-tuff-breccia has abundant quartz crystals, subangular to angular porphyritic lithic lapilli, and numerous fiamme enclosed in a light reddish-brown to white or light green-white magmatic matrix. This unit occurs in the northeast portion of the Project area, and includes intervals of felsic crystal±lapilli-tuff up to several meters wide that locally contain that contains feldspar crystals. Feldspar crystals and lithic clasts are variably altered to clays.

Sedimentary Rocks: Sedimentary rocks include conglomerate, sandstone, and minor siltstone and shale.

Sedimentary Breccia and Conglomerate: Greenish-grey to brown, matrix to clast supported, moderately to poorly, and locally well sorted polylithic sedimentary breccia to conglomerate is the most widespread sedimentary lithology. Weak carbonate and hematite cement are common. Disseminated, fine-grained euhedral pyrite is commonly observed within the cement. Locally the matrix is completely replaced by epidote.

Sandstone: Sandstone forms lenses that are typically less than 6 m thick, and occur most commonly associated with the conglomerate. Irregular, thicker sandstone horizons up to 20 m thick also occur in the upper parts of the volcanic stratigraphy in the northwestern part of the property. Locally, sandstone grades into siltstone and shale beds that are generally less than 2 m thick.

Metasedimentary Rocks: Metasedimentary rocks include muscovite-schist and quartz-muscovite-schist that form the basement to the overlying sedimentary and volcanic sequence. The upper contact of the metasedimentary rocks is commonly veined, intruded by subvolcanic dikes and sills, and faulted. The schist contains two well-developed foliations, one of which is folded by the other.

Subvolcanic Intrusions: The entire volcanic, sedimentary and metasedimentary sequence is intruded by dikes and sills of felsic to intermediate composition. The subvolcanic intrusions are most abundant along the contacts between basement metasedimentary rocks and the overlying conglomerate. Dike and sill contacts are typically strongly clay altered.

Light to dark colored feldspar porphyry dikes and sills are sparsely porphyritic, but locally have few to moderate feldspar phenocrysts and few mafic phenocrysts that are completely converted to pyrite and clays. Feldspar phenocrysts are variably clay altered. Groundmass is commonly moderately sericitized.

7.3.3 Alteration

The principal visible alteration facies observed in the Property consist of:

- Patchy albite-epidote±chlorite flanks the deposit to the west, north, and southeast, and produces strengthening of the rock. Chlorite, brucite, and epidote are the most common minerals present in this facies.
- Silica-sericite-pyrite occurs along northwest and east-northeast trending corridors, and does not appear to be intense enough to affect rock strength. Illite and muscovite are the most common minerals in this facies.
- Pseudomorphic clays and carbonate after phenocrysts occurs throughout all porphyritic lithologies and does not define specific trends or affect rock strength. Illite, phengite, and montmorillonite, iron-carbonates, and lesser chlorite are the most common in this facies.
- Pervasive texture destructive silica defining northwest-trends are often defined by zones of breccia.
- Fault-fill clays are restricted to post-mineral faults. Montmorillonite is the most common mineral in this facies.

The most frequently occurring are (in order of decreasing frequency): montmorillonite, illite, phengite, iron carbonate, silica, chlorite, brucite, muscovite, kaolinite, calcite, and epidote. Among the alteration minerals, muscovite is the mineral that appears to have best spatial correlation with faults.

The Deposit lacks a distinct halo of a high illite crystallinity surrounding mineralization. This lack of an alteration halo is interpreted as being due to a combination of the strong lithological control over illite crystallinity, and to the scale of this alteration study, which was conducted along the main mineralized zones. It is possible that a broad zone of illite crystallinity high would be defined at a more regional scale.

7.4 MINERALIZATION

The area has been cut by numerous structures, both NW and NE oriented (to NNW and NNE), as well as ~E-W; post-Oligocene extension resulting in graben-style faults, possibly with low-angle listric-type movement. Subsequent

mineralization occurred along these low and high-angle faults, and also followed the low-angle contact of the basement or conglomerate with the tuff. The Martha vein, with a dip of ~20° to the SW, defines the unconformity at depth. The shallowly dipping Abundancia vein dips ~50° to the WNW, and the high-angle La Gloria vein, in the west, dips ~75° to the WSW. Internal to this main system of veins, are also areas of veinlets and stockwork, which constitute most of the mineralization.

Mineralization is controlled by three types of structures:

- Type 1: structures commonly associated with faults and exhibit crustiform, cockade, and colloform textures that are representative of multiple vein opening stages. These veins generally have widths of greater than 30 centimeters and can form vein systems up to several meters wide. Cavities are also common in these veins. Quartz stockwork comprising millimeter- to centimeter-scale quartz veinlets is common both in the hanging wall and footwall of Type 1 vein systems.
- Type 2: structures consist of veins that range in width from 1 centimeter to several tens of centimeters, and rarely include veins up to 6 meters wide (e.g., Abundancia and La Gloria veins). Type 2 veins are dominated by colloform textures with sugary quartz and euhedral crystals projecting into cavities along the vein centers. Dilation (or jigsaw) breccia veins are also common, with angular clasts of wall rock (typically fine-grained volcaniclastic rock) in quartz and (or) calcite cement. Colloform textures and crystal growth into cavities are characteristics of open-space filling which commonly occurs in extensional settings.
- Type 3: structures are commonly associated with abundant hematite alteration of the host rock, breccia, minor stockwork development, and patchy or narrow quartz vein development. Type 3 structures are typically fault zones up to several meters wide with variably developed quartz-carbonate-calcite veins and fault breccia.

As previously mentioned, the mineralization in the area occurs in veins, veinlets, and stockwork. These veins average in true width under 15 m (Martha Vein) consist of several stages of banded, crustiform to colloform, quartz (and cryptocrystalline quartz at shallow depths), adularia, barite and typically later carbonates (both calcite and rhodochrosite); illite commonly replaces the adularia. There are variable amounts of pyrite, sphalerite and galena plus argentite, and variable amounts of tetrahedrite - tennantite, freibergite and Ag sulfosalt.

7.4.1 Local Mineralization

The district has many characteristics that are typical of epithermal veins in Mexico, particularly of the Ag-rich variety. Quartz veins are accompanied by adularia, barite, calcite, rhodochrosite of variable timing, as well as acanthite, freibergite, Ag sulfosalts and minor electrum, plus variable amounts of pyrite, honey-colored sphalerite, tennantite/tetrahedrite, chalcopyrite and galena, and supergene Fe and Mn oxides; the hypogene minerals are characteristic of intermediate-sulphidation deposits in Mexico. Mineralization is believed to be Tertiary in age both the LVS and UVS are mineralized, but the basalts are recent and not mineralized.

Petrographic studies of the veins in Deposit, find that multiple stages of silver and base metal mineralization are associated with repeated fluid boiling and mixing events, defined by crustiform banded fill/cement assemblages within a framework of intermittent and more significant fracturing/rupturing of wall rock and pre-existing vein/cement assemblages. There is a repetition of common hydrothermal fill/cement mineralogy, including ore minerals, such that correlation of vein/cement assemblages/events between drillhole intersections would be difficult.

The occurrence of adularia and style of early quartz and chalcedonic quartz replacement amongst wall rock replacement and fracture-fill/cement assemblages confirms silver and base metal mineralization associated with low sulphidation, epithermal style systems developed on the Martha and Olin structures at the Project. Significant widths of mineralized quartz and carbonate dominated fracture-fill and breccia cement assemblages have developed as a result of extended episodes of hydrothermal fluid flow and repeated rupturing of wall rock and pre-existing vein/cement

assemblages. Internal crustiform banding within the different voluminous fill/cement assemblages represents incremental opening and filling of fractures/cavities between major rupturing events.

The Martha vein is the largest vein in the deposit by far, with at least 3 times the volume of the next largest vein, La Abundancia. Both veins are low angle, the Martha vein dips ~20-30°, following the SW-dipping contact of volcaniclastic rocks overlying an immature conglomeratic unit (consisting mainly of polylithic clast-supported fragmental rock with angular to sub-rounded clasts), or the underlying schist.

There are also high-angle veins in the west on the ridge, such as La Gloria vein, the largest of this set of veins. These high-angle veins can be considered as a mineralized zone or lode of stock work, silicification, breccias, veins, vein breccias, veinlets, and a general mix of multiple styles of mineralization. Within this broader zone, for example the Martha lode ranges from 1 m to 35 m thicknesses and averages approximately 5 m.

7.4.2 Structural Geology

There are three main types of syn-mineralization veins and faults in the property:

- Type 1 – Silver-gold bearing, south–southeast– and south–striking, shallow west-dipping structures (e.g., the Martha fault zone).
  - These structures are commonly associated with faults and exhibit crustiform, cockade, and colloform textures that are representative of multiple vein opening stages. Veins generally have widths greater than 30 cm and can form vein zones up to several meters wide;
  - Steep down-dip (i.e., shallow west-plunging) mineral lineation and associated steps indicate that these structures developed as normal faults.
- Type 2 – Silver–gold bearing, south–southeast– to south-southwest–striking, moderate to sub vertical west-dipping structures (e.g., the Abundancia and La Gloria veins).
  - These structures contain veins that range in width from 1 cm to several tens of cm, and include rare up to 6 meters wide veins. Vein textures comprise colloform banding, dilation (jig-saw) breccia, and euhedral crystals projecting into cavities along the vein centers typical of extensional veins. Few faults are associated with these veins, although vein walls are sometimes characterized by smooth and striated post-mineralization faults;
  - Type 2 veins developed as extensional veins in the hanging wall and footwall of Type 1 structures. Rare syn-mineralization faults display steep-west plunging mineral lineation and associated steps indicating normal dip-slip movement.
- Type 3 - East–southeast–striking, moderate to steep south-dipping structures (e.g., two ESE structures, La Plomosa, and Transversal veins) with sporadic silver-gold bearing quartz veins. These structures are up to several meters wide, consisting of fault zones with variably developed quartz-carbonate-calcite veins and fault breccia commonly associated with hematite alteration of the host rock;
- Dominantly moderate to steeply west-plunging mineral lineation and associated steps along southwest-dipping veins indicate that these structures developed as normal-dextral oblique-slip faults.

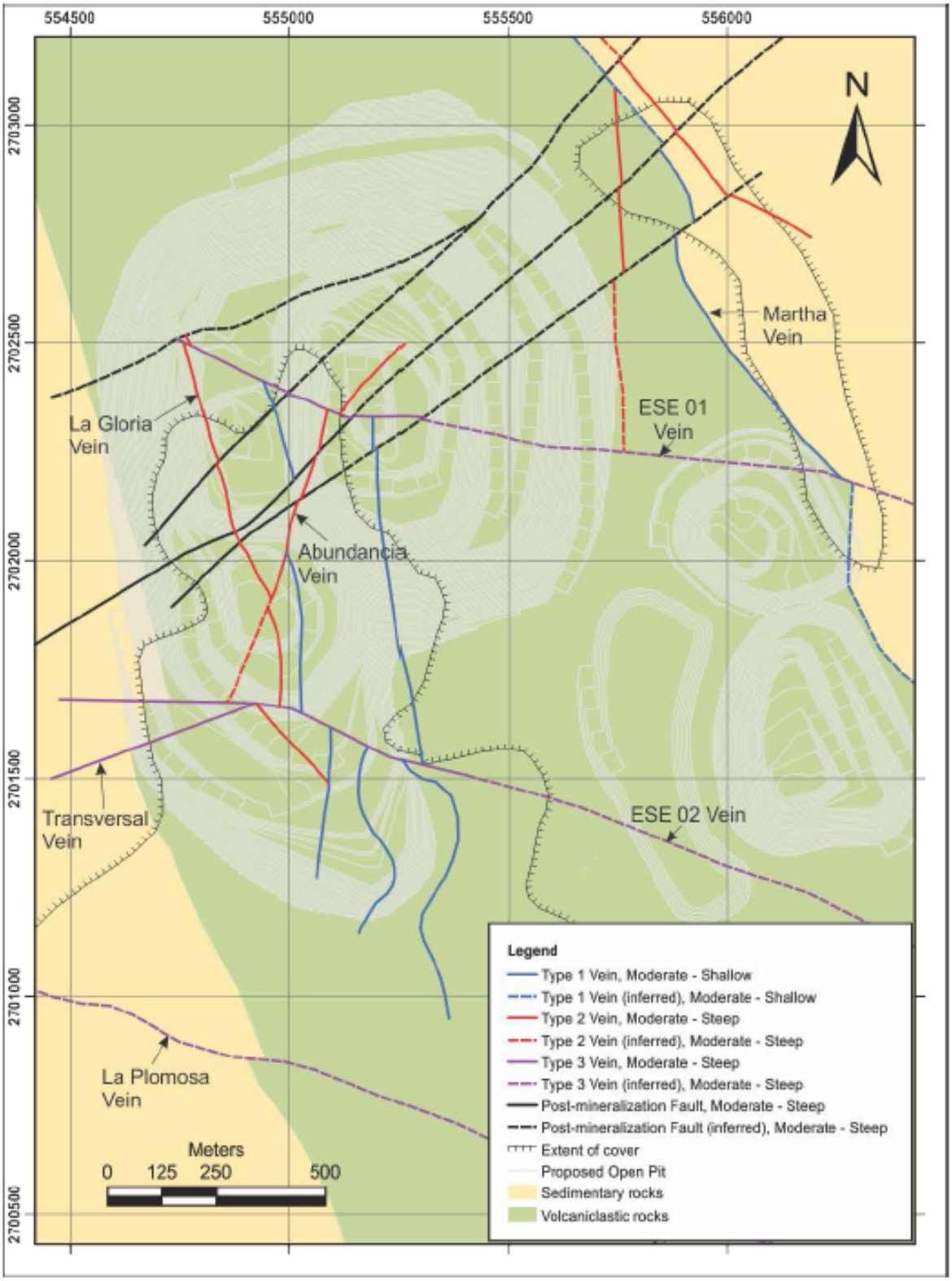


Figure 7-6: Structural Geology Map for the Project (SRK, 2014)

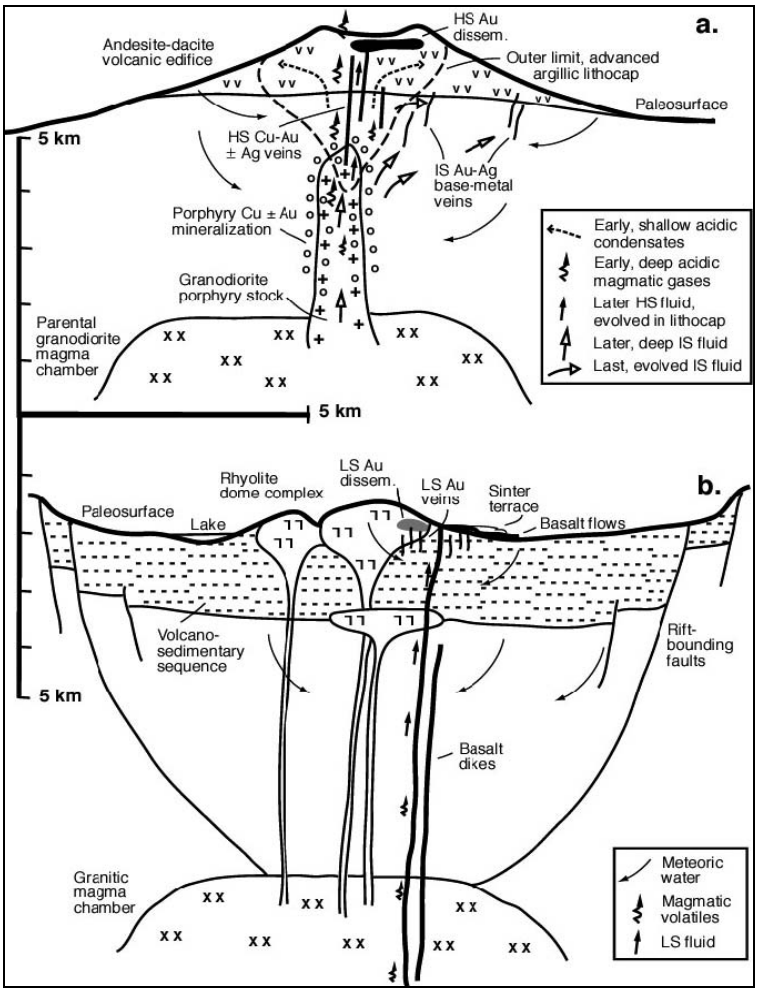
8 DEPOSIT TYPES

Silver and gold mineralization in the Project area can be grouped into the low sulphidation epithermal model of precious metal deposits. These types of deposits are found worldwide and have been commonly formed during the Cretaceous to Holocene.

Low sulphidation deposits occur as veins, breccias, and disseminated precious metal mineralization deposited by the circulation of neutral to weakly acidic hydrothermal fluids along regional fault structures, fracture zones, or through highly permeable lithologies such as ignimbrite and agglomerate. Because the fluids are relatively neutral, very little alteration is evident and the veins and nearby wall rock may commonly include illite, sericite, and adularia. Generally this style of mineralization is distal from a heat source.

Sillitoe and Hedenquist (2003) subdivide epithermal deposits into High (HS-), Low (LS-) and Intermediate-sulphidation (IS) types based on mineralogy, deposit morphology, associated alteration, and geologic setting (Figure 8-1).

Type IS epithermal deposits occur in a broadly similar spectrum (to HS deposits) of andesitic-dacitic arcs, but commonly do not show such a close connection with porphyry Cu deposits as do many of the HS deposits. However, high silica igneous rocks such as rhyolite are related to only a few IS deposits. IS deposits form from fluids spanning broadly the same salinity range as those responsible for the HS type, although Au-Ag, Ag-Au, and base-metal rich Ag-(Au) subtypes reveal progressively higher ore-fluid salinities.



a. Calc-alkaline volcanic arc with neutral to mildly extensional stress state showing relationships between high- and intermediate-sulphidation epithermal and porphyry deposits (note that the complete spectrum need not be present everywhere). Early magmatic volatiles are absorbed into ground water within the volcanic edifice (shown here as a stratovolcano, but it may also be a dome setting) to produce acidic fluid for lithocap generation, over and/or supra-adjacent to the causative intrusion. Later, less acidic intermediate-sulphidation fluid gives rise to intermediate-sulphidation mineralization, both adjacent to and distal from the advanced argillic lithocap. Where the intermediate-sulphidation fluid flows through the leached lithocap environment, it evolves to a high-sulphidation fluid (Einaudi et al., 2003) to produce high-sulphidation veins or disseminated mineralization, depending on the nature of the structural and lithologic permeability. The high-sulphidation fluid may evolve back to intermediate-sulphidation stability during late stages, supported by paragenetic relationships and lateral transitions of high- to intermediate-sulfidation mineralogy. b. Rift with bimodal volcanism and low-sulphidation deposits. Deep neutralization of magmatic volatiles, typically reduced, results in a low-sulphidation fluid for shallow low-sulphidation vein and/or disseminated mineralization and related sinter formation (Sillitoe and Hedenquist, 2003).

Figure 8-1: Schematic sections of end-member volcanotectonic settings and associated epithermal and related mineralization types.

The veins in the Project area consist of several stages of banded, crustiform (to colloform; quartz and cryptocrystalline quartz at shallow depths, adularia, barite and typically later carbonates both calcite and rhodochrosite, "illitic clay" (illite) commonly replaces the adularia (Coote, 2010). There are variable amounts of pyrite, sphalerite and galena plus argentite, and variable amounts of tetrahedrite-tennantite, freibergite and Ag sulfosalts.

The Ag:Au ratio is high, approximately 500:1 for the resource. Supergene oxidation extends to at least 300 m depth, and includes manganese oxide. There is abundant adularia, bladed calcite textures and coexisting vapor-rich and liquid-rich inclusions, all indicating an ascending, boiling fluid, consistent with the abundant evidence for brecciation, which suggests that that mixing caused metal deposition and carbonate formation.

9 EXPLORATION

9.1 SUMMARY OF PAST EXPLORATION

Exploration and other work at the Project date back to mining in the late 1800s on the Abundancia and La Gloria veins, two prominent veins exposed on the surface of La Preciosa Ridge. This work, which ceased in the early 1900's, and small-scale underground mining in the 1970s, resulted in the production of a small amount of material from these two veins, estimated by Mining Plus (Head and Collins, 2012) to be less than 30,000 tonnes. This tonnage estimation was not validated by Coeur but site inspections support that a small amount of mining was previously done.

The majority of work at the Project that is material to the mineral resources is from contemporary exploration, mainly drilling, conducted by Luismin, Orko, and PAS (Table 9-1). In addition to the drilling completed by these companies, other exploration activities, consisting of:

1. Prospect sampling by Orko in 2004, followed by geologic mapping by Orko geologists;
2. Completion of three IP (Induced Polarization) ground geophysical surveys in 2005 that totaled 40 line-km. The resistivity data did not appear to be a useful product of this work, but the chargeability component did identify an anomaly in the valley between La Preciosa Ridge and Zona Oriente; and
3. A large geochemical soil sampling program over a grid spanning 5 km north to south and 2 km east to west. This program produced anomalous analytical results from areas near shallowly covered veins such as Veta Nueva, Orito, and Nancy.

Historic exploration (along with recognition of late 1800s/early 1900s mining) was responsible for the identification of anomalous silver and gold in soils and outcropping veins.

9.2 COEUR EXPLORATION AND DEVELOPMENT

Coeur's 2013-2014 drilling program was divided into three types:

- Type I drilling: completion of 21 RC drillholes to test and condemn waste dumps and tailings impoundment areas, drilling commenced January 2014 and completed February 2014.
- Type II drilling: infill core drilling between February 2014 and mid-April 2014 completed a 75-hole drilling program totaling 11,437m. Drilling targeted the first three years of the mine plan to convert inferred to indicated resources and reduce risk in achieving the early mine plan. All drilling was concentrated around the Abundancia Ridge area.
- Type III drilling: from December 2013 to March 2014 Major Drilling, under KP supervision, completed seven HQ3 core holes specifically to obtain geotechnical data in the area of the design pits, tailings impoundment, and process plant footprint. Subsequently, these holes also were sampled for geochemical data.

Coeur has completed development and exploration work at the Project in 2013-2014, as shown in Table 9-1.

Table 9-1: 2013-2014 Coeur Exploration and Development Work Summary

Quantity	Data type	Totals	Target
75 drillholes	Core Holes	11,437 m	In-fill drilling, resource conversion
21 drillholes	RC Holes	8,543 m	Waste dumps and tailings, condemnation drilling
7 drillholes	Core Holes	2,244 m	Geotechnical information
103 drillholes	Drill samples	12,358 samples	New assay samples from older drillholes
NA	Geophysical	300 km^2	Magnetic survey for lithological and structural domains
NA	Geologic mapping (surface and underground)		Define structural geology
NA	Drillholes (scanned 109 new and old drillholes)	35,754 m	IR measurements to define alteration
NA	Drillholes (scanned 26 new and old drillholes)	6,166 m	Televiewer scans for structural geology and geotechnical data

In the opinion of the QP, Coeur's drilling, sampling, and logging was done to industry standards. A total of 25.908 meters of RC samples, or 17 intervals, was logged as NR (no return), which is 0.3% of the total amount of RC drilled in 2014 (Table 9-1). RC drilling was specifically focused on exploring sites for waste rock and tailings facilities. Core recovery is reported as 100 percent, no NR intervals were reported. Because the 2014 core drilling program was designed to infill between existing drillholes, the resulting samples are representative of the mineralization as a whole and are not biased in their location, orientation, sampling method, or metal grade. Since the core drilling infilled the area designed to be mined in the first three years of the mine plan, the spatial density of sampling is good, sufficient for much of the material to be classified as indicated or measured.

In the opinion of the QP, the quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs completed by Coeur, Orko, PAS, and Lusmin are sufficient to support Mineral Resource and Mineral Reserve estimation as follows:

- Core logging meets industry standards for gold exploration.
- Collar surveys have been performed using industry-standard instrumentation.
- Downhole surveys were performed using industry-standard instrumentation.
- Recovery data from core drill programs are acceptable.
- Geotechnical logging of drill core meets industry standards for planned open pit operations.
- Drill orientations are generally appropriate for the mineralization style, and have been drilled at orientations that are optimal for the orientation of mineralization for the bulk of the deposit area.

Core logging did not reveal any unusual geologic features that have not been observed in previous logging in the Project area. Assay results and location of mineralized intercepts are consistent in spatial location and grade of previous drilling in the Project area and no unusally high-grade intercepts or previously unknown mineralized areas were encountered, i.e. the distribution of sample grades from the 2014 drill program are similar to distributions of grades from previous drill programs.

10 DRILLING

In early 2014, Coeur planned and drilled 75 HQ diamond drillholes for a total of 11,437m, with an average depth of 150m, and core recoveries of 85%. Drilling was done by Layne de Mexico S.A de C.V.

In addition to infill core drilling, an additional 2,244m of core was drilled for the geotechnical investigation by Major. These core holes were logged and on completion of geotechnical work, the core was split, sampled, and assayed; however, the assay and geology data were not available in time for use in the resource model.

All drillholes, with the exception of reverse circulation drillholes for condemnation drilling, are diamond core holes of varying diameters, mainly HQ and some NQ diameter drillholes. Exploration and development drilling to delineate mineral resources has been performed in sequential campaigns by Luismin, Orko, PAS, and Coeur as summarized in Table 10-1 (excludes RC drilling because RC was not used in resource estimation).

Table 10-1: Drilling Summary

Company	Years	Area	Number of Drillholes	Meters of Drilling	Hole Number Prefixes
Luismin	1981, 1982, 1994	La Preciosa	8	1,630	BP
Orko	2006	Orito	7	2,326	BO
	2007	San Juan	8	3,554	SJ
	2005	La Preciosa	1	451	BC
	2006		6	1,910	BB
	2005-2008		366	144,126	BP05-BP08
PAS	2009-2010		363	91,095	BP09-BP-10
Orko	2011 -2012		5	500	BP11-BP12
Coeur	2013-2014	La Preciosa	103	22, 324	CLP14, KP14, KP13, DH13
		TOTALS	**867**	**267,916**	

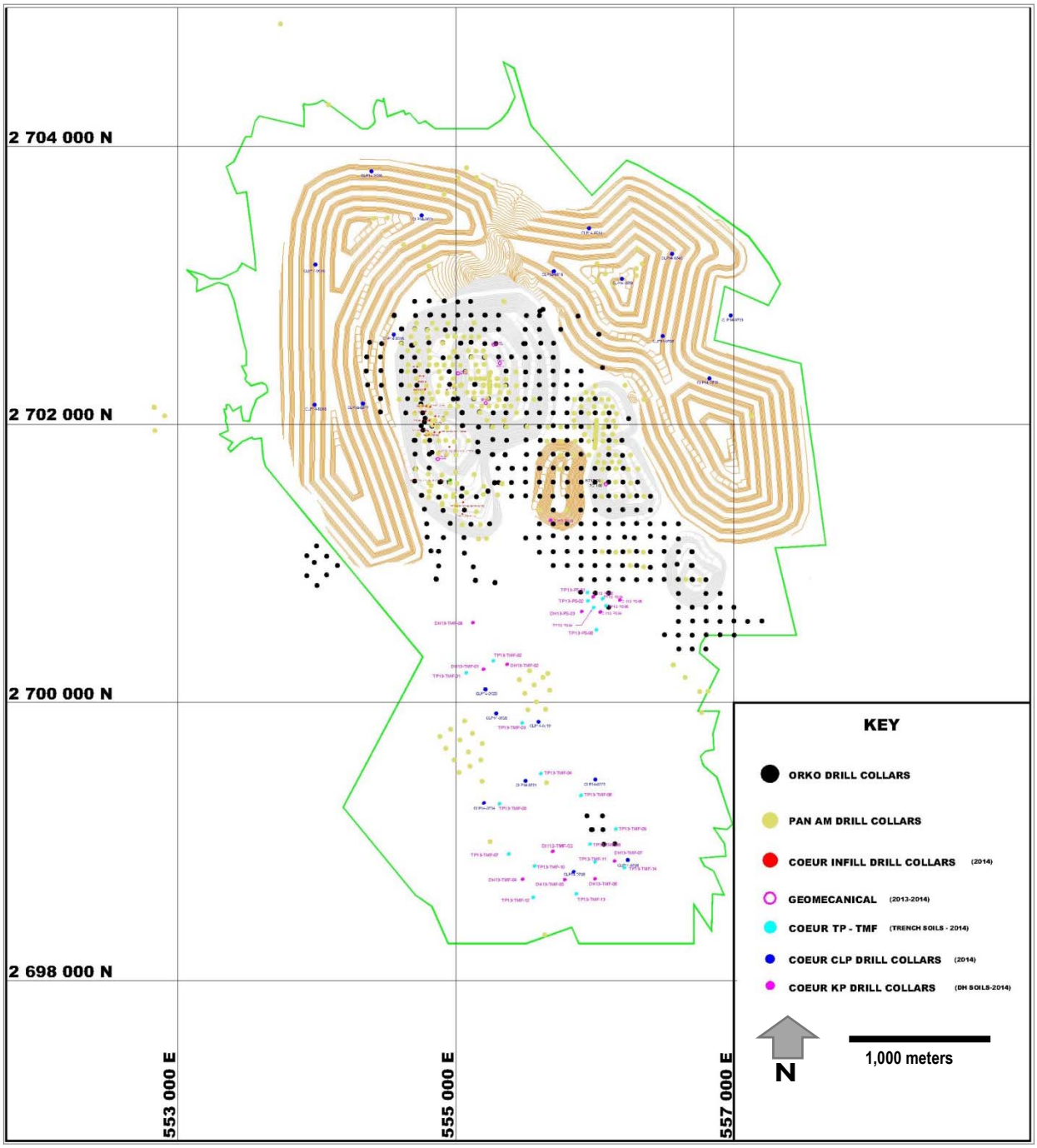


Figure 10-1: Drillhole Location Map (Coeur, 2014)

10.1 DRILLING BY LUISMIN

Of the seven Luismin drillholes in the Project database, two were drilled from underground workings and five from the surface. The primary targets were the Abundancia and La Gloria veins, which run semi-parallel to the NNW-striking Abundancia Ridge, at depths of 50 to 75 m below the primary underground workings on the 2065m level (elevation). Lusmin drilled one additional drillhole 313 m deep in 1994 in the eastern vein breccia system, but data for this drillhole are not available. There are no available details on the Luismin drilling procedures, except that the drill core was either small-diameter BQ or AX size. The remaining half-core from these holes is stored in the original core boxes on site.

10.2 DRILLING BY ORKO (2005 TO 2008, 2011, 2012)

Orko began drilling in March 2005, ultimately completing 388 diamond drillholes totaling 152,368 m of core, spaced on roughly 100 m centers, with all but 16 of the holes targeting various veins. Orko used Major for all of its drilling using Longyear 44, 38A, and 38B core drills. Drill core diameters started at HQ-diameter, with reductions to NQ-diameter at around 260 m downhole. Between rod changes the drillers inserted a wooden “run” block in the core boxes marked with the downhole depth in both feet and meters. Downhole surveys were taken approximately every 50 m down the hole with a Reflex survey instrument. The results of these surveys indicated only moderate deviation in downhole azimuths and inclinations.

Drill core was collected on a daily basis from the drill rig by Orko technicians, who taped the boxes shut prior to transporting the core to the site core shed. Once at the shed, technicians cleaned the boxes and core, marked the boxes with the hole number, box number, and the depth intervals, and reconciled these data with the depths marked on the driller’s core run blocks.

After completion of each hole, a PVC pipe was placed in the hole collar and a concrete cap was poured around the collar PVC pipe, and a length of PVC pipe was left protruding above the concrete cap. The concrete cap was inscribed with the drillhole number, total hole depth, and the azimuth and inclination of the hole at the collar. An independent surveyor was contracted to survey the collar coordinates on a regular basis.

10.3 DRILLING BY PAS

PAS began drilling in June 2009, under the terms of PAS’s Option Agreement to acquire a joint venture interest in the Project from Orko, and PAS completed 331 diamond drillholes. The drilling focused on in-filling the 100 m center grid previously completed by Orko. PAS’s drilling resulted in a spacing of 50 m on every other section (100 m apart) over an area approximately 800 m × 800 m. This selective tighter spaced drilling area is located in the northern part of the deposit. Additionally, infill drillholes were drilled on selected sections as well as on two 15- to 20-meter close-spaced fences to assess the short-range continuity of geology and mineralization. Major was also used by PAS to do the drilling program, which resulted in similar drilling and downhole surveying procedures as Orko, although greater capacity drill rigs were employed which resulted in fewer NQ-diameter drillholes. Beginning in early 2010, selected drillholes were surveyed using a Reflex ACT/QPQ orientation tool to obtain oriented drill core for geotechnical purposes. The drillhole collar monuments and the survey of collar coordinates followed the same procedures established earlier by Orko.

10.4 DRILLING BY COEUR

Between January and April 2014 Coeur drilled a total of 75 HQ core drill, 21 reverse circulation (RC) holes totaling 19,980 m, and 7 geotechnical core holes totaling 2,244 m. A majority of Coeur’s drillholes were oriented west to east at varying dips, depending on the target vein orientation, to minimize the drillhole intersection angle with the vein. In general, the downhole length of the drill intersection approximates the true thickness of the vein, but this length can vary from hole to hole. Most of the 2013 drillholes were completed in the main deposit area, an area approximately 3000 m north to south by 2000 m east to west.

RC condemnation drillholes were designed to test potential mineralized targets within the boundaries of waste dumps and tailings facilities. RC drillholes were drilled on -45° to -60°, to -90° inclination with N-NE and SW-W azimuths. Infill drillholes were designed to increase the amount of Measured and Inferred material in the first three years of the mine plan (an area encompassing Abundancia Ridge). These drillholes were inclined from -40° to -85°, with azimuths from N-NE to SW-W.

Downhole surveying in core and RC drillholes was done from top to bottom approximately every 10 meters downhole using a gyroscope survey instrument. Results of these surveys indicated only minor deviation in azimuth and inclination. Collars were surveyed with total station instrument using WGS 84 coordinate system.

Drill core was collected on daily from the drill rig by Coeur technicians, who taped the boxes shut prior to transporting the core to the onsite core shed. In the core shed technicians cleaned the boxes and core, marked the boxes with the drillhole number, box sequence number, depth intervals, and checked recorded depths against the depths marked on the driller's core run blocks. Color digital photographs of each core box were taken before the core was split and sampled. Core was then laid out and logged, using paper logging forms, by project geologists, who also marked sampling intervals according to Coeur QA/QC sampling protocols. Briefly sampling criteria requires samples to be greater than 50 cm and less than 200 cm in length. Logging describes all common features, such as rock type, alteration, mineralization, faults, etc.

RC drill sampling was done on 5 foot or 1.5 m intervals. Samples were collected after passing through a cyclone under both wet and dry conditions. Samples were placed in plastic bags when drilling dry material and in Micropore bags when the drilling was wet.

After completion of each hole, a PVC pipe was placed in the drillhole collar and a cement cap or monument was poured around the collar PVC pipe and inscribed with the drillhole number, total drillhole depth, and the azimuth and inclination of the hole at the collar. An independent surveyor was contracted to survey the coordinates of each collar on a regular basis.

10.5 CORE RECOVERY AND ROCK QUALITY DESIGNATION (RQD)

Only Orko drilling had drillhole core recovery and RQD values recorded in the acQuire database, these types of data were not recorded by PAS or Luismin. Although Coeur recorded core recovery and RQD measurements, at the time of this report was written those data had not been entered into the acQuire database, thus no analysis was done.

Orko's recovery values are reasonable with a mean core recovery of 94.5 percent and a mean RQD of 54.3%. For both silver and gold there is a decrease in Ag and Au grade with increasing core recovery where recovery is >20 percent, which suggests that there is a small sampling bias with loss of material in higher grade zones. Grades decrease slightly with increasing core recovery, however maximum grades increase with increasing core recovery because of the greater number of intervals with better core recovery. It is important to note that the data are not normalized for the number of recovery measurements. Because the highest Ag and Au grades are typically found in quartz veins, and core recovery in quartz veins tends to be lower because the veins are fractured, the grade-recovery relationship is expected. Given the small number of recovery measurements in the range of 0-40 percent recovery, and lower maximum grades in the 0-40 percent range, the impact of core recovery on the resource estimate is insignificant.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLE COLLECTION METHODS

11.1.1 Luismin (1981, 1982, 1994)

No documents/reports are available that describe the sampling methods used by Luismin. Based on descriptions provided by visual inspection of Luismin core by previous Qualified Persons, the core was split using a conventional manual splitter and properly "marked and neatly stored". Luismin reportedly collected a total of only 130 samples for assay, with variable sample lengths that ranged from 0.5 to 2.0 meters. The breaks between samples reportedly respected geologic features.

11.1.2 Orko (2003-2008, 2011, 2012)

Orko technicians transported core from the drill rigs on a daily basis to the core logging facility, where it was cleaned and the core boxes marked with the hole number, box number and from-to depth intervals. Each box of core was then photographed and moved to a rack for examination by a geologist who logged lithology, structure, alteration, mineralization type, intensity and sulfide percentage and oxidation and assigned codes for rock types, structures, and veins. Logging was done manually on paper logging forms. Following geologic logging, geotechnical data including core recovery, was recorded. After logging was complete, the geologist marked the sample intervals on the core and on the core box dividers with a permanent marker, along with a cutting line along the longitudinal axis of the core and recorded the sample interval depths and corresponding sample numbers on the geological log. The core was then sawn in half along the cut line by an Orko technician using a water-cooled diamond saw, after which one half of the interval was placed in a plastic sample bag along with a sample tag. The remaining half was returned to the core boxes that then were placed on numbered racks in a large, secure, storage shed at the Project site.

In order to determine material density, a single piece of the sampled core was removed from each sample sack, allowed to air-dry, and then dry weighed for measurement of specific gravity. Once measured, the core piece was returned to the appropriate sample bag and the whole sample was placed in a rice sack for transport to the Inspectorate de Mexico sample preparation facility in the city of Durango. Prior to transport, each rice sack was weighed and the total weight recorded. All samples were in the possession of Orko personnel from the diamond drill rigs to the Inspectorate lab.

11.1.3 PAS (2008-2010)

PAS followed essentially the same drillhole logging and sampling procedures and protocols developed by Orko, beginning with PAS drillhole BP10-458 onwards. The geologists determined the diamond core sample intervals and marked the positions of the intervals on both the core and the core box dividers. The core was then cut along the cut line marked on the core by the geologists using a water cooled diamond bladed saw, and both halves were placed back in the core boxes for transport to the core sampling area. Sample bags and sample tags were labeled with the consecutive sample numbers assigned to the sample intervals, with numbers reserved for insertion of QA/QC samples. The pieces of half core to be assayed were then placed in the appropriate labeled sample bags along with the corresponding sample tag, and then the bags containing the individual samples were inserted in groups of ten into labeled rice sacks along with the labeled standard and blank QA/QC samples. The rice sacks filled with samples were stored on site until transported by a PAS employee to the SGS de Mexico laboratory in the city of Durango, Mexico.

11.1.4 Coeur (2013-2014)

The Coeur development program consisted of reverse circulation (RC) and core drilling. The RC drilling program was conducted by two drill rigs contracted from Layne de Mexico. One geologist was assigned to each active drilling shift. Geologists were provided with a package of sample tags which indicated the sample identification and the interval. Sample tags were included inside each sample bag and a permanent marker was used to note the sample identification

and interval on each sample bag. A geologic description was recorded on a paper log at the drill rig, including any additional notes on the drilling or sample. This log was later transferred to an electronic format. Coeur's company protocol for quality control (Coeur, 2012) was applied throughout the sample collection process. All RC samples were collected by Coeur technicians, accompanied by a project geologist. Samples were collected at 1.5 m intervals in two five gallon buckets. The entire sample was weighed, with typical weights ranging from 100-125 kg. The sample was initially split in half using a single Jones-type splitter with one half of this split bagged for analysis at the commercial laboratory. The remaining sample is split once more, retaining 1/8 of the original sample, and bagged for storage in the project warehouse.

Core was collected at the drill rig and transported to the core logging facility on a daily basis, where it was cleaned and the boxes were marked with hole number, box number, and the sample interval. Each box of core was photographed and moved to a rack for examination by a geologist. After geologic and geotechnical logging were complete, geologists then marked sample intervals on the core and on the core box dividers with a permanent marker, along with the cutting line along the longitudinal axis of the core. All sample intervals and corresponding sample numbers were recorded on the geologic log. The core was then sawn in half along the cut line by a Coeur technician using a water cooled diamond saw. One half of the interval was placed in a plastic sample bag along with a sample tag. The remaining half was returned to the core box and placed on numbered racks in a large, secure, storage shed at the project site.

11.2 SAMPLE PREPARATION AND ANALYSIS PROCEDURES

11.2.1 Luismin (1981, 1982, 1994)

The drillhole samples collected by Luismin were transported to the company's in-house laboratory in Durango. No written records of the chain of custody, sample preparation, or sample analysis procedures are known to exist.

11.2.2 Orko (2003-2008, 2011, 2012)

11.2.2.1 Sample Preparation

Orko used two-sample preparation labs located in the city of Durango – Inspectorate and SGS. For most of 2005 to 2007, SGS was the primary lab used and Inspectorate served as the secondary lab. After completion of hole BP07-93 or thereabouts, the primary and secondary lab designations were switched and Inspectorate became the primary lab in order to improve assay turn-around times. Upon receipt at both the SGS and Inspectorate sample preparation laboratories, the samples were placed in order according to sample number, and then crushed, and a sub-sample split was taken for pulverization. The remaining coarse rejects were returned to the project site and stored. Neither preparation lab was ISO nor IEC certified at the time the Project samples were processed.

11.2.2.2 Sample Analysis

The sample pulps were sent to Inspectorate's analytical laboratory in Reno, Nevada, USA, which was ISO 9001:2008 certified, and to the SGS analytical laboratory in Toronto, Canada, which was accredited by ISO/IEC 17025. Sample pulps representing check assays also were sent to these analytical facilities, as well as to ALS Chemex in North Vancouver, Canada and ALS Chemex in Reno, Nevada, USA, each of whom is independent of Coeur. At the SGS analytical laboratory in Toronto, the pulps were analyzed by several methods. Gold content was determined by fire assay at a detection limit of 5 ppb Au. Silver was analyzed by atomic absorption spectrometry (AAS), at a calibrated detection limit of 0.3 g/t Ag and an upper limit threshold of 300 g/t Ag. Samples with silver values greater than 300 g/t Ag based on this analytical method were re-run by fire assay with a gravimetric finish. All samples also were subjected to strong acid digestion followed by a 40 element Inductively Coupled Plasma (ICP) analyses, including silver.

Some of the elements in the ICP package have threshold limits for ICP analysis. Examples include silver, which due to its 10 g/t upper ICP threshold does not allow the method to be used for this Project because over half of all samples

exceed this value. Similarly, the base metals Pb and Zn and the element Ba have an upper threshold of > 10,000 g/t, (or 1.0%), which also precludes the use of ICP analysis for these elements. For the minerals containing any of the 40 elements that are totally digestible by strong acids, such as oxide, sulfide and carbonate species, the ICP analysis method works well. However, for minerals containing any of these elements that are resistant to the strong acid digestion, only partial values will result.

The laboratory procedures used at the Inspectorate lab in Reno were similar to those used by SGS and described above. However, silver was an exception where, due to more precise instrument calibration, the detection limits were a lower 0.1 g/t (g/t) Ag and the upper threshold limit was 200 g/t Ag. As a result, samples having silver contents greater than 200 g/t were subsequently re-analyzed by fire assay with a gravimetric finish.

Orko completed two drillholes in 2011 and three drillholes in 2012 for a total of 500 meters. Only 29.2 meters of this drilling was sampled and assayed, according to the Orko database. No record of QA/QC procedures and results exists for these drill campaigns.

11.2.3 PAS (2008-2010)

11.2.3.1 Sample Preparation

With the exception of the pulp duplicate samples, all PAS samples were prepared and assayed by SGS Laboratories in Durango, México. Upon arrival at the SGS laboratory, the samples were assembled in numerical order according to the sample tag numbers, individually crushed, then riffle split to provide a sub-sample for pulverizing. The pulverized, approximately 200 g sub-sample, was placed in a small labeled paper packet. After the required assay aliquots were removed, the residual material remaining in the packet was returned to PAS for storage on site at the Project, along with the coarse reject that remained after splitting of the assay sub-sample.

Pulp duplicate samples were analyzed at Inspectorate's lab in Sparks, Nevada.

11.2.3.2 Sample Analysis

Sample pulps analyzed at the SGS laboratory used the following procedures:

- For gold analyses at SGS, all samples were initially assayed using fire assay procedures with atomic absorption spectroscopy (AAS) finish. The detection limit for this procedure was 0.005 g/t and the maximum assay threshold was 10 g/t. For samples initially assaying more than 10 g/t Au, these were rerun using a fire assay with gravimetric finish procedure having a detection limit of 3 g/t Au,
- For silver analyses at SGS, all samples were initially analyzed using 3-acid digestion with an AAS finish (0.3 g/t detection limit). For samples with analyses greater than the 300 gram Ag threshold limit, the samples were rerun using a fire assay with gravimetric finish procedure having a detection limit of 5 g/t Ag. In addition, 33-element trace analyses using a 2-acid digestion and ICP finish having a 2 g/t detection limit and a threshold of 10 g/t for silver were completed for all samples,
- For gold analyses at Inspectorate, all samples were run by fire assay with a gravimetric finish that had a detection limit of 3 g/t Au, and
- Silver analyses for all samples run at Inspectorate were initially run using a 4-acid digestion with ICP finish (0.1 g/t Ag detection limit) that had a 200 g/t Ag upper threshold limit. For samples with analyses greater than 200 g/t, the samples were rerun using fire assay with a gravimetric finish that had a detection limit of 5 g/t and an upper threshold limit of 5,000 grams per tonne Au.

11.2.4 Coeur (2013-2014)

11.2.4.1 Sample Preparation

All Coeur samples in 2013 and 2014 were submitted to an accredited commercial laboratory. Coeur contracted ALS Laboratory in Zacatecas, ZAC, MX to complete all sample preparation on RC cuttings and split HQ drill core. The sample is logged in the tracking system, weighed, dried and finely crushed to better than 70% passing a 2 mm screen. A riffle split of up to 250 g is taken and pulverized to better than 85% passing a 75 micron screen. The method is appropriate for both RC cuttings and drill core.

11.2.4.2 Sample Analysis

Sample pulps were created in Zacatecas and sent to ALS's analytical laboratory in Vancouver, BC, CA which is ISO 9001:2008 certified. Orko era pulps representing re-assays were sent to ALS Vancouver, as well as to SGS in Lakefield, ON, CA which is ISO 17025 certified. Both labs are independent of Coeur.

11.2.4.2.1 Silver Detection

At ALS Laboratory, silver content was determined by Inductively Coupled Plasma-Atomic Emission Spectroscopy (ICP-AES). A 0.25 g sample is digested with perchloric, nitric, hydrofluoric, and hydrochloric acids. The residue is topped up with dilute hydrochloric acid and the resulting solution is analyzed. The lower and upper detection limits for this method are 0.5 ppm and 100 ppm, respectively. At 100 ppm the sample triggers an additional 4-Acid Digestion ICP-AES analysis that is optimized for accuracy and precision at high metal concentrations. This method utilizes the same acids as the prior method, but includes additional stages of heating and drying, along with the addition of de-ionized water to aid in further digestion. The lower and upper detection limits for this method are 1 ppm and 1500 ppm, respectively.

At SGS Laboratory, silver content was determined by Inductively Coupled Plasma-Atomic Absorption Finish. This is a 4-Acid digestion. A 2 g sample is digested with perchloric, nitric, hydrofluoric, and hydrochloric acids. The lower and upper detection limits are 0.3 g/t and 300 g/t respectively.

11.2.4.2.2 Gold Detection

Gold content was determined by Inductively Coupled Plasma-Atomic Emission Spectroscopy (ICP-AES), following an initial Fire Assay Fusion of a precious metal bead. The sample bead is digested in 0.5 ml dilute nitric acid in the microwave oven. 0.5 ml of concentrated hydrochloric acid is then added for further digestion. The lower and upper detection limits of this method are 0.001 ppm and 10 ppm, respectively. At 10 ppm the sample triggers an additional Gravimetric analysis. This includes the creation of a lead button containing the 30 g sample, which is then cupelled to remove the lead. The remaining gold and silver bead is parted in dilute nitric acid and weighed as gold. The lower and upper detection limits for this method are 0.05 ppm and 1000 ppm, respectively.

At SGS Laboratory, gold content was determined by exploration grade fire assay. This is a 30 g fire assay with an ICP finish. The lower and upper detection limits are 1 ppb and 10000 ppb, respectively.

11.2.4.2.3 Multi-Element Detection

At ALS Laboratory, an additional suite of 40 elements were also analyzed by ICP for all new samples created in 2013 and 2014. The drilling also encountered base metal values of zinc and lead which triggered the multi-element over limit method. Table 11-1 lists the elements and associated units, and detection limits.

Table 11-1: Multi-Element ICP Package Analyzed by Coeur

Element	Symbol	Units	Lower Limit	Upper Limit
Aluminum	Al	%	0.01	50
Arsenic	As	ppm	5	10000
Barium	Ba	ppm	10	10000
Beryllium	Be	ppm	0.5	1000
Bismuth	Bi	ppm	2	10000
Calcium	Ca	%	0.01	50
Cadmium	Cd	ppm	0.5	500
Cobalt	Co	ppm	1	10000
Chromium	Cr	ppm	1	10000
Copper	Cu	ppm	1	10000
Iron	Fe	%	0.01	50
Gallium	Ga	ppm	10	10000
Potassium	K	%	0.01	10
Lanthanum	La	ppm	10	10000
Magnesium	Mg	%	0.01	50
Manganese	Mn	ppm	5	100000
Molybdenum	Mo	ppm	1	10000
Sodium	Na	%	0.01	10
Nickel	Ni	ppm	1	10000
Phosphorus	P	ppm	10	10000
Lead	Pb	ppm	2	10000
Sulfur	S	%	0.01	10
Antimony	Sb	ppm	5	10000
Scandium	Sc	ppm	1	10000
Strontium	Sr	ppm	1	10000
Thorium	THz	ppm	20	10000
Titanium	Ti	%	0.01	10
Thallium	Tl	ppm	10	10000
Uranium	U	ppm	10	10000
Vanadium	V	ppm	1	10000
Tungsten	W	ppm	10	10000
Zinc	Zn	ppm	2	10000

11.3 SAMPLE SECURITY

RC samples are collected and bagged at the drill rig by Coeur technicians, accompanied by a project geologist. The bags are labeled with a unique sample ID and the interval meterage. Core samples were bagged by Coeur technicians at the project logging and storage facility. Samples were delivered daily by Coeur technicians and a project geologist to the ALS laboratory in Zacatecas, ZAC, MX. Sample prep was completed Zacatecas and pulps samples were shipped to ALS Vancouver, BC, CA for analytical test work.

Chain of custody for delivery is established by transmittal sheets and sample receipt documents from the lab. Final chain of custody is ensured through electronic delivery of work orders and PDF assay certificates.

Hard copies of assay certificates and the geologic logs are stored at the project office in Durango, MX. The geologic logs include the sample sequence list, including inserted QAQC. Electronic copiers of all data are stored on a corporate server in Chicago. This server is backed up regularly. Ultimately all data are stored in an acQuire database located on an independent, backed-up server.

Coarse reject and sample pulps are returned to the Project site by laboratory staff and stored onsite in multiple secure storage facilities.

11.4 ANALYTICAL RESULTS

11.4.1 Assay Methods

Table 11-2 contains a listing of the assay methods and associated metadata used by Coeur. The sample preparation, security and analytical procedures are adequate and within industry accepted norms.

Table 11-2: Assay Methods

Analytical Laboratory	Element	Analytical Method	Units	Lower Limit	Upper Limit
ALS	Ag	ME-ICP41	ppm	0.2	100
ALS	Ag	ME-ICP61	ppm	0.5	100
ALS	Ag	ME-OG46	ppm	1	1,500
ALS	Ag	ME-OG62	ppm	1	1,500
ALS	Ag	GRA21	ppm	5	10,000
SGS	Ag	GE-AAS42E	g/t	0.3	300
ALS	Au	ICP21	ppm	0.001	10
ALS	Au	GRA21	ppm	0.005	1,000
SGS	Au	GE-FAI313	ppb	1	10,000

11.4.2 Data Delivery and Storage

Following the completion of analyses at the commercial laboratory, electronic results are delivered via email to a distribution list of Coeur recipients, approved by the project manager. ALS also provides secure online access to review the status of work orders and offers the ability to download data files and certificates.

Data are loaded into the acQuire database by a database manager or geologist with sufficient acQuire permissions. AcQuire is designed to securely store all original data. Acquire uses calculated and derived fields to produce data in a consistent format that can be uploaded into a 3D modeling package which allows for a further visual review of the data.

11.5 QUALITY ASSURANCE AND QUALITY CONTROL (QA/QC), CHECK SAMPLES, AND CHECK ASSAYS

11.5.1 Luismin QA/QC (1981, 1982, 1994)

There are no records of any QA/QC programs or protocols prior to 2003.

11.5.2 Orko QA/QC (2003-2008, 2011, 2012)

11.5.2.1 Orko QA/QC Procedures

Orko maintained a QA/QC program during its tenure that consisted primarily of inserting standards and blanks into the sample sequence. Although no duplicates were included in the regularly submitted sample batches, duplicates (check samples) were submitted to a secondary laboratory in separate batches to check for systematic bias by the primary assay laboratory.

According to earlier Technical Reports (MDA, 2009, Snowden 2011a, Mining Plus 2012), alternating standards and blanks were inserted every tenth sample in the sample sequence, equivalent to a 5% insertion rate for each sample type. Mining Plus noted that based on the 88,235 core samples submitted by Orko to the primary laboratories, a 5% insertion rate is roughly equivalent to 4,400 blanks and an equal number of standards. However, Mining Plus stated that in the QA/QC files in the database provided for its mineral resource estimate, there were data for a slightly smaller number of standards (3,994 standards, 4.3% of the total samples), but a significantly lesser number of blanks (1,127 blanks, or 1.3% of the total samples) and 1,103 duplicates. Similar quantities of Orko QA/QC data were reported by MDA and Snowden. The data verification process performed by the Qualified Persons responsible for this Report also detected a shortfall in the amount of expected QA/QC data, which is discussed in the following Section 12 of this Technical Report.

Mining Plus and Snowden stated in their Technical Reports that Orko's blank samples consisted of a combination of basalt core drilled during its exploration program and material collected from basalt boulders found on the property. The basalt blanks reportedly used were designated as Orko-2, Orko-4, Orko-5, Orko-7, and Orko-9. MDA noted in its Technical Report that SGS provided certificates for the basalt blanks based on approximately 150 analyses by aqua regia digestion and ICP-AES finish, but that no round-robin multi-laboratory analyses were done to substantiate the SGS values for the blanks.

Early on in Orko's exploration program, two commercial standards were used in a small number of sample batches, one of which was certified for gold and silver, and the other for gold only. The accepted values for these standards are unknown. For most of Orko's tenure, custom standards were used that were prepared from a stockpile of mineralized material situated near the Luismin portal. An unspecified amount of this material was sent to SGS's metallurgical division for certification prior to preparation of the standards. MDA reported that this certification was based on approximately 150 analyses. As with the SGS-certified basalt blanks, MDA noted that values established for the standards were not supported by round-robin testing at multiple labs. Over the course of Orko's exploration activities, four such custom standards were compiled, numbered as follows with accepted values in parenthesis: Orko-1 (0.210 g/t Au, 293.40 g/t Ag), Orko-3 (0.068 g/t Au, 112.00 g/t Ag), Orko-6 (0.072 g/t Au, 146.10 g/t Ag), and Orko-8 (0.134 g/t Au, 237.90 g/t Ag). In addition to these standards, in accordance with recommendations by MDA, a fifth standard was compiled that was subjected to round-robin multi-lab analyses. This standard, Orko-10, did not have a final certified value at the time MDA issued its Technical Report (results from one of the five round-robin labs had not been received). This standard subsequently saw limited use by PAS, and is discussed in Section 11.3.3.1 of this Technical Report.

11.5.2.2 Orko QA/QC Results

The results of Orko's QA/QC results are not easily interpreted. The standards were problematic because they were not certified, and record keeping was reported to have been inadequate such that the identity of the standards in the sample batches and assay certificates was not always certain (MDA, 2009). The standard deviation of some of the standards was unusually high, particularly silver for Orko-1 and gold for all standards. The relative closeness of the accepted silver values for the high grade standards (Orko-1: 293.40 g/t Ag, and (Orko-8: 237.90 g/t Ag) and the moderate grade standards (Orko-3: 112.00 g/t Ag and Orko-6: 146.10 g/t Ag) result in overlapping of the two-standard deviation ranges for the standard pairs, making it unclear whether some observed "failures" falling outside of these ranges were due to inconsistencies in the standards themselves, mislabeling of the standards during sample submission (as noted by MDA), or actual errors in the assay analyses. Snowden's graphical plots of the standard results suggest that there may have been some switching of standard labels for standards Orko–3 and Orko-8.

MDA (whose personnel were the only independent Qualified Persons involved one-on-one with Orko personnel during exploration drilling) reported that although Orko examined the standard sample assay results on a batch-by-batch basis, these results were not systematically charted over time, and as a consequence analytical failures were not investigated in a timely manner. This was compounded by the uncertainty as to the identities of some standards (as

described above), which made the identification of actual failures all the more difficult. As a result, trends in the results of standards during drilling were not identified. Also, in a number of cases the standards were analyzed by an analytical method that differed from the method used to analyze certain drill core samples in a batch. This occurred at the Inspectorate lab, where all initial silver assays that fell below 200 g/t Ag were analyzed using an ICP method (which occurred for two out of the four standards), whereas those samples with initial ICP assays greater than 200 g/t Ag were analyzed by fire assay with gravimetric finish. However, after reviewing the results of the Orko standards analyses, MDA concluded that the data neither revealed any systematic analytical biases, nor did it provided assurance that no such problems existed.

Four percent of the samples representing the Orko-2 blank and 10% of the basalt drill core samples exceeded the allowable value for silver, while the basalt drill core samples experienced no failures for gold. However, due to the lack of true round-robin certification of the blanks, in Snowden's opinion the blank samples could not be considered to actually be void of mineralization, making it impossible to determine whether blank sample failures were a result of contamination or background silver grade. Thus, Snowden judged the blanks to be unreliable for determining whether contamination was an issue in the sample preparation and analytical laboratories. However, Snowden did recognize an important point – if the blank failures were totally due to contamination, their magnitude did not indicate "significant concern with sample contamination".

A discussion of Orko's QA/QC duplicate samples can be found in Section 12 of this Technical Report.

Orko completed two drillholes in 2011 and three drillholes in 2012 for a total of 500 meters. Only 29.2 meters of this drilling was sampled and assayed, according to the Orko database. Therefore no record of QA/QC procedures and results exists for these drill campaigns.

11.5.3 PAS QA/QC (2008-2010)

The descriptions and discussions of results presented in this section rely on those provided by Snowden in its 2011 Technical Report prepared for Orko and PAS (Snowden, 2011a).

11.5.3.1 PAS QA/QC Procedures

Relative to the Orko QA/QC program, the PAS QA/QC procedures provided for a much more systematic insertion of blanks, standards, and pulp duplicates into the batches of drill core samples. PAS batches consisted of 50 samples submitted to the SGS laboratory, within which sample numbers ending with 10 or 60 were silver standards, sample numbers ending in 30 or 80 were gold standards, and sample numbers ending in 20, 40, 70, or 90 were blanks. Duplicate samples were collected by the laboratory by taking a pulp duplicate split of every sample represented by sample numbers in the sequence ending with 49 or 99, and these pulp duplicates were subsequently assigned the next sample numbers (ending in 50 and 00, respectively), which had been reserved in the sample numbering sequence by the geologists. These duplicates were subsequently submitted in separate batches of pulps to an umpire (secondary) laboratory.

The blank samples used by PAS consisted of half-core basalt. A total of 652 blank samples were inserted as described in the previous paragraph, resulting in a 4% insertion rate. A total of 662 standards were inserted as described, resulting in a similar 4% insertion rate. PAS used three different standards summarized as follows, with expected values shown in parenthesis - Orko-10 (145.47 g/t Ag, 0.057 g/t Au), GBM908-13 (151.4 g/t Ag), and G308-7 (0.27 g/t Au). Standards GBM908-13 and G308-7 were commercial standards purchased from Geostats Pty. Ltd. Standard GBM908-13 was a base metal standard that was also certified for silver and sulfur, while standard G308-7 was certified for gold only. Standard Orko-10 was the custom standard described earlier in this Report. Snowden confirmed that Standard Orko-10 was prepared by SGS in Durango from material obtained from stockpiles at the Project site, and noted that although this standard was round-robin tested in five independent laboratories from which expected values for gold and silver were derived, it was never officially certified. A total of only 21 samples from this standard were inserted in sample

batches, all of which were from drillholes BP09-355 to BP09-364. PAS commissioned SGS Peru to evaluate the Orko-10 standard material, and SGS Peru concluded that this standard had unacceptably high variances, probably due to the presence of native silver in the sample. As a result, PAS discontinued its use of standard Orko-10 beginning with hole BP09-365 and thereafter.

Snowden reported that PAS geologists regularly monitored the performance of standard and blank assays received from SGS by plotting values on line graphs in Excel as soon as each batch of assays was reported by the laboratory. Whenever any of the standard results exceeded three standard deviations from the expected value, the entire batch of assays was re-submitted for analysis.

11.5.3.2 PAS QA/QC Results

Snowden reported failures for only two gold standards and five silver standards which represent failure rates of 0.3% for gold, and 0.8% for silver. These results indicate that laboratory contamination was not a material concern for the samples from the PAS drilling campaigns.

The results for both silver and gold from certified commercial standards GBM908-13 and G308-7 displayed a bias on the high side of the expected values. Although the results for GBM908-13 fell within acceptable limits, a consistent high bias relative to the expected mean was present. Results for standard G308-7 displayed a very slight bias towards the under-reporting of the gold grades. However, all results were within two standard deviations of the certified expected values, indicating acceptable levels of laboratory accuracy.

11.5.4 Coeur QA/QC Program (2013-2014)

Coeur maintained a QA/QC program that was structured on guidelines set forth in the written company QA/QC policy. QA/QC consisted of routine insertion of standards, blanks and duplicates into the primary sample stream for both RC and core samples. Umpire check assays have been commissioned in 2014. Table 11-3 defines the suggested Test % for QA/QC sample insertion based upon the total count of primary samples.

Table 11-3: Coeur Development Program QA/QC Recommendations

	Primary Lab Control					External Control Samples				
Sample Type	Duplicates			Standards	Blanks	Pulps	Rejects	Standards	Blanks	Duplicates
Duplicate Type	Sample	Prep	Analytical							
Suggested Test %	2.5%	2.5%	2.5%	5.0%	5.0%	10.0%	1.0%	1.0%	1.0%	0.5%

11.5.4.1 Certified Standards and Blanks

The 2013 and 2014 assay campaigns utilized five certified commercial standards and one round robin tested standard. The campaigns used one certified blank and one round robin tested blank. The certified standards were purchased from CDN Resource Laboratories Ltd., in Langley, B.C., Canada and SGS de Mexico, in Durango, Mexico. The blank was purchased from Rocklabs, in Auckland, New Zealand. Table 11-4 lists the standards and blank and their certified silver and gold values.

Table 11-4: Coeur Certified Standards and Blanks

Certified Standards and Blanks				
Standard ID	**Certifying Lab**	**Element**	**Standard Value (ppm)**	**1 Standard Deviation (ppm)**
CDN-ME-1101	CDN	Ag	68	2.3
CDN-ME-1101	CDN	Au	0.564	0.28
CDN-ME-1205	CDN	Ag	25.6	1.2
CDN-ME-1205	CDN	Au	2.20	0.14
CDN-ME-1303	CDN	Ag	152	10
CDN-ME-1303	CDN	Au	0.924	0.1
CDN-ME-1304	CDN	Ag	34.0	1.6
CDN-ME-1304	CDN	Au	1.8	0.06
HGRS-02	SGS	Ag	98.7	4.0
HGRS-02	SGS	Au	0.01	0.003
ORKO-10	SGS (round robin)	Ag	145.466	4.227
ORKO-10	SGS (round robin)	Au	0.057	0.005
AuBlank58	RockLabs	Au	< 0.002	NA
AuBlank58	RockLabs	Ag	NA	NA
BLANK-5	Acme, ALS, SGS (round robin)	Ag	<5	NA
BLANK-5	Acme, ALS, SGS (round robin)	Au	<0.004	NA

QA/QC comparison analyses are completed in the acQuire database using separate tools for blanks, standards, and duplicates. Performance of the standards was tracked over time and against lower and upper cut-off limits. Run plots were generated, with multiple user controlled options. Plots generated for this report include the assay value plotted against the certificate number, which depicts the standard's performance over time. The plots also contain error lines indicating the acceptable minimum and maximum values for the given standard and assay method. Coeur policy recognizes QA/QC failures as ± 3 standard deviations for standards and ± 5 times the lower detection of the assay method for blanks.

When a blank or standard fails QA/QC it is moved to a "rejected" status in acQuire, along with all primary, duplicate, and lab QA/QC samples above and below the failure, and up to the next or previous passing blank or standard. This partial batch of samples must be re-run at the original laboratory, and with the same analysis method as the failed method. In the case of the Project, the QA/QC focuses on silver and gold, although the primary analysis includes multi-element ICP. Coeur does not re-run the multi-element ICP on failed sample batches. If the blank or standard fails a second re-assay the entire batch must be analyzed a third time at a second commercial laboratory, using an analysis method similar to that of the original test work.

11.5.4.2 Coeur QA/QC Results

11.5.4.2.1 Blanks and Standards

In 2013-2014, Coeur submitted 21,991 primary samples for assay at two commercial laboratories. 1,018 blanks and 1,369 standards where inserted into the sample streams, representing insertion rates of 4.63% and 6.23%, respectively. The combined insertion rates exceed the total of 10% standards and blanks suggested by Coeur protocol and included in Table 11-3. All standards and blanks were analyzed at ALS in Vancouver, BC, CA and/or SGS in Lakefield, ON, CA. Table 11-5 is an actual example of a QA/QC report exported directly from the acQuire database. The report tables the statistical performance of all standards and blanks, defined the assaying laboratory and by the analytical method utilized. The column # *Outside Limit* is the count of failed standard or blanks.

Table 11-5: acQuire Standards QA/QC Report, ALS Laboratory

Assay Field	Std ID	# of Analyses	# Outside Limit	% Outside Limit	Mean	Median	Min	Max	Standard Deviation	% Rel Std Dev	Standard Error	% Rel Std Err	Total Bias
Ag_Ag_GRA21_ppm	ORKO-10	6	0	0	134.33	136	120	141	6.97	5.19	2.84	2.12	0
Ag_Ag_OG46_ppm	ORKO-10	6	0	0	137	135.5	120	161	12.18	8.89	4.97	3.63	0
Ag_Ag_OG62_ppm	ORKO-10	137	38	27.74	145.75	143	124	185	11.47	7.87	0.98	0.67	0
Ag_Ag_OG62_ppm	HGRS-02	238	1	0.42	101.95	102	96	141	3.53	3.46	0.23	0.22	0.03
Ag_ME_ICP61_ppm	HGRS-02	282	13	4.61	95.13	97.55	36.1	100	10.69	11.24	0.64	0.67	-0.04
Ag_ME_ICP61_ppm	CDN-ME-1101	285	0	0	68.74	68.9	59.5	78.7	2.75	4	0.16	0.24	0.01
Ag_ME_ICP61_ppm	CDN-ME-1205	172	0	0	26.39	26.3	23.8	29.8	1.15	4.36	0.09	0.33	0.03
Ag_ME_ICP61_ppm	CDN-ME-1304	225	3	1.33	34.98	34.8	20.8	65.1	3.04	8.7	0.2	0.58	0.03
Au_Au_GRA21_ppm	CDN-ME-1205	157	0	0	2.31	2.3	1.91	2.88	0.18	7.82	0.01	0.62	0.05
Au_Au_GRA21_ppm	CDN-ME-1304	2	2	100	2.5	2.5	2.24	2.75	0.26	10.22	0.18	7.23	0.39
Au_Au_ICP21_ppm	ORKO-10	143	0	0	0.06	0.06	0.05	0.07	0	5.16	0	0.43	0.02
Au_Au_ICP21_ppm	HGRS-02	508	18	3.54	0.01	0	0	1.81	0.08	822.12	0	36.48	-0.01
Au_Au_ICP21_ppm	CDN-ME-1101	285	0	0	0.58	0.58	0.49	0.68	0.03	5.77	0	0.34	0.03
Au_Au_ICP21_ppm	CDN-ME-1205	172	0	0	2.22	2.21	1.84	2.63	0.17	7.48	0.01	0.57	0.01
Au_Au_ICP21_ppm	CDN-ME-1304	225	3	1.33	1.78	1.81	0.01	2.1	0.21	11.75	0.01	0.78	-0.01

The manual insertion of standards and blanks includes an inherent risk for mislabeled or incorrectly inserted samples. Original standards and blanks failed QA/QC and the sample batches were reassayed. Further review was conducted on the failed standards and blanks by Coeur and Amec. The result of the review identified 189 standards and seven blanks that were likely assigned an incorrect standard ID or blank ID. These samples were reclassified based on the original assay results which produced silver and gold values that were associated with another active project standard or blank. The result of the reclassification was the approval of the Round 1 QA/QC for these samples.

Further review of the results of standard Orko-10 indicated a resulting bias to the overall QA/QC performance. The PEA indicated that the use of Orko-10 had been discontinued by PAS after a reevaluation of the standard resulted in unacceptably high variances (M3, 2013). The Orko-10 standard was inserted 143 times into the 2013-2014 campaign. The decision to use the standard was the result of recent performance indicating low standard deviations, based on 161 sample results; and its immediate availability to the Project. The standard failed 38 silver analyses, a 26.6% failure rate. The run plots in Figure 11-1 illustrates that the standard performed consistently when analyzed by Ag_OG62_ppm, by failing both above and below the standard limits, with locally larger magnitude failures above the maximum acceptable limit. In order to quantify these findings, Table 11-6 and Table 11-7 contain values in parentheses that indicate QA/QC results which exclude the entire Orko-10 sample set. Coeur has moved to discontinue the used of standard Orko-10, and dispose of any remaining sample material.

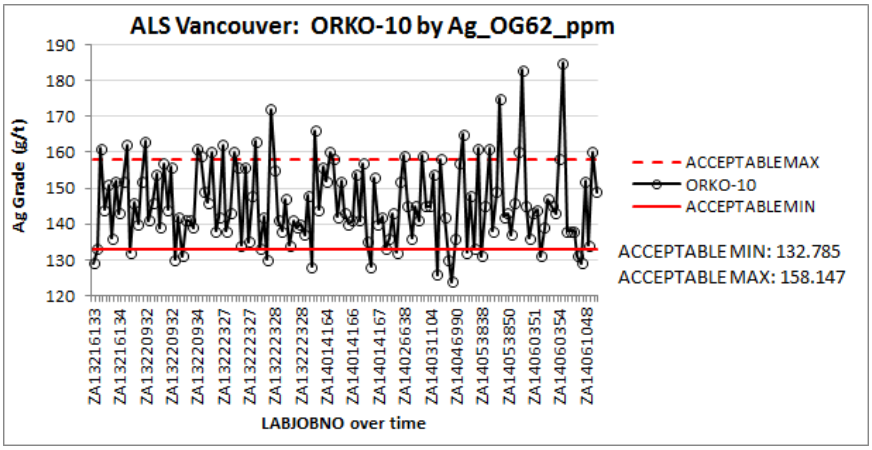


Figure 11-1: Orko-10 Silver Standard. Medium population, with multiple failures outside the acceptable minimum and maximum.

Table 11-6 contains a Round 1 summary of Coeur QAQC results for company inserted standards and blanks. Round 1 results are those submitted with the original sample batches for the 2013-2014 campaign. The Total Failure is the combined failure rate of standards and blanks with respect to the total submitted primary samples for the assay campaign. The values in parentheses in Table 11-6 indicate the resulting standard and failure count after all ORKO-10 standards are removed from the Round 1 QAQC statistics. The resulting recalculated failure rate is in parentheses. The Round 1 sample statistics indicate a low (<2%) failure rate for the Coeur campaign after removal of the ORKO-10 subset.

Table 11-6: Summary of Round 1 QA/QC Results

Group	Count / Rate
Primary Samples	21,991
Blanks	1,018
Blank Failures	16
Standards	1,369 (1,226)
Standard Failures	60 (22)
Failure Rate	3.27% (1.78%)

The run plot examples in this report (Figure 11-2 through Figure 11-5) illustrate various graphical representations of the performance of actual blanks and standards. The run plot demonstrates assay value on the y-axis, plotted against the LABJOBNO on the x-axis. The LABJOBNO on the x-axis is also a depiction of the standard performance over time.

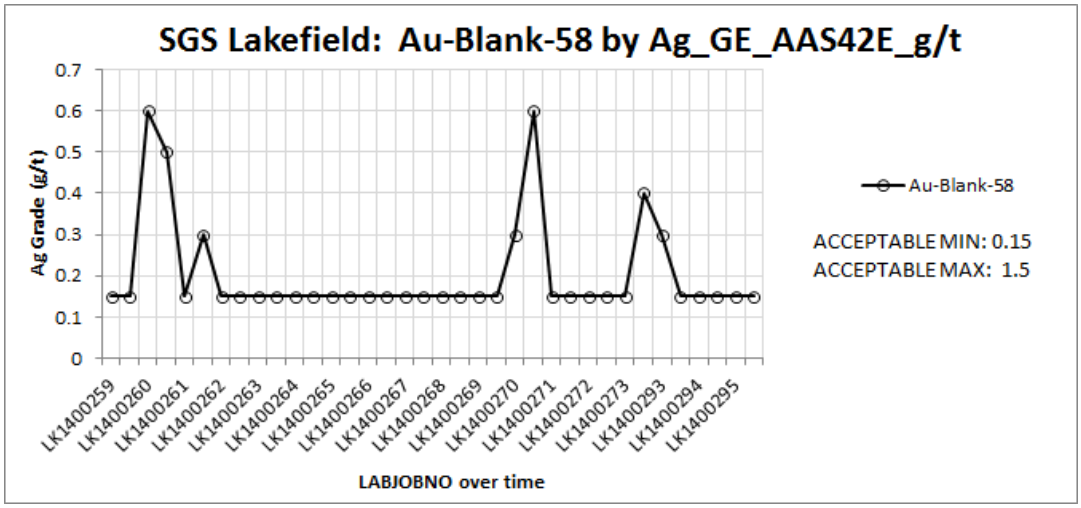


Figure 11-2: Silver blanks. Small population, zero failures. Baseline, or ACCEPTABLEMIN is ½ the LOWERDETECTION of the assay method.

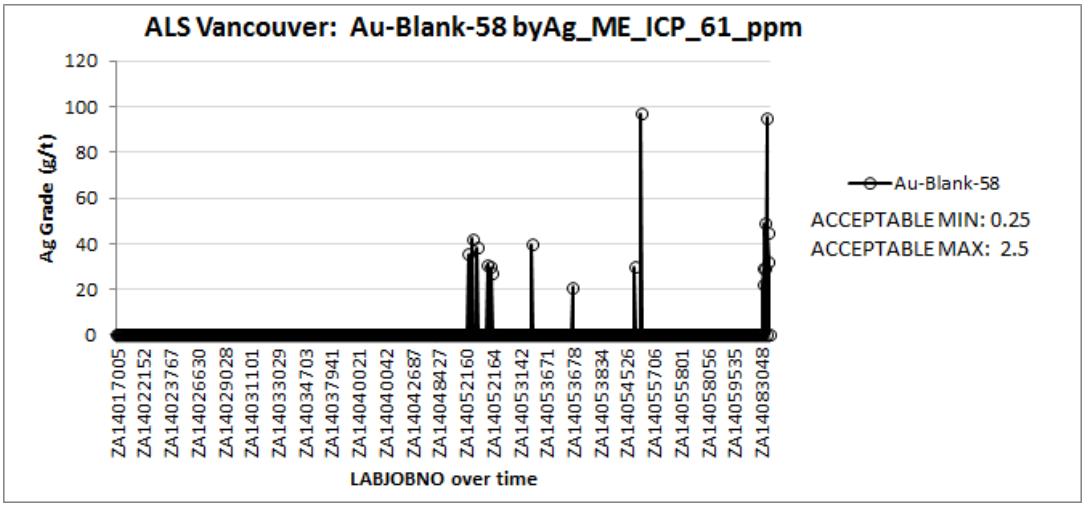


Figure 11-3: Silver blanks. Large population with multiple failures of large magnitude. Baseline, or ACCEPTABLEMIN is ½ the LOWERDETECTION of the assay method.

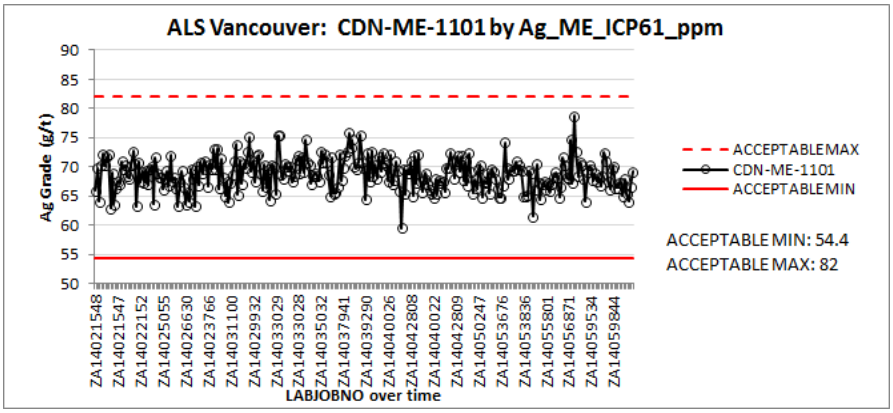


Figure 11-4: Silver standard. Large population with zero failures. Results trend along the true standard value.

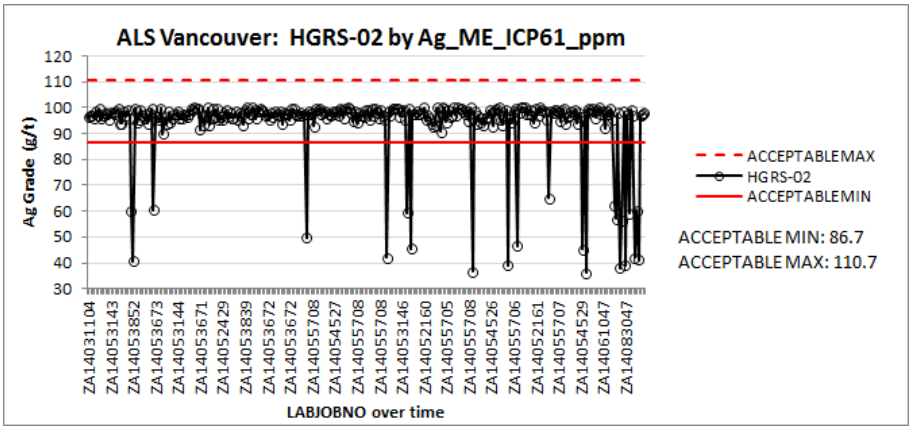


Figure 11-5: Silver standard. Large population with multiple failures of large magnitude below the ACCEPTABLEMIN.

A percentage of failures of Round 1 QA/QC were subjected to reassay at the same laboratory using the same assay method as the original sample. Table 11-7 summarizes the result and status of standards and blanks that have received a Round 2 analysis. Samples pending results of a Round 2 analysis are also tabled. At the time of this report, 11 blanks and 30 standards are pending results of the Round 2 analyses. As in Table 11-6, values in parentheses represent sample counts that exclude Orko-10 standards. All samples that failed Round 2 QA/QC, and their associated sample batches, will be analyzed at a second commercial laboratory with an analysis method equivalent to that used by ALS.

Table 11-7: Summary of Round 2 QA/QC Results

		Development Drilling
Blanks	Reassayed Round 2	6
	Passed Round 2	0
	Failed Round 2	5
	Pending Round 2 Results	11
Standards	Reassayed Round 2	30 (15)
	Passed Round 2	16 (3)
	Failed Round 2	14 (12)
	Pending Round 2 Results	28 (5)
	Total Percent Passing Round 2	44% (14%)

All QA/QC samples that have not successfully passed Coeur's QA/QC procedures remain in a "rejected" status in the acQuire database. Additionally all primary samples associated with these failed control samples remain in a "rejected" status and are unavailable for use in the resource evaluation. When a control sample passes QA/QC, the new primary assay value of the associated primary samples will be moved to an "accepted" status in the acQuire database.

11.5.4.2.2 Duplicates

In 2013-2014 Coeur submitted 15,107 new primary samples to ALS Global. These samples are subjected to the company duplicate QA/QC protocol, included in Table 11-3. The protocol suggests an equal distribution of duplicate samples at various check stages. These include the sample (S), prep (C), and analytical (P) stages. Coeur submitted 1,347 duplicates resulting in an insertion rate of 8.92% which exceeded the total suggested insertion rate of 7.5%. Check stage totals and resulting failures are included in Table 11-8. The *Threshold* is a dynamic value that eliminates sample pairs from the population based on their assay value's proximity to the lower detection level of the method used. For this exercise the *Threshold* is equal to 10 times the lower detection of the method. Any primary sample with an assay value less than this is considered below the *Threshold* and is removed from the analysis. Failed duplicates will be reassayed in 2014 with the primary sample and all associated duplicates per Coeur QA/QC protocol.

Table 11-8: Duplicate Sample Summary

Check Stage	Sample Type	Count	Count above Threshold	% Acceptable Difference	Failed Duplicates	% Failure
Sample	(S) Sample	661	236	NA	NA	NA
Prep	(C) Crush	343	173	25%	14	8.1%
Analytical	(P) Pulp	343	172	20%	18	10.5%
Eligible Primary Samples		**Duplicate Samples**			**Insertion rate**	
15,107		1,347			8.92%	

Figure 11-6 is a scatter plot of the primary sample grade on the x-axis and the duplicate sample grade on the y-axis. The plot is segregated by check stage. The plot illustrates R^2 values for each check stage that indicate moderate correlation between the sample duplicates (S) and excellent correlation between the analytical (P) duplicates.

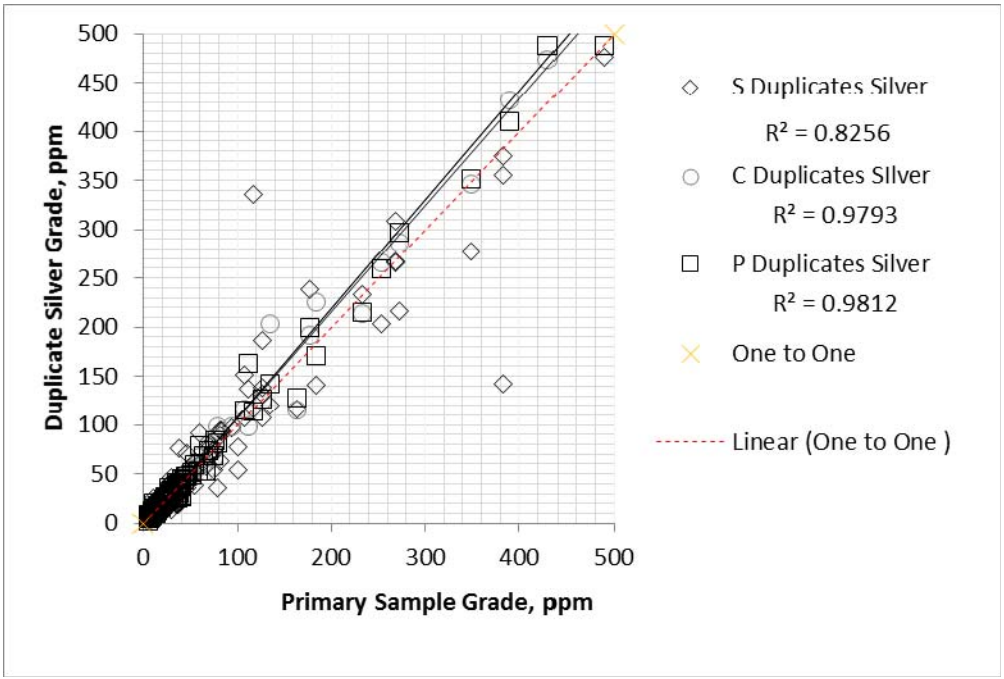


Figure 11-6: Scatter Plot of Primary vs Duplicate Values by Check Stage

11.5.4.2.3 Umpire Assays

Umpire assays for the 2013-2014 sample campaigns are currently in progress. Per Coeur policy, a random selection of pulps chosen throughout the range of grades should be selected from each assay certificate from the primary lab and sent to another laboratory for check analysis; using the same analytical digestion method and instrumental finish.

11.5.5 Opinions and Recommendations of the Qualified Person

In the opinion of the QP responsible for this section of this Report, the sample collection procedures, sample preparation, sample analysis procedures, and chain of custody procedures were adequate and acceptable for operators before Coeur. The QP acknowledges the issues related to the Orko blanks and standards, but in the QP's opinion these do not constitute a material issue. The QP does note that the insertion rates for blanks and to a lesser degree for standards was below the 5% of total samples rate that is generally accepted in North American and Australasian jurisdictions. However, based on the very low number of blank failures, in the opinion of the QP this is not a material issue.

The QP acknowledges that Coeur was successful in addressing the following recommendations presented in the PEA.

- The deficiency in standard insertion rates exists in pre-Coeur campaigns. The 2013-2014 Coeur campaign met or exceeded the generally accepted 5% insertion rate.
- A new set of commercial standards and blanks were purchased, addressing the criteria of the necessary variable grades.

The QP acknowledges Coeur has achieved insertion rates for quality control samples consistent with Coeur and industry requirements. The QP acknowledges that Coeur has completed or is in the process of completing QAQC procedures on all assay campaigns, with methods consistent with those defined in company protocol. These include accepting and rejecting assays based on standard and blank performance, reassaying of failed sample batches, and the completion of umpire lab check analyses. The QP does note that conditions were identified with quality control

samples that initially affected the integrity of the assay results, and also that these conditions have been addressed and corrected. The following are recommendations for work to be completed by Coeur.

- Continue and complete the QA/QC review of Orko-era SGS assay data. This includes identification of unknown standards and blanks; and identification of missing samples located only on laboratory data files.
- Complete and finalize the standard, blank, and duplicate QA/QC review for 2013-2014 drill data. This includes the secondary and tertiary rounds for failed standards and blanks; and their associated sample batches.
- Complete a Coeur compliant check assay campaign for the 2013-2014 Coeur development and exploration sampling campaigns at an accredited umpire laboratory. (Initiated July 2014)

12 DATA VERIFICATION

The Project drillhole database has been validated by the Coeur technical services group. It has been verified and deemed appropriate for resource modeling. A review and validation of the 2013-2014 assay, collar coordinate, downhole survey, and assay data has been performed by Coeur.

The historic drillhole database has been verified by Orko (pre-2009), MDA (2009), PAS (2008-2010), Snowden (2011), Mining Plus (2012), and IMC (2013). The following section 12.1 is taken from the PEA compiled by M3.

12.1 HISTORIC DATA VERIFICATION PROCEDURES

12.1.1 Orko – Pre-2009

Prior to 2009, Orko reportedly sent 331 duplicate sample pulps from five of its drillholes to ALS Global (formerly ALS Chemex) in North Vancouver, B.C., Canada, as a check against Inspectorate's primary assay results for these holes. Although the analytical methods used by ALS Global (ALS) for silver for some of the check samples reportedly differed slightly from those methods used by Inspectorate, the results for these check samples indicated a only slight high bias on the part of Inspectorate for silver grades less than 30 g/t Ag, and a corresponding very slight high bias across the board for Inspectorate gold assays, as determined using fire assay with gravimetric finish methods. The QP responsible for this section of the Report reviewed scatterplots of 317 of these 331 gold and silver check analyses provided in earlier Technical Reports (which did not provide reasons for the 14 check assays missing from scatterplot comparisons). Based on these reviews, in the opinion of the QP, these check assay data fall within acceptable limits (M3, 2013).

In addition to the 331 duplicate samples, Orko submitted coarse rejects to SGS for 134 samples from drillhole BP07-102 that were originally prepared and assayed by Inspectorate. SGS in turn prepared pulp duplicates for these samples that were subsequently submitted to ALS. Mining Plus, consultants to Orko and the Qualified Persons for the November 5, 2012 Technical Report on the Project, created Q-Q plots of assays for 120 of the 134 samples from the three labs that were available in the Mining Plus database. These plots indicated reasonable correlation with no biases between Inspectorate and SGS for gold or silver. The Q-Q plots for Inspectorate versus ALS showed similar correlations, but with an apparent slight high bias for silver in the Inspectorate assays. In the opinion of the QP responsible for this section of the Report, these comparisons are acceptable for both gold and silver between all three laboratories (M3, 2013).

12.1.2 Mine Development Associates (MDA) – 2009

As a follow-up to the pre-2009 check assay programs conducted by Orko, Mine Development Associates (MDA) in 2009 conducted a comprehensive check assay program that included pulp and coarse reject samples reportedly representing each of the mineralized vein intercepts. Submitted by MDA to ALS Chemex in Reno, Nevada, these checks samples consisted of 240 pulp rejects, of which 61 original pulps were assayed by SGS and 179 original pulps, which were assayed by Inspectorate. Q-Q plots of the results revealed acceptable correlation between the two primary laboratories (SGS and Inspectorate) and the secondary laboratory (ALS), with no indication of biases. In conjunction with this check assay effort, MDA inserted QA/QC blanks and gold and silver standard samples into the check assay batches at select but unequal intervals. The QA/QC results for the blank samples indicated no failures for silver and two failures (out of 10) for gold. All standard assays fell within the acceptable ranges (M3, 2013).

For determination of material density, Orko had routinely conducted one density measurement from each sample sent for assay, using a single piece of half-core removed from each sample and a water immersion method that resulted in a set of recorded densities that exceeded 88,000 in number. After each density analysis, the samples used were returned to the appropriate sample sacks for shipment to the laboratory for assay. Concerned that Orko's density determinations were possibly biased high because the method used did not account for the presence of vugs in the

vein samples, MDA had Orko complete an additional 92 density determinations using a dry analysis technique on whole core representing the Martha vein and other lesser veins from the deposit. In the opinion of the Qualified Persons responsible for this section of the Technical Report, this density validation testing generated specific gravity values that are not significantly different than the average specific gravities obtained by Orko's analyses (M3, 2013).

12.1.3 PAS – 2008-2010

The Snowden (September 2011) and Mining Plus (November 2012) Technical Reports both make reference to a suite of pulp duplicate samples representing the Martha vein from both earlier Orko and PAS drillholes. According to the Snowden Technical Report, "To eliminate any concerns about the quality of Orko data, PAS undertook a specific testing program of original data by reassaying drillhole samples and by comparing recent PAS drillhole sample grades with earlier Orko sample grades, which also showed grade biases". The Snowden Technical Report further states in Table 11.3 that the duplicate samples, which totaled 146 in number, were submitted "because of problems in correlating mineralization over short distances between Orko and Pan American holes". The Mining Plus Technical Report added that of the 146 duplicate pulps, 43 of the pulps were originally assayed by Inspectorate, while 103 were originally assayed by SGS. The duplicate assays for all 146 pulps were generated by SGS. However, neither report provides any details of the results of comparisons between the duplicate sample pairs (M3, 2013).

To follow up on MDA's validations of material density, PAS applied four different testing methods to the same individual rock samples removed from 252 different sample intervals in the remaining half core. These included 133 samples from veins and adjacent silicified material and 119 samples from un-silicified andesite, the most common host rock to the Project mineralization. The selection of each of the 252 samples considered the variable degree of oxidation in the deposit by taking approximately 40 samples of vein/silicified material and approximately 40 samples of andesite from shallow (highly oxidized), middle (moderately oxidized), and deep (weakly oxidized to unoxidized) portions of the deposit. One of the four measurement techniques employed included data for determination of a "void index" that could be used to derive bulk density. The selected samples (most of which were previously measured for specific gravity by Orko) weighed between 400 grams and 600 grams in order to reduce measurement error. Prior to testing, each sample was geologically described. The resulting specific gravity measurements reportedly were made by a technician in the metallurgical laboratory at PAS's La Colorado mine operations in Zacatecas, Mexico. The results of these measurements indicated that the Orko specific gravity data were suitable for use in Mineral Resource estimation, provided that a bulk density conversion factor of 0.99 was applied to the average specific gravity for each of the material types (vein, vein silicification, and various host rocks). In the opinion of the QP responsible for this section of the Report, this fine tuning of the measured specific gravities for the various material types is acceptable, but not material to estimation of Mineral Resources in the Project deposit (M3, 2013).

12.1.4 Snowden – 2011

The Snowden Technical Report (Snowden 2011a; 2011b) does not mention the collection by Snowden of any independent drill core samples or existing coarse reject or pulp duplicates for check assay. Snowden reportedly reviewed original assay certificates from the SGS laboratory in Durango, México and from Inspectorate's laboratory in Sparks, Nevada, which included 441 assays from PAS's drilling and 3,188 assays from Orko drillholes. In total, 44 errors were noted in the PAS database assay files (an error rate of 1.4%), 41 of which were determined to be related to a single assay batch that apparently was subsequently reassayed. Two of the three remaining errors reportedly involved the entry of incorrect assay detection limits. Nine errors were noted in the Orko assay database, seven of which were reportedly due to cases where the average values of acceptable duplicate assay pairs were entered rather than the primary sample assays.

Snowden did not perform or recommend additional material density (specific gravity) testing (M3, 2013).

12.1.5 Mining Plus – 2012

Mining Plus independently collected 74 samples consisting of 23 samples of half (sawn) core and 46 existing coarse rejects. Although the Mining Plus Technical Report states that, "*All results from this program returned values well within acceptable limits*", no actual data for the duplicate sample analyses were provided. Mining Plus also reportedly compared the results of 3,285 assays representing the major zones of mineralization reported on laboratory certificates with entries in the assay database and found six errors (an error rate of only 0.2%), and concluded that none of these errors were materially significant with respect to mineral resource estimation (M3, 2013).

Mining Plus checked collar coordinates for 17 drillholes using a hand-held GPS unit and noted no significant discrepancies (taking into account the accuracy of the GPS unit) with the values in the project database. Downhole survey data also were reviewed by Mining Plus, which found errors believed to have been caused either by inaccurate data transcription or by errors in the actual survey measurements. Where original data could be located, these downhole survey data were verified or corrected. Mining Plus also noted variances between drillhole azimuths and inclinations at the drillhole collars and the initial downhole survey data points. In Mining Plus's view, the reason for these discrepancies probably was due to the drill set-ups not corresponding to the planned azimuths and inclinations. To address these differences, Mining Plus reported that hole collar markers were removed and new collar measurements made of drillhole azimuths and inclinations, except where prevented by deterioration of the drillhole collars due to caving and/or the presence of the magnetic basalts on the eastern portion of the project drill pattern. Also, Mining Plus reported removing the downhole survey data for several drillholes for which it determined that the trace of these holes as defined by the downhole surveys was physically impossible. In summary, outside of the errors described above, Mining Plus noted that with the collection of downhole survey data on 50 m intervals, the potential impact of individual survey errors is limited.

Mining Plus did not perform or recommend additional material density (specific gravity) testing (M3, 2013).

12.1.6 IMC – 2013

Only five additional drillholes that were not included in the Mining Plus Mineral Resource estimate were considered for inclusion in the IMC Mineral Resource Estimate summarized in Section 14, (holes 1, 2, 3, 4 and 5) and only one of these has assay data (BP12-718). In the opinion of the QPs responsible for this Report no additional independent validations of these data were warranted. This opinion is based on the lack of any significant amount of additional data that post-dates the Mining Plus study and Technical Report, and in light of the documented acceptable agreement between duplicate sampling and assaying exercises and material density validations conducted by previous Qualified Persons (M3, MDA, Snowden and Mining Plus).

The QA/QC information that has been described in this section was loaded into the IMC system and analyzed to confirm previous work.

In summary, the primary basis for confirmation of the drillhole database are the inserted standards that have been completed during the assay programs. There were also a number of inserted blank samples. As described earlier, there are a total of 1,103 duplicate assays in the QA/QC database. These are a mixture of:

Duplicate Pulps to the same lab:	43 samples
Check Pulps to a second lab:	793 samples
Coarse Rejects to the same lab:	192 samples
Coarse Rejects to a second lab:	75 samples

The analysis by previous Qualified Persons and IMC do not indicate any particular issues with bias in the above data sets. However, none of the above programs are sufficiently consistent in procedure, data distribution, or purpose to

be considered as a major component of the QA/QC data. The above duplicate samples represent 1.1% of the total assay database.

The reliability of the entire drill program is therefore relying on the inserted standards and blank samples.

- Standards:

  There are 4,580 standards inserted into 104,720 assays (4.4%) of which 547 standards were inserted into the 12,955 assays above 25 g/t (4.2%).

- Blanks:

  There are 1,765 blanks out of 104,720 assays (1.7%) of which 348 blanks were inserted into the 12,955 assays above 25 g/t (2.7%).

Blanks were not inserted on a regular basis, although it appears that blanks were likely inserted after high-grade intercepts on a visual judgment basis rather than on a consistent insertion basis.

Standards were generally inserted throughout the database. The percentage inclusion is similar in both mineralized material and waste components of the deposit. IMC prepared maps and sections of the holes that contain standards and found them to cover the area of the mineral resource on a relatively consistent basis.

Analysis of tested standards versus the published Standard Value in the Mining Plus Report (MP Report) dated 5 November 2012 do not show any immediate issues regarding assay lab bias, regardless of the lab used for the primary assay. There are some confusing data points in the QA/QC database that present different certified values for some standards compared to the MP Report. IMC has chosen to use the certified values as published in the MP Report.

The statistical analysis of the QA/QC database indicates that the project database can be accepted for estimation of mineral resources, based almost entirely on the reliable results from inserted standards. However, the QA/QC database in general does not meet industry “best practices” in the opinion of IMC.

12.2 VALIDATION OF COEUR DRILL DATA

12.2.1 Assay Data Validation

Assay data are imported into an acQuire database using assay import object. The commercial laboratories provide assay data in a pre-constructed CSV template that imports seamlessly into the acQuire database, when no errors are present. AcQuire imports adhere to very strict rules relating to sample ID’s, assay data, and lab job numbers. When these rules are violated, an error report is generated. There is no manual data entry related to the assay import process. In addition, continuous comparison of the database values against the original PDF certificates is a valuable check on the database integrity. These checks should be conducted on an annual basis.

12.2.2 Geologic Data Validation

In 2014 Coeur initiated and completed a comprehensive review and data entry campaign for all drillhole geologic logs from Orko, PAS, and Coeur. The review included 843 drillholes for a total of 259,919 meters. The Project was contracted to HRC and included review and data entry of hardcopy and scanned geologic drill logs. HRC inspected four database tables; lithology, alteration, mineralization, and structure. A list of accepted logging codes was provided by Coeur with edits applied as needed as the Project proceeded. HRC supplemented the data process with an internal validation process that reviewed the logs for legibility, completeness, and consistency with regards to geologic interpretation. Select drillholes were reviewed in 3D to identify potential error in interpretation. Mechanical audits were completed to identify overlapping intervals, gaps in geology, and inconsistencies in drill depths.

12.2.3 Collar Survey Checks

Drillhole collars are checked visually in MineSight and on topographic based maps to confirm that holes are on the correct drill pads and map coordinates.

12.2.4 Downhole Survey Checks

Downhole survey data are imported into an acQuire database using an import object. The 2013-2104 downhole surveys were completed by IDS Mexico. IDS provided CSV data files that, in most cases, imported seamlessly into acQuire. On several occasions the CSV file was constructed from a different template, and additional manual formatting was required and completed by the geologic database manager. On import, the data are checked for overlapping intervals and intervals below the recorded length of the drillhole.

12.3 COEUR ONGOING VALIDATION OF HISTORICAL DRILLHOLE DATA

Coeur continued to verify Orko era data in 2013 and 2014. A 2013 report includes a review of the original Orko database and reviews subsets of the data loaded into the acQuire Database. This review included drillhole collar, downhole survey, density, RQD, and sample ID and assays. The review also outlines the general acQuire database structure. Recommendations are proposed and have been addressed by Coeur, or are included in this report as recommendations for further work.

Coeur identified the need to verify assay data against original hardcopy assay certificates. Coeur initiated this project in 2014. Verification of samples against hard copy assay certificates was conducted manually at both the Chihuahua Exploration office and the Chicago Corporate office. The initial comparisons checked 298 standards and 497 primary samples that were analyzed at Inspectorate Laboratory from 2006 through 2008. No discrepancies between the two datasets were identified. This amounts to verification of 2.9% of QAQC samples and 0.7% of primary samples.

The Orko and PAS assay results required verification for use in resource estimation. Further evaluation of the Orko master database was completed in 2014.

Inspectorate Laboratory provided official data files and certificates for 100% of the lab jobs completed from 2007-2009. The Inspectorate results were loaded to acQuire directly from the data files and are presumed validated, but will benefit from an additional check against the hardcopy certificate. The SGS Laboratory data files provided by Orko had been modified. Coeur was able to obtain 426 of 753 (57%) original data files and 634 of the original PDF certificates from the SGS Laboratory. The data files received were loaded directly into acQuire and the resulting data are considered validated.

QA/QC procedures were completed on the imported SGS data files, per Coeur's company protocol. A combined silver and gold total of 3,180 identifiable QA/QC values were analyzed in acQuire, resulting in 30 failures. Seven pairs of data were excluded from the analysis due to either the standard value exceeding the upper detection limit (UDL) of the assay method, or because the standard was not certified for a gravimetric analysis. Coeur policy mandates that a standard fails when the value exceeds ± 3 standard deviations (SD) of the standard value and a blank fails if it exceeds ± five times the lower detection limit (LDL) of the analysis method. Failure rates for silver and gold for from the SGS data files were 0.7% and 1.3%, respectively.

Assay pairs from the Orko master database and the SGS Laboratory data files were compared in acQuire to validate the primary assay database. A combination of 46,551 sample pairs of silver and gold were analyzed using x-y scatter plots. (Figure 12-1 and Figure 12-2) The silver and gold values should be exact matches and therefore a failure is defined as any deviation greater than zero. The comparison resulted in 510 failures, or 1.1% of the pairs. Figure 12-1 and Figure 12-2 visually indicate that the failures are of relatively low magnitude. Coeur is confident 54 of the silver failures are the result of Orko gravimetric results being merged into a fire assay field. These are not true failures, but

the result of manual data manipulation. The remaining silver and gold failures are assumed to be attributed to manual error or discrepancies with reporting of assays at or near the detection level of the assay method.

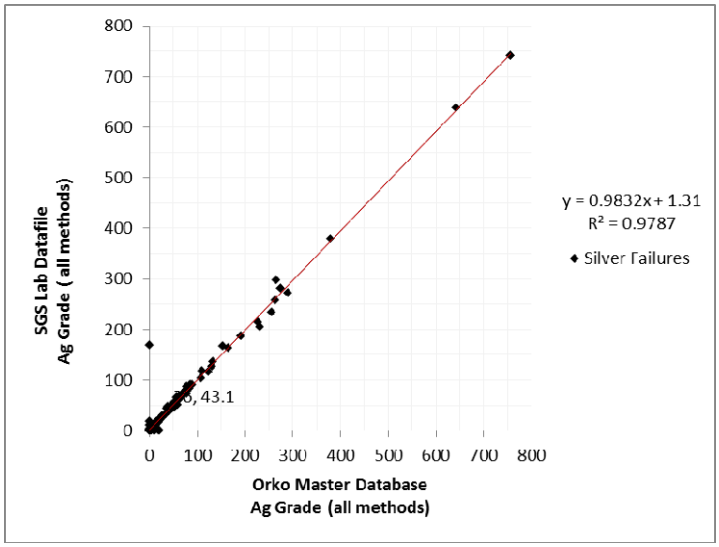


Figure 12-1: Scatter Plot of Failed Silver Pairs

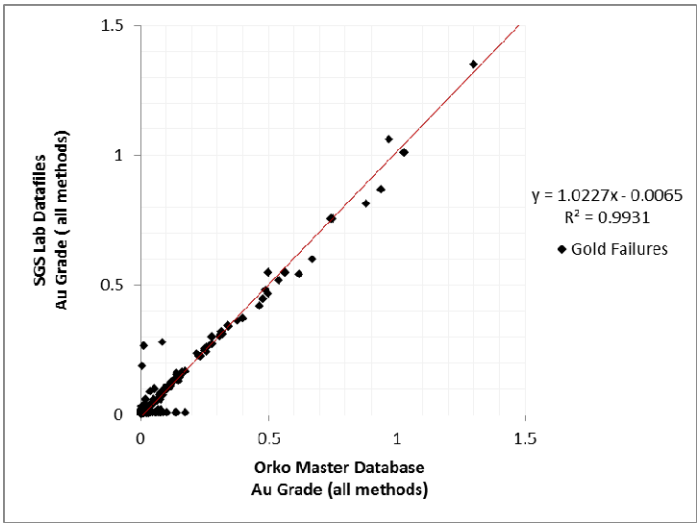


Figure 12-2: Scatter Plot of Failed Gold Pairs

The data and statistics presented in this section illustrate that Coeur was able to verify 60% of the primary SGS sample count and was able to verify a 43% increase in the QA/QC sample count. The QA/QC insertion rate of 12%, from the SGS data files indicated that the validated data contained an acceptable quantity of QA/QC samples which exceeded Coeur's current internal requirement. The QA/QC completed on the SGS data performed well, with low failure rates. The primary assay comparison produced very low failure rates and overall low magnitude failures. In summary, these comparisons represent 60% of the primary assay database and 57% of the total lab jobs stored in acQuire. The remainders of the outstanding original data files are considered to be unobtainable. Therefore, Coeur considered the performance of the Orko data that was verified by the SGS data files to be indicative of the performance of the entire Orko master database, and subsequently acceptable for use in resource evaluation.

12.4 COEUR RECOMMENDATIONS FOR FURTHER DATA VERIFICATION WORK

- Field check of drillhole collar coordinates for a random 10% of 2014 drillholes.
- Submit two batches of SGS samples to ALS in Vancouver to check for lab bias when analyzing samples with a 4-Acid ICP digestion. (Initiated July 2014).
- Completion of database to assay certificate verification of random silver and gold assay results from 2005 through 2014 assay certificates It is recommended that 5% of the total sample population for both primary and QA/QC samples be verified.
- Further review of 510 failures reported from the Orko and SGS database comparison.
- Further review and report on missing samples from the Orko master database. Develop an action plan to include, exclude, or re-define the samples.

12.5 COEUR DATA COLLECTION CAMPAIGNS

12.5.1 Density

The Orko master density database contained 89,226 records of calculated density, with an average density of 2.51 g/cm^3. In 2014, Coeur developed a standardized procedure to improve the accuracy of density measurements. In March 2014 Coeur employees completed 1,667 density measurements using the new procedure. The average density of these measurements was 2.52 g/cm^3. The 2014 density measurements were also coded to specific lithologies for use in the resource model shown in Table 12-1.

Table 12-1: Density Values Applied to Resource Model

Lithology	Density (g/cm^3)
Basalt	2.38
Metamorphic	2.67
Sedimentary	2.50
Volcanic	2.48

12.5.2 Geomechanical

In 2013 KP initiated a geomechanical drill program. The drilling consisted of two holes completed in 2013 and four holes completed in 2014. The drilling was completed by Major Drilling using HQ3 diameter core using triple tube techniques. The geomechanical core was oriented using the Reflex ACTIII device. The drill core was split and assayed using Coeur's procedures discussed in this report. The assay results for these drillholes are not included in the resource estimation data set.

12.5.3 Reassay of Pulps

Initial silver assaying by Orko at SGS used a 3-acid digestion with ICP-AES finish for all samples, followed by fire assays for samples exceeding 300 g/t Ag. In 2013, 313 pulp samples were reassayed and demonstrated that using a 4-acid digestion with an ICP finish resulted in a more complete sample digestion and, on average, higher Ag grades.

In 2014 Coeur selected all Orko era samples within the 25 - 100 ppm silver grade range for reassay by the 4-Acid ICP method. Coeur submitted 6,059 and 825 pulp samples to ALS and SGS laboratories, respectively. Reassay results from ALS (Figure 12-3) show the graphical representation of the average increase in Ag grade of 11 percent when comparing Coeur's 4-acid digestion ICP results to Orko's 3-acid digestion results. Reassay results from SGS (Figure 12-3 and Figure 12-4) show an average increase in Ag grade of 2.2 percent when comparing Coeur's 4-acid digestion ICP results to Orko's 3-acid digestion results.

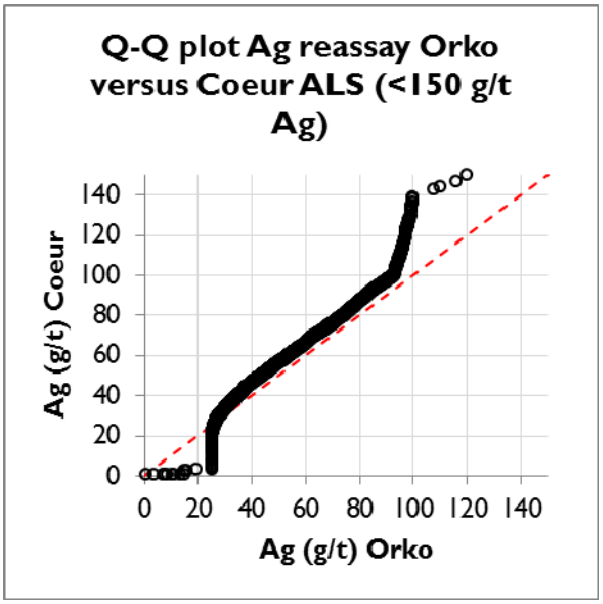


Figure 12-3: Q-Q Plots, Orko versus Coeur (ALS 4-Acid)

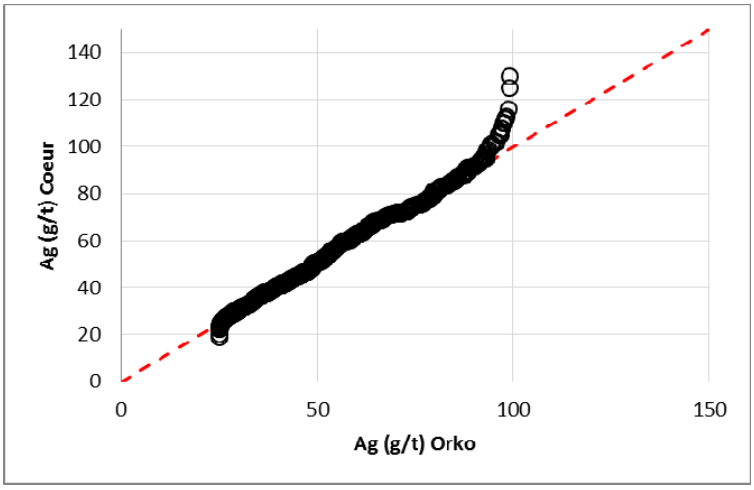


Figure 12-4: QQ Plot of Orko 3-Acid versus Coeur (SGS 4-Acid)

12.5.4 Orko Era Drill Core Sampling

Core drilled during the Orko and PAS campaigns was selectively sampled based on geologic observation during the core logging process. In 2013 Coeur initiated a campaign to re-log, sample, and assay portions of the previously un-split core. The program originated as a consequence of the identification of inconsistencies between the core data and the geologic interpretation in cross sections (Figure 12-5). During the review of the core and logs, Coeur geologists identified samples that ended in mineralized material as well as structures and alteration that were unsampled. The geologists collected these new samples from 35 drillholes along three parallel sections. The program resulted in 3,520 new primary samples added to the acQuire database.

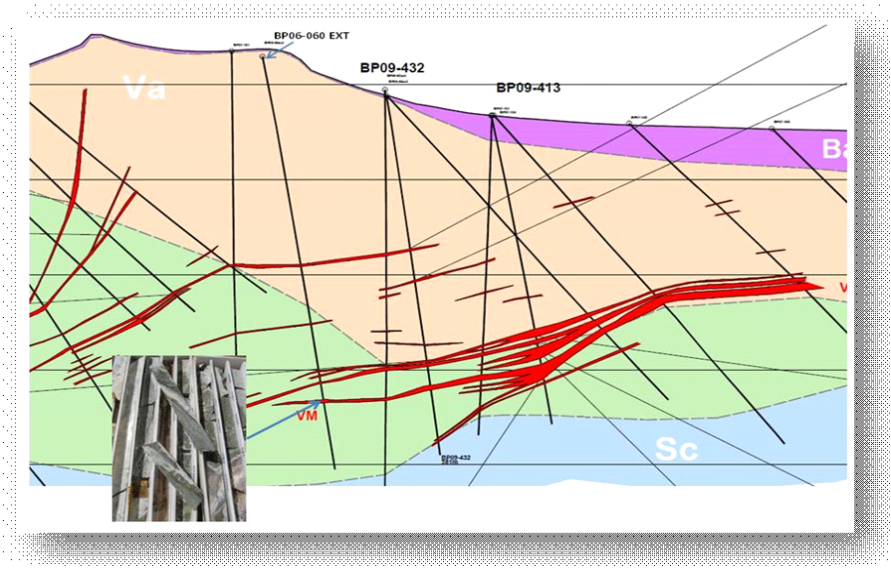


Figure 12-5: Example of the discrepancy between the orientation of structures as seen in the core and the interpretation in the cross section.

12.5.4.1 Orko Era Quartz Vein Sampling

In addition to the legacy drill core sampling, Coeur geologists reviewed the database to identify un-sampled intervals logged as quartz vein. A total of 95 samples, representing 120 meters of core length, were identified in the drillhole database. If the material was available and un-sampled, the vein interval and several meters of core above and below it would be sampled and assayed.

12.5.5 Spectroscopy Study

In 2014, Coeur contracted SRK Consulting of Toronto, Ontario to conduct an Infrared Absorption scan for alteration on the Project drill core. SRK scanned 76 drillholes for a combined 8,568 intervals totaling 24,987 meters. The spectral data identifies the presence or absence of clay species and is useful for geotechnical purposes such as identifying fault zones, zones of high cohesive strength, and depth to water. The spectral dataset is stored in the project acQuire database.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

Metallurgical studies for recovery of silver and gold from the Project have been completed. The deposit may be processed in a conventional crushing, grinding, and Merrill-Crow silver recovery circuit with detoxified tailings reporting to a tailings storage facility.

The following sections outline test programs completed to support the feasibility study. The feasibility process is presented in Figure 13-1.

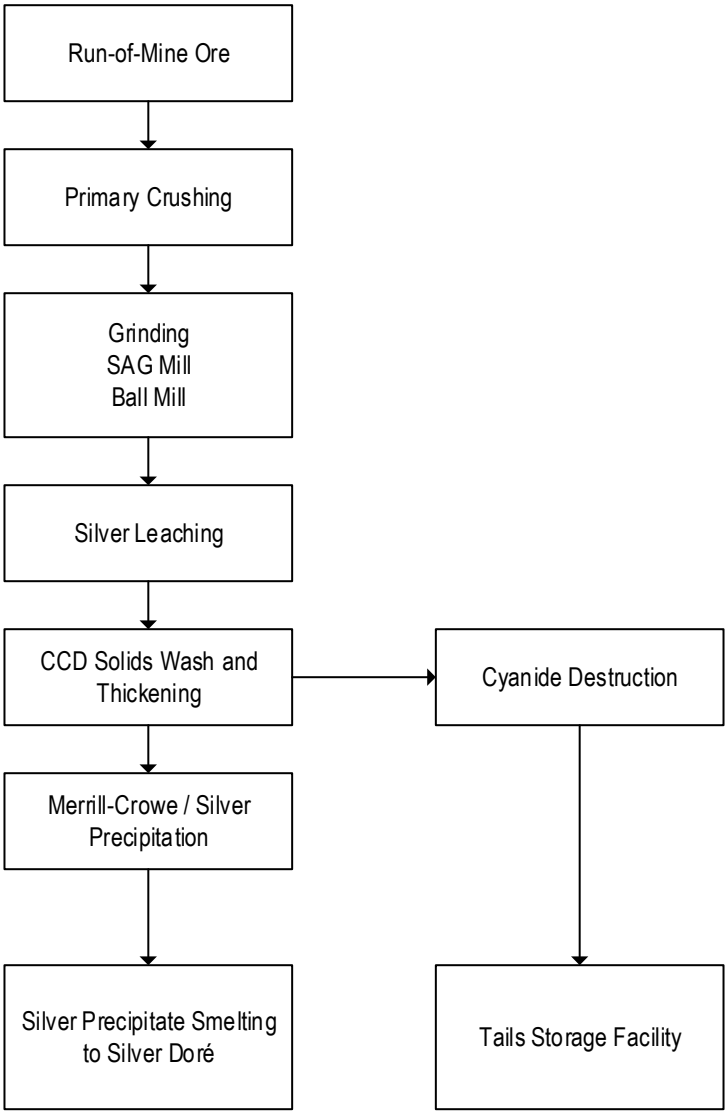


Figure 13-1: Project – Feasibility Study Process Block Diagram

13.2 PREVIOUS METALLURGICAL TEST PROGRAMS

Luismin initiated the first metallurgical test work on project samples in 1988. Process Research Associates (PRA, now Inspectorate) laboratory in Richmond, BC, and Westcoast Mineral Testing Inc. (WMT) of North Vancouver, BC

completed tests from 2007 and 2009. In the period from 2007 to 2010, a total of 865 samples from diamond drill core were used to prepare 44 metallurgical composites, including 28-variability composites. Orko and PAS, in 2009, shipped composites to the SGS metallurgical laboratory in Durango, Mexico. The composites were used to prepare a new master composite that was used in the first of four programs. A second master composite and variability composites were prepared for testing at SGS.

PAS initiated a test program at McClelland Laboratory, Inc. in 2012. The McClelland test program and results were reviewed in this study.

13.3 METALLURGICAL TEST PROGRAMS

Metallurgical development for the Feasibility study was completed at several laboratories and with third party consultants. The following laboratories or contractors contributed to the metallurgical program:

- ALS Metallurgy, Kamloops (ALS) - mineralogy, and flotation.
- FLSmidth - Bond crusher work index tests.
- Hazen Research Incorporated (Hazen) - sodium cyanide leaching, Merrill-Crow simulation, cyanide destruction, QEMSCAN analysis, clay analysis and comminution.
- McClelland Laboratories, Inc. (McClelland) program for PAS – XRD analysis and sodium cyanide leach optimization.
- Pocock Industrial, Inc. (Pocock) - flocculant screening, conventional, high-rate and paste thickening, vacuum and pressure filtration, pressure clarification, and slurry rheology.
- Spectral International, Inc. - near infrared analysis; identification of minerals, oxidation, and clays.

13.3.1 Hazen Metallurgical Program

The metallurgical program at Hazen achieve the following:

- Examined mineralogy; silver, copper, lead, and zinc deportment by QEMSCAN for Variability and Lithology Composites.
- Completed comminution testing; SAG Mill Comminution Testing (SMC), Bond abrasion testing (Ai), Bond rod mill work index (RWi), and Bond ball mill work index (BWi).
- Examined a Quick, diagnostic cyanide leach test to obtain data for correlation to bottle rolls.
- Validated leach parameters established in previous metallurgical work.
- Evaluated alternative slurry conditioning and leaching conditions.
- Completed sodium cyanide bottle roll leach testing of Variability composites and Lithology elevation composites.
- Completed sodium cyanide detoxification of leached slurry using SO_2/Air conditions.
- Completed sodium cyanide leach and detoxification of large bulk composites.
- Completed Merrill-Crowe precipitation of leach solution.
- Prepared composite samples for solid/liquid separation evaluation.
- Prepared composite samples for flotation evaluation.

13.3.2 McClelland Laboratories – PAS Metallurgical Composites

McClelland received 602 drill core intervals from PAS in January 2012. Thirty (30) individual composites and one Master composite were developed from core samples. Samples for comminution testing were provided; five drums of 1/2 core were blended, and five buckets of whole core were provided for crusher work index testing, CWi.

McClelland was requested to complete the final report, and ship remaining samples to Hazen. The following tasks were completed at Hazen:

- Master composite samples were subjected for a Quick Leach optimization test program.
- Whole core samples were submitted for Crusher work index testing, CWi.
- Comminution testing was completed for the five comminution composites.

13.3.3 Hazen Metallurgical Composites – Variability Composites

Coeur collected two composite groups from the Project site. The first group was identified as **Variability Composites;** open pit areas were divided into five different rock types; volcanic, sedimentary, quartz vein, metamorphic and intrusive. The composites were grouped by depth and two silver grade ranges.

The silver grades were:

- Low-Grade (LG), silver grade ranging from 30 to 55 gpt.
- Average-Grade (AVG), silver grade ranging from 55 to 120 gpt.

Fifty-three (53) composites were identified, and 1307 drill core intervals were selected, quartered, bagged at the Project site, and shipped to Hazen.

13.3.4 Hazen Metallurgical Composites – Lithology Composites

The second composite group, based on the Geological database in November 2013, were separated by **Lithology**; Quartz Vein, Volcanic, and Sedimentary. The composites covered the entire database range by combining the data into 10m or 20m elevation bench levels. The analysis resulted in the identification of 79 individual Lithology elevation composites; Quartz Vein – 26, Volcanic – 31, and Sedimentary – 22. A total of 2739 samples of drill core, representing 1967 m, were prepared at the Project site. The selected samples were logged, split to provide ¼ core sections, and shipped to Hazen.

Additionally,

- The weighted silver grade per elevation bench was determined. Samples were randomly selected to match the weighted grade.
- The mass of material collected was weighted by the distribution of samples, in the elevation, to provide a total composite weight of ~ 700-800 kg.
- Some elevation-benches were combined to reduce the number of composites.
- Intervals from previous metallurgical tests programs were omitted.
- An additional 20-30 kg of 1/2 core, for comminution testing, was collected. The elevation distribution of samples was used to weight the number of ½ core samples selected for comminution tests for each lithology.

13.4 METALLURGICAL HEAD ANALYSIS

The composites were subjected to different analytical techniques depending on the laboratory and the composite. Analyses included: head assay gold and silver – with a fire assay (AA finish), silver, copper, and zinc – with a 4-acid digestion/A.A. finish, cyanide shake test to determine soluble gold, silver, and copper, multi-element ICP analysis, mercury (cold vapor/A.A), sulfur speciation analysis, XRD analysis, semi quantitative x-ray fluorescence (XRF) whole rock analysis, QEMSCAN analysis, clay analysis, and Cation-Exchange–Capacity (CEC).

13.5 MINERALOGY

13.5.1 Historical Mineralogy Investigations

Mineralogy was reviewed by M3 in the 2013 PEA. The following summarized points are from the various mineralogical programs:

- Mineralogy work identified the various contained minerals including the metallic minerals, but was of limited use since there was no quantitative data.
- Gold was not identified, as expected, because of the low concentration of that metal.
- Silver minerals in the Martha Oxide zone were identified as argentite (Ag_2S) and stromeyerite (AgCuS), the latter of which was the sole copper mineral.
- In the Martha Sulfide zone, the primary silver mineral was argentite.
- In the Abundancia composite the silver minerals were bromo-argentite (AgBr), argentite, and anglesite (a lead sulfate mineral with less than 9% Ag in solid solution). The various silver minerals range in size from 15 to 135 microns.

13.5.2 Feasibility Study Mineralogy

The main gangue minerals identified were quartz and feldspar, predominantly K-feldspar. Quartz had the highest concentration in the Quartz Vein composite, and feldspar was most prominent in the Volcanic composite. The Sedimentary composite exhibited the highest carbonate concentration, predominantly calcite. Other gangue minerals were the micas; (biotite and muscovite), clinochlore and chlorite–smectite clays, barite, iron oxide or iron hydroxide, titanium oxide, and trace kaolinite.

Silver was found to be present in a wide variety of silver sulfide minerals. The Quartz Vein had the highest detected occurrences of liberated silver and silver sulfide minerals, in binary association, with sulfide minerals, compared to the other two composites. The Volcanic and Sedimentary composites had a higher percentage of detected minerals in association with gangue minerals.

In the Quartz Vein Lithology composite, the major portion of the silver occurred as silver sulfide, acanthite, Ag_2S. Traces of silver–selenium sulfide, possibly aguilarite, and complex Ag–Sb–Zn sulfide were also observed. The largest silver-bearing grain observed was about 100 µm in size. The silver-bearing grains observed were locked in quartz or were associated with pyrite, galena and lead carbonate, and iron oxide or iron hydroxide. Acanthite also occurred as very fine veins within quartz.

In Volcanic and Sedimentary composites, the major portion of the silver occurred as silver sulfide, acanthite, Ag_2S. Traces of silver–copper sulfide, possibly stromeyerite, were also observed. The largest acanthite grain observed was about 60 by 15 µm in size. The silver-bearing grains were associated with manganese-rich carbonate, quartz, and sphalerite, or other sulfides.

Due to the low head grade of gold in the samples, gravity concentrations were conducted via a Knelson Concentrator to increase the likelihood of finding occurrences. The majority of the located gold occurrences in the samples would be considered liberated, these liberated particles were in general larger in diameter. Gold inclusions were found within the pyrite or gangue particles, and may not be amenable to cyanide leaching. Gold was detected in association with silver sulfide minerals.

In the Quartz Vein, the average gold concentration was measured at 0.3 g/t, two gold grains, less than 1 µm in size, were observed. The gold grains were contained in iron hydroxide, enclosed in quartz-rich particles.

Pyrite was the most abundant sulfide, followed by sphalerite and galena. The total sulfide concentration was highest in the Sedimentary Composite.

Lead occurred as galena and was also observed as manganese-rich barium–lead hydroxides, as lead carbonate, cerussite, and as a lead–zinc–iron–copper–vanadium oxide.

Zinc was observed in sphalerite, but also occurred in oxidized zinc phases; zinc-bearing clay, zinc silicate, and zinc-bearing carbonate. Low levels of zinc were also observed in the silicate clinochlore and iron hydroxide.

The clay analysis indicated the samples contained clinochlore proper, a non-swelling mineral, and two mixed-layer clinochlore-smectite clays, one of which has a higher portion of smectite layers. Quantitative mineral abundance analysis using QEMSCAN indicated the total concentration of clinochlore and mixed chlorite-smectite clay for the Quartz Vein composites ranged from 1.3–2.2%, for the Volcanic composites 4.5-6.6%, and for Sedimentary composites 4.2-4.4%.

Near infrared analysis of core samples indicated that the oxidation state of the sample did not correlate with silver dissolution.

13.6 COMMINUTION PARAMETERS

Three previous comminution Bond ball mill work index (BWi) tests were completed indicating work indices of 12.2, 14.1, and 16.1 kWh/t for three composites. The Bond abrasion index ranged from 0.721-0.764 g.

Comminution testing for this feasibility study consisted of: Bond crusher work Index (CWi), Bond abrasion index (Ai), Bond rod mill work index, (RWi), Bond ball mill work index (BWi), and Semi-autogenous grinding mill comminution testing (SMC).

The Bond crusher work index values were measured from 65 whole core samples obtained from the McClelland-PAS test program. The samples were split by lithology and depth and submitted to FLSmidth for analysis. The Bond crusher work index averaged 9.7 kWh/t ± 3.4 and varied from 21.3 to 5.2 kWh/t for all samples tested.

The Bond Ai tests were completed on 47 samples. Bond abrasion index, Ai, ranged from 0.37-1.27 g, and averaged 0.74 ± 0.25g.

The Bond BWi tests were completed on 47 samples. Bond ball mill work index, BWi, ranged from 14.7–18.2 kWh/t and averaged 16.1 ± 1.0 kW/t.

The Bond RWi tests were completed on 36 samples. Bond roll mill work index, RWi, ranged from 12.7–18.1 kWh/t and averaged 15.3 ± 1.4 kW/t.

Sag mill comminution tests were completed on eight ½-core composites.

13.7 FLOTATION

ALS Metallurgy evaluated flotation as part of feasibility study. Sequential lead-zinc separation and bulk flowsheets yielded poor silver and gold recoveries when compared to whole ore cyanidation; the test program was suspended.

13.8 SODIUM CYANIDE LEACHING

In this study, McClelland completed a report for a test program initiated by PAS in 2012. Hazen completed five leach programs; Quick Leach study, Development tests, Variability study, Exploratory tests, and Bulk leach tests (2014d).

13.8.1 McClelland Test Results - 2012

Master composite head analysis indicated silver and gold concentrations of 175 gpt and 0.30 gpt, respectively. Silver recoveries ranged from 80.3% to 87.7%. Gold recoveries ranged from 74.1 to 79.3%. Sodium cyanide consumption ranged from 0.16 – 0.91 kg NaCN/t. Lime consumption ranged from 0.3-1.3 kg CaO/t. Leach kinetics indicated silver leaching was increasing after 96 hours, and gold leaching was complete after 50 hours.

Optimum leach parameters suggest a grind particle size P_{80} 63 microns, initial cyanide concentration of 3 gpl, pH of 10.3, pulp density was inconclusive, and lead nitrate addition of 30 gpt.

13.8.2 Hazen Quick Leach Development

Quick leach development tests evaluated initial sodium cyanide concentration, particle size, leach time, leach temperature, and lead nitrate addition.

The conditions selected for the variability quick leach tests were a particle size P_{80} of 9 µm, an initial cyanide concentration of 4 gpl NaCN, ambient temperature, residence time of 2 hours, 0 lead nitrate, pH 11.5, and a pulp density of 20% solids.

Material was ground to a particle size P_{80} 9.4 -11.2 µm. Average silver head grades silver ranged from 71.9-161.5 gpt. The average residues ranged from 16.6-29.5 gpt. Silver recovery ranged from 57.5%-89.8% and averaged 74.6%-77.1%. Sodium cyanide consumption averaged 0.5-0.9 kg/t. Lime consumption averaged 1.8-2.4 kg/t. The data indicate a slightly increasing tail grade and recovery with increasing head grade.

Quick leach results did not correlate with bottle roll recoveries.

13.8.3 Hazen – Variability, Lithology, and Exploratory Bottle Roll Results

The variability study consisted of 141 bottle roll tests with samples ground to a particle size P_{80} of ~45 µm and 23 bottle roll tests with the samples ground to a particle size P_{80} of ~74 µm. The ground materials were leached for 72 hours, at a pulp density of 40% solids, in 1.5 gpl NaCN, and at a pH of 11.5. Lead nitrate was not added.

The Exploratory cyanide leach series investigated various alternative leaching schemes to determine silver and gold recoveries, or improve metal leaching kinetics. The Exploratory leach tests included feed and residue size classification, baseline cyanide leaching, two-stage leaching, hot leaching, leaching of sand and slime size fractions, O_2 enriched leaching, and pressure conditioning and pressure leaching.

Quartz Vein Variability and Lithology composite bottle roll tests indicate silver recovery ranged from 67.7% to 92.8%. Average silver dissolutions in Low-Grade Variability, Average-Grade Variability, and Lithology elevation composites were 74%, 85% and 87%, respectively. Gold recovery ranged from 41% to 100%. Average gold dissolutions in Low-Grade Variability, Average-Grade Variability, and Lithology elevation composites were 48%, 53%, and 66%, respectively.

Volcanic Variability and Lithology composite bottle roll tests indicate silver recovery ranged from 54% to 89%. Average silver dissolutions in Low-Grade Variability, Average Grade Variability, and Lithology elevation composites were 66%, 78%, and 76%, respectively. Gold recovery ranged from 0% to 100%. Average gold dissolution in Low-Grade Variability, Average-Grade Variability, and Lithology elevation composites were 0%, 55% and 45%, respectively.

Sedimentary Variability and Lithology composite bottle roll tests indicate silver recovery ranged from 29% to 90%. Average silver dissolutions in Low-Grade Variability, Average Grade Variability, and Lithology elevation composites

were 72%, 62% and 80%, respectively. Gold recovery ranged from 0% to 100%. Average gold dissolutions in Low-Grade Variability, Average-Grade Variability, and Lithology elevation composites were 0%, 63% and 49%, respectively.

Exploratory results in Quartz Vein material indicated;

- Silver dissolution with a particle size P_{80} of 77 and 49 µm, decreased 4% with a decrease in sodium cyanide concentration from 4 gpl to 2 gpl.
- Silver dissolution increased 2%, with a decrease in particle size P_{80} from 77 µm to 49 µm.
- Sodium cyanide consumption ranged from 2.7 to 3.3 kg/t with an initial sodium cyanide concentration of 4 gpl. Sodium cyanide consumption ranged from 1.6 to 2.6 kg/t with an initial sodium cyanide concentration of 2 gpl.
- Copper and zinc dissolutions ranged from 43% to 52% and 4% to 6%, respectively.
- Lithology Quartz Vein Rock Type composites completed silver leaching after 72 hours. Copper dissolution was near completion after 30 hours.

Exploratory results in Sedimentary material indicated;

- Silver dissolution with a particle size P_{80} of 69 µm, decreased 3% with a decrease in sodium cyanide concentration from 4 gpl to 2 gpl.
- Silver dissolution increased 3%, with a decrease in particle size P_{80} from 69 µm to 45 µm, and increased 9%, with a decrease in particle size P_{80} from 69 µm to 29 µm.
- Sodium cyanide consumption ranged from 1.9 to 2.5 kg/t with an initial sodium cyanide concentration of 4 gpl. Sodium cyanide consumption was 0.7 kg/t with an initial sodium cyanide concentration of 2 gpl.
- Copper and zinc dissolutions ranged from 49% to 61% and 2% to 3%, respectively.
- Lithology Sedimentary Rock Type composites with a particle size P_{80} of 29 µm and 45 µm indicated the fastest leach kinetics and highest silver dissolutions. Silver leaching was complete after 48 hours. Copper dissolution was near completion after 10 hours.

Exploratory results in Volcanic material indicated;

- The silver dissolution, decreased 8%, with a decrease in sodium cyanide from 4 to 2 gpl. (Duplicate tests should confirm this result.)
- Silver dissolution did not change with a decrease in particle size P_{80} from 64 µm to 49 µm, and increased 5% with a decrease in particle size P_{80} from 64 µm to 27 µm.
- Sodium cyanide consumption ranged from 1.4 to 3.3 kg/t with an initial sodium cyanide concentration of 4 gpl. Sodium cyanide consumption was 0.6 kg/t with an initial sodium cyanide concentration of 2 gpl.
- Copper and zinc dissolutions ranged from 24% to 36% and 1% to 2%, respectively.
- Lithology Volcanic Rock Type composites with a particle size P_{80} of 27 and 49 µm indicated the fastest leach kinetics, and highest silver dissolutions. Silver dissolution was completed after 48 hours. Copper dissolution was near completion after 10 hours.

Exploratory leach test results in simulated commercial tank pressure indicated;

- Pressure leach tests that simulated the commercial tank oxygen partial pressure improved silver dissolution.
- Silver dissolution in the Lithology Master Composite under atmospheric air pressure was 81%, and with 15 psig air pressure, 85%.
- Sodium cyanide consumption was 1.2 kg/t in atmospheric air pressure, and 1.1 kg/t in the leach test at 15 psig. Lime consumption was 2.7 kg/t in atmospheric air pressure, and 1.5 kg/t in the leach tests at 15 psig.
- Silver leaching was completed in 48 hours with 15 psig air pressure, and in 60-72 hours with atmospheric air pressure.

13.8.4 Bulk Leach – Variability Master Composite

Hazen conducted a bulk leach test on a 2 kg sample of the Variability Master composite (Hazen, 2014d). The leach parameters were a particle size P_{80} 45 µm, pulp density of 40 wt.% solids, for 96 hours, pH 11.4, and an initial cyanide concentration of 4 gpl.

The silver, gold, copper and zinc extractions were 88%, 65%, 41%, and 3%, respectively. The sodium cyanide addition and lime addition rates were 2.1 kg/t and 0.8 kg/t, respectively. Leach kinetics were not determined.

The slurry was filtered, washed and prepared for a sodium cyanide destruction test. The washed filter cake was re-pulped in primary filtrate and wash liquor. The liquors were combined to provide 1,000 mg/l free cyanide concentration, and a pulp density of 50 wt.% solids. The prepared slurry was subjected to a cyanide detoxification test and shipped to ACZ Laboratories for geochemical characterization.

The cyanide detoxification process was simulated with the addition of reagents and air to the slurry. The WAD cyanide measured less than 10 mg/l at the conclusion of the test. Reagents additions were not optimized. Sodium metabisulfite was added at a rate of 23 kg/t ore. Lime at a rate of 6.5 kg/t ore, and copper sulfate at a rate of 0.2 kg/t ore.

13.8.5 Bulk Leach – Lithology Master Composite

Bulk leach tests on the Lithology Master composite were completed at two primary grind sizes, P_{80} 45 µm and 74 µm. The leach tests were split into two leaches at P_{80} 74 µm, and two leaches at 45 µm. The leach parameters were a pulp density of 40 wt.% solids, for 72 hours, pH 11.4, and an initial cyanide concentration of 1.54 gpl.

The average silver, gold, copper and zinc extractions, at a particle size P_{80} 74 µm, were 79%, 63%, 47%, and 3%, respectively. The sodium cyanide and lime consumptions were 1.1 kg/t and 1.4 kg/t, respectively.

The average silver, gold, copper and zinc extractions, at a particle size P_{80} 45 µm, were 84%, 69%, 49%, and 3%, respectively. The sodium cyanide and lime consumptions were 2.0 kg/t and 1.0 kg/t, respectively.

Silver leaching was not complete after 72 hours.

The slurry was filtered, washed, and the filter cake prepared for sodium cyanide destruction tests. The filter cake was repulped in filtrate and wash liquor to provide 1,000 mg/l free cyanide concentration, and a pulp density of 50 wt.% solids. The prepared slurry was subjected to cyanide detoxification and shipped to ACZ Laboratories for geochemical characterization.

Solutions were submitted for ICP analysis.

13.8.6 1 kg Leach – Lithology Master Composite

A 1 kg bottle roll leach tests on the Lithology Master composite was completed at a primary grind size, P_{80} 82 µm. The leach parameters were a pulp density of 45 wt.% solids, 72 hours, pH 11.5, and an initial cyanide concentration of 4 gpl. Silver, gold, copper and zinc extractions were 81%, 49%, 49%, and 3%, respectively. Sodium cyanide and lime consumptions were 4.2 kg/t and 2.7 kg/t, respectively. Gold dissolution was complete after 24 hours. Silver dissolution was nearly complete after 72 hours.

13.8.7 Heap Leach Evaluation – ½" Bottle Roll Results

Twenty bottle roll tests were completed at a particle size P_{80} 8,950 µm to evaluate low grade material for heap leaching. The average head grade was 50 gpt. Silver dissolution averaged 26%. Heap leaching of low grade material was not recommended.

13.8.8 Feasibility Metal Recovery Methodology

13.8.8.1 Silver Dissolution

Silver recovery was based on head grade versus tail grade bottle roll results. The Variability Average-Grade, Lithology Elevation bench, and selected Exploratory bottle roll test data were evaluated.

The following adjustments to the data set were employed:

- A multiple linear regression was completed using; particle size P_{80} diameter, Area (P_{80}), and head grade versus tail grade, the data correlated poorly.
- The regression equations were used to adjust the tail grade data to a grind size of P_{80} of 74 µm.
- The final head grade versus tail grade regression correlation was based on a recovery adjustment for commercial size tank, based on the agitated pressure leach test.

Figure 13-2 and Figure 13-3 present head grade versus tail grade data, and silver dissolution, respectively. Silver recovery from life-of-mine plan head grades are indicated.

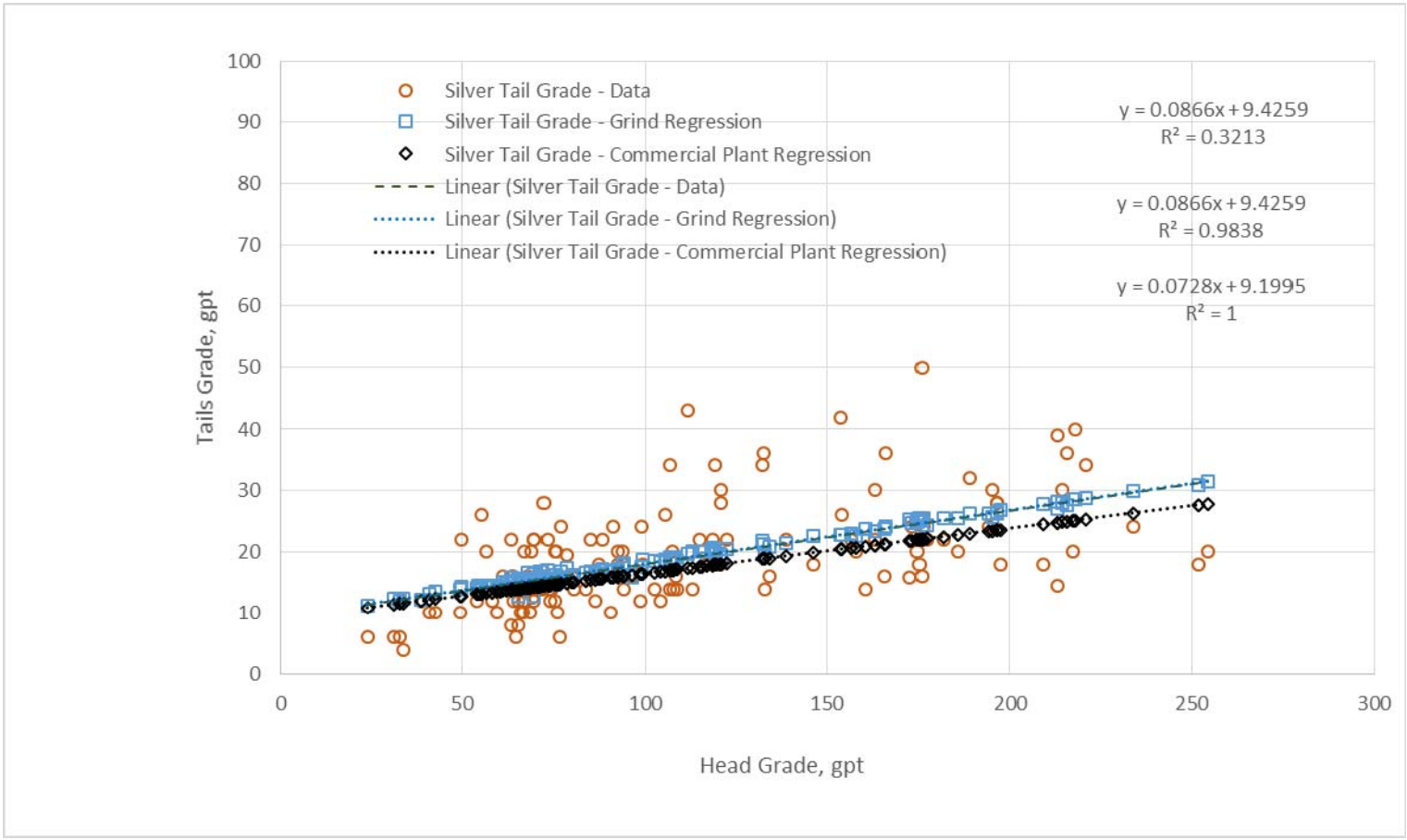


Figure 13-2: Feasibility Silver Recovery – Head Grade – Tail Grade Curve

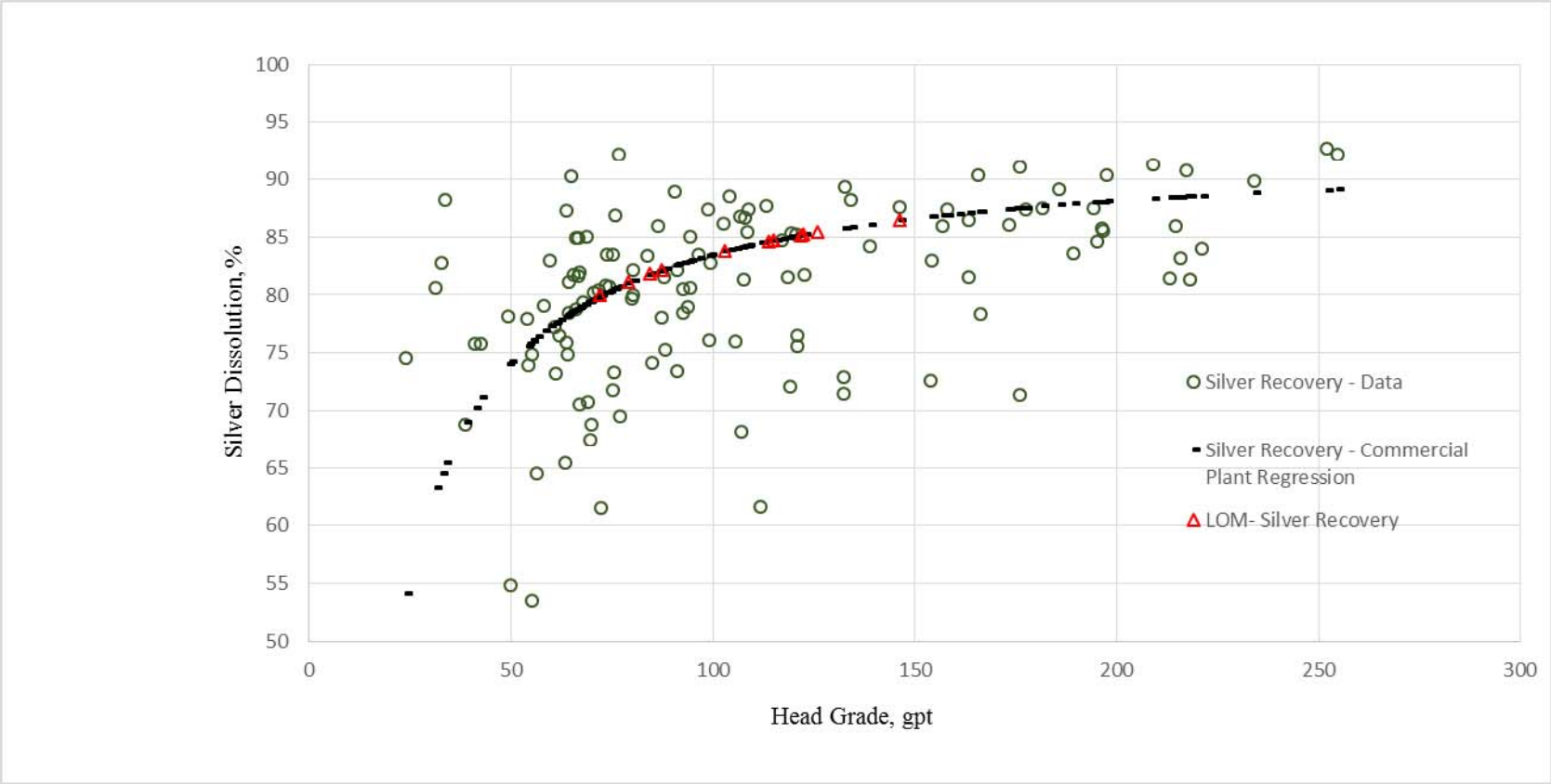


Figure 13-3: Feasibility Silver Recovery – Head Grade – Silver Dissolution

13.8.8.2 Gold Dissolution

Gold dissolution was based on head grade versus tail grade regression of the Variability Average Grade, Lithology Elevation bench, and selected Exploratory bottle roll test results.

The following methodology was employed:

- Due to low gold grade, poor analytical data resulted in many analyses at or near the detection limits for gold. Assay head grade data were select for the data set. A head grade versus tail grade regression was completed and the results reviewed. The review indicated some negative recovery points, which were replaced with adjacent data.

Figure 13-4 and Figure 13-5 present gold head grade versus tail grade data and recovery, respectively.

Silver and gold recoveries, based on mine plan head grades, were determined to be 84.1% and 61.2%, respectively.

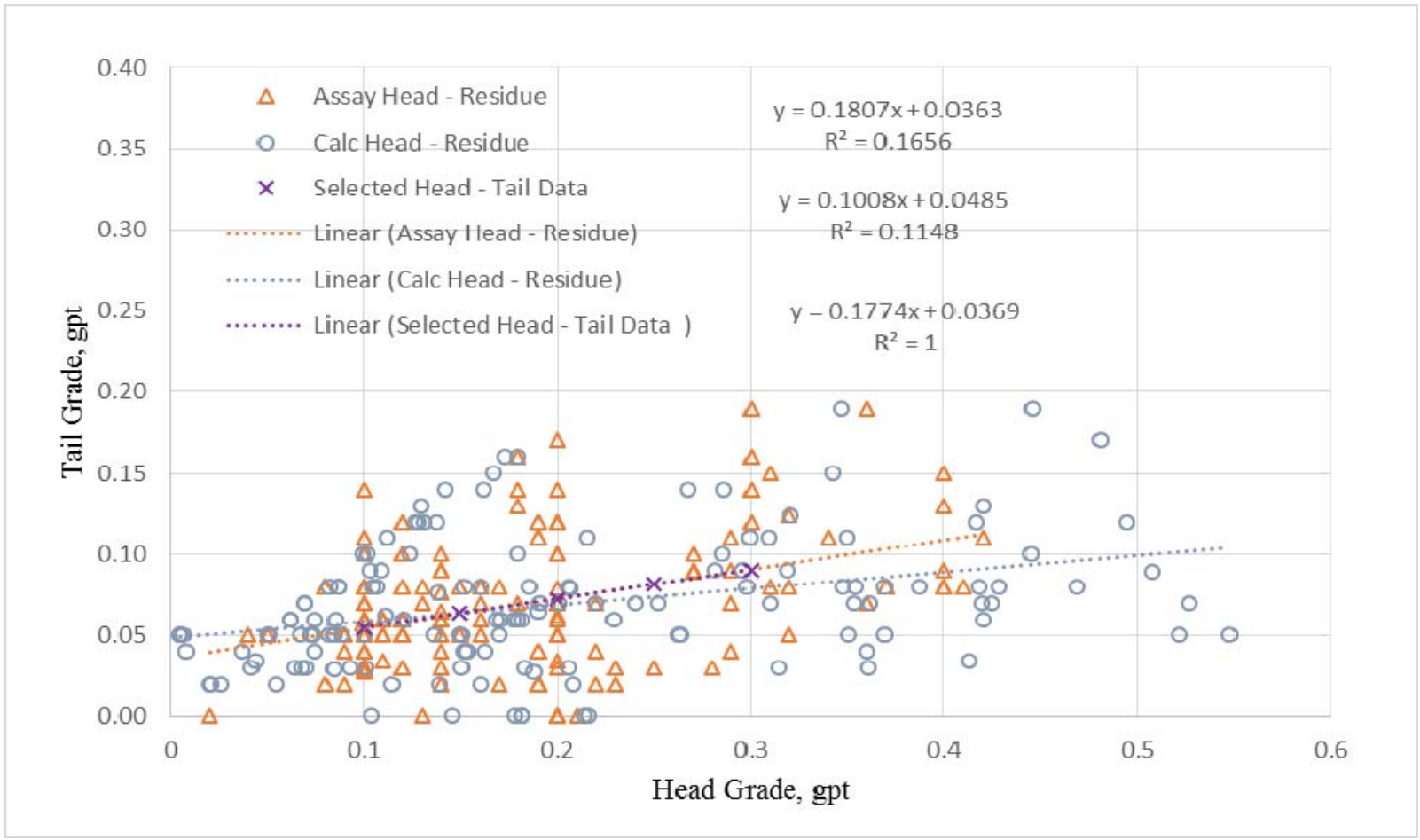


Figure 13-4: Feasibility Gold Recovery – Head Grade – Tail Grade Curve

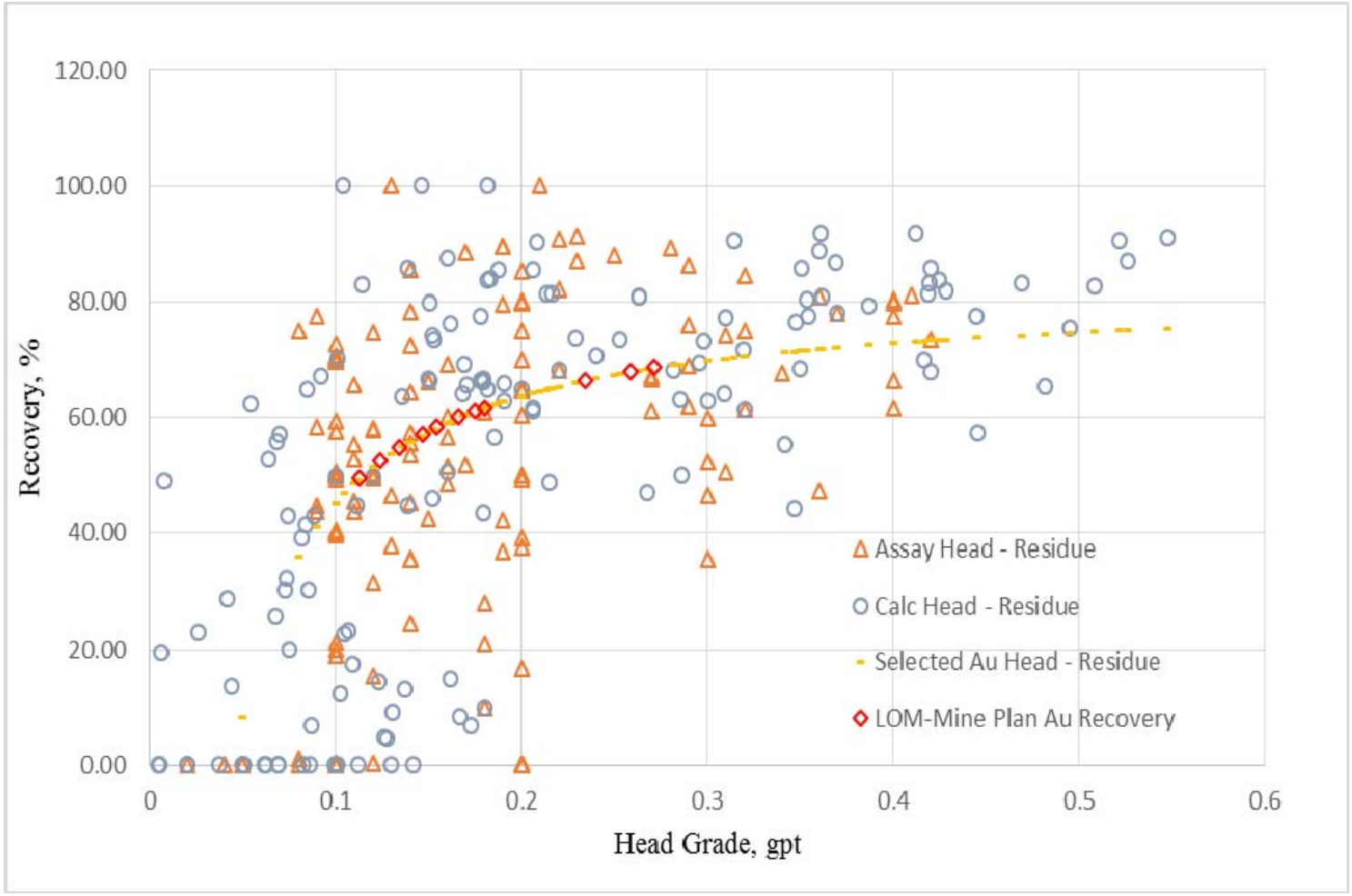


Figure 13-5: Feasibility Gold Recovery – Head Grade – Recovery

13.9 SOLID-LIQUID SEPARATION TESTING

Pocock conducted solids-liquid separation tests on six samples of Lithology Rock Type composites. Slurry was provided at two primary grind size P_{80} 45 µm and 75 µm. The test program included particle size analyses, flocculant screening, conventional, high-rate and paste thickening, underflow viscosity tests, pressure and vacuum filtration and pressure clarification.

13.10 TAILINGS CYANIDE DETOXIFICATION

The tests indicated that optimum conditions for obtaining a WAD CN concentration less than 100 mg/l are a residence time of 99 minutes, with 0.8 kg sodium metabisulfite/tonne ore, and 0.07 kg copper sulfate pentahydrate/tonne ore.

13.11 MERRILL-CROWE

Three Merrill-Crowe experiments were performed to demonstrate that silver could be precipitated from solution using standard technology. Silver-bearing leach liquors proved to be amenable to the simulated Merrill Crowe process.

13.12 FEASIBILITY STUDY REAGENT CONSUMPTION

The average cyanide consumption was determined to be 1.0 kg/t.

Lime consumption was determined to be 1.4 kg/t.

The flocculant consumption was determined to be 0.085 kg/tonne.

Cyanide destruction reagent consumptions were determined to be 0.8 kg sodium metabisulfite/t and 0.07 kg copper sulfate pentahydrate/t.

13.13 REPRESENTATIVE METALLURGICAL COMPOSITES

In this study, new composites, based on variability and lithology methodologies, and new metallurgical results, were developed independently from previous test work. Metallurgical results obtained from the composites are considered representative of the mineralization represented by the selected core samples. The metallurgical results indicate variable metallurgical response in the mineralized zones.

14 MINERAL RESOURCE ESTIMATES

14.1 SUMMARY

A new mineral resource model was built for this feasibility study, using updated drillhole data and multiple indicator kriging (MIK) was used to estimate both silver and gold grades. MIK was used to check for any potential biases in previous resource models and because MIK required less time to implement. Using the new resource model for the feasibility study the resulting Mineral Resources, exclusive of Mineral Reserves, are shown in Table 14-1.

Table 14-1: La Preciosa Mineral Resources, exlcusive of Mineral Reserves, as of 29 July 2014

Category	Tonnes	Average Grade (g/tonne)		Contained Ounces	
		Au	Ag	Au	Ag
Measured	6,839,000	0.186	84.1	40,900	18,485,000
Indicated	10,540,000	0.160	88.3	54,100	29,920,000
Total M&I	**17,379,000**	**0.170**	**86.6**	**95,000**	**48,405,000**
Inferred	1,889,000	0.126	77.5	7,700	4,705,000

1. Metal prices used for the estimation of Mineral Resources were $25 per troy ounce of silver and $1,400 per troy ounce of gold.
2. An NSR cutoff of $21.93/tonne ($24.17/short tonne) was used, based on the following parameters: NSR = [(Ag price per ounce - refining charge) × plant recovery × payable recovery] + [(Au price per ounce -refining charge) × plant recovery × payable recovery]
3. Rounding of short tonnes, grades and troy ounces, as required by reporting guidelines, may result in apparent differences between tonnes, grades and contained metal contents.
4. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be considered for estimation of Mineral Reserves.
5. U.S. Investors are cautioned that the term “mineral resource” is not defined or recognized by the U.S. Securities and Exchange Commission.
6. Mineral Resources are exclusive of Mineral Reserves.

14.1.1 Methodology

The estimation methodology used for this resource model was MIK, which was used for both the silver and gold estimates. A result of using MIK is a probabilistic model that represents the probability of finding ore in a model block rather than a deterministic model that tags each block as ore or waste. There are two major advantages to probabilistic methods, first they are quicker and less prone to biases that can be inadvertently introduced to the model when using deterministic methods and second, probabilistic models allow for a rigorous implementation of estimating mining dilution. It is much more difficult to achieve both items using deterministic models without the introduction of somewhat arbitrary factors.

14.1.2 Significant Changes from Other Models

The most significant change from the PEA resource model to the feasibility study resource model was the change from using ordinary kriging (OK) to MIK. Previous models used deterministic methods, i.e. hard boundaries were interpreted in three dimensions to represent the extent of the vein mineralization. Interpretation of these boundaries is a time-consuming task. Using discrete boundaries generally works well where there are very distinct and well defined veins, however there are a number of potential problems with this method. Other than the time required, the biggest potential problem is that it is easy to introduce biases into the resource tonnage and grade with very small changes in the interpreted boundaries. In cases where there are distinct well defined veins, there are inevitably splays and stockworks of smaller veins, as seen at the Project. It can be difficult to account for these smaller veins and the interpretation can become very subjective. For most deposits, an experienced geologist can do an excellent job of building a deterministic

model that takes into account smaller veins and stockwork. With the potential problems involved, it is difficult to validate the deterministic model. This was one of the primary reasons that the method was changed.

14.2 GEOLOGICAL DATABASE

14.2.1 Data Validation

Following Coeur's QA/QC protocols the database was thoroughly validated by Coeur (Section 11.0) prior to being used to develop the resource model. For the purposes of the resource model the data were essentially used as received from Coeur. No data checks were made as part of the resource modeling process, instead data validation was done before resource modeling. Early in the process a number of problems were found with drillhole surveys and some holes with missing data. All of these potential problems were resolved prior to the final model being estimated.

14.2.2 Un-Assayed Intervals in the Database

One significant issue was found with the pre-Coeur drillhole data is that only 37 percent of the meters drilled were sampled and assayed because of sampling decisions made by previous operators. Many of these un-assayed intervals are away from the known mineralization, but a significant number of un-assayed intervals are intermixed with assayed intervals in the mineralized zones. Ignoring these un-assayed intervals and treating them as missing data would lead to a significant selection bias in the estimate. To address the issue of un-assayed intervals all of the un-assayed intervals were filled with zero values except those that were logged as having no core recovery. It is anticipated that many of the intervals actually would have silver and gold values above zero, but the actual grade would likely be significantly less than the ore cutoff, so the impact of using a zero value rather than a non-zero value is likely to have minimal impact on the resource estimate.

Coeur recognizes that the presence of null values in the data is problematical, has identified the un-assayed intervals within the first few years of mining and has instituted a program of sampling and assaying the missing intervals. As a standard practice all intervals in Coeur's drill campaign are sampled.

14.2.3 Lithology Model

The lithology model developed by SRK Consulting of Toronto was used to code the blocks in the resource model. Fault zones interpreted by SRK were used directly as hard boundaries within the resource model. Lithology itself does not appear to be a factor in controlling the mineralization, thus the lithology model was used only to establish the rock density and was not used directly in controlling the interpolation.

14.2.4 Topographic Information

Site topography was done by PhotoSat™ (at the request of PAS) using high definition satellite photos (color ortho photos with 50 cm resolution acquired on 14 October 2011) from which a digital elevation model (DEM) was built, and the DEM then reduced to a topographic map with 1, 5, 10, and 50 m contour intervals. PhotoSat reports an accuracy of ±30 cm on a 1 m grid. An unknown number of control points were established on the ground to register the satellite image to the project datum (datum used is WGS84).

Contours were provided by PhotoSat in several digital formats, and the digital topographic data were imported into MineSight for use in the resource model. An indirect method of validation of topography by PhotoSat was done during PhotoSat's analysis of 759 drillhole collars. This drillhole collar validation for PAS (Section 12.2.3) was done by comparing surveyed collar coordinates against collar coordinates determined from digitized collars using PhotoSat's images. Although done for validating collar coordinates, this validation process works equally well for validating topography. In general the surveyed collar coordinates are on average within ±2.5 m of the DEM model.

14.2.5 Densities

Densities applied to the resource model, the QA/QC results, and measurement methodologies are discussed in Section 3.7.3.1. Densities were assigned to the resource model based on SRK's simplified lithology model. In the lithology model each block was coded as metamorphic (2.67 t/m^3), sedimentary (2.50 t/m^3), volcanic (2.48 t/m^3) or basalt (2.38 t/m^3). After the lithology types were coded into the blocks in the resource model, the densities were stored in the model based on the lithology code. There is one lithology and one density per block based on the majority coding of the lithology. Densities were then used directly in the resource summary and for mine planning.

14.3 STATISTICAL ANALYSIS

14.3.1 Exploratory Data Analysis (EDA)

The assay and geological database consists of data from 866 holes representing a total of 267,921 m of drilling. Silver and gold assays are present in 827 holes for a total of 100,446 m assayed. The average interval length is 0.84 m. Only 37% of the drilled meters have been assayed. In addition to assays, there are codes for mineralization, alteration and lithology. EDA focused primarily on analysis of controls on mineralization associated with lithology and the relationship between silver and gold. In addition, multi-element assays were done on a significant number of intervals, but those data were not used in EDA or mineral resource estimate.

14.3.1.1 Lithological Controls

The geological logging of lithology used 65 unique designations that were subsequently grouped into 10 unique stratigraphic units (Table 14-2); mineralization is present in all of the lithological units. Veining is heavily mineralized and significant mineralization also appears in the sedimentary and volcanic series.

Table 14-2: Occurrence of Straigraphic Units

Stratigraphic Unit	Designation	Occurrence (%)	Average Silver (ppm)	Average Gold (ppm)
Soil and Alluvium	9	0.1	13.0	0.010
Veining	8	6.4	127.0	0.227
Felsic Intrusive	7	1.0	2.6	0.017
Intermediate Intrusive	6	2.2	3.1	0.013
Mafic Event	5	1.0	1.3	0.006
Upper Volcanic Series	4	1.7	6.0	0.016
Lower Volcanic Series	3	54.5	11.2	0.023
Sedimentary Package	2	22.6	11.7	0.034
Metamorphic Basement Package	1	5.4	4.3	0.020
Non-stratigraphic Codes	0	4.6	9.9	0.030

Potential "ore" intervals were examined by looking at all intervals where the silver grade was greater than 31.1 ppm (Table 14-3). The veining appears to be the source of the mineralization, but significant mineralization occurs away from the veins within the adjacent volcanic and sedimentary rocks that are host to the veining.

Table 14-3: Occurrence of High-grade Intercepts.

Unit	Occurrence (m)	Relative Percent	Silver (ppm)	Gold (ppm)
Vein	3616	36	198.5	0.348
Volcanic	4067	41	99.6	0.143
Sedimentary	2248	23	119.5	0.208
Total	**9931**	**100**	**140.1**	**0.232**

14.3.1.2 Correlation between Silver and Gold

Correlation between silver and gold is important to the resource evaluation as both contribute to the economic value. Distributions of silver and gold were estimated independently then evaluated jointly on a block by block basis. The correlation was examined by stratigraphic unit (Table 14-4), and it can be seen that the correlation is relatively poor between silver and gold, but within the ore-bearing units the correlation is not unreasonable. A correlation of 0.5 could easily be explained by the nugget effect seen in the sampling and assaying process where the silver and gold are actually highly correlated.

Table 14-4: Correlation between Silver and Gold by Stratigraphic Unit

Stratigraphic Unit	Occurrence (%)	Correlation	Average Silver (ppm)	Average Gold (ppm)
Soil and Alluvium	0.1	0.38	13.0	0.010
Veining	6.4	0.45	127.0	0.227
Felsic Intrusive	1.0	0.37	2.6	0.017
Intermediate Intrusive	2.2	0.39	3.1	0.013
Mafic Event	1.0	0.28	1.3	0.006
Upper Volcanic Series	1.7	0.70	6.0	0.016
Lower Volcanic Series	54.5	0.53	11.2	0.023
Sedimentary Package	22.6	0.51	11.7	0.034
Metamorphic Basement Package	5.4	0.17	4.3	0.020

14.3.1.3 Composites

Fixed length 2.5 m composites were used for the evaluation of the resource. A 2.5 m composite was used with a 5 m bench height because it is anticipated that the mining will be highly selective. The composites were declustered using a cell declustering technique then tagged with lithology and estimation domain using the appropriate geometric solids. Within the model volume there are 95,143 composites from 786 drillholes.

Declustered weighted statistics for resource modeling domains are shown for the 2.5 m composites in Table 14-5. Methodology to establish resource modeling domains is discussed below. It can be seen that both the silver and gold grades are highly variable and that the correlation between the silver and gold is moderate to low.

Table 14-5: Composite Statistics

	Mean (ppm)		Std. Dev.			Coef. of Var. (CV)	
Domain	**Ag**	**Au**	**Ag**	**Au**	**Correlation**	**Ag**	**Au**
Martha North	8.3	0.023	41.0	0.151	0.44	5.0	6.5
Martha South	4.9	0.012	31.3	0.111	0.47	6.4	8.9
Gloria and Abundancia	7.8	0.014	36.2	0.119	0.41	4.6	8.3
Undifferentiated North	4.0	0.008	15.0	0.088	0.18	3.8	11.3
Undifferentiated South	1.6	0.004	11.0	0.066	0.20	6.7	15.1
All Domains	5.1	0.012	29.7	0.109	0.41	5.8	9.0

Cross variograms were calculated to examine the spatial relationship between silver and gold for each of the domains. The spatial correlation between silver and gold was very good, very similar to the silver and gold variograms. Nugget effects of the fitted models inferred greater correlation than calculated with the declustered statistics. Because of the nugget effect it is believed that the actual correlation between the silver and gold is higher than what is inferred by the declustered statistics and that the lower correlation coefficients can be explained by the nugget effect inherent in the variance associated with the sampling and assaying process.

14.3.2 Spatial Data Analysis

All of the spatial analysis was based on the 2.5 m fixed length composites. Correlograms were used exclusively to model the spatial structure of the data; and correlograms were calculated for each domain. All of the data in a given domain were used but data pairs that crossed fault boundaries were not allowed. In this way the faults were honored as hard boundaries without partitioning the domains. If the domains are made too small then it becomes very difficult to infer the spatial structure since there are not enough data pairs to dampen out the noise, using all the data in a domain while honoring the faults avoided this problem.

All fitted variogram models used spherical models. The procedure that was followed to fit models was the same for all variograms. Data were first paired into a three dimensional block matrix. The major, intermediate and minor directions of continuity were determined by slicing the three dimensional matrix to produce three orthogonal two dimensional variogram maps. Once the rotations were determined, three one-dimensional orthogonal variograms were created along the rotation axes. These variograms were then fitted with spherical models using Excel.

14.3.2.1 Silver Variography

Silver variograms were fitted for the grade data and for the indicators for each domain. The orientation of the structures remained constant for each domain with the exception of North Martha, which changed slightly with the upper indicator variables (Table 14-6).

Table 14-6: Structural Orientation of Estimation Domains

Domain	Strike (Azimuth)	Dip (degrees)
North Martha (Ind. 0 to 4)	N15E	30 to West
North Martha (Ind. 5 to 11)	N10E	40 to West
South Martha	N15W	25 to West
Gloria and Abundancia	NS	30 to West

It became impossible to fit variograms models at the higher indicator thresholds because the noise in the variogram overwhelmed any structure present at that point. The highest fitted variogram model was also used for all subsequent indicators. The fitting of variograms in the undifferentiated domain proved to be problematical because a large number of the unassayed intervals fall in this domain. Visual inspection of the domain shows that the mineralization present appeared to parallel mineralization seen in the Martha. The North and South Martha orientations and variograms were used to estimate the undifferentiated domain.

Structural continuity shown in the silver variograms is generally quite good, and the nugget effect seen in the variogram models are relatively low when compared to other precious metal deposits. This good continuity and low nugget is primarily because the mineralization is structurally controlled. These factors should significantly enhance the mines' ability to separate ore and waste with a high level of mining selectivity.

14.3.2.2 Gold Variography

Gold variograms were fitted to the grade data and for the indicators for each domain. The orientation of the structures for gold was identical to that of silver. The fitted variogram models for the gold variograms are very similar to that of the silver.

14.4 RESOURCE MODEL

14.4.1 Summary

The dimensions of the resource model are shown in Table 14-7. The model origin and dimensions are slightly different from previous models but were adjusted to be compatible with the SRK geologic model. A block height of 5 m was chosen to be compatible with the anticipated mining bench height.

Table 14-7: Model Schema

Dimension	Minimum (UTM m)	Maximum (UTM m)	Block Size (m)	No. of Blocks
East	554,500	557,000	10	250
North	2,700,050	2,703,200	25	126
Elevation	1,550	2,270	5	144

14.4.1.1 Modeling Domains

The resource model was originally separated into three main domains: the Gloria-Abunduncia, the Martha and the undifferentiated volume in between. Orientation of the Martha domain changes slightly from the south end to the north end, so the Martha domain was split into North and South domains. A large part of the drilling in the undifferentiated volume has not been sampled or assayed, thus it was virtually impossible to derive variograms for this undifferentiated volume. It appears that any mineralization in this undifferentiated volume is close to parallel to the Martha vein system. The undifferentiated volume was combined with the Martha domain and the two domains were estimated together using the variograms and orientations derived for the two Martha domains.

Fault boundaries, developed as part of the geological model, were used to subdivide the domains into fault blocks. The fault boundaries were treated as hard boundaries, that is data search was not allowed to cross the fault boundaries during estimation. Fault offsets appear to be relatively small, on the order of 100 meters, but are locally significant in the interpolation of the data. Domain boundaries that are not fault-controlled were treated as soft boundaries. Using soft boundaries the variograms and search orientations change for each domain but the data search was allowed to cross the domain boundaries.

14.4.1.2 General Kriging Plan

The general kriging plan was identical for all variables (silver and gold); only the search orientation and variograms used were changed by domain. A true octant search was used with a maximum of 12 composites per octant. A maximum of 12 composites were allowed from any given drillhole and a minimum of 26 and a maximum of 64 composites were used for interpolation. The limits of the number of composites per octant and per drillhole forced all estimated blocks to be estimated with a minimum of 3 octants populated and a minimum of three drillholes used. Block kriging was used with a block discretization of 4 × 8 × 2 for the 10 m × 25 m × 5 m block. A search ellipsoid with radii of 500 m along strike, 200 m down dip and 40 m perpendicular to the structure was used for block estimation. The orientation of the search ellipsoid was changed for each domain to be identical with the orientation of the variograms. Because of the limits on the number of composites used, the search limits were rarely extended to their maximum limits because the number of composite criteria was met before the search limit criteria were met. The number of composites used appears large but 2.5 m composites were used with the 5 m blocks instead of 5 m composites. The data support would be the same with 5 m composites and use half the number of composites.

Composites that fell within the model bounds were coded by fault zone and all of the blocks in the model were also coded by fault zone using majority coding. When the blocks were estimated, only composites within the same fault zone were used, i.e. the faults were treated as hard boundaries during estimation.

The estimation variance was stored in the block model as the third indicator for both silver and gold. In addition the following items were stored in the block model: number of composites, number of drillholes, distance to the closest composite, and the average distance to all composites was retained in the model for both the silver and gold indicator kriging.

14.4.2 Multiple Indicator Kriging

MIK was chosen as the method used to estimate the assayed values for all of the various domains, both veins and non-vein domains. Both silver and gold values were estimated using MIK, but silver and gold were estimated independently of each other. MIK was chosen because it is capable of producing reasonable estimates when there are highly skewed distributions of grades. For the Project, there are a relatively small number of high-grade samples that represent a large portion of the contained metal.

It is difficult to produce reasonable estimates that are not overly smooth using ordinary kriging (OK) when dealing with highly skewed data. The proper level of smoothing can be achieved by adopting methods that estimate a distribution rather than a single value for a block. Methods, such as, MIK or OK followed by uniform conditioning (UC) can do this. MIK was chosen primarily because it is an appropriate method that will work well and because of familiarity with the method.

MIK is a probabilistic method that produces a probabilistic view of the deposit, which is distinctly different from traditional models that attempt to produce a deterministic model. With a traditional deterministic model the actual location of mineralized material and the expected grade at every point is predicted. A resource model is produced that “looks like” the deposit. In contrast with a probabilistic model, the actual location of mineralized material is not determined. What is estimated is the probability of encountering ore as it is mined. As a large area is mined, the probability can be treated as a proportion of the tonnes and grade of ore and waste can be determined. The resulting resource model that is produced will be a “fuzzy” probabilistic model.

In a deterministic model a single grade is estimated into each block in the model, in contrast in an MIK model the entire grade probability distribution is estimated into each block. With MIK it is possible to apply any cutoff grade to a block and determine the probability of encountering material above that cutoff grade.

The grade distribution is estimated by using indicator variables where an indicator is a variable that is set to zero or one. An example of the use of an indicator would be to create an indicator variable for the composites where it is set to 1 if the grade is less than or equal to a threshold value of, say 30 ppm and to 0 if it is greater than 30 ppm. This variable is estimated into the resource model using OK, the result is a continuous value between 0 and 1 that represents the probability of encountering a composite less than or equal to the threshold of additional samples taken at the location being estimated. This estimation process is then done for multiple indicators set at a series of increasing discrete thresholds.

The result obtained from MIK is a conditional cumulative distribution function (CDF) for each block estimated. The estimated CDF for each block allows the portion of the block that is above any arbitrary cutoff and the average grade of that portion to be calculated.

A distribution directly estimated with MIK is the probability distribution of composites within the blocks. What is desired is the distribution on minable sized units within a block, not composite sized units. As the size of the volume being estimated increases, the variability of the estimated value will decrease, i.e. the variability of gold grades of a large volume of rock will be much less than the variability of the gold grades of individual composites within a given block.

This is known as the volume variance relationship and correcting for this relationship is known as “change of support”. To turn the estimated probability distribution of composites into a distribution of minable units (SMU), it is necessary to reduce the variance of the distribution. This variance reduction is done by using an indirect lognormal change of support with the application of a variance reduction factor (VRF) on a block by block basis. The change of support and the application of the working cutoff grade to the blocks is known as “post processing” the MIK distribution.

The block CDF is modeled by estimating the probability of obtaining a sample less than or equal to a discrete number of thresholds. For practical considerations, there need to be a relatively small number of discrete thresholds. The thresholds need to be chosen such that the CDF is adequately represented. If thresholds are correctly selected then the average grade can be reasonably calculated and change of support can be properly applied. Experience has shown that the best way to choose these thresholds is to distribute the thresholds such that in the low grade portion of the distribution, the amount of data between each threshold is approximately the same, and in the high grade portion of the distribution, the amount of metal between each threshold is approximately the same.

The post-processing of the indicator kriging is done using a parameter free method. This technique breaks the estimated CDF into a probability density function (pdf) for each block. The pdf is fitted using a piecewise linear function that is consistent with the estimated CDF. A cutoff grade can then be applied by integrating the density function above the cutoff. This technique eliminates the use of class means and the interpolation of grades between the thresholds; however it is necessary to provide a maximum grade of the distribution. For the integration of the density function, the top end of the distribution, between the last threshold and the maximum grade is considered a triangular distribution. The median of a triangular distribution is 29% of the way between the minimum and the maximum while the mean is 1/3 of the way between the minimum and maximum. The mean was used to establish the maximum grades.

14.4.3 Silver and Gold Estimation

Both silver and gold values were estimated using MIK, but silver and gold were estimated independently of each other and they are the only economic metal variables estimated. It is possible to implement a full bivariate MIK but it is not worthwhile at this time. The two distributions were combined on a block by block basis as part of the change-of-support process.

14.4.3.1 Silver Model

The silver model was produced using twelve indicator variables; with eleven indicator thresholds chosen to provide a reasonable balance between the quantity of tonnes and the quantity of metal in each class. With the large number of unassayed intervals, a twelfth indicator was added at 0.0 to properly reflect this portion of the population. The same thresholds were used for all domains. The highest two thresholds were not estimated in the undifferentiated domain and the maximum value of the distribution was adjusted for each domain. Since MIK will treat all data above the highest threshold equally, it is not necessary to consider top-cutting assay values.

Indicator thresholds were applied to the declustered distribution of the 2.5 m composites. Composites were declustered using cell declustering with a cell size of 50 m east-west, 100 m north-south and 10 m vertically. Silver distribution is extremely skewed with 65% of the metal in the deposit represented by approximately 1% of the data.

The maximum grades used in the post-processing of the indicator distributions were calculated for each domain using the class median of the top class, which assumes a distribution of the top class. This approach resulted in maximum silver grades of 1501 ppm for North Martha, 1018 ppm for South Martha, 946 for Gloria and Abundancia, and 459 ppm for the undifferentiated domain.

14.4.3.2 Gold Model

The gold model was produced using 12 indicator variables; with 11 indicator thresholds chosen to provide a reasonable balance between the quantity of tonnes and the quantity of metal in each class. With the large number of unassayed intervals, a twelfth indicator was added at 0.0 to properly reflect this portion of the population. The same thresholds were used for all domains. Unlike the silver model, all thresholds were estimated for all domains in the gold model.

The indicator thresholds were applied to the declustered distribution of the 2.5 m composites. Composites were declustered using cell declustering with a cell size of 50 m east-west, 100 m north-south and 10 m vertically. Gold distribution is extremely skewed with 65% of the metal in the deposit represented by approximately 1% of the data.

The maximum grades used in the post-processing of the indicator distributions were calculated using the class median of the top class assuming a triangular distribution of the top class, which resulted in a maximum grade of 2.87 ppm gold.

14.4.3.3 Post-processing of MIK Models – Change of Support

The results from MIK create an estimation of the distribution of composites within a block in the resource model. Although the estimates were created using a composite block size in the resource model, the blocks will not be mined the size they are represented in the resource model; instead they will be mined as much larger volumes, commonly referred to as a selective mining unit (SMU). Thus it's necessary to adjust the estimated grade distribution so that it reflects the distribution of SMUs rather than composites. This adjustment is referred to as a volume-variance correction, or change of support. The adjustment consists of reducing the variance of the distribution to represent the larger SMU volumes. A significant benefit of this process is that the resource model should represent what will be achieved at the time of mining, i.e. the predicted resource is fully diluted to represent what will be mined. Factoring in additional dilution is not required or recommended.

Once the adjustment is made to the indicator distribution, it is possible to apply any arbitrary cutoff grade and determine the proportion of ore and the average ore grade for a block in the model. The cutoff grade used for the Project is an economic cutoff grade with anticipated revenues from both silver and gold. Thus it is necessary to look at both the silver and gold distributions simultaneously to determine the revenue contribution of each to determine the economic cutoff. Treating the silver as the primary variable and gold as a secondary variable is reasonable because the majority of the revenue is from silver. The process of determining the economic portion of the block carried out using an iterative approach with a FORTRAN program. A cutoff grade was calculated for the silver using the average gold grade and the resulting ore portion was applied to the gold distribution and a gold grade for the ore portion was calculated. A new cutoff grade for the silver was calculated using the new gold grade which resulted in a new ore portion and new gold grade and this process was continued until the ore portion converged. This process used not only the silver and gold distributions but also the economic parameters associated with the project and the metallurgical model discussed below.

A net result of the process is that each estimated block will be populated with the proportion of the block that would be expected to be ore and the silver and gold grades of the ore portion of the block (Figure 14-1 and Figure 14-2).

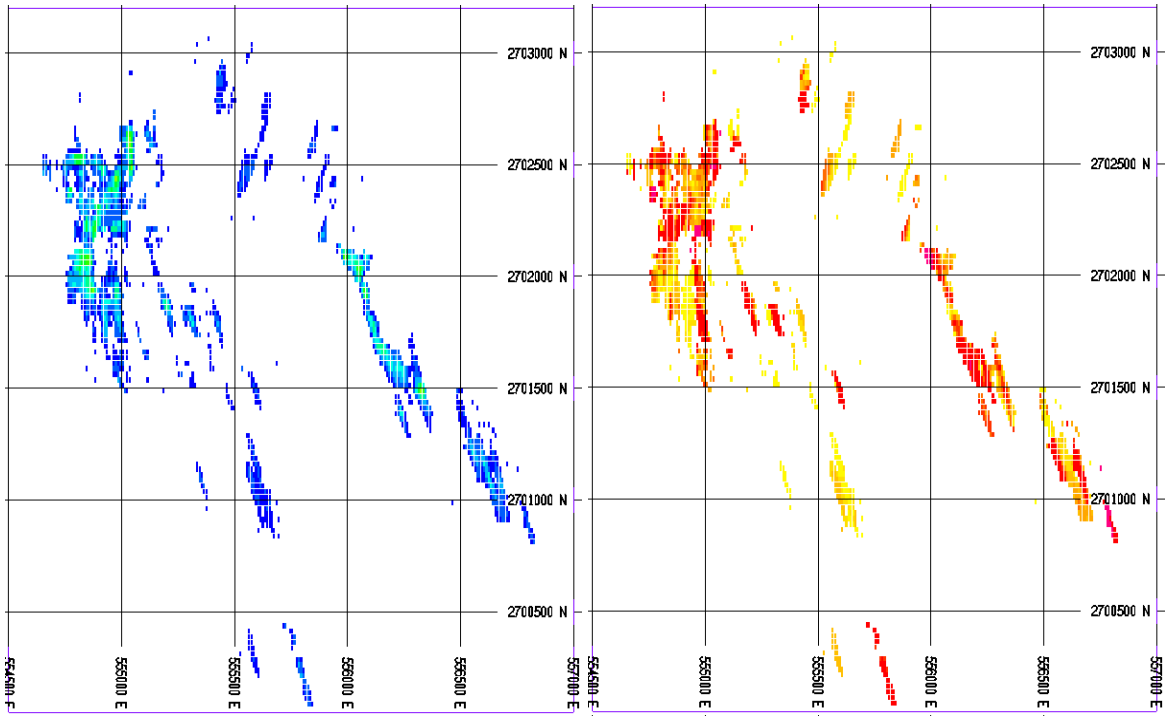


Figure 14-1: Block Model Silver and Gold Ore Portion

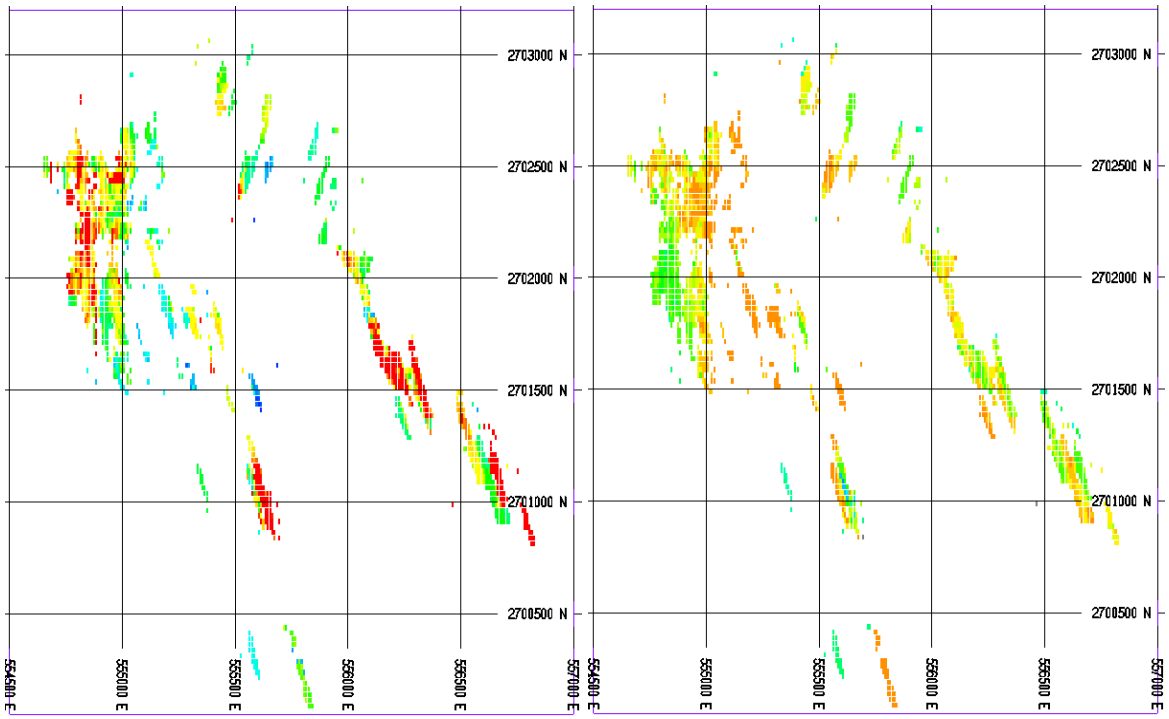


Figure 14-2: Ore Fraction, Ag Ore, Au Ore, and Equivalent Silver Cutoff – 1950 Bench

The change of support process requires an anticipated SMU size which is dictated by a number of factors, including the size of the mining equipment and the ore control practices employed. If the SMU size used is too small then the model will be under diluted and will predict too few tonnes at too low a grade. Conversely if the SMU size used is too large the model will be over diluted and will predict too many tonnes at too low a grade. It can be seen that it is critical to the veracity of the model and subsequent mine planning that the SMU size used be as close as possible to what will be achieved at the time of mining. The SMU will be used to calculate the amount of variance reduction that will be imposed on the indicator distribution. At operating properties, the required variance reduction is relatively easy to determine by reconciling the resource model to the production model. Prior to production it is necessary to calculate the variance reduction theoretically from the SMU size using the variogram. Variance reduction factors are calculated using variograms of the grade. Grade variograms were fitted by domain and the variance reduction factors calculated for an anticipated SMU of 5 m × 5 m ×5 m, which is a relatively small SMU by open pit mining standards, but reflects the expectation that a high level of care will be exercised in ore control and mining (Table 14-8).

Table 14-8: Variance Reduction Factors by Domain

Domain	Silver Variance Reduction	Gold Variance Reduction
North Martha	0.66	0.53
Gloria-Abundancia	0.60	0.48
Undifferentiated	0.60	0.48
South Martha	0.65	0.52

14.4.4 Metallurgical Model

The metallurgical model used is primarily dependent on the lithological composition of the ore. The metallurgy recoveries were grouped into three distinct rock types: vein quartz, volcanic, and sediments. It was necessary to attempt to predict the relative proportions of the various rock types that mill feed would be composed of, but the geological model was not sufficiently detailed to enable it to be used directly to estimate its composition. An indicator model was used to predict the rock type proportions associated with the ore by using the estimated proportions. Using the estimated proportions of rock type a linear weighted average of the metallurgical recovery parameters was calculated for each block in the model. A separate silver and gold recovery were calculated based on the grade of the ore portion of the block and stored back into the model.

14.4.4.1 Metallurgical Parameters

The originally proposed metallurgical recovery parameters consisted of a simple constant tail fraction with steps at various grades. Post-processing of the grade distributions requires that the metallurgical recovery be a continuous function rather than a step function and to accomplish this, a linear regression function was fitted to the tail grade versus the calculated head grade of the bottle roll tests. Separate regressions were done for each of the three rock types for silver and gold and the regressions were used to predict the recovered grade as a function of the head grade.

Recovered Grade = Head Grade – Tail Grade (Equation 1)

Recovered Grade = Head Grade – [Head Grade × slope + intercept] (Equation 2)

Recovered Grade = Head Grade × (1.0-slope) – intercept (Equation 3)

The derived parameters are shown in Table 14-9.

Table 14-9: Silver and Gold Recovery Parameters

Silver	Constant Tail	Recovery
Quartz Vein	8.9	0.916
Volcanics	6.8	0.852
Sedimentary	0.0	0.791
Gold		
Quartz Vein	0.020	0.794
Volcanics	0.050	0.956
Sedimentary	0.054	0.881

This was not done originally because the correlation coefficients of the linear regressions are not high, so an average tail and head value were used over groups of the data. This resulted in a non-continuous step function of recovery which is difficult to use effectively. Because of the scatter seen in the data, it may be desirable to attempt to determine if there are other geological attributes that may be possible to explain the different recoveries.

14.4.4.2 Estimation of the Ore composition

Ore composition was estimated using three indicator variables, one each for vein quartz, volcanic, and sediments. It was desired to estimate the composition of the ore portion of a block, not the relative proportions of the entire block. Indicator variables were established using only the assays that were potentially ore, whether an assay will actually be used in the estimation of ore is difficult to determine ahead of time, which is why the potential for ore was used. An assay was classified as potentially ore if the value was greater than 31.1 ppm Ag. Two calculated sample variables were added to the assay database, one for vein quartz and one for volcanic. If an assay interval was potentially ore and logged as vein quartz then the indicator variable was set to 1, if it was potentially ore and not vein quartz it was set to zero, and if it was not potentially ore it was set to missing, the volcanic indicator was set in a similar fashion. A separate composite file was created for the ore composition indicators using fixed length 2.5 m composites. The valid length (the length of the defined portion) of the composites was also stored. After the composites were created, a third indicator was added to represent the sediments as one minus the vein quartz and volcanic indicators.

Since the indicators only represent lithological proportions of the ore, it was impossible to derive variograms for them because the indicators were estimated by domain, with the fault zones as hard boundaries, just as the grade indicators were. The variograms modeled for the third silver indicator for each domain were used to estimate the ore composition indicators.

The estimation of the indicator values for a block was weighted by the valid length of the composite along with the kriging weights. This was done because the length of the potential ore within a composite would be volumetrically proportional to the composition of the ore.

14.4.4.3 Determination of Silver and Gold Recoveries by Block

A result of the estimation of the ore composition is the three indicator values should nominally sum to 1.0, and to insure they would sum to 1.0 the indicator values were normalized. A weighted linear average of the metallurgical parameters was then calculated using the predicted relative proportions and these proportions were used in the post-processing of the indicator distributions to determine the ore portion and ore grade of a block. Once the ore grade of the block was determined, it was possible to calculate the final recovery for both gold and silver and the metallurgical recoveries were stored back into the block.

14.4.5 Model Classification

Estimated blocks within the resource model were classified as Measured, Indicated, or Inferred based primarily on data support and closeness to available composite data. These two factors are indirectly represented by the kriging estimation variance, thus the final resource classification was based on this estimation variance. This resource classification approach is consistent with accepted and legal definitions of Measured, Indicated and Inferred.

Estimation variance for the third silver indicator is generally preferred for use as the criterion for resource classification as it will directly reflect the geometric configuration of the data. When the estimation is interpolating between data, the estimation variance will generally stay relatively low, reflecting higher confidence. When the estimation starts extrapolating away from data, the estimation variance will rise rapidly, reflecting low confidence. The estimation variance is basically made up of three components; first is the natural variability of the data; secondly is the distance from the block being estimated to the data being used; and thirdly is how well the data points are spread out in space. The distribution of the average distance to the closest composite is shown in Figure 14-3. There is some overlap in the distance to the closest composite but this only reflects one aspect of the data configuration and not how all of the data used is spatially configured. The distribution of the estimation variance versus the distance to the closest composite is shown on Figure 14-4. For this study, the thresholds chosen are 0.2 for Measured and 0.4 for Indicated, with everything greater than 0.4 being Inferred.

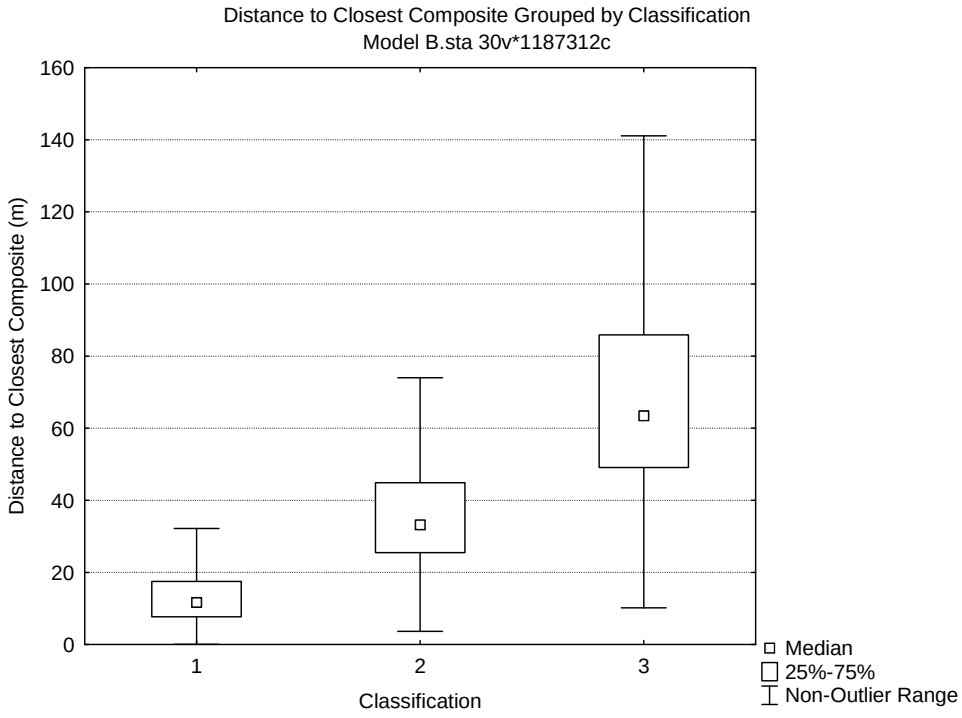


Figure 14-3: Distance to Closest Composite by Classification

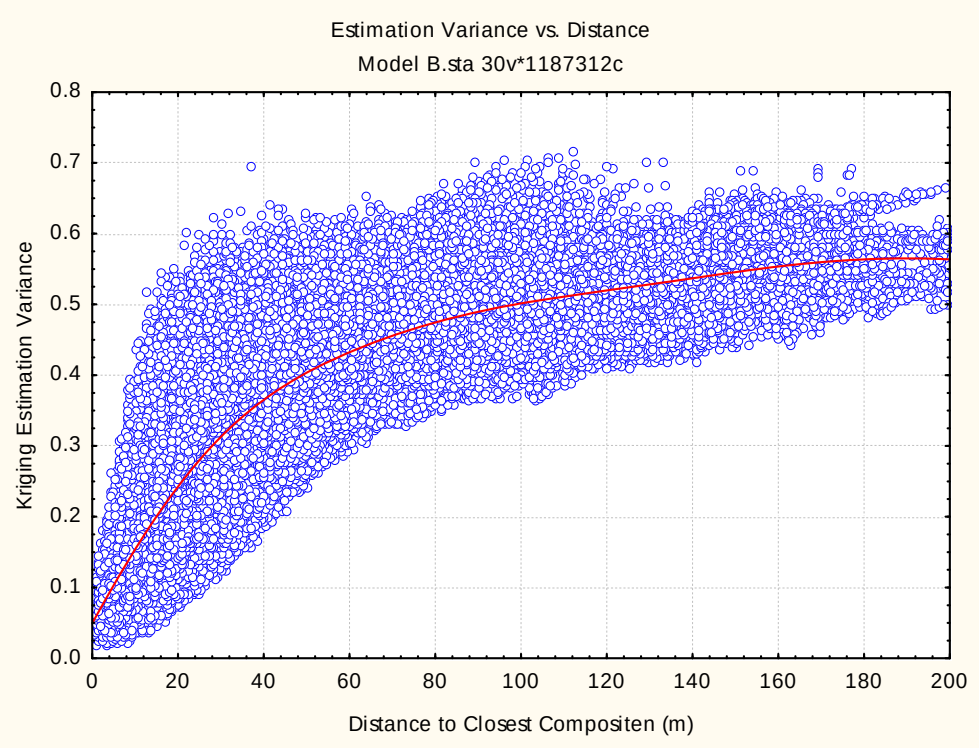


Figure 14-4: Estimation Variance vs. Distance to Closest Composite

The final classification is shown in Figure 14-5. An example of the final classification was based on the distance to the nearest composite. Where the estimation variance is less than 0.2, the block has been classified as Measured, where it is greater than or equal to 0.2 and less than 0.4, the block has been classified an Indicated; and when greater than or equal to 0.4 it is Inferred.

There is actually a relatively small amount of Inferred material estimated into the resource model. This is due to a large part of the function of the kriging plan because the kriging plan requires data to be present in at least three octants and from at least three drillholes. These limitations prevent excessive extrapolation of the model away from available data. Allowing further extrapolation away from the data would result in estimates that would be considered basically unsupported. An example of the estimation variance and final classification is shown if Figure 14-5 from the 1950 bench.

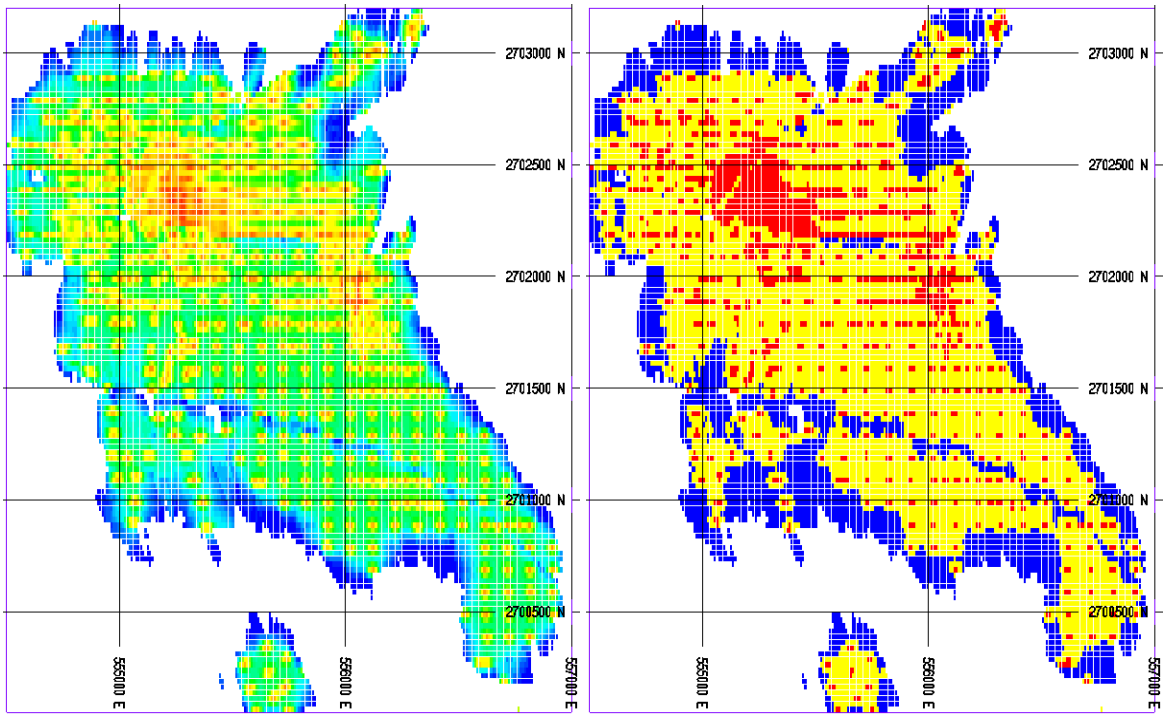


Figure 14-5: Estimation Variance (left) and Final Classification (right) – Example from 1950 Bench

14.4.6 Model Validation

Three levels of model validation were done: first the model was checked to ensure that it was globally unbiased, second it was checked to ensure that it was spatially unbiased, and finally the resource model was visually checked in cross-section and plan to ensure that the model compared reasonably to the drilling.

The MIK model was compared to a NN model to check for unbiasedness. The NN model will provide poor local estimates and should never be used for local estimates, but the NN model will provide a superior unbiased global estimate of the grade distribution within the model volume. Therefore, over large volumes, the NN estimate can be used to provide an excellent check against unbiasedness in the model. The average grade of each block was calculated using the estimated indicator distribution, the e-type estimate. The NN estimate was populated into the model by taking the nearest composite to the block center for each block. The nearest neighbor estimate honored the fault boundaries as did the kriged model.

Global unbiasedness was checked by averaging all of the estimated Measured and Indicated blocks in the model as summarized in Table 14-10. It can be seen that the e-type estimate for silver is slightly higher than the NN estimate while the gold estimate is slightly lower. These differences are within acceptable limits.

Table 14-10: Nearest Neighbor versus E-Type Comparison

	Nearest-Neighbor	E-Type	Difference
Silver (ppm)	4.87	5.03	103%
Gold (ppm)	0.0115	0.0111	97%

Swath plots were developed by averaging all of the block values within a given east-west section, north-south section or given elevation. The values for the nearest-neighbor and e-type estimates were plotted for each of the orthogonal directions. The swath plots were examined by estimation domain and globally. The global plots are shown in Figure

14-6 through Figure 14-8. It is expected that the kriged values would be a little smoother than the NN estimates which is the case. It can be seen that the kriged estimates follow the trends of the NN estimates in all three directions. There does not appear to be any spatial bias in the estimate. The exact same validation process was done for the gold estimate with similar results.

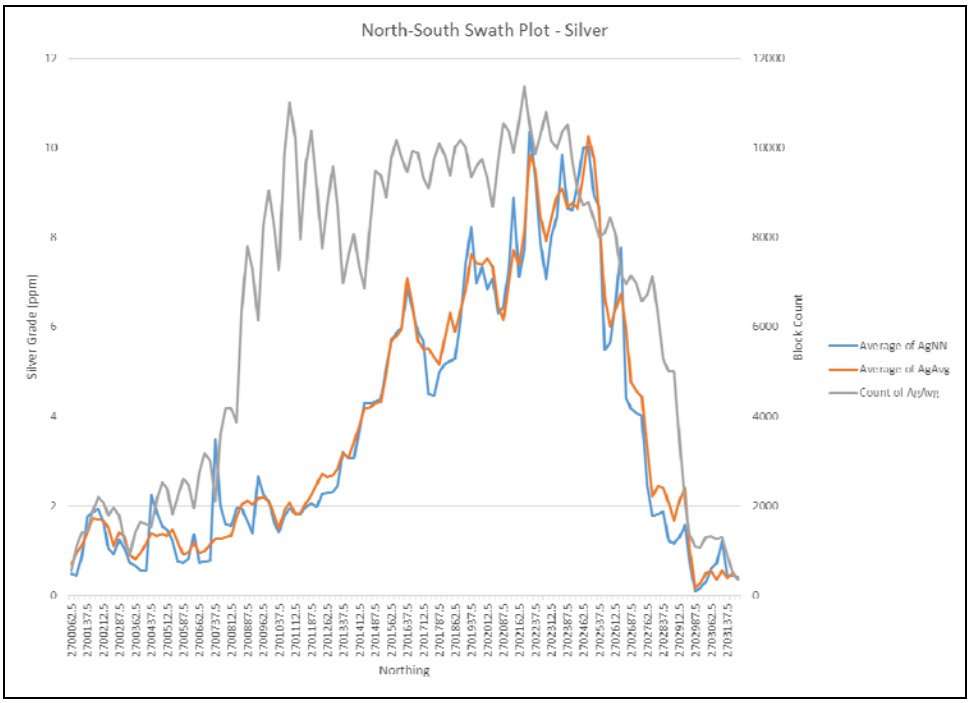


Figure 14-6: North-South Swath Plot – Silver 1

As a final validation the resource model was compared in cross-section and in plan against the drilling data using computer visualization. The resource model appears to do a very reasonable job of representing the original drilling data. The model also appears to be consistent with the faulting and domain models, as intended.

The ability of the mine operator to separate ore from waste selectively and the continuitiy or lack of continuity in the veining could reduce the average grade mined. There are no other mining, metallurgical, infrastructure, permitting or other relevant factors that would materially affect the mineral resource estimate.

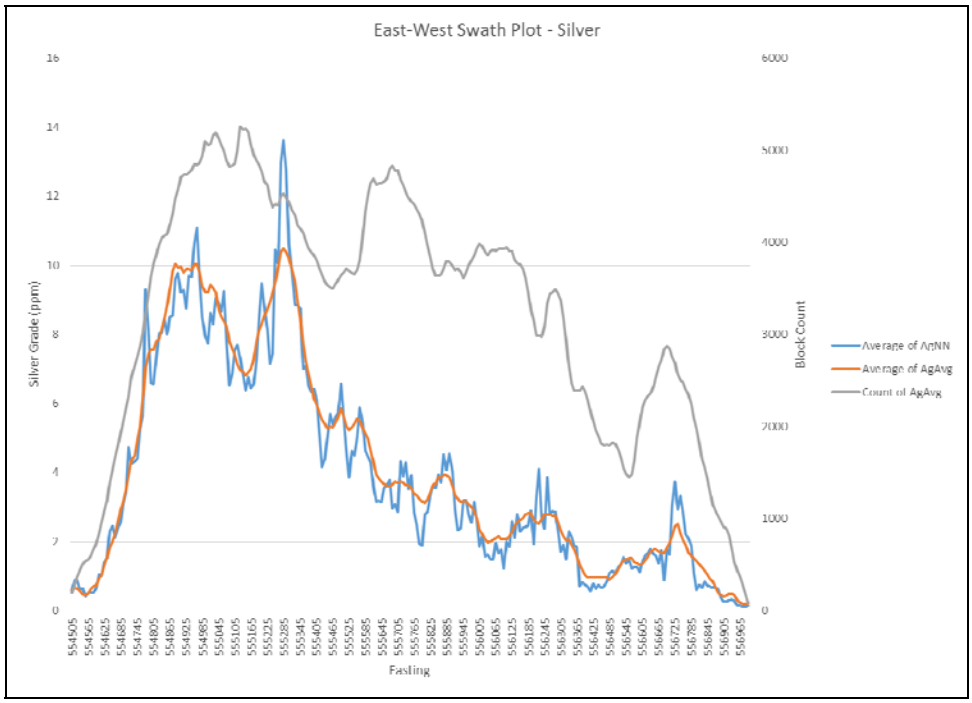


Figure 14-7: East-West Swath Plot – Silver 2

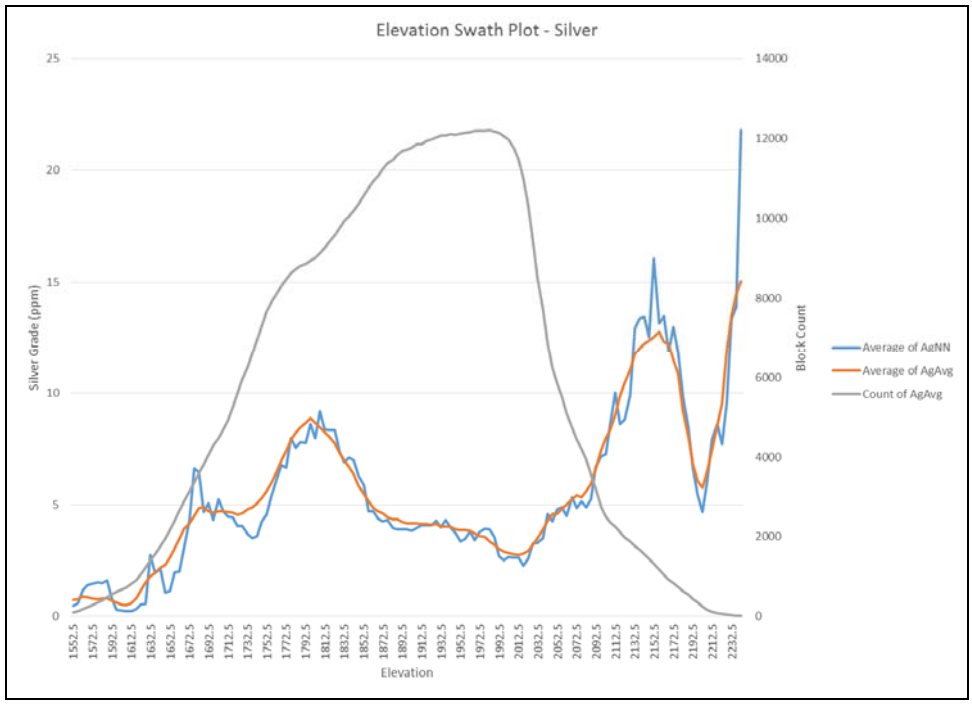


Figure 14-8: Elevation Swath Plot – Silver 3

15 MINERAL RESERVE ESTIMATES

Table 15-1 presents the mineral reserves for the Project based on the mine and plant production schedules developed for the project. The mineral reserves amount to 37.3 million ore tonnes at 105.4 g/t silver and 0.174 g/t gold. Contained metal amounts to 126.4 million ounces of silver and 208,500 ounces of gold. Measured and indicated mineral resources in the production schedule are converted to proven and probable mineral reserves, respectively.

Table 15-1: La Preciosa Mineral Reserves as of 29 July 2014

Category	Tonnes	Average Grade (g/tonne)		Contained Ounces	
		Au	Ag	Au	Ag
Proven	18,365,000	0.200	113.3	118,100	66,920,000
Probable	18,959,000	0.148	97.7	90,400	59,523,000
Total P+P	**37,324,000**	**0.174**	**105.4**	**208,500**	**126,443,000**

1. W. David Tyler is the QP for these reserve estimates. Metal prices used for estimation of Mineral Reserves were $22 per troy ounce of silver and $1,350 per troy ounce of gold. Metal prices used for the estimation of Mineral Resources were $25 per troy ounce of silver and $1,400 per troy ounce of gold.
2. An NSR cutoff of $21.93/tonne ($24.17/short tonne) was used, based on the following parameters: NSR = [(Ag price per ounce -refining charge) × plant recovery × payable recovery] + [(Au price per ounce -refining charge) × plant recovery × payable recovery]
3. Rounding of short tonnes, grades and troy ounces, as required by reporting guidelines, may result in apparent differences between tonnes, grades and contained metal contents.
4. U.S. Investors are cautioned that the term “mineral resource” is not defined or recognized by the U.S. Securities and Exchange Commission.

15.1 DESIGN ECONOMICS

Table 15-2 shows the economics for mine design and production scheduling. It should be noted that these are initial estimates used to initiate mine design and scheduling and are not the final economics developed for the Feasibility Study. The final pit design was based on silver and gold prices of $22 and $1,350 per ounce, respectively.

The mining, processing and G&A costs were estimated using the costs included in the 2013 PEA. The costs were not escalated between the PEA and the Feasibility Study. Average recoveries for silver and gold were 84.1% and 61.2%, respectively, based on the metallurgical model provided in the resource model.

The NSR calculation for the mine plan and mineral resource per tonne is:

NSR = [(Ag price per gram-refining charge) x silver recovery x refinery payable x silver grade] +

[(Au price per gram-refining charge) x gold recovery x refinery payable x gold grade]

NSR = [($22.00 / 31.1035) – 0.0122] x Silver Recovery x 0.997 x Ag grade (g/t)] + *Silver Contribution*

[($1350 / 31.1035) – 0.0122] x Gold Recovery x 0.9950 x Au grade (g/t)] *Gold Contribution*

NSR is calculated in US$ per tonne of ore. The breakeven cutoff grade would include the processing, G&A and the mining cost per tonne and totals $17.43 per tonne. The incremental cutoff grade assumes that the mining cost is sunk for blocks that have to be moved from the pit. The internal cutoff grade includes processing plus the G&A cost and equals $16.11/tonne.

Table 15-2: Economic Parameters

Parameter	Units	
Silver Price Per Ounce	US$	$22.00
Gold Price Per Ounce	US$	$1,350
Mining Cost Per Total Tonne	US$	$1.32
Ore Crushing & Process Per Ore Tonne	US$	$14.52
G&A Per Ore Tonne	US$	$1.59
Average Silver Recovery	%	84.1%
Average Gold Recovery	%	61.2%
Gold Payable at Refining	%	99.90%
Silver Payable at Refining	%	99.70%
Gold Refining Cost Per Ounce	US$	$0.38
Silver Refining Cost Per Ounce	US$	$0.38

As will be discussed in Section 16, Mining Methods, the mine production schedule and resultant mineral reserves estimate, is based on cutoff grades that vary from year to year to balance mine and plant production capacities. The cutoff grades are higher than the breakeven and internal cutoff grades.

Mining advance rates are high for the ore benches and will be further discussed in Section 16. There are no other mining, metallurgical, infrastructure, permitting or other relevant factors that would materially affect the mineral reserve estimate.

16 MINING METHODS

The Project mine plan was developed using conventional hard rock open pit mining methods. No underground resource was envisioned in this mine plan. While a smaller, higher grade portion of the resource could be recovered in an underground mine, the scale and percent recovery of the resource would be substantially diminished. In addition, the complicated structure of the low angle Martha vein which makes up 60% of the resource would make underground mining complicated and costly.

Scoping studies indicated that a production schedule filling the mill at 10,000 tpd maximized the Project return on investment. The total material rate is tied to equipment productivity. The total material moved ramps up to 214,000 tonnes per day in year 6, and averages 60,400 ktonnes/yr or 167,700 tpd for the first 10 years. The mine is scheduled to operate 365 days/yr with two, 12-hour shifts/day.

The mine plan was developed with a phased approach to facilitate sufficient ore release to provide the mill feed desired. In addition to the phases, mining was envisioned on 5 m in ore and 10 m in waste to increase productivity and ore release. The approach to phase designs, mine schedule, and mine equipment requirements are summarized below.

16.1 PHASE DESIGN

A Lerchs Grossman algorithm using a 10% discount rate was used as a guide to the design of the phases or pushbacks. Multiple economic pits were developed using the costs, slope angles and recoveries outlined in the Section below. The slope angles used include an allowance for roads and recommended catch benches in the overall slopes. Metal prices were varied from low to high in order to establish a series of multiple nested pit geometries and resulting NPVs. The results of this work indicated the starting point, final pit and the extraction sequence that maximized the NPV throughout the mine life.

The evaluation was based on a $22 per ounce silver and $1,350 per ounce gold price (Table 16-1). Recoveries were based on the metallurgical model. Refining costs and payabilities were based on quotes received for refining the Project doré. For pit design purposes, prices from the PEA were used to establish the pit limits. This pit was designed on only Measured and Indicated resources. Inferred resources were not included.

The benches were discounted according to a twelve 10-m bench per year annual advance rate.

Table 16-1: Input Parameters for Lerchs Grossman Economic Pit Evaluation

	Final Mine Plan
Gold Price ($/oz)	$1,350
Silver Price ($/oz)	$22.00
Gold Refining Cost ($/oz)	$0.38
Silver Refining Cost ($/oz)	$0.38
Mining ($/tonne)	$1.32
Ore Crushing & Process ($/ore tonne)	$14.52
G&A ($/ore tonne)	$1.59
Average Silver Recovery (%)	84.1%
Average Gold Recovery (%)	61.2%
Gold Payable (%)	99.90%
Silver Payable (%)	99.70%

Discounting of the ultimate pit by use of a discount to the value throughout the depth of the pit reduces the likelihood that uneconomic material will be mined at some future date. The discounting prior to estimating the economic pit is applied in order to maximize the NPV of the Project.

Seven phases were designed for Project with a minimum of 100 m of operating width on each bench within a phase, in general. Phase 1 was sized to contain roughly one year of mineralized material. The following summarizes the basic parameters used for mine design. The challenge in mining the Property is that there frequently is not one month of ore available on any one bench. General design criteria are listed in Table 16-2.

Table 16-2: Phase Design Parameters

Haul Road Width	30 meters
Haul Road Grade	10% Maximum
Interramp Slope Angles	by sector
Operating width between pushbacks	100 meters nominal

The phases were tabulated from the block model and those tabulations were used as input to the development of the mine production schedule. Figure 16-1 illustrates the relative position of the phases at surface. Table 16-3 summarizes the material in each phase.

Phase design was challenging since the majority of the ore is in the Martha Vein in the base of the deepest phases. The initial phases were focused on mining the more vertical sectors of the mine, in the Abundancia and Gloria veins. This is illustrated in Figure 16-1. This shows in the phase numbering from low to high, with Phases 1 through 3 mining the vertical veins, providing ore for mill feed while waste is removed from Phase 4.

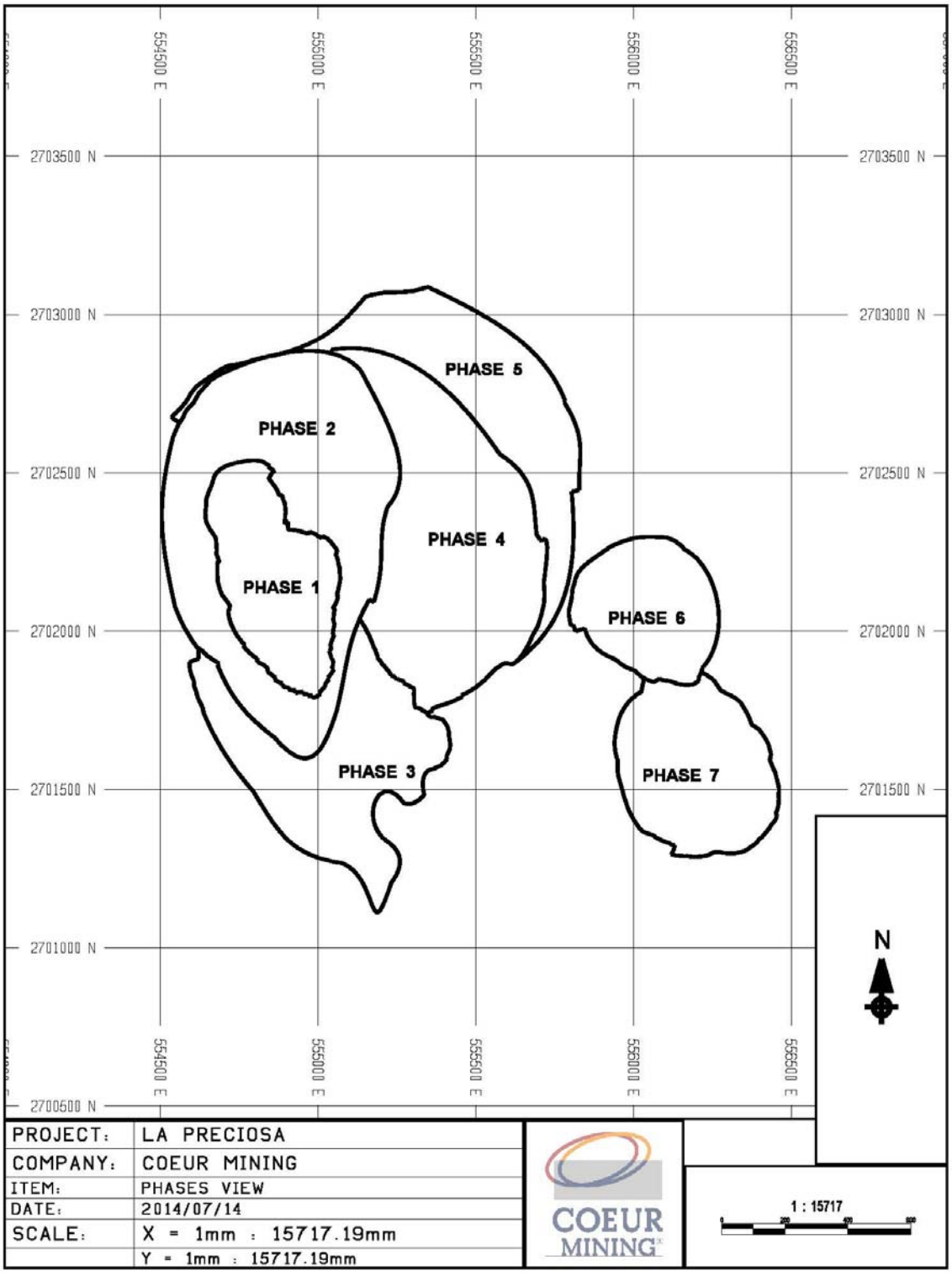


Figure 16-1: Relative Location of Phase Designs

Table 16-3: Phase Design Summary

Material By Phase (ktonnes)								
	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5	Phase 6	Phase 7	TOTAL
Ore	3,649	8,861	5,003	6,601	8,258	1,228	1,389	34,988
Ag grade (g/t)	115.2	95.5	87.2	125.3	105.6	90.2	129.6	105.4
Au grade (g/t)	0.146	0.147	0.100	0.229	0.215	0.130	0.262	0.174
Waste	26,348	130,281	81,480	176,578	135,692	16,478	34,188	601,046
TOTAL	**29,997**	**139,142**	**86,483**	**183,179**	**143,950**	**17,706**	**35,577**	**636,034**

16.2 METAL PRICE SENSITIVITY ANALYSIS

A comparison of LG pits was made to show the sensitivity of the resource to metal prices. The same economic and recovery factors were used as well as the preliminary slope angle recommendations. The exceptions are the slope angles were estimated using a 40 degree slope in the area between the Abundancia and the satellite pits. The LG analysis used a 10% discount rate in every case. The economic pits and the strip ratio calculated include the inferred material as waste.

Table 16-4: Metal Price Sensitivity Analysis

	Measured & Indicated			Inferred			Waste	
Silver Price	Tonnes	Grade Ag (g/t)	Grade Au (g/t)	Tonnes	Grade Ag (g/t)	Grade Au (g/t)	Tonnes	Strip Ratio
$18	31,881,974	111.82	0.182	1,165,356	85.68	0.132	510,368,510	17.5
$20	34,828,461	110.85	0.183	1,232,945	86.00	0.132	576,165,551	16.6
$22	36,001,772	110.14	0.183	1,273,816	85.59	0.133	600,683,114	15.5
$25	46,933,004	109.82	0.189	1,576,432	85.11	0.138	935,590,050	16.9
$30	50,600,923	108.30	0.188	1,852,295	83.96	0.143	1,044,560,861	16.9

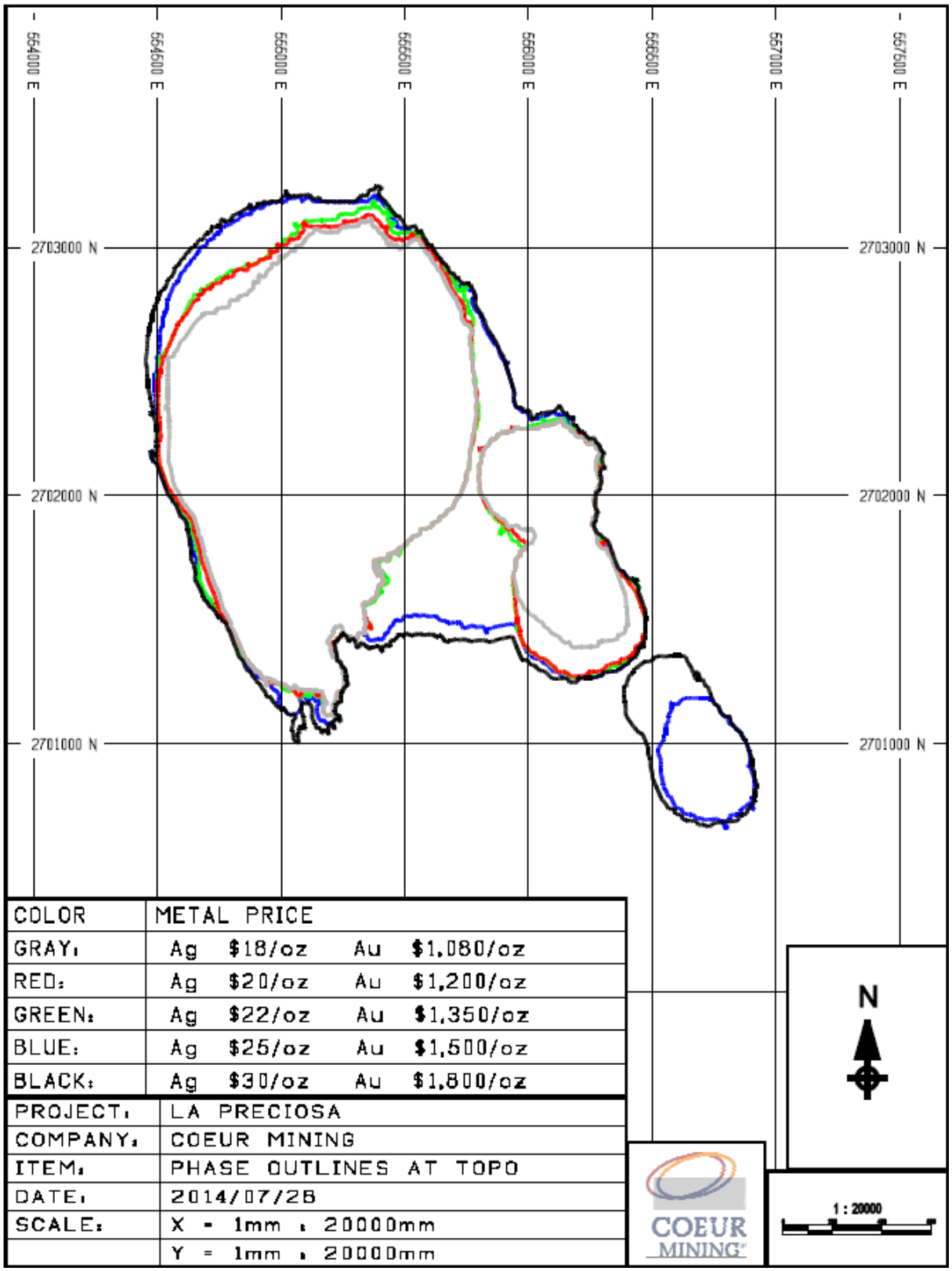


Figure 16-2: Metal Price Sensitivity Analysis

16.3 MINE PRODUCTION SCHEDULE

The mine production schedule was developed with the goal of filling the mill at 10,000 tpd and maximizing the Project return on investment. Multiple mine production schedules were developed that analyzed alternative cutoff grade strategies versus mine total material movement. Total material rates were tied to the size and number of loading units so that the final selected schedule would provide efficient use of the capital equipment employed.

The multiple schedules were evaluated on a NPV basis at the project design prices that were used to establish the mineral resource. The best overall production schedule on an economic and practical basis was selected and is summarized in Table 16-6.

This schedule utilizes a variable-cutoff scheme to maximize the grade moving to the processing facility. Five cutoffs were selected to allow the schedule optimization program to calculate the best solution to keep the mill filled to capacity. In order to move waste more efficiently, it was decided to mine waste from 10-m benches. This kept the vertical advance rates in waste to a reasonable 10-12 benches per year. To provide for selectivity, ore was planned to be mined from 5-m benches. Rates on 5-m benches are aggressive at 10,000 tonnes per day ore feed rates. The vertical advance in ore is 18-20 benches per year.

Pre-production is estimated to require 12 months, starting at 1 shift per day of operation. Once the first shift of operators is trained, a second shift will start training in the last four months of that year. A third shift will be added two months after that. A staff of seasoned operators will be hired to be a nucleus of trainers during the pre-production period and then to transition to senior operators.

Preproduction stripping will occur in Phases 1, 2 and 3. In many cases there is less than 1 month of mineralized material per 5m bench within a phase at the Project. Consequently, the mine plan maintains 2 phases in mineralized material in order to assure that mineralized material is released and available to assure plant feed.

The mineralized material that is encountered during pre-production (509 ktonnes) will be stockpiled near the crusher (or in the pit) and delivered to the mill during Year 1. The mineralized material in the upper benches of Phase 1 and Phase 2 is limited in quantity. This material is accumulated and stockpiled to ensure there is sufficient mineralized material release in Phase 1, Phase 2, and the stockpile combined to assure mill production.

Low grade material that is above the internal cutoff of $16.11/tonne NSR and less than mill feed cutoff in the early years will be stored in the low grade stockpile that is located northeast of the crusher. That material is planned to be re-mined and delivered to the mill in later years. Several cutoffs were used in to optimize the ore feed to the mill. The cutoffs used, by year, are listed in Table 16-7. Those cutoffs were based on lower metal prices and are summarized in Table 16-5. The bottom of Table 16-6 illustrates the mineralized material planned for processing inclusive of stockpile re-handling.

Table 16-5: Incremental Cutoffs used for Schedule Optimization

	case 1	case 2	case 3	case 4	case 5
Metal Price					
Silver ($/OZ)	22	19	16	13	10
Gold ($/Oz)	1350	1140	960	780	600
Average grade					
Silver (gr/tonne)	41.64	47.25	54.96	67.06	156.98
Gold (gr/tonne)	0.078	0.086	0.100	0.116	0.250
Average Recoveries					
Silver (%)	70.63%	73.25%	75.98%	79.00%	86.86%
Gold (%)	34.67%	39.52%	45.26%	50.52%	67.53%
Average NSR $/tonne	**21.93**	**22.35**	**22.83**	**23.57**	**46.99**

The cutoff grade for the mine schedule uses an NSR calculation that includes a credit for both the silver and gold. Using the parameters in Table 16-1, and using the metallurgical recovery model, an initial silver cutoff is calculated using the average gold grade in the block. Using the block percent ore, the average grade of gold is calculated for that portion of the grade distribution. The recoveries and resulting revenues are recalculated to re-estimate a new silver cutoff. This process is iterated until it converges.

The mine plan for the Project includes only Measured, and Indicated category mineralization. The economics do not include a credit from Inferred mineralization. The mine plan material in Table 16-6 contains 37.3 million tonnes of mill feed as shown in Table 16-7.

In addition, there are 3.4 million tonnes of mineralized material which is inferred. This resource offers potential metal production if the material can be converted to Measured or Indicated by additional drilling, within the economic pit shell and between the pits.

Once the pit was designed, incremental cutoffs were used to decide if direct mining of higher grade material could sustain the mill feed requirements. The desire was to feed the highest grade directly to the mill and stockpile lower grade material until the direct mining of that material could not fill the mill. Then stockpiled material would be added to the mill feed, from the highest grade stockpile first.

Table 16-6: Forecast Mine Production Schedule at 10,000 tpd

	From MINE			Stockpiles					
Mining Year	ORE 5m (kt)	HG 5m (kt)	LG 5m (kt)	WASTE 5m (kt)	WASTE 10m (kt)	HG to Mill (kt)	LG to Mill (kt)	Total from Mine (kt)	Total Moved (kt)
-1		304	205	764	9,237			10,509	10,509
1	3,002	654	979	6,953	38,861	304		50,448	50,752
2	3,259	0	956	6,323	67,214	390		77,751	78,141
3	3,649	0	1,296	7,418	66,245			78,607	78,607
4	3,651	0	494	6,218	67,651			78,013	78,013
5	2,886	0	0	4,329	70,020	264	500	77,235	77,999
6	2,749	0	0	4,124	70,405		900	77,277	78,177
7	3,651	0	0	5,477	64,436			73,563	73,563
8	1,434	0	0	2,151	44,310		2,217	47,895	50,112
9	3,338	0	0	5,007	20,176		313	28,521	28,834
10	3,651	0	0	5,477	5,236			14,363	14,363
11	1,168	0	0	1,752	3,898			6,818	6,818
Total	**32,438**	**958**	**3,930**	**55,989**	**527,685**	**958**	**3,930**	**621,000**	**625,888**

Table 16-7: Mill Feed Schedule

Year	Cutoff Grade NSR/mt	From Mine to Mill				From HG Skpl to Mill				From LG Skpl to Mill				Total Mill Feed				Waste Ktonnes
		Mineralized material Ktonnes	NSR $/tonne	Silver g/t	Gold g/t	HG Ktonnes	NSR $/tonne	Silver g/t	Gold g/t	LG Ktonnes	NSR $/tonne	Silver g/t	Gold g/t	Mineralized material Ktonnes	NSR $/tonne	Silver g/t	Gold g/t	
Prep-1																		10,000
1	23.57	3,002	73.02	117.3	0.182	304	46.14	79.037	0.109					3,306	70.55	113.77	0.175	45,813
2	22.83	3,259	79.43	130.4	0.128	390	29.41	53.95	0.086					3,649	74.08	122.25	0.124	73,536
3	22.83	3,649	74.51	121.6	0.147									3,649	74.51	121.61	0.147	73,662
4	22.83	3,651	63.27	102.8	0.166									3,651	63.27	102.83	0.166	73,868
5	22.35	2,886	60.29	97.7	0.174	264	29.41	53.95	0.086	500	23.3	44.995	0.074	3,650	52.99	87.3	0.154	74,349
6	21.93	2,749	53.88	90.3	0.126					900	23.3	44.995	0.074	3,649	46.34	79.11	0.113	74,528
7	21.93	3,651	94.6	146.3	0.271									3,651	94.6	146.32	0.271	69,912
8	21.93	1,434	72.48	113.9	0.228					2217	23.3	44.995	0.074	3,651	42.62	72.05	0.134	46,461
9	21.93	3,338	54.79	88.1	0.19					313	23.3	44.995	0.074	3,651	52.09	84.42	0.18	25,183
10	21.93	3,651	81.13	125.8	0.259									3,651	81.13	125.76	0.259	10,712
11	21.93	1,168	73.47	115.1	0.234									1,168	73.47	115.09	0.234	5,650
TOTAL		**32,438**	**$71.44**	**114.4**	**0.19**	**958**	**$34.72**	**61.9**	**0.09**	**3,930**	**$23.30**	**45**	**0.07**	**37,326**	**$65.43**	**105.8**	**0.17**	**583,674**

This Feasibility Study is based only on Measured and Indicated resources, which have been converted to mineral reserves. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Please see Section 14 of this report for a discussion of the qualifications and assumptions made by the Qualified Persons (QPs) responsible for this Report pertaining to mineral resource estimates.

16.3.1 Waste and Low-Grade Storage

The annual mine schedule drawings at the end of this section illustrate the location and size development of the waste dumps and the low-grade stockpile.

The low-grade stockpile is located adjacent to and east of the crusher and southeast of the Abundancia (main) pit. The location was selected to be a reasonable haul during the mine life for the storage of the material as well as a short haul distance to the crusher at the end of the mine life. The stockpile is planned to be re-mined and is not designed for future reclamation. The maximum total storage planned for the low-grade stockpile is 4,888 ktonnes of low-grade mineralized material that will be re-handled in subsequent years as the schedule requires.

There are two waste dumps planned at this time. The west dump is located 100 m northwest of the pit and the east dump is located 700 meters east of the pit. Waste storage was designed at 2.5 to 1 overall slopes (21.8 degrees) to facilitate reclamation at the end of the mine life. Dump lifts are expected to be 30 m high at the angle of repose (37 degrees) with 35 m setbacks between lifts. The total storage of waste material over the mine life is 583.7 million tonnes. The west dump is designed to contain 354.7 million tonnes and the east dump will contain 472.8 million tonnes, for a total dump capacity of 851.9 million tonnes.

There is no provision for re-contouring of the waste dumps within the mine operating costs. Re-mining of the low-grade stockpile is included in the mine operating costs.

16.3.2 Mine Equipment Requirements

Mine equipment is expected to be comprised of standard commercially available units. Principal mining equipment is a fleet of hydraulic mining shovels and 184 tonne class mechanical drive rock trucks.

Drilling on 5-m benches is expected to be completed with rotary blast hole rigs with 180 kN pull down capacity. Drilling on 10-m benches is expected to be completed with rotary blast hole rigs with 200 kN pull down capacity. A down-the-hole drill will be used for drilling trim wall patterns.

Drill capacity was based on a 0.2 kilogram of explosives per tonne of rock powder factor. This necessitated the use of the maximum drill bit sizes for the 5 and 10-m drills. The drill bit used for the 5-m benches was sized at 20.3 centimeter diameter. The drill bit for the 10-m benches was sized at 25.4 cm diameter.

The blasted rock will be loaded into 184-tonne haul trucks using three 26 m^3 hydraulic shovels and one 18.5 m^3 front-end loader. The loading equipment was selected to be able to mine on 5 m and 10 m benches. Truck fleet requirements were developed from haul time simulation over profiles measured for each material type, by phase, for each year of the mine plan. The trucks can be loaded with the hydraulic shovels and loaders.

The equipment fleet includes a large hydraulic excavator with a backhoe configuration that can be used to load mineralized material in narrow veins or used as an auxiliary support unit. The excavator has been scheduled for heavy use in all years so that the operating costs reflect the potential use as a material loading unit. A 10.5 m^3 loader and two 100-tonne haul trucks will be used for mining of high grade within the pit to improve selectivity and rehandling of stockpiled material.

Table 16-8 summarizes the major mine equipment units that are planned to be on site throughout the mine life. Additional minor units are included in the capital cost in Section 21 that will be required to maintain and sustain mine operations.

Table 16-8: Mine Major Equipment Fleet On Hand (Units owned based on fleet build up and replacement)

Equipment Type	Time Period											
	-1	1	2	3	4	5	6	7	8	9	10	11
5m Blast Hole Drill (180kN pulldown)	1	3	3	3	3	3	3	3	3	3	3	3
10m Blast Hole Drill (200kN pulldown)	1	3	5	5	5	5	5	5	5	5	5	5
Hydraulic Front Shovel (26 m^3)	0	2	3	3	3	3	3	3	3	3	3	3
Rubber-tired Front End Loader (18 m^3)	1	1	1	1	1	1	1	1	1	1	1	1
Haul Truck (177 Tonne)	3	11	18	19	23	24	30	30	30	30	30	30
Track Dozer (574 hp)	3	4	4	4	4	4	4	4	4	3	3	3
Wheel Dozer (354 hp)	3	3	3	3	3	3	3	3	3	3	3	3
Motor Grader (16 ft)	3	3	3	3	3	3	3	3	3	3	3	3
Water Truck (75,000 Liters)	3	3	3	3	3	3	3	3	3	3	3	0
Aux Front End Loader (11 m^3)	1	1	1	1	1	1	1	1	1	1	1	1
Aux Haul Truck (90 Tonne)	2	2	2	2	2	2	2	2	2	2	0	0
Down-The-Hole Percussion Drill (330 KW)	1	1	1	1	1	1	1	1	1	0	0	0
Hydraulic Excavator (4.5 m^3)	1	1	1	1	1	1	1	1	1	1	1	1
TOTAL	**23**	**38**	**48**	**49**	**53**	**54**	**60**	**60**	**60**	**58**	**56**	**53**

16.3.3 Major Equipment Productivities

Operating time per shift has been derated from a nominal 12-hour shift to arrive at a metered operating time of 11 hours. This has further been derated by an efficiency factor per metered hour of 83.3%, to arrive at the net productive operating time per shift of 9.2 hours. The details are in the table below.

Table 16-9: Summary of Operating Time Per Shift

	Operating Time (minutes)
Scheduled Time per Shift (minutes)	720
Less Scheduled Non-productive times (minutes)	
Travel Time/Shift Change/Blasting	10
Equipment Inspection	10
Lunch/Breaks	30
Fueling, Lube & Service	10
Net Scheduled Productive Time (Metered Operating Time) (minutes)	660
Job Efficiency (50 minutes Productive Time per Metered Hour) (minutes)	83.3%
Net Productive Operating Time per Shift (minutes)	550

Major equipment has had a mechanical availability applied over the life of the Project of 85-86%. The utilization of available time varies by class of equipment from 75% to 95%, as shown in Table 16-10 below.

Table 16-10: Utilization and Availability of Mining Equipment

Equipment Type	Mechanical Availability	Utilization of Availability	Maximum Utilization
5m Blast Hole Drill (180kN pulldown)	85%	90%	76.5%
10m Blast Hole Drill (200kN pulldown)	85%	90%	76.5%
Hydraulic Front Shovel (26 m^3)	86%	95%	81.7%
Rubber-tired Front End Loader (18 m^3)	86%	95%	81.7%
Haul Truck (177 Tonne)	85%	95%	80.8%
Track Dozer (574 hp)	85%	75%	63.8%
Wheel Dozer (354 hp)	85%	75%	63.8%
Motor Grader (16 ft)	85%	75%	63.8%
Water Truck (75,000 Liters)	85%	75%	63.8%
Aux Front End Loader (11 m^3)	85%	75%	63.8%
Aux Haul Truck (90 Tonne)	85%	75%	63.8%
Down-The-Hole Percussion Drill (330 KW)	85%	75%	63.8%
Hydraulic Excavator (4.5 cuM)	86%	95%	81.7%

Equipment replacement was also considered. The equipment replacement was estimated based on the maximum utilization and an expected annual equipment hours per year. Those equipment hours were divided into an expected equipment life. The 11-year mine life is within the design life of the principal loading and haulage equipment. Those are noted in Table 16-11 below.

Table 16-11: Equipment Life and Years to Replacement

Equipment Type	Max Utilization	Hours per year	Equip Life (hours)	Years to Replace	Notes
5m Blast Hole Drill (180kN pulldown)	76.50%	6,143	40,000	7	
10m Blast Hole Drill (200kN pulldown)	76.50%	6,143	60,000	10	not replaced
Hydraulic Front Shovel (26 CuM)	81.70%	6,561	80,000	12	not replaced
Rubber-tired Front End Loader (18 CuM)	81.70%	6,561	30,000	5	
Haul Truck (177 Tonne)	80.80%	6,488	60,000	9	not replaced
Track Dozer (574 hp)	63.80%	5,123	30,000	6	
Wheel Dozer (354 hp)	63.80%	5,123	30,000	6	
Motor Grader (16 ft)	63.80%	5,123	30,000	6	
Water Truck (75,000 Liters)	63.80%	5,123	48,000	9	not replaced
Aux Front End Loader (xx CuM)	63.80%	5,123	30,000	6	
Aux Haul Truck (90 Tonne)	63.80%	5,123	48,000	9	not replaced
Down-The-Hole Percussion Drill (330 KW)	63.80%	5,123	60,000	12	not replaced
Hydraulic Excavator (4.5 cuM)	81.70%	6,561	20,000	3	

16.3.4 Minor Support Equipment List

Minor equipment was included to support the blasting, dewatering, maintenance and communications for the operations. Additionally, a spare shovel bucket was included to allow rebuilds of the bucket off of the machine.

16.3.5 Manpower Requirements

Mine total manpower is based on having a 3 panel roster of crews to man the mine on a 2 by 12-hour shift basis. The rotations are anticipated to be 5 days of dayshift, 5 days of nightshift and 5 days off, with 24 hours between the end of day shift and the start of night shift. This allows for one crew to be off at all times.

Operating manpower numbers were set by the numbers of equipment, multiplied by the 3 panels. Maintenance manpower numbers were set by using experience and a ratio of about 60%. These numbers do not include the contracted services for diesel supply, tire maintenance and explosives delivery. A blasting crew was put on to design and load the explosives.

Labor rates are modeled after Coeur Mexico's Palmarejo operations, and include fringe benefits. An allowance of 10% of the overall manning schedule was added to allow for vacation, sickness and absenteeism.

16.3.6 Supervisory – Salaried Labor

A project supervisor is assigned to the general manager to accomplish minor projects during the life of mine. Maintenance supervision also doubles as warehouse and purchasing supervision, at the shop/warehouse complex, due to their close proximity.

Table 16-12: Mine Hourly Labor Requirements

JOB TITLE	Annual Cost	-1	1	2	3	4	5	6	7	8	9	10	11
MINE OPERATIONS:													
Drill Operator	18,103	3	13	17	18	17	16	16	16	10	8	6	2
Shovel Operator	18,103	0	5	9	9	9	9	9	8	6	3	2	0
Loader Operator	18,103	3	3	3	3	3	3	3	3	2	1	1	2
Haul Truck Driver	18,103	7	29	48	53	63	67	82	72	58	45	27	13
Track Dozer Operator	18,103	7	9	9	9	9	9	9	9	9	7	7	3
RTD Operator (Wheel Dozer)	18,103	5	7	7	7	7	7	7	7	7	5	5	2
Grader Operator	18,103	7	7	7	7	7	7	7	7	7	7	5	2
Water Truck Operator	15,318	7	7	7	7	7	7	7	7	7	7	5	2
Utility Equip Operator (Service Crew)	15,318	14	14	14	14	14	14	14	14	14	14	12	4
Blasting Crew	15,318	4	6	6	6	6	6	6	6	6	6	6	
Dispatch Operator	19,495	1	3	3	3	3	3	3	3	3	3	3	3
Laborer	12,533	2	6	6	6	6	6	6	6	6	6	6	6
Operations Total		60	109	136	142	151	154	169	158	135	112	85	39
MINE MAINTENANCE:													
Mechanics I	42,810	13	28	36	38	39	41	45	42	35	29	21	9
Mechanics II	18,103	5	11	15	15	16	17	18	17	14	12	9	4
Welder	42,810	5	10	12	13	14	14	16	14	12	10	8	3
Electronic Tech.	42,810	5	10	12	13	14	14	16	14	12	10	8	3
Fuel & Lube Crew	18,103	6	6	6	6	6	6	6	6	6	6	6	2
Tire Crew	18,103	0	0	0	0	0	0	0	0	0	0	0	0
Laborer Maintenance	12,533	3	3	3	3	3	3	3	3	3	3	3	1
Maintenance Total		37	68	84	88	92	95	104	96	82	70	55	22
VS&A at		10	18	22	23	24	25	27	25	22	18	14	6
TOTAL LABOR REQUIREMENT		107	195	242	253	267	274	300	279	239	200	154	67
Maintenance /Operations Ratio		0.62	0.62	0.62	0.62	0.61	0.62	0.62	0.61	0.61	0.63	0.65	0.56

Notes:
1. Utility Crew operates Aux Loader, Aux Trucks, Rock Drill, Excavators, the extra Water Truck, etc.
2. VSA Basis: 10%

Table 16-13: Salaried Staff Labor Requirements

JOB TITLE	Annual Cost ($US)	-1	1	2	3	4	5	6	7	8	9	10	11
General Manager	199,372	1	1	1	1	1	1	1	1	1	0	0	0
Project Supervisor	44,009	1	1	1	1	1	1	1	1	1	1	1	0
Secretary (Admin. Assist.)	25,583	1	1	1	1	1	1	1	1	1	1	1	1
Total		3	3	3	3	3	3	3	3	3	2	2	1
MINE OPERATIONS:													
Operations Manager	135,880	1	1	1	1	1	1	1	1	1	1	1	1
Mine Manager	97,221	1	1	1	1	1	1	1	1	1	1	1	1
FL Supervisors	66,386	1	3	3	3	3	3	3	3	3	3	3	3
Drill & Blasting Supervisor	66,386	1	1	1	1	1	1	1	1	1	0	0	0
Mine Clerk	22,516	1	1	1	1	1	1	1	1	1	0	0	0
Mine Trainer	40,914	1	1	1	1	1	1	1	1	1	0	0	0
Safety Manager	104,453	1	1	1	1	1	1	1	1	1	1	0	0
Safety Pro.	31,796	1	1	1	1	1	1	1	1	1	1	0	0
Nurse	31,796	1	1	1	1	1	1	1	1	1	1	1	1
Ambulance/EMT	30,369	1	1	1	1	1	1	1	1	1	1	1	1
Mine Operations Total		10	12	12	12	12	12	12	12	12	9	7	7
MINE MAINTENANCE:													
Maintenance Manager	96,016	1	1	1	1	1	1	1	1	1	0	0	0
Purchasing Manager (General Foreman Mnt)	93,038	1	1	1	1	1	1	1	1	1	1	0	0
Warehouse supt. (FL Supervisors Mnt)	52,726	1	3	3	3	3	3	3	3	3	2	1	1
Purchasing Staff (Maintenance Planners)	28,341	1	1	1	1	1	1	1	1	1	0	0	0
Maintenance Trainer	40,914	1	1	1	1	1	1	1	1	1	0	0	0
Maintenance Clerk	22,516	1	1	1	1	1	1	1	1	0	0	0	0
Warehouse worker	10,786	1	1	1	1	1	1	1	1	1	1	1	1
Mine Maintenance Total		7	9	9	9	9	9	9	9	8	4	2	2
MINE ENGINEERING:													
Engineering Manger	105,660	1	1	1	1	1	1	1	1	1	1	0	0
Engineers	90,000	1	2	2	2	2	2	2	2	2	2	0	0
GIS	25,235	1	1	1	1	1	1	1	1	1	1	1	1
Sr. Surveyor	66,386	1	1	1	1	1	1	1	1	1	1	1	1
Surveyor	41,639	1	1	1	1	1	1	1	1	1	0	0	0
Surveyor Helper	15,654	1	1	1	1	1	1	1	1	1	1	1	1
Mine Clerk	22,516	1	1	1	1	1	1	1	1	1	0	0	0
Mine Engineering Total		7	8	8	8	8	8	8	8	8	6	3	3
ENVIRONMENTAL:													
Environmental Manager	108,503	1	1	1	1	1	1	1	1	1	1	1	1
Enviro. Suptt.	66,386	1	1	1	1	1	1	1	1	1	1	1	1
Enviro. Specialist	41,639	1	1	1	1	1	1	1	1	1	1	1	1
Enviro. Sampler	15,654	1	1	1	1	1	1	1	1	1	1	1	1
Enviro. Records management	15,654	1	1	1	1	1	1	1	1	1	1	1	1
Mine Environmental Total		5	5	5	5	5	5	5	5	5	5	5	5
MINE GEOLOGY:													
Geology Manager	135,880	1	1	1	1	1	1	1	1	1	0	0	0
Mine Geologist	80,000	1	1	1	1	1	1	1	1	1	0	0	0
Sr Geotechnical Engineer	80,000	1	1	1	1	1	1	1	1	1	0	0	0
Geotechnical Engineer	80,000	1	1	1	1	1	1	1	1	1	0	0	0
Sampler	15,654	2	2	2	2	2	2	2	2	2	1	1	1
Mine Geology Total		6	6	6	6	6	6	6	6	6	1	1	1
TOTAL PERSONNEL		33	38	38	38	38	38	38	38	37	22	15	14

Note: Annual Cost includes Fringe Benefits

16.3.7 Mine and Waste Storage Plans

Figure 16-3 through Figure 16-8 illustrate the mine plans along with the low-grade stockpile and waste storage plans from the end of pre-production through the ultimate configuration of the pits and dumps.

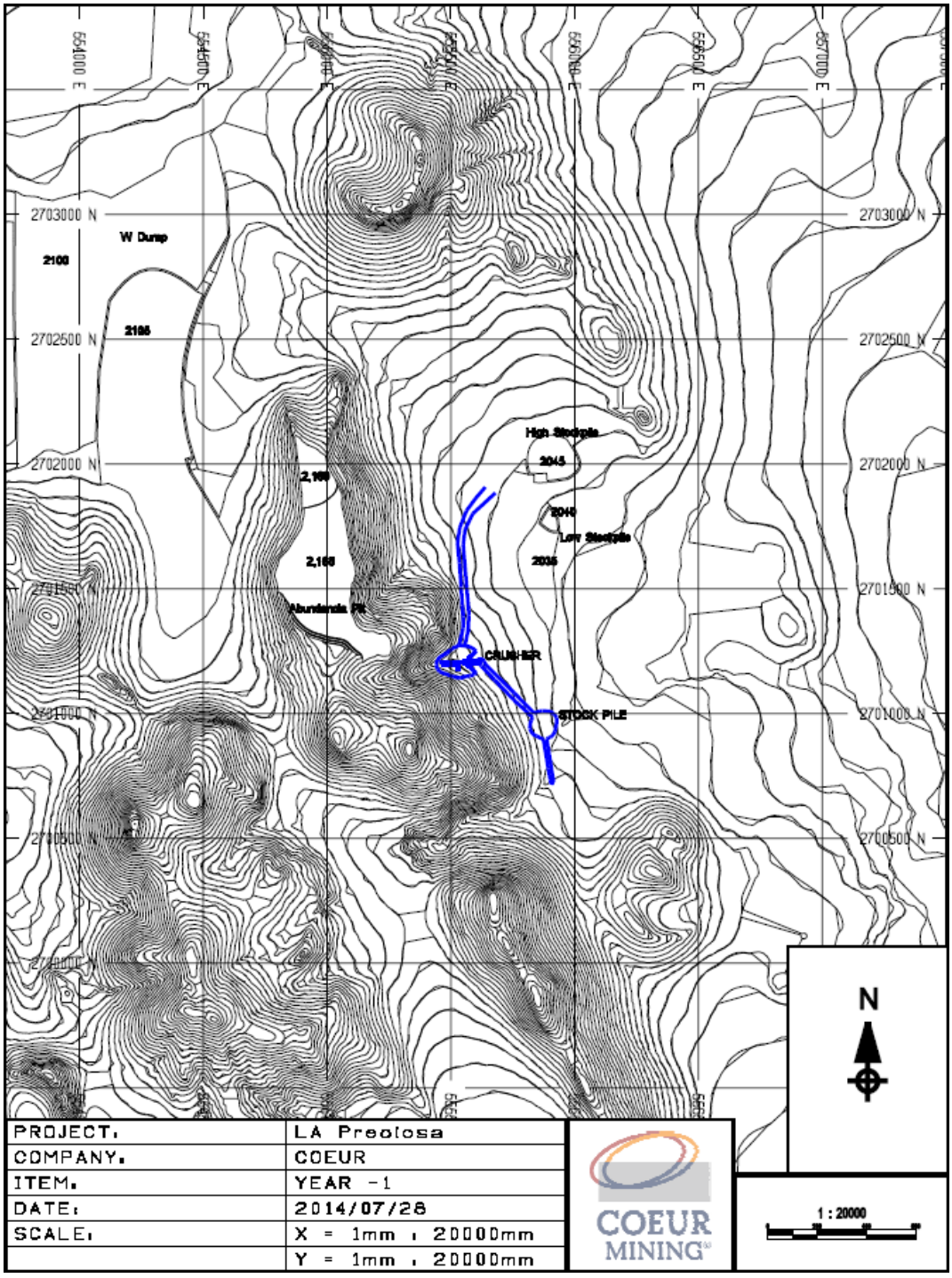


Figure 16-3: End of Pre-Production

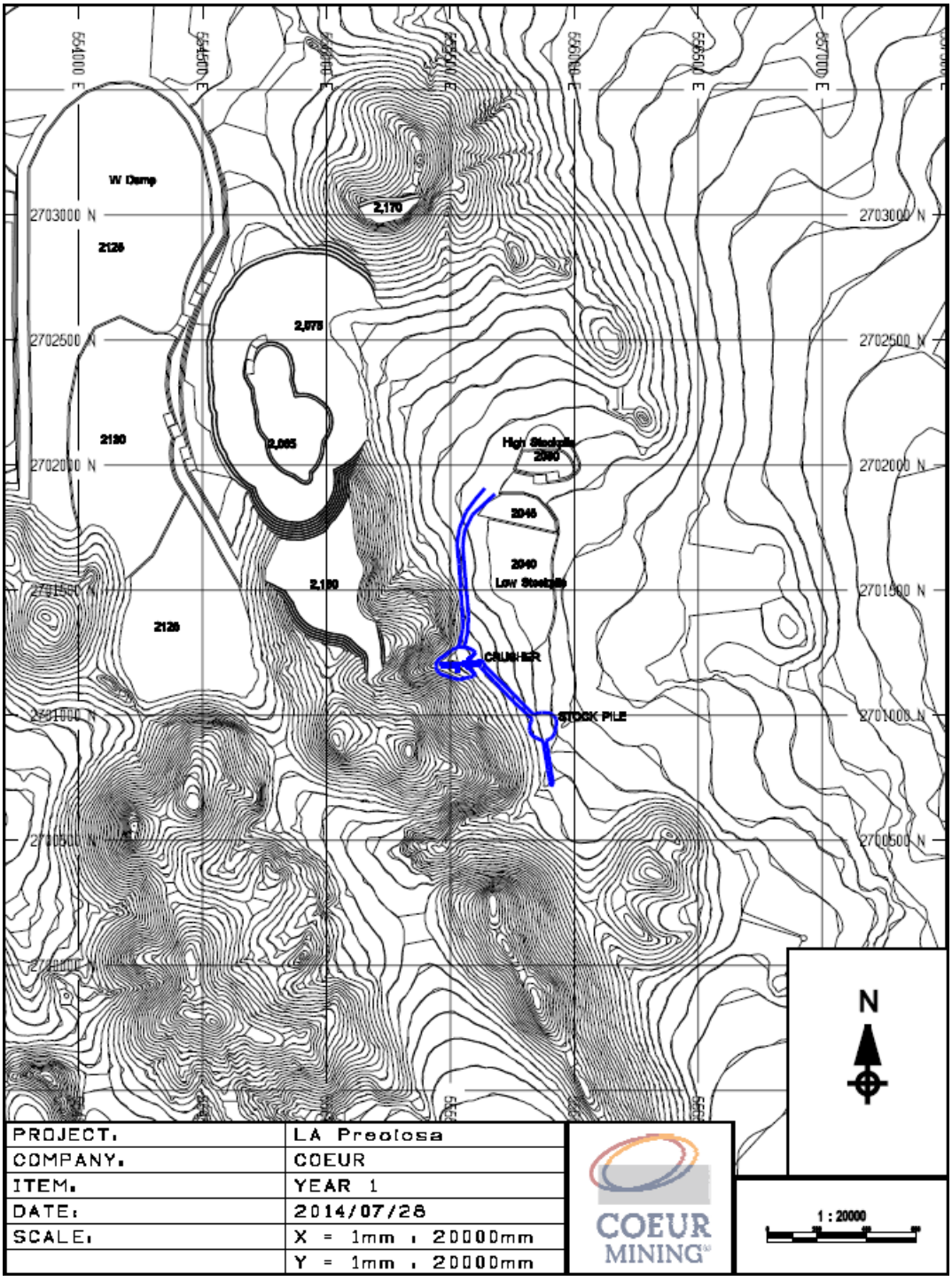


Figure 16-4: End of Year 1

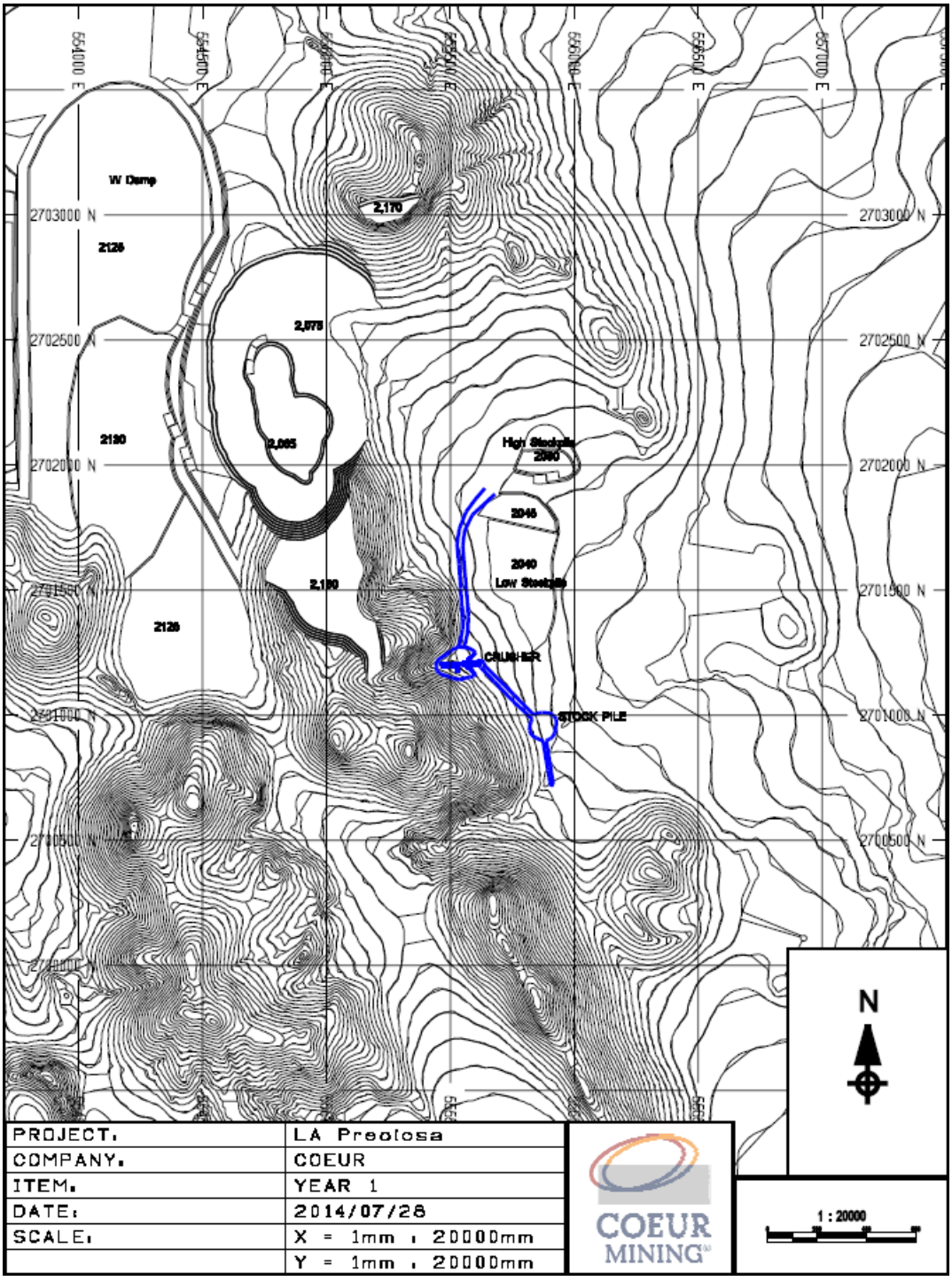


Figure 16-5: End of Year 3

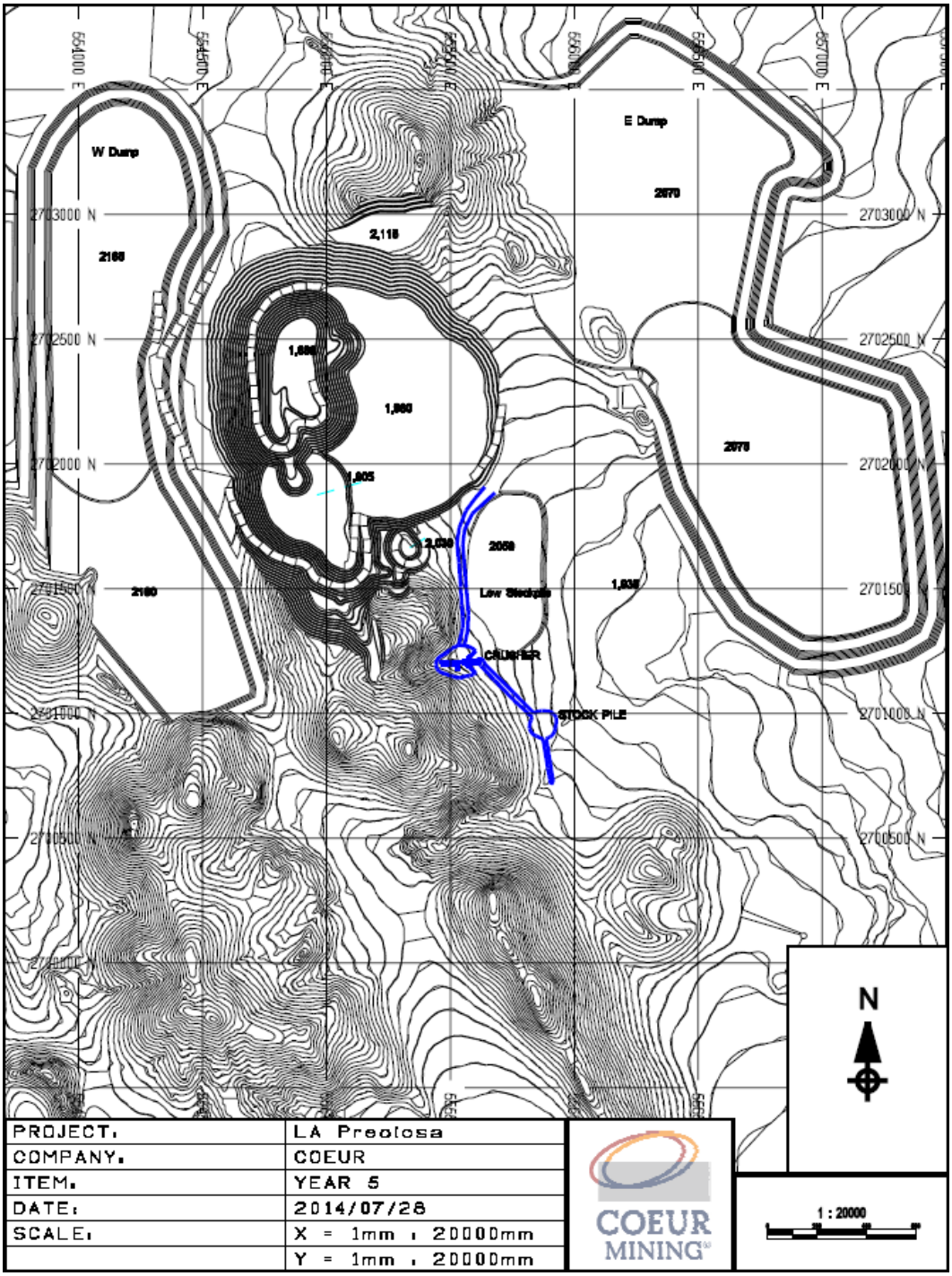


Figure 16-6: End of Year 5

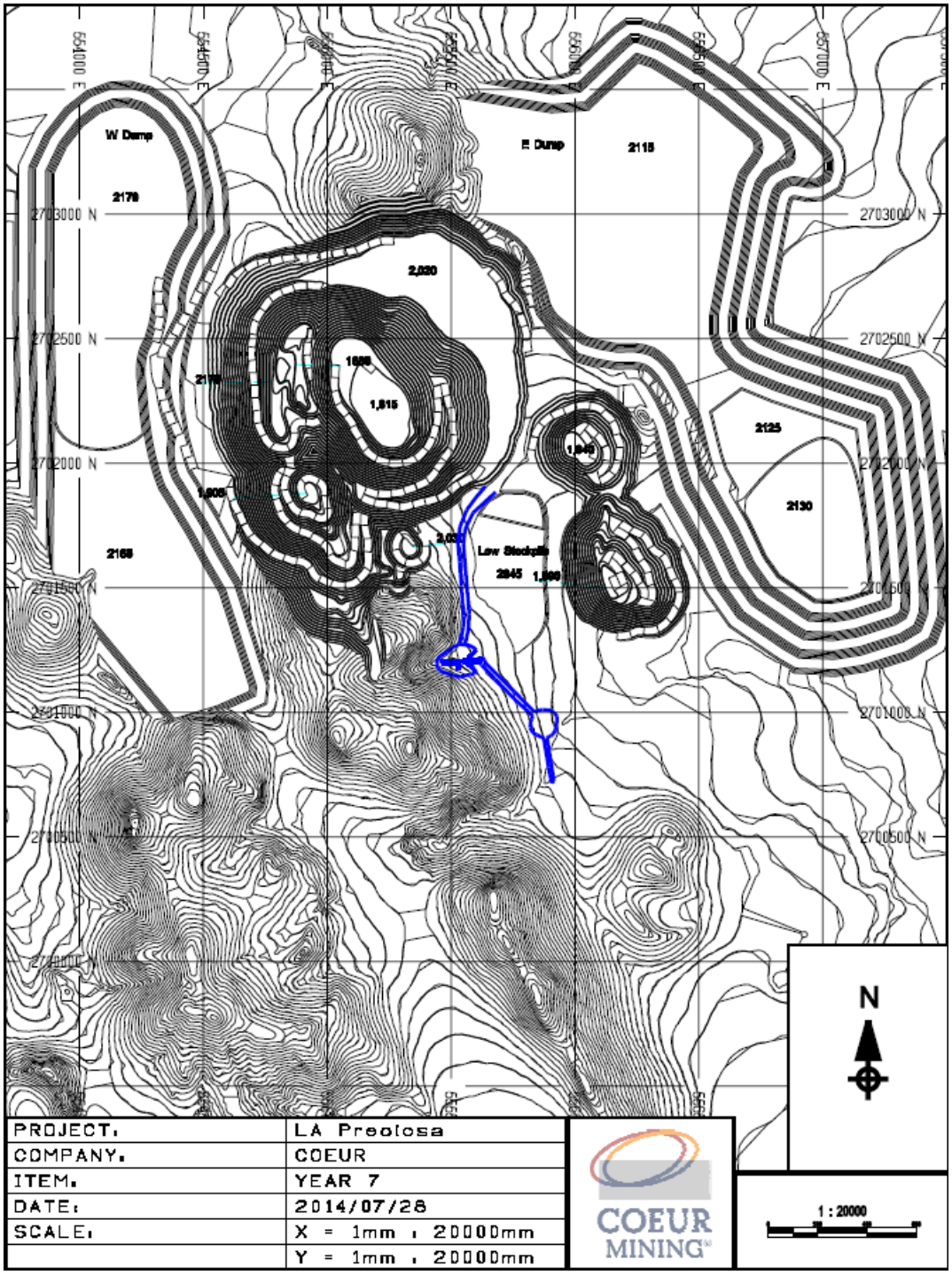


Figure 16-7: End of Year 7

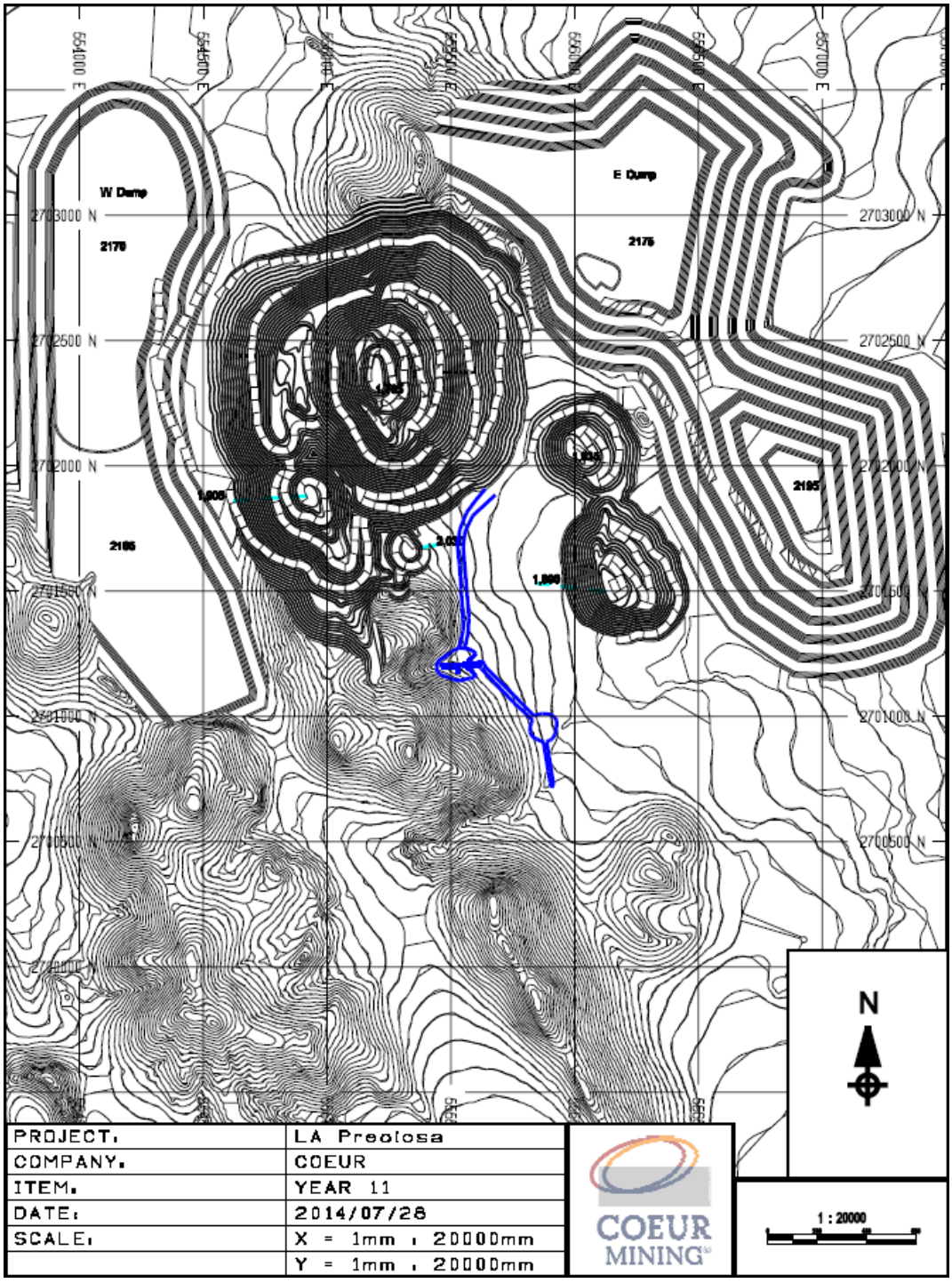


Figure 16-8: Final Pit – End of Mine Life

16.4 WASTE STORAGE DESIGNS

16.4.1 Dump Design Criteria (Geotechnical)

The west and east waste storage facilities (WSF) have been designed with these parameters. Both WSFs have been designed at 2.5 to 1 overall slope angles, 30 m lifts, at an overall slope angle which is angle of repose (37 degrees) and includes a 35 m catch bench. The WSFs have been designed to accommodate a 30-m wide haul road. The toes of the WSF's are approximately 25 m from the property boundaries. The WSF's have also been designed to respect the maximum LG pit. The WSF capacities and designs are shown in Table 16-14 and Figure 16-9, respectively. Dump designs have been redesigned since the PEA to now include a 30-meter setback from the pit rim and from property boundaries under Coeur control. This will allow for seepage collection structures at the toe of the dumps.

Table 16-14: Waste Dump Capacity

	Color	Capacity (ktonnes)
West WSF	Green	354,729
East WSF		
Base of WSF	Blue	343,767
Northern Top	Pink	95,287
Southern Top	Orange	33,064
Total		**851,889**

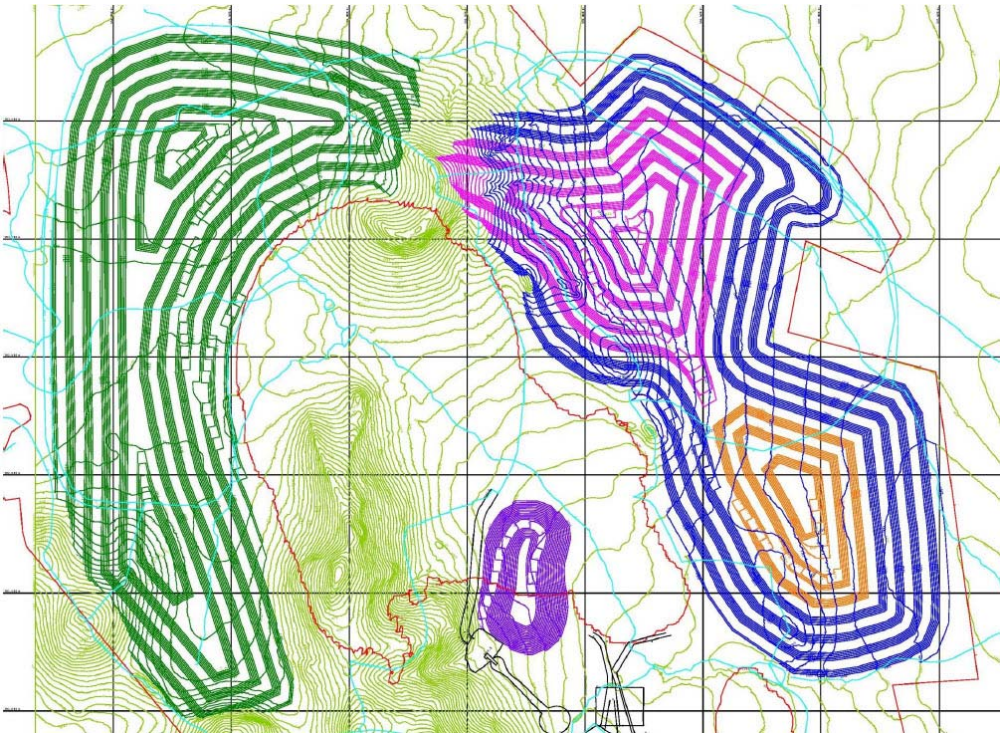


Figure 16-9: West and East Waste Storage Facility Design

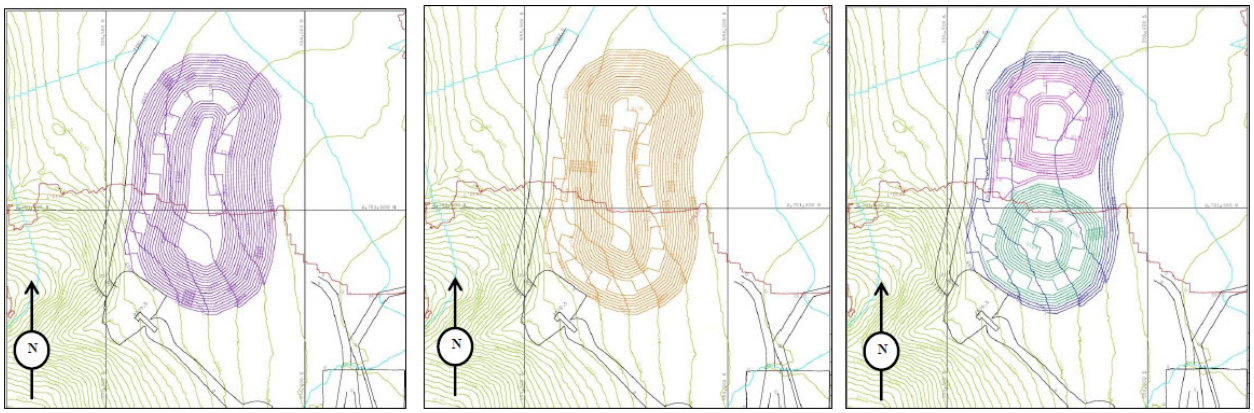
Three options have been developed for the Low-Grade Stockpile (LGS) (Table 16-15). The LGS is assumed to be built in single 5 m lifts at the angle of repose (37 degrees). The LGS has been designed to accommodate a 30 m wide haul road. The LGS access is from the crusher pad in all three options. Option 2 has dual access from the pit. The 3rd option includes an allocation for splitting the stockpile between high and low grade material. The low-grade stockpile

designs do not respect the maximum LG resource pit shell ($24.50 per ounce silver & $1,470 per ounce gold pit). If prices increase, the stockpiles will have to be consumed or relocated.

Table 16-15: Summary of Stockpile Design Options

	Color	Capacity (ktonnes)
Option 1 – South Entrance	Purple	20,135
Option 2 – N & S Entrance	Orange	18,580
Option 3 – Hi & Lo Grade Stockpiles		14,315
Base	Blue	7,183
Northern Top	Pink	3,712
Southern Top	Green	3,420

All waste storage and stockpile designs are based on a dry bank density of 2.5 tonnes per cubic meter and a swell factor of 30 percent.



Option 1 **Option 2** **Option 3**

Figure 16-10: Stockpile Design Options

16.4.2 ARD Considerations (If Applicable)

Geochemical testing of waste materials showed that 16% (6 of 37) of volcanic waste rock samples analyzed are acid-generating as defined by regulations published by SEMARNAT for characterization of mining wastes and hazardous wastes (NOM-157) (SRK, 2014K). Additional testwork will be required to assess the portion of the volcanic rock types in the future waste rock dump that will be acid generating and will need to be managed through selective handling, placement, and water management. No extraordinary waste rock management was assumed for the purposes of this study.

Numerous constituents in waste rock exceed the dry base maximum permissible limits of NOM-157. However, when subjected to the leaching analysis (PECT test), no analyses exceeded the maximum permissible limits for the PECT. By definition in NOM-157, none of the waste rock is therefore classified as “dangerous” due to metals toxicity.

16.4.3 Recommendations

Suggestions are to use an indicator to estimate the percentage of mineral within blocks. This mineral percentage can be used as a boundary to separate the data sets for estimation at some cutoff. Pit designs should be updated to reflect the latest geotechnical recommendations. When expedient, a bench through the scoria should be excavated to generate data to allow geotechnical design recommendations. Slope stability monitoring while mining initial phases

should also be planned using technology such as the use of reflectorless surveying or radar monitoring. This will allow the final slopes to be steepened opportunistically and reduce design conservatism.

In addition, a number of design changes were noted during the planning and costing exercise which should be implemented. These include:

- The discounting in the economic pit shells should be adjusted to account for the advance rates seen during this schedule.
- A method should be developed to allow for the Abundancia pit to be scheduled as two pits mining concurrently. This would allow an improvement in the production profile, by accessing ore at the bottom of phase 4 sooner, while still mining mill feed from phases 1 through 3. A potential solution may be available using linear programming.
- Pit ramps should be adjusted to allow for continuous access to adjacent phases, when possible, as the subsequent phases are mined. This will allow for backhaul of waste material to fill areas where mining has been completed.
- Haulage routes should be optimized to minimize material movements.
- Drill density in the Tajitos should be reviewed to more fully develop the resources in those pits.
- Review of mining costs, using haulage simulations, should be done to better estimate the incremental haulage cost on a bench-by-bench basis.
- The use of a geotechnical model could provide for a more optimum slope design.

Mine plans should be detailed on a month-by-month basis for the first 3 years, to match the long-range plans developed. A strategy should be developed that considers a protore cutoff for low-grade material and accounts for a storage location for that protore.

Finally, plans for drilling for ore control should be more developed to allow planning of the use of RC drilling.

16.5 GEOMECHANICAL – OPEN PIT

16.5.1 Introduction

Coeur is currently evaluating the feasibility of mining the deposit using several open pits:

- Abundancia Pit (the main pit, 460 m deep)
- North Satellite Pit (105 m deep)
- Central Satellite Pit (130 m deep)
- South Satellite Pit (115 m deep)

KP was retained by Coeur to complete the geomechanical and hydrogeological work needed to support feasibility level slope design for the Abundancia pit. Preliminary recommendations that were based on limited site investigations were also provided for the satellite pits. The completed work included:

- A review of all available geological and structural information
- A geomechanical and hydrogeological site investigation program
- Laboratory strength testing
- Characterization of the engineering properties of the encountered rock masses
- Slope stability analyses
- The development of slope recommendations for the final pit walls

This report summarizes the completed work and presents feasibility level final pit slope recommendations for the Abundancia pit.

16.5.2 Site Investigation Program

The geomechanical and hydrogeological site investigation program was designed to support the pit slope design of the proposed open pits. The site investigation program was completed between December 2013 and March 2014, and included the following:

- Six (6) oriented and triple-tubed geomechanical drillholes with associated detailed geomechanical logging and hydraulic conductivity (packer) testing
- Installation of multi-point vibrating wire piezometers in three (3) geomechanical drillholes
- Detailed geomechanical logging of intervals of full core from seven (7) exploration drillholes
- Underground line mapping at two (2) locations, and surface line and spot mapping at one (1) and two (2) locations, respectively
- Collection of 123 samples for laboratory strength testing. Unconfined Compressive Strength (UCS), Triaxial and Direct-Shear tests were completed
- On-site evaluation of the susceptibility of the rock mass to time-dependent degradation
- Borehole televiewer surveys of eight (8) exploration drillholes and one (1) geomechanical drillhole conducted by Groupe Qualitas Inc.
- Training Bufete Minero y Servicios de Ingenería, S.A. de C.V. geologists in drill supervision and Expromin S.C. geologists in detailed geomechanical logging

16.5.3 Rock Mass Characterization

The data collected from the geomechanical site investigations and laboratory strength testing allowed several geomechanical domains to be defined based on rock mass characteristics. Two possible domain definitions were considered and, ultimately, it was decided that domains would be defined by lithology. A summary of the rock mass characteristics for each domain is included below.

- **Scoria** - The characteristics and spatial extents of the Scoria are currently not well understood, as limited information is available for this unit. No recommendations have been provided for the Scoria.
- **Basalt** - This domain is characterized by an average UCS value of 120 MPa and a mi value of 13. The quality of the Basalt varies with the quantity of vesicles, but is generally GOOD quality rock with relatively short (< 5 m) intervals of reduced quality. The basalt has a RMR_{89} design value of 60.
- **Paleosoil** - The Paleosoil and the associated zone of paleo-weathering in the underlying Basalt or Volcanics is characterized by an average UCS of 25 MPa and is classified as POOR quality rock with a RMR_{89} design value of 30. A mi value was not estimated. Due to its limited thickness and flat orientation, the Paleosoil was not considered in the analyses.
- **Volcanics** - This domain is characterized by an average UCS of 80 MPa and a mi value of 10. The rock mass quality varies from FAIR to GOOD with a RMR_{89} design value of 55.
- **Sediments** - This domain is characterized by an average UCS of 55 MPa and a mi value of 9. This domain is classified as GOOD to VERY GOOD quality rock with a RMR_{89} design value of 70.
- **Metasediments** - This domain is characterized by an average UCS of 60 MPa and a mi value of 5. This domain is classified as FAIR to GOOD quality rock with a RMR_{89} design value of 50. The contact between the Metasediments and the Sediments is faulted and thought to have an increased susceptibility to time-dependent rock mass degradation.
- **Martha Vein and Hangingwall Zone** - This domain consists of the Martha Vein and a 20 m thick zone of reduced rock mass quality in the hangingwall of the upper vein. The domain is characterized by an average

UCS of 65 MPa. A mi value of 10 was assumed based on the results for the Volcanics and Sediments. This domain is classified as POOR quality rock with a RMR_{89} design value of 35.

- **Other Veins** - The Abundancia, La Gloria and Transversal veins are characterized by an average UCS of 65 MPa. A mi value of 10 was assumed based on the results for the Volcanics and Sediments. This domain is classified as POOR to FAIR quality rock with a RMR_{89} design value of 35.

The overburden is expected to form only a minor part of the proposed pit slopes and was not considered in the domain definition process and subsequent analyses.

16.5.4 Stability Analyses and Pit Slope Design

Based on the location and characteristics of the geomechanical domains and the pit design provided by Coeur, 11 design sectors were identified. Slope stability analyses were undertaken on each sector to define achievable slope configurations. These analyses included Kinematic and Limit-Equilibrium analyses. The results from these analyses provided guidance on achievable bench face, inter-ramp and overall slope angles for each design sector.

A slope design summary is included below:

- Bench Face Angle: 55 to 70°
- Bench Width: 6.5 to 10 m
- Bench Height: 10 to 15 m (double or triple benching 5 m benches)
- Inter Ramp Angle: 36 to 49° (for heights up to 150 m)
- Overall Slope Angle: 31 to 45°

16.5.5 Geomechanical Conclusions and Recommendations

The provided pit slope design recommendations are based upon the currently available geological, structural, geomechanical and hydrogeological data. The results of the analyses and a review of precedent practice suggest that the recommended geometries are reasonable and appropriate. To achieve these angles, the design assumes that controlled blasting and proactive geotechnical monitoring will be undertaken, along with an on-going commitment to geomechanical data collection and analyses. The results of the analyses suggest that groundwater depressurization is not likely to strongly influence overall slope stability; however, the phreatic surface that develops behind the pit walls should be carefully monitored and depressurization implemented on an as-needed basis. Maintaining flexibility in the mine plan will be important to accommodate any slope stability issues.

Future work should include a review of the updated pit design when it becomes available, as well as the work required to support detailed design. The detailed design level work is expected to include more detailed analyses based on additional and/or updated data for the deposit. It is anticipated that the current 3D lithological and/or structural models will be updated to incorporate the results of any additional exploration drilling and/or an improved understanding of the deposit geology.

Additional data requirements to be obtained through future site investigations (e.g., drilling, surface mapping, hydrogeological testing, laboratory testing, etc.) include:

- Characterization of the engineering properties of the Scoria through a focussed drilling program and/or the excavation of test trenches in the slopes of the cinder cone.
- Definition of the relative extents of the Scoria and Basalt through additional exploration/infill or geotechnical drilling, potentially supplemented by geophysical investigations.

- Confirmation of the lithologies and structures present behind the North wall of the Abundancia pit through additional exploration/infill drilling. The geological and structural models are currently extrapolated in this area.
- Additional characterization of the rock masses in the Northeast, East, Central (West), Southeast, West (South) and West (North) sectors where potential stability concerns have been identified. In these cases, the achievable pit slope geometry is sensitive to the discontinuity properties (persistence, spacing, strength and orientation).
- Additional characterization of the DB1_ENE_Fault, including defining the spatial extents and significance of its zone of influence.

Any further infill drilling at the Deposit should be reviewed for opportunities to provide additional information on the deposit rock masses and large-scale structures. Stripping and the initial stages of mining will also present opportunities for the collection of additional information.

The domain definition, stability analyses and slope recommendations should be updated to account for the results of the recommended site investigations and any changes to the geological models, large-scale structural interpretations and/or pit wall geometry. Additional analyses are also recommended to advance the slope recommendations to support detailed design. These analyses include:

- Further consideration of joint persistence and spacing and their influence on the results of the stability analyses
- Developing slope recommendations for the Scoria
- Reviewing the stability and configuration of the interim pit designs

The feasibility level recommendations have only been provided for the Abundancia pit. Geomechanical and hydrogeological site investigations, along with the associated analyses, will be required to advance the preliminary slope recommendations for the satellite pits to a similar level of design.

17 RECOVERY METHODS

The results of metallurgical testing have indicated that conventional cyanide leaching technology can be used to recover silver and gold from the Project feed material. The design basis for the feed processing facility is 10,000 dry metric tonnes per day (dtpd) or 3.65 million dry metric tonnes per year (dtpy). It is contemplated that feed will be transported from the mine to the concentrator facility by off-highway haulage trucks. The mineralized material would then be processed to produce silver and gold doré that would be loaded onto highway haul trucks and transported to a metal refinery.

The proposed process operations are summarized as follows:

- Crushing of the feed by primary jaw crusher to reduce the feed size from run of mine to minus 152 mm.
- Stockpiling primary crushed feed in a coarse feed stockpile and then reclaiming by feeders and conveyor belt.
- Grinding feed in a SAG mill / ball mill circuit prior to processing in a cyanide leach circuit. The SAG mill will operate in closed circuit with a vibrating screen. Crushed pebbles will be stacked on the ground for return to the SAG mill. The ball mill will operate in closed circuit with a hydro-cyclone to produce the desired grinding product size distribution of 80% (P_{80}) passing 74 microns.
- Leaching of the slurry at 40% solids by weight with cyanide solution in agitated leach tanks to dissolve the silver and gold contained in the slurry.
- Recovering soluble silver and gold using multi-stage counter current decantation technology (CCD). Barren solution will be added in the final CCD thickener as the washing solution.
- Clarifying of the pregnant solution followed by adding zinc dust to the solution to obtain a precipitate containing the recovered precious metals.
- Filtering and batch smelting of the precipitate to produce a silver and gold doré.
- Thickening of the leached tailing slurry and the recovery of the cyanide solution before detoxifying the thickened tailing slurry using oxygen and sulfur, with copper sulfate as a catalyst, prior to disposal in a tailings pond.
- Recycling water from the tailing pond for re-use in the process plant. Plant water stream types include barren solution, reclaim solution, fresh water, and potable water.
- Storing, preparing, and distributing reagents to be used in the process.

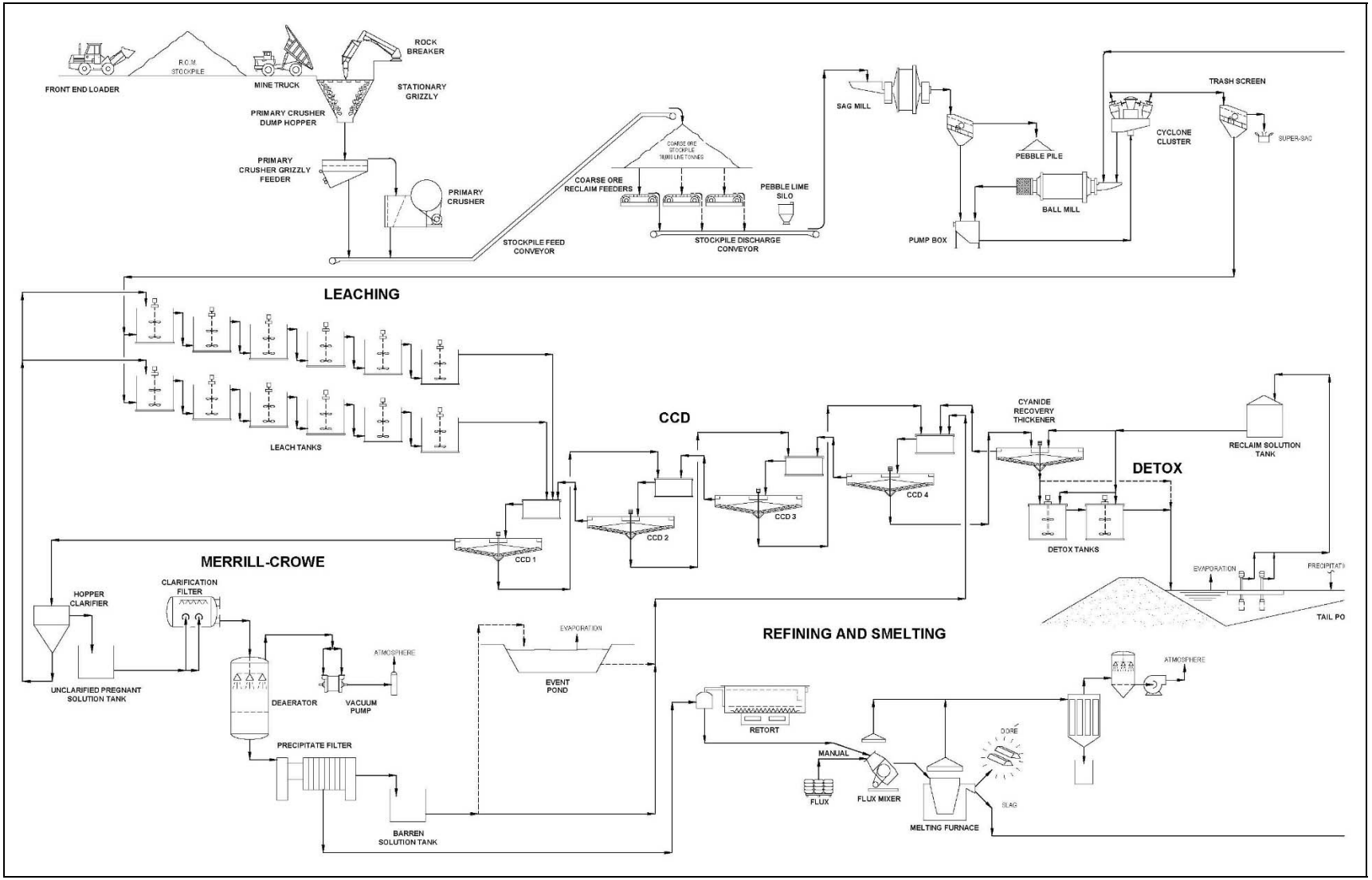
The overall process flow sheet is shown in Figure 17-1. Reagents to be used in the process and other main process consumable items are listed in Table 17-1 and the general design criteria used for equipment selection are identified in Table 17-2.

Table 17-1: Process Consumables

Item	kg/tonne feed
Reagents	
Anti-scalant	0.05
Copper sulfate	0.07
Diatomaceous earth	0.05
Flocculant	0.11
Flux	0.33
Lead nitrate	0.00
Lime (CaO)	1.50
Sodium cyanide	1.00
Sodium meta-bisulfite	0.85
Zinc dust	0.09
Grinding media	
Grinding balls, SAG mill	0.71
Grinding balls, ball mill	1.71
Utilities	
Fresh water	71.5 liters per second
Power	25.5 megawatts

Table 17-2: Process Design Criteria

Item and Unit	Value
General	
Feed Assay	
Gold (g/tonne)	0.174
Silver (g/tonne)	105.77
Feed Abrasion Index (Ai)	0.6362
Feed Work Index (kWh/tonne)	
Crushing, (CWi)	9.4
Rod Mill, (Cwi)	16.7
Ball Mill, (Bwi)	17.2
Feed Moisture Content (%)	
Design	3
Minimum	1
Maximum	5
Production Schedule	
Average (dmtpy)	3,650,000
Average (dmtpd)	10,000
Metal Production Schedule	
Silver Recovery (%)	84.0
Gold Recovery (%)	61.0
Primary Crushing	
Days per Year	365
Hours per Day	12
% Availability	75
Feed Crushing Rate (dmtph)	1,111.10
Crusher Feed, F_{80}, mm	500
Crusher Product, P_{80}, mm	152
Grinding	
Days per Year	365
Hours per Day	24
% Availability	92
Milling Rate (dmtph)	452.9
Primary Grinding	
Mode of Operation	Closed Circuit w/Vibrating Screen
Feed Size, microns	152,400
Transfer Size, microns	3,000
Pebble Circulating Load (%)	0
Secondary Grinding	
Mode of Operation	Closed Circuit w/Vibrating Screen
Product Size, microns	74
Recirculating Load (%)	300
CCD/Cyanide Recovery	
Days per Year	365
Hours per Day	24
% Availability (%)	92
Unit Area, $m^3/m^2/hr$	4.16 - 5.12
Slurry, % solids w/w, design	15
Cyanide Leach	
Days per Year	365
Hours per Day	24
% Availability	92
Tank Feed Rate (m^3/h)	866.5
Slurry, % solids w/w, design	40
Residence Time, hr., total	66
Mode of Operation	Series
Merrill-Crowe	
Days per Year	365
Hours per Day	24
% Availability (%)	96
Feed Rate (m^3/h)	2,185
Metal Recovery from Solution (%)	
Gold	99
Silver	99



(Source: M3, 2014)

Figure 17-1: Project Process Flow Sheet

18 PROJECT INFRASTRUCTURE

18.1 FACILITY LAYOUT

The main processing facilities for the Project are to be located in a compact arrangement to limit the amount of required earthworks. These facilities are shown in Figure 18-1 and include primary crushing, grinding, leaching, CCD thickeners, Merrill-Crowe processing, and refining facilities.

In addition to the main process facilities, it is contemplated that there will be several surface buildings constructed to support the mining and process operations. These facilities include administration, guardhouse, truck shop, warehouse, change house, explosives storage buildings, and truck wash and mill maintenance facilities.

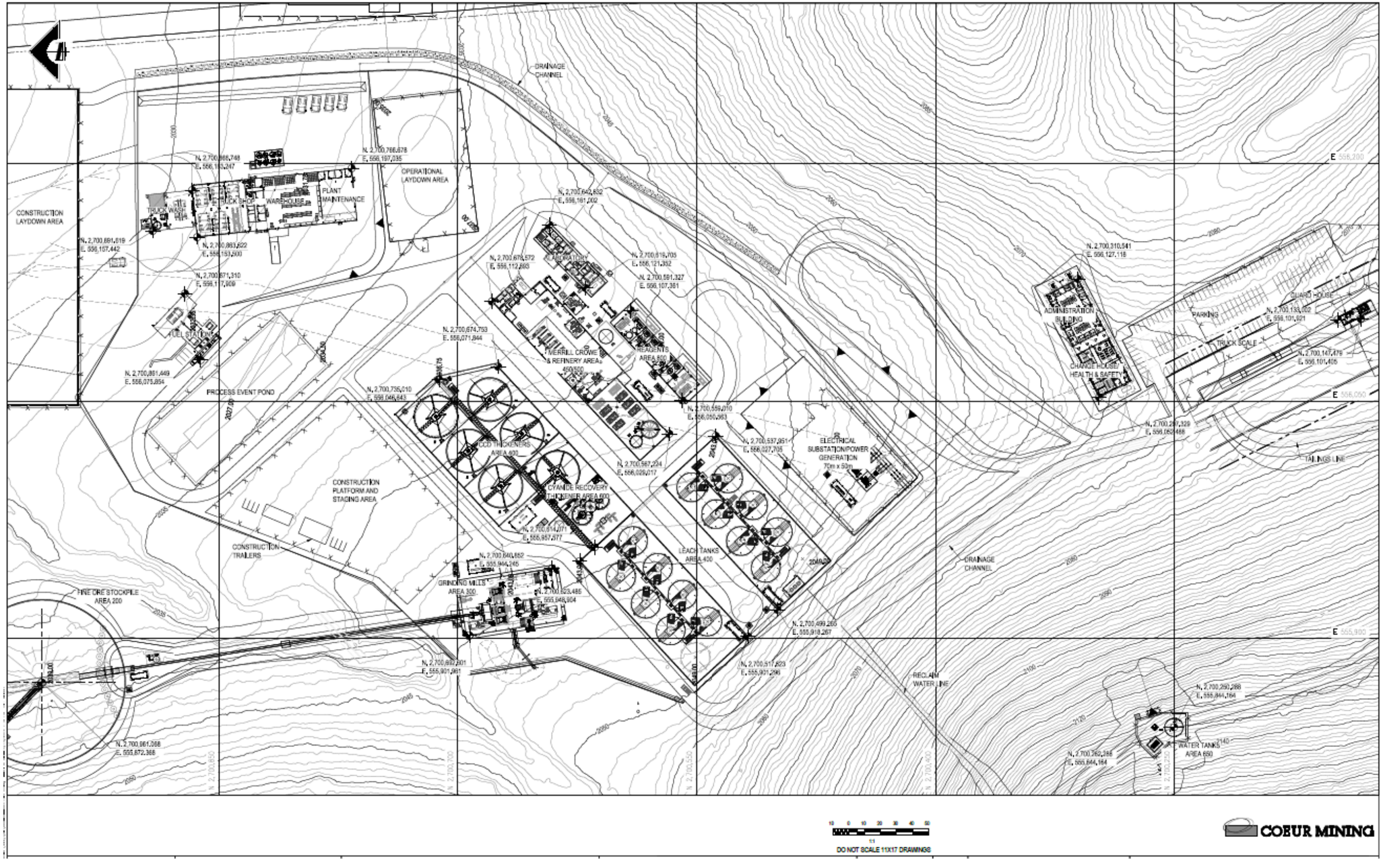


Figure 18-1: Project Processing Facility Plan

18.2 ACCESS TO SITE

The Project is located 85 km by road northeast of Durango and can be accessed by vehicle in approximately 90 minutes. Figure 18-1 shows the primary access by Mexican Federal Highway Route 40, an excellent all weather asphalt road connecting two major cities of Torreon to the Northwest, and Durango to the Southwest.

A new access road and utility corridor has been anticipated for the site, and is shown on Figure 18-2. This road will connect to the site from an existing exit off of Mexican Federal Highway Route 40 at the exit to the community of Vicente Suarez. This mine access road will use a portion of the existing community road and then deviate from that road to take a more direct route to the mine. Total road to be constructed is ~14 km of which ~12.5 km will be new construction and 1.1 km of the existing road will be improved. The final road will be about 65 km from Durango to the mine.

Due to heavy transport to the site during construction, the road is anticipated to be 10 m wide of driving surface, consisting of improved and compacted gravel, with drainage ditches. With exception of replacement or by-pass of one existing bridge on the portion of the existing road to Vicente Suarez, the remaining arroyo crossings are assumed to be “low water” crossings with concrete base.

Costs for this road are estimated to be $1.1 million.

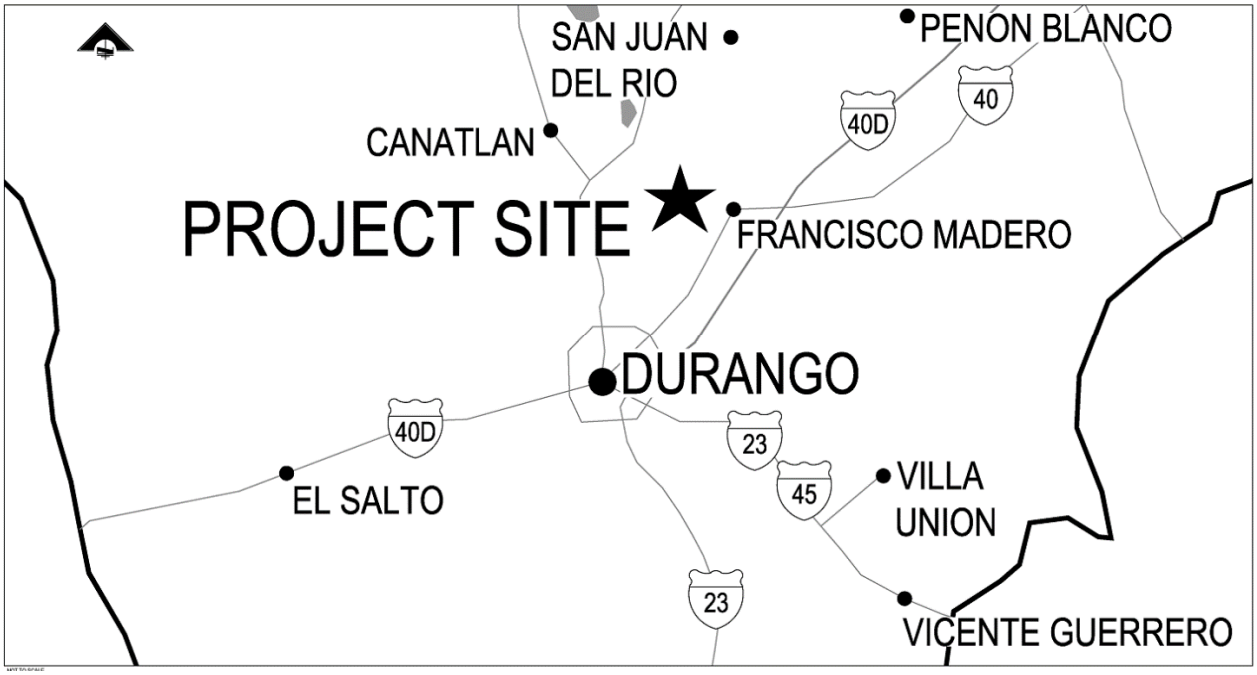
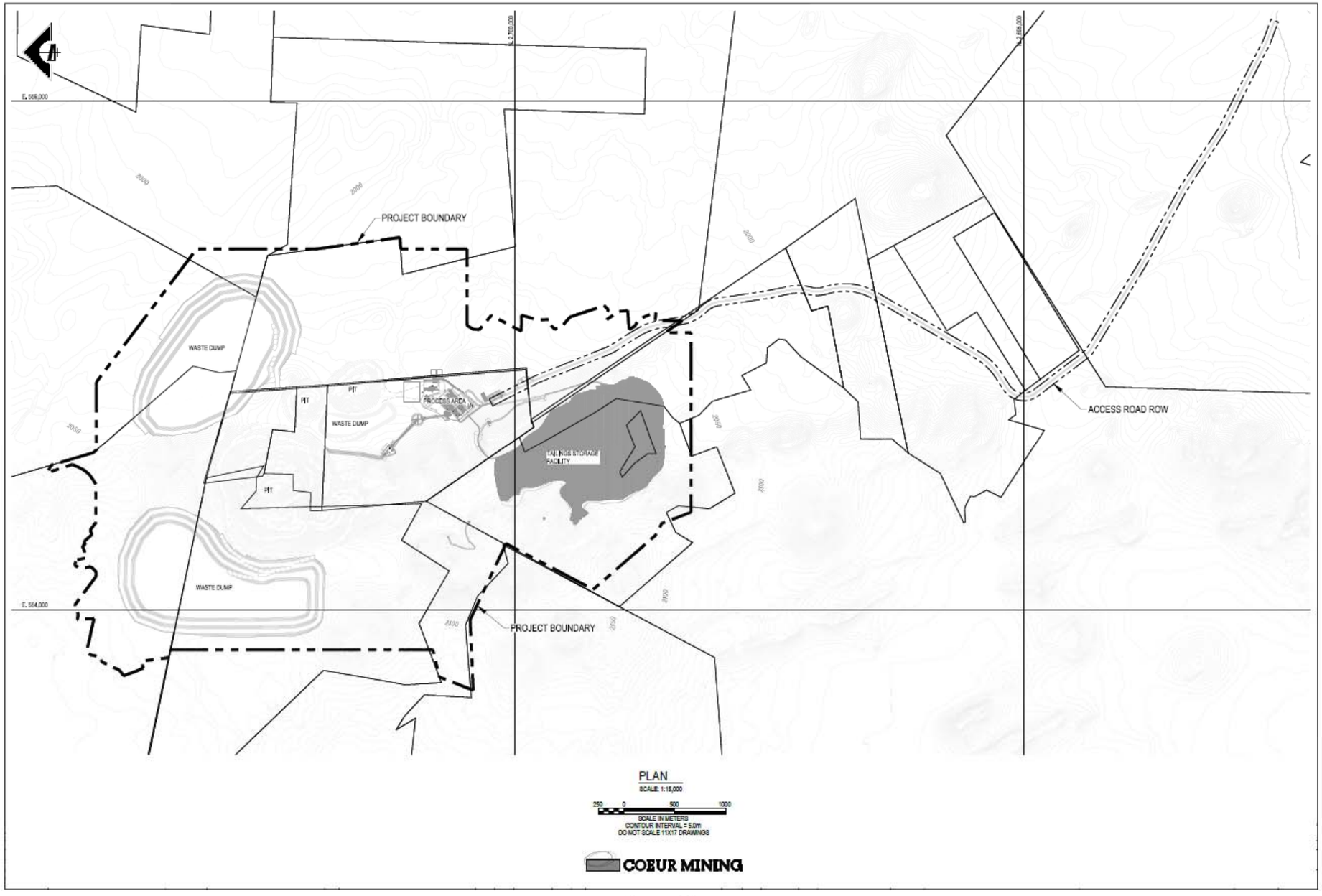


Figure 18-2: Proposed Access Route from Durango to the Project Site



(Source: M3)

Figure 18-3: Project Local Area Map

18.3 COMMUNICATIONS

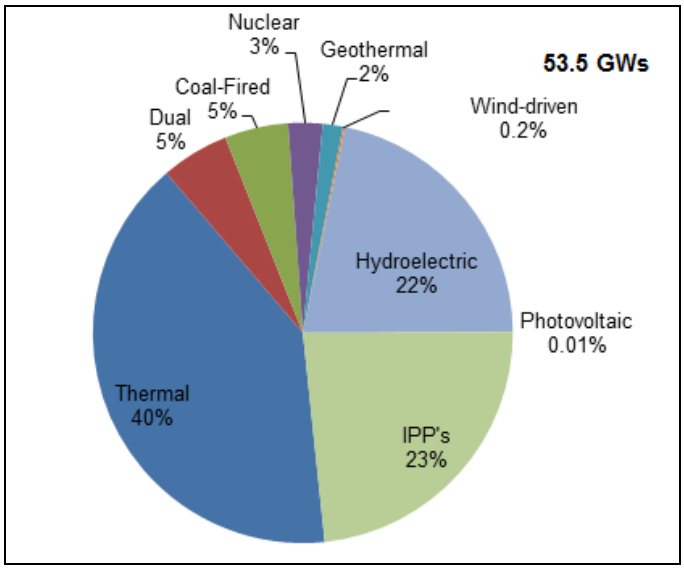
The site is currently equipped with a radio communications tower that belongs to the Project and has radio capacity to communicate back to the town of Durango. This radio tower was constructed by prior owners and consists of a dedicated 34.5-m high fenced tower operated by solar power, with multi frequency and repeater capacity. There is a second tower located on the roof of the Durango Coeur office building to assure 2-way communications to the site.

Costs for an internet and phone communications network were defined for the site and anticipate the construction of relay towers and a system of up to 30 Mbps can be constructed between the site and Durango for a cost of approximately US$135,000.

18.4 POWER

18.4.1 Power Supply and Regulatory Framework

According to the Mexican Secretary of Energy (SENER), Mexico had 53.5 gigawatts (GW) of effective generation capacity in 2013. Over three-quarters of power generation in Mexico is supplied by fossil fuels with natural gas thermal plants providing the largest component of power generation among the fossil fuels group (oil, natural gas, coal). Hydroelectricity represented 22% of installed capacity and provided 11% of energy in 2013 with production being highly dependent on annual rainfall. Non-hydro renewables (wind, solar, geothermal, biofuels) account for approximately 3% of power generation, and the country's sole nuclear power plant, Laguna Verde, has an installed capacity of 1.6 GW and contributes the remaining ~10% of generation in Mexico. Recently, through government support initiatives, Mexico is poised to become one of the fastest growing wind energy producers, with plans to add 3 GW of installed capacity by 2020.



Source: Pace Global (June 2014)

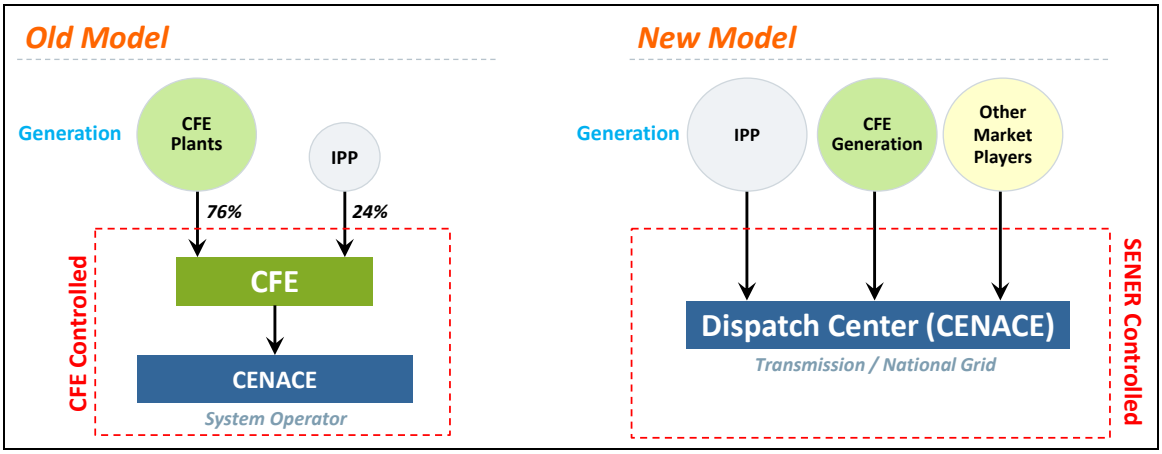
Figure 18-4: 2013 Mexico Power Generation Capacity

The state-owned utility, CFE, is the dominant player in the power sector, controlling over three-quarters of the country's installed generating capacity and controls all transmission and distribution of electricity in Mexico. In addition to the Public Sector (CFE), 29 units totaling 13 GW are owned by Independent Power Producers (IPPs). 33% of power generation in Mexico was from private entities as of 2013.

Mexico's electric power sector was partially liberalized in 1992-1993. The Energy Reform 2013 was published the 21st of December, 2013, and focuses on changes to Articles 27 and 28 of the Constitution with the following objectives:

- To reduce system costs and make headroom to lower the CFE tariffs
- To add renewable generation in large amounts to meet the legal regulations of the Climate Change and Renewable Laws

Article 27 is to be modified to create a competitive generation (capacity + energy) market and an Independent System Operator (ISO) to dispatch the fleet. The State maintains control of the power sector and ownership and operation of the transmission and distribution infrastructure through CFE, who will also be able to compete in the wholesale market. The below graphic illustrates the changes this Reform will have on the previous Mexican power market model.



Source: Siemens (June 2014)

Figure 18-5: Changes to the Mexican Power Market by the New Energy Reforms

The primary differentiators between the "Old" and "New" models are:

- PEMEX (state-owned petroleum company and natural gas supplier) and CFE will become "State Productive Entities" – commercially-oriented organizations coupled with subsequent tax reform that will allow them to operate on a for-profit basis and reinvest in its projects, rather than operating as a revenue unit of the federal government.
- National Center for Energy Control (CENACE) – will be an independent entity which controls the operations of the national electricity system and the electricity wholesale market.
- Private parties may participate with appropriate permits.
- Private parties may take part via contracts exclusively with CFE with no concessions issued.

Mexico's Congress will be reviewing and voting on the proposed secondary energy reform laws starting in June 2014, which further define the legislative detail governing the Reform and the final step towards passage of the law. This could result in a blackout period that could last as long as two years until the final regulations are passed.[9]

[9] It is important for all potential applicants for self-supply agreements applying under the existing scheme to initiate agreements before the laws are signed to avoid any permitting delays that may result from a potential blackout period.

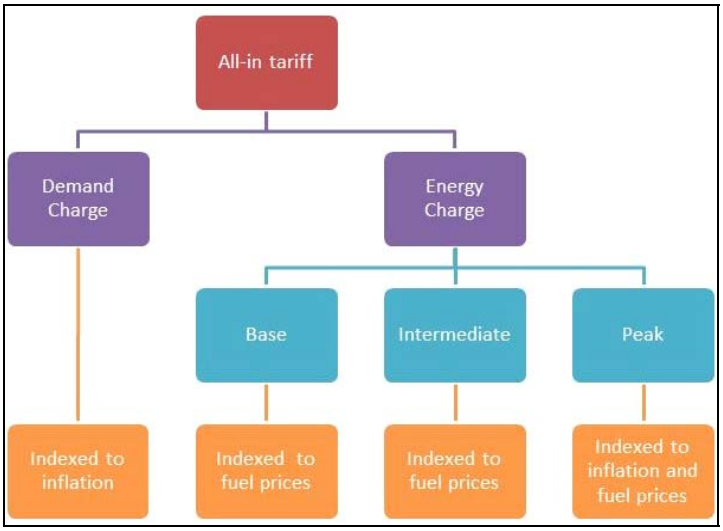
18.4.2 Power Market Considerations

18.4.2.1 Options to Commercially Available Power

Purchase Power from the CFE

CFE is a state-owned company which generates, transmits, and distributes electricity across Mexico. The grid, which is owned, operated, and maintained by CFE, stretches across the country, interconnecting all of Mexico with the exception of Baja California. CFE controls roughly 85% of all generation in Mexico.

CFE electricity rates are published tariffs, set by SENER. CFE offers two basic types of tariffs: 1) specific use tariffs, and 2) general tariffs. Specific tariffs are set for designated user types: residential, public service, agricultural, and temporary. General tariffs are set by voltage and region. All tariffs offered by CFE are full service, which include capacity, energy, transmission, and delivery services. Most medium voltage and all high voltage tariffs are time-of-use rates, with energy charges that vary by hour and season. All CFE tariffs are adjusted monthly to reflect changes in the cost of generation.



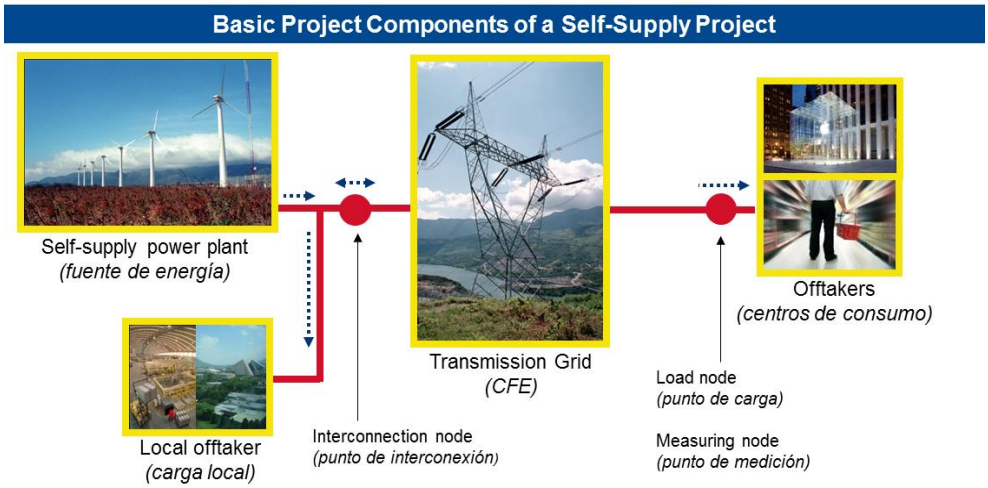
Source: Pace Global

Figure 18-6: CFE Tariff Adjustment Methodology for High Voltage Users

If the Project maintains its current plan to interconnect to the CFE transmission grid at 230 kV, the applicable tariff will be the H-T tariff for the North region. Based on a 90% load factor, the cost of CFE electricity in May 2014 would have been $108/MWh.

Purchase from Third Party via Self-Supply

Through the 1992-3 reform, a regulatory law permits private investment in specific types of generation such as IPPs and self-supply generation or cogeneration. IPP projects are private power projects which are developed exclusively to serve CFE load under long-term Power Purchase Agreements (PPAs). Self-supply projects are private power projects in which all the off-takers are shareholders, which enable sale of power directly to end-users.



Source: Pace Global (June 2014)

Figure 18-7: Self-Supply Projects Are Not Required to Be On-Site

Self-supply projects must receive permits from CFE and CRE in order to interconnect and operate on the transmission grid. Once approved, CFE is obligated to receive the power at the point of interconnection for the power plant and deliver the power to the load point(s). Typically, these projects are constructed only after PPAs are signed with the off-takers, which provide the banks the certainty they require to finance the project. The timeline for self-supply projects can range from one to three years, depending on project complexity and development progress. In addition to the PPA, three other contracts are required between the power project and CFE.

Table 18-1: Contracts Required for a Self-Supply Project

Interconnection Agreement	Sets the terms and conditions of the project interconnect to CFE's grid, describes location of injection and cascade schedule of the project's delivery points, metering equipment, invoicing and arbitration procedures.
Back-up Agreement	Sets the procedures, cost, metering, etc. by which CFE will provide back-up service and energy to the self-supply project (both for planned and forced outages).
Wheeling Agreement	Sets the procedures, cost, metering, etc. by which CFE will transport the electricity from the interconnection point to the load nodes of the consumer. Several wheeling options available: 5-yr renewable for normal and minimal charges, hedging option for normal and minimal charges.

Source: Pace Global

Current self-supply indicative pricing offers received show significant potential savings when compared to normal CFE high voltage tariffs. Below is a list of indicative pricing offers received for the Project. Prices are based on "all-in" cost, inclusive of any back-up and/or supplemental power required from CFE. The pricing structures offered fell into three categories:

1. "Cost Plus" – fixed monthly capacity charge + variable energy charge + transmission + back-up. Vendor's margin is typically included in the fixed capacity charge to achieve a certain return on investment in the asset.
2. Fixed Discount to CFE – a fixed percentage discount off the prevailing CFE H-S tariff
3. Fixed Price – fixed price of energy (minimum contract volume applies), escalating with inflation

A detailed analysis based on the prices quoted by six IPP's in formal proposals, indicates that power costs can range from a low of US$83/MWh to US$104/MWh and represent an "all-in" price including transportation charges. The pricing

and fee structures were presented under confidentiality agreements by several vendors, so specifics are not disclosed in this document. A comparable cost for power acquired from CFE is US$112/MWh.

Purchase Power from Third Party via On-Site Generation

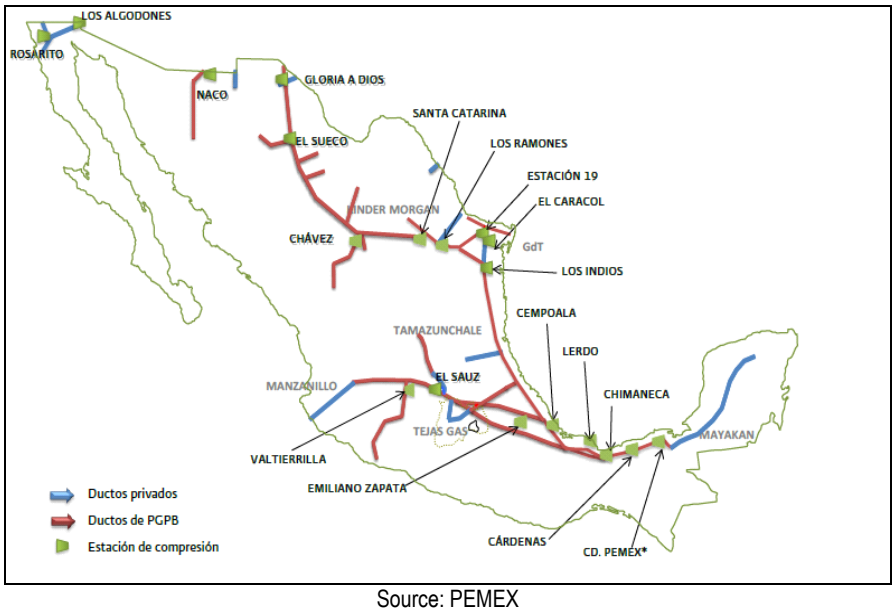
Self-supply regulations also permit on-site generation. In many cases, large industrials prefer not to focus on owning and operating power plants as it is outside of their core business and expertise. Instead, a self-supply project can be established on-site with a third-party developer who will finance, construct, own, and operate the plant, selling electricity to the industrial. Candidate sites for on-site generation require ready access to firm natural gas supply, water, and, in many cases, will rely on CFE grid interconnection for back-up power. If electrical grid interconnection is unfeasible or uneconomical, additional back-up capacity should be installed on-site to provide power during periods of scheduled or forced outages of generation units.

The Property location, northeast of Durango, is limited in its ability to secure firm pipeline natural gas supply due to the unavailability of firm capacity in the nearby pipeline. Other options for natural gas delivery, including Compressed Natural Gas (CNG) and Liquefied Natural Gas (LNG), could provide firm gas supply, but would result in an all-in power cost that exceed published CFE tariffs for electricity.

18.4.2.2 Pemex Natural Gas Pipeline

Availability

Mexico's state owned gas producer and supplier, PEMEX, owns and operates the bulk of the natural gas transmission infrastructure. Mexico currently imports 25% of its gas supply via pipeline from the U.S., primarily through south Texas. The rest of the natural gas is supplied from domestic production and 9% from LNG, imported through Manzanillo. La Preciosa is located near the end of PEMEX's 16" diameter pipeline which supplies all loads south of the Chavez compression station (between Torreon and Durango).



Source: PEMEX

Figure 18-8: Map of Existing Natural Gas Pipelines

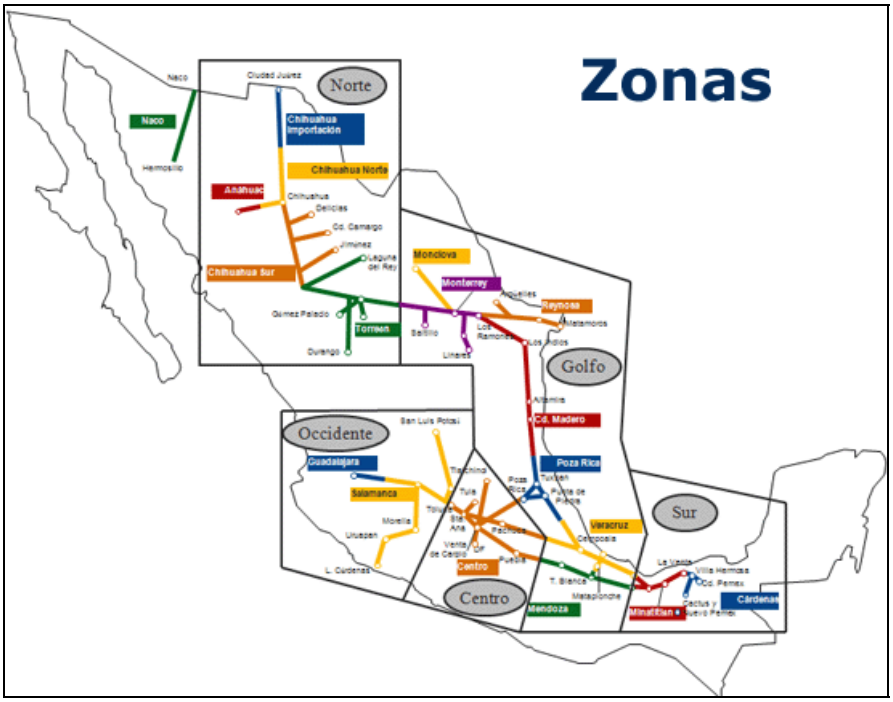
According to PEMEX officials, that particular section of pipe is highly constrained. The pipe has a total capacity of 92 million cubic feet per day (MMCFD) and currently serves the following loads:

- Norte I – power plant (firm capacity)
- Biopapel – paper mill
- Energas – currently produces/sells CNG
- Ecogas – local gas distributor in Durango (firm capacity)

There is no firm capacity available in the pipe at this time. Compression station upgrades could increase the firm capacity available on the Torreon-Durango pipeline, but Coeur would likely be the entity responsible for bearing the cost of these upgrades. The upgrades would increase capacity well beyond the needs for the Project, which would require coordinating partners (other gas users) to share in the expense of the upgrades, which would add to the development timeline and increase project complexity and risk. As an alternative, it is possible that a gas storage solution at the mine site could be used to convert the existing interruptible gas supply to firm/semi-firm service, though the option would carry its own risks and associated costs, including potential interconnections of new gas off-takers or increases in usage from existing users.

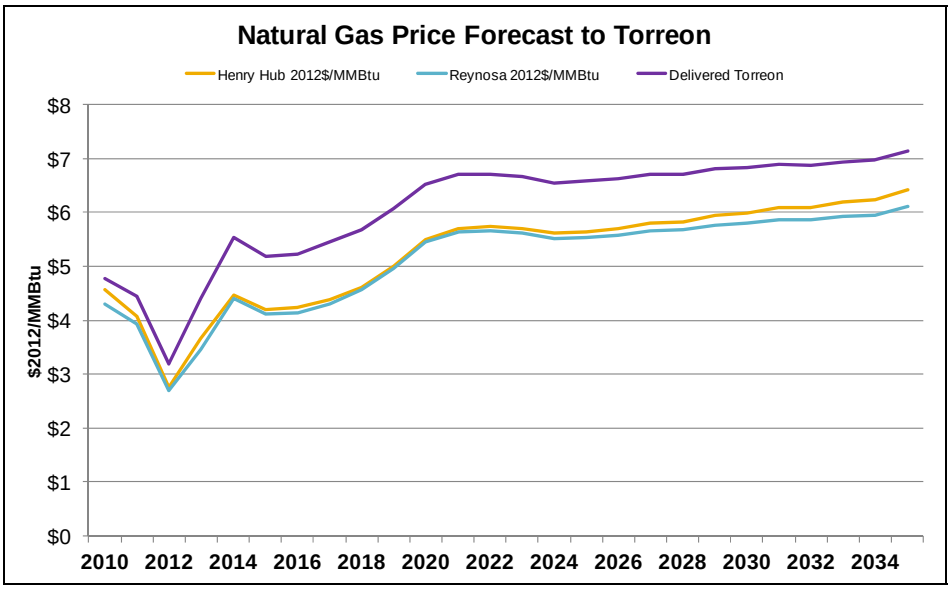
Pricing

Natural gas prices in Mexico are regulated by the CRE and are set by PEMEX based on the cost of imported gas from south Texas at Reynosa. First-hand gas sales are based on five distinct regions: Norte, Golfo, Occidente, Centro, and Sur. In addition to the zonal charges, all first-hand sales include an LNG premium which essentially socializes the higher cost of LNG imports across all gas users in Mexico. Under the energy reform, private participation in the natural gas Exploration & Production (E&P) and transportation will be allowed. This change will likely impact natural gas pricing structure across Mexico.



Source: PEMEX

Figure 18-9: Map of Existing Natural Gas Pipelines



Source: Pace Global

Figure 18-10: Historical and Forecasted Natural Gas Prices Delivered to Torreon

First-hand gas sales price formula uses a netback methodology from South Texas to determine the monthly and daily gas index prices at Reynosa and the PEMEX gas processing site located near Tabasco. To calculate the regional gas prices, the CRE issues postage transportation tariffs. Prices account for the gas commodity, transportation, service and distribution. Exhibit 12 above shows the calculated price of natural gas, delivered to Torreon based on this methodology and forecasted through 2035[10]. As shown in the chart, prices to Torreon are expected to soften in the near term to US$5.20 per MMBtu[11] in 2015 then increasing as high as US$6.70 per MMBtu by 2020.

Offsite Power Generation Utilizing Natural Gas

Because of limited pipeline capacity, an alternative, offsite option considered is power produced upstream of the Durango pipeline (for example at Torreon), and wheeled via transmission lines to the mine site was also considered as an option.

Pemex has confirmed the ability to deliver gas to the Project via Durango pipeline; but firm service would require multiple compression upgrades upstream from the power generating station, at a cost yet to be confirmed, but likely in excess of US$100 million.

18.4.3 Grid Capabilities

Near the mine site at about 21 km to the south east there is an existing power plant, natural gas fired combined cycle, owned by Union Fenosa. This power plant is capable of generating 450 MW, 400MW are committed and delivered to CFE via a 230 kV substation and its associated power lines. Besides the above mentioned power plant, the state of Durango is interconnected with CFE's national grid via their 230kV power lines.

The 230 kV line supplied by Union Fenosa passes within a straight line distance of about 9km from the mine site, roughly parallel to Route 40. Also, there is a 115 kV line at the same location and distance from the mine site. According

[10] Based on Pace Global forecast for natural gas supplied out of South Texas
[11] Million British thermal units

to CFE however, power is not available from this line because the conductors are too small and have insufficient capacity to supply mine power requirements.

A 115kV line at the village of Canatlán, at a distance of 42km from the mine site has sufficient capacity to provide for the mine's requirements.

18.4.4 Project Transmission Line

Three route options have been considered for this study:

1. CFE has proposed to provide power from the future Canatlán Potencia substation located about 42 km west-northwest from the site, Canatlán Potencia substation is planned to have a capacity of 100MW, and will step-down voltage level from 230 kV to 115 kV, from the substation a 42 km long, 115kV transmission line will feed the Project. A 115 kV power line could be constructed in wood poles or steel towers as favored by CFE with a 477 kCMIL size, ACSR-type conductor. CFE offered to build the transmission line in 34 months, including the new Canatlán Potencia substation. It is typical for mining projects for the client, or their contractor, to build the transmission lines and related substations and then turn them over to CFE for their operation and maintenance. The construction of any infrastructure that needs to be turned over to CFE, switching substation and transmission line will need to be inspected and accepted by CFE technical personnel during and after construction and commissioning of the facilities.
2. There is an existing 115 kV power line at a distance of about 9 km from the mine site. It is currently providing power to the city of Guadalupe Victoria. This option would require construction of a new branch, running to the mine site at a length of about 11 km. The power line does not have sufficient capacity to provide the power required by the mine, connecting to this power line it is not feasible since it does not have excess power capacity.
3. The 230 kV power line connected to the national grid is located at a distance of about 9 km from the mine site. Construction of a branch line with a length of about 11 km will be required. Positive discussions are continuing with CFE. This option will also require construction of a new switching substation nearby route 40. The new 230 kV transmission line to the Project will follow the Project new access road to the north. The line will need to be designed and constructed on steel tower with a minimum ACSR conductor size of 1113 circular mils (kcmil). Following the same logic as indicated in the first option, CFE may allow for a credit for the cost of construction to be applied against the rate in future billings.

Option 3 is preferred due to shortened construction, potentially lower capital cost, and lessened right of way (ROW) acquisition issues. Engineering could start once CFE approves final connection point and route is defined, design specifications and requirements will be based on CFE normalized standards based on substations and transmission line voltages.

18.4.5 Capital Cost

Table 18-2: 230kV Switching Substation near Route 40 –Electrical Infrastructure

Summary	
230kv Switching Substation	$ 3,520,700.00
New Power Line, 230kv, 11 Km	$ 2,529,938.14
New Project Substation	$ 5,364,900.00
Total Estimated Direct Cost For Power System	**$11,415,538.14**

Table 18-3: Canatlan –Electrical Infrastructure (CFE Approved Connection Point)

Summary With Steel Tower Structures (As Indicated By CFE)	
Canatlán Substation Modifications	$ 578,629.00
New Power Line, Steel Towers, 115kv, 42 Km	$ 8,376,492.87
New Project Substation	$ 4,455,241.20
Total Estimated Power System	**$13,410,363.07**

ROW acquisition and other indirect cost of constructing a new power line are not included in the above noted estimates.

18.4.6 Site Power Distribution

18.4.6.1 Substation and step down transformers

The Substation will consist of two 30 MVA 230 kV to 13.2 kV power transformers. Each transformer is sufficient to power the entire plant load allowing one transformer to be taken down for maintenance at any given time without affecting the plant operations. Step-down transformers from 13.2 kV to 4160V or 480V will be located at the individual process areas.

18.4.6.2 Site-wide power distribution

Site power distribution will be at 13.2 kV and will consist of both underground distribution and overhead lines. Underground distribution will be used to supply power to the main process areas near the substation while overhead lines will be used to provide power to the outlying areas such as primary crushing, stockpile, water tanks, and tailing. Wells and well water booster pumps will be fed by a 34.5 kV overhead line from a 13.2 kV-34.5 kV step-up transformer in the main substation.

18.4.7 Auxiliary Generating Capacity

Two 1000 KW generators will be located in the Project Substation to provide emergency power to critical loads.

18.5 WATER

A water source for the Project has been located in the Valle de Guadiana aquifer, and two wells were drilled to supply the anticipated ~80 liter per second (L/s) of plant and mine demand. Those two wells were drilled on farm land and all agreements are in place with the land-owners for the use of the land. Water rights have been obtained from the government agency CONAGUA or are in the process of being obtained.

Power for the two wells is anticipated to cost approximately US$180k and consist of extension of existing power lines that go to Vicente Suarez and also service other agricultural wells in the area. This will include posts, wire, transformers and control for the primary well pumps and the booster pumps.

A booster tank will be located so as to serve both the existing wells with an installation of two in-line booster pumps such that water can be pumped directly to the site.

Water pipeline design from the well booster tank to the site will cross easements with local landowners and the Vicente Suarez Ejido and will be constructed of steel in the lower portion and HDPE in the upper portion as pressure requirements drop.

A review of local mine dewatering needs, performed by KP, indicates that the processing plant will require use of the wells for water in the initial operation of the mine but that mine dewatering may be able to provide all required plant water in mid to late mine life.

18.6 TAILING STORAGE FACILITY

KP completed a feasibility level design for the TSF for the Project (KP, 2014). The TSF will provide permanent and secure tailings storage, in addition to water storage and control to ensure protection of the environment during operations and in the long-term (after closure). The design has been carried out based on the preferred TSF option as provided by Coeur.

The TSF site lies to the southwest of the proposed open pit and process plant. The ultimate configuration of the TSF is shown on Figure 18-11 with respect to the overall site layout.

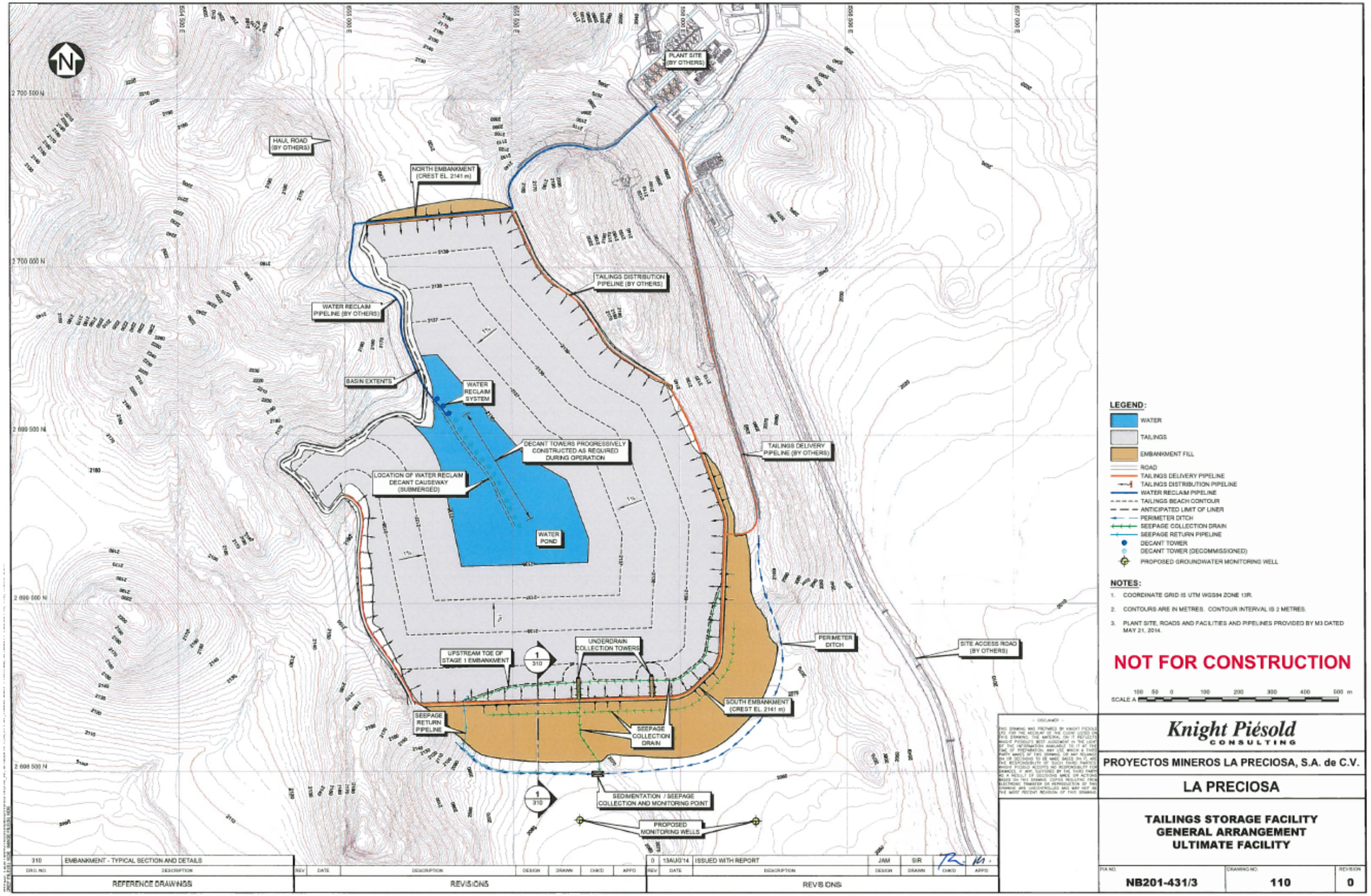


Figure 18-11: Tailings Storage Facility – General Arrangement- Ultimate Facility

18.6.1 Design Basis Overview

The principal objective of the TSF design is to provide tailings and water storage, ensure protection of the environment during operations and following closure, and to achieve effective reclamation at mine closure. The feasibility design of the TSF has taken into account the following requirements:

- Permanent, safe and secure long-term management of tailings and water
- Control, collection and decant of free draining water from the tailings during operations, for recycling as process water to the maximum extent practical
- The inclusion of appropriate monitoring strategies for all elements of the facility to ensure performance goals are achieved and that the design criteria and assumptions are met over the mine life
- Secure reclamation and closure of the impoundment after mining is complete

Embankment stability and design storm events chosen for the TSF are based on relevant criteria including Mexican standards (i.e. SEMARNAT NOM-141 Standards (2004)) and the guidelines of the Canadian Dam Association (CDA, 2007), including the following:

- Environmental Design Storm: 1 in 25 yr., 24-hour storm event
- Inflow Design Flood: 1/3 between 1/1000 and PMF, 24-hour storm event
- Minimum Freeboard Allowance: Site plots in “Humid Zone”. Minimum freeboard is 2 m
- Maximum Design Earthquake: 1 in 2,500 year (0.045 g)

The feasibility study design for the TSF is based on a projected 17-y mine life at a processing rate of approximately 10,000 dtpd. The size is due to performing the design work in parallel with mine engineering work. The life of mine storage capacity is 59.12 Mt in this design.

18.6.2 Tailings Physical Properties

Tailings samples were produced and subjected to geotechnical testing (KP, 2014). Based on this test work, the following tailings properties have been adopted for the feasibility design:

- Specific Gravity: 2.77
- Approximate Grain Size: 81.7% passing 74 µm (No. 200 sieve); SILT (69%), some sand (18%), some clay (13%)
- Average Settled Dry Density: 1.3 t/m^3

18.6.3 Tailings Management

The TSF includes the lined tailings basin for settlement of solids and supernatant water storage. Two (2) embankments will be constructed to establish the TSF, including a main embankment along the south side of the basin and a smaller embankment constructed later at the north side of the basin. Natural topographical containment will form the northeast and west sides of the TSF.

Tailings at 50% solids content (approximately, by weight) will be delivered via pump and pipeline from the plant site for storage at the TSF. Tailings will be deposited sub-aerially using multiple spigot offtakes at regular spaced intervals starting in the southwest corner and cycling around the south, east and north sides of the facility. Perimeter discharge from the embankments and natural topography will maintain the supernatant pond near the central west side of the tailings basin away from the embankments.

The TSF has been sized to permanently store approximately 45.5 million m^3 of tailings at an average settled dry density of 1.3 t/m^3.

18.6.4 Water Management

The Project is located where water resources are scarce and water supply can be expensive. In addition to minimizing the downstream impacts of the facility on the environment, the primary water management objective for the TSF is to collect all available water for re-use in the process in order to minimize make-up water requirements.

Supernatant water will be removed from the TSF via a decant tower system. Water recovered from the decant system will be pumped back to the plant for re-use in the process circuits. The operating pond will be maintained at the minimum volume required to provide adequate clarification prior to reclaim.

The design incorporates a tailings underdrain collection system comprising a series of perforated drainage pipes over the TSF basin floor. Free water draining from the tailings mass will be collected by the underdrain collection system and will report to reinforced concrete collection towers located upstream of the south embankment toe. Submersible pumps installed within these towers will return the water to the supernatant pond.

A monthly operational water balance was completed for the TSF and the results indicate that the TSF will operate at a net water deficit during all years of operations. Only a portion of the process water requirements can be satisfied by water reclaimed from the TSF. Additional make-up water, from other sources, is required in all years of operation.

In addition to strategic tailing deposition and minimizing operating pond size, a number of seepage reduction strategies have been incorporated into the design and operation of the facility, including the installation of the liner and underdrain collection system to collect any available free water above the liner. Additionally, seepage collection drains will be installed along the base of the south embankment.

Discharge from the embankments and basin perimeter will maintain the supernatant pond near the central west side of the tailings basin away from the embankments. A water reclaim system will maintain the pond with limited size. Under these conditions sufficient storage capacity will be provided within the TSF to store any tailings supernatant water, run off and the design storm events, in addition to an allowance for freeboard. Embankment construction will be staged and the construction schedule will ensure that there is always sufficient storage capacity available in the facility to avoid overtopping and negate the need for interim spillways for each embankment stage.

During the final year of operation, the supernatant pond will be 'pushed' toward the southwest corner of the facility by depositing tailings primarily from the east-northeast side of the facility. The closure spillway will be constructed as part of the last embankment stage, prior to closure, and will be designed to provide long-term passive water management for the closed facility. The permanent spillway will be constructed in such a manner as to allow rainfall run off from the surface of the rehabilitated facility to flow into the surrounding natural drainage system. The spillway has been sized to safely pass run off from the TSF catchment resulting from rainfall events up to and including the PMP.

18.6.5 Facility Construction

The TSF design section will include an initial starter embankment (Stage 1) for the south embankment with ongoing raises completed for the south and north embankments using centerline construction methods throughout the life of the facility. The initial starter embankment at the south, will be constructed of zoned rockfill with a composite liner on the upstream slope. The centerline raises will consist of zoned rockfill with a low permeability core zone. Transition zones will be established between the core zone and the embankment rockfill to ensure internal stability. The Stage 1 TSF will provide storage for two (2) years of tailings deposition. Construction of the north embankment will not be required until Year 9 of operations. The embankments will be extended and raised using rockfill from the mine over the life of the facility.

The design includes a composite liner consisting of a 60 mil HDPE geomembrane overlying a low permeability soil layer placed on the upstream slope of the starter embankment and over the entire basin footprint.

Design analyses completed for the dams, included seepage and stability analyses.

18.6.6 Monitoring

Monitoring of TSF operations will be active and ongoing to ensure optimum performance and allow refinement of the operation, as necessary. The achieved density of the tailings will be dependent on the tailings deposition practices and supernatant pond configuration. Field information on the in situ tailings densities will be collected during the initial months of operations for comparison with the design values. This information will allow further optimization of the tailings deposition strategy and water balance model, adjustments to the pond operating levels and, if necessary refinement to ongoing operations and future design. Other performance monitoring items include: groundwater monitoring well sampling and testing, analyzing piezometer levels, analyzing settlement gauge data, monitoring movement monuments, completing embankment surveys, water flow measurements.

18.7 SURFACE GEOTECHNICAL

Geotechnical investigations were completed by KP in 2014 to support the feasibility designs for the Project surface infrastructure. The detailed results of the geotechnical site investigations are summarized in KP's geotechnical investigation report (KP, 2014b).

The focus of the 2014 geotechnical site investigation program was to characterize the general soil and bedrock conditions within the vicinity of the proposed Plant Site and TSF.

18.7.1 Investigation Methods

The completed site investigations consisted of geotechnical drilling (soils and bedrock), test pit excavation, collection of representative samples, geotechnical logging, geomechanical logging, in situ testing and laboratory index testing.

18.7.1.1 Geotechnical Drilling

A total of 17 vertical drillholes were completed to depths ranging from 11.8 m to 37.30 m, with an average depth of 22.3 m. One additional drillhole with an azimuth of 90° and a dip of 50° was completed to a length of 185.3 m to evaluate if hydrogeological conditions and/or geologic structures are present that could convey seepage from the TSF to the Open Pit.

The geotechnical drilling, sampling and in situ testing was completed by Major using a diamond drill rig equipped with HQ3 diameter drill rods.

Each drillhole was advanced through the overburden by augering or coring and sampling via SPT until refusal. Advancement in bedrock was accomplished by casing and coring. The casing was advanced through the overburden and seated in the bedrock and the rock cored using an HQ3-diameter drill bit.

18.7.1.2 Test Pit Excavation

A total of 24 test pits were excavated during the site investigation program. Test pits varied in depth from 0.0 m (bedrock at surface) to 3.8 m (the maximum reach of the backhoe), with an average depth of approximately 2.0 m.

Test pits were excavated using a CAT 430E backhoe supplied and operated by Coeur. The excavation was advanced until refusal due to bedrock or the limits of the backhoe power and/or reach. The test pits were logged, sampled, and photographed prior to being backfilled.

18.7.1.3 Overburden Logging

Logging, sampling and testing of the overburden was undertaken in accordance with the Canadian Foundation Engineering Manual (Canadian Geotechnical Society, 2006) and the Standard Practice for Description Identification of Soils (ASTM, 2009).

Approximately 166 m of overburden in the proposed Plant Site and TSF areas was drilled or test pitted and characterized. The overburden was logged and photographed in the field as the samples were recovered (factual logging) from the drill core tube, split spoon or test pit walls.

Only samples considered to be representative of the in situ conditions were collected. Representative samples were placed in double plastic bags, sealed and labelled in preparation for future testing at a Mexican soils laboratory.

18.7.1.4 Bedrock Logging

Recovered bedrock drill core was placed in core boxes and labeled (drillhole identification, run identification and depths). KP personnel were present during bedrock drilling activities.

Bedrock core logging included estimates of key intact rock and discontinuity characteristics. The logging results were used to estimate the quality of the encountered rock masses using the Rock Mass Rating (RMR89) system (Bienawski, 1989). Rock mass characteristics were collected to support foundation design.

Approximately 263 m of bedrock was logged. The assigned lithologies are based on field descriptions and consultation with the Coeur geologists.

18.7.1.5 Standard Penetration Tests (SPTs)

SPTs were conducted within the overburden in all drillholes at an interval of approximately 0.75 m (2.5 ft.) starting from surface. The tests were completed in order to obtain relative density information and allow for collection of representative samples. The SPT tests were completed using a standard manual hammer and a standard split spoon sampler. The “standard penetration resistance” or the “N” value, reported in blows/foot was recorded during each test.

18.7.1.6 Packer Hydraulic Conductivity Testing

Hydraulic conductivity testing was completed using packers within the bedrock in the TSF drillholes. The packer testing was conducted by Major under the direction of KP personnel, using nitrogen or hydraulic inflatable packers. The tests were performed using the Lugeon or constant head method, typically isolating the bedrock zone as the drillhole was advanced. Note, groundwater was not encountered in any of the geotechnical drillholes and the packer testing was completed above the groundwater table, as such, these conductivities are relative estimates only and cannot be directly compared to conductivities measured below the groundwater table.

18.7.1.7 Laboratory Index Testing

Laboratory index testing was completed on select overburden samples from the drillholes and test pits. The purpose of the testing was to confirm general soil characteristics and to classify the various overburden materials encountered. Laboratory testing of the overburden was completed by Ingeniería Geotécnica del Norte, S.A. de C.V., in Durango, Mexico. The Laboratory test work included the following:

- Natural Moisture Content (ASTM D2216-10, 2010a)
- Particle Size Analysis of Soils using Sieve and Hydrometer (ASTM D422-63, 2007)
- Atterberg Limits (ASTM D4318-10, 2010b)

18.7.2 Site Investigation Results

A total of 18 drillholes and 24 test pits were completed to characterize the overburden and near surface bedrock conditions at the proposed Plant Site and TSF locations.

The following paragraphs provide a brief summary of the geotechnical site investigations findings.

Plant Site - A total of five drillholes and eight test pits were completed in the Plant Site area in order to evaluate the foundation conditions and to characterize the overburden and near surface bedrock. Main observations included:

- The depth to bedrock was relatively deep and typically ranged from 15 m to 20 m. The depth to bedrock at one drillhole location was shallow at 4 m.
- The overburden encountered in the area was generally found to comprise of nominal surficial layer of organic silt underlain by sand, with varying amounts of clay or gravel overlying bedrock. All of the soil samples collected from the Plant Site area, except for one, exhibited plasticity with plasticity index values ranging from 9.4% to 26.2%.
- Bedrock underlying the overburden at the Plant Site mainly consisted of andesite which ranged in quality from POOR to FAIR. Basalt was encountered at one drillhole location.

Tailings Storage Facility - A total of 12 drillholes and 16 test pits were completed in the TSF area in order to characterize the overburden and near-surface bedrock and evaluate the shallow bedrock hydraulic conductivity. Main observations included:

- The depth to bedrock was shallow (i.e. less than 5 m) in all the drillholes except on the west side of the TSF, where up to 15.5 m of overburden was locally encountered.
- The overburden encountered in the TSF area generally consisted of a nominal surficial layer of organic silt underlain by sand, with varying amounts of clay or gravel overlying bedrock. All of the soil samples collected from the TSF area exhibited plasticity, with plasticity index values ranging from 8.4% to 31%.
- The bedrock consisted of basaltic flows with a well-developed vesicular structure, dense basalt with a weak to moderately developed vesicular structure, scoria and andesite. The encountered rock mass qualities typically ranged from POOR to FAIR.
- A total of 23 packer tests were performed at 12 drillhole locations. Hydraulic conductivity values were generally high especially within the shallow bedrock. This likely is attributed to the well-developed vesicular structure in the basalt, broken and rubble zones. A small area in the southwest corner of the TSF that is underlain by andesite exhibited relatively low hydraulic conductivities. Intervals with no recovered bedrock core were encountered in drillholes at the north end of the facility potentially indicating the presence of a lava tube or similar structure (void).

Geologic Structure between TSF and Open Pit - One deeper, inclined drillhole was advanced into bedrock within the northern area of the TSF to evaluate if the hydrogeological conditions and/or geologic structures were present that could convey seepage from the TSF to the Open Pit. Main observations included:

- Five possible faults were identified in this drillhole with consistent north-south or northeast-southwest striking fault orientations similar to the fault orientations identified in the area of the Open Pits. Faults with these orientations could represent a potential conduit for seepage between the TSF and the Open Pit.

- The measured hydraulic conductivity of the encountered faults was generally one to two orders of magnitude higher than the deep host rock, but several orders of magnitude lower than the shallow bedrock. Given the expected hydraulic conductivities, it is likely any seepage from the TSF will flow through the highly permeable shallow vesicular basalt and not be conveyed to the Open Pit via any specific geologic structure.

18.7.3 Future Investigations

Additional site investigations are recommended to provide supporting data for future design studies. Any previously uninvestigated, new or modified infrastructure locations should also be investigated to provide the required data. The proposed site investigations may include the following:

- Completing test pits and drilling for the Waste Rock Dumps, Low Grade Ore Stockpile, the scoria deposit forming the North walls of the Abundancia pit and any other ancillary facilities associated with the Project to confirm subsurface foundation conditions and design assumptions.
- Completing the additional investigations needed to evaluate potential construction materials and borrow sources. Borrow material will be necessary for site grading, embankment construction and roads. Laboratory testing of fine grained soils should be performed to assess the suitability for low permeability fill.
- Completing the additional investigations needed to evaluate foundation conditions, with particular focus on the potential for large voids to be present in the near surface bedrock within the TSF basin and below the embankments.

18.8 WATER BALANCE

A site-wide water balance has been prepared to represent average steady state conditions during the life of mine (SRK, 2014g). The site-wide water balance summarizes water balance calculations completed for individual work areas (plant area, process circuit, mine area, TSF) prepared by Coeur and Coeur's consultants. SRK estimated stormwater run off, direct precipitation, and evaporation related to the plant and mine areas. M3 and other Coeur consultants estimated the steady state water balance calculations for the process circuit. KP prepared a variable and steady state water balance for the TSF (KP, 2014b) and estimated inflows to the Abundancia pit as the pit deepens (KP, 2014c).

For each identified area, the major water flow rates included stormwater run off, direct precipitation of ponds, well water for process circuit, TSF supernatant pond pump-back to reclaim solution tank, potable water needs, dust suppression, tailing slurry, entrained water in tailings, TSF seepage, groundwater pit inflow, and evaporation. The estimated water usage by area is presented in Table 18-4 and the steady state water balance flow rates are presented in Table 18-5.

The total average plant area water inflow rate is 81.7 cmh from run off and potable water and the average water outflow rate is estimated to be 2.6 cmh due to evaporation and septic leach field discharges to the subsurface. For the process circuit, the total average water inflow rate is 535.1 cmh based on well water and TSF reclaim water sources, and the total average water outflow rate is estimated to be 469.3 cmh with losses due to fresh water usage at the mine administration offices and shops, dust suppression, and TSF discharges. The total average TSF water inflow and outflow rates are balanced at 479.2 cmh. The water retained in tailings voids is estimated to be 170.1 cmh, the total average evaporation loss is 29.6 cmh, and a small volume of seepage is assumed (1.8 cmh). The total average TSF reclaim water, therefore, is 277.7 cmh so that the process circuit will require 138.9 cmh of fresh make-up water. The total average mine area water inflow rate is 324.7 cmh basin run off and groundwater, and the total average water outflow rate due to evaporation is estimated to be 2 cmh.

The results from the average steady-state water balance indicate that the Project is a net water user and will rely on the make-up water addition pumped from the well field, stormwater run off, and Abundancia Pit dewatering. The water balance for the plant area, process circuit, and TSF indicate an estimated water loss of 257.7 cmh. Fresh water demand from the water supply wells will be reduced slightly as make-up water from pit dewatering becomes available in

approximately Year 4. This inflow volume will increase as the pit deepens during the mine life and more water flows into the pit.

Table 18-4: Estimated Steady-State Water Usage

Plant Area	Quantity	Units
Plant drainage basin area	13.5	Ha
Plant run off coefficient (CN)	99	
Upgradient drainage basin area	119.6	Ha
Upgradient run off coefficient (CN)	70	
Estimated Plant basin storage	0.10	
Estimated Upgradient basin storage	4.29	
Yearly estimated Plant run off volume	71,780	m^3
Yearly estimated Upgradient run off volume	632,328	m^3
Event Pond Area	2,146	m^2
Yearly estimate Event Pond Precipitation	1,145	m^3
Yearly estimate Event Pond Evaporation	5,101	m^3
Stormwater Pond Area	3,600	m^2
Yearly estimate Stormwater Pond Precipitation	1,921	m^3
Yearly estimate Stormwater Pond Evaporation	8,557	m^3
Mine Admin Offices & Shops (Human Use)		
Number of people on site	30	people
Usage rate per person	0.4	m^3/day
Estimate usage	12	m^3/day
Mine Area		
Abundancia Pit drainage basin area	156.6	Ha
Abundancia Pit run off coefficient (CN)	100	
Upgradient drainage basin area	105.5	Ha
Upgradient run off coefficient (CN)	70	
Estimated Abundancia Pit basin storage	0.00	
Estimated Upgradient basin storage	4.29	
Yearly estimated Abundancia Pit run off volume	835,696	m^3
Yearly estimated Upgradient run off volume	557,611	m^3
Abundancia Pit pond area	7,384	m^2
Yearly estimate Abundancia Pit pond Evaporation	17,551	m^3

Source: Compiled by SRK, 2014

Table 18-5: Estimated Steady-State Water Balance Flow Rates

Process	Qty	Unit	Source
Well Water	257.4	m³/hr	M3
ROM Ore	34.4	m³/hr	M3
Reclaim Water	277.7	m³/hr	KP
Plant Area			
Mine Admin Offices & Shops[1]	1.0	m³/hr	SRK
Plant Area Run off[2]	8.2	m³/hr	M3
Upgradient Basin Run off[2]	72.2	m³/hr	M3
Event Pond Precipitation[2]	0.1	m³/hr	M3
Event Pond Evaporation[2]	0.6	m³/hr	M3
Stormwater Pond Precipitation[2]	0.2	m³/hr	M3
Stormwater Pond Evaporation[2]	1.0	m³/hr	M3
Septic Leach Field	1.0	m³/hr	SRK
Tailings Storage Facility			
Tailings Slurry Water (Inflow)	416.7	m³/hr	KP
Pond Precipitation[2]	6.7	m³/hr	KP
Pond Evaporation[2]	29.6	m³/hr	KP
Beach Run off[2]	7.0	m³/hr	KP
Run off from dry TSF area[2]	22.8	m³/hr	KP
Run off from within TSF[2]	16.2	m³/hr	KP
Upgradient Basin Run off[2]	9.8	m³/hr	KP
Entrained	170.1	m³/hr	KP
Seepage through liner	1.8	m³/hr	KP
Miscellaneous Water Usage			
Dust Suppression (Roads)	51.6	m³/hr	M3
Mine Area			
Abundancia Pit Run off[2]	95.4	m³/hr	M3
Upgradient Basin Run off[2]	63.7	m³/hr	M3
Average pit groundwater inflow (from Year 4-15)	165.6	m³/hr	KP
Abundancia Pit pond Evaporation	2.0	m³/hr	SRK

Source: Compiled by SRK, 2014; Notes: 1 = Based on 12-hr operations day. 2 = Based on yearly average volumes

19 MARKET STUDIES AND CONTRACTS

19.1 MARKET STUDIES

The Project will produce silver and gold doré, which will be trucked from the mine site to the refinery.

Coeur has received quotes for the secure transportation and refining of the silver-gold doré bars produced at the Project from parties with whom it has existing business relationships. Given the project's proximity to Coeur's Palmarejo mine, the rates for secure transportation are similar to what Coeur is currently paying its secure transportation provider in the region. The quote included in the financial analysis includes a fixed charge of $14,500 per shipment (up to 7,500 kg) plus a liability charge of $0.30 per thousand of amount declared.

Likewise, the doré bars that will be produced at Project are expected to be similar in grade and purity to those produced at Coeur's San Bartolome mine. As such, the refining charges quoted for the Project's production are similar to those currently quoted for the refining of San Bartolome's bars. Under the proposed contract terms, the refiner will credit the refined silver and gold to PMLP's account, at which point it will become available to sell. Coeur sells its silver and gold production on behalf of its subsidiaries on a spot or forward basis, primarily to banks and metal trading houses. After a sale is made, Coeur will send the refiner a bullion transfer order thereby directing the refiner to deliver the sold metal to the trade counterparty on the trade settlement date. The proceeds from said sales will be credited to PMLP's USD bank account upon delivery of the refined silver or gold bullion to the counterparty.

The expected precious metal and trace metal compositions of the doré are shown in Table 19-1 and Table 19-2, respectively. Refining terms include (i) a treatment charge of $0.17/oz based on the net weight of the doré bars received by the refinery; (ii) a metal return of 99.7% of recoverable gold; and (iii) 99.92% of recoverable silver. A 0.5% refining loss attributable to melt losses and differences in assays with the refiner was assumed. This is based on Coeur's refiner loss experience at Coeur's San Bartolome mine. No penalties are expected to be assessed against the refining. The refined silver and gold is expected to be available for Coeur to sell 8 business days after receipt of the doré bars by the refinery.

Table 19-1: Expected Dore Composition

Ag	99.87 wt%
Au	0.13 wt%

Table 19-2: Traces

Fe	0 - 0.1- 0.3 wt%
As	Trace
Sb	Trace
Hg	Trace

The estimated doré transportation charges and refining costs used for the financial analysis were based on quotes mentioned above.

Coeur plans to sell the refined silver and gold from the Project primarily to financial institutions, including multi-national banks and bullion trading houses. The markets for both refined silver and gold are highly liquid.

19.2 CONTRACTS

There are no established contracts for the sale of the doré currently in place for the Project. Coeur has a number of existing agreements with refiners in the United States and Europe. An agreement for the Project will be negotiated at the point the project is deemed to be approaching production.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Section 20 describes factors related to the Project and the preliminary results of environmental studies to support the permitting activities. The section summarizes the environmental issues identified on site, existing permit status, and environmental monitoring that is either in place or planned. Social and community programs are also discussed.

20.1 FACTORS RELATED TO THE PROJECT

This section addresses the general context of Mexican regulations related to mining law, land, environmental protection and management, and health and safety programs.

20.1.1 Mexican Mining Law

Under the Mexican Constitution Article 27, minerals are part of the national patrimony and are under the jurisdiction of the federal government. Mineral exploration and mining in Mexico are regulated by the Mining Law of June 26,1992, which established that all minerals found in Mexican territory are owned by the nation and that private parties may exploit these minerals (with the exception of oil and nuclear resources) through mining concessions granted by the federal government.

The right to explore for or exploit minerals is granted to private parties, through mining concessions issued by the Federal executive branch (Secretaría de Economía). Amendments to the Mining Law in April 2005, and put into effect in January 2006, provide for all mining concessions to be valid for a period of 50 years. A concession may be extended for an equal period on5 years advance notice prior to expiration and if no other cause for cancellation of the concession is evident. Amendments to the Mining Law were most recently approved by the Mexican Senate in October 2013 when mining tax reforms were passed, including a 0.5% tax on gross revenues of gold, silver, and platinum mines and a 7.5% tax on mining sales, minus certain deductions. These mining tax reforms received final approval by President Peña Nieto in December 2013 and went into effect on 1 January 2014.

A mining concession in Mexico does not confer any ownership of surface rights. Surface rights must be secured through land purchase or land lease; the legal documentation must be certified by the Agrarian Ministry. Mining concessions may only be granted to Mexican nationals and companies, ejidos, agrarian communities and communes, and indigenous communities. Eligible companies must be based in Mexico, and foreign participation in the ownership of such companies must comply with the Foreign Investment Law that allows companies to be owned by a foreign interest.

The mining concessions are held by PMLP, which is domiciled in México and owned by Coeur La Preciosa Silver Corp. (formerly, Orko Silver Corp.), a Canadian entity and La Preciosa Silver, S.A. de C.V., a Mexican entity, both of which are wholly owned by Coeur Mining, Inc. The bylaws and charter of PMLP includes the exploration or exploitation of minerals and substances that are subject to the Mining Law.

In accordance with a 2005 Mining Law amendment, there is no difference between exploration and exploitation mining concessions. The law allows owners of mining concessions to perform exploration activities with the purpose of identifying mineral deposits and quantifying and evaluating economically usable reserves and conducting work to prepare and develop areas containing mineral deposits; and to exploit the deposits (that is, mine the mineral products). Articles 29 and 39 from the Mining Law clearly stipulate that all mining concessions and projects must adhere to applicable regulations and permitting, including and not limited to labor, environment, explosives, etc.

The Mining Law requires the mining concession owner to:

- Start operations of exploration or exploitation 90 days following the recorded date of the mining concession, and incur and show evidence of certain minimum investments or obtain economically useful minerals;

- Pay mining concession fees (fiscal requirements include corporate income tax and value added tax, but no royalties);
- Comply with applicable regulations and standards, including but not limited to labor, safety, explosives, and the environment;
- Maintain permanent fortification works, shoring and other installations needed for stability and safety;
- Preserve landmarks;
- Provide the Secretary of Economy with statistical, technical and accounting reports;
- Allow inspections by the Secretary of the Economy;
- Provide the Secretary of the Economy with technical reports when the mining concession is cancelled;
- Provide the Mexican Geological Service with semi-annual reports on work and production if the concession is granted through a bid process; and
- File annual reports detailing production statistics for the previous calendar year to the Secretary of the Economy.

20.1.2 Key Mexican Environmental Statutes and Regulations

At the federal level, the key environmental statutes and regulations are under the following laws:

- General Law for the Ecological Equilibrium and Environmental Protection and its Regulations in various matters (Ley General del Equilibrio Ecológico y la Protección al Ambiente (LGEEPA));
- General Law for the Prevention and Integral Management of Waste and its Regulations (Ley General para la Prevención y Gestión Integral de los Residuos);
- Law of National Waters and its Regulations (Ley de Aguas Nacionales);
- General Law for Forestry Sustainable Development and its Regulations (Ley General de Desarrollo Forestal Sustentable);
- General Law of Wildlife and its Regulations (Ley General de Vida Silvestre);
- General Law of Climate Change (Ley General de Cambio Climático);
- Federal Law of Environmental Liability (Ley Federal de Responsabilidad Ambiental (LFRA));
- Mexican Official Standards, which contain technical provisions in environmental matters (Normas Oficiales Mexicanas (NOMs)).

Environmental regulations, compliance criteria, and other environmental guidance related to mining operations are promulgated primarily on a federal level by the Ministry of Environment and Natural Resources Secretaría de Medio Ambiente y Recursos Naturales (SEMARNAT).

The authorization to disturb the environment is through an Environmental Impact and Risk Authorization, change-of-use permit for forest land, or other comparable authorization by SEMARNAT. The Environmental Impact Evaluation is performed by SEMARNAT upon receipt of an Environmental Impact Statement [Manifestación de Impacto Ambiental (MIA)] study prepared by the mining company. The MIA contains property ownership and land/legal information, technical information about the pre-mine ambient conditions related to the environment, a description of the environmental impacts and disturbance that will result from the mining and processing operations, and mitigation plans presented by the company to reduce impacts to the environment. The MIA is submitted along with the Technical Study of Forest Land use Modification (ETJ) and a Risk Assessment (ER) when the Project involves hazardous activities, such as the storage and handling of cyanide.

As established by the federal environmental regulations, when naturally occurring vegetation or natural land is totally or partially cleared in order to use such land for non-forestry activities, the Project developer is required to obtain a Forest Land-Use Change from SEMARNAT. Important factors to note about the handling of the submitted ETJ document to request the authorization for a change in forest land use include the following:

- ETJ environmental documents are denied by default and only those projects that justify an exception are authorized;
- The ETJ authorization involves a compensation fee, currently at $1,750 USD per hectare;
- The ETJ authorization will most likely result in high compensation measures to keep an adequate balance between forest clearing and environmental services (i.e. reforestation in areas around or near the Project site will be required);
- SEMARNAT requests a technical opinion from the State Forest Council; and
- The required legal documentation for ejido land must be certified by the Agrarian Ministry. The General Ejido Assembly grants permission for the Forest Land Use Modification to Mining. The Ejido contract, leases, occupation, and expropriation must be certified by the Agrarian Ministry.

The MIA process in Mexico is maturing towards a more stringent permitting policy with increasing compliance criteria, especially for high profile projects involving open pit mining, use of cyanide, and deforestation. The MIA submittal and resolution process involves the following:

- Meeting a number of environmental conditions and requirements for pre-construction, construction, operation, closure, and post-closure activities;
- Submission of periodic compliance reports;
- Proof of a financial instrument to ensure full compliance to environmental conditions (environmental insurance, environmental bond, environmental trust fund, or other viable instruments);
- Continuous verifications (i.e. site visits, audits) by the Federal Environmental Protection Agency [Procuraduría Federal de Protección al Ambiente (PROFEPA)]; and
- Preparation and periodic update of specific management, monitoring, and training programs.

Provisions of LFRA are designed to protect, preserve, and restore the environment and ecological equilibrium in order to guarantee the human right to a healthy environment for all individuals. LFRA defines the concept of environmental damage and the responsibility of corporate entities causing that damage, exemptions, and liabilities related to illicit acts or omissions, and compensation requirements when damage cannot be restored to the original state. A suit can be filed by (1) inhabitants of the community adjacent to an area where environmental damage has occurred, (2) Mexican non-profit organizations representing a community inhabitant, (3) PROFEPA or (4) a state or federal environmental agency or institution within their territorial jurisdiction working in partnership with PROFEPA. In addition to repairs, fines can be levied against individuals and corporate entities. LFRA defines the judicial process for civil claims, makes a provision for an Environmental Liability Fund [Fondo de Responsabilidad Ambiental (FORA)] and alternative mechanisms of resolving disputes. LFRA creates new Environmental Judicial Courts that will begin operation as of 7 July 2015.

Several of the specific environmental standards and regulations [Normas Oficiales Mexicanas (NOMs)] that apply to mining and beneficiation operations, including closure for these same facilities are listed below.

- NOM-035-SEMARNAT-1993 establishes the measurement methods to determine the concentration of suspended particles in the air to monitor air quality. It requires data for temperature, barometric pressure, wind direction and speed to determine the concentration of total suspended particles.
- NOM-083-SEMARNAT-2003 requires that the landfill facilities should be instrumented to monitor gas emissions and that any leachate should be characterized to determine pH, biochemical oxygen demand (BOD), and chemical oxygen demand (COD), as well as heavy metals. The aquifer should be monitored for changes in the water gradient as well as the natural variations of the flow due to the seasonal changes; and a baseline study should be conducted to determine groundwater quality. At closure the landfill must be closed

using an engineered cover. This cover must isolate the waste, minimize infiltration of fluids into the ground, control the flow of biogas, minimize erosion, and provide adequate drainage.

The regulation applies to landfills that reach their final height and have an area of at least one hectare. The closure of the landfill should include restrictions regarding the use of the site, slope stability, land boundaries, details about the closure engineered cover, surface drainage and infrastructure to control leachate and biogas. A program for post-closure monitoring and maintenance of all facilities of the landfill should be developed, for a period of at least 20 years. This period may be reduced if it is shown that there is no risk to health and the environment. The program should include the maintenance of the final cover and procedures to repair it as well as procedures to handle subsidence caused by the degradation of the solid and hazardous waste. Also damage from erosion (caused by stormwater run off and wind) should be taken into account when preparing the monitoring and maintenance program.

- NOM-138-SEMARNAT/SS-2003 establishes maximum permissible limits for hydrocarbons in soil. Should limits be exceeded, an environmental and human health risk assessment may be conducted to determine remediation options.
- NOM-141-SEMARNAT-2003 establishes procedures to characterize tailings, as well as providing specifications and criteria for tailings dam siting, design, construction, operation, and closure. The post-closure requirements for tailings impoundments requires that measures be taken to ensure that the closed facilities are not liberating particulates to the atmosphere, that discharges are not impacting surface water or groundwater and that the impoundments are physically stable. If the tailings are potentially acid-generating, then the tailings should be covered or submerged to prevent acid drainage formation, or they should be neutralized using other materials. If mitigation of acid drainage is required, then the measures taken should prevent impacts to water, soils, and sediments. The slopes needed to be stabilized.
- NOM-147-SEMARNAT/SSA1-2004 establishes soil remediation levels for concentrations of arsenic, barium, beryllium, cadmium, hexavalent chromium, mercury, nickel, silver, lead, selenium, thallium, and vanadium. The regulation includes specifications for site characterization (such as the number of samples), a conceptual site model, and an alternative method to determine remediation levels based on a risk assessment.
- NOM-157-SEMARNAT-2009 establishes the requirements for mine waste management plans. Section 5.6 of the regulation describes the criteria for storage and final deposition of wastes. The criteria include identification of the site environment that could be impacted by operations; the engineering and maintenance specifications to maintain physical stability; control measures to avoid wind and water erosion; and measures to prevent acid drainage, leaching, and run off. Post-closure criteria include monitoring of water bodies that could be impacted and reforestation using stockpiled soil and native species of the area.

20.1.3 State Laws and Ordinances

The Durango State Ecological Ordinance is the primary state law that will be applied to the Project. The type of project proposed at the Property falls under two environmental categories: A) Conservation and B) Land usage. The Durango State ordinance does not set limiting statutes for mining under these two categories and clearly states that these activities, in terms of ecological ordinances, must be reviewed at municipal or local scale.

The Project will need to file for a Land Use License. A permit is generally required by the Local Urban Development Regulations, if they exist, stating that the use of the land where a given mining project is located is compatible with the local or regional urban development plan. The Project will also need to secure a construction permit for the installation of the required infrastructure.

20.1.4 Site Permitting Requirements

The environmental permits will be primarily issued by SEMARNAT. The Environmental Impact Evaluation will take place in Mexico City (Central Offices), as the current applicable SEMARNAT criteria is that all projects that involve a hazardous activity, must be evaluated by SEMARNAT´s Central Offices. The Project will need a number of permits for commercial operation including:

- A Change of Use of Forest Land to Mining use;
- An Environmental Impact and Risk Authorization;
- Environmental License (Licencia Ambiental Única), which will allow commercial mining extraction and mineral processing after the MIA application is approved;
- Land Use License and Construction Permit;
- Permit for the Use of Explosives;
- Hazardous Waste Generator Permit (Generadora de Residuos Peligrosos); and
- Water concession permits for extraction and use of mine water.

A Cédula de Operación Anual (COA) – an annual report regarding the air emissions and contaminants generated on site, is submitted as part of the environmental license requirement. The COA requires an update to describe the mining and processing operation every 5 years.

20.1.5 Site Permitting Status

The Project currently has two exploration drilling permits from SEMARNAT. The permits needed for commercial operations will first require approval by SEMARNAT of the MIA documents.

20.1.5.1 Baseline Studies to Support MIA Applications

Coeur has initiated the baseline studies on the major topics (i.e. forest, wildlife, soil, landscape, hydrogeology, waste characterization) in support of preparing two MIA applications that will be submitted to SEMARNAT in the future. The baseline studies include information collected in different seasons and years (2010, 2011, 2013, and 2014), based on the location of the mine site and expected access road and services. The environmental baseline is composed of data collected on a regional level as well as physical, chemical, and biological data collected on a local scale. The environmental baseline was scaled to ensure suitable coverage by collecting regional and site-specific data, including the definition of the local environmental system, based mostly on nano and micro basins directly upgradient and downgradient of the Project infrastructure and operations, as well as additional surface areas that have similar conditions or natural barriers.

The “Environmental System” needed for permitting evaluations has been identified, covering 14,261 hectares that comprise a mosaic of agricultural land, scrub forest, the mine site, and the immediate communities; the Environmental System considers the current land use and the natural hydrologic basins and aquifers. The first MIA application will address disturbance to the road and utility access corridor from Federal Highway 40 to the site. The second MIA application will address the disturbance to areas at the Project site required for mining, processing, and disposal of waste rock and tailings.

20.1.5.2 Future Permitting Phases

The Project is planned to be executed in two permitting phases:

Phase 1: Access road and services. Once the mine development moves forwards, a suitable and dedicated mine road is needed ready in advance for construction. The environmental documents and authorization that must be submitted and obtained prior to execution are a MIA and an ETJ.

Phase 2: Development of mine site (preparation, construction, exploitation, processing, and closure)

The environmental documents and authorization that must be submitted and obtained prior to execution of Phase 2 are a MIA, a Risk Assessment (ER) and an ETJ. Mine site preparation and construction will need a dedicated access road and infrastructure; the two phase approach should enable the Project to initiate construction as soon as an authorization is issued.

The Project is in the process of completing the environmental documents. The final documents will include the following:

- Project updates and latest design improvements (geotechnical, resource modeling, water balance, infrastructure components and final master plan)
- Legal documentation for surface land (ownership, lease, documentation ratified by Agrarian Ministry)

The required evaluation period for the environmental authority is 60 workings days. Certain projects, owing to their complexity, profile, and need for additional information, may have an expanded review period of up to 60 additional working days (for a total of 6 months).

If the Project requires modifications and additional areas, a new MIA, ETJ and an updated ER may need to be submitted in order to consider new and total environmental impact for a new environmental authorization.

20.1.6 Social License in Mexico

Under the Mexican legal system, the concept of the "social license to operate" does not exist. It must be considered, however, that whereas the Federal Government issues permits for mine development and operations, it is the Municipal (local) Government that grants a Land Use Permit and a Construction License, in accordance with the local legislation that allows or prohibits certain activities or land uses.

In the past years, increased concern has arisen related to the topic of indigenous groups. This issue is addressed by the government in the current criteria considered during the authorization of the MIA. The La Preciosa region and municipality around the Project fall under the "scattered indigenous population" category; less than 40 percent of the population is indigenous. Locally, the largest concentrations of indigenous people (55 percent) reside in the locality of Francisco I. Madero, approximately 13.75 km east of the Property.

In terms of the role that society plays in the environmental legislation in Mexico, a public consultation can take place within the Environmental Impact Evaluation Procedure of the Project whereby any person interested in the Project can make observations to the MIA and propose further mitigation and compensation measures. Furthermore, a public assembly can take place with all stakeholders.

As discussed in Section 20.1.2, any person, social group, non-governmental organization, or association can file claims before the environmental authorities in accordance with the LGEEPA when an activity threatens to cause damages to the environment. This law allows third parties to submit appeals against administrative acts that result in illegal actions related to environmental law.

20.1.7 Site Health and Safety Program and Training

Health and safety issues are being monitored by Coeur's Safety Department, and training is and will be provided by the staff to employees, contractors, and others working on site The Safety Department is currently developing a standardized Health and Safety system for training and for tracking and reporting incidents (PMLP, 2014). Since purchasing the Project, PMLP has worked without a lost time injury or medical treatment case.

20.2 ENVIRONMENTAL STUDY INVESTIGATIONS AND RESULTS

A number of site characterization and engineering studies have been completed or are currently in progress to support the MIA applications. The completed environmental studies and inventories have been compiled for the project area and adjacent land in compliance with requirements for the presentation of the main environmental documents that must be submitted prior to mine development: the MIA, the ETJ and ER. The information collected will be used for the preparation of the MIA and for other environmental documents to be submitted to the authorities for environmental permitting. Further environmental work will continue at the site in the form of pre-mining baseline monitoring programs. Brief summaries are provided in Sections 20.2.1 through 20.2.8 of the pre-mine site conditions and the findings from these studies.

20.2.1 Local and Site Climate Studies

Local climatological data, acquired from the climatological network of CONAGUA, was compiled to estimate climate conditions for the Project site. Data from four weather stations located around the Project were reviewed; the locations of the stations are listed in Table 20-1 and shown in Figure 20-1. Each station represents more than 25 years of precipitation data. Proximity to the Project, a comparison of elevation, and suitability of the data collected at each station were the criteria used to select the data set that would best represent climate conditions at the Project. The Peña del Águila weather station was the furthest from the site, but was selected based on similar elevation, and the relevance and integrity of the data records from this station (longer period of precipitation data and more complete).

A compilation of monthly average total precipitation and evaporation is presented in Table 20-2. The local precipitation data were used as design criteria inputs for the Project, and were used to estimate the probable maximum precipitation 24-hour rainfall depth (302 mm) (SRK, 2014a), to determine the various design storms (SRK, 2014b), and to estimate the steady-state water balance discussed in Section 20.2.5 (SRK, 2014j).

A site weather station was installed in 2014 at the core sheds (future location of the Abundancia pit) to monitor the climate conditions at the Project. Measurements for temperature, rainfall depths, rainfall time interval (intensity), humidity, solar radiation, wind speed and wind direction are being recorded and downloaded from a data logger for compilation and monthly review. Daily maximum and minimum values are being compiled. Evaporation pan equipment has not yet been installed.

Table 20-1: Weather Stations Nearby the Project Site

Station No.	Name	Township	Latitude	Longitude	Elevation (masl)	Years of Data
10063	San Bartolo	Canatlán	24° 31' 37" N	-104° 39' 20" W	2,000	18 (1990-2007)
10027	Francisco I. Madero	Panuco de Coronado	24° 24' 2" N	-104° 19' 8" W	1,960	51 (1954-2004)
10052	Panuco de Coronado	Panuco de Coronado	24° 32' 18" N	-104° 19' 59" W	2,134	27 (1982-2008)
10054	Peña del Águila	Durango	24° 12' 19" N	-104° 39' 29" W	1,890	45 (1963-2007)

Source: Compiled by SRK, 2014 using data from the climatological network of Comisión Nacional del Agua, (CONAGUAS)

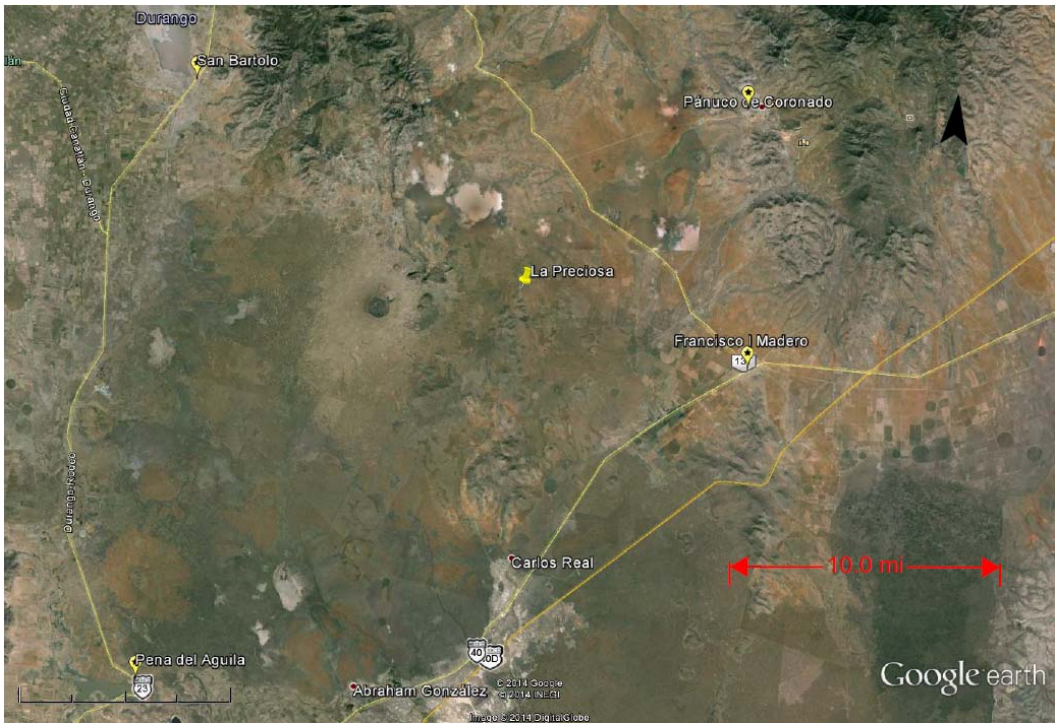


Figure 20-1: Location of CONAGUA's weather stations closest to the Project (SRK)

Table 20-2: Regional Average Monthly Total Climate Data

Month	Days	Precipitation (mm)	Evaporation (mm)
January	31	16.47	141.4
February	28	5.1	174.1
March	31	3.66	268.2
April	30	2.14	301.7
May	31	10.08	320.4
June	30	69.76	256.8
July	31	124.61	186.5
August	31	130.11	163.2
September	30	113.15	140.5
October	31	32.62	152.3
November	30	12.97	143.6
December	31	12.98	128.2
Yearly Average:		**533.7**	**2377**

Source: Compiled by SRK (2014) from Pena del Águila Station, 45 years.

20.2.2 Baseline Air Quality

The existing air quality at the Project site is typical of the farming regions of Central Durango State. The air quality in the area is variable. While there are few fixed-point emission sources, fugitive dust emissions are a common characteristic of the area, especially during dry season and during the preparation of agricultural plots prior to the rain season. No formal study of the baseline air quality at the site has yet been undertaken.

Farming activities in the area immediately surrounding the project site include dry farming for beans and corn. There is a plowing season that occurs in late May and into June, with planting of the desired seeds immediately prior to the monsoonal rainy season. Harvest of the beans and corn begins after the rainy season, with that farm activity occurring for some months as a primary and gleaning activities occur. Thus, there are two extended times during the year when plowing and harvesting activities will likely cause considerable dust. Two other times during the year generate very little dust including the rainy season and the inactive agricultural season during which there is limited activity in the fields.

20.2.3 Groundwater Characterization

The Project is within the limits between Madero-Guadalupe Aquifer and the Valle del Guadiana Aquifer as defined by CONAGUAS (2009). As part of the preliminary baseline groundwater characterization, SRK located and sampled 21 wells and 3 springs that are located within 10 km of the Project site (SRK, 2014n). These wells and springs are used for drinking water supply, agriculture, and livestock. At each source, SRK measured field parameters (temperature, pH, and conductivity) and collected samples for analysis of total dissolved solids, total suspended solids, nitrates, chloride, alkalinity, total hardness, sulfates, and dissolved metals. The laboratory results show that the groundwater in the region is generally of good quality. For reference purposes the results were compared against the permissible limits established by NOM-001 for human drinking water. Nearly all of the wells and springs were compliant with the drinking water standards; however, two wells and one spring located approximately 7 km south of the Project had iron and fluoride levels in excess of the standards. These elevated levels are believed to be naturally occurring. The water supply wells for the Project are within 500 meters of one of these wells.

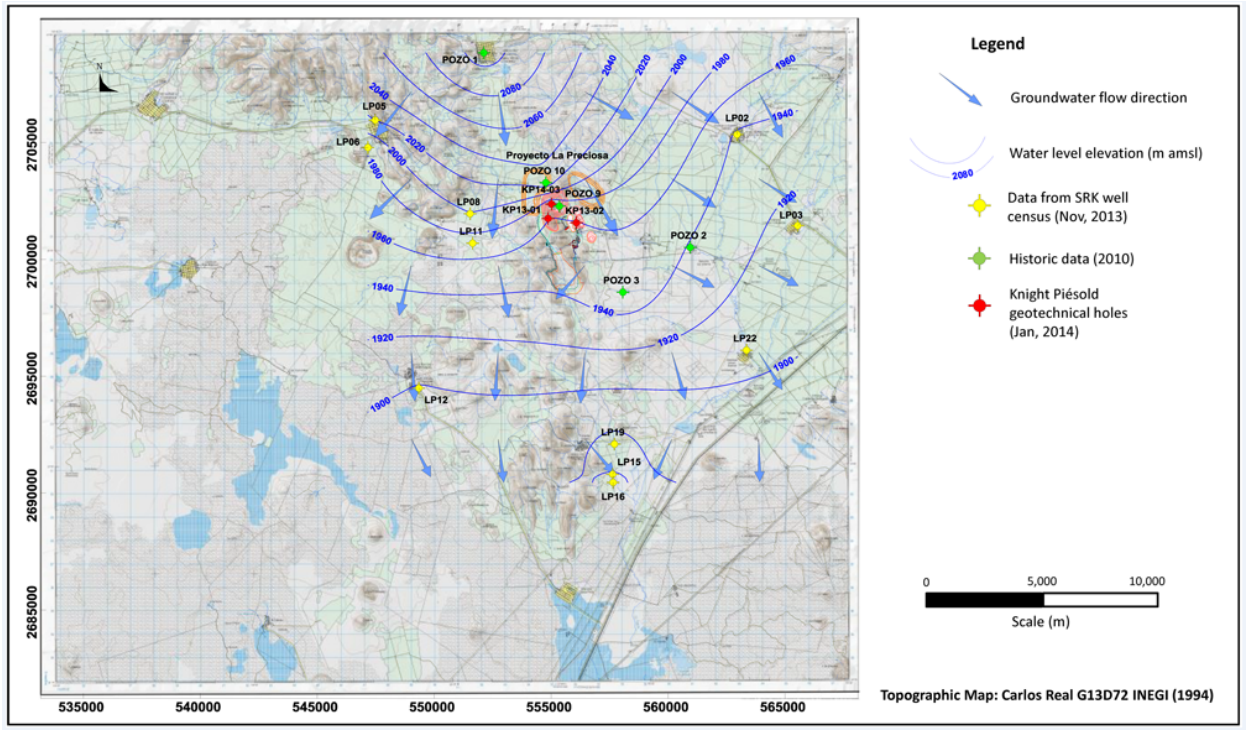
In addition to the samples taken, SRK recorded water levels where possible and constructed a map of the regional potentiometric surface. From the available well data listed in Table 20-3, the groundwater appears to flow from the north-northwest to the south-southeast, mimicking the regional topography (Figure 16-1). KP also documented site water levels in geotechnical boreholes drilled in 2014 and from the vibrating wire piezometers (VWPs) they installed. The three VWPs installed in the Abundancia pit area indicate the water level is approximately 1,960 masl (ranges from 1960 masl to 1,963 masl) or 8.8 to 200.8 m below ground surface (bgs). The average water level beneath the Abundancia pit and the TSF is approximately 100 m and 150 m bgs, respectively.

In the Project area, the most important aquifer is the “volcanic” unit which hosts most of the mineralization. Increased fracturing is associated with localized vein contacts, and jointing and fracturing is more prevalent in the volcanic unit than in the underlying conglomerate and schist units. The irregular fracturing and vein density make the hydraulic conductivity of this unit quite variable. While permeability is generally low, the hydraulic conductivity based on packer testing performed ranges from 4E-08 to 3E-04 m/s (KP, 2014b).

Table 20-3: Water Level Data from Nearby Wells and Piezometers

Name	Easting (m)	Northing (m)	Elevation (m amsl)	Measurement Date	Depth (m bgs)	Casing Stickup (m)	Water Level (m bgs)	Water Level Elevation (m amsl)
LP02	562951	2705345	1988	13-Nov	72	0.18	50.8	1937.4
LP03	565541	2701443	1970	13-Nov	180	0	56.5	1913.5
LP05	547478	2705958	2040	13-Nov	-	0	18.9	2021.1
LP06	547165	2704786	2022	13-Nov	100	0.12	40.9	1981.2
LP08	551535	2701953	2040	13-Nov	-	0.3	41	1999.4
LP11	551643	2700692	2040	13-Nov	133	0.15	69.6	1970.6
LP12	549343	2694490	1935	13-Nov	-	0.37	36.7	1898.7
LP15	557627	2690809	1890	13-Nov	150	0.6	10	1880.6
LP16	557649	2690456	1890	13-Nov	150	0.4	8.8	1881.6
LP19	557705	2692098	1894	13-Nov	150	0.55	20.4	1874.2
LP22	563355	2696116	1924	13-Nov	-	0.18	17.5	1906.7
KP14-03	555015	2702367	2106	14-Jan	385	1	140.6	1966.4
KP13-01	554869	2701751	2160	14-Jan	249	1	200.8	1960.2
KP13-02	556079	2701571	2024	14-Jan	169	1	66.5	1958.5
POZO 1	552118	2708838	2150	10-Oct	191	0	40.3	2109.7
POZO 2	560945	2700514	1958	10-Oct	50	0	19.4	1938.6
POZO 3	558067	2698596	1996	10-Oct	180	0	42.2	1953.8
POZO 9	555355	2702277	2053	10-Oct	252	0	81.9	1971.1
POZO 10	554772	2703280	2099	10-Oct	391	0	67.4	2031.6

Source: Compiled by SRK from data provided by Coeur and SRK



Source: SRK, 2014j

Figure 20-2: Regional Potentiometric Surface Map and Groundwater Flow Direction

20.2.4 Surface Water and Storm Water

Surface run off is mostly to the south and east and all the surface run off is seasonal, lacking permanent surface water bodies in the area (SRK, 2014i). One perennial stream flows past the village of Francisco Madero. The remaining streams in the Project area are ephemeral with short-duration seasonal flow in washes and arroyos that are dry for most of the year. The ephemeral stream flow is captured at numerous locations in the Project site and immediate environs by man-made reservoirs created for agriculture and livestock usage. A semi-permanent source of water is found in the crater of the nearby Jaguey volcano in the La Breña-El Jaguey Maar Complex. In October 2013, SRK took surface water quality samples from the Jaguey crater, a nearby spring, and from selected nearby water reservoirs (SRK, 2014n). The samples were intended to provide a preliminary analysis of pre-mine background conditions for local surface water. None of these water sources appear to be used for human drinking water. For reference purposes, the results were compared against the permissible limits established by NOM-001 for human drinking water. Arsenic and sodium were elevated above permissible limits in the water sampled from the nearby crater. Results from three of the agricultural reservoirs showed concentrations of aluminum and iron that exceeded permissible limits for human consumption. One of the reservoir samples had elevated total dissolved solids.

20.2.5 Site Water Balance Estimate

As discussed in Section 18.8, a steady-state site-wide water balance was prepared for inclusion in the second MIA application (SRK, 2014j). The water balance estimate indicates the site will be a net water user and that the Project will rely on make-up water pumped from the well field, stormwater run off ponds, and from pit dewatering. Water usage from water supply wells will decrease with time owing to the contribution of groundwater inflows as the pits deepen.

20.2.6 Geochemical Assessment – Waste Rock and Tailings

A preliminary geochemical assessment of mine waste materials was performed by SRK in 2014 (SRK, 2014k). The primary objective of the program was to assess the acid rock drainage and metal leaching potential (ARD/ML) of waste rock and tailings that have been or will be produced during mining and processing and that could potentially impair local surface and groundwater quality. The methods that were used comply with Mexican regulations pertaining to geochemical characterization of mining wastes, and the results will provide documentation to support the site-wide MIA.

The geochemical analytical program was based primarily on regulations published by SEMARNAT for characterization of mining wastes and hazardous wastes. These include NOM 141 SEMARNAT 2003 (SEMARNAT 2004), NOM 157-SEMARNAT-2009 (SEMARNAT 2011), and NOM 052 SEMARNAT 2005 (SEMARNAT 2006). SRK also included analyses for pH, moisture, and neutralization potential per the Modified Sobek method.

20.2.6.1 Waste Rock

Representative samples of waste rock were generated from historic and recent drill core through a review of the assay, rock, and oxidation data in the drillhole database and the location of drill samples within the 3D pit shells. The results of the preliminary waste characterization indicate that some of the mining wastes generated at the Project have the potential to cause environmental impacts and may require mitigation measures. Seven of the 50 waste rock samples are acid generating by the criterion defined in NOM-157. Six of these are volcanic and one is quartz vein. Based on data and field observations, the basalt, sedimentary rock, and paleosoil rock units are non-acid generating. The six acid generating samples of the 37 volcanic rock samples represent 16% of the volcanic sample set. The samples were selected randomly with no known geologic or spatial bias. Unless supplementary kinetic testing proves that the rocks are not acid generating, waste rock management methods will be considered where necessary to limit stormwater contact with potentially acid generating waste rock and to minimize run off and release of acid rock drainage.

The extent to which the measured neutralization potential compares to the effective neutralization potential may be an important factor in the generation of ARD/ML from waste rock. Measured NP includes non-carbonate minerals such as silicates that can neutralize acid in a laboratory setting. Effective NP is the NP that can neutralize internal and external acidity inputs sufficiently to maintain a near-neutral drainage pH (Price, 2009). NP measured in static tests may be higher than effective NP for several reasons. At high rates of acid generation, silicate minerals, which can report as neutralizing minerals in ABA, may not be available for neutralization. This may be relevant at the Project which, owing to the volcanic host, is characterized by relatively large proportions of silicate minerals. Kinetic testing can help quantify effective NP, which is why SRK recommended conducting humidity cells on selected samples (SRK, 2014k)..

With respect to leaching potential of metals and metalloids, numerous constituents exceed the dry base maximum permissible limits of NOM-157. However, when subjected to the leachate test (PECT), there were no analytes that exceeded the maximum permissible limits for the PECT. By definition in NOM-157, none of the waste rock is classified as dangerous owing to metals toxicity.

20.2.6.2 Tailings

Two representative tailings samples were generated from historic and recent drill cores through a review of assay (above cutoff grade), rock and oxidation data in the drillhole database and the location of the proposed drill core intervals with the 3D pit shells. Both of the samples of tailings materials generated through laboratory test work were measured to be acid neutralizing. No follow-up testing is required per Mexican regulations, and no mitigation measures appear to be necessary to comply with the mandates of Section 5.7.2 in NOM-141. From the standpoint of regulatory compliance for permitting, the PECT data are the only leachate data that are relevant, and the leachate data show no analytes that exceed regulatory limits.

However, the ore processing methodology currently being evaluated has the potential to leave residual constituents that might necessitate mitigation measures at some time in the future depending on circumstances such as exposure of tailings to humans or wildlife. More specifically, based on supernatant analyses, tailings have the potential to carry cyanide at concentrations that exceed NOM-001 limits for aquatic life. Although analyzing the supernatant solution component and submitting this data is not required for permitting documents, routine monitoring of the TSF liquid fraction will be expected over the life of the mine, and these data are the best indicator of the long-term conditions expected. With a lined TSF, possible groundwater or surface water discharge is a minor concern assuming the liner functions as designed. The primary concern requiring monitoring is the potential for wildlife to access and ingest the tailings supernatant fluid.

20.2.7 Biological and Wildlife Assessment

There are several conservation designations that have the potential to impact potential land use if the Project were to fall within one of these specially protected lands. No portion of the Project area is designated for special environmental protection. There are three national Protected Natural Areas [Área Natural Protegida (ANP)] within the state of Durango, but the closest ANP is 100 km from the Project. The Project also falls outside any protected area classified as a critical terrestrial region [Regiones Terrestres Prioritarias (RTP)]; the closest RTP is found 20 km away near the municipality of Canatlán (M3, 2013). There are two areas in Durango set aside for conservation of bird species (particularly migratory water fowl) [Areas de Importancia para la Conservación de las Aves (AICA)]. The two AICAs, however, are 8 km and 48 km distant from the site.

The vegetation in the Project area is characterized by large expanses of agricultural fields, thorny acacia, scrub brush, succulent woody scrub brush (*stegnosperma*), ocotillo, cacti (*opuntia*), yucca; the foothills contain limited woody vegetation. The most prevalent woody species is acacia, which is quite abundant in the non-agricultural lowlands. Baseline studies in 2009 identified the area to be heavily impacted by intensive agricultural disturbances (farming, grazing). Although the technical vegetative designation of the area is a scrub oak forest, there are very few oak trees

in the immediate area including the rolling hills that outcrop between the agricultural files. The area adjacent to the Project support more oak forests than in the immediate Project area (M3, 2013).

Baseline surveys at the Project site were initiated by Clifton Associates Ltd (Clifton) of Guadalajara, Mexico in 2009 for a previous owner (Clifton, 2009). These studies reviewed all regional and local site characteristics (climate, geology, vegetation zones/types, soil zones/types, and the plant and vertebrate species). The 2009 survey identified 91 local vertebrate species including 67 birds, 11 mammals, 1 amphibian, and 12 reptiles. Nine of the species within the Project area were identified as protected fauna under Mexican Law.

In 2014, Clifton updated the vegetation surveys and wildlife inventories on a total of 14,261 ha (Clifton, 2014). The inventoried land comprises scrub forest, natural and planted grassland, seasonal dry-land agriculture, and irrigated agriculture. The following species were identified:

- 41 plant species; and
- 121 vertebrates including 87 birds,15 mammals, 3 amphibians, and 16 reptiles.

Thirteen of the animal species were identified as protected fauna and two of the cacti species are protected flora.

20.2.8 Historical and Archaeological Resources

In order to assess the presence of any historical and/or archaeological resources within the Project area, the site was first outlined on a map based on the preliminary plans for the open pits, waste rock dumps, low-grade stockpile, TSF, and processing plant facilities (M3, 201) plus a half kilometer buffer zone. This potential disturbance area was sent for review to the Mexican National Institute of Anthropology and History [Instituto Nacional de Antropolgía e Historía (INAH)] in Durango; INAH reviews plans for industrial or mining disturbance. INAH maintains national databases for resources related to anthropology, history, archaeology, paleontology and ethnography and is responsible for the conservation of those resources. The Durango INAH Durango office reviewed the outline of proposed disturbance and compared the outline with known resources catalogued in the Sistema de Registro Público de Monumentos y Zonas Arqueológicos e Históricicos. Five archaeological sites were noted in the database in the general region but none were known to exist within the project area (INAH, 2013). A site specific inventory is being contracted.

The project site and the access road route were reviewed in 2013-2014 by field consultants from Clifton for the initiation of the permitting process for the Project. Their focus was the documentation of flora, fauna and other resources. No historic or archeological resources, however, were reported during those field surveys.

20.3 ENVIRONMENTAL MONITORING AND MITIGATION PLANS

This section summarizes the planned programs to characterize baseline conditions and for routine monitoring and management during construction, operations, and closure.

20.3.1 General

The mitigation plans for disturbance at the Project cover three main areas:

- Project designs based on site-specific geotechnical, geochemical, natural resources, and land-use considerations and Coeur's commitment to minimize disturbance where ever possible;
- Environmental monitoring and mitigation plans for each project stage, activity and project component; and
- A closure plan that is periodically updated for site restoration and elimination of residual impacts and reclamation of disturbed land.

The development during preparation, construction, and operations will be monitored based on a strict control over the authorized disturbance areas, actual disturbances, and compensation measures to offset the expected disturbance. This requires creation of an environmental management system at site and a GIS-based monitoring plan through the use of remote sensors to be updated during the life of mine.

The main tool for environmental compliance and monitoring actions throughout the life of mine is the Environmental Surveillance Program (PVA). The PVA, will be an essential tool for environmental compliance and performance monitoring. The PVA will integrate the following:

- Prevention, Control and Mitigation measures for environmental impacts;
- Terms and conditions derived from environmental authorizations; and
- Social, Community and Corporate policies, guidelines and reporting protocols.

The PVA will integrate various operational, environmental, and health and safety aspects including:

- Stability monitoring of the TSF, waste rock piles, and open pits;
- Soil erosion prevention, topsoil recovery and the installation of soil retention structures;
- Storm water pollution prevention plan to manage, protect and divert run off and avoid mixing with contact water (from waste rock piles, stockpile, mining areas);
- Materials management plan, based on characteristics, volume, production, handling and disposal;
- Vegetation protection and restoration plans;
- Fauna protocols: Inspections, animal chase off, capture and release, burrow holes and nest inspections; specific protocols for capturing, handling and relocating protected species; identification protocols and new species reporting;
- Forest material: Monitoring the usage of non-commercial forest resources;
- Land and soil: Securing and controlling toilet services, delimitation of each work area and land disturbance boundary, training personnel for land clearing, topsoil recovery and handling, recovery and integration of plant material to topsoil, application of land conservation measures;
- Hydrology: Methods for surface water conveyance, protection and monitoring; protection and segregation of clean and contact stormwater;
- Spill Control and Chemical Handling: Training employees to prevent spills, handle chemicals, and perform emergency response actions, if necessary.
- Air emissions: Dust monitoring and dust abatement;
- Noise and Vibration: Monitoring and control of excessive noise and vibrations; daily and periodic equipment checks, monitoring and emissions controls from process operations;
- Climate monitoring: Operation, maintenance, and database management of the site weather station;
- Waste management: Preparation and updating a waste management plan including waste classification, waste separation, adequate and labeled waste containers, continuous monitoring and cleanup activities, waste minimization program (generation, reutilization, recycling), storage and handling, pollution prevention, training, transportation, security, and emergency response.
- Preventive maintenance program: Asset management program;
- Signage: Design and implementation of environmental and health and safety protection signs, identification of restricted use areas (authorized for mine development, conservation area, recovery, reforestation, etc.), educational and informative signs;
- Reforestation and reclamation of disturbed land: Reforestation surface, species, reporting by parcels, surface, density, species, survival rate, phytosanitary conditions; plant nursery operation and production;
- Financial instruments: Update of a technical and financial assurance documents to secure environmental bonds, insurance, or other financial instrument;
- Mine Closure: Periodic update of mine closure plan, progressive closure measures; and

- Additional programs: Personnel training, community relations, environmental budget review and update, follow up and periodic reviews, environmental reporting.

The sections below provide details on specific mitigation and monitoring plans.

20.3.2 Climate Monitoring

The Project staff will continue to maintain and monitor the site weather station to provide information relevant to operations and environmental management of the site. Daily measurements for temperature, rainfall, humidity, solar evaporation, wind speed, and wind direction will be measured and downloaded monthly from the data logger for compilation and reporting monthly. Pan evaporation equipment is recommended to be added in the future.

20.3.3 Air Quality Monitoring and Dust Abatement

Dust monitoring will begin in the area 6 months prior to site preparation, using a certified air emissions laboratory for Total Suspended Particles (TSP) and Particulate Matter (PM-10). Once mine development initiates, laboratory samples will be collected each trimester. The Project will operate an internal program for dust monitoring though the use of portable low volume samplers for TSP and PM-10.

The Project will effectively minimize, monitor, and control dust and particulate generation during both the construction and operation phases. Construction activities will include clearing and grubbing of soils during site preparation of the plant site, TSF, mine waste dumps and roads, and other features. The dust generated during construction activities will be controlled with application of additional water or other seasonal measures, as required.

Control of dust during mine operations will be via standard industry practices and techniques. This includes adding water as part of all drilling activities and installing enclosed skirting around the drill bits to minimize dust generation. Road dust will be minimized through use of dust control agents on haulage and mine roads and applying water using large water trucks. Environmentally friendly surface stabilization products will be used where needed.

20.3.4 Groundwater Monitoring Plan

No permanent groundwater monitoring wells exist at the site. One well has been proposed to be installed to monitor background conditions upgradient of the Project. One well has been proposed downgradient of the Plant Site and two wells have been proposed to be installed downgradient of the TSF (SRK, 2014m). The wells will be monitored monthly for a minimum of one year to establish the pre-mining baseline groundwater quality and water level conditions. During operations and for a minimum of 10 years post-closure, the wells will be routinely analyzed for an extensive suite of constituents on an annual basis. A limited list of indicator parameters will be analyzed on a quarterly basis during operations to provide more frequent operational monitoring. Water level measurements will be used to prepare an annual potentiometric surface map and vertical profiles through the pit area to monitor water level conditions. A data management system will be initiated to review the analytical results, prepare hydrographs and water quality graphs, and to assess the occurrence of any trends that may occur in groundwater quality.

Groundwater or methane monitoring wells may be required for monitoring downgradient of the solid waste landfill, but will be reviewed once a design has been prepared for the landfill.

20.3.5 Surface Water Monitoring and Storm Water Management Plan

Stormwater run on from upgradient watershed basins will be managed to minimize run on into the open pits and to the plant area, waste rock dumps, and the TSF. Preliminary designs for channels and berms have been developed to prevent unimpacted stormwater from contacting mining wastes. Precipitation and stormwater falling directly on or within the plant or tailings areas will be considered to be impacted through contact with potentially impacted soils and mined

waste materials. Precipitation within the open pits, plant area, or on the TSF will be collected and pumped to the plant site as make-up water.

A *Surface Water and Storm Water Management and Monitoring Plan* will be developed prior to the initiation of construction activities and will be updated, as needed, during operations (KP, 2014f). The plan is intended to:

- protect local surface waters from impacts owing to the potential erosion of sediments from newly disturbed construction areas and fines from tailings and waste rock dumps;
- divert unimpacted stormwater to the greatest extent possible to adjacent natural drainages to minimize losses to the downgradient arroyos and agricultural users;
- provide guidance on best management practices to control stormwater run on and run off, and to prevent loss of fines (i.e. berms, channels, ditches, sumps, sediment collection basins);
- provide guidance on routine inspections to ensure the stormwater control features are operating properly; and
- provide guidance for stormwater outfall monitoring for any potential impacted stormwater that is discharged from the site.

20.3.6 Petroleum and Chemical Spill Control Plan

A *Chemical and Petroleum Spill Prevention and Control Plan (Spill Control Plan)* has been prepared to ensure the safe response to potential release of chemical and petroleum products to the environment (SRK, 2014d). It also describes procedures to prevent, control, and mitigates releases of diesel fuel, gasoline, lubrication oils, hydraulic oils, anti-freeze/water mixes, transformer oils, plant reagents, and liquid lab wastes to the environment.

The Spill Control Plan was prepared in to be in accordance with Mexican environmental regulations including: Ley General de Equilibrio Ecológico y Protección al Ambiente; Reglamento de la Ley General de Equilibrio Ecológico y Protección al Ambiente en Materia de Impacto Ambiental; Ley General para la Prevención y Gestión Integral de los Residuos y su Reglamento. The Plan incorporates additional industry best practices using requirements promulgated by both Mexico and the United States Environmental Protection Agency 40 Code of Federal Regulations (CFR), Part 112, Oil Pollution Oil Pollution Prevention and Response; Non-Transportation-Related Onshore and Offshore Facilities; Final Rule. La Presciosa must comply with these minimum requirements because the facility will store greater than 1,320 gallons of chemical and petroleum products above ground. The *Spill Control Plan* will be implemented at the beginning of pre-mine construction and updated during operations as needed.

20.3.7 Hazardous and Solid Waste Disposal

Prior to start up, the Project will implement a waste management program. A consolidated *Waste Management Plan* will be prepared and submitted to SEMARNAT. The management program will focus on separate components:

- Hazardous waste management (prevention, reduction, classification, storage, disposal).
- Non-hazardous special and solid waste management (reduction, recycling, re-utilization, and disposal).

A specific building or area will be designated for temporary on-site storage of hazardous materials and special wastes (batteries, fluorescent lights, electronic waste, used oils), or other materials requiring recycling. Manifests will be prepared for hazardous wastes to document the container sizes, residual quantity of each material requiring disposal, and the destination of each shipment to a permitted storage. Wastes will be shipped by a licensed transporter.

An on-site solid waste landfill will be managed for disposal of non-hazardous wastes including office trash, food waste, and building and wood debris. Cardboard and plastics will be recycled, if practical, rather than be placed in the solid

waste landfill. No asbestos-containing waste is expected to be generated or stored on site. A tire burial cell will needed on-site to dispose of large truck tires. Waste water disposal will be handled through on-site septic systems and disposal.

20.3.8 Waste Rock Management and Monitoring Plan

A *Waste Rock Monitoring Plan* has been prepared (SRK, 2014l) and will be updated as needed. The plan addresses management of stormwater run on and run off through the use of berms, ditches, and diversion channels to minimize the stormwater contact with sub-grade, but metal-bearing, waste rock. The plan provides for the creation of sediment collection basins at the downgradient toe of the waste rock storage facilities to prevent fines from being transported into surface water drainages. Monitoring activities will focus on assessing the stability of the rock storage facilities, inspecting stormwater structures, and performing geochemical analyses on representative samples from each dump. The geochemical data will be provided to SEMARNAT in an annual monitoring report and will be used to assess the need for blending of materials during operations to minimize potential for acid rock drainage and to assist in closure planning.

20.3.9 Tailings Management and Monitoring Plan

A preliminary Tailings Management Plan (KP, 2014) has been prepared to address construction, operations, and maintenance of the TSF including the TSF itself and associated pumping and piping systems. A *Monitoring Plan for the Tailings Storage Facility* (SRK, 2014e) was prepared to provide protocols for routine site inspections (stability, freeboard, stormwater management structures). The Monitoring Plan provides guidance and requirements for documenting the results of the inspections, addressing repair issues as needed, and providing clear contingency actions in the event upset conditions occur. The two plans will be updated and/or consolidated as needed during operations.

20.3.10 Vegetation Protection and Restoration Plan

The Project staff will develop a *Vegetation Protection and Restoration Plan* that will provide protocols to be used during the initial site preparation activities (clearing and grubbing), concurrent reclamation of the TSF embankment and other disturbed areas as appropriate during operations, and the restoration methods at closure. The vegetation protection protocols will specify methods for rescue and transplanting of protected species, salvaging wood or edible plants, methods for performing mechanical and manual land clearing, and plant handling and relocation. An on-site nursery will be designed and constructed to provide plants for restoration purposes.

20.4 MINE CLOSURE AND RECLAMATION

Mine reclamation is addressed in Article 27 of the Mexican Constitution, which sets two broad standards for reclamation:

(1) Mexico retains ownership of the land at all times and concession holders only have rights to mined materials. As such, the Nation may establish the conditions of reclamation.
(2) Mexico has an obligation to take mitigation measures to protect natural resources and restore the ecological balance.

Mexican mining law has no requirements or technical guidance pertaining to the site-wide closure, reclamation, and post-closure of a mining and beneficiation facility. Post-closure requirements, standards, and guidelines related to specific facilities such as landfills, tailings impoundments, and mining waste storage facilities are specified in the NOMs. The maximum permissible limits for hydrocarbons and soil remediation levels for concentrations of metals are set in NOM-138-SEMARNAT/SS-2003 and NOM-147-SEMARNAT/SSA1-2004, respectively. Many mining corporations in Mexico, however, do prepare site-wide closure plans in accordance with the requirements of their corporate

environmental charters, international best closure practices, and prescriptive environmental protection and management codes.

The Project duration is expected to be approximately 11 years based on the current LOM plan shown in Table 20-4. Detailed closure planning typically begins in the latter part of the mine life within three years of mine closure. A *Conceptual Closure and Reclamation Plan* (Closure Plan) was prepared by SRK (2014c) for the Project. The plan provides a conceptual level outline of the proposed closure activities and associated costs, and incorporates current best practices for rehabilitation and closure. It incorporates facility rehabilitation options, strategies, and details pertaining to construction, operations, and final facility closure. This document was prepared as a component of the MIA.

Table 20-4: Summary of Mine Material Movements

	From MINE			Stockpiles					
Mining Year	ORE 5m (kt)	HG 5m (kt)	LG 5m (kt)	WASTE 5m (kt)	WASTE 10m (kt)	HG to Mill (kt)	LG to Mill (kt)	Total from Mine (kt)	Total Moved (kt)
-1		304	205	764	9,237			10,509	10,509
1	3,002	654	979	6,953	38,861	304		50,448	50,752
2	3,259	0	956	6,323	67,214	390		77,751	78,141
3	3,649	0	1,296	7,418	66,245			78,607	78,607
4	3,651	0	494	6,218	67,651			78,013	78,013
5	2,886	0	0	4,329	70,020	264	500	77,235	77,999
6	2,749	0	0	4,124	70,405		900	77,277	78,177
7	3,651	0	0	5,477	64,436			73,563	73,563
8	1,434	0	0	2,151	44,310		2,217	47,895	50,112
9	3,338	0	0	5,007	20,176		313	28,521	28,834
10	3,651	0	0	5,477	5,236			14,363	14,363
11	1,168	0	0	1,752	3,898			6,818	6,818
Total	**32,438**	**958**	**3,930**	**55,989**	**527,685**	**958**	**3,930**	**621,000**	**625,888**

20.4.1 Closure and Reclamation Plan

The main processing facilities to be constructed and eventually closed include primary crushing, grinding, leaching, CCD thickeners, Merrill-Crowe processing, and refining facilities. The mining and waste disposal facilities will also include four open pits, two rock dumps, a temporary low-grade stockpile, and a TSF. In addition to the main processing facilities, there will be several surface buildings constructed to support the mining and process operations. These facilities include administration, laboratory, guardhouse, truck shop, warehouse, change house, explosives storage buildings, truck wash, and mill maintenance facilities.

The conceptual closure plan outlines the following key activities:

- Concurrent reclamation, where feasible, of the tailings embankment raise and disturbed land during operations;
- Internal closure planning and closure-related engineering and characterization studies and permitting activities in the 1- to 3-year period prior to closure;
- Decommissioning of equipment and management of salvageable, recyclable, and waste materials at closure;
- Demolition of physical structures and management of infrastructure to be removed or that will remain;

- Earthworks regrading of the TSF, waste storage facilities, and plant area, as needed, to shed stormwater and re-establish natural drainages;
- Reclamation and re-vegetation of disturbed land; and
- Post-closure environmental monitoring and maintenance.

At present it is anticipated that the post-closure land use will be wildlife habitat, grazing, and potential future mining of underground resources. Undisturbed portions may also be used for dry farming uses. The areas of the open pits, tailings, and former plant site will be restricted from public access.

The closure process will include decommissioning, demolition, and rehabilitation. The decommissioning process would initiate at the early stages of closure and would include the decommissioning of all cyanide materials and equipment. Appropriate methods would be put in place for decommissioning procedures for hazardous materials and equipment.

The general reclamation process is to break/perforate and backfill foundations and sumps, recontour for positive drainage, cover with growth medium, re-establish natural drainages, and revegetate with native species. Further details for closure procedures are explained in the Closure Plan (SRK, 2014c). The site closure and reclamation activities are estimated to take approximately three years assuming some concurrent reclamation.

20.4.2 Post-Closure Period

Post-closure requirements will be established based on a review of facility closure actions. A post-closure plan will be prepared at the time of closure to address post-closure maintenance and monitoring actions. The plan will describe the sampling, monitoring, and inspection procedures to be followed after closure and to describe actions to be taken if a discharge results in exceedances of permissible limits, or if the physical stability of a facility deteriorates. Post-closure monitoring will include inspections of physical stability of remaining facilities and monitoring for environmental impacts. Annual site monitoring will be conducted for 10 years, or until compliance with applicable standards is achieved. Revegetation efforts will be monitored annually for at least 5 years after closure, or more depending on the revegetation study recommendations.

20.4.3 Closure and Post-Closure Costs

A reclamation and closure cost was estimated for third-party implementation of the closure process (SRK, 2014c). The cost of closing the various mine facilities is a function of the closure design plan chosen for each facility. The life-of-mine configuration of the facilities was used as the basis of this cost estimate; closure design plans will be prepared prior to closure. Rehabilitation of the major facilities was estimated in the cost estimate for the process plant, tailing impoundments, waste rock areas, infrastructure (buildings, roads, and landfill), contaminated areas, and utilities.

A total reclamation and closure cost is presented in Table 20-5. The table provides a summary of associated cost for the major mine facilities. The estimated cost is approximately US$11.1 million with a contingency of +/- 20%, which applies to the closure cost (Item Nos. 1.00 through 6.00) and not to the post-closure care and maintenance (Item No., 7.00).

Table 20-5: Summary of Reclamation and Closure Costs

Item No.	Item Description	Cost
1.00	Tailings Impoundment	$8,473,078
2.00	Plant and Ancillary Facilities	$5,915,028
3.00	Waste Rock Dumps	$2,868,033
4.00	Open Pit Areas	$112,816
5.00	Technical Studies & Engineering Designs	$2,442,896
6.00	Closure Management	$3,036,722
	Subtotal of All Capital Cost Items	$22,848,573
	Salvage Value	$16,400,000
	Subtotal Minus Salvage Value	$6,448,573
	Contingencies (20%)	$1,289,715
	Subtotal with Contingencies	$7,738,288
7.00	Post-Closure Care & Maintenance	$3,397,500
	Total Reclamation and Closure Cost (USD)	**$11,135,788**

Source: Compiled by SRK, 2014c

20.5 SOCIAL AND COMMUNITY

The key stakeholders interested in, benefitted by, or impacted by the Project are listed in Table 20-6. The Community Relations Department will address all grievances and concerns regarding Project operations in order to prevent potential community conflicts. Guidance for the interactions and relationships of Coeur with the local communities and landowners who live adjacent to the Project are addressed under the general mission, vision, and values statements prepared for all Coeur operations. These statements are as follows:

Mission: Contribute to increasing the social, mental, and physical capabilities of the people in the surrounding areas of the mine, with the intention of bettering their quality of life, and sustaining social development.

Vision: To achieve recognition from the communities surrounding the mine, contributing to the institutional strengthening and improvement of the quality of life of the local population, increasing the inclusive development and sustainability of the region in order to achieve community sustainability at the end of the mining activity.

Values:

- *Security - It is our main motivation to promote, facilitate, and provide security in our daily activities, because the personal quality of life of our stakeholders and the communities where we operate, depends on this value. This includes environmental protection and ensuring the continuity of the operations and businesses;*
- *Responsibility - We act with responsibility, ever conscious that the commitment to act in a timely and efficient manner for the local interest groups, falls to us;*
- *Ethics - Our behavior is based on loyalty and transparency in each of our actions, rejecting any act of corruption in all levels of the company. Confidentiality and discretion are fundamental pillars in the management of the information;*
- *Honesty - Our behavior is based on truth and is committed to the policies and procedures of the company with transparency and sincerity in each of the actions we take; and*
- *Respect - We promote and practice personal, labor, and business relationships based on mutual respect, which ensures a long-term commitment.*

Coeur has developed a community relations program to support the corporate mission, vision, and values. Annual objectives are developed in consultation with local property owners, ejidos, and other leaders. Activities are mainly focused on four local communities, four ejidos, including community use areas and 20 parcels, and nine private properties.

The principal community relations activities by Coeur include assisting with the following:

- Identification of community needs and expectations;
- Acquiring property to support the land required for mining, processing, and waste disposal operations;
- Developing relations with governmental institutions;
- Developing relationships with political and moral leaders;
- Developing relationships with the local ejidos and communities;
- Preparing a system to address grievances and prevent conflict;
- Creating a culture of care for the environment and health; and
- Developing programs and activities of a social nature.

Activities in 2013 included an evaluation of the social situation in the local communities and a review of governmental social development programs. The status of prior commitments by Orko was reviewed, and communications efforts were undertaken by Coeur to initiate awareness in the local communities about the new owner.

Community relations objectives include the following:

- Secure private land use agreements;
- Secure agreements for temporary use and right of way access in the ejidos;
- Strengthen and maintain consistent and constant communication with the communities;
- Provide frequent information and training about care for the environment;
- Assist in preparing components of the Environmental Impact Statement (MIA);
- Implement medical visits to the communities;
- Continue assisting local education activities;
- Increase assistance to social programs and activities;
- Implement a socioeconomic study of the region;
- Strengthen ties with the government;
- Start the mine project and become a local employer of choice; and
- Develop training to match the anticipated job demands of the Project.

Table 20-6: Key stakeholders (Coeur, 2014)

STAKEHOLDER DIAGNOSTIC										
#		Key Stakeholder	Role in Organization	Impact of Project on stakeholder (H,M, L)	Current / Desired Support					Reasons for Support or Resistance
					Strongly Opposed	Opposed	Neutral	Supportive	Strongly supportive	
1	Federal Government	Secretaría de Medio Ambiente y Recursos Naturales (SEMARNAT)	Authority	H			X			
2		Procuraduría de Federal de Protección al Ambiente (PROFEPA)	Authority	H			X			
3		Comisión Nacional del Agua (CONAGUA)	Authority	H			X			
4		Secretaría de Economía	Authority	H			X			
5		Comisión Federal de Electricidad (CFE)	Authority	H			X			
6		Registro Agrario Nacional (RAN)	Authority	H			X			
7		Agricultural Land Attorney	Authority	H			X			
8		Agricultural Land Registry	Authority	H			X			
9		Instituto Nacional de Antropología e Historia (INAH)	Authority	H			X			
10		Secretaría de Comunicaciones y Transportes SCT	Authority	H			X			
11		Secretaría del Trabajo y Previsión Social (STPS)	Authority	M				X		Inspectors favor the creation of new employment sources
12		Comisión Nacional de Seguridad Nuclear y Salvaguardias (CNSNS)	Authority	M			X			
13		Secretaría de Salud (SSA)	Authority	M			X			
14		Secretaria de la Defensa Nacional (SEDENA)	Authority	H			X			
15		Secretaría de Minas	Authority	H			X			
16	State Government	State Government of Durango	Authority	H				X		Government favors the creation of new employment sources
17		Property Public Records of the State of Durango	Authority	H			X			
18		State Cadastre	Authority	H			X			
19		Secretary of Natural Resources and the Environment	Authority	H			X			
20	Municipal Government	Town Planning Dept. of Durango	Authority	H			X			
21		Durango Municipality	Authority	H			X			
22		Canatlán Municipality	Authority	H			X			
23		Pánuco de Coronado Municipality	Authority	H				X		Government favors the creation of new employment sources
24	Population / Public	Ricardo Flores Magón	Interested	H					X	Population is interested in the creation of new employment sources near their homes
25		Francisco Javier Mina	Interested	H					X	Population is interested in the creation of new employment sources near their homes
26		Vicente Suárez	Interested	H					X	Population is interested in the creation of new employment sources near their homes
27		Francisco I. Madero	Interested	H					X	Population is interested in the creation of new employment sources near their homes
28		Primero de Mayo	Interested	H					X	Population is interested in the creation of new employment sources near their homes
29		Vulnerable Group - Children in the Communities	Affected	M			X			
30		Vulnerable Group - Elderly in the Communities	Affected	M			X			
31		Vulnerable Group - Women in the Communities	Affected	M				X		Population is interested in the creation of new employment sources near their homes
32	Town	Ricardo Flores Magón **339** Members	Affected	H					X	Population is interested in the creation of new employment sources near their homes
33		Francisco Javier Mina **221** Members	Affected	H					X	Population is interested in the creation of new employment sources near their homes
34		Francisco R. Serrano **106** Members	Affected	H					X	Population is interested in the creation of new employment sources near their homes

STAKEHOLDER DIAGNOSTIC										
#		Key Stakeholder	Role in Organization	Impact of Project on stakeholder (H,M, L)	Current / Desired Support					Reasons for Support or Resistance
					Strongly Opposed	Opposed	Neutral	Supportive	Strongly supportive	
35		Lázaro Cárdenas **113** Members	Affected	H					X	Population is interested in the creation of new employment sources near their homes
36		Vicente Suárez **70** Members	Affected	H					X	Population is interested in the creation of new employment sources near their homes
37	**Town Petitioner**	Ricardo Flores Magón - Señor Ernesto Berúmena has support of **70%** of the town population	Interested	H				X		Population is interested in the creation of new employment sources near their homes
38		Ricardo Flores Magón - Señor Guadalupe Guzmán Meza has support of **20%** of the town population	Interested	H				X		Population is interested in the creation of new employment sources near their homes
39		Ricardo Flores Magón - 1**0%** of the town population has relocated to the USA	Interested	H			X			Has not commented
40	**Private Owners**	22 Identified Proprietors	Interested	H				X		Population is interested in the creation of new employment sources near their homes
41	**Company / Supplier**	Pemex	Interested	M			X			Business interest
42		Fenosa	Interested	M				X		Business interest
43		Lubricant Supplier	Interested	H					X	Business interest
44		Gas Supplier	Interested	M					X	Business interest
45		Explosives Supplier	Interested	H					X	Business interest
46		Reactive Chemicals Supplier	Interested	H					X	Business interest
47		Electric Power Supplier	Interested	H					X	Business interest
48		Maintenance Supplies Supplier	Interested	H					X	Business interest
49		General Labor Supplier	Interested	H					X	Business interest
50		Hazardous Waste Management Supplier	Interested	M					X	Business interest
51		Non-Hazardous Waste Management Supplier	Interested	M					X	Business interest
52		Equipment Maintenance Supplier	Interested	M					X	Business interest
53		Construction Supplier	Interested	M					X	Business interest
54		Supplier for Environmental Services	Interested	M					X	Business interest
55	**Other**	Producers of Corn and Beans in the area	Affected	H			X			Has not commented
56		Senior Groups	Interested	L				X		Benefits for the Group
57		Groups Organized for Community Clinics	Interested	L				X		Benefits for clinics and trained employees

20.6 RECOMMENDATIONS

Progress has been made in the last year to address gaps in the site knowledge of pre-mining conditions, identify environmental liabilities, prepare documentation to support the site permits, and to prepare management plans to mitigate potential environmental impacts to the environment. Some studies are currently in progress and others remain to be done, or are planned for the future to support two MIA studies. Presented below are recommendations related to specific environmental studies or monitoring programs. These are designed to meet Mexican regulations and international environmental best management practices to reduce potential environmental impacts and risks to employees and nearby communities during operations and closure.

20.6.1 Baseline Characterization Programs

Baseline characterization programs need to be continued or initiated to document the pre-mine conditions as a requirement for the environmental permitting process (MIA) and to provide site information to refine the operational designs and management strategies.

20.6.1.1 Ambient Air Quality Monitoring

A pre-development dust characterization program will need to be implemented prior to site preparation, ideally commencing six months in advance of road-building or other construction activities and using a certified emissions laboratory. The baseline program should include PM 10 and PM 2.5 particulates monitoring in addition to total suspended solids.

20.6.1.2 Ambient Groundwater Monitoring

Water quality monitoring has been performed on a number of wells in the local region. Water quality monitoring wells should be installed within the next few months at the Project site to characterize ambient groundwater quality and water levels prior to start up. Sufficient sampling needs to be done to address potential variability on an annual or seasonal basis.

20.6.1.3 Surface Water and Stormwater Monitoring

One set of surface water quality samples was collected in late 2013 from several local reservoirs. A second set of samples is recommended to establish the seasonal water quality of these man-made ponds and ephemeral washes such as during the rainy season. Collecting storm water samples in the ephemeral washes will require installation of sampling stations and equipment and the ability to mobilize a technician on short notice to take custody of the sample.

20.6.1.4 Data Management

An environmental data management and reporting system should be implemented prior to start up to itemize sampling frequencies and requirements, inspection requirements, record results, and prepare routine reporting forms. All boreholes drilled on the site including exploration drillholes, geotechnical borings, and water wells (supply, dewatering, monitoring) should be entered into a unified drillhole database systems for use by operations and environmental staff.

20.6.2 Additional Geochemical Testwork of Mining Wastes

Additional samples, generated through metallurgical test work and of the future actual tailings, are recommended to confirm the geochemical behavior of the tailings materials. The samples could be composited by production years, dominant ore types, or other method to ensure representative samples are collected and analyzed.

The preliminary analysis of waste rock materials indicates that a percentage of the rock materials do have the potential to generate acidic rock drainage and contain metal concentrations that exceed dry base maximum permissible limits. However, when the materials were subjected to a leachate test (PECT), the analyses confirm the materials are not classified as dangerous owing to metals toxicity levels. Kinetic testwork (humidity cells) is recommended on selected rock types to confirm that the rock materials will not generate acid or metal-bearing leachate that will require mitigation measures. This will also confirm the suitability of the waste rock storage facilities as potential sources of non-toxic closure cover and rock armor materials.

20.6.3 Archaeological Surveys

The Project does not have archaeological features that may pose a cultural liability to mine development. However, the area does contain historic hunting grounds and rock artifacts that are commonly found in the region. An archaeological land survey will be completed by the INAH prior to site preparation. In the event that artifacts are detected, the survey will need to incorporate salvage activities prior to the intended land liberation resolution.

20.6.4 Wildlife Management

As part of the MIA study and permitting process, it is recommended that the Project develop a wildlife management program (protection, identification, monitoring, conflict prevention, relocation and identification of potential risks). The program should be appropriate for each stage of mine development.

20.6.5 Forest Land Use Compensation

The Project requires a change in forest land use (scrub forest) to mining land use; this will require the execution of internal compensation measures to ensure continuity of environmental services in the area, mostly by the application of reforestation parcels. It is highly recommended that the Project explores potential reforestation areas, ideally as a buffer zone to separate mining activities from agricultural land. It is anticipated that a large portion of land will be required in order to maintain forest cover equilibrium between mine development and surrounding areas.

20.6.6 Reforestation Program

Owing to the required surface area and reforestation intensity for the environmental compensation, it is advisable to design, install and operate an on-site plant nursery (Mezquite, Acacia and Pine species) with a plant production capacity that is adequate for each stage of project development as well as the inclusion adequate space for seedlings maintenance.

20.6.7 Social and Community Engagement

It is important for the Project to identify a communication strategy in order to monitor stakeholder development and dynamics. Additional mechanisms are recommended to inform the general public and specific groups about the mine development plans and to ensure an adequate mechanism to integrate stakeholders and maintain a positive equilibrium in terms of project acceptance.

20.6.8 Land Ownership

Addressing land ownership and legal documentation issues, especially for Ejido land, tends to be a cumbersome process that may require more time than originally anticipated. Each property and parcel needs adequate documentation / titles, the land owners need to be present, and internal proprietor disputes can halt certain transactions, and/or stimulate real estate speculation. The land ownership and documentation issues have the potential to delay submittal of and requests for MIA approvals so should receive priority attention.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

The total evaluated project cost is estimated to be approximately $327.6 million. A summary of the direct capital costs is shown in Table 21-1 by process area and the indirect costs and total evaluated project costs are shown in Table 21-2.

Table 21-1: Direct Capital Cost Summary by Area

Plant Area	Description	Man-hours	Plant Equipment	Material	Labor	Construction Equipment	Total
000	General Site	212,646	$3,976,376	$4,551,077	$2,228,842	$991,603	$11,747,897
050	Mine	0	$0	$0	$0	$0	$0
100	Primary Crushing	97,531	$2,047,412	$2,615,146	$1,086,057	$553,920	$6,302,534
150	Primary Crusher Discharge	53,217	$2,849,747	$1,392,849	$608,348	$405,056	$5,256,000
200	SAG Feed Conveyor	80,694	$3,321,920	$1,703,518	$886,391	$386,764	$6,298,593
300	Grinding	330,915	$20,278,195	$6,248,933	$3,739,916	$2,021,443	$32,288,486
400	Leach & CCD	393,188	$21,272,444	$9,526,045	$4,599,897	$2,025,118	$37,423,504
450	Agitation Leach	133,490	$12,019,763	$3,774,976	$1,756,021	$617,921	$18,168,682
500	Refinery	29,345	$1,154,881	$974,557	$297,140	$109,522	$2,536,099
600	Tailing Facility	564,925	$3,740,537	$5,565,262	$6,010,781	$4,586,724	$19,903,304
650	Fresh Water Supply	102,421	$1,631,360	$2,668,001	$1,372,403	$474,144	$6,145,908
700	Electrical Utilities	321,129	$2,641,000	$9,044,517	$3,311,420	$38,879	$15,035,815
800	Reagents	79,249	$1,326,381	$2,060,957	$956,614	$369,999	$4,713,951
910	Administration Building	16,886	$38,000	$733,084	$174,657	$64,293	$1,010,034
930	Guard House	6,348	$26,000	$103,952	$69,318	$31,741	$231,010
940	Laboratory	26,364	$1,073,752	$786,862	$293,812	$27,006	$2,181,431
960	Mill Maintenance Building	65,923	$410,420	$1,963,337	$717,589	$71,424	$3,162,770
970	Truck/Mine Shop	72,553	$673,251	$2,249,518	$810,527	$126,211	$3,859,507
975	Truck Wash	11,786	$398,000	$240,194	$123,693	$38,149	$800,035
980	Fueling Station	10,729	$61,883	$254,674	$108,437	$22,108	$447,102
990	Powder Magazine Storage Building	3,660	$0	$60,045	$32,500	$6,131	$98,676
995	Detonator Storage Building	1,617	$0	$27,995	$14,268	$2,293	$44,556
	Freight		$6,315,306	$4,523,640			$10,838,946
	Taxes & Duties		$1,578,826	$1,130,910			$2,709,736
	Subtotal DIRECT COST	**2,614,617**	**$86,835,454**	**$62,200,048**	**$29,198,628**	**$12,970,447**	**$191,204,578**

Table 21-2: Indirect Capital Costs & Total Evaluated Project Costs

Total Direct Field Cost w/o Mine	**$191,204,600**
Mobilization	$956,023
Busing Costs (@$2.00 per MH)	$5,229,234
Contractor Fee (2)	In Directs
Construction Power	$478,012
Total Constructed Cost	**$197,867,868**
Management & Accounting (3)	$1,434,035
Engineering (4)	$12,428,299
Project Services (5)	$1,912,046
Project Control (6)	$1,434,035
Construction Management (7)	$11,472,276
EPCM Fee	$2,868,069
EPCM Construction Trailers	$382,409
Supervision of Specialty Construction	$868,355
Precommissioning	$260,506
Commissioning	$260,506
Temporary Facilities & Support	$956,023
Total Contracted Cost	**$232,144,427**
Capital & Commissioning Spare Parts (8)	2,170,900
Contingency (9)	28,117,839
Added Owner's Cost (10)	
Total Contracted and Owner's Cost	**$262,433,166**
Mining Equipment Cost, and Pre-production stripping	$40,976,679
Royalty Purchase	$12,000,000
Owners Costs	$12,150,000
Total Evaluated Project Cost	**$327,559,845**

- The capital cost estimate for the mine facilities was provided by Independent Mining Consultants (IMC) based on the major mining equipment being under lease with an option to buy.
- Capital costs for the TSF estimated from Bill of Materials provided by KP.
- Power line costs based on assumption of CFE approval of access to 230 kVA grid transmission lines and brought to site along the access road corridor to an onsite substation and transformer.
- M3 provided the capital cost estimate for the remaining process and ancillary facilities at the mine and mill areas.

The areas of the capital cost estimate include the following facilities, with the corresponding plant area:

000 Site General

This area includes systems or facilities that cross multiple areas of the plant. Included are the overall site plan, access roads, in-plant roads, site fencing, utility distribution, overland pipelines, drainage trenches and storm water diversions that are not associated with other areas of the project.

050 Mining
This area includes costs for mobile equipment and rolling stock not leased (included in operating cost).

100 Primary Crusher
This area consists of the primary crushing, auxiliary systems such as lube, screens, feeders and control room.

150 Primary Crusher Discharge Conveyor
This area comprises one conveyor for mill ores from the primary crushing facility conveyor to the crushed ore stockpile.

200 SAG Feed Conveyor
This area includes the feed system from crushed stockpile to SAG mill. It includes stockpile, reclaim tunnel under the coarse ore stockpile, the reclaim feeders, lime silo, and conveyor feeding the SAG mill.

300 Grinding & Classification
This area consists of a grinding line sized for 10,000 tpd including a SAG mill, ball mills, pebble conveyor, and cyclone classifiers.

400 Leach and CCD
This area consists of twelve 18 m X 19 m leach tanks with agitators to allow for 66 hours of leach residence time. Two parallel trains will treat 5000 tpd each. Four counter current decantation thickeners (CCD), pumps and tanks for recovery of dissolved metals and cyanide.

450 Merrill-Crowe
This area consists of all equipment necessary for clarification of pregnant solutions, de-aeration, and zinc addition steps.

500 Refinery
This area consists of precious metal recovery plant including filtration, retorts, and smelting to doré bars.

600 Tailing Facility
This area consists of cyanide detoxification, TSF, and process water reclaim lines.

650 Water Supply Systems
This area consists of the fresh water system, plant process water system, potable water system, and fire water system.

700 Electrical Utilities
This area consists of the main transformers, switchgear and related equipment in the main substation, overhead power transmission lines, and the distribution lines to the various mine and process areas. The electrical equipment rooms and low voltage distribution lines from the motor control centers to the process equipment are in the various process areas.

800 Reagents
This area consists of the reagent storage and distribution systems required for the process.

910 Administration Building
This area consists of the ancillary facilities required for the mine and mill; including administration/safety/medical building, change house/auditorium, training, laboratory, maintenance,

warehouse, scale house, truck wash, truck shop, tire change, mine truck fueling station, light vehicle fueling station, core shed, detonator and explosive magazine storage, a helipad and bus terminal.

930 Guard House
Security entry gate onto mine site with truck scale.

940 Laboratory
Assay laboratory and related equipment

960 Mill Maintenance Building
Office and work space to support mill maintenance facility. Building is contiguous with 970 Mine Truck shop and warehouse.

970 Truck Mine Shop
Maintenance facility with offices for rolling stock and site warehouse building.

975 Truck Wash
Heavy duty mobile equipment wash station with equipment for recycling waste water.

980 Fueling Station
Fuel tanks and pumps for diesel and gasoline service.

990 Powder Magazine Building
Explosives storage building meeting regulatory requirements.

995 Detonator Storage Building
Explosives storage building meeting regulatory requirements.

Capitalized Mining Costs
This area consists of the mining costs prior to Year 1 of the mine production schedule. These costs are capitalized and depreciated along with other capital expenditure line items. The mine maintenance facilities and mine truck shop are included in the ancillary facilities in Area 970 and 975. Partial mining preproduction costs are captured in project siteworks bulk fill costs, including tailings core fill requirements.

Owner's Costs
This area consists of owner's costs line items such as project team and overhead, permitting activities, legal fees, land acquisition and re-classification, early staffing, initial fills, etc.

21.2 EXCLUSIONS

The following items are excluded from the capital estimate unless otherwise noted in the Owner's cost:

a) IVA (Mexico VAT tax)
b) Finance and interest charges
c) Depreciation and depletion allowances
d) Environmental permits
e) Performance bond
f) Pre-operations expense
g) Land acquisition

h) Water rights acquisition
i) Coeur project management
j) Hiring and relocation
k) Public relations
l) Sunk costs prior to this estimate
m) Mine equipment
n) Mobile and shop equipment
o) Office equipment
p) Allowance for further expansion
q) Any propane tanks
r) Sprinkler system
s) Operating spare parts
t) Road repair and maintenance
u) Reagents (cost included in OPEX estimate)
v) Grinding or process media
w) First fill of lubricants
x) First fill Reagents
y) Plant Rescue Operations

21.3 BASIS OF ESTIMATE

21.3.1 General Conditions and Parameters

21.3.1.1 General Costs

The estimated costs are based on the Project being executed by experienced EPCM contractor(s) in the hard rock mining industry with a recent record of bringing projects on budget or under budget. In addition, it is assumed that all contracts and subcontracts are based on a lump-sum basis or a competitively bid unit cost basis, such as per m^3 of concrete placed. In particular, no time and material contracts are anticipated, nor should they be allowed in order to ensure that this budget is best maintained.

The general costs include the following:

- Mobilization 0.5% of direct field costs
- Bussing $2.00 / manhour
- Construction Power 0.25% of direct labor cost
- EPCM Trailers with Utility Setup 0.2% of direct field cost
- Owner Costs are listed based on data provided by the Project Manager

For horizontal contracts in process areas, it is assumed that at least two sufficiently sized self-performing local contractors are in place for all trades, such as civil, concrete, steel, architectural, mechanical, electrical, instrumentation and controls, and process piping. Certain contractors will have multiple trade capabilities.

The estimate assumes that vertical contracts encompassing all trades will be the method of execution for ancillary buildings such as labs and offices. Often smaller local contractors are suitable for this effort.

Sanitary facilities will be provided by contractor. Any existing or newly constructed facilities are for PMLP's use and in general will not be available to construction personnel.

No allowance has been made for fire protection during construction. PMLP will provide security services, access to construction water, communication links, and power for the contractors. The contractors will provide their own

communication system, equipment and radio frequencies. The contractors are responsible for their own drinking water and portable toilets, all utility hookups (e.g., satellite communications and power into construction trailer), delivery of construction water, all construction (portable or temporary) power.

PMLP will not supply any construction equipment (such as forklifts and crane for unloading or water trucks for dust suppression) to the Project.

The construction site will be available to contractors 24 hours per day Monday morning through Saturday afternoon.

It has been assumed that construction work areas would be accessible during all scheduled working hours. Allowance is not included in this estimate for stand-by time for inefficiencies resulting from work stoppages or interferences initiated by operations or revisions.

Contractors can have trailers and laydown yards in designated areas near the construction site. Construction personnel can park their construction vehicles in designated areas near the construction site. Personal vehicles will be parked near the contractor designated gate and personnel bused to the site by the contractors.

The contractors are responsible for all receiving of materials and equipment supplied by them. Any such items that have been received prior to construction, and received by EPCM or PMLP, must be loaded and transported to the construction site from the receiving area by the contractors responsible for installation of the equipment. In general, PMLP personnel will not receive shipments.

Contractors shall be responsible for security of received material including supply and preparation of fenced areas.

Contractors shall be responsible for quality control of safety. The EPCM shall be responsible for quality assurance of safety.

Conversion rates used in this estimate to convert from foreign currencies to US Dollars were:

1.00 US Dollar equals 0.72 Euros
1.00 US Dollar equals 13 Mexican Pesos
1.00 US Dollar equals 1.08 Canadian Dollar

21.3.1.2 Labor

Labor rates are based on prevailing merit shop wages. The costs for indirect labor hours as well as any contractor profit are captured in the direct labor hour rates. Craft labor has been estimated at the following overall average rates.

Table 21-3: Average Craft Labor Rates

	Direct Costs	Indirect Costs	Total Hourly Rate (US$)
Carpenter	6.11	included	6.11
Cement Mason	6.11	included	6.11
Electrician	11.36	included	11.36
Iron Worker	11.36	included	11.36
Laborer	6.11	included	6.11
Millwright	23.60	included	23.60
Operator	11.36	included	11.36
Painter	11.36	included	11.36
Plumber	23.60	included	23.60
Sheet Metal Worker	11.36	included	11.36
Truck Driver	11.36	included	11.36

21.3.1.3 Material Take-off and Field Labor

Material unit prices for the project were developed using costs gained through contacts with local regional suppliers, information from recently constructed projects and M3 in-house data.

Civil work quantities of general excavation, grading and backfill were taken off the site plot plan, general arrangement drawings and grading plans. In mountainous and hilly areas, in the absence of better information, the top 5 m of excavation has been assumed to be rippable, and thereafter blasting has been assumed. In alluvial areas, normal excavations were assumed. Takeoffs are broken into the categories of alluvium, rippable, and blastable. The civil engineering department performs rough grading takeoffs and the structural department handles foundation excavations and finish grading. The following factors are typically used on neat quantities. These are to be added by engineering and clearly spelled out.

a. Major Excavations add 5%
b. Finish grading add 50%
c. “Small” building foundation excavations add 100%
d. Roads including sub-base and graveling add 10%

Construction borrow material sources are assumed to be located within 6 kms. Excess borrow material haulage is not included. Bulk fill material will be supplied by Coeur with material from pre-production stripping.

Tailings core fill is by PMLP, supplied in 2 m lifts at a cost of $2.00 per m^3.

Road construction by PMLP is included in direct charges.

Tailings MTO supplied by PMLP’s representative (KP).

Civil miscellaneous items include:

- Liners
- Culverts
- Anticipated compaction water supply location
- ABC
- Paving – asphaltic or concrete, if any

- Fencing and gates
- Civil specialties – man holes, fire hydrants, drainage piping, multi-plate tunnels, bored tunnels, etc.
- Hilfiker walls must include their required backfill as a separate number
- Cattle guard
- Access road drainage serves as fill requirements.

Demolition costs have been estimated based on specific tasks; e.g., known items requiring removal.

Concrete quantities were developed from general arrangement and/or concrete drawings and experience with similar projects. An allowance has been made for lean concrete.

Building architectural costs are based on material takeoffs for new construction and allowances for any refurbishing and modification of existing buildings.

Structural steel quantities were developed from the general arrangement drawings and experience with similar installations. Quantities include allowances for miscellaneous steel including base plates, bracing, bolts and gussets.

- Mill floor deck design based on 300#/ft^2 loading.

Takeoffs have been made for mechanical steel including platework, abrasion resistant liners, ductwork, etc. based on the general arrangement drawings, mechanical drawings, equipment list and experience with similar installations.

Structural steel has four contributing components:

- Detailing of steel by M3 – used $0.20/pound (2014 prices)
- Fabrication of steel
- Erection of steel
- Freight from fabricator to Project site

Communications tower (30 MB) is supplied by PMLP. Existing road suffices for access.

Mechanical steel also has four contributing components:

- Detailing of steel by M3 – used $0.20/pound (2014 prices)
- Fabrication of steel
- Erection of steel
- Freight from fabricator to project site

General piping quantities were estimated based on experience with similar installations and MTO's provided from PID diagrams. The piping account material cost includes 3% of material costs for spool detailing. In addition, allowances have been made for the following specific systems:

- Large and custom fittings by estimate
- Medium bore pipes by estimate
- Small bore pipes by estimate
- Bolts allowance add 50%
- Special equipment such as hydraulic power units and HDPE fusion machines
- Tanks are to be tagged and estimated, including tanks and all fittings

Electrical takeoffs were performed using experience with similar installations and general arrangement drawings. An allowance has been made for duct bank concrete. This includes:

- Normal takeoff of power cable, control cable, conduit and cable tray
- Emergency power
- Lightning protection
- Offsite power lines - Cost estimate based on fair weather assumption of power route from Highway 40.
- Offsite substations - Cost estimate supplied by CFE
- Main plant substation
- Miscellaneous electrical gear
- High voltage transformers are redundant. Medium and Low Voltage transformers are not redundant.

Instrumentation allowance was based on experience with similar installations. Control valves are estimated from PID MTO's.

Construction equipment costs vary according to the tasks performed and the crew hours involved. Construction equipment is included as a direct cost which averages approximately $5/hour. Fuel costs are included in the equipment estimate. All liner handlers are included for both SAG and ball mill.

An additional 2% of total direct costs have been added for commodities in each area for quality assurance testing (e.g., compaction) for civil, concrete, piping, steel, and electrical. An additional 1% of total direct costs has been added for all commodities in each area for surveying of just the civil, concrete, and steel. Programming charges are included as a direct cost as part of the 000 Area. These costs have been estimated as 0.2% of the total direct cost.

21.3.2 Project Specific Interfaces and Conditions

The Project specific interfaces and conditions include the following

- Fiber optic network.
  - On site – included in estimate
  - Off site – not included in estimate – 30 MB Microwave estimate at $200,000 supplied by PMLP.
- Safety signage and road markings will be authorized by Coeur Safety Department. An allowance is made in the estimate for such signage.
- A leach field with septic tank has been estimated for sanitary facilities.
- Office will initially be comprised of modular units. Those units will be utilized for permanent administration facilities after construction.
- Truck scale. Included in estimate.
- The guard house will have manual gates. Guard house will be point of control for truck scale.
- Fencing includes the following:
  - Chain link fences 100 feet to each side of automated gates.
  - Chain link fence around substations and secure areas and substations.
  - Barb wire fence, four feet high at property line, estimated at 25,000 total meters.
- Fire protection system shall include the following:
  - Fire suppression loop/supply included in estimate
  - Specific building fire suppression sprinklers not included
- Fuel supply, storage, and containment systems include the following:
  - Tankage supplied by vendor (allowance in estimate)
  - Fuel containment areas by EPCM (allowance in estimate).

  - Construction fuel cost included in direct equipment cost estimate. Coeur may elect to supply fuel for environmental control.
- No allowance has been made for lost operations time.
- Water system interfaces are as follows:
  - Pump purchase/supply included in estimate. Coeur may elect to execute works.
  - Pump electrical supply included in estimate. Coeur may elect to execute
  - Pipeline by EPCM contractor.
- Plant air system as follows:
  - Plant air with receivers and dryers.
  - Instrument air with receivers and dryers.
- Crusher Liners
  - Initial set as Project Cost.
  - First replacement set as Coeur's Cost.
- Mill Liners
  - Initial set as Project Cost.
  - First replacement set as Coeur's Cost.
- Electrical Rooms
  - Most of the electrical rooms are assumed as pre-fabricated.
  - The electrical room for the mills will be concrete blocks.
- Fair weather assumption that CFE supplied high voltage power supply line 230 kVA from Route 40 along mine access road.

21.3.3 Mine Capital Basis

Estimated mining capital costs are summarized in Table 21-4. Contingency is not included in mining costs.

21.3.3.1 Cost Basis

The costs listed in Table 21-4 for all the major equipment were based on vendor supplied quotes and most of the minor equipment is based on historical quotes and estimates. Much of the historical cost information was current during 2013 and 2014. Older quotes for minor units were inflated as required.

Equipment quotes were received from several vendors, including Caterpillar, Komatsu, Atlas Copco and Sandvik. Quotes included delivery to the mine site, erection, import duties and taxes.

Equipment supplied under contracted blasting and tire service was not included in this estimate. The equipment from the explosives vendor includes the supply of ANFO storage tanks, powder and cap magazine buildings, and the explosives delivery vehicle. Under the tire service contract, tire handling equipment will be supplied by the vendor.

21.3.3.2 Total Capital Summary

Table 21-4: Project Preliminary Mining Capital Cost Estimates

Item	Value ($M)
Pre-Production Mining	24.70
Misc. Mining Capital	7.92
Equipment Leasing during pre-production	8.35
TOTAL	**$40.97**

21.3.3.3 Mine Capital Costs

The mining equipment capital costs are based the major mining equipment required for the pre-production year and the first three years of operations being leased for a five-year term. At the end of the lease, the leased equipment is purchased, with the exception of the 994 loader and the 390D excavator. Details for the proposed capital costs, are not included due to the proprietary nature of those costs. Equipment requirements during the first four years is also included as an operating cost. The lease costs were calculated from the equipment list shown on Table 16-8. Escalation and contingency are not included in capital costs. Lease costs are discussed in the operating cost summary.

Mine capital costs include:

1. The mine mobile equipment fleet.
2. An allowance for initial shop tools.
3. An allowance for initial spare parts inventory.
4. Mine engineering equipment (computers, survey equipment etc.).
5. Mine communication network & system.

Mine capital costs do not include:

1. Mine office buildings, or shop facilities. They are included elsewhere in the project capital list.
2. Mobile equipment that is not required by the mine. (i.e. no mobile units for the plant)
3. Infrastructure or process plant related costs.

A capital cost build up was created that includes the capital purchase of major and minor mining equipment. The table is not included due to the confidentiality of the majority of those costs. End-of-lease capital and costs were included in the capital cost estimates as a residual purchase.

21.3.4 Tailing Basis of Estimate

The principal objective of the TSF design is to ensure protection of the environment during operations and following mine closure. The design of the TSF takes into account the following requirements:

- Safe and secure long-term management of tailings and water
- Control, collection, preservation and removal of free draining water from the tailings during operations, for;
- Recycling as process water to the maximum extent practical
- The inclusion of monitoring features within the facility to ensure performance goals are achieved and design
- Criteria and assumptions are met; and
- Secure reclamation and closure of the impoundment after mining is complete

The TSF design will include an initial starter embankment and ongoing raises throughout the life of the facility.

This offers a number of advantages as follows:

- The ability to reduce capital costs and defer some capital expenditures until the mine is operating.
- The ability to refine design, construction and operating methodologies as experience is gained with local conditions and constraints.
- The ability to adjust plans at a future date to remain current with evolving best practice (engineering and environmental).

- To allow the observational approach to be utilized in the ongoing design, construction and operation of the facility. The observational approach can deliver substantial cost savings and a high level of safety. It also enhances knowledge and understanding of site-specific conditions.

Two embankments will be constructed to establish the TSF, including a main embankment along the south side of the basin and a smaller embankment constructed later at the north side of the basin. Natural topographical containment will form the northeast and west sides of the TSF. The TSF design section will include an initial starter embankment (Stage 1) for the south embankment with ongoing raises completed for the south and north embankments using centerline construction methods throughout the life of the facility. The initial starter embankment at the south, will be constructed of zoned rockfill with a composite liner on the upstream slope. The centerline raises will consist of zoned rockfill with a low permeability core zone. Transition zones will be established between the core zone and the embankment rockfill to ensure internal stability. The Stage 1 TSF will provide storage for two (2) years of tailings deposition.

The cost basis uses the feasibility design MTO's provided by Knight Piésold and estimated costs by M3 based on typical contractor costs in Mexico. The initial capital expenditure for the first stage of tailings operation is included in the initial capital cost estimate. Costs are based on a qualified contractor performing preparation, excavation, minor haulage, and compaction of materials. The cost basis for the core of the tailings dam assumes suitable material from the pre-stripping of waste material from the Abundancia pit and placed into 2 m lifts, spread by contractor, and compacted with the aid of the heavy haulage equipment. The supply cost for this material is $2 per m^3 placed. Similarly, a limited amount of suitable materials will be excavated from the core of the tailings impoundment by the Coeur.

Tailings pipelines, and recovered water piping and pumps, as well as earthwork for access roadways is included in the M3 estimate.

The feasibility design by Knight Piésold provides for up to 17 years of productive mine life, although eleven were used for the feasibility economic model. The tailings construction is planned for 8 phases, and the unit costs similar to the initial capital cost and applied to the sustaining capital in the economic model.

21.3.5 Indirect Cost Basis of Estimate

EPCM indirect costs are factored as percentages of constructed costs, which include direct costs, plus mobilization, plus any construction facilities and utilities:

Table 21-5: EPCM Indirect Costs

Activity	Cost Codes	Mexico
Management	100, 101, 105, 110	0.75%
Engineering	120 → 123, 130 → 140	6.50%
Project Services	150 → 155	1.00%
Project Controls	160 → 166	0.75%
Construction Management	200 → 215, 220, 250, 251, 255	6.00%
EPCM Fee	990.0901	1.50%
EPCM Construction Trailers	991.9102	0.20%

Vendors' representatives' costs during fabrication and construction are included in the general allowance listed on the summary page unless the Coeur requests that an additional amount be added to the line item for specific process equipment item in the estimate details.

Vendor Support has been included as follows:

- Supervision of specialty construction - 1% of Plant Equipment Costs
- Precommissioning - 0.3% of Plant Equipment Costs
- Commissioning - 0.3% of Plant Equipment Costs

The engineering and procurement included is based on the proposed activities for the current scope of work. An allowance has been made for field engineering. For the tabulation of this estimate, indirect field labor costs are included as direct field costs. The contingency included in this estimate is for the Scope of Work items as defined. It is not for items outside the present Scope of Work. The contingency is calculated at a percentage of the total contracted cost including commissioning and spare parts.

Costs are included for plant acceptance and initiation of operations as per the following:

- Mechanical completion – by Contractor/EPCM
- Pre-Commissioning of Unit Operations – by Coeur/EPCM
- Commissioning – by Coeur/EPCM
- Initial fills – by Coeur
- Startup – by Coeur/EPCM/Contractor
- Ramp-up – by Coeur
- Demonstration test – by Coeur

Taxes include the following:

- Payroll Taxes
- Gross Receipts Taxes (Sales and Use Taxes)

No escalation is included.

The accuracy of this estimate is assumed to be in the range of 15% plus 10% minus; i.e., the cost could be 10% lower to 15% percent higher than the estimate to the estimate total. Accuracy is an issue separate from contingency and is largely dependent on the bidding climate and the likely duration of time between preparing the estimate and the bidding of the contracts.

21.3.5.1 Freight Costs

Freight has been included as a percentage of equipment cost in the estimate for domestic sourced equipment. Some equipment quotes may include international sourced equipment freight which has been priced and listed separately

The 10% comprises the following components on an average basis:

Factory or in-transit warehousing	1%	
Freight	7%	
Duties, customs, taxes	2%	(Latin America)
	10%	

21.3.5.2 Temporary Construction Facilities

The estimate includes 0.5% of direct costs for temporary EPCM construction support facilities. This indirect cost item is meant to accommodate costs for temporary power lines, temporary water lines, general communications

and any emergency requirements associated with construction. This cost bucket is in addition to the construction camp.

Camp costs, including any busing and meals, are estimated at $42/man hour.

Construction power is included at 0.25% of direct costs. Mobilization is included at 0.25% of direct costs for subcontractors.

21.3.5.3 Spare parts

Capital or insurance spares as specified are direct costs. Such spares are typically identified (and sometimes tagged) individually and the costs for such captured within the capital cost estimate. These have been included as 2% of plant equipment costs. Spare mill motors are included in this allowance.

Commissioning spares are 0.5% of plant equipment costs. 2-year spares are part of operating costs and as such are **not** included in the capital cost estimates but are included in the financial model.

Operating or consignment spares are also **not** included as part of the capital cost estimate.

21.3.6 Coeur's Costs Basis of Estimate

Table 21-6: Coeur's Capital Costs

ITEM	COST	
Operator Training	$	498,417
Geotechnical	$	200,000
Security During Construction	$	627,692
First Aid and Medical during Construction	$	208,646
Construction Water	$	525,600
Coeur's Project Management	$	1,000,000
Early Staffing	$	1,875,000
Community Relations / Charity	$	598,500
Coeurs office fees	$	1,611,200
Personnel Safety Equipment	$	100,000
Builder's All Risk Insurance	$	1,200,000
ENVIRONMENTAL		
Development Impact Fees (SEMARNAT or EIS Fees)	$	1,915,961
Permits/consultants/other fees	$	606,495
COEUR'S WORKING CAPITAL COST SPARES		
SAG Mill Liners & Hardware – 1 set of spares	$	606,000
Ball Mill Liners & Hardware – 1 sets of spares	$	301,000
Primary Crusher Liner – 1 set	$	25,000
Shop Tools and Furnishings	$	250,000

21.3.7 Sustaining Capital Cost Estimate

Table 21-7: Sustaining Capital ($ million)

Year	1	2	3	4	5	6	7	8	9	10	11	LOM total
Mine	$4.22	$2.83	$0.25	$14.43	$19.74	$36.89	$7.14	$1.96	$0.04	$ -	$ -	$87.50
Process Plant	$ -	$9.46	$0.56	$0.56	$10.66	$1.04	$ -	$0.40	$0.40	$14.42	$1.64	$39.14

21.4 OPERATING AND MAINTENANCE COST ESTIMATE

The operating cost estimate was developed for the Project in support of 10,000 tpd milling operation and a pit design optimized to maximize economic grade and tonnes.

21.4.1 Mine Operating Cost Summary

Mine operating costs estimates are based on first principals for the scheduled production, equipment lease requirements, operating hours, hourly equipment operating costs, and manpower requirements. Mine operating costs were developed based on first principals for the mine plan and the equipment list. Neither escalation nor contingency are included in operating costs.

It is expected that the mining fleet will be leased for a 5 year period with an option to purchase after the lease has expired. Sustaining capital includes the purchase of the leased mining fleet after the 5th year. The unit costs for mining labor were developed based on labor rates for Coeur's Palmarejo mine. Fuel costs were set at $0.77 USD per liter based on publically available price forecasts, excluding VAT.

The annual lease payments have been shown as a line item in the Financial Model in Table 22-8 starting in Year -1 of mine operations. The preproduction cost of mine equipment due to the equipment leased during that time is included in the cash flow as a capitalized expense. The details of the lease costs are in operating expenses and were developed based on discussions with vendors that Coeur has had business relationships with in the past. The details are not presented here due to the confidential nature of those costs.

The mine plan stipulates that all ore will be mined on 5 m bench heights and most of the waste will be mined on 10 m bench heights. The amount of waste mined on 5 m benches is equivalent to 150% of the ore on each bench. The total material mined on 5 m benches equals 250% of all ore.

The equipment selection has been designed to accommodate the two bench heights (5 m and 10 m). The drilling equipment was split into two different size drill fleets and paired with each bench height. The drill fleet requirements were calculated based on the amount material mined on each bench height. The shovel requirements are applied only to the 10 m bench waste material. The front end loader is applied to 5 m benches in pit, stockpiles and will occasionally assist with the 10 m waste benches as needed. A large excavator will also assist with the extraction of 5 m ore.

Table 21-8 summarizes the total mine operating cost per time period along with the total mine capital cost. This table should provide a clear indication of the mine operating costs by year of operation.

Table 21-8: Mine Capital Cost and Operating Cost by Year

($US x 1000)						
	Mine Equipment					
Year	Initial Capital Cost	Sustaining Capital Cost	Mine Preproduction Development	Total Mine Capital	Operating Cost	TOTAL COST
-1	7,919		24,705	32,624		32,624
1		4,224		4,224	63,254	67,478
2		2,831		2,831	89,071	91,902
3		254		254	92,658	92,912
4		14,548		14,548	97,719	112,267
5		19,740		19,740	99,571	119,311
6		37,017		37,017	108,584	145,601
7		7,135		7,135	100,779	107,914
8		1,963		1,963	77,818	79,781
9		35		35	57,371	57,406
10		0		0	37,079	37,079
11		0		0	16,859	16,859
TOTAL	**7,919**	**87,747**	**24,705**	**120,371**	**840,762**	**961,133**

Table 21-9 summarizes the mine operating costs by the unit operations per time period. Preproduction is established to be 12 months in duration operating 1 shift/day. The cost per tonne in all periods is based on the total tonnage moved within the mine plan.

The estimated mine operating costs include:

1. Drilling, blasting, loading, and hauling of material from the mine to the crusher, low grade stockpile or waste storage facilities. Maintenance of the waste storage areas and stockpiles is included in the mining costs. Maintenance of mine mobile equipment is included in the operating costs.
2. Mine supervision, mine engineering, mine safety/emergency team, environmental team, geology and ore control are included in the G&A category.
3. Operating labor and maintenance labor for the mine mobile equipment are included.
4. Contractor cost for tire service is included in the general mine maintenance cost.
5. The contracted cost for the blasting supplied and for the blasting related equipment (i.e. ANFO/slurry truck, stemmer and explosive storage tanks) is included in the blasting operating cost.
6. Mine access road construction and maintenance is included. If mine haul trucks drive on the road, its cost and maintenance is included in the mine operating costs.
7. The small stockpile (958 ktonnes) that is generated during preproduction stripping is re-handed to the plant in Years 1 & 2.
8. The low grade stockpile (3,930 ktonnes) that is generated until year four is re-handled to the plant as needed throughout the mine life. The cost of re-mining the low grade stockpile is included in the operating costs.
9. A general mine allowance is included that is intended to cover mine pumping costs and general operating supplies that cannot be assigned to one of the unit operations. Included in the general mine cost is an allowance for ore control assaying, dispatch software, mine planning software, general labor and dispatch personnel cost.
10. The excavator operating costs include an allocation that is equivalent to moving 1/2 of the ore. However, the actual loading and hauling costs for ore are based on the loader productivity.

11. A general maintenance allowance is included that is intended to cover the general operating supplies of the maintenance group.
12. Allocations have been made for 10% wet hole blasting, a mine communications network and 10% for VS&A.
13. Haul truck fleet and operating requirements were based on haul time simulation. Haul profiles for input to simulation were measured from mine plans provided by Coeur.

IMC assisted Coeur with the development of the equipment requirements, operating and capital costs. The final selection and application of mining costs were established by Coeur personnel. The mine schedule was provided to IMC in several iterations. Haul profiles were measured from "Mineplan4". Later, the mine plan was updated and insufficient detail was available to update the haul profiles. The profiles applied to the truck requirements do not have a direct correlation to mine schedule but are a reasonable approximation to the haulage schedule developed.

The schedule that was used to determine the truck requirements did not maintain access to all working phases of the mine. The following assumptions were applied to resolve instances when access was cut off.

The mine operating costs in Table 21-9 do not include:

1. Lease costs for mine major and mobile equipment. A breakdown of the mine equipment, which is assumed to be leased, including lease costs.
2. Crushing, conveying or processing.
3. Road base material, man camp cost or travel expenses.
4. Reclamation recontouring costs.
5. IVA [Mexian Value Added Tax (VAT)] taxes.
6. A small auxiliary excavator for projects.

The mine is planned to work 2 shifts per day for 365 days per year. Five days (10 shifts) of loss time are assumed due to weather delays. The mine will operate using 3 crews.

Table 21-9: Summary of Mine Operating Costs (excluding equipment lease costs) – Per Total Tonne ($US x 1000)

Mining Year	Total Material (kt)	Drilled/ Blasted (kt)	Drilling	Blasting	Loading	Hauling	Auxiliary	General Mine	General Maintenance	G&A	TOTAL	Total Cost
-1	10,509	10,509	0.114	0.260	0.200	0.399	0.953	0.080	0.109	0.237	2.351	24,705
1	50,752	50,448	0.126	0.201	0.183	0.338	0.227	0.063	0.048	0.059	1.246	63,254
2	78,141	77,751	0.114	0.196	0.182	0.359	0.147	0.059	0.043	0.040	1.140	89,071
3	78,607	78,607	0.118	0.196	0.182	0.394	0.147	0.059	0.043	0.040	1.179	92,658
4	78,013	78,013	0.114	0.197	0.182	0.470	0.147	0.059	0.043	0.041	1.253	97,719
5	77,999	77,235	0.108	0.195	0.182	0.502	0.148	0.058	0.043	0.041	1.277	99,571
6	78,177	77,277	0.107	0.194	0.182	0.615	0.147	0.058	0.043	0.042	1.389	108,584
7	73,563	73,563	0.113	0.197	0.182	0.575	0.156	0.059	0.044	0.044	1.370	100,779
8	50,112	47,895	0.102	0.194	0.182	0.674	0.230	0.060	0.050	0.062	1.553	77,818
9	28,834	28,521	0.134	0.212	0.182	0.909	0.351	0.070	0.063	0.070	1.990	57,371
10	14,363	14,363	0.181	0.242	0.183	1.103	0.598	0.089	0.092	0.093	2.582	37,079
11	6,818	6,818	0.141	0.195	0.194	1.282	0.464	0.079	0.066	0.052	2.473	16,859
TOTAL	**625,888**	**621,000**	**0.116**	**0.199**	**0.183**	**0.529**	**0.198**	**0.061**	**0.047**	**0.050**	**1.383**	**865,467**
PERCENT			**8.4%**	**14.4%**	**13.2%**	**38.3%**	**14.3%**	**4.4%**	**3.4%**	**3.6%**	**100.0%**	
Per Tonne Drilled/Blasted			**0.117**	**0.201**							**1.394**	

21.4.2 Process Plant Operating Cost Summary

This section addresses the following costs:

- Process Plant Operating & Maintenance Cost
- General and Administrative Cost

The process plant operating costs are summarized by area and then by cost element of labor, electric power, reagents, maintenance parts and supplies and services. Below is a summary of the cost by area for the life of mine (Table 21-10).

Table 21-10: Process Operating Cost Summary

	LOM Production	
Process Tonnes	37,326,000	
	LOM Cost	**Unit Cost per Tonne**
Primary Crushing	$13,791,511	$0.37
Grinding	$249,351,177	$6.68
Leach & CCD	$140,342,316	$3.76
Merrill-Crowe	$34,808,681	$0.93
Refining & Smelting	$18,773,563	$0.50
Tailings Disposal	$51,357,659	$1.38
Ancillary	$40,105,825	$1.07
Total	**$548,530,732**	**$14.70**

21.4.2.1 Process Labor and Fringes

The process plant operating and maintenance labor costs were derived from a staffing plan and are based on labor rates from an industry survey for this region and modified where necessary. The annual salaries include overtime and benefits for both salaried and hourly employees. The benefit rate used is 42% for salary and 31% for hourly. A summary of the labor annual cost is shown in Table 21-11.

Table 21-11: Process Labor Summary

	Personnel	**Annual Cost**
Operations	49	$822,480
Maintenance	49	$895,828
Lab	5	$75,576
Total	**103**	**$1,793,884**

21.4.2.2 Reagents

Consumption rates were determined from the metallurgical test data or industry practice. Budget quotations were received for reagents supplied from local sources where available, with an allowance for freight to site or from historical data from other projects.

A summary of process reagent consumption and cost is shown in Table 21-12.

Table 21-12: Process Plant Reagents

Reagents	kg/t	$/kg
Grinding		
Flocculant	-	$4.50
Leaching & CCD		
Lime	1.40	$0.14
Sodium Cyanide	1.00	$2.29
Flocculant	0.09	$4.32
Lead Nitrate	-	$3.26
Merrill-Crowe		
Zinc Dust	900.00	$3.75
Diatomaceous Earth	0.05	$0.77
Flocculant	0.01	$4.32
Refining and Smelting		
Flux	0.33	$1.25
Tailings Disposal		
Lime	0.10	$0.14
Sodium Cyanide	-	$2.29
Sodium Meta-bisulfite	0.8	$0.84
Copper Sulfate	0.07	$3.85
Flocculant	0.01	$4.32
Ancillary		
Antiscalant	0.05	$2.30

21.4.2.3 Maintenance Wear Parts and Consumables

Grinding media were estimated on a kilogram/tonne basis. The SAG mill and Ball mill liners were estimated on a set basis, while the primary crusher liner was based on a kilogram/tonne basis. These consumption rates and costs are shown in Table 21-13.

Table 21-13: Grinding Media and Wear Items

Grinding Media & Wear Parts	kg/t	$/kg
Primary Crusher Liners	0.01	$5.02
SAG Mill Liners	0.05	$4.93
Ball Mill Liners	0.13	$3.15
SAG Mill - Balls	0.71	$1.17
Ball Mill - Balls	1.71	$1.29

An allowance was made to cover the cost of maintenance for the facilities and all items not specifically identified. The allowance made as a percent of the direct capital cost of equipment for each area; the rate used was 5%. The annual cost is estimated to be $4.2 million.

21.4.2.3.1 Electrical Power

Electrical power costs were based on current pricing at a rate of $0.083 per kWh. The electric power consumption was based on the equipment list connected kW, discounted for operating time and the anticipated operating load level. The estimated annual power cost is $13.5 million.

21.4.2.3.2 Process Supplies and Services

An annual allowance was estimated for items such as lubricants, diesel fuel, safety items and tools. Also included is the water charges which are estimated at $1.5 million. The allowances were estimated from historical information or from other operations and projects. The annual cost is estimated to be $3.8 million.

21.4.2.3.3 General and Administration (G&A)

G&A costs include labor and fringes for the administrative employees, accounting department, purchasing, human resources, community relations, safety and environmental departments. Also included are office supplies, communications, legal fees, community relations, and insurance cost and outside services. These costs are summarized in Table 21-14.

A staff of 86 employees for the departments mentioned above and labor rates that are based on an industry survey for this region and modified where necessary are used in calculating the labor cost. The benefit rate used is 42% for salary and 49% for hourly. All other G&A costs are annual allowances.

Table 21-14: General Administration Summary

Cost Item	LOM	
	Cost	$/tonne ore
Labor & Fringes	$33,934,389	$0.91
Accounting (excluding labor)	$525,000	$0.01
Safety & Environmental (excluding labor)	$525,000	$0.01
Human Resources (excluding labor)	$525,000	$0.01
Security (excluding labor)	$4,725,000	$0.13
Office Operating Supplies and Postage	$1,100,000	$0.03
Maintenance Supplies	$1,192,215	$0.03
Maintenance Labor, Fringes, and Allocations	$215,671	$0.01
Power Allocation 15% Process Ancillary Facilities	$0	$0.00
Water Charges	$310,403	$0.01
Propane/Fuel	$1,050,000	$0.03
Communications	$1,050,000	$0.03
Small Vehicles	$785,000	$0.02
Legal & Audit	$6,050,000	$0.16
Consultants	$4,200,000	$0.11
Community Relations	$7,850,000	$0.21
Janitorial Services	$525,000	$0.01
Insurances	$10,500,000	$0.28
Meals	$3,796,000	$0.10
Subs, Dues, PR, and Donations	$525,000	$0.01
Travel, Lodging, and Meals	$1,100,000	$0.03
Recruiting/Relocation	$1,000,000	$0.03
Total General & Administrative Cost	**$81,483,678**	**$2.18**

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

The financial evaluation presents the determination of the projected NPV, payback period (time in years to recapture the initial capital investment), and the 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 silver and gold bullion. The estimates of capital expenditures and site production costs have been developed specifically for the Project and have been presented in earlier sections of this report.

22.2 MINE PRODUCTION STATISTICS

Mine production is reported as mineralized material and overburden from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this Report.

The life of mine mineralized material quantities and mineralized material grades are presented Table 22-1.

Table 22-1: Life of Mine Mineralized Material, Waste Quantities, and Grade

	Tonnes (kt)	Gold (g/t)	Silver (g/t)
Mill Feed Mineralized material	37,326	0.17	105.77
Low Grade Mineralized material	3,930	0.07	45.00
Waste	583,674		

The table displays material mined, it does not display material re-handled from stockpile. The mined material is either placed on the stockpile or fed directly to the plant.

22.3 PLANT PRODUCTION STATISTICS

The design basis for the process plant is 10,000 tonnes per day at 92% mill availability. The recoveries are projected to average for the life of the mine as follows:

- Gold 61.0%
- Silver 84.0%

The estimated life of mine metal production is presented in Table 22-2.

Table 22-2: Life of Mine Production

Gold (kozs)	128
Silver (kozs)	106,643

22.3.1 Marketing Terms

The silver and gold bullion is assumed to be shipped to refinery and the terms are negotiable at the time of the agreement. The refinery charges and payable metal calculated in the financial evaluation are presented in Table 22-3.

Table 22-3: Refinery Terms

Silver and Gold Bullion	
Payable Silver (%)	99.42%
Payable Gold (%)	99.20%
Treatment Charge ($/oz net weight received)	$0.17
Refining Charge ($/oz per payable gold)	$0
Transportation Charges ($/shipment)	$14,500.00
Shipment per year (Low, High)	16-65
Insurance $/$1,000	$0.30

22.3.2 Capital Expenditure

22.3.2.1 Initial Capital

The financial indicators have been determined assuming internal financing and leasing of the mine mobile equipment for the initial capital. Any acquisition cost or expenditures prior to start of the full Project period have been treated as "sunk" cost and have not been included in the analysis.

The total initial capital carried in the financial model for new construction and pre-production mine development is $327M expended over a 2 year period. The initial capital includes all pre-production capital expenditures for design, procurement and construction of project facilities, including owner's costs and contingency. The cash flow will be expended in the years before production and a small amount carried over into the first production year.

The initial capital includes all pre-production capital expenditures for design, procurement and construction of project facilities, including owner's costs and contingency. Declaration of commercial production is defined as at the end of the third consecutive month of 60% of nameplate or greater mill tonnage throughput.

The initial capital is presented in Table 22-4.

Table 22-4: Initial Capital

	$ in millions
Mining (includes preproduction)	$53.0
Process Plant	$262.4
Owner's Cost	$12.2
Total	**$327.6**

22.3.3 Sustaining Capital

A schedule of capital cost expenditures during the production period was estimated and included in the financial analysis under the category of sustaining capital. The total life of mine sustaining capital is estimated to be $126.6 million. This capital will be expended during an 11 year period.

22.3.4 Working Capital

All working capital is recaptured at the end of the mine life and the final value of these accounts is $0. During the mine life, working capital is comprised of the following components:

Trade Receivables: Receipts on doré sales are assumed to be received 15 days after revenue recognition.

Supplies Inventory: An allowance of $3.0 million for plant consumable inventory is estimated in each of years -1 and 1. It is assumed that this stock will remain constant throughout the production life.

Trade and Other Accounts Payable: Trade payables are equal to 30 days of annual non-labor expenditures.

The year-to-year working capital impacts are shown in the cash flow model in Table 22-8.

22.3.5 Salvage Value

An allowance for salvage value has been included in the cash flow analysis of approximately $11.4 million.

22.3.6 IVA Taxes

The VAT tax (in Spanish, the "Impuesto al Valor Agregado" (IVA)) is assumed to be recovered within the year spent and was not included in the cash flow.

22.3.7 Revenue

Annual revenue is determined by applying estimated metal prices to the annual payable metal estimated for each operating year. Sales prices have been applied to all life of mine production without escalation or hedging. The revenue is the gross value of payable metals sold before treatment charges and transportation charges. Metal sales prices used in the evaluation are as follows:

- Gold $1,350.00/ounce
- Silver $22.00/ounce

These price assumptions are considered conservative compared to allowances made for calculating NI 43-101 metal price assumptions. This model allows 60% for an average of previous 36 month metal pricing and 40% component based on futures forecasts for 24 months.

22.3.8 Exchange Rate

Currency used in this report is in United States dollars. The Mexican peso (MXN) to United States dollar (USD) forecast exchange rate was determined based on publicly published forecasts by leading US and Canadian financial institutions. Within the financial model, exchange rate was assumed to equal 13$MXN per $USD.

22.4 TOTAL OPERATING COST

The total operating cost over the life of the mine is estimated to be $43.32 per tonne of mineralized material processed, excluding the cost of the capitalized pre-stripping. The total operating cost includes mine operations, process plant operations, general administrative cost, refining charges and shipping charges. The table below shows the estimated total operating cost by area per metric tonne of mineralized material processed.

Table 22-5: Life of Mine Total Operating Cost

Total Operating Cost	$/mineralized material tonne
Mine	$22.52
Equipment Lease	$3.08
Process Plant	$14.70
General Administration	$2.33
Refining Charges	$0.69
Total Operating Cost	**$43.32**

22.5 TOTAL PRODUCTION COST

The average Total Cash Production Cost over the life of the mine is estimated to be $45.59 per tonne of mineralized material processed. Total Cash Production Cost is the Total Operating Cost plus royalty, salvage value, reclamation and closure costs, and the new mining royalties. Total Production Costs which includes depreciation plus the Total Cash Production Cost equals $57.76 per tonne of mineralized material processed.

22.5.1 Mining Royalties

Production costs include two mining royalty taxes:

- A 7.5% royalty tax has been applied to include income from mining activities. The tax is calculated on a basis of earnings before interest, income taxes, depreciation and amortization (i.e. EBITDA),
- a 0.5% royalty tax has been applied on revenue from precious metals

Both of these taxes are assumed to be deductible against income before the calculation of corporate income tax.

22.5.2 Net Smelter Return Royalty

The purchase of the NSR royalty by Coeur is treated as a capital cost for the purposes of this evaluation.

22.5.3 Reclamation and Closure

An allowance for reclamation and closure was included in year 12 of the cash flow of $15.7 million.

22.5.4 Salvage Value

At end of the mine life an estimated salvage value was shown of $11.4 million which was based on 10% of the cost of capital equipment.

22.6 TAXATION

22.6.1 Income Tax

The corporate income tax rate of 30% in effect in Mexico for 2014 was included in the economic model. This is applied to net profits of the company.

Income taxes paid over the life of the mine are estimated to be $102.1 million.

22.6.2 Depreciation for Tax

Capital expenditures are assumed to be depreciated for calculation of Mexican Corporate Income Tax on a straight line basis over a ten year period.

22.6.3 Tax Loss Carry Forward Balances

Tax loss carry forwards for Mexican corporate income tax purposes equal to $39.5 million MXN are available (with varying expiry periods) as at the start of the construction period. Tax losses can be carried forward for a period of ten years.

22.6.4 Inflation/Deflation of Asset Balance

The financial model is computed in real dollars. Accordingly, in the financial model, zero inflation or deflation is applied to calculations of depreciable asset balances and loss carry forwards. (In accordance with Mexican income tax laws, depreciable asset balances and loss carry forwards will increase each year based upon inflation and therefore retain their full value in real terms).

22.7 EXCLUDED COSTS

Sunk costs (e.g. drilling costs and corporate overheads) incurred prior to project construction commencement are excluded from the economic analysis.

22.8 BONUS/PROFIT SHARING

Profit sharing is not applied to the Project.

22.9 PROJECT FINANCING

The Project is assumed to be internally financed, with the exception of the mining mobile equipment, which is financed for 5 years and then purchased.

22.10 NET INCOME AFTER TAX

Net income after tax amount over the life of the Project is estimated to be $240.6 million.

22.11 NET PRESENT VALUE, INTERNAL RATE OF RETURN, PAYBACK

The economic analysis indicates that the Project has an NPV using a 5% discount rate of $87.6 million and a projected IRR of 9.5%. Payback of the initial capital occurs at a period of 6.9 years after commencement of commercial production. Over the life of the Project, cash operating costs are $15.29/oz payable silver and production costs of $20.38/oz payable silver.

22.12 SENSITIVITY ANALYSIS

Table 22-6 compares the base case project financial indicators with the financial indicators when different variables are applied. By comparing the results it can be seen that the metal prices have the most impact on the Project while the capital and operating cost have a lesser impact.

Table 22-6: Sensitivity Analysis

		NPV @ 5% (thousands)	IRR	Payback (years)
Base Case		$87,605	9.5%	6.9
Metal Prices	+20%	$326,665	20.3%	3.0
	-20%	-$179,304	-5.5%	N/A
Operating Cost	+20%	-$54,409	2.1%	9.8
	-20%	$225,335	15.9%	4.3
Capital Cost	+20%	$23,443	6.0%	8.8
	-20%	$143,607	14.0%	6.5
Recovery	+20%	$323,192	20.2%	3.0
	-20%	-$174,362	-5.2%	N/A

Table 22-7: Metal Price Sensitivity Analysis

Silver Price	NPV @ 5%	NPV @ 10%	IRR %	Payback
	(millions)			(years)
$31	$541.8	$337.9	28.9%	2.3
$28	$391.7	$223.7	23.0%	2.7
$25	$240.1	$108.5	16.6%	3.8
$22	**$87.6**	**-$7.7**	**9.5%**	**6.9**
$20	-$15.2	-$86.5	4.2%	9.4
$18	-$131.1	-$174.7	-2.4%	N/A
$16	-$273.4	-$282.3	-11.9%	N/A

Table 22-8: Financial Model

	Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22
Mine Operations																										
Mine to Mill Ore																										
Beginning Inventory (kt)	32,438	32,438	32,438	32,438	32,438	29,436	26,177	22,528	18,877	15,991	13,242	9,591	8,157	4,819	1,168	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	32,438	-	-	-	3,002	3,259	3,649	3,651	2,886	2,749	3,651	1,434	3,338	3,651	1,168	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	32,438	32,438	32,438	29,436	26,177	22,528	18,877	15,991	13,242	9,591	8,157	4,819	1,168	-	-	-	-	-	-	-	-	-	-	-	-
Gold (g/t)	0.19	-	-	-	0.18	0.13	0.15	0.17	0.17	0.13	0.27	0.23	0.19	0.26	0.23	-	-	-	-	-	-	-	-	-	-	-
Silver (g/t)	114.42	-	-	-	117.29	130.42	121.61	102.83	97.68	90.28	146.32	113.87	88.11	125.76	115.09	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs.)	197	-	-	-	18	13	17	19	16	11	32	11	20	30	9	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs.)	119,333	-	-	-	11,321	13,665	14,267	12,071	9,064	7,979	17,175	5,250	9,456	14,762	4,322	-	-	-	-	-	-	-	-	-	-	-
High Grade Ore to Stockpile																										
Beginning Inventory (kt)	958	958	958	958	654	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	958	-	-	304	654	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	958	958	654	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold (g/t)	0.09	-	-	0.11	0.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Silver (g/t)	61.91	-	-	79.04	53.95	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs.)	3	-	-	1	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs.)	1,907	-	-	772	1,134	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Low Grade Ore to Stockpile																										
Beginning Inventory (kt)	3,930	3,930	3,930	3,930	3,725	2,746	1,790	494	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	3,930	-	-	205	979	956	1,296	494	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	3,930	3,930	3,725	2,746	1,790	494	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold (g/t)	0.07	-	-	0.07	0.07	0.07	0.08	0.09	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Silver (g/t)	44.99	-	-	43.34	43.50	47.44	46.00	41.28	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs.)	9	-	-	0	2	2	3	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs.)	5,685	-	-	286	1,369	1,458	1,917	656	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Ore																										
Beginning Inventory (kt)	37,326	37,326	37,326	37,326	36,817	32,182	27,967	23,022	18,877	15,991	13,242	9,591	8,157	4,819	1,168	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	37,326	-	-	509	4,635	4,215	4,945	4,145	2,886	2,749	3,651	1,434	3,338	3,651	1,168	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	37,326	37,326	36,817	32,182	27,967	23,022	18,877	15,991	13,242	9,591	8,157	4,819	1,168	-	-	-	-	-	-	-	-	-	-	-	-
Gold (g/t)	0.17	-	-	0.09	0.15	0.11	0.13	0.16	0.17	0.13	0.27	0.23	0.19	0.26	0.23	-	-	-	-	-	-	-	-	-	-	-
Silver (g/t)	105.77	-	-	64.66	92.77	111.60	101.80	95.50	97.68	90.28	146.32	113.87	88.11	125.76	115.09	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs.)	209	-	-	1	22	16	20	21	16	11	32	11	20	30	9	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs.)	126,925	-	-	1,058	13,824	15,124	16,184	12,726	9,064	7,979	17,175	5,250	9,456	14,762	4,322	-	-	-	-	-	-	-	-	-	-	-
Waste (ktonnes)	583,674	-	-	10,000	45,813	73,536	73,662	73,868	74,349	74,528	69,912	46,461	25,183	10,712	5,650	-	-	-	-	-	-	-	-	-	-	-
Total Material Mined	621,000	-	-	10,509	50,448	77,751	78,607	78,013	77,235	77,277	73,563	47,895	28,521	14,363	6,818	-	-	-	-	-	-	-	-	-	-	-
Rehandle Ore (ktonnes)	4,888	-	-	-	304	390	-	-	764	900	-	2,217	313	-	-	-	-	-	-	-	-	-	-	-	-	-
166.73	625,888	-	-	10,509	50,752	78,141	78,607	78,013	77,999	78,177	73,563	50,112	28,834	14,363	6,818											
Process Plant																										
Ore processed (kt)	37,326	-	-	-	3,306	3,649	3,649	3,651	3,650	3,649	3,651	3,651	3,651	3,651	1,168	-	-	-	-	-	-	-	-	-	-	-
Gold (g/t)	0.174	-	-	-	0.175	0.124	0.147	0.166	0.154	0.113	0.271	0.134	0.180	0.259	0.234	-	-	-	-	-	-	-	-	-	-	-
Silver (g/t)	105.77	-	-	-	113.77	122.25	121.61	102.83	87.30	79.11	146.32	72.05	84.42	125.76	115.09	-	-	-	-	-	-	-	-	-	-	-
Contained Gold (kozs.)	209	-	-	-	19	14	17	19	18	13	32	16	21	30	9	-	-	-	-	-	-	-	-	-	-	-
Contained Silver (kozs.)	126,925	-	-	-	12,093	14,342	14,267	12,071	10,245	9,281	17,175	8,457	9,909	14,762	4,322	-	-	-	-	-	-	-	-	-	-	-
Gold & Silver Bullion																										
Recovery Gold (%)	61%	0%	0%	0%	61%	52%	57%	60%	58%	50%	69%	55%	62%	68%	66%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Recovery Silver (%)	84%	0%	0%	0%	85%	85%	85%	84%	82%	81%	86%	80%	82%	85%	85%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%	0%
Recovered Gold (kozs.)	128	-	-	-	11	8	10	12	11	7	22	9	13	21	6	-	-	-	-	-	-	-	-	-	-	-
Recovered Silver (kozs)	106,643	-	-	-	10,235	12,218	12,149	10,112	8,419	7,526	14,845	6,761	8,108	12,608	3,662	-	-	-	-	-	-	-	-	-	-	-
Payable Metal																										
Payable Gold (kozs.)	127	-	-	-	11	8	10	12	10	7	22	9	13	21	6	-	-	-	-	-	-	-	-	-	-	-
Payable Silver (kozs.)	105,790	-	-	-	10,153	12,120	12,052	10,031	8,352	7,466	14,726	6,707	8,043	12,507	3,633	-	-	-	-	-	-	-	-	-	-	-

	Total	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22
Cash Flow Summary ($000)																										
Revenues																										
Gold Revenue ($ 000)	$171,081				$15,274	$10,168	$13,203	$15,667	$14,099	$8,832	$29,244	$11,592	$17,484	$27,692	$7,825	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Silver Revenue ($ 000)	$2,327,375				$223,361	$266,650	$265,145	$220,684	$183,745	$164,250	$323,973	$147,562	$176,941	$275,148	$79,916	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Revenues	$2,498,456	$0	$0	$0	$238,635	$276,818	$278,348	$236,351	$197,844	$173,082	$353,217	$159,154	$194,425	$302,840	$87,741	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Operating Cost	$1,617,076	$0	$0	$0	$129,950	$178,583	$181,300	$185,995	$180,615	$183,066	$168,664	$142,351	$121,946	$103,202	$41,404	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Salvage Value	-$11,394				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	-$11,394	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Reclamation & Closure	$15,698				$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$15,698	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Total Production Cost	$1,621,381	$0	$0	$0	$129,950	$178,583	$181,300	$185,995	$180,615	$183,066	$168,664	$142,351	$121,946	$103,202	$30,010	$15,698	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Operating Income	$877,075	$0	$0	$0	$108,685	$98,235	$97,049	$50,356	$17,230	-$9,984	$184,552	$16,803	$72,479	$199,638	$57,731	-$15,698	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Government Royalty and Taxes	$80,199	$0	$0	$0	$0	$9,345	$8,752	$8,670	$4,958	$2,281	$865	$15,608	$2,056	$6,408	$16,487	$4,769	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Working Capital	$0	$0	$0	-$1,520	-$646	$2,428	$160	$2,112	$1,140	$1,219	-$8,586	$5,812	-$3,127	-$5,996	$3,760	$3,242	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Capital Expenditures																										
Initial Capital	$327,560	$0	$185,838	$141,721	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Sustaining Capital	$126,644	$0	$0	$0	$4,224	$12,292	$819	$14,994	$30,398	$37,930	$7,135	$2,365	$437	$14,416	$1,635	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Income Taxes	$102,098	$0	$0	$0	$10,808	$16,345	$16,142	$1,709	$0	$0	$17,492	$0	$0	$39,601	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Cash Flow after Taxes	$240,574	$0	-$185,838	-$143,241	$93,007	$62,681	$71,496	$27,095	-$16,987	-$48,976	$150,473	$4,643	$66,860	$133,217	$43,369	-$17,225	$0	$0	$0	$0	$0	$0	$0	$0	$0	$0
Cummulative Cash Flow after Taxes		$0	-$185,838	-$329,079	-$236,073	-$173,391	-$101,896	-$74,801	-$91,787	-$140,763	$9,710	$14,353	$81,213	$214,430	$257,799	$240,574	$240,574	$240,574	$240,574	$240,574	$240,574	$240,574	$240,574	$240,574	$240,574	$240,574
						1.0	1.0	1.0	1.0	1.0	1.0	0.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Economic Indicators after Taxes																										
NPV @ 0%	0%	$ 240,574																								
NPV @ 5%	5%	$ 87,605																								
NPV @ 10%	10%	$ (7,699)																								
IRR		9.5%																								
Payback		6.9																								

23 ADJACENT PROPERTIES

Arcelia Gold Corp., through its wholly owned subsidiary, Arcelia Gold, S.A. de C.V. owns two (2) Mining Concessions (La Peña, Título #204828 and El Niño, Título #236219) that are adjacent to and contiguous with the El Choque Tres, El Choque Cuatro, La Preciosa, and San Juan Mining Concessions of PMLP;

Canasil Resources, Inc., through its wholly owned subsidiary, Minera Canasil, S.A. de C.V. owns two (2) Mining Concessions (Carina, Título #233344 and Reducción Victoria Fracción B, Título #235845) that are adjacent to and contiguous with the San Juan Mining Concession of PMLP.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION PLAN

24.1.1 Introduction

This section of the Report provides the outline for the Project Execution Plan which forms part of the framework for the project schedule and capital cost estimate. The execution plan covers the period post feasibility and application of environmental permitting, and finishing with plant commissioning and reaching full production.

Going forward, Coeur will advance the Project by completing a number of tasks (Figure 24-1). These include receiving permits and Environmental Assessment (MIA) to construct and operate the mine and finally constructing the mine site.

The major assumptions for execution of the Project include the following:

- Coeur Board of Director approval for expenditure prior to advancing this project.
- Rights-of-way and land rights are finalized for the Project prior to construction activities.
- All construction and environmental permits required to commence construction are in place.
- Adequate project financing is in place at the start of construction.
- Basic engineering is started a minimum of six months prior to initiating construction of major earthwork aspects.
- Major equipment specifications and procurement are prepared in the first three months of basic engineering so that vendor certified drawings can be obtained and long-lead items can begin manufacture.
- Construction contracts are developed for earthworks and civil concrete during basic design. Contracts are bid and subsequently awarded four to six months after design begins.
- The power line and fresh water system are designed, bid and awarded midway through basic engineering.
- The access road is constructed immediately after environmental approval.
- Construction water is sourced from new wells near the site. Construction power is supplied by diesel-fired generators until power is available from the power line.
- Separate contracts are awarded for earthworks, concrete, structural steel erection, mechanical installation, electrical installation, instrumentation and controls. Unit quantity bids will largely be employed. Contracts may or may not be with the same contractor (i.e. mechanical/electrical).
- An EPCM construction approach will be utilized as outlined above.

Table 24-1: La Preciosa Project Execution Plan

	Y-2				Y-1				Y1	
	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2
Permitting										
Basic Engineering										
Long Lead time Procurement										
Construct facility										
Plant Commissioning										
First Commercial Production										

24.1.2 Description

The project execution plan describes, at a high level, how the Project will be carried out. This plan contains an overall description of what the main work focuses are, Project organization, the estimated schedule, and where important aspects of the Project will be carried out.

The project execution plan proposed incorporates an integrated strategy for EPCM. The primary objective of the execution methodology is to deliver the Project at the lowest capital cost, on schedule, and consistent with the Project standards for quality, safety, and environmental compliance.

24.1.3 Objectives

The Project execution plan has been established with the following objectives:

- To maintain the highest standard of safety so as to prevent incidents and accidents;
- To design and construct a process plant, together with the associated infrastructure, that is cost-effective, achieves performance specifications and is built to high quality standards;
- To design and operate the mine using proven methodologies and equipment;
- To optimize the project schedule to achieve an operating plant in the most efficient and timely manner within the various constraints placed upon the Project; and
- To comply with the requirements of the conditions for the construction and operating license approvals.

24.1.4 Plan of Approach

24.1.4.1 Philosophy

This section describes the execution plan for advancing the Project from the current Feasibility Report stage to production. The project execution plan will ensure that key project processes and procedures are in place that will:

- Develop a project schedule from completion of the feasibility study through permitting, construction and commissioning;
- Consider significant project logistics;
- Develop and implement site communications, construction infrastructure, and water supply for an early and efficient startup;
- Plan for early construction mobilization;
- Develop and execute project control procedures and processes;
- Perform constructability reviews;
- Define Work Breakdown Structure (WBS);
- Implement project accounting and cost control best practices suitable for downstream owner usage;
- Issue a cost control plan and a control budget, and monitored continually; and
- Oversee project accounting.

Coeur intends to utilize an EPCM approach utilizing multiple hard money and low unit cost prime contracts for construction management, as the recommended method for executing the Project. The capital cost estimate is based on this methodology. The Project is located in an area with sufficient qualified contractors, and experienced mining personnel. Construction is to be performed by Mexican companies, normally with at least two such companies for each of the skill trades.

Mine development pre-production work activities as well as site road construction will be performed by Coeur.

Some usual items which could affect the Project are:

- Ability to start work early that does not require significant engineering; e.g., civil rough grading;
- Experience of the qualified firms considered and their typical and proposed approach; and
- An approach that utilizes the best resources available (matching contractors to the size of each contract).

An EPCM approach is the basis for the capital cost estimate. This approach provides for contracts that would include the skill trades of civil, concrete, structural steel, mechanical, piping, electrical and instrumentation.

The majority of mechanical and electrical equipment required for the Project will be procured within North America. Concrete and building construction materials will be sourced locally in northern Mexico. Structural and miscellaneous steel, piping, tanks, electrical and miscellaneous process equipment will be sourced within Mexico, to the extent practical.

The Coeur PMO group will generally provide corporate level project oversight and control through the process of detailed engineering / construction / and project hand-off to operations. Important duties of the PMO will include, but not be limited to such things as monitoring adherence to accepted Coeur procurement (and other general business, safety, environmental, ethics, etc.) policies, engineering approvals, inspection and approval of work completed, management of change requests, reporting of project earned value and schedule performance, contract management, and verification and final approval of contractor invoices for payment.

The PMO will coordinate monitoring project performance using Primavera P6 with an interface to the Coeur's Oracle Enterprise Resource Planning (ERP) system.

The PMO, comprised of an anticipated four to six Coeur employees, most likely stationed at the project site and Durango office, will form the staff of the vice president accountable for project delivery.

24.1.4.2 Engineering

The detailed engineering schedule is based on interim approval to be granted in early basic engineering starting in Q1 Y-2, and full EPCM release in Q3 Y-2. The design is scheduled to be 60% complete by the end of Q4 Y-2. Engineering will be completed upon receipt of all pertinent vendor data.

Engineering will be done to match the plant protocol for drawing titles, equipment numbers and area numbers. Design will produce drawings in metric format. Preliminary drawings and specifications will be done in English, while field specifications approved for construction will be in dual versions with Spanish prevailing contractually.

A site conditions specification will be done to ensure that vendors are aware of the site conditions. Individual equipment specifications will be done. Individual equipment specifications will be based on performance requirements plus minimum hardware requirements.

Engineering control will be maintained through drawing lists, specification lists, equipment lists, pipeline lists and instrument lists. Control of Engineering Requisitions for Quote will be performed through an anticipated purchase orders list. Progress will be tracked through the use of the lists mentioned.

Concrete reinforcing steel drawings will be done using customary bar available in Mexico. Reinforcing bar will be fully detailed to allow either site or shop fabrication. Shop fabrication is usually preferred due to delivery and cost advantages.

Structural steel will be detailed by the EPCM contractor using 3D computer aided design software (e.g. TEKLA). Mechanical steel will be detailed by the EPCM utilizing a similar approach. This will allow fabrication of steel prior to the award of steel installation contracts.

Coeur review of engineering progress and design philosophy will be a continuous process.

24.1.4.3 Procurement

Procurement of long delivery equipment and materials is scheduled with their relevant engineering tasks. This will ensure that the appropriate vendor information is incorporated into the design drawings and that the equipment will be delivered to site and supports the overall project schedule. Particular emphasis will be placed on procuring the material and contract services required to establish the temporary construction infrastructure required for the construction program.

Under general supervision of Coeur, procurement of major process equipment will be by the EPCM contractor, acting as Agent for Coeur through the use of owner approved purchase order forms. This will include all of the equipment in the equipment list as well as all of the instruments in the instrument list. Some instruments will be part of vendor equipment packages. In addition, structural steel, electrical panels, electrical lighting, major cable quantities, specialty valves and special pipe will be purchased. Contractors will be responsible for the purchase of common materials only.

The EPCM contractor will establish a list of recommended pre-qualified vendors for each major item of equipment for approval by Coeur. The EPCM contractor will prepare the tender documents, issue the equipment packages for the bid, prepare a technical and commercial evaluation, and issue a letter of recommendation for purchase for approval by Coeur. Coeur through the assistance of the EPCM contractor will conduct the commercial negotiations with the recommended vendor and advise the EPCM contractor of the negotiated terms for preparation of the purchase documents. When approved, the EPCM contractor will issue the purchase order, track the order, and expedite the engineering information and formally accept delivery of the equipment to the site maintaining care custody and control of the equipment until it is turned over to the installation contractor.

Major equipment may be sourced on a world-wide basis, assessed on the best delivered price and delivery schedule, fit-for-purpose basis. Equipment and bulk material suppliers will be selected via a competitive bidding process.

Equipment will be typically purchased FOB at the point of manufacture for domestic suppliers or nearest shipping port for international shipments. The EPCM working in association with Coeur's logistics coordinator will arrange all shipments of equipment and materials for the Project and arrange for ocean and overland freight to the job site.

The EPCM contractor will be responsible for the receipt of the major equipment and materials at site; with the quantities and conditions of the items received reviewed and accepted by a representative of the Company's PMO. The EPCM contractor will manage and care and custody of materials until the equipment and materials are turned over to the installation contractor for storage and safe keeping until installed. Bulk piping and electrical materials and some minor equipment will be made part of the construction contracts, and as such will be supplied by the various construction contractors. It is expected that each construction contractor provide for the receipt, storage, and distribution of materials and minor equipment they purchased.

24.1.4.4 Inspection

The EPCM contractor will be responsible to conduct QA/QC inspections for major equipment during the fabrication process to ensure the quality of manufacture and adherence to specifications. Levels of inspection for major equipment will be identified during the bidding stage, which may range from receipt and review of the manufacturer's quality control procedures to visits to the vendor's shops for inspection and witnessing of shop tests prior to shipment of the equipment. Where possible, third-party independent inspectors close to the point of fabrication will be contracted to

perform this service in order to minimize the travel cost for the Project. Some assistance will be provided by the EPCM engineering design team if and as required. The EPCM shall ensure a qualified person performs inspection of major components.

24.1.4.5 Expediting and Logistics

The EPCM contractor will also be responsible to expedite the receipt of vendor drawings to support the engineering effort as well as expediting the fabrication and delivery of major equipment to the site. An expediting report will be issued at regular intervals outlining the status of each purchase order in order to alert the Project of any delays in the expected shipping date or issue of critical vendor drawings. Corrective action will be taken to mitigate any delay.

The EPCM contractor will also be responsible to coordinate the logistic of the equipment and material shipments from point of manufacture to site, including international shipments through customs.

24.1.4.6 Project Services

The EPCM contractor will be responsible for management and control of all Project service activities and ensure that the team has appropriate resources to accomplish Coeur's objectives. Coeur will provide corporate level oversight through a PMO.

24.1.5 Construction

24.1.5.1 Construction Methodology

The construction program is scheduled to start in in the third quarter after commencing basic engineering. Contract bid calls will start in in the second quarter following commencement of basic engineering. The work includes clearing and grubbing of the plant site, mass earthwork for site development, project access road and in-plant roads. Concrete foundations and underground utilities for the process buildings and other support structures will be constructed beginning in early Q2 Y-1.

Construction work is scheduled for 18 months from ground breaking to the commencement of commissioning. Earthworks for process facilities will commence as soon as the contractor can be mobilized to the field, after tall required permits have been obtained. This work will include surface water control diversions, process building foundations and tailings facilities.

24.1.5.2 Construction Management

Construction Management will be performed by the EPCM contractor using prime contracts for civil/concrete and structural/mechanical/electrical/piping/instrumentation. The contracting plan is based on utilizing local contractors to execute the construction work packages to minimize mobilization and travel costs. The EPCM contractor will pre-qualify local contractors and prepare tender documents to bid and select the most qualified contractor for the various work packages. Some work packages will include the design, supply, and erection for specific facilities which are specialized in nature. The EPCM contractor team will be comprised of individuals capable of coordinating the construction effort, supervising and inspecting the work, performing field engineering functions, administering contracts, managing warehouse and material management functions, and performing cost control and schedule control functions. These activities will be under the direction of a Coeur resident construction manager and a team of engineers, supervisors, and technicians. There would also be a commissioning team to do final checkout of the Project. Some site services will be contracted to third party specialists, working under the direction of the resident EPCM construction manager. Construction service contracts identified at this time include the following:

- Field survey services;

- QA/QC testing services
  - Concrete
  - Welding and x-ray
  - bolt torque
  - instrument calibration

There would also be a commissioning team to do final checkout of the Project. The commissioning stages from mechanical completion to startup are as follows.

- **Mechanical Completion –** Mechanical Completion consists of the confirmation of fabrication, assembly and nonfunctional testing of a specific system(s) or facility to confirm the integrity of the construction and installation. Verification of system(s) or facilities mechanical completion includes confirmation that the final construction is in accordance with the project drawings, specifications, and industry standards as well as all regulatory requirements from any authorities having jurisdiction.

  Small-scale operational checks such as bumping motors, circuit checks, piping/instrumentation valves and controls and dry running (without any processing or test material) of conveyors and other equipment are also included. Mechanical Completion is the precursor to the Pre-Commissioning and Commissioning stages of a project.

- **Pre-Commissioning** – Pre-commissioning activities are preliminary efforts to test and check the individual units, systems and/or sub-systems within a logical area (Filter Press within Area 100 is one example). This is done to ensure that the components of any given system and/or sub-system perform as designed. An activity essential to the pre-commissioning process is the testing with water or other liquids of those systems and/or sub-systems that contain a process flow during facility operations.

- **Mechanical Completion Punch List** – The Mechanical Completion Punch List is a list of items (structural, mechanical, piping, electrical, and instrumentation) that were found to be deficient or incomplete during the Pre-Commissioning testing and checking phase. During this phase these problems will be corrected and indicated as such on the punch list. When punch list items are complete, they should be logged and so indicated to ensure the full functionality of the items/systems before Commissioning. All punch list work should be complete including any additional work requested by the Coeur (where possible) prior to commencement of the Commissioning phase.

- **Commissioning** – The Commissioning phase follows completion of the Pre-Commissioning activities performed by the Contractors and other Pre-Commissioning team members. Commissioning involves testing, checking and start-up at the system-level. Commissioning occurs prior to the facility actually becoming operational. Commissioning entails running and testing all areas of each system to ensure the system performs as anticipated and to correct any remaining problems prior to Plant Operational Start-up. Testing and calibration of the facility occurs with water only. Material handling systems, including conveyor belts, are run empty.

- **Start-up** – Start-up constitutes the commencement of actual plant operations. During the Start-up phase, systems that were tested in isolation in the Commissioning phase are initialized together so that product may be introduced and the plant put into production mode. The order in which systems are initialized during Start-up should be determined beforehand and followed explicitly to prevent damage to plant equipment. The Start-up order should be organized to ensure that when product is introduced into the system the follow on systems or processes are also functional and in an operational mode. During Start-up, the plant runs continuously with product to ensure that the plant and facilities are running and producing the end product as originally intended. Coeur's PMO will coordinate this phase in preparation of handoff to operations.

24.1.6 Contracting Plan

Contracting is an integral function in the Project's overall execution. A combination of vertical (e.g., all trades for laboratory), horizontal (e.g., electrical trade for process area), and design construct (e.g., fire protection) contracts may be employed as best suits the work to be performed, degree of engineering and scope definition available at the time of award.

Concrete batch plants are expected to be based on-site, to supply specified concrete mixes to all construction contractors. There will not be a dedicated construction camp as Durango is located 45 minutes away by asphalt roads.

The civil/concrete contract will cover clearing, grubbing, bulk excavation, engineered fill, geomembrane lining, and all concrete forming, rebar, and placement. This approach will result in economy of scale and eliminate interfacing issues which would arise if multiple contractors were employed. The contractor will require only one major mobilization for all work.

As part of the contracting strategy, a list of proposed contract work packages has been developed to identify items of work anticipated to be assembled into a contract bid package. Depending upon how the Project is ultimately executed and the timing, several work packages may be combined to form one contract bid package. The following table represents highlights of the proposed contract work package list:

Table 24-2: Proposed Contract Work Package List

No.	Bid Packages:	Comments
1	Materials Testing	Soils, Concrete & Structural Materials
2	Survey	Confirm Existing Terrain. Create Topo of Roadway, Plant Site Areas, etc., and QA/QC of contractor installations.
3	230 kV Power Lines	#3,4 Packaged together
4	230 kV Substations	#3,4 Packaged together
5	Field Electrical Distribution - Sub Station to Process Areas, Camp & Water Pumping	Duct Banks for on-site electrical installations.
6	Water Wells & Supply System -	Coeur prerogative to self-manage work
7	Site Piping Distribution	Including Firewater
8	Septic System - Sewer Piping, Plant & Distribution Field	
9	Clearing, Grubbing, Site Excavation & Site Preparation - All Areas	Some work possibly by Coeur
10	Concrete Work	Form, Supply, Place (including rebar and embeds)
11	Mechanical #1, Mechanical #2	Including crushing & mills
12	Electrical #1, Electrical #2	
13	Piping #1, Piping #2	
14	Field Erected Tanks	Should Include Field Weld Specifications
15	Instrumentation	
16	Merrill-Crowe/Smelting building.	
17	Structural Steel Erection #1	
18	Structural Steel Erection #2	
19	Instrument Calibration	

24.1.7 Project Schedule

The schedule consists of milestones, permitting, basic engineering & long-lead procurement, detail engineering and construction & start up activities. The summary schedule is shown at the end of this section.

The construction schedule is to occur within two years including start-up and ramp-up and assumes that Basic Engineering, permitting and feasibility engineering will commence Q1 of Y-2. The key milestone assumptions that drive the schedule are the completion of environmental permitting in Q2 of Y-2 leading to a full project release in Q3 of Y-2. The schedule allows for 5 days of weather delays to weather during the seasonal rains.

Construction/Project execution of 18 months is expected. Coeur intends to engage an EPCM contractor with significant mining project experience in Mexico, which involved large projects and contractors to insure timely project delivery. Construction Completion and Turn-over Procedure

The Construction Completion and Turn-over Procedure is part of the Project Execution Plan. Contractors are to enter into contractual agreements jointly with Coeur and the EPCM contractor acting as agent for Coeur Mining to perform certain portions of the work, which includes quality control of their work.

The PMO will manage the turnover between construction and operations.

24.1.8 Quality Plan

A project specific, quality plan will be developed, incorporated into the project execution plan, and implemented before mobilization. The quality plan is a management tool for the EPCM contractor, through the construction contractors, to maintain the quality of construction and installation. The plan, which consists of different manuals and subcategories, will be developed during the engineering phase and available prior to the start of construction. The Coeur PMO will strictly monitor the Quality Plan for compliance.

24.1.9 Commissioning Plan

The Commissioning Plan will be developed, incorporated into the Project Execution Plan and implemented to insure a step-by-step, documented process and procedure for all mechanical, process, electrical/instrumentation completion, checkout and pre-operational testing. Pre-operational testing and commissioning will take place concurrent with mechanical completion. Pre-operational testing is currently scheduled to commence in Q4 Y-1 and wet commissioning and start-up is scheduled to commence in Q1 Y1.

The Commissioning Plan will also be project specific and is characterized as the transition of the constructed facilities from a status of “mechanically” or “substantially” complete to operational as defined by the subsystem list that will be developed for the Project. The commissioning group will systemically verify the functionality of plant equipment, piping, electrical power and controls. This test and check phase will be conducted by discrete facility subsystems. The tested subsystems will be combined until the plant is fully functional. Start-up, also a commissioning group responsibility, will progressively move the functional facilities to operational status and handover to operations.

24.1.10 Health and Safety Plan

The Health and Safety Plan (HASP) will be established for the construction of the Project and any other authorized work at the Project site. The HASP covers all contractor, EPCM, and Coeur personnel working at the site. All contractors will be contractually bound to plan and perform all work in compliance with the conditions of the HASP.

The HASP specifies regulatory compliance requirements, training, certifications and medical requirements necessary to complete the project for all personnel and contractors involved in the Project. A written safety plan will be required of each contractor as a deliverable before mobilization to site.

24.1.11 Project Organization

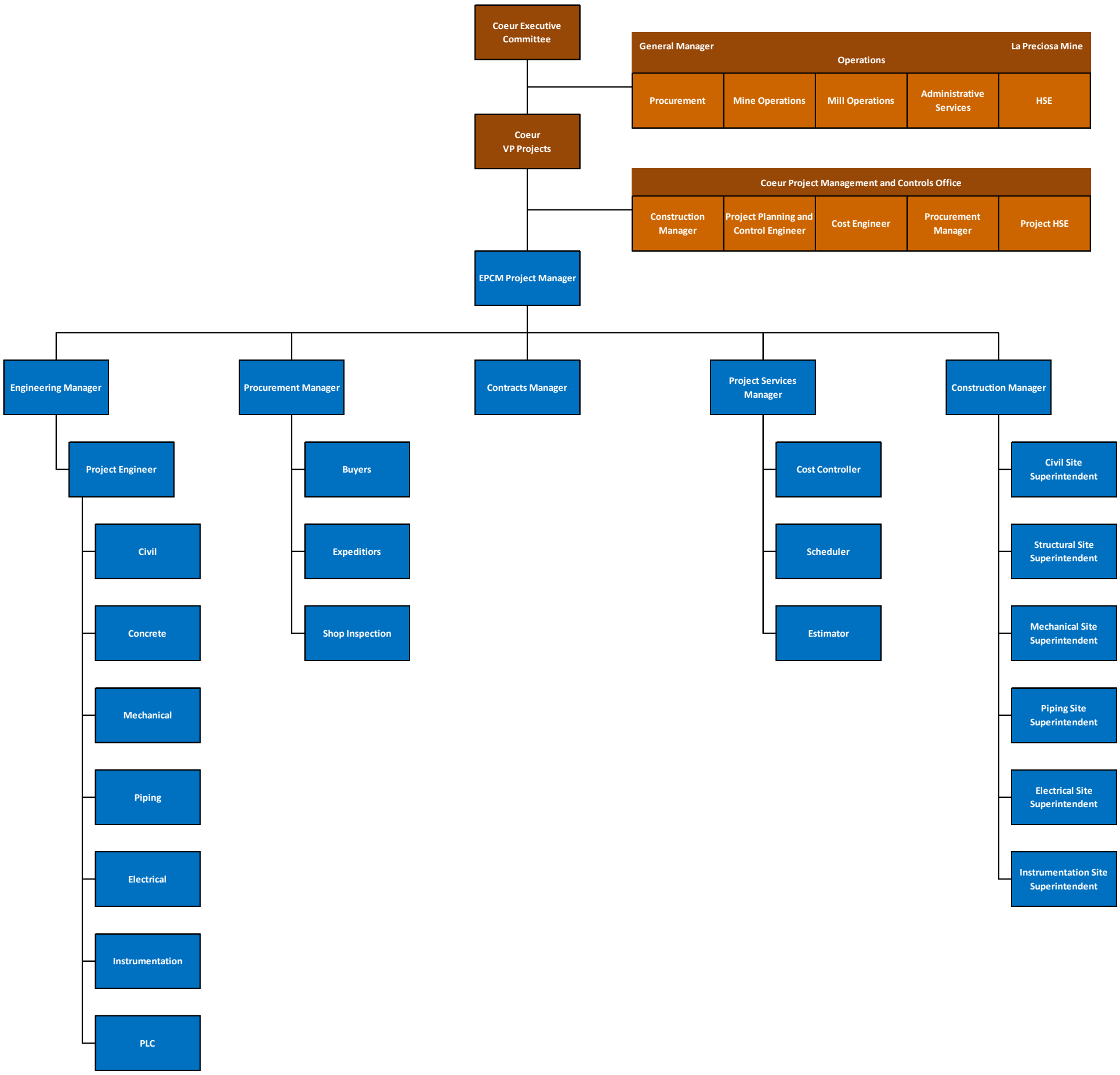


Figure 24-1: Project Organization Block Diagram

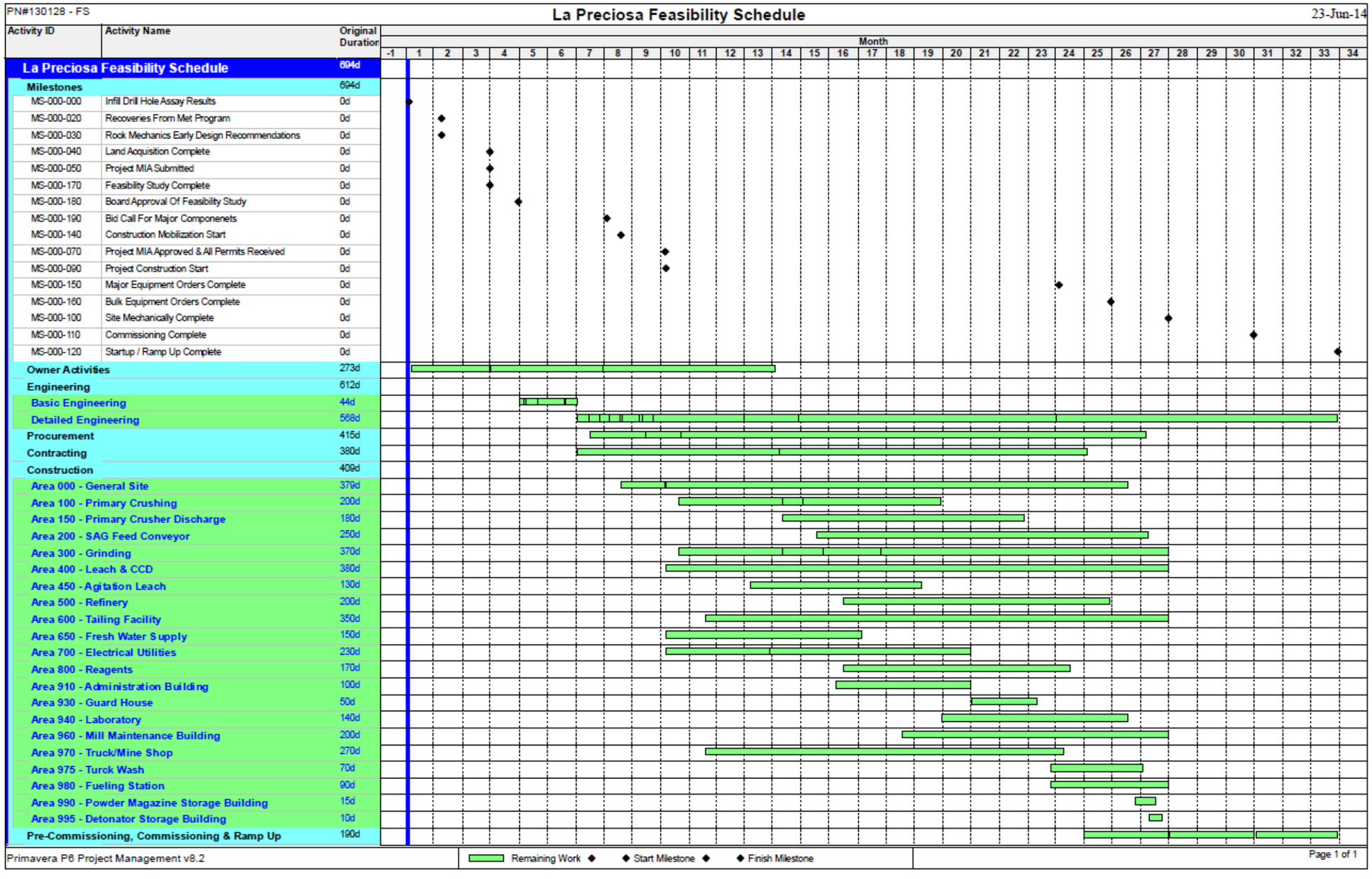


Figure 24-2: Project Summary Schedule

24.2 METALLURGICAL RISKS AND MITIGATION

Metallurgical samples received for the test programs fall within the current open pit outline and were representative of the material that would be mined.

Individual metallurgical results are highly variable, and the data taken collectively, correlate poorly in linear relationships used to established recovery curves. However, the data do establish head grade - tail grade trends that are likely to be observed in commercial operations. Optimization of leach parameters during detailed engineering may reduce variability risk.

25 INTERPRETATIONS AND CONCLUSIONS

This Feasibility Study of the Project indicates that the project has economics at $22/ounce Ag and $1,350 Au that indicate an IRR of 9.5%, and a payback of 6.9 years. The Project has an IRR that increases to 18% at $25/ounce Ag and $1,500/ounce Au, with a payback of 3.5 years.

The Project is located in an area with moderate climate, workable topography and regional work force that has experience in construction and operations of mining projects. The Project is located near a major population center and has proximity to infrastructure. The permitting process in Mexico is relatively straightforward and a reasonable permitting schedule is achievable. The preparation of permitting documents is essentially complete, with only the source of power and final water permit to be resolved, prior to submission.

The Project mine plan was developed using conventional hard rock open pit mining methods. The mine production schedule was developed with the goal of filling the mill at 10,000 tpd and maximizing the Project's return on investment. Further mine planning work needs to be done to improve the mining vertical advance rates to minimize the risk.

Detailed capital and operating costs were developed for the Project based on budgetary equipment quotations, in house data and material quantity take-offs.

The mineral resources for the Project were developed using a computer block model, the drillhole data, and the geologic and structural interpretations. The mineral reserves and mineral resources are summarized in Table 25-1 and Table 25-2.

Table 25-1: Project Mineral Reserves – July 29, 2014

Category	Tonnes	Average Grade (g/tonne)		Contained Ounces	
		Au	Ag	Au	Ag
Proven	18,365,000	0.200	113.3	118,100	66,920,000
Probable	18,959,000	0.148	97.7	90,400	59,523,000
Total P+P	**37,324,000**	**0.174**	**105.4**	**208,500**	**126,443,000**

Table 25-2: Project Mineral Resources, exclusive of Reserves as of 29 July 2014

Category	Tonnes	Average Grade (g/tonne)		Contained Ounces	
		Au	Ag	Au	Ag
Measured	6,839,000	0.186	84.1	40,900	18,485,000
Indicated	10,540,000	0.160	88.3	54,100	29,920,000
Total M&I	**17,379,000**	**0.170**	**86.6**	**95,000**	**48,405,000**
Inferred	1,889,000	0.126	77.5	7,700	4,705,000

1. Metal prices used for estimation of Mineral Reserves were $22 per troy ounce of silver and $1,350 per troy ounce of gold. Metal prices used for the estimation of Mineral Resources were $25 per troy ounce of silver and $1,400 per troy ounce of gold.
2. A NSR cutoff of $21.93/tonne ($24.17/short ton) was used, based on the following parameters: NSR = [(Ag price per ounce - refining charge) × plant recovery × payable recovery] + [(Au price per ounce -refining charge) × plant recovery × payable recovery]
3. Rounding of short tonnes, grades and troy ounces, as required by reporting guidelines, may result in apparent differences between tonnes, grades and contained metal contents.
4. Inferred Mineral Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be considered for estimation of Mineral Reserves.
5. U.S. Investors are cautioned that the term "mineral resource" is not defined or recognized by the U.S. Securities and Exchange Commission.

25.1 PROJECT RISKS AND OPPORTUNITIES

The Project is a well explored and studied project located close to centers of population and infrastructure. Considerable knowledge and understanding of the deposit has been obtained in the last several years that have improved the prospects of this large scale open pit silver mine. Many of the risks and opportunities of this project are common to all such mining projects located in Mexico.

Recent studies have largely eliminated typical project risks of processing design and recovery issues, slope stability issues, and have optimized mine and plant design. Mine plans are flexible and can offer a variety of mine life durations based on metals prices. The plant design is conventional in nature with no esoteric new processes or chemicals and the mine plan anticipates conventional drill, blast, excavation, haul technology with well-known costs and performance.

25.1.1 Risks to the Project

The Project is anticipated to be a large scale (150,000 mtpd) open pit operation feeding about 10,000 mtpd to a processing plant. The majority of the ore mineralization is contained within veins that will be mined and separated from the waste rock. There is risk in the continuity and grade of those veins and risk in the high strip ratio if mining costs are not well controlled.

Control of surface land for the mining activity has lagged due to issues with clear titles on most parcels of land that have caused delays in obtaining those surface rights required prior to initiating mining/construction activities. The majority of the land is now controlled and the MIAs (environmental documents) can be pursued when some immaterial issues regarding water and power are resolved.

Water rights have been difficult to obtain from the local office of CONAGUA and additional rights remain to be resolved. The existing well locations and pipeline location will need to be approved and included in the MIA for both access road construction and for project operation. The two wells seem to be adequate for the projected processing plant needs, but alternative water sources are being evaluated.

Electrical Power is available from the local grid, either from an existing 230 kV power line located approximately 12 km south of the Project of via a connection to a 115 kV power line located approximately 41 km to the west of the Project. However, the local governmental agency responsible for power supply and connections (CFE) has not responded to requests for connections that have been proposed by the company. Power can be obtained at lower rates than is being offered by CFE via wheeling of power from other sources through the CFE grid.

Environmental permit activities need to be completed and the anticipated approval time would be about 6-9 months.

Mining concession maintenance fees and activities are required to maintain the status of mining concessions. This involves conducting exploration or other activities on the property as well as filing annual reports. In the event of Project delay, those activities would have to be included in the on-going activities and expenses at the site.

Mexican mining tax and royalty rates increased in recent years but regulations governing those new laws are still not available and may change interpretations of how those laws will be enforced or put into effect. Those tax and royalty laws were recently enacted and at a time of low metal prices. There are approximately two more years of the current government before the next major elections. Those elections may result in a relaxation, a continuation or other changes in those tax and royalty laws.

25.1.2 Opportunities

Due to the current lower metals price environment, the availability of mining and processing equipment and associated terms and conditions for acquiring that equipment are very advantageous.

The start-up of the Project production will take approximately 2 years from date of approval. This is a quick, but attainable, timetable for bringing the production on-line. The plant and mine design are state-of-the-art but also well-established technologies.

Extended mine life is likely with improved metals prices due to the nature of the deposit. The deposit is not closed at depth and still has some inferred resources that could be added to the ultimate mine production. There are multiple veins within the deposit and a quality ore control program could bring additional resources into the production matrix. There is additional exploration potential in the region and Coeur holds a large mining concession position.

The Property is close to population centers without being too close to generate opposition, is close to power and transportation infrastructure, and has a ready workforce available locally and in the region.

Reducing purchased water needs are through a combination of water in-flow into the pit at depth and from the capture of rain water during the rainy season.

The opportunity to stockpile waste rock within mined out portions of the pits can reduce the haulage profile and there-by reduce the mining costs and equipment acquisition costs for the Project.

Power costs are available at better rates than are being offered by CFE. The quoted reduced rates are included in the current evaluation but there is a highly competitive market developing in the energy sector that could result in further reductions in those costs.

25.2 METALLURGY CONCLUSIONS

Feasibility level metallurgical studies for the recovery of silver and gold from the Project have been completed. The deposit may be processed in a conventional crushing, grinding, and Merrill-Crowe silver recovery circuit with detoxified tailings reporting to a tailings storage facility.

Metallurgical results obtained from the composites in this study were considered representative of the mineralization represented by the selected core samples. The metallurgical results indicate variable metallurgical response in the mineralized zones.

Mineralogy indicated the major portion of the silver occurs as silver sulfide, acanthite, Ag_2S. Traces of silver–selenium sulfide, possibly aguilarite, and complex silver–antimony–zinc sulfide were observed. Silver-bearing grains ranged in size from 15 -100 μm. The silver-bearing grains, were locked in quartz, or were associated with pyrite, galena, lead carbonate, and iron oxide or iron hydroxide. Acanthite occurred as very fine veins within quartz. Gold grains were contained in iron hydroxide that was enclosed in quartz-rich particles. The majority of the located gold occurrences in the samples would be considered liberated. Gold particles, in binary associations, with pyrite and non-sulfide gangue minerals, were generally low in gold content, and were of smaller average diameter than the liberated gold particles. Gold was found as inclusions within the pyrite or gangue particle, and in association with silver sulfide minerals.

The Bond crusher work index averaged 9.7 kWh/t ± 3.4 and varied from 21.3 to 5.2 kWh/t for all samples tested. Bond abrasion index, Ai, ranged from 0.37-1.27 g, and averaged 0.74 ± 0.25g. Bond ball mill work index, BWi, ranged from 14.7–18.2 kWh/t and averaged 16.1 ± 1.0 kW/t. Bond rod mill work index, RWi, ranged from 12.7–18.1 kWh/t and averaged 15.3 ± 1.4 kW/t. Sag mill comminution tests (SMC) were completed.

Feasibility Study silver and gold recoveries were based on head grade versus tail grade regression of over 154 bottle roll tests from the Variability Average Grade, Lithology Elevation bench, and selected Exploratory bottle roll test programs. The weighted silver and gold recoveries, based on life of mine head grades, were determined to be 84% and 61%, respectively.

The average cyanide consumption was determined to be 1.0 kg/t. Lime consumption was determined to be 1.4 kg/t. The flocculant consumption was determined to be 0.085 kg/tonne. Sodium cyanide detoxification reagent consumptions were determined to be 0.8 kg sodium metabisulfite/t and 0.07 kg copper sulfate pentahydrate/t.

26 RECOMMENDATIONS

On the basis of the results of this Feasibility Study, it is recommended to consider completing basic engineering. The Project has an estimated net present value of $87.6 million at a 5% discount rate, with an estimated internal rate of return of 9.5%, and a projected Project payback period of 6.9 years after taxes at the metal prices used for this Report. Having basic engineering completed will allow the rapid start-up of construction, when the company decides to proceed with the Project.

Key up-front activities during the basic engineering period include:

Environmental

- Continue baseline environmental monitoring, including the monitoring of the piezometers and weather station. Initiate the air monitoring with contractors to have baseline information. Confirm geochemical characteristics of tailings and waste rock with kinetic testing to assess long-term geochemical behavior.
- Complete archeological survey.
- Reinitiate MIA applications for road construction and Project.

Land

- Continue with land acquisitions and ROWs for road and ancillary activities (power/water).
- Maintain mining claim rights through exploration programs.

Power

- Pursue 230 kVA connection point approval with CFE.

Metallurgy

It is recommended metallurgical developments continue to support optimization of process parameters:

- Optimization of leach parameters with lithology and variability composites; commercial tank pressure leach, temperature, and particle size.
- Evaluation of alternate technology: CELP, or equivalent, provided the development is stepwise, and without commercial obligations prior to proving the technology.
- Continue development of the geometallurgy model with additional sampling, bottle roll, and QEMSCAN analysis.
- An allowance for additional metallurgical test work is approximately $600,000.

Plant Design

- Initiate basic plant design.

Water Rights

- Continue process to obtain water rights or confirmation of water rights dependent upon operation/approval of the Project.

Tailing Design

- Collection of site specific meteorological and hydrology data. These data will be used to refine seasonal run off values and design storms to be used in future detailed design work.
- Optimization of the water balance to incorporate updated run off and process flow estimates, in order to maximize the available reclaim back to the process. The TSF water balance should be combined with the site wide water balance for the Project.
- Additional site investigations should be completed as part of detailed design. The investigations should focus on construction materials and foundation conditions, with particular focus on the potential for large voids to be present in the near surface bedrock within the basin and below the embankments.
- The tailings materials and properties should be reviewed during detailed design to be sure they are representative, especially if any changes to the process occur or as more representative tailings samples become available.
- Development of a full closure plan for the TSF based on the final design configuration.

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SRK Consulting, 2014m. Plan de muestreo y análisis – Proyectos La Preciosa: unpublished groundwater monitoring plan prepared for Proyectos Mineros La Preciosa S.A. de C.V., 25 July 2014, 15 p.

SRK Consulting, 2014n. Hydrogeologic and surface water site characterization report: unpublished report by A. Pérez Morán, M. Salguero Lazo de la Vega, J. Begay, and D. Russin, 8 August 2014, 31 p., 2 appendices.

APPENDIX A: FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE of QUALIFIED PERSON

I, Daniel H. Neff, P.E., do hereby certify that:

1. I am currently employed as President by:
   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Ste. 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of the University of Arizona and received a Bachelor of Science degree in Civil Engineering in 1973 and a Master of Science degree in Civil Engineering in 1981.

3. I am a:

   - Registered Professional Engineer in the State of Arizona (No. 11804 and 13848)

4. I have practiced civil and structural engineering and project management for 39 years. I have worked for engineering consulting companies for 14 years and for M3 Engineering & Technology Corporation for 27 years.

5. I have read the definition of "qualified person" set out in National instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for Section 1 of the technical report titled "La Preciosa Silver-Gold Project NI 43-101 Feasibility Study" (Technical Report), dated effective July 29, 2014, prepared for Coeur Mining, Inc.

7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was the preparation of a "Preliminary Economic Assessment- Technical Report" dated effective 30 June 2011.

8. I visited La Preciosa Site on June 8, 2010.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 29 day of July, 2014.

(Signed) (Sealed) "*Daniel H. Neff*"
Signature of Qualified Person

Daniel H. Neff
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Conrad E. Huss, P.E., Ph.D., do hereby certify that:

1. I am Senior Vice President and Chairman of the Board of:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Rd., Suite 101
   Tucson, Arizona 85704
   U.S.A.

2. I graduated with a Bachelor's of Science in Mathematics and a Bachelor's of Art in English from the University of Illinois in 1963. I graduated with a Master's of Science in Engineering Mechanics from the University of Arizona in 1968. In addition, I earned a Doctor of Philosophy in Engineering Mechanics from the University of Arizona in 1970.

3. I am a Professional Engineer in good standing in the State of Arizona in the areas of Civil (No. 9648) and Structural (No. 9733) engineering. I am also registered as a professional engineer in the States of California, Illinois, Maine, Minnesota, Missouri, Montana, New Mexico, Oklahoma, Texas, Utah, and Wyoming.

4. I have worked as an engineer for a total of forty-three years. My experience as an engineer includes over 36 years designing and managing mine development and expansion projects including material handling, reclamation, water treatment, base metal and precious metal process plants, industrial minerals, smelters, special structures, and audits.

5. I have read the definition of "Qualified Person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I am the principal author for the preparation of the technical report titled "La Preciosa Silver-Gold Project NI 43-101 Feasibility Study" (Technical Report), dated July 29, 2014, prepared for Coeur Mining, Inc.; and am responsible for Sections 2, 3, 4, 5, 18, 19, 21, 22, 24, 25, 26, and 27.

7. I have not visited the La Preciosa project site.

8. I have not had prior involvement with the property that is the subject of the Technical Report.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 29th day of July, 2014.

(Signed) (Sealed) "*Conrad E. Huss*"
Signature of Qualified Person

Conrad E. Huss, P.E., Ph.D.
Print Name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, W. David Tyler, do hereby certify that:

1. I am currently employed as a Vice President and Project Manager by:

   Coeur Mining Inc.
   104 S. Michigan Ave, Suite 900
   Chicago, Illinois 60603

2. This certificate applies to the technical report titled "La Preciosa Silver-Gold Project NI 43-101 Feasibility Study" (Technical Report), dated effective July 29, 2014, prepared for Coeur Mining, Inc. (Company).

3. I am a graduate of the Colorado School of Mines.

4. I am a Registered Member of the Society for Mining, Metallurgy and Exploration, Inc., member number 3288830.

5. I have practiced engineering and project management for 33 years for open pit and underground mines. I have worked on study projects, constructed mines, and have been the Chief Engineer and Mining Manager at mines that are similar to the current project in size and/or in the type of mineralization and process plant designs.

6. I have read the definition of "qualified person" set out in National instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

7. I am responsible for the Sections 15 and 16, and corresponding subsections of Sections 1, 25 and 26 of the Technical Report.

8. I am an employee of the issuer and not independent of the issuer.

9. I have had prior involvement with the property that is the subject of the Technical Report. During the period from May 2013 to present, I was employed as the Vice President of Technical Services for Coeur Mining, responsible for the technical evaluation of this project, including the Preliminary Economic Assessment.

10. I have visited La Preciosa Site on several occasions during this time. My last site visit was on February 13, 2014.

11. I have read National Instrument 43-101 and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument.

12. As of the date of this certificate, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the Technical Report not misleading.

13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 29 day of July, 2014.

(Signed) (Sealed) "*W. David Tyler*"
Signature of Qualified Person

W. David Tyler
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Robert Bruce Kennedy, do hereby certify that:

1. I am currently employed as a Project Manager by:

   Coeur Mining Inc.
   104 S. Michigan Ave, Suite 900
   Chicago, Illinois 60603

   Assigned to work at:

   Proyectos Mineros La Preciosa, S.A. de C.V.
   Durango, Mexico

2. I am a graduate of the New Mexico Institute of Mining and Technology.

3. I am a:

   - Registered Professional Engineer in the State of New Mexico, License number 11023, and a Professional Member of the Society for Mining, Metallurgy and Exploration, Inc. member number 4022866

4. I have practiced engineering and project management for over 40 years, including consulting engineering, Mine and Processing Plant Management, for open pit and underground mines. I have been General Manager at mines that are similar to the current project in size and/or in the type of mineralization and process plant designs.

5. I have read the definition of "qualified person" set out in National instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the Sections 6 and 23, and corresponding subsections in Sections 1, 25, and 26 of the technical report titled "La Preciosa Silver-Gold Project NI 43-101 Feasibility Study" (Technical Report), dated effective July 29, 2014, prepared for Coeur Mining, Inc.

7. I have had prior involvement with the property that is the subject of the Technical Report. During the period from June 2013 to present, I was employed as the Project Manager/General Manager of this project.

8. I have visited La Preciosa Site frequently during that time period. My last site visit was on July 22, 2014.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am an employee of the issuer and not independent of the issuer.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 29 day of July, 2014.

(Signed) (Sealed) "*Robert Bruce Kennedy*"
Signature of Qualified Person

Robert Bruce Kennedy, P.E.
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Erin L. Patterson, P.E. do hereby certify that:

1. I am currently employed as an Engineer by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, Suite 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of the University of Arizona and received a Bachelor of Science in Chemical Engineering in 2005.

3. I am a:

   - Registered Professional Engineer in the State of Arizona (No. 54243)

4. I have practiced engineering at M3 Engineering & Technology Corporation for six years.

5. I have read the definition of "qualified person" set out in National instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the Section 17 and corresponding subsection in Section 1 of the technical report titled "La Preciosa Silver-Gold Project NI 43-101 Feasibility Study" (Technical Report), dated July 29, 2014, prepared for Coeur Mining, Inc.

7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was the preparation of a "Preliminary Economic Assessment-Technical Report" dated effective 30 June 2011 and a "Preliminary Economic Assessment" dated effective 26 July 2013.

8. I have not visited the La Preciosa Site.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 29 day of July, 2014.

(Signed) (Sealed) *"Erin L. Patterson"*
Signature of Qualified Person

Erin L. Patterson
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Tracy Edward Barnes, do hereby certify that:

1. I am currently employed as a President and Principal Mining Engineer by:

   Barnes Engineering Services, Inc.
   12945 W 15th Drive
   Golden, CO 80401-3572

2. I am a graduate of the University of Washington with a B.S. Mining Engineering (1975) and the Colorado School of Mines with a M.S. Mining Engineering (2009).

3. I am a:
   - Professional Engineer in the State of Colorado (Registration No. 33381)
   - SME Founding Registered Member (No. 159490 RM)

4. I have practiced engineering continuously since 1975, the last 18 years in my current position.

5. I have read the definition of "qualified person" set out in National instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the Section 14 and corresponding subsection in sections 1, 25 and 26 of the technical report titled "La Preciosa Silver-Gold Project NI 43-101 Feasibility Study" (Technical Report), dated effective July 29, 2014, prepared for Coeur Mining, Inc.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have not visited La Preciosa Site.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 29th day of July, 2014.

(Signed) (Sealed) "*Tracy Edward Barnes*"
Signature of Qualified Person

Tracy Edward Barnes, P.E.
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Christopher L. Easton, do hereby certify that:

1. I am currently employed as President by:

   Easton Process Consulting, Inc.
   9532 S. Desert Willow Way
   Highlands Ranch, Colorado 80129
   U.S.A.

2. I am a graduate of University of Wyoming, and received a Bachelor of Science degree in Chemical Engineering in 1987.

3. I am a:

   - Registered Qualified Person of the Mining and Metallurgical Society of America (MMSA).
   - A lifetime Member in good standing of the American Institute of Chemical Engineering (AICHE).
   - Member in good standing of the Society for Mining, Metallurgy, and Exploration, Inc. (SME).

4. I have practiced metallurgical and mineral processing engineering for 25 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Sections 13 and also responsible in part for Sections 1, 24, 25, and 26 of the technical report titled "La Preciosa Silver-Gold Project, NI 43-101 Feasibility Study" (Technical Report), dated effective July 29, 2014, prepared for Coeur Mining, Inc.

7. I have not previously had prior involvement with the property that is the subject of the Technical Report.

8. I visited the La Preciosa Project Site in November 4-8 and December 2-6, 2013.

9. As of the date of this certificate, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 29th day of July, 2014.

(Signed) "*Christopher L. Easton*"
Signature of Qualified Person

Christopher L. Easton
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Dana C. Willis, do hereby certify that:

1. I am currently employed as a geologist by:

   Coeur Mining, Inc.
   104 South Michigan Ave., Suite 900
   Chicago, IL 60603

2. I am a graduate of Sonoma State University (BS Geology 1983) and Oregon State University (MS Economic Geology 1992).

3. I am a: Registered Member of the Society for Mining, Metallurgy and Exploration, Inc. (SME), member number 3510270.

4. I have practiced geology since 1982 and have been previously involved with geology (geothermal and mineral exploration; mine and development geology; geology and resource models; and geostatistics) for a variety of precious and base metal deposits in North and South America, Africa, Australia, New Zealand, and Indonesia. I have been directly involved with gold and silver resource models and resource estimation since 2003.

5. I have read the definition of "qualified person" set out in National instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the Sections 7, 8, 9, 10, 11, and 12 and those portions of the Summary, Interpretations and Conclusions and Recommendations that pertain to those Sections of the Technical Report titled "La Preciosa Silver-Gold Project NI 43-101 Feasibility Study" (Technical Report), dated effective July 29, 2014, prepared for Coeur Mining, Inc.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have visited La Preciosa Site on the following dates: 5-7 November 2013, 3-5 December 2013, and 22-23 January 2014.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am an employee of the issuer and not independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 29 day of July, 2014.

(Signed) (Sealed) "*Dana C. Willis*"
Signature of Qualified Person

Dana C. Willis
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, COROLLA HOAG, do hereby certify that:

1. I am currently employed as a Practice Leader and Principal Consultant (Geology) by:

   SRK Consulting (U.S.), Inc.
   3275 W. Ina Road, Suite 240
   Tucson, Arizona 85741, USA

2. I graduated with a Bachelor of Science degree in Geology from Western Washington University, Bellingham, Washington in 1983. I obtained a Master of Science degree in Economic Geology from The University of Arizona, Tucson in 1991.

3. I have been a Certified Professional Geologist through the American Institute of Professional Geologists (CPG – 11205) since August 2008. I am a Founding Registered Member (1455400RM) with the Society of Mining, Metallurgy, & Exploration and have been since July 2006.

4. I have worked as a Geologist for 26 years in the fields of copper and gold exploration; mine feasibility studies for open pit, in-situ leach, and underground operations; and environmental permitting for mine development, operations, and closure.

5. I have read the definition of "qualified person" set out in National instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for Section 20 and the corresponding subsection in Sections 1, 25, and 26 of the technical report titled "La Preciosa Silver-Gold Project NI 43-101 Feasibility Study" (Technical Report), dated effective July 29, 2014, prepared for Coeur Mining, Inc.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I visited La Preciosa Site for four days on October 29, 2013 through November 1, 2013.

9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.

11. I have read National Instrument 43-101 and Form 43-101F1, and those portions of the Technical Report for which I am responsible have been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 29 day of July, 2014.

(Signed) (Sealed) "*Corolla Hoag*"
Signature of Qualified Person

Corolla Hoag, C.P.G., SME-RM
Print name of Qualified Person