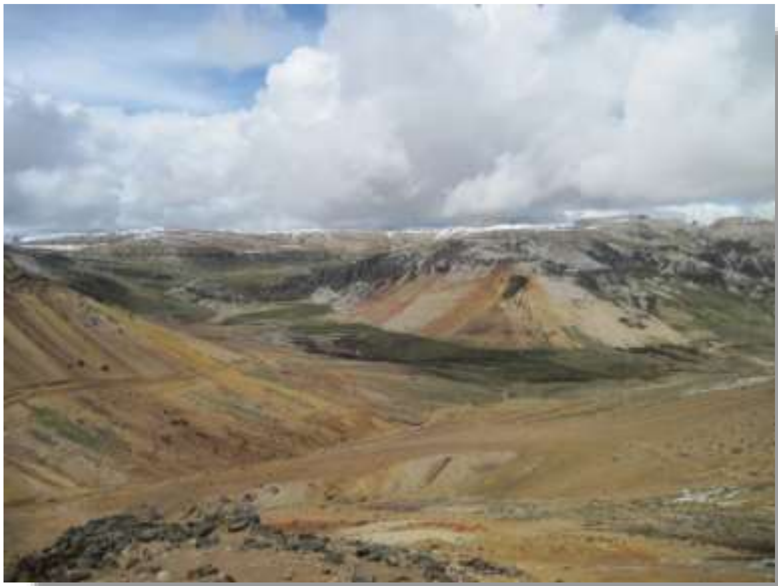
M3-PN140135
Effective Date:
May 30th, 2015

Optimized and Final Feasibility Study



Corani Project

Puno, Peru

Form 43-101F1 Technical Report

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DATE AND SIGNATURES PAGE

The effective date of this Report is May 30, 2015. The issue date of this Report is July 17, 2015. See Appendix A, Feasibility Study Contributors and Professional Qualifications, for certificates of Qualified Persons.

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CORANI PROJECT
FORM 43-101F1 TECHNICAL REPORT

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1 SUMMARY

This Feasibility Study was prepared for Bear Creek Mining Corporation by M3 Engineering & Technology Corp. (M3) in cooperation with Global Resource Engineering Ltd (GRE), a consulting company based in Denver, Colorado; Tom Shouldice independent metallurgist consultant; and Christian Rios, independent geological consultant. The technical report presented here is the National Instrument 43-101 project report that summarizes the feasibility study update. This Report is based on the outcomes of an engineering study completed to Feasibility Study (FS) standards.

Several project components were optimized subsequent to the previous Technical Report (M3, 2011). Detailed engineering studies, site investigation work, and laboratory testing programs were ongoing at the time of the 2011 report. The optimizations envisaged in 2011 have been advanced through additional fieldwork and detailed engineering to support the optimization concepts presented in this study. A brief summary of the work performed subsequent to the 2011 report for the present 2015 study is presented below.

- Additional metallurgical and geotechnical drilling within the Corani Project area
- Re-logging and re-interpolation of the mineralogy
- An updated mineralogical database and block model
- Geometallurgical model for predicting recovery within the block model
- Updated capital and operating cost estimation
- Mine plan optimization
- Geotechnical site investigations and waste characterization test work and studies
- Detailed process review
- Project-wide water balance studies
- Engineering and design to advance the project to feasibility study level
- Updated quantity and cost estimation

1.1 PROPERTY DESCRIPTION AND OWNERSHIP

1.1.1 Location

The Project site is located in the Andes Mountains of south-eastern Peru at elevations of 4800 to 5100 meters above sea level (masl), specifically within the Cordillera Vilcanota of the Eastern Cordillera. The site is located in the Department of Puno immediately east of the continental divide separating the Pacific and Atlantic drainages.

The site location is approximately 160 kilometers (km) in a direct line to the southeast of the major city of Cusco, with Universal Transverse Mercator (UTM) coordinate ranges of 312,000E to 322,000E and 8,443,000N to 8,451,000N, using UTM, Zone 19S, Provisional South American datum, PSAD 56.

Access to the mining operations will be via a new 46 km road to be built over generally flat and gently sloping topography. The new mine access road will connect at the town of Macusani to the Interoceanic Highway; a two-lane, paved highway connecting to the Peruvian highway system and to the Port of Matarani.

1.1.2 Description

The Project has favorable infrastructure. The mine is 30 km from a new high-voltage power line with abundant capacity to meet the Project needs. The project has technically and environmentally favorable sites for waste rock and filtered tailing co-disposal storage. Additionally the mine plan is amenable to sequenced backfilling of the pit, reducing operating costs and eliminating environmental pit lake liability at closure.

1.1.3 Mineral Tenure

The land status of the Project is a series of twelve mineral claims or concessions. Mineral concessions in Peru are filed with the Instituto Nacional de Concesiones y Catastro Minera (INACC) which is part of the Ministerio de Energía y Minas (MINEM) in Peru. Claims can vary in size from 100 to 1,000 ha. Concessions are defined by limits parallel to the UTM grid system employed in the district to form rectangular areas.

Claim monuments need not be maintained in the field as the primary documentation exists as the filed boundaries at INACC. The 12 claims at Corani Project are located in the districts of Corani, Macusani and Nuñoa, provinces of Carabaya and Melgar, department of Puno, in Peru, and cover an aggregate extent of 5,180 hectares. The general location map and list of claims are provided on Figure 4-2 and Table 4-1 respectively.

1.2 GEOLOGY AND MINERALIZATION

1.2.1 Regional Geology

The regional geology in the Project area is characterized by volcanic flows overlying a thick sequence of sedimentary basement rocks. All units have been affected by Pleistocene glaciations forming U-shaped valleys and arêtes, typical of alpine glacial terrain.

1.2.2 Property Geology

The basement units in the Corani area are a series of upper Paleozoic (320 Ma) sandstone and shale units of the Grupo Ambo Formation that have been weakly metamorphosed into quartzite and phyllitic shales. The resistant quartzite units are often ridge formers. The weathered shales generally form subdued, generally slope-forming outcrops. Within the Project resource area, the sediments are generally red to gray shales.

A sequence of Tertiary (23.1Ma +/- 0.2Ma), pre-mineral volcanic tuffs unconformably overly the sediments. These tuffs are generally crystal and crystal-lithic with quartz-eyes ranging up to 5 mm in diameter. These rocks range from well bedded to massive. In the upper parts of the pre-mineral sequence are andesite volcanic flows, which are generally more bedded than the underlying tuffs. The variations in the stratigraphic makeup of the pre-mineral tuffs and andesite flows do not appear to have any controlling effect on the mineralization. All of the resource within the Corani district is hosted in the pre-mineral tuffs and andesite flows.

Unconformably overlying the pre-mineral units, the Tertiary post-mineral tuff (10.2Ma +/- 0.1Ma), consisting of crystal tuffs with similar characteristics to the lower pre-mineral tuffs, is effectively barren. The post-mineral sequence forms prominent spires and thickens to the north (from 0 meters to over 200 meters within the project area).

Alteration in the project area consists of a broad, 5 x 2 kilometer zone of illite-kaolinite alteration of the pre-mineral tuffs. More specific to the mineralization are illite, kaolinite, smectite/chlorite/celadonite and gangue minerals including quartz (massive to banded), barite, chalcedony and iron and manganese oxides. Each of the three mineralized areas, namely Corani Este, Minas Corani and Main Corani, exhibit differences in alteration and gangue, including:

- Corani Este: strong barite, minor quartz and chalcedony, moderate smectite/chlorite/celadonite, brecciation, strong iron oxides and no manganese oxides;

- Minas Corani: strong smectite/chlorite/celadonite, moderate chalcedony and barite with strong iron oxides and moderate manganese oxides; and

- Main Corani: banded quartz, strong barite, iron oxides and minor manganese oxides.

Structurally, the Project area is marked by a stacked sequence of listric normal faults striking dominantly north to north-northwest with moderate to shallow (50 to <10 degrees) westerly dips. The hanging walls of the listric faults are extensively fractured and brecciated, forming sites for metal deposition. The stacked sequences are more prominent in Minas and Main Corani with Este showing a single listric fault with a more extensively fractured and brecciated hanging wall.

Mineralization in the Project area is comprised of freibergite (silver-bearing tetrahedrite), galena (not argentiferous), sphalerite (white to dark-colored), pyrite, marcasite, other silver sulfosalts (myrargyrite, pyrargyrite-proustite (ruby silver)), boulangerite, acanthite and minor native silver.

The ore body can be split into a three principal metallurgical types: first, a mixed sulfide group that is composed of relatively coarse to very fine sulfide mineralization; second, a transitional mineralization where the sulfide minerals have been partly oxidized, with some of the lead having been remobilized into a lead-phosphate mineral and much of the zinc removed from the ore; and third, an oxide zone. This metallurgical zonation mimics the south west dipping nature of the listric faults and as such forms a tilted layer cake with the oxides occurring on the far west of the project, the transitional ores in the middle west and the sulfides in the center of the deposit and to the east.

Southeast of the principal areas of mineralization there are smaller areas of mineralization that are referred to as the Gold Zone and Antimony Zones. Neither of these has been included in any published resource and are not included in this Report.

1.3 EXPLORATION STATUS

Prior to the early 1950s, mineral exploration in the district consisted of shallow prospect pits and adits in the northern portion of the current Project. These prospects are of unknown age and may date back to colonial Spanish time. Antimony prospects south and east of the property reportedly were active in the early 1900's with limited antimony production (C.R. Petersen, 1967).

The first modern evaluation of silver-lead mineralization began with the location of mineral concessions in 1951 by Augusto Leon y Leon. Compañía Minera Korani was formed in 1956 to develop the silver-lead mineralization previously prospected. The mines were developed and operated from 1956 up to at least 1967; initially mining 80 tpd of ore. In 1965, Compañía Minera Korani sought to increase production from 80 tpd to 300 tpd. In 1967, Compañía Minera Korani was owned two-thirds by Compañía Minera Palca and one-third by M. Hochschild. In early 1967, estimated mine production was reported at about 3,400 short tons per month, with grades of 7.0-9.0% lead, 2.3% zinc, and 8.0 to 11.0 oz/ton silver (C.R. Petersen, 1967). Total historical production is uncertain, but is estimated at 100,000 t of silver-lead-zinc ore.

Historical maps of the underground workings show development on four levels (4820, 4843, 4860 and 4870 m levels for 50 meters vertically) that extend over an area of approximately 500 meters in a general north-south direction (parallel to strike) by about 150 meters in an east-west direction. It is not known when operations of Compañía Minera Korani ceased, but it is presumed to be in the late 1960s or early 1970s. When the mining stopped at the Project the previous operations were abandoned and several environmental liabilities still exist at the areas related to mining and processing.

Subsequent exploration activity was performed by Minsur a private Peruvian company, whose exploration program was reported to include 40 shallow drill holes in various locations, including a number of close proximity holes in the gold zone (located south of the current resource area). None of Minsur's exploration information is available or verifiable; although reportedly gold mineralization was encountered in much of Minsur's drilling.

In late 2003 and early 2004, Rio Tinto Mining and Exploration began a surface exploration program for porphyry copper mineralization. During 2004, Rio Tinto conducted surface mapping, sampling, and ground magnetic surveys, and developed access roads into the area. The initial work by Rio Tinto defined anomalous silver and lead mineralization to the south of the Korani mines, and also defined a zone of anomalous gold mineralization in rock and soils.

The concession ownership by Compañía Minera Korani apparently lapsed during the 1970s. The ownership of Minsur also lapsed prior to Rio Tinto's exploration in the project area. Rio Tinto re-established some of the older concessions in their name beginning in 2003. BCM has added two concessions early in 2005 to create the current land position.

In early 2005 when BCM entered into an option agreement with Rio Tinto. BCM completed the option agreement in January 2008 with the final payment of the $5.4 million required under the agreement for a 70% share of the Project. In April of 2008 BCM entered into a purchase agreement for the remaining 30% that was controlled by Rio Tinto, and BCM completed the purchase of Rio Tinto's 30% interest in February 2011.

Since acquiring the project BCM has completed approximately 93 km of drilling, prepared detailed geological maps, performed extensive metallurgical testing from all areas of the deposit, continually operated an exploration camp, engaged in development projects with the two nearby communities and completed several environmental studies. The results of this work have been the completion of five NI43-101 reports (2006-2011) where the resource has grown and culminates with this Report.

1.4 DEVELOPMENT AND OPERATIONS

1.4.1 Production Schedule

The mine requires pre-production waste stripping of 22.8 million tonnes, all within Year -1 of the mine production schedule. After that, a mining sequence that directly feeds the mill with 7.875 million tonnes of ore per year was developed.

The mining sequence calls for the waste stripping to average 2.11:1 (waste to ore) for the first five years and then the stripping ratio will reduce to 1.31 for the following 13 years. The LOM stripping ratio averages of 1.68:1.

1.4.2 Mine Equipment

Mining will be performed using conventional open pit methods using 181 t trucks and 18 m^3 hydraulic shovels mining on 8 meter-high benches.

1.5 METALLURGY

The Corani deposit is a silver-lead-zinc deposit with relatively complex mineralogy. Upon review of the metallurgical testing data, it is clear that performance of Corani mineralization to conventional flotation and cyanidation processing was widely variable. The geological classifications provided some delineation of metallurgical response: Samples representing the CSC ore type consistently responded well to a conventional sequential lead / silver – zinc flotation. Conversely, FeOx and MnO responded very poorly to flotation, but generally responded better to cyanide leaching for silver.

The geological classification FBS which represents a large amount of the estimated resource tonnage, had a broad range of metallurgical response. The variable response was shown to be related to the fine texture of the mineralization and presence of non-sulphide lead mineral forms. However, the geological classifications alone were not able to delineate the texture or quantity of non-sulphide lead minerals.

To better predict the metallurgical response, a geo-metallurgy approach was investigated to link metallurgical response to block modelling parameters. The statistical analysis indicated several key parameters could be used to generate metallurgical response. For the purpose of the analysis, the metallurgical process was restricted to only sequential flotation of silver bearing lead concentrate followed by the flotation of a zinc concentrate, also containing some silver.

With metallurgical response linked to block modelling parameters, the mine plan could be optimized to maximize the revenue for the Project. The table below displays the estimated metal recoveries by mine schedule.

Table 1-1: Recovery Predictions for Mine Schedule

Production Year	Tonnes (000)	Feed Grade - % or g/t			Recto Pb Con - %		Rec to Zn Con - %	
		Ag	Pb	Zn	Ag	Pb	Ag	Zn
Year 1	5,675	96	1.17	0.48	67	69	2	54
Year 2	7,744	84	1.43	0.88	70	69	5	63
Year 3	7,897	73	1.20	0.85	70	71	6	69
Year 4 to 5	15,745	80	1.12	0.97	71	74	7	76
Year 6 to 10	39,393	55	0.98	0.38	62	45	2	49
Year 11 to 18	69,120	27	0.61	0.50	69	72	7	56

Note: Ag grades are denoted in g/tonne, all other assays are in percent.

1.6 PROCESS

The Project processing facility is designed to treat 22,500 tpd of silver-lead-zinc ore at an operational availability of 92 percent. The processing flow sheet for the Project is a standard flow sheet that is commonly used in the mining industry, including conventional flotation recovery methods typical for lead-zinc ores. Figure 1-1 below is a simplified schematic of the process. M3 completed the process design based on the results of several 2009 and 2011 metallurgical testing programs (Blue Coast, 2011; DJB Consultants, 2011; SGS, 2007, 2008a, 2008b, 2009a, 2009b, and 2010) and new metallurgical testwork and analysis for grinding, sedimentation, and filtration in 2014 (Alex G. Doll Consulting Ltd., 2014; ALS Metallurgy Kamloops, 2014; Outotec Canada, 2014a; and Outotec Canada, 2014b).

The ore will be crushed in a primary jaw crusher that is located adjacent to the open pit mine. From there it will be conveyed to the processing facilities where it will be ground to 80 percent finer than 106 microns in a semi-autogenous grinding (SAG) and ball milling circuit.

The ore is further processed in a flotation circuit consisting of lead flotation followed by zinc flotation. The majority of the silver will be recovered in the lead flotation circuit and some silver will also be collected in the zinc flotation circuit. Lead sulfide will be recovered in a one-pass rougher flotation bank, producing a concentrate that will be upgraded to smelter specifications in three stages of cleaning. Tails from the lead flotation section will then be conditioned for zinc sulfide flotation. The process scheme for zinc flotation also includes a rougher bank and three stages of cleaning using both mechanical cells and column flotation to produce smelter-grade zinc concentrates. For both lead and zinc sections, the rougher flotation concentrates will be reground to 80 percent finer than 25 microns prior to cleaner flotation to liberate the sulfides for further upgrading.

Tailing from the lead and zinc flotation circuit will be thickened, filtered and conveyed to a stockpile at the plant. From there, the filtered tailing will be trucked to the Main Waste Dump where it will be co-disposed with mine waste during the first six years of operation. After Year 6, filtered tailing will be disposed of as backfill into the Corani Este pit with additional waste rock.

Water will be reclaimed from the tailing thickener overflow and from the tailing filtrate. Process make-up water will be pumped from the contact water section and fresh water section of the Plant Water Supply Pond.

Lead and zinc concentrates will be thickened, filtered, and bagged into supersacks for shipment. They will then be loaded into flatbed trucks and enclosed vans to be trucked to the Port of Matarani for ocean shipment to smelters.

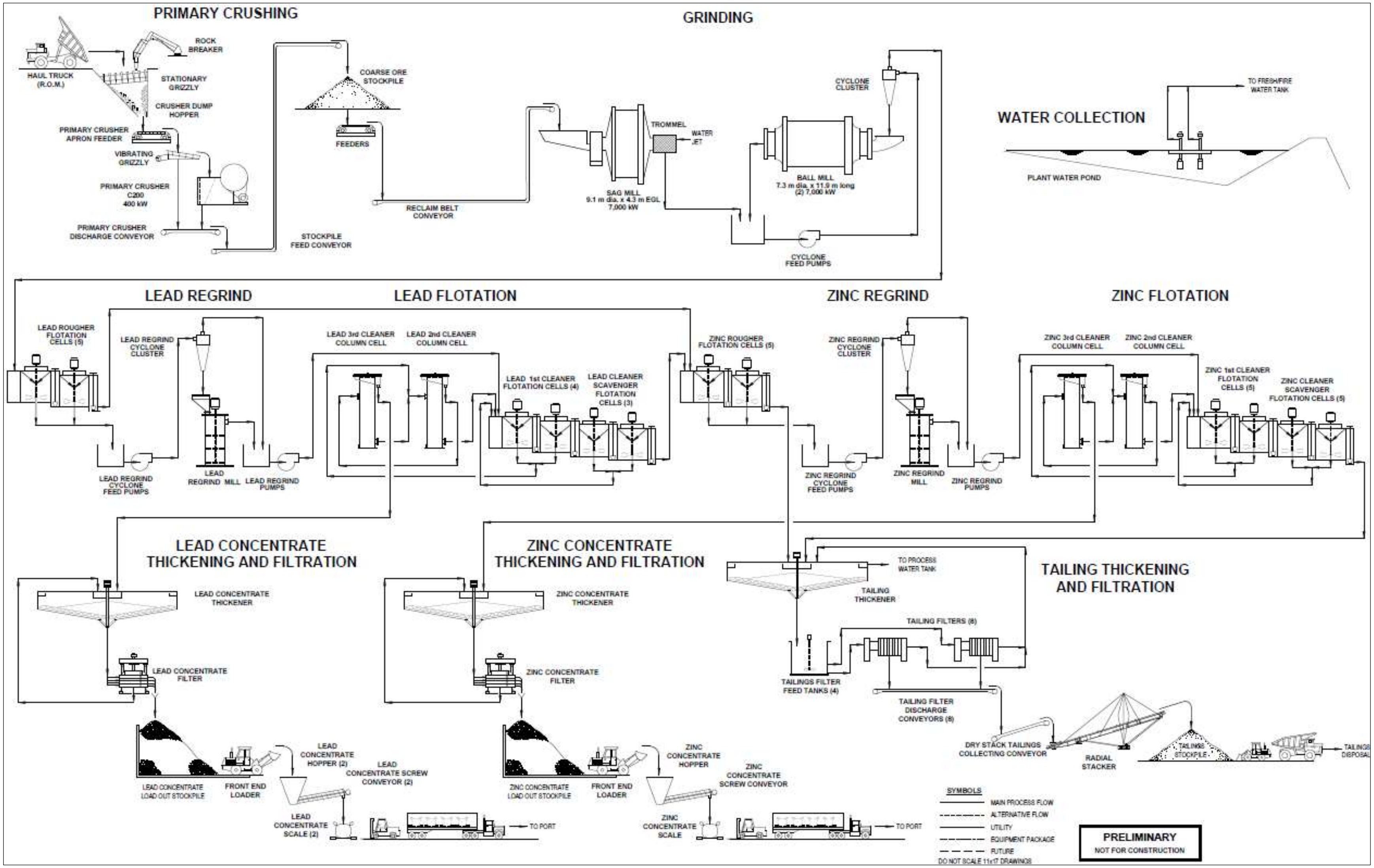


Figure 1-1: Simplified Process Flow Diagram for the Corani Project

1.7 ENVIRONMENTAL AND PERMITTING

Due to the mineralogy of the area, the main environmental considerations associated with the Project are primarily related to the management of surface and groundwater that has contacted exposed mineralized material. During operations, impacted water will be recycled to the plant or temporarily stored in the Plant Water Pond, preventing release of any impacted water. During closure, surface and groundwater will be treated as necessary before being released to the downstream. In addition, due to historic mining activities, a number of environmental liabilities are present on the Project site. These can be resolved, to the extent practical, as the site is developed.

1.8 RECLAMATION AND CLOSURE

The Project received approval for an Environmental and Social Impact Assessment (ESIA) in 2013 based on the mine plan presented in the 2011 Technical Report (M3, 2011). The new project configuration developed for the current study will be incorporated into a modification of the ESIA to be completed subsequent to the publication of this report. Project optimization performed for the current study resulted in a reduced project footprint, a reduction in water consumption, and other changes which are anticipated to reduce environmental impacts associated with project development relative to the previous study. In a number of cases, the development of the Project is anticipated to improve existing environmental conditions.

1.9 PROJECT EXECUTION

1.9.1 Overview

The purpose of the Execution Plan is to provide a comprehensive plan for the development and implementation of the Project. The Execution Plan provides a tactical plan for engineering, procurement, construction, commissioning and start-up of the plant facilities and infrastructure.

A preliminary execution plan is provided in Section 24.1 which addresses the overall Project including objectives, scope and strategies.

1.9.2 Project Schedule

A conceptual level EPC schedule was developed to identify critical project milestones. The following engineering, test work and permitting durations were developed based on consultant input, client input and historical project data. Construction durations were based on quantities and man-hours developed in the capital cost estimate:

- Basic Engineering – 6 months
- Detailed Engineering – 15 months
- Permitting – 16 months
- Major Offsite Contracts (Camp, Power Line, Access Road) – 13 months
- Mine Construction/Prestripping – 12 months
- Plant Construction – 17 months
- Commissioning and Start-Up – 4 months

The relationships between the tasks are shown in the simplified schedule below (Figure 1-2). The total time from receiving financing to start-up is estimated to be approximately 31 months.

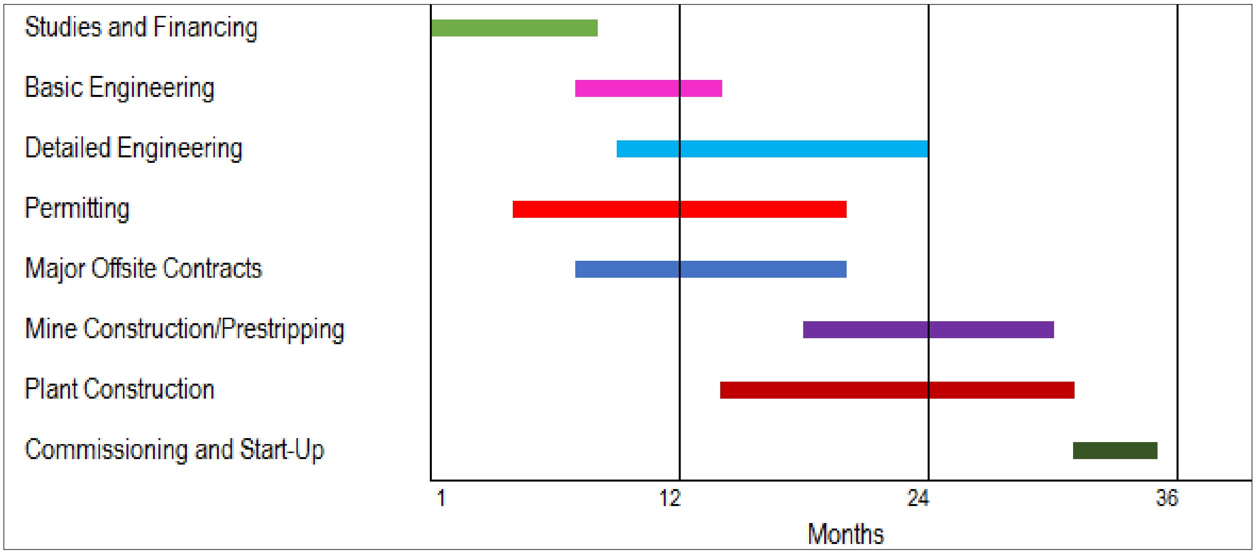


Figure 1-2: Simplified Schedule

The summary Project Schedule shown in Figure 24-2 indicates the critical activities and milestones.

1.9.3 Objectives

The project would be executed in accordance with the Execution Plan which is designed to achieve the following objectives:

- Conformance to the budget
- On-schedule completion
- Compliance with project quality standards
- Uncompromised safety
- Inclusion of Peruvian participation
- Environmental compliance

1.9.4 Project Management

An internationally experienced EPCM team would be assembled to manage the development of the project. This team would develop and implement the Project Procedures Manual that would include the following information:

- Project Management Plan;
- Engineering Management Plan;
- Procurement Plan;
- Logistics and Transportation Plan;
- Construction Plan;
- Commissioning and Startup Plan;
- Quality Assurance Plan;
- Environmental, Health and Safety Plan;
- Communication Plan;
- Project Controls Plan;

- Project Schedule; and
- Project Close-Out Plan.

1.9.5 Engineering

Some design packages, such as roads and power supply could be executed in Peru. The Project engineering would be developed in two-phases:

- A Basic Engineering phase that would confirm and expand on the feasibility designs and initiate the procurement of long-lead equipment items,

- A Detailed Engineering phase that would be carried out by a leading international engineering company following the completion of the Basic Engineering phase. As detail engineering designs and quantity take-offs are completed these would be transferred to the procurement and contracts groups for purchase and contracting and to the construction team at the project site.

1.9.6 Procurement and Contracting

Due to the location and altitude of the site, pre-fabrication and skid-mounted packages would be considered to the greatest extent possible. Pre-fabricated modules would be equipped with piping and valves, wiring and instrumentation to reduce onsite labor.

Sourcing of the majority of equipment and materials is expected to be from USA, Canada, Europe, Chile and China. Some major and minor mechanical equipment and material would be procured from Peruvian suppliers.

Working with the project construction management team a detailed contracting plan indicating scope breakdown and contract type will be developed during the project detail engineering phase.

1.9.7 Construction

The construction management team would manage the site activities of all onsite general contractors and specialty construction contractors.

Specific timing for all engineering work packages and construction ERFP packages would be included in the project master schedule.

1.9.8 Commissioning and Startup

The commissioning and start up team is planned to be an integrated organization of plant start-up professionals.

Commissioning includes those activities necessary for an effective transition between construction and mechanical completion when systems are turned over to the commissioning and start-up team. These activities include the following:

- Ensure that equipment is operationally ready for start-up (i.e. to accept feed);

- Sequence starting and running of tested logical groups of equipment;

- Wet and dry runs of systems;

- Demonstration of the suitability of the facilities to be ready for processing and production; and
- Coordinate with and assist the owner to achieve hand over of the completed facilities.

1.10 OPERATING COST ESTIMATE

Mining costs were prepared on a year by year basis with costs varying mostly due to changing haulage distances. The life-of-mine average mining costs will be $5.19 per tonne of total waste and ore mined. This cost also includes haulage of filtered tailings to the Main Waste Dump for co-disposal. A detailed breakdown is shown in Section 21.

The process costs are estimated to be $8.76 per tonne of processed ore, which includes the addition of the tailings filtration plant. Process costs include labor, maintenance, spare parts, and services. G&A costs are estimated to be $1.55 per tonne of processed ore. See Table 1-2.

Table 1-2: Life of Mine Operating Cost

Operating Cost	$/ore tonne
Mine	$5.19
Process Plant	$8.76
General Administration	$1.55
Smelting/Refining Treatment & Concentrate Transport	$6.40
Total Operating Cost	**$21.90**

1.10.1 Mine Operating Cost

Operating cost is based on labor and equipment usage for each mining area and time period. Hourly labor and annual salaried personel wages were provided by Bear Creek Mining Company and M3. Peruvian quotes were obtained for consumables like fuel, explosives, and tires. Replacement parts and maintenance materials are from Infomine. Detailed estimates of equipment productivity were made to obtain the annual operating costs. The life of mine operating costs are shown in Table 1-3 below.

Table 1-3: Mine Operating Cost

	Total	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5
OPEX - Total	**$ 714,528,400**	**$ -**	**$ 67,975,489**	**$ 57,514,661**	**$ 50,852,129**	**$ 47,580,246**	**$ 33,434,668**
OPEX - Direct Op	$ 580,997,182	$ -	$ 48,287,730	$ 38,105,553	$ 31,730,084	$ 28,535,784	$ 31,382,631
OPEX - Overhaul	$ 23,438,493	$ -	$ 1,955,918	$ 1,565,697	$ 1,278,634	$ 1,127,835	$ 1,269,183
OPEX - Equip. Leasing	$ 110,092,726	$ -	$ 17,731,841	$ 17,843,411	$ 17,843,411	$ 17,916,627	$ 782,855
Tonnes (Ore + Waste)	$ 369,219,550	22,822,468	35,068,685	27,254,649	19,192,936	17,464,821	16,133,470
Op Cost per Tonne	$ 1.94	$ -	$ 1.94	$ 2.11	$ 2.65	$ 2.72	$ 2.07
CAPEX - Total	**$ 75,742,090**	**$ 56,928,551**	**$ -**	**$ -**	**$ -**	**$ 13,049,083**	**$ -**
CAPEX - Initial Lease	$ 17,731,841	$ 17,731,841	$ -	$ -	$ -	$ -	$ -
CAPEX - Prestripping	$ 35,521,863	$ 35,521,863	$ -	$ -	$ -	$ -	$ -
CAPEX - Initial (Other)	$ 3,674,847	$ 3,674,847	$ -	$ -	$ -	$ -	$ -
CAPEX - Sustaining (Balloon payments on leased equipment)	$ 18,813,539	$ -	$ -	$ -	$ -	$ 13,049,083	$ -

	YR 6	YR 7	YR 8	YR 9	YR 10	YR 11	YR 12
OPEX - Total	**$ 44,785,931**	**$ 43,868,807**	**$ 40,670,474**	**$ 34,217,611**	**$ 41,613,879**	**$ 38,036,272**	**$ 29,954,480**
OPEX - Direct Op	$ 40,969,496	$ 36,904,181	$ 33,747,389	$ 27,175,150	$ 34,267,348	$ 31,461,954	$ 26,957,370
OPEX - Overhaul	$ 1,701,153	$ 1,525,637	$ 1,372,526	$ 1,067,627	$ 1,404,210	$ 1,271,431	$ 1,059,388
OPEX - Equip. Leasing	$ 2,115,282	$ 5,438,988	$ 5,550,559	$ 5,974,835	$ 5,942,320	$ 5,302,888	$ 1,937,722
Tonnes (Ore + Waste)	24,792,058	23,307,190	21,343,423	14,831,190	21,697,278	19,503,455	16,601,176
Op Cost per Tonne	$ 1.81	$ 1.88	$ 1.91	$ 2.31	$ 1.92	$ 1.95	$ 1.80
CAPEX - Total	**$ -**	**$ 82,106**	**$ -**	**$ 53,880**	**$ 440,126**	**$ 980,550**	**$ 2,528,063**
CAPEX - Initial Lease	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Prestripping	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Initial (Other)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Sustaining (Balloon payments on leased equipment)	$ -	$ 82,106	$ -	$ 53,880	$ 440,126	$ 980,550	$ 2,528,063

	YR 13	YR 14	YR 15	YR 16	YR 17	YR 18	YR 19
OPEX - Total	**$ 31,577,934**	**$ 29,187,854**	**$ 38,934,841**	**$ 29,989,209**	**$ 28,172,586**	**$ 26,161,328**	**$ -**
OPEX - Direct Op	$ 28,399,213	$ 26,473,242	$ 36,315,041	$ 28,446,309	$ 26,770,161	$ 25,068,547	$ -
OPEX - Overhaul	$ 1,119,349	$ 1,041,161	$ 1,511,902	$ 1,127,997	$ 1,057,634	$ 981,211	$ -
OPEX - Equip. Leasing	$ 2,059,372	$ 1,673,452	$ 1,107,898	$ 414,902	$ 344,791	$ 111,571	$ -
Tonnes (Ore + Waste)	17,766,344	14,477,051	19,099,531	14,787,056	13,178,581	9,898,189	-
Op Cost per Tonne	$ 1.78	$ 2.02	$ 2.04	$ 2.03	$ 2.14	$ 2.64	$ -
CAPEX - Total	**$ 82,106**	**$ 366,110**	**$ 416,198**	**$ 509,984**	**$ 51,596**	**$ 171,630**	**$ 82,106**
CAPEX - Initial Lease	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Prestripping	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Initial (Other)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Sustaining (Balloon payments on leased equipment)	$ 82,106	$ 366,110	$ 416,198	$ 509,984	$ 51,596	$ 171,630	$ 82,106

1.10.2 Plant Operating Cost

The process plant operating costs are summarized by areas of the plant and then by cost elements of labor, power, reagents, grinding media, wear items, maintenance parts and supplies and services. A summary of the process plant operating costs for LOM operations is shown in Table 1-4.

Labor costs were developed from current mining projects in Peru. Power is based on a quoted cost of $0.051 per kWh. Reagents and grinding media were priced in Peru while maintenance parts, supplies and services were estimated as factors on the cost of plant equipment.

Table 1-4: Process Plant Operating Cost

Cost Item	LOM (000s)	$/Tonne Ore
Primary Crushing	$34,127	$0.248
Grinding	$383,410	$2.784
Flotation	$546,383	$3.968
Concentrate & Tailings Thickening & Filtration	$189,506	$1.376
Ancillary	$53,488	$0.388
Total Process Plant	**$1,206,914**	**$8.764**

1.10.3 G&A Cost

The operating cost for the General Administration areas were determined and summarized by cost element. The cost elements include labor (136 employees), supplies, support infrastructure, services, and other expenses. In addition to these cost a contingency was added in the amount of $1.0 million. The departments included are as follows:

- Administration
- Controller's
- Human Resources
- Purchasing
- Safety & Environmental

Table 1-5 lists the estimated annual costs in an LOM of production and the cost per tonne of ore for G&A costs.

Table 1-5: G&A Operating Cost

Cost Item	LOM Cost (000s)	$/tonne
Labor & Fringes	$55,601	0.404
Power	$4,682	0.034
Vehicle Operating & Maintenance	$13,050	0.095
Communications	$2,727	0.020
Safety Supplies / Incentives	$3,240	0.024
Offsite Training & Conferences	$648	0.005
Insurance	$19,332	0.140
Corporate Services and Travel	$16,092	0.117
Environmental	$1,944	0.014
Security & Medical	$4,455	0.032
Professional Membership Costs	$108	0.001
Community Development	$5,400	0.039
Bussing (150 weekly and 40 per day)	$4,032	0.029
Staff Living Expenses (250 people at the camp)	$29,565	0.215
Consultants	$1,215	0.009

Cost Item	LOM Cost (000s)	$/tonne
Computer Equipment/Software	$675	0.005
Misc. Office Supplies	$324	0.002
Misc. Freight & Couriers	$324	0.002
Recruiting and Relocation	$3,564	0.026
Mine Access Road Maintenance	$7,042	0.051
Legal, Permits, Fees	$4,950	0.036
Contingency (10%)	$17,902	0.130
Total General & Administrative Cost	**$196,873**	**$1.430**

1.11 CAPITAL COST ESTIMATE

The project capital cost estimate has been prepared by two independent engineering companies, M3 and GRE. The mining costs were prepared by GRE, the process and portions of the infrastructure capital cost have been prepared by M3 and the mining capital costs, waste/filtered tailings co-disposal facility and remaining infrastructure costs have been prepared by GRE. In addition, capital costs for the mine access road (Anddes & HC&A), the power transmission line (Promotora), and the operating and construction camp (EMSA) were supplied by qualified Peruvian companies. The initial startup capital is estimated to be $625M as summarized on Table 1-6.

The sustaining capital cost is estimated to be $39M total or $2.2M annually over the life of mine. The capital costs include detailed long-term plans for mining fleet buyouts of leased equipment, a new haul road in Year 7, surface water management changes as the site develops, and an allowance for plant equipment maintenance.

Table 1-6: Initial Capital Cost Summary

AREA	2015 TOTAL ($000)
Direct Cost	**$352,062**
General Site	11,021
Mine Capital + Preproduction	63,066
Primary Crushing	24,335
Reclaim Stockpile	8,413
Grinding	48,921
Flotation and Regrind	49,200
Concentrate Thickening & Filtration	16,174
Tailing Thickening & Tailings Pond (See Note 1)	9,443
Tailings Filtration (See Note 1)	55,263
Fresh Water/ Plant Water	11,491
Power Supply Infrastructure	8,551
Reagents	12,881
Ancillaries	33,302
Indirect Cost	**$104,735**
Contractor Indirects	20,436
EPCM Services	45,732
Commissioning and Vendor Reps	1,976
Capital & Commissioning Spare Parts & Initial Fills	11,913
Freight, Duties	24,679
Owners Costs	**103,180**
General Owner's Cost Items	26,718
Operating and Construction Camp	27,918
Mine Access Road (All sections)	32,623
Power Transmission Line	15,921
Contingency	**65,150**
Contingency (Process Plant)	59,746
Contingency (Mine)	5,404
Total	**$625,127**

1) No tailings pond in 2015 plant design as tailings will be filtered.

1.12 ECONOMIC ANALYSIS

The economic analysis was performed using a Discounted Cash Flow (DCF) which is a standard industry practice. The key assumptions used for the study are shown in Table 1-7 and establish a "Base Case". The table provided the life-of-project averages for grade recovery and these values vary over the life of the project depending on the head grades and split between mixed sulfide ore and transition ore.

Table 1-7: Key Assumptions for the Corani Project – Base Case

Annual ore production – years 1 to end of life (ktonnes)	7,875
Overall process recovery – silver – into both lead and zinc cons	71.9%
Overall process recovery – lead – into lead cons	62.8%
Overall process recovery – zinc – into zinc cons	60.1%
Total processed ktonnes	137,698
Average silver grade (g/t)	51.6 g/t
Average lead grade (%)	0.91%
Average zinc grade (%)	0.59%
Payable ounces of silver net of smelter payment terms (total)	151 million
Payable pounds of lead net of smelter payment terms (total)	1.65 billion
Payable pounds of zinc net of smelter payment terms (total)	910 million
Overall stripping ratio	1.68 to 1
Life-of-Mine years	18

The results of the economic analysis for the project has an after-tax internal rate of return (IRR) of 20.6%, net present value of $643 million at 5% discount rate based upon metal prices of $20.00 per ounce silver, $0.95 per pound for lead and $1.00 per pound zinc.

1.13 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The mineral resources were developed from a computer block model of the mineralization and the development of a potentially mineable geometry to establish the component of the deposit with reasonable prospects of economic extraction.

The mineral reserve is a subset of the mineral resource and is comprised of the proven and probable category ore that is planned for processing over the life of the mine plan. No economic credit has been applied to inferred mineralization in the development of the mineral reserve.

The detailed economic, process recovery, and slope angle information that was used to define the mineral reserves and mineral resources are summarized in Sections 14 and 15.

Mineral reserves were developed with metal prices of $20.00/oz silver, $0.95/lb lead, and $1.00/lb zinc. The economic cutoff for mineral reserves with those prices is $11.00/tonne NSR (Net Smelter Return).

Mineral resources were developed with metal prices of $30.00/oz silver, $1.425/lb lead, and $1.50/lb zinc. The economic cutoff for mineral resources was $11.00/tonne NSR, with a leachable silver cutoff of 15g/tonne silver above 4900 masl.

Table 1-8 summarizes the mineral reserves and the mineral resources in addition to the mineral reserves.

Table 1-8: Mineral Reserves and Mineral Resources

Mineral Reserves, variable $23.00-11.00 NSR cut-off							
Total	Ktonnes	Silver gpt	Lead %	Zinc %	Silver Million oz	Lead million lb	Zinc million lb
Proven	19,855	69.1	1.09	0.72	44.1	478.7	313.4
Probable	117,843	48.6	0.88	0.57	184.3	2289.2	1470.7
Proven & Probable	137,698	51.6	0.91	0.59	228	2,768	1,784

Mineral Resources in Addition to Reserves, $11.00 NSR cut-off, 15 g/tonne Ag cutoff (oxide)							
Total	Ktonnes	Silver gpt	Lead %	Zinc %	Silver million oz	Lead million lb	Zinc million lb
Measured	14,360	32.01	0.34	0.19	14.8	108.4	61.6
Indicated	83,749	25.37	0.37	0.28	68.3	682.2	512.8
M&I	98,109	26.34	0.37	0.27	83.1	790.6	574.4
Inferred	39,953	37.20	0.58	0.40	47.8	510.6	352.4

Mineral Reserves are based on metal prices of $20/oz silver, $0.95 per pound for lead and $1.00 per pound for zinc using variable NSR cutoffs throughout the project life. The mineral resources uses a Whittle pit shell generated with metal prices of $30/oz for silver, $1.425/lb for lead and $1.50/lb zinc. The Mineral Resource NSR cut-off was $11.00/tonne. The Mineral Resource includes potentially leachable mixed oxide material that fell within the Mineral Resource pit shell using a silver cut-off grade of 15g/tonne and block elevation above 4900 meters.

The mineral resources and mineral reserves were developed by GRE with Terre Lane, QP. acting as the qualified person. Metal price changes or significant changes in costs or recoveries could materially change the estimated mineral resources in either a positive or negative way. At this time, there are no unique situations relative to environmental or socio-economic conditions that would put the Corani mineral reserves or mineral resources at a higher level of risk than any other developing resource in Peru. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

1.14 CONCLUSIONS

The Project has an after-tax internal rate of return (IRR) of 20.6%, net present value of $643 million at a 5% discount rate based upon metals prices of $20 per ounce silver, $0.95 per pound for lead and $1.00 per pound for zinc.

Payable silver production averages 13.5 million ounces per year for the first 5 years. The project will produce 151 million ounces of payable silver (average of 8.4 million payable ounces of silver annually year), 1.6 billion pounds of lead (91.8 million pounds of lead annually) and 910 million lbs of zinc (51 million pounds of zinc annually) over the 18 year mine life.

Total Life-of-Mine cash cost is $3.80/oz silver, net of base metal credits.

Table 1-9: Life of Mine Total Cash Cost

ITEM	Total
Direct Mining Costs	$2,135,293
Transportation and Refining Charges	$881,731
Subtotal	**$3,017,024**
Lead Payable Revenue	($1,569,256)
Zinc Payable Revenue	($909,579)
Total Cash Cost, Net of Lead and Zinc Revenues	**$538,189**
Reclamation	$36,292
Total Cash Cost, Including Reclamation	**$574,481**
Payable Silver Ounces	151,048
Total Cash Cost per Ounce of Payable Silver, Net of Lead and Zinc Revenues	**$3.56**
Total Cash Cost per Ounce of Payable Silver, Including Reclamation and Net of Lead and Zinc Revenues, and Reclamation	**$3.80**

The initial capital investment on the project is estimated to be $625 million with sustaining capital expenditures during mine operations averaging $2.2 million per year over the 18 year mine life. The project achieves payback of capital in 3.6 years using base case metal prices.

The FS is based upon assumptions derived from mine planning sequences completed by GRE and metallurgical test work performed by SGS Laboratories in Vancouver, BC, and other established labs and reviewed by TS Metallurgical Services. The mining sequence primarily derives ore from the higher-grade mining phases in the early years and moves to lower-grade phases in the later years of production. Operations are for 18 years based on current reserves.

In the mine sequence, 228.4 million ounces contained within 137.7 million tonnes have been used as reserves in this mine plan. An additional 98.1 million tonnes of measured and indicated resource (containing 83 million ounces of silver at 26.3 g/t) and 40.0 million tonnes of inferred resource (containing 47.8 million ounces of silver at 37.2 g/t) remain that could be included in later plans of operations. This includes the potentially leachable and the flotation process resource.

1.15 RECOMMENDATIONS

Additional work is required to evaluate assumptions made during this study and provide input to basic and detailed engineering leading up to the start of construction. The recommendations provided below address areas that require more complete definition to inform and optimize the detailed engineering design.

1.15.1 Site Geotechnical

It is recommended that additional work be done to ensure that the currently planned site layout is feasible from a geotechnical standpoint. Some of the assumptions made in designing project facilities require field verification. Specific areas requiring additional field evaluation for detailed design include:

- Building foundations;
- Primary crusher structure, conveyor supports;
- Project support facilities foundation requirement review;
- Roadways;
- Main Dump foundation;

- Pit slopes.

1.15.2 Mine Geotechnical

Additional geotechnical drilling should be completed within the planned pit. This will confirm the current pit slope design basis, and potentially allow an increase in the pit slope angles. The pit will intersect the unconsolidated sediments lining the floor of the upper bofedal and lower bofedal areas. Additional drilling, testing, and analysis are required to design the pit slopes within the bofedal soils and to develop a detailed plan for dewatering and mining the bofedal soil material.

1.15.3 Plant Water Pond

Site investigation and geotechnical design of the Plant Water Pond are required for permitting and for detailed design of the pond system. The process water ponds are required very early in the Project development schedule as a source of water and for sediment control during construction. Geotechnical drilling of the impoundment and embankment dam areas, as well as test pitting of shallow soil cover areas, must be performed.

1.15.4 Metallurgy

It should be verified that smelters selected for the study have the capacity and ability to accept the proposed quantity and quality of produced lead and zinc concentrates. As part of the program, concentrate analysis should be completed to improve the quality of the concentrate and quantify its properties for filtration, transportation, and sale.

Additional metallurgical testing should be completed to optimize the known flotation test conditions. The reagent scheme should be optimized and additional testing could explore the possibility of higher $ZnSO_4$ dosages on difficult samples in addition to removing and/or reducing other depressants.

The geometallurgical model was developed using all metallurgical testing data, including tests representing non-optimized conditions. Additional testing should be performed during detailed engineering and samples representing optimized test conditions are available, the statistical model should be re-evaluated to ensure estimated recoveries represent optimal conditions.

The selected process flowsheet should be re-tested to optimize:

- Primary grind size
- Regrind size
- Residence time for each flotation stage
- Reagent selection and dosage

Additional lock cycle testing is recommended for each deposit, particularly material representing moderate to low zinc grades which is under-represented in the current test database. This will allow for validation of the final estimated recoveries and the selected concentrate grades. This testing should include analysis of minor elements; limited test data is available regarding the concentration of minor elements in the final concentrate.

The geotechnical testwork on filtered tailings described in Section 26.1 may lead to the need for additional tailing filtration testing. The filtration equipment needed for mechanical placement of tailings is a significant capital and operating cost to the project. It will be important to further determine the filtered tailing physical properties so that the equipment can be specified to achieve the optimal results.

Testing of an acid brine leaching process for oxide mineral resources should be considered, or testing of an alternative flowsheet, for the non-floatable resource areas of the deposit. If this were successful, it could potentially add value to the project.

1.15.5 Constructability

Generation of a site-wide cut-and-fill material balance is recommended, including specialized construction materials such as clays, concrete aggregate, drain rock, road base, and rip rap.

A detailed Project execution plan and schedule should be produced.

1.15.6 Optimization

It may be possible to reduce the cost of delivering tailings to the Main Dump and pit backfill disposal sites by varying the proportion of tailings delivered by conveyor systems and by trucks during the period when tailings are being produced. It is recommended that an optimization study be carried out to determine this, and a detailed plan should be devised. Tailings will be co-disposed with waste rock in the Main Dump and pit backfills. In general, it is expected that it will be cheaper to use conveyors instead of trucks to deliver tailings to ultimate disposal destinations, but exclusive use of conveyors may be less practical for tailings destined for the pit backfill. During the period when pit backfilling will be taking place, the current schedule indicates that truck capacity will be available; therefore, an optimization study should specify the ideal mix of conveyor/truck transport of tailings over time, depending partly on truck availability.

Use of a maintenance-and repair-contract (MARC) for the mine fleet should be investigated. This could reduce the skilled-trade staff.

A study should be conducted to match operating equipment to the high-altitude conditions, potentially identifying equipment outfitted with pressurized cabs and other worker comfort and performance additions. Caterpillar equipment offers high-altitude arrangements (HAA), and these modifications allow their power ratings to be valid to 4,877 m.

1.16 RISKS

The following risks are identified:

- The high altitude of the site may have a greater-than-expected negative impact on worker productivity.
- The high altitude of the site may result in greater-than-expected impacts on the function and capacity of diesel-powered equipment and electrical components.
- As with any large scale mine development, there is a risk that additional capital may be difficult to raise in the event that costs increase during the preproduction period.
- A currency exchange risk exists. While a weakening of the Peruvian Nuevo Sol (PEN) would lower the cost of in-country expenses, imported materials, priced in USD, would become higher. However, during operations the sale of metals, priced in USD, would benefit from a weaker PEN.
- Although local communities have generally supported the project development, there is a risk that sentiments could change, or that special interest groups from outside the community could mobilize opposition to the project.

1.17 OPPORTUNITIES

The following opportunities are identified:

- The potential may exist to introduce an oxide circuit to allow treatment of the non-floatable material.
- It may be possible to improve metal recoveries by optimizing flotation work. Testing under optimized conditions could increase recovery over that predicted by the geometallurgical model.
- Data generated during additional geotechnical drilling may show that it is feasible to steepen pit slopes.
- It may be possible to improve concentrate grades and increase the net smelter return.
- Operating cost improvements may be derived from using conveyor systems to transport tailings to the disposal sites.

1.17.1 Gold Zone

The Gold Zone is an advanced exploration target on the Corani Property. To advance the Gold Zone target, additional metallurgical test work should be undertaken to identify an appropriate recovery method so that capital and operating costs can be developed for a recovery plant. When a recovery strategy is determined, a mineral resource should be estimated for this area so that scoping level studies can be undertaken to evaluate the Gold Zone's economic potential.

2 INTRODUCTION

2.1 PURPOSE

Bear Creek Mining Corporation (BCM) commissioned M3 Engineering and Technology Corporation (M3) to prepare a feasibility study update for the Corani Project in Puno Region, Peru. The technical report presented here is the National Instrument 43-101 project report that summarizes the feasibility study update. This Report is based on the outcomes of an engineering study completed by several authors (described below) to Feasibility Study (FS) standards.

The Project includes grinding and flotation of mixed sulfide, and transitional ores, for proven and probable Mineral Reserves containing 137.7 million tonnes of ore having a Life-of-Mine average grade of 51.6 ounces per tonne of silver, 0.91% lead, and 0.59% zinc, containing, 228.4 million ounces of silver, 2.8 billion pounds of lead and 1.8 billion pounds of zinc over a period of 18 years of mining.

This Report was prepared in conjunction with an updated NI 43-101 resource estimate completed by Global Resource Engineering Ltd (GRE), a consulting company based in Denver, Colorado, Tom Shouldice independent Metallurgist Consultant and with Christian Rios independent geological consultant. The Report has been prepared in accordance with "Form 43-101F1, Technical Report" of the Canadian Securities Administrators National Instrument 43-101 (NI 43-101).

A scoping level study was completed in 2008 for the Project and a pre-feasibility level study in 2009. In 2011, a NI 43-101 compliant feasibility study was prepared followed by additional optimization work. This updated feasibility study documents the culmination of the optimization work to investigate and reduce the capital costs, raise the internal rate of return, and to reduce risk by streamlining the project and performing additional engineering and optimization of the site layout.

Since the 2011 feasibility study the following have been completed:

- Drilled 12new metallurgical holes;
- Undertaken new metallurgical studies including throughput study, grinding power, geotechnical tests, sedimentation and filtration tests on tailings samples.
- Developed a new geometallurgical model that allows better prediction of metal recoveries based on geological and mineralogical parameters;
- Developed a new mine plan and schedule to optimize the current resources and eliminate marginal mineral reserves.
- Redesigned specific areas of the plant to optimize capital costs including changing the primary crusher, grinding mills, flotation arrangement, regrind mills, and switching from conventional wet tailings to filtered tailings.
- Developed a design and methodology for waste rock and filtered tailings co-disposal in the Main Waste Dump followed by backfilling of the Corani Este pit;
- Eliminated the East Waste Dump, Tailings Storage Facility (TSF), and the South Water Supply Pond;
- Developed a new site water balance model and enlarged the Plant Water Supply Pond;
- Re-organized the site plan and plant ancillary buildings to better accommodate the new design and phasing;
- Incorporated a new mine access road design that significantly deviates for the previous design by the addition of a tunnel to cut off approximately 9 km of switchbacks;
- Included a new power transmission line design to reduce costs;
- Incorporated a new operations camp design;
- Updated capital and operating costs based on current equipment and material supply prices and current labor rates;

- Prepared a new financial model based on all of the above.

The Feasibility Study incorporates an updated resource estimation and mine design performed in April 2015 by GRE based upon 85,198 meters of drilling and sampling in 495 diamond drill holes and 25 trenches totaling 1297 meters completed through January of 2012. The studies have included revision of the mineral resources and mineral reserve estimates, open pit optimization and pit design, metallurgical test work and process design, preliminary geotechnical/hydrological investigations and design for mine waste management and infrastructure, estimates of capital and operating costs, and assessment of the economics to develop the project as an open pit mine and process plant. The information available to May 2015 provided the basis for this Report.

2.2 SOURCES OF INFORMATION

This Report is the product of technical contributions from a number of consultants; together with BCM personnel. Listed below are the primary "Qualified Persons" (as defined in the National Instrument 43-101) that compiled different sections of the report. Table 2-1 describes the primary contributors by section.

- Daniel H. Neff, P.E., M3 will be the principal Qualified Person ("QP") and author of the study.
- Terre Lane, P.E., Global Resource Engineering Ltd. (GRE) – Resource and Reserve Estimation and Mine Engineering;
- Tom Shouldice, C.Eng., TS Technical Services (TS) – Metallurgy;
- Chris Chapman, P.E., Global Resource Engineering Ltd. (GRE) – Geotechnical, Water and Environmental
- Christian Rios, Consulting Geologist, Exploration, Drilling, & Data Collection,
- Rick Moritz, P.Eng., Global Resource Engineering Ltd. - Geometallurgy
- Laurie Tahija, M3 - Process Engineering

Table 2-1: List of Contributing Authors

SECTION	SECTION NAME	Main Contributor	Qualified Person
1	Summary	M3, GRE & TS	Daniel H. Neff - M3, Chris Chapman - GRE & Tom Shouldice –TS
2	Introduction	M3	Daniel H. Neff – M3
3	Reliance on Other Experts	M3	Daniel H. Neff – M3
4	Property Description and Location	BCM	Daniel H. Neff – M3
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	BCM	Daniel H. Neff – M3
6	History	BCM	Daniel H. Neff – M3
7	Geological Setting and Mineralization	Consultant	Christian Rios
8	Deposit Types	Consultant	Christian Rios
9	Exploration	Consultant	Christian Rios
10	Drilling	Consultant	Christian Rios
11	Sample Preparation, Analyses and Security	Consultant	Christian Rios
12	Data Verification	Consultant	Christian Rios
13	Mineral Processing and Metallurgical Testing	TS & GRE	Tom Shouldice - TS, & Rick Moritz - GRE.
14	Mineral Resource Estimates	GRE	Terre Lane – GRE
15	Mineral Reserve Estimates	GRE	Terre Lane – GRE
16	Mining Methods	GRE	Terre Lane – GRE
17	Recovery Methods	M3	Laurie Tahija – M3
18	Project Infrastructure	M3 & GRE	Daniel H. Neff – M3 & Chris Chapman – GRE
19	Market Studies and Contracts	M3 & BCM	Daniel H. Neff – M3
20	Environmental Studies, Permitting and Social or Community Impact	GRE & BCM	Chris Chapman – GRE
21	Capital and Operating Costs	M3 & GRE	Daniel H. Neff – M3 Terre Lane – GRE
22	Economic Analysis	M3	Daniel H. Neff – M3
23	Adjacent Properties	BCM	Christian Rios
24	Other Relevant Data and Information	M3	Daniel H. Neff – M3
25	Interpretation and Conclusions	All	Daniel H. Neff – M3
26	Recommendations	All	Daniel H. Neff – M3
27	References	M3	Daniel H. Neff – M3

Abbreviations: ALL – All QP Contributors; BCM – Bear Creek Mining Corporation; GRE – Global Resource Engineering Ltd; M3 – M3 Engineering & Technology Corporation, TS – TS Technical Services
Note: Where multiple authors are cited, refer to author certificate (Appendix A) for specific responsibilities.

This Report has been compiled for BCM by M3, GRE, TS Metallurgical Services, and Christian Rios, collectively the Authors. The Report is based on information and data supplied to the Authors by BCM and other parties. The Authors have relied upon the data and information supplied by the various qualified persons listed above as being accurate and complete.

The Authors have relied on information provided by BCM and on information provided from previous studies. Where possible, the Authors have confirmed the information provided by comparison against other data sources, similar projects in Peru and South America or by field verification.

Where checks and confirmations were not possible, the Authors have assumed that all information supplied in the previous technical report is complete and reliable within normally accepted limits of error. During the normal course of the review, the Authors have not discovered any reason to doubt that assumption.

This Report conforms to the standards of a Feasibility Study and a NI43-101 Technical Report. This Report, based on the work completed to date, is intended to summarize the work performed to date on the Project and to evaluate the economics of the plant operating at 22,500 MTPD. The study sets forth conclusions and recommendations, based on the Authors' experience and professional opinion, which result from their analysis of work and data collected.

In accordance with the feasibility nature of the Report, M3 and the other contributors have used estimates and approximations based on experience and expertise. Where such estimates and approximations have been used, it is so noted and the assumptions made in making such estimates and approximations are so noted.

This Report should be construed in light of the methodology, procedures and techniques used for its preparation, and should be read in original context - all readers should refer to referenced documents for clarification of the original context.

2.3 SITE VISIT & PERSONAL INSPECTIONS

The following site visits were made by the groups and individuals listed below. Tom Shouldice, C.Eng., of TS Metallurgical Services, has not visited the site but has visited SGS Vancouver on several occasions where most of the metallurgical test work has been performed.

Daniel H. Neff, of M3 Engineering and Technology, visited the Project site from December 8 to 9, 2010. The primary focus of the site visit was to observe firsthand the project site and gain an understanding of potential issues related to the general infrastructure, mine access road, process plant location options and availability of services required for plant operation.

Chris Chapman, PE, of Global Resource Engineering Ltd. visited the project site on seven occasions from November 2010 to May 2012. Mr. Chapman was responsible for management of the site investigation programs performed by GRE in 2011 and 2012.

Terre Lane and Rick Moritz of Global Resource Engineering Ltd. have not visited the project site.

Christian Rios, P.G., Consulting Geologist, worked at the site 3 years, and visited the project on several occasions from December 2004 to March 2015.

Other BCM and consultant personnel have made visits to the site. In summary, the following groups have made site visits, including Qualified Persons and others:

- M3
    - December 8-9, 2010 by Dan Neff

- GRE
    - Multiple site visits over the period from November 2010 to May 2012.

2.4 TERMS OF REFERENCE

The units of production in this report are metric unless otherwise noted. Production is in tonnes (t). All monetary amounts are in 2nd Quarter 2015 US dollars along with other variables such as the price of silver, lead and zinc, unless otherwise noted.

Table 2-2: List of Acronyms

Abbreviation	Definition
ARD	Acid Rock Drainage
BCM	Bear Creek Mining Corporation
CERTAG	Silver Value from Assay Certificate
CIF FO	Cost Insurance & Freight Free Out
DCS	Distributed Control System
EDO	Emulsified Diesel Oil
EPC	Engineering Procurement and Construction
EPCM	Engineering Procurement and Construction Management
ERFP	Engineering Requisition for Purchase
ESIA	Environmental and Social Impact Assessment
FS	Feasibility Study
GA	General Arrangement
GFA	General Facilities Arrangement
GRE	Global Resource Engineering Ltd.
IGV	Impuesto General a las Ventas (Peruvian value added tax)
INACC	Instituto Nacional de Concesiones y Catastro Minera
IRR	Internal Rate of Return
M3	M3 Engineering & Technology Corporation
MINEM	Ministerio de Energía y Minas
NAG	Non-acid generating
NPV	Net Present Value
NSR	Net Smelter Return
PAG	Potentially Acid Generating
PDS	Power Distribution Center
PE	Plan of Execution
PEA	Preliminary Economic Assessment
PFS	Prefeasibility Study
PMT	Post Mineral Tuff
QEMSCAN	Quantitative Evaluation of Minerals by Scanning Electron Microscopy.
RDi	Resource Development, Inc. (Wheat Ridge, Colorado)
ROI	Return on Investment

Abbreviation	Definition
SAMPNO	Sample Number
SOW	Scope of Work
TS	TS Technical Services (Tom Shouldice)
WRF	Waste Rock Facility
PEN	Peruvian New Sol
R^2	Coefficient of Determination
Ma	Million Years
RF	Revenue Factor (Whittle)
TSF	Tailings Storage Facility
QA/QC	Quality Analysis/Quality Control
bcm	Bank Cubic Meter
MARC	Maintenance and Repair Contract
SGS	SGS Mineral Services Laboratory
G&T	G&T Metallurgical Services
LCT	Locked Cycle Test
UTM	Universal Transverse Mercator

Table 2-3: Glossary

Term	Definition
Bofedal	Organic soil found in the wet areas in the central parts of the valley
Campesino	A term in Spanish meaning farmer.
Quebrada	A term in Spanish, meaning gorge, valley or draw.
Tailing	Finely ground materials from which the desired mineral values have been largely extracted. Typically, approximately 98 per cent of the material mined for processing is discharged as tailing.
Waste rock	Material such as soils, barren or uneconomic mineralized rock that surrounds a mineral ore body and must be removed in order to mine the ore. This is generally referred to as waste rock in metalliferous mines.

Table 2-4: Units of Measure

Unit Abbreviation	Definition
Ag	Chemical Symbol for Silver
cm	centimeter
d	day
dmt	dry metric tonne
ft	foot
g	gram
g/t	gram per tonne (metric)
gm/t	gram per tonne (metric); alternate spelling
h	hour

Unit Abbreviation	Definition
ha	hectare
hp	horsepower
kg	kilogram
kg/t	kilogram per tonne (metric)
km	kilometer
kph	kilometers per hour
kt	kilotonne
ktpy	kilotonnes per year
kW	kilowatt
kWh	kilowatt hours
kWh/t	kilowatt hours per tonne (metric)
lb	pound
m	meter
m^2	square meter
m^3	cubic meter
min	minutes
mm	millimeter
opy	ounces per year
oz	ounces
Pb	Chemical Symbol for Lead
t	tonne (metric)
tpd	tonnes (metric) per day
tph	tonnes (metric) per hour
tpy	tonnes (metric) per year
µm	micrometer
wmt	wet metric tonne
Zn	Chemical Symbol for Zinc

3 RELIANCE ON OTHER EXPERTS

The Qualified Persons (QPs) for this report have relied upon certain reports, opinions and statements of legal and technical experts who are not considered "Qualified Persons", as defined by NI 43-101. Reports received from other experts have been reviewed for factual errors by the relevant QPs and determined that they conform to industry standards, are professionally sound, and are acceptable for use in this Report. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

3.1 MINING CONCESSIONS

Legal review concerning the status of mineral concessions covering the Corani Project was conducted by Estudio Grau Abogados. The status report (Estudio Grau, 2015) with respect to the mining concessions on the Project property stated that the claims are in good standing and that BCM owns the title.

3.2 POWER SUPPLY

A design and cost estimate for a 138 kV power transmission line and substation connecting the Corani Project Main Substation to the Peruvian national power grid was provided by PROMOTORA de Proyectos S.A.C. (2015a). PROMOTORA is a consulting engineering and EPCM based in Peru that design and constructs power lines. PROMOTORA also provided a design and cost estimate for a 13.8 kV distribution line between the Corani Main Substation and the Corani Residential Camp (PROMOTORA, 2015b). The QPs for this report have relied upon these documents for the design and capital cost of this critical infrastructure improvement.

3.3 MINE ACCESS ROAD

A design and cost estimate for an access road from the Interoceanic Highway (Carretera Interoceánica) to the Corani Residential Camp was provided by Anddes Asociados, S.A.C. (Anddes, 2015). A design and cost estimate for the access road from near the Corani Residential Camp to the Corani Mine Entrance was provided by H.C. & Asociados (HC&A, 2012).

3.4 CORANI RESIDENTIAL CAMP

A design and cost estimate for a residential camp for the Corani Project was provided by EMSA, S.A. (EMSA, 2015). The design included water supply, water treatment, residential housing, cooking and cleaning facilities, administrative offices, utilities (power, water, fire suppression, and telecommunications distribution), parking, and wastewater treatment.

3.5 MARKETING STUDIES AND TREATMENT TERMS

Andes Mining Research conducted a review of the lead and zinc concentrate markets, smelting charges, penalties, concentrate handling and land and ocean transportation costs.

3.6 GEOTECHNICAL

Anddes Asociados S.A.C. carried out plant geotechnical site investigations and laboratory testing based on the plant configuration presented in the 2011 Feasibility Study (Anddes Asociados S.A.C., 2012). Although some plant facilities have been relocated outside the Anddes study area, the study remains applicable to most of the plant complex considered in this study.

In 2009, Vector Peru performed a Pit Slope Stability Study. In 2012, McDonald Engineering Services, LLC performed the geotechnical evaluation provided the pit slope recommendations adopted for this study (McDonald Engineering Services, LLC, 2012).

The QPs for this report have relied upon these documents for the design of the plant site foundations and pit slopes.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 LOCATION

The Project site is located in the Andes Mountains of south-eastern Peru at elevations of 4800 to 5100 meters above sea level (masl), specifically within the Cordillera Vilcanota of the Eastern Cordillera. The site is located in the Region of Puno, immediately northeast of the continental divide that separates Pacific drainages from Atlantic drainages. The site location is approximately 160 km in a direct line to southeast of the major city of Cusco, with Universal Transverse Mercator (UTM) coordinate ranges of 312,000E to 322,000E and 8,443,000N to 8,451,000N. Figure 4-1 illustrates the general location on the map of Peru. The nearest town of significant size and infrastructure is Macusani, which is located around 30 km to the east of the Project.

Figure 4-1: Corani Project Location in Peru

4.2 MINERAL TENURE

4.2.1 Summary

The land status of the Project is a series of twelve mineral claims or concessions. Mineral concessions in Peru are filed with the Instituto Nacional De Concessiones Y Catastro Minera (INACC) which is part of the Ministerio de Energía y Minas in Peru (MINEM). Claims can vary in size from 100 to 1,000 ha. They are rectangular geometries parallel to the UTM grid system employed in the district. The Corani Project is located in the districts of Corani, Macusani and Nuñoa, provinces of Carabaya and Melgar, department of Puno, in Peru, and covers an aggregate extent of 5,180.1213 hectares.

4.2.2 Purchase Agreements

On March 15, 2007, the Company (BCM or “Bear Creek”) and Rio Tinto Mining and Exploration Ltd. ("**Rio Tinto**") executed a definitive option and shareholders’ agreement (the "**Option Agreement**") in respect of the Corani Property. The Option Agreement formally defines and confirms the terms as set out in the Letter of Understanding signed between the parties on January 19, 2005. Refer also to "Mineral Projects – Corani Silver-Zinc-Lead Property".

On January 15, 2008, the Company made the final US$3 million payment to Rio Tinto under the Option Agreement, resulting in the Company owning 70% of the Corani Property, subject to certain success payments, purchase rights provisions, and claw-back rights as previously disclosed. Under the terms of the Option Agreement, upon Bear Creek earning its 70% interest, the parties were required, within 100 days of January 15, 2008, to enter into a joint venture agreement, with Rio Tinto having a 30% interest, and dilution provisions for each party which conform to industry practices.

On March 6, 2008, Bear Creek entered into an agreement (the "**Purchase and Sale Agreement**") with Rio Tinto, which was subsequently amended, as described below, to purchase Rio Tinto’s remaining 30% interest in the Corani Project and extinguish all of Bear Creek’s future payment obligations, royalties and Rio Tinto’s back-in rights under the Option Agreement. Bear Creek agreed to pay Rio Tinto US$45 million and to issue Rio Tinto 3,871,000 common shares as follows: (i) 3,871,000 Bear Creek common shares which were issued on July 16, 2008; (ii) US$5 million in cash payable by December 31, 2008; (iii) US$15 million in cash payable by the earlier of December 31, 2008 or 15 business days following a change of control of the Company; and (iv) US$25.0 million in cash payable on the earlier of December 31, 2009 or 180 days following a change of control of Bear Creek.

On July 17, 2008, the Company amended the terms of the Purchase And Sale Agreement and agreed to issue an additional 120,000 common shares to Rio Tinto, in consideration for which Rio Tinto extended US$15 million of the US$20 million cash payment which has been required to be made under the Purchase And Sale Agreement by the earlier of December 31, 2008 and 15 business days following a change of control of the Company, to the earlier of September 30, 2009 and 90 days following a change of control of the Company. This increased the number of common shares issuable to a total of 3,991,000 shares.

On February 27, 2009, the Company entered into an amendment agreement (the "**Amendment Agreement**") with Rio Tinto with respect to its purchase of Rio Tinto’s remaining 30% interest in the Corani Project. Under the Amendment Agreement, Rio Tinto agreed to restructure the final two cash payments of US$15 million previously due on September 30, 2009 and US$25 million previously due on the earlier of December 31, 2009 or 180 days following a change of control of Bear Creek. In consideration for deferring the majority of these payments out several years, the purchase price increased from US$75 million to US$77.2 million, representing an increase of US$2.2 million, of which US$36.1 million had been already paid in shares or cash. The restructured remaining payments were then:

- US$10 million due 30 September 2011; and

- US$15 million due 30 June 2012.

Bear Creek agreed to make the following additional payments in consideration for the restructuring payable in either cash or shares, at the option of Bear Creek:

- US$1.1 million upon signing of the Amendment Agreement or as soon thereafter as TSX-V acceptance was received, which was paid on March 11, 2009 by the issuance of 1,021,266 shares of Bear Creek; and
- US$1.1 million cash, which was paid on January 10, 2011.

Additionally, the Amendment Agreement immediately removed the accelerated payment condition upon change of control of Bear Creek. The Amendment Agreement also provided for the reduction of the security against the balance of the payments, to security charging only the Corani Property upon completion of US$10 million.

On February 3, 2011, the Company entered into an additional amendment agreement (the "**2011 Amendment Agreement**") whereby Rio Tinto agreed to accept a final payment of US$23 million in lieu of and in full satisfaction of the remaining two cash payments of US$10 million due on September 30, 2011 and US$15 million due on June 30, 2012. Accordingly, the Company has paid the entire obligation for its 100% interest in the Corani project early and received a discount of 8% of the balance of the required payments for doing so.

This final payment extinguished all security interests, share pledges and other encumbrances that Rio Tinto held over the Corani Project and Company's other assets. Copies of the Purchase and Sale Agreement, the Amendment Agreement and the 2011 Amendment Agreement may be obtained under the Company's profile on the SEDAR website (www.sedar.com).

4.2.3 Property Identification

According to Estudio Grau (2015), Bear Creek and its subsidiaries own 100% of the title to the twelve (12) mineral concessions comprising the Corani Project, listed in Table 4-1.

Table 4-1: Mineral Concessions comprising the Corani Project

Name	Identification Code	Available Extent (Hectares)
Corani I	010289403	300.0000
Corani II	010289503	300.0000
Corani III	010021905	300.0074
Corani 100	010251005	5.0000
Corani 200	010251105	21.9730
Corani 5	010068505	93.2601
Minazpata 1	010289203	1,000.0000
Minazpata 2	010289303	300.0000
Minazpata 3	010038904	1,000.0000
Minazpata 4	010357604	159.8808
Chaupitera	010250805	800.0000
Pacusani	010250905	900.0000
Total		**5,180.1213**

a. The Corani Project comprises the twelve (12) metallic mineral concessions (collectively the "Corani Project")

b. The Corani Project is located in the districts of Corani, Macusani and Nuñoa, provinces of Carabaya and Melgar, department of Puno, in Peru, and covers an aggregate available extent of 5,180.1213 hectares.

c. The location of the Corani Project is fixed, for all legal purposes, by the UTM coordinates (Datum PSAD 56) for each of their vertices shown on Exhibit III to this legal opinion, all of them duly recorded in the Public Registry.

Figure 4-1 shows the location of the project within Peru. Figure 4-2 shows a map of the Corani mineral concessions within the area.

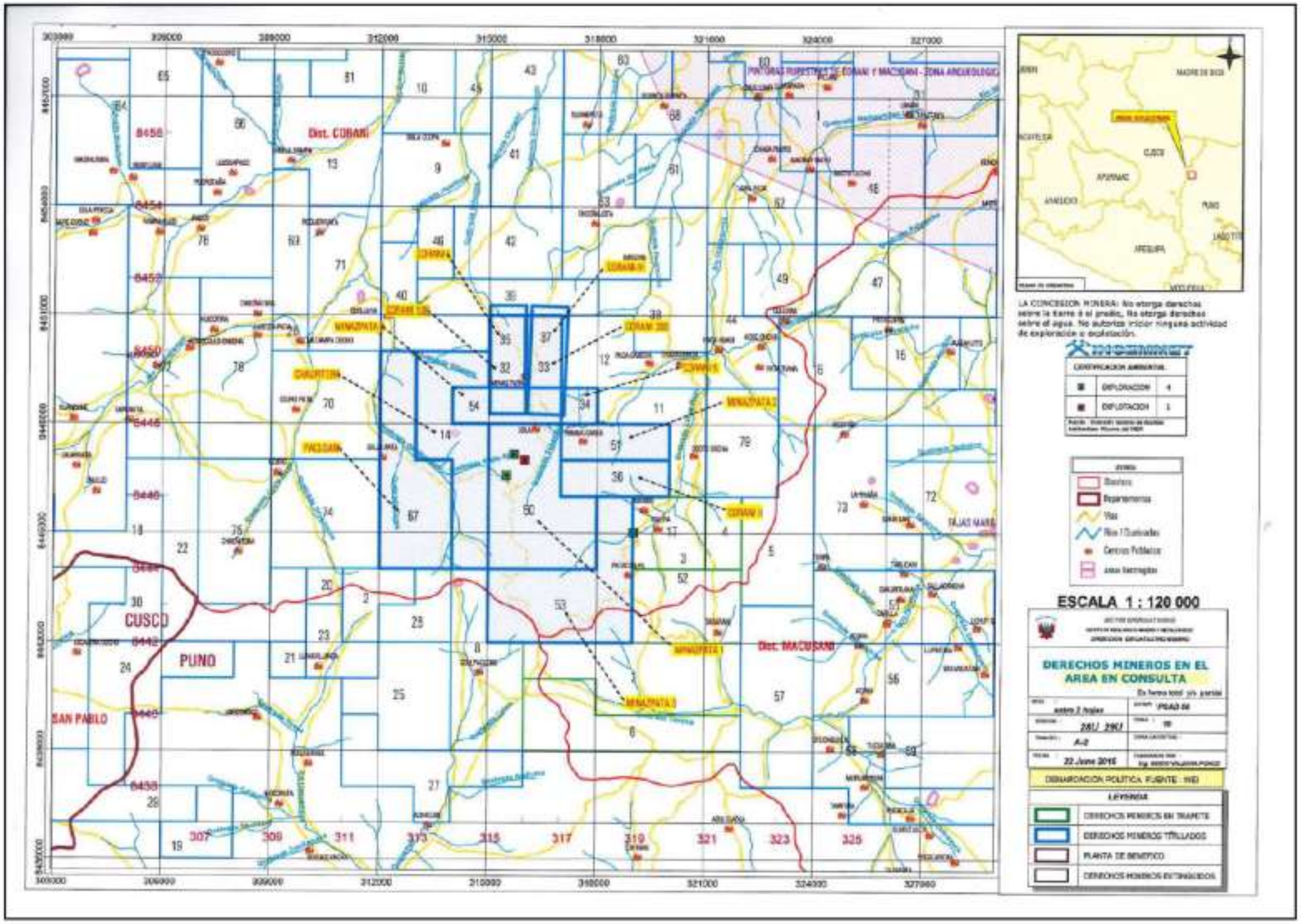


Figure 4-2: Map of Corani Mineral Concessions

4.2.4 Maintenance Obligations

The twelve (12) mineral concessions comprising the Corani Project are subject to compliance with payment of annual license fees in the amount of US$3.00 per hectare ("License Fees").

The mineral concessions comprising the Corani Project are also subject to compliance with either of the following alternative obligations: minimum required levels of annual production of at least US$100 per hectare in gross sales ("Minimum Production"); or payment of an additional amount referred as Penalty of US$6.00 per hectare for the 7th through 11th year following the granting of the concession, and of US$20.00 per hectare thereafter; or exploration expenditures of 10 times the Penalty. Compliance with one of these three maintenance obligations, together with timely payment of License Fees, is required to them in good standing. Failure to comply with License Fee payments or Penalty payments for two consecutive years causes the forfeiture of the mineral concessions.

The twelve (12) mineral concessions comprising the Corani Project are part of an Administrative Economic Unit under the name of Corani ("UEA Corani") duly approved by Geologic Mining and Metallurgic Institute - INGEMMET as the competent governmental agency. Bear Creek is authorized to comply with its maintenance and reporting obligations to the Peruvian State, applicable to the Corani Project through the UEA Corani. This includes the right to report exploration expenditures incurred in one or more mineral concessions to the benefit of the UEA Corani as a whole.

Table 4-2 below shows the projected annual amounts for each of the alternative maintenance obligations to keep the Corani Project in good standing from 2012 through 2016.

Table 4-2: Corani Mineral Concessions Maintenance Obligations

Mineral Concession	Annual License Fees Years 2012-2016 (US$)	Alternative Annual Maintenance Obligations Years 2012-2016				
		Minimum Production in gross sales from 2012-2016 (US$)	Minimum Exploration Expenditures (US$)		Penalty Payment (US$)	
			2012-2013	2014-2016	2012-2014	2015-onwards
Corani I	900.00	30,000.00	18,000.00	60,000.00	1,800.00	6,000.00
Corani II	900.00	30,000.00	18,000.00	60,000.00	1,800.00	6,000.00
Corani III	900.02	30,000.74	18,000.44	60,001.48	1,800.04	6,000.15
Corani 100	15.00	500.00	300.00	1,000.00	30.00	100.00
Corani 200	65.92	2,197.30	1,318.38	4,394.60	131.84	439.46
Corani 5	279.78	9,326.01	5,595.61	18,652.02	559.56	1,865.20
Minazpata 1	3,000.00	100,000.00	60,000.00	200,000.00	6,000.00	20,000.00
Minazpata 2	900.00	30,000.00	18,000.00	60,000.00	1,800.00	6,000.00
Minazpata 3	3,000.00	100,000.00	60,000.00	200,000.00	6,000.00	20,000.00
Minazpata 4	479.64	15,988.08	9,592.85	31,976.16	959.28	3,197.62
Chaupitera	2,400.00	80,000.00	48,000.00	160,000.00	4,800.00	16,000.00
Pacusani	2,700.00	90,000.00	54,000.00	180,000.00	5,400.00	18,000.00
Total	**15,540.36**	**518,012.13**	**310,807.28**	**1,036,024.26**	**31,080.73**	**103,602.43**

4.2.5 Legal Standing

According to Estudio Grau (2015),

(a) The twelve (12) mineral concessions comprising the Corani Project are valid and in good standing.

(b) They were validly applied for and granted title to concession by the competent governmental authority.

(c) Each of the twelve (12) mineral concessions comprising the Corani Project is designated metallic as a mineral concession and allows its titleholder or lessee the exclusive right to explore and exploit all metallic minerals located within their internal boundaries. These mineral concessions are separate from the surface right.

(d) The mineral concessions comprising the Corani Project have been granted to the titleholders for an indefinite period of time, provided that maintenance obligations, including license fee payments, minimum production, investment and/or payment of applicable penalties are attained when due. The mineral concessions comprising the Corani Project will therefore remain valid through the maximum legal deadline to be put into production as long as the titleholder or lessee continues complying with annual license fee payments, qualified investments and/or applicable penalties.

(e) Exercise of the rights derived from the twelve (12) mineral concessions comprising the Corani Project, including the right to explore, develop and further exploit, on an exclusive basis only the designated minerals within the internal boundaries of the mineral concession, is subject to the awarding of the required permits, authorizations and approvals, including relevant surface lands.

4.3 ENVIRONMENTAL LIABILITIES

Historical mining activities have been carried out in the vicinity of the proposed mine and associated facilities. The history of the Project site including ownership, and any known mineral exploration and production, are described in Section 6 of this report.

In accordance with Peruvian Law 28271, generators of environmental liabilities are responsible for remediation activities. Therefore, if historical environmental liabilities are defined, responsibility for these lies with the generator; however BCM can assume responsibility for them in order to expedite the development of the site.

In December 2010, Walsh undertook an environmental-liabilities study in order to declare to the Ministry of Energy and Mines (MINEM) the existence of liabilities left from previous mining activities. These previous mining activities have left excavations, stopes, test pits, and mine portals on the site. During the study site visit, the location of each liability (either previously known or discovered during the study) was inventoried and registered with the MINEM. A total of 141 liabilities were recorded, however it is possible there are others. A number of the environmental liabilities are located within the boundaries of proposed project components; therefore, in order for groundbreaking and or development activities to occur, BCM will need to assume responsibility for these.

Environmental liabilities associated with development of the property (past and future) are managed through an Environmental Closure Plan or Plans. Environmental Liabilities Closure Plan or Plans were approved by the Peruvian Government in April 2015 and must be reviewed within three years following the Peruvian legislation.

4.4 PERMITTING

BCM obtained the permits required for the previous field exploration activities and have identified the permits required for the construction, exploitation and closure phases. An outline of the national, territorial and municipal legislation, and the associated approvals and permits which apply to the Project has been already compiled and referenced in a previously filed technical report. It is considered that there has not been any material change in the permit requirements, notwithstanding some changes to the project description. Table 4-3 presents a summary of the permits required.

Furthermore, as the environmental impact of the proposed Corani operation has been reduced as a result of the modifications described in this Optimized and Final Feasibility Study, the Company anticipates final permitting timelines will shorten and costs will be lower than previously anticipated.

4.5 WATER SUPPLY

The updated water balance and water model prepared will be used to prepare a technical document required to obtain the authorization and permit from the Water National Authority (ANA), designated approving authority. This is the regular procedure in Peru.

4.6 ENVIRONMENTAL AND PERMITTING

The main environmental approval required in order to begin mining activities is an Environmental and Social Impact Assessment (ESIA). In 2013, the Ministry of Energy and Mines approved the ESIA based on the Feasibility Study prepared in 2011.

The design and operating improvements incorporated in the 2015 Corani Feasibility Study are expected to require only a modification of the existing approved ESIA, without the necessity for additional public hearings, as they are entirely located within the previously approved project footprint. Furthermore, as the environmental impact of the proposed Corani operation has been reduced as a result of the modifications described within this report, the Company anticipates final permitting timelines will shorten and costs will be lower than previously anticipated.

Bear Creek's plans for the Corani project are to focus on preparing for development of the project starting with the preparation and submission of the amended ESIA in the third quarter of 2015.

Table 4-3: Summary of Permit Requirements by Phase

	CONSTRUCTION	EXPLOITATION	CLOSURE
ESIA *(modifications through the life of the mine may apply)*	X		
Certificate of Mining Operations *(for explosives use purposes)*	X		
Positive technical opinion of DIGESA *(for the construction of landfills outside the area of mining concessions)*	X		
Closure Plan *(modifications through the life of the mine may apply)*	X		
Certificate of Non Existence of Archaeological Remains – CIRA	X		
Surface water use license	X		
Groundwater use license	X		
Sanitary authorization for wastewater treatment system and discharge	X		
Sanitary authorization for drinking water treatment system	X		
Fuel Direct Consumer's favorable technical report	X		
Registration as a direct consumer of liquid fuels - fixed or mobile facilities	X		
Authorization for eventual explosives use	X		
Explosives shack operation license	X		
License for explosives handlers	X		
Authorization for explosives transportation	X		
Identification code for users of Restricted Chemicals	X		
Verifying deed for the purchase and transportation of IQPFs used by companies	X		
Authorization for opening Special Registry of IQPFs	X		
Incorporation in the Unique Registry for IQPFs	X		
Monthly reports of IQPF Special Registry	X		
Annual Opinion for making, marketing, and warehousing explosives of civilian use and related goods	X		
Definitive Concession for energy transmission line	X		
Individual license for radioactive facilities' handling	X		
Installation license for the operation of fixed nuclear measuring equipment	X		
Transportation Guide of hazardous materials and wastes	X		
Insurance for transportation of hazardous materials and wastes	X		
Certification of transportation personnel	X		
Registry for ground transportation	X		
Special drivers license	X		
Beneficiation Concession		X	
Authorization to start operation		X	
Posting Financial Assurance		X	
Final Closure Plan *(2 years before final closure)*			X
Final Closure Certificate *("Exit ticket")*			X

The main environmental approval required in order to begin mining activities is an Environmental and Social Impact Assessment (ESIA). In 2013, the Ministry of Energy and Mines approved the ESIA based on the Feasibility Study prepared in 2011. The Closure Plan was approved in April, 2015.

The design and operating improvements incorporated in the 2015 Corani Feasibility Study are expected to require only a modification of the existing approved ESIA, without the necessity for additional public hearings, as they are entirely located within the previously approved project footprint. Furthermore, as the environmental impact of the proposed

Corani operation has been reduced as a result of the modifications described above, the Company anticipates final permitting timelines will shorten and costs will be lower than previously anticipated.

Bear Creek's plans for the Corani project are to focus on preparing for development of the project starting with the preparation and submission of the amended ESIA in the third quarter of 2015.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY OTHER RELEVANT DATA AND INFORMATION

5.1 ACCESSIBILITY

Existing access to the Project site is primarily by road from the town of Macusani (located on the paved dual lane Interoceanic Highway) which is more readily accessible from the town of Juliaca, also serviced by commercial airlines from Lima. This route typically takes 4.5 to 5 hours and is also indicated on Figure 5-1. From Juliaca, the route generally aligns north towards the city of Azángaro on the paved Interoceanic Highway. The Interoceanic highway extends approximately 180 km between Azángaro and Macusani. At Macusani, the route extends west and northwest for approximately 60 km to the mine site on improved gravel roads.

Figure 5-1: Map of Existing Access to the Project

There are other access routes to the site from Cusco, taking approximately 6 hours by vehicle on increasingly primitive roads approaching the site. The access route is shown above in Figure 5-1 and passes through Sicuani and the town of Santa Rosa, which is located approximately 208 km southeast of Cusco on a good paved road. From there the route extends approximately 33 km northeast on improved gravel roads to the village of Nuñoa, and continues northeast for around 27 km on a less improved gravel road to the small village of Huaycho. From Huaycho, the access route continues north on an unimproved gravel road for approximately 70 km and ascends a mountain pass to the Project site. The City of Cusco is serviced by commercial airlines from Lima.

5.2 LOCAL RESOURCES AND EXISTING INFRASTRUCTURE

5.2.1 General

The nearest town of significant size and with significant infrastructure is Macusani, which is the capital of the Province of Carabaya in the Region of Puno. Macusani is approximately 30 km east of the Project in a direct line. The access road from Huiquisa Bridge to the Permanent Camp will be improved. The length of the proposed Mine Access Road connecting the process Plant to Macusani is anticipated to be approximately 64 km.

Macusani has a total area of approximately 1,030 km^2 (no data was found specific to the urban area) and its economy is based mainly on agriculture and transportation of agricultural products.

Infrastructure in the town of Macusani includes a national highway - the Interoceanic Highway - currently complete from the Peruvian Port of Matarani to the town of Macusani. Other paved and unpaved roads, trails and footpaths allow access to most areas of the municipality.

The Project site is located in the district of Corani, also in the Province of Carabaya. The closely orientated campesino communities of Chacaconiza and Quelcaya, which have a joint population of approximately 200 families (80 and 120 families respectively), will be directly impacted by the mine development, in terms of landholding, rights to water, employment, etc.

Chacaconiza and Quelcaya are communities that maintain a fragile, high altitude economy. Both communities are below the poverty line with few resources for economic and social development.

The main economic activity in these communities is the raising of alpacas. Approximately 90% of their economy is dependent on this activity, which is augmented to a very marginal degree by trading and seasonal migration.

5.2.2 Available Labor Force

The community consultation undertaken with the Chacaconiza and Quelcaya communities to date has included a proposal for mining employment, generating widespread acceptance, mainly among younger community members, the teachers at local educational facilities and community leaders. The current labor force is generally unskilled, mainly working on highway remediation and maintenance. A technical training program is planned to develop the skills of community members to fulfill employment requirements of the Project, which will include agreements with universities and institutes to improve the local population’s vocational skill levels. The training program will include a system of scholarships that will allow the most successful students to occupy positions of greater responsibility on the project. The training program has been designed to be conducted over a 5 year period. After this, BCM will continue to support the training program.

Mining and services related training will be segmented by age group, to allow older people to be trained for simpler tasks, while younger people will have access to jobs that demand more knowledge and specialization, such as the operation and maintenance of heavy machinery, woodworking and electrical work.

The Project's requirement for labor will exceed the labor resources available in the Chacaconiza and Quelcaya communities. A ranking system will be developed with regard to geographical location of employment applicants, together with categorization and quantification of the labor force required.

5.2.3 Power

The National Interconnected Electric System (SEIN) is the source of power supply for the project. The San Gabán II Hydroelectric Power Station is located on the San Gabán River, some 260 km north-west of the city of Puno and 100 km east of Cusco. The 138 kV power transmission line that connects the Hydroelectric Power Station of San Gabán II (CH San Gabán II) with the SEIN at the Azángaro Substation (SE Azángaro), passes through the neighboring areas of the Project, near the town of Macusani. Therefore, the Project's recommended access to power supply is from the SEIN, connected to this transmission line.

The main facilities that have a direct impact on the Project are described below, and are represented schematically in Figure 5-2.

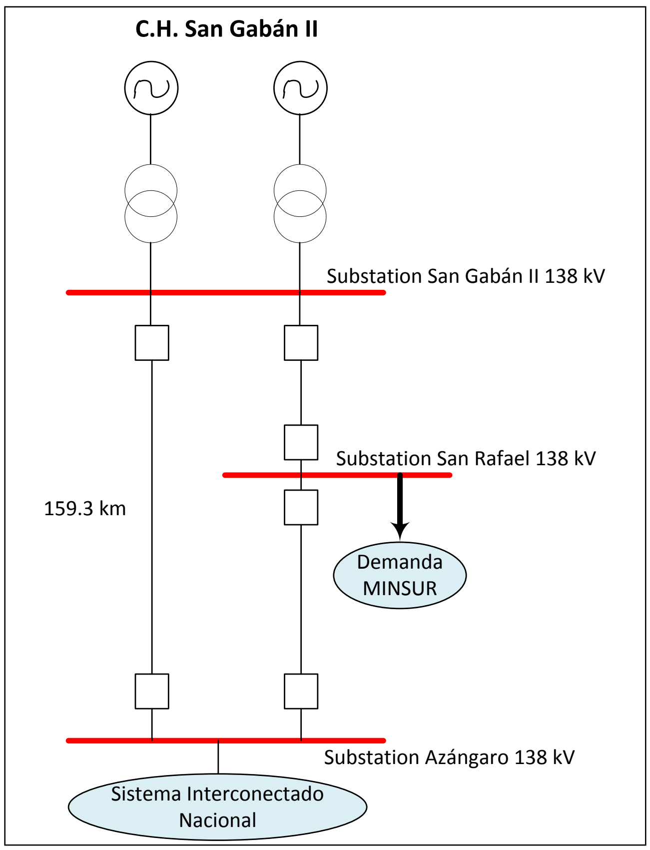


Figure 5-2: Existing Facilities Schematic

San Gabán II Hydroelectric Power Station

This hydroelectric power station is owned by "Empresa de Generación Eléctrica San Gabán", (EGESG), a state-owned company in charge of the operation of the plant's facilities since the end of its construction in the year 2000. The plant's characteristics are the following:

- Number of units : 2
- Power (each) : 54MW
- Energy (annual average) : 800GWh

- **138 kV transmission line CH San Gabán 2 – Substation Azángaro**

The energy produced by the hydroelectric power station is delivered to the SEIN at the Substation Azángaro. The characteristics of this transmission line are the following:

- Number of circuits: 2
- Denomination of the circuits:
    - L-1010 (first circuit)
    - L-1013 and L-1009 (second circuit)

- Tension: 138 kV
- Length: 159.3 km
- Capacity by circuit: 120 MVA

Azángaro Substation

The Azángaro substation is a state-owned substation, granted in concession to Red Eléctrica del Perú (REP), it is part of the southern transmission ring and has a three winding transformer with a capacity of 12/12/5 MVA and voltage of 138/60/22.9 kV. The Substation San Rafael is also connected from this substation through a 60 kV transmission line, which is out of service at the time of this writing.

Southern Transmission Ring

The Peruvian Southern Transmission Ring interconnects the cities of Azángaro, Juliaca, Puno, Moquegua, Arequipa, Tintaya and Ayaviri. The Southern Transmission Ring is represented in Figure 5-3. At the Socabaya Substation of Arequipa city, the southern ring gets interconnected with the electrical system of the center-north, integrating the National Interconnected Electric System (SEIN). The lines between Puno, Azángaro, Tintaya and Arequipa are 138kV and the lines between Puno, Moquegua and Arequipa are 220 kV.

Figure 5-3: Southern Electric Ring Schematic

5.2.4 Water

Surface water in the region is typically taken for farming and livestock watering, as well as essential human needs, such as a drinking and bathing. Surface water and water from springs and marshlands is collected in catchments in the communities of Chacaconiza and Quelcaya and is distributed to basic water supply systems which conduct water to distribution points located in public squares of each respective community.

Nearby the Project site, hydraulic works related to irrigation or water storage were not observed. It was noted that local inhabitants have constructed handmade canals to irrigate pastoral areas, although the structures are considered temporary and many have been abandoned depending on irrigation requirements.

5.3 CLIMATE

The Corani Project metrological station is located in the vicinity of the proposed plant site. Six years of data are available since the station was commissioned in December 2008.

The climate at the project site is characterized by an estimated average annual precipitation of 635 mm, with the highest values recorded between October and April (89% of the annual precipitation). The annual average evaporation was determined to be on the order of 1415 mm with the highest monthly evaporation rates occurring in October (145 mm) and the lowest monthly evaporation occurring in April (87 mm).

The average annual temperature was 1.4°C. The maximum average monthly temperature was 4.0°C during the month of February, while the minimum average monthly temperature was -1.2°C in July. The lowest recorded temperature was -20.8°C in the month of March.

The average relative humidity is around 68%, with monthly averages ranging from a low of 47% in June to a high of 83% in February. The annual average wind speed is estimated to be 2.3 m/s with monthly averages ranging from 1.7 m/s in February to 2.6 m/s in September. The wind direction is generally from the southeast.

Limited comparison of the site data may be drawn to other weather stations in the region. Several regional weather stations have relatively long data records. However, all of the available stations are a significant distance from the project, and only general seasonal trends correlate with the project meteorology station.

Firsthand observation of weather conditions during the 2011 drilling program indicates conditions vary significantly across the Corani Project site. Terrain and elevation have been observed to exert considerable influence on precipitation, temperature, and wind patterns.

Engineering designs should employ appropriate conservatism based on the limited site climate data available. The site climate will allow for year-round operations, with normal operating delays for conditions such as snow and fog. Freeze protection shall be required for all hydraulic works.

5.4 PHYSIOGRAPHY AND VEGETATION

The Project site is located in the eastern Andes mountain range, between 4600 and 5200 m above sea level (masl). The area is characterized by mountainous terrain dominated by volcanic rock above which sits glacial gravel. The lithologic and climatic conditions have given rise to a series of cirques or bowl-shaped, steep-walled basins. During periods of rainfall the valley floors collect precipitation allowing the generation of small wetlands (bofedales).

Apart from the vegetation associated with the wetlands mentioned above, areas of 'puna' or alpine tussock grassland occupies the valleys and moderate to steep slopes. The areas above 4700 masl mostly consist of steep mountainous slopes where erosion and climatic conditions largely prevent the development of soils or vegetation. These areas are scarcely vegetated with species specially evolved to withstand the harsh conditions. The naturally occurring acidic soils related to oxidation of sulfide bearing materials and the resulting ARD from exposed mineralized zones within the project area has also prevented the development of vegetation where these conditions occur.

6 HISTORY

6.1 PRIOR OWNERSHIP AND PRODUCTION

Prior to the early 1950s, mineral exploration in the Corani district consisted of shallow prospect Pits and adits in the northern portion of the current project area. These prospects are of unknown age and may date back to colonial Spanish time. Antimony prospects south and east of the property reportedly were active in the early 1900s, when there was limited antimony production (C.R. Petersen, 1967).

The first modern evaluation of silver-lead mineralization began with the location of mineral concessions in 1951 by Augusto Leon y Leon. In 1953, Fernando de Las Casas visited the site and prepared a geological report titled “The Negrominas – Corani District”. He mentioned that the rocks exposed in the area covered by the Negrominas Claims consist principally of a series of rhyolitic tuffs, breccias and flows tilted to the Northeast. Also, he determined that two main types of ore bearing structures are distinguished at Negrominas. Compañía Minera Korani was formed in 1956 to develop the silver-lead mineralization previously prospected. The mines were developed and operated from 1956 to at least until 1967; initially producing 80 tpd of ore. In 1967, Compañía Minera Korani was owned two-thirds by Compañía Minera Palca and one-third by M. Hochschild. Total historical production is uncertain, but is estimated at 100,000 t of silver-lead-zinc ore. In early 1967, estimated mine production was reported at about 3,400 short tons per month, with grades of 7.0-9.0% lead, 2.3% zinc, and 8.0 to 11.0 oz/ton silver (C.R. Petersen, 1967).

Figure 6-1: Part of the old flotation circuit

Historical maps of the underground workings show development on four levels (4820, 4843, 4860 and 4870 m levels for 50 meters vertically) that extend over an area of approximately 500 meters in a general north-south direction (parallel to strike) by about 150 meters in an east-west direction. It is not known when operations of Compañía Minera Korani ceased, but presumably they ceased in the late 1960s or early 1970s. This mining operation left behind many mine portals, waste piles, and mine tailing that continually produce ARD. Smaller portals are located near the TSF that also emit ARD into the Collpa Mayo drainage.

The next exploration activity was by a private Peruvian company, Minsur. That exploration was reported to include 40 shallow drill holes in various locations, including a number of close proximity holes in the gold zone (located south of the current resource area). Although Minsur is an active mining company in Peru; attempts by BCM to secure copies of Minsur's exploration data have been unsuccessful. None of Minsur's exploration information is available or verifiable; although reportedly gold mineralization was encountered in some of Minsur's drilling.

In late 2003 and early 2004, Rio Tinto Mining and Exploration began a surface exploration program for porphyry copper mineralization. During 2004, Rio Tinto conducted surface mapping, sampling, and ground magnetic surveys, and developed access roads into the area. That initial work by Rio Tinto defined anomalous silver and lead mineralization to the south of the Korani mines, and also defined a zone of anomalous gold mineralization in rock and soils.

The concession ownership by Compañía Minera Korani apparently lapsed during the 1970s. The ownership of Minsur also lapsed prior to Rio Tinto's exploration activities after 2000. Rio Tinto re-established some of the older concessions in their name beginning in 2003. BCM has added two concessions early in 2005 to create the current land position described in Section 4.

6.2 HISTORICAL EXPLORATION AND ESTIMATES

There have been five previous mineral resource estimates for the Project. This document presents the first statement of mineral reserves. The four previous resource statements are summarized below:

1) March 31, 2006, National Instrument 43-101 Technical Report, Initial Resource Estimate for Corani Silver-Gold Exploration Project. SRK Consulting. Tucson, Arizona, United States.
2) October 4, 2006, Corani Project Mineral Resource Technical Report, Independent Mining Consultants, Inc, Tucson, Arizona, United States.
3) May 12, 2008, Technical Report, Corani Resource Estimate and PEA, Independent Mining Consultants, Inc. Tucson, Arizona, United States.
4) October 14, 2009, NI43-101 Technical Report, Prefeasibility Study Corani Project Puno Perú, Vector Perú S.A.C.
5) December 22, 2011, NI43-101 Technical Report, Feasibility Study. Corani Project. M3 Engineering & Technology Corporation, Tucson, Arizona, United States.

The mineral resource tables from each report are summarized in Table 6-1, Table 6-2 and Table 6-3 below in order to illustrate the development of the deposit and the progression of mineral resources. The corresponding resource tables from the earlier IMC updates are summarized on Table 6-2 and Table 6-3. The corresponding reserves tables from the IMC estimates are summarized in Table 6-4 and Table 6-5.

Table 6-1: Mineral Resources - March 2006

		SILVER	LEAD	ZINC
Category	Kilotonnes	g/t	%	%
Measured	7,759	65.12	1.081	0.162
Indicated	20,123	43.61	0.678	0.251
Measured + Indicated	27,882	49.60	0.790	0.230
Inferred	87,627	72.91	1.032	0.578

Table 6-2: Mineral Resource - October 2007

DEPOSIT	CATEGORY	KILOTONNES	SILVER g/t	LEAD %	ZINC %	SILVER Million ozs	LEAD Million lbs	ZINC Million lbs
Main	Measured	7,899	52.5	0.93	0.29	13.3	162.0	50.5
	Indicated	44,196	40.7	0.70	0.39	57.8	682.0	380.0
	Meas+Ind	52,095	42.5	0.73	0.37	**71.1**	844.0	430.5
	Inferred	11,898	49.7	0.64	0.26	**19.0**	167.9	68.2
Minas	Measured	2,487	77.1	1.41	0.53	6.2	77.3	29.1
	Indicated	39,405	52.2	1.03	0.40	66.1	894.8	347.5
	Meas+Ind	41,892	53.7	1.05	0.41	**72.3**	972.1	376.6
	Inferred	20,713	47.3	0.74	0.30	**31.5**	337.9	137.0
Este	Measured	14,558	82.7	1.07	0.76	38.7	343.4	243.9
	Indicated	31,856	72.6	0.91	0.75	74.4	639.1	526.7
	Meas+Ind	46,414	75.8	0.96	0.75	**113.2**	982.5	770.6
	Inferred	5,326	55.9	0.41	0.25	**9.6**	48.1	29.4
Total	Measured	24,944	72.6	1.06	0.59	58.2	582.7	323.5
All	Indicated	115,457	53.4	0.87	0.49	198.3	2,215.9	1,254.2
Deposits	**Meas+Ind**	**140,401**	**56.9**	**0.90**	**0.51**	**256.5**	**2,798.6**	**1,577.7**
	Inferred	**37,937**	**49.3**	**0.66**	**0.28**	**60.1**	**553.9**	**234.6**

Source: (Published by Independent Mining Consultants, Inc. Based on 16 g/t Silver Cut-off Grade Contained Within an Approximate Open Pit)

Table 6-3: Historic Mineral Resource - May 2008

DEPOSIT	CATEGORY	KILOTONNES	SILVER g/t	LEAD %	ZINC %	SILVER Million ozs	LEAD Million lbs	ZINC Million lbs
Main	Measured	10,025	42.3	0.80	0.37	13.6	176.8	81.8
	Indicated	64,250	30.0	0.57	0.43	62.0	807.4	609.1
	Meas+Ind	74,275	31.7	0.60	0.42	75.6	984.2	690.9
	Inferred	11,928	33.1	0.57	0.36	12.7	149.9	94.7
Minas	Measured	6,168	53.4	1.05	0.44	10.6	142.8	59.8
	Indicated	106,970	38.2	0.75	0.38	131.4	1,768.7	896.1
	Meas+Ind	113,138	39.0	0.77	0.38	142.0	1,911.5	955.9
	Inferred	19,698	32.5	0.54	0.39	20.6	234.5	169.4
Este	Measured	20,523	63.3	0.91	0.69	41.8	411.7	312.2
	Indicated	40,485	52.0	0.75	0.57	67.7	669.4	508.7
	Meas+Ind	61,008	55.8	0.80	0.61	109.5	1,081.1	820.9
	Inferred	1,526	30.4	0.41	0.21	1.5	13.8	7.1
Total All Deposits	Measured	36,716	55.9	0.90	0.56	66.0	731.3	453.8
	Indicated	211,705	38.4	0.70	0.43	261.1	3,245.5	2,013.9
	Meas+Ind	**248,421**	**40.9**	**0.73**	**0.45**	**327.1**	**3,976.8**	**2,467.7**
	Inferred	33,152	32.6	0.54	0.37	34.8	398.2	271.2

Source: (Published by Independent Mining Consultants, Inc. Based on $9.35/t NSR Cut-off Grade Contained Within an Approximate Open Pit)

Table 6-4: Mineral Reserve and Resource - August, 2009

Mineral Reserves, $9.10 NSR cut-off									
					Contained Metal			Equivalent Ounces	
Category	Ktonnes	Silver Gm/t	Lead %	Zinc %	Silver Million Ozs	Lead Million Lbs	Zinc Million Lbs	Eq. Silver Million Ozs	Eq. Silver Gm/t
Proven	27,957	70.2	1.08	0.59	63.1	665.7	363.6	115.0	127.9
Probable	111,666	54.3	0.90	0.43	194.9	2,215.6	1,058.6	360.3	100.4
Proven + Probable	139,623	57.5	0.94	0.46	258.0	2,881.3	1,422.2	475.3	105.9

Mineral Resources in Addition to Reserves, $7.85 NSR cut-off									
					Contained Metal			Equivalent Ounces	
Category	Ktonnes	Silver Gm/t	Lead %	Zinc %	Silver Million Ozs	Lead Million Lbs	Zinc Million Lbs	Eq. Silver Million Ozs	Eq. Silver Gm/t
Measured	10,791	16.7	0.43	0.45	5.8	102.3	107.1	16.2	46.8
Indicated	99,626	20.6	0.45	0.39	66.0	988.4	856.6	158.2	49.4
Measured + Indicated	110,417	20.2	0.45	0.40	71.8	1,090.7	963.7	174.4	49.1
Inferred	34,215	32.4	0.54	0.34	35.6	407.3	256.5	69.0	62.7

Note: for this reserve resource statement silver equivalency calculation represents the contained equivalent silver ounces sent to concentrate and is based on the resource metal prices assumptions of $13.00/oz Ag, 0.70/lb Pb and 0.65/lb Zn and recoveries to concentrate of 74.5% for silver and 71.7% for lead and 71.3% for zinc. The calculation does not take into account the net smelter payment terms for the different metals in the two separate concentrates. The resulting equivalency is 1 oz Ag = 19.3 lb. Pb and 1 oz Ag = 20.9 lb. Zn.

Table 6-5 Mineral Reserve and Resource - October, 2011

Mineral Reserves, $10.54/tonne NSR Cutoff					Contained Metal		
Category	Ktonnes	Silver Gm/t	Lead %	Zinc %	Silver Million Ozs	Lead Million Lbs	Zinc Million Lbs
Proven	30,083	66.60	1.041	0.603	64.4	690.4	399.9
Probable	126,047	50.73	0.872	0.467	205.6	2,422.6	1,297.7
Proven + Probable	156,130	53.79	0.904	0.493	270.0	3,113.0	1,697.6
Mineral Resouces in Addition to Reserves							
$9.20 NSR Cutoff					Contained Metal		
Category	Ktonnes	Silver Gm/t	Lead %	Zinc %	Silver Million Ozs	Lead Million Lbs	Zinc Million Lbs
Measured	10,878	17.50	0.380	0.330	6.1	91.1	79.1
Indicated	123,583	20.80	0.380	0.290	82.6	1,035.3	790.1
Measured + Indicated	134,461	20.50	0.380	0.290	88.7	1,126.4	869.2
Inferred	49,793	30.00	0.464	0.278	48.0	509.4	305.2

Metal Prices: For Mineral Reserve: $18.00/oz Silver, $0.85/lb Lead, $0.85/lb Zinc
For Mineral Resource: $30.00/oz Silver, $1.00/lb Lead, $1.00/lb Zinc

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

Regionally, a thick section of sandstone, shale, and minor limestone has been weakly metamorphosed to predominantly quartzite and phyllitic shale. Approaching the project area from the south, the sedimentary section has been folded and faulted, with stratigraphic units locally vertical. Generally, units are relatively shallowly dipping in gentle folds. Quartzite units develop into prominent outcrops on the slopes and ridge tops, while shale units are weathered and commonly do not outcrop.

The entire sequence of sedimentary rocks of probable Paleozoic age is overlain in the higher portions of the ranges by multiple events of volcanic flows and air-fall tuffs of Tertiary age. In the southern part of the Project minor occurrences of volcanic breccias have been identified but major occurrences of intrusive rocks are rare. All units have been affected by Pleistocene glaciation forming U-shaped valleys and arêtes, typical of alpine glacial terrain. The bedrock geology in the area of the deposit is shown on Figure 7-1 which also illustrates the regional lithologic units.

7.2 LOCAL GEOLOGY

7.2.1 Lithology

The basement units in the Project area are a series of upper Paleozoic (320 million years old [Ma]) sandstone and shale units of the Grupo Ambo Formation that have been weakly metamorphosed into quartzite and phyllitic shales. The resistant quartzite units are often ridge formers. The weathered shales generally form subdued, generally slope-forming outcrops. Within the Project resource area, the sediments are generally red to gray shales.

There has been folding and faulting of this meta-sedimentary sequence due to tectonics. Within the project area the regional strike is northwest with broad northeast and southwest dips of 11 to 30 degrees.

A sequence of Tertiary (23.1Ma +/- 0.2Ma), pre-mineral volcanic tuffs unconformably overly the meta-sediments. These tuffs are generally crystal and crystal-lithic with quartz-eyes ranging up to 5 mm in diameter and range from well bedded to massive. In the upper parts of the pre-mineral sequence are andesite volcanic flows, which are generally more bedded than the underlying tuffs. This pre-mineral unit belongs to the Quenamari Formation, Chancacuniza Member.

The variations in the stratigraphic makeup of the pre-mineral tuffs and andesite flows do not appear to have any controlling effect on the mineralization. All of the resource within the Corani district is hosted in the pre-mineral tuffs and andesite flows.

Unconformably overlying the pre-mineral units, the Tertiary post-mineral tuff (10.2Ma +/- 0.1Ma), consisting of crystal tuffs identical to the lower tuffs, is effectively barren. The post-mineral sequence forms prominent spires and thickens to the north (from 0 meters to over 200 meters thick, according to project drilling). This unit belongs to the Quenamari Formation, Yapamayo Member.

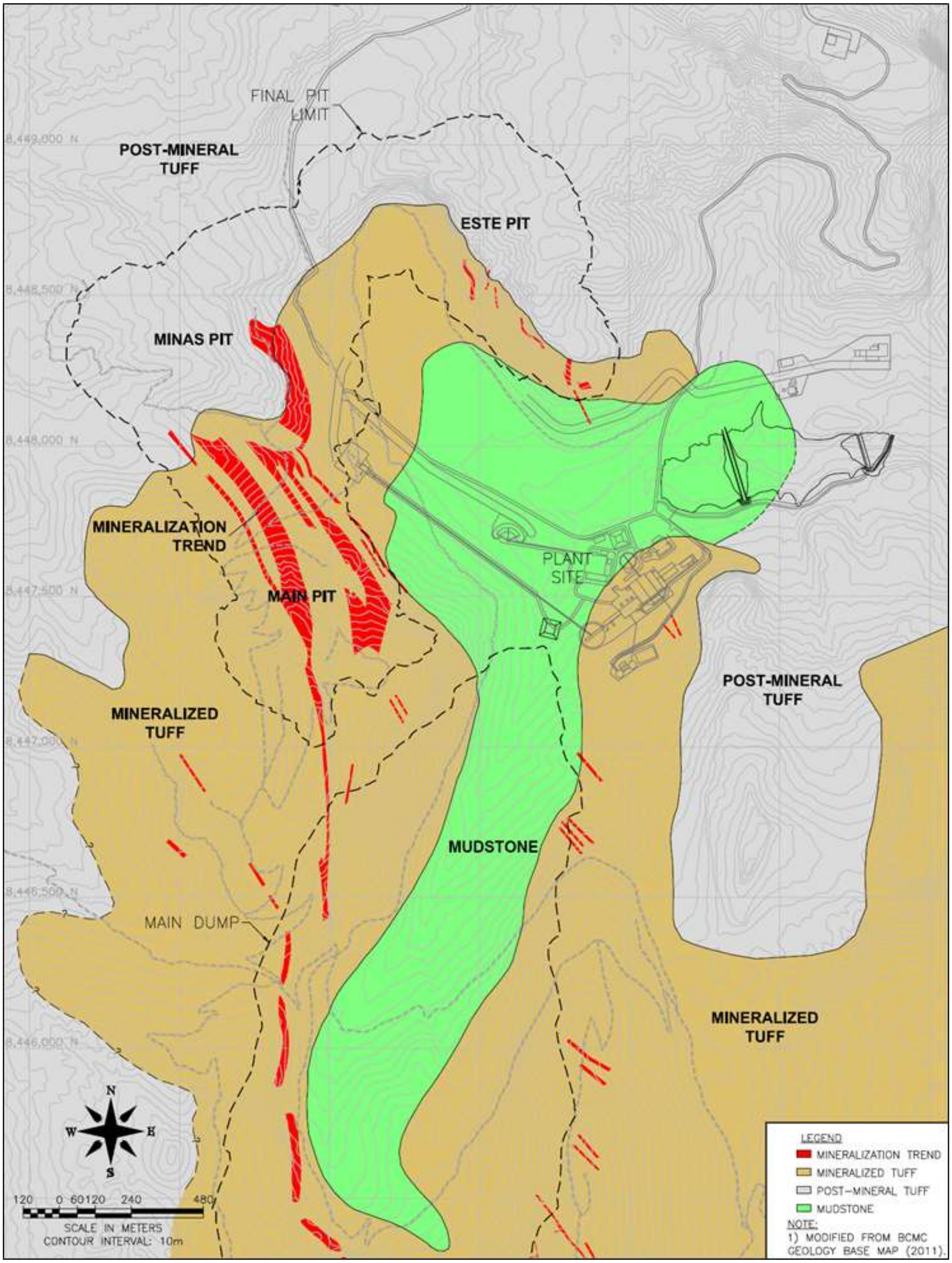


Figure 7-1: Bedrock Geology in Model Area

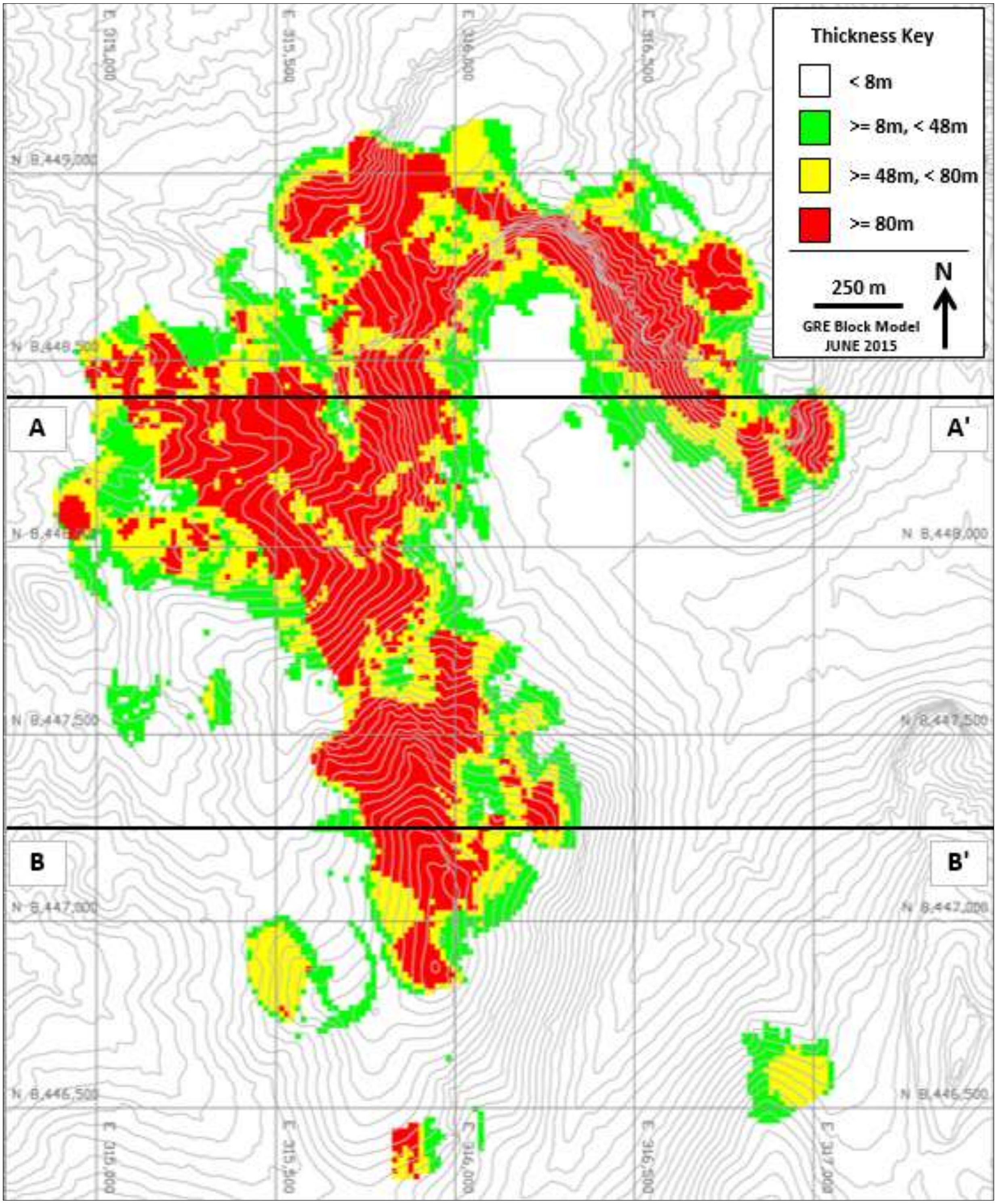


Figure 7-2: Thickness Map of Silver and Mineralization

7.2.2 Alteration

Alteration in the Project area consists of a broad (5 x 2 km) zone of illite-kaolinite alteration of the pre-mineral tuffs. More specific to the mineralization are illite, kaolinite, smectite/chlorite/celadonite and gangue minerals including massive to banded quartz, barite, chalcedony and iron and manganese oxides. Each of the three mineralized areas, namely Corani Este, Minas Corani and Main Corani, exhibit differences in alteration and gangue.

- **Corani Este:** strong barite, minor quartz and chalcedony; moderate smectite/chlorite/celadonite, brecciation, strong iron oxides and no manganese oxides;

- **Minas Corani:** strong smectite/chlorite/celadonite, moderate chalcedony and barite with strong iron oxides and moderate manganese oxides; and

- **Main Corani:** banded quartz, strong barite, iron oxides and minor manganese oxides.

Structurally, the Project area is marked by a stacked sequence of listric (concave upward) normal faults striking dominantly north to north-northwest with moderate to shallow (50º to <10º) westerly dips. The hanging walls of the listric faults are extensively fractured and brecciated, forming ideal sites for metal deposition. The stacked sequences are more prominent in Minas and Main Corani with Este showing a single listric fault with a more extensively fractured and brecciated hanging wall.

7.2.3 Mineralization

There are three main zones of mineralization in the Project area. They have been named Este, Minas, and Main, as identified above. Those zones are broadly illustrated on Figure 7-1 and Figure 7-2. The Minas and Main zones have each expanded so that mineralization is generally continuous between the two areas.

The general forms of economic mineralization in the Project area are comprised of freibergite (silver-bearing tetrahedrite), galena (not argentiferous), sphalerite (white to dark-colored), pyrite, marcasite, other silver sulfosalts (myrargyrite, pyrargyrite-proustite [ruby silver], and boulangerite), acanthite, and minor native silver. Specific mineralization styles have been identified in the Project area and each displays metallurgical characteristics that will be discussed in more detail later in this study. These styles include:

- Coarse silica-sulfide-celadonite has easily discernible galena-sphalerite locally with celadonite-tetrahedrite.

- Freibergite ores are characterized by recognizable late stage coarse-grained tetrahedrite cutting earlier sulfides and display the highest Ag contents.

- Pyrite-marcasite ± quartz mineralization typically occurs as an early stage of polymetallic mineralization, likened to the quartz-sulfide style typical of these deposits and contains little Ag-Pb-Zn mineralization but often has higher zinc values with low lead and silver content.

- Fine black silica-sulfide represents very fine grained mineralization from a rapidly cooled (quenched) ore fluid with highly variable metal contents and forms the largest portion of the mineral resources / reserves.

- Crystalline quartz-sulfide + barite is interpreted as early fault fill. Although galena-bearing, it is only well mineralized where it is cut by later tetrahedrite-bearing fractures. These veins may be transitional to the Au-bearing quartz-pyrite veins in southern Corani.

- Transitional mineralization that includes iron oxides mixed with fine black sulfide mineralization, as described above.

- Iron oxide mineralization with locally elevated Ag and typically low Pb-Zn.

- Manganese oxides which contain Ag ±Pb-Zn mineralization.

Approximately 3 km south of the Project resource area, mineralization in the Gold Zone is hosted in the same Tertiary crystal tuffs cut by a north-south striking listric normal fault with a west dip and typical quartz, barite, and iron oxide gangue as in the silver/base metal zones to the north. BCM drilled 47 diamond core holes in the Gold zone with the best intervals described as follows.

- 25m @ 4.1 g/t Au and 24.9 g/t Ag (Hole C-36A);
- 14m @ 11.7 g/t Au and 49.8 g/t Ag (Hole C-52); and
- 12m @ 4.8 g/t Au and 90.2 g/t Ag (Hole C-60B).

However, mineralization in the Gold Zone is erratic and discontinuous. Southeast of the Gold Zone another 1.5 km is an area of quartz-stibnite mineralization in discrete, narrow, veins, and breccias zones referred to as the Antimony Zone. BCM has drilled condemnation and additional exploration drill holes in the southern part (5 km from the current ore body) and it has encountered mineralization. BCM drilled two holes in the Antimony zone and neither hole had a significant mineralization. Previous Technical Reports have described the Gold and Antimony Zones in more detail (Nelson, 2006; Corbett 2007). Neither the Gold nor Antimony Zones are included in the resources or in the operational plan of the Feasibility Study.

7.2.4 Veining

The veining is found in the 23 Ma mineralized tuff package. The mineralized veins occur in two forms; in large veins associated with the predominant listric fault structures and in stockwork veins found in the surrounding rocks. The larger principle veins are generally rich in quartz / silica and barite and contain typical higher lead-zinc-silver mineralization and can be several meters in width and are quite continuous along strike and down dip. The stockwork veins occur in the wall rock between the faults and also contain significant lead-zinc-silver mineralization. The stockwork veins tend to be mineralized with fine black sulfide mineralization, indicating rapid emplacement and quenching of the mineralizing fluids. The stockwork veins tend to be randomly oriented suggesting that vein formation was controlled by fluid movement into the rock mass that was broken or weakened by the regional tectonic activity forming the listric faults.

7.2.5 Structure

The structure of the rock units varies greatly between the three principal rock types. The basement sediments are massive in structure and have little jointing. The sedimentary rock in the area of the ore zone is a strongly altered siltstone that still shows remnant bedding but the rock has been strongly altered and considerable pyrite mineralization has occurred in this rock. From the standpoint of lead-zinc-silver mineralization, the rock is virtually barren.

The structure in the mineralized tuff mirrors the description of the veining discussed in the previous section. The principal listric faults have been interpreted in drill core to occur approximately every 150 m and are continuous along strike through the deposit.

In between the principal faults, the rock mass has been broken and weakened, forming a random orientation of structures. Since the principal vein-forming faults are listric faults, they are curved and flatten down-dip, becoming horizontal. This horizontal coalescing of the principle veins forms a well-defined bottom to the mineralization in the mineralized tuff unit. Above the “bottom-forming” fault the rock is hydrothermally altered and moderately to strongly deformed. The rock below the “bottom-forming” fault appears very fresh, relatively unaltered, and undeformed. This “bottom-forming” fault is always within the mineralized tuff unit and is found above the sediment mineralized tuff contact. The thickest veins and best mineralization occurs in the listric faults when they have a steeper dip. At the bottom of the listric faults, where they all join together along the “bottom” fault, there appears to have been very little dilation so the mineralization is weak.

The 10 Ma Post-Mineral Tuff (PMT) was deposited after the tectonics of the listric faults and the mineralizing event. The rock is very fresh and completely unaltered. The rocks have a vertical joint pattern which causes the formation of rock towers and steep cliffs. This rock unit and the characteristic rock tower formation extends for tens of kilometers from the site.

7.3 MINERALIZATION

The Project resource is comprised of low to intermediate sulfidation epithermal silver, lead, and zinc mineralization within stockworks, veins, and breccias. The Main, Minas, and Este areas are generally structurally controlled along a general north-northwest strike. The Este area is limited by the overlying post mineral tuff.

Figure 7-2 is a map illustrating the thickness of the silver mineralization within the Project resource at an approximate cut-off grade of 15 g/t silver. The strike length of the silver mineralization for each area is roughly:

- Main and Minas combined 2 km; and
- Este 1.5 km.

Figure 7-3 presents cross sections through the silver values across the deposits on two sections indicating the general thickness and dip of the mineralized zone. All zones of mineralization appear to be hosted by westward dipping listric structures with vertical components to the mineralization.

The abrupt boundary on the top of the Este mineralization reflects the unconformable contact with the post-mineral tuffs. That upper limit causes the Este deposit to appear to have a more horizontal component than the other deposits.

7.3.1 Mineralogy of Economic Metals

Mineral test work by Hazen Research, Inc (Hazen, 2006) has identified the most abundant silver bearing mineral as fine grained argentian tetrahedrite, also called freibergite. Other minor sources of silver are acanthite and one or more members of the lead-silver sulfosalt group such as adorite and diaphorite. Boulangerite and galena do not appear to be a significant source of silver.

The primary sulfides are pyrite-marcasite, boulangerite (a lead antimony sulfosalt), sphalerite (high and low Fe component), and galena. Zinc mineralization in the form of sphalerite may or may not be colocated with silver mineralization. There are zones, particularly in the Minas area, where zinc mineralization is outside of the silver mineralization. The majority of the lead occurs as galena but lead can also occur as plumbogummite, a lead aluminum phosphate, which has a lower flotation recovery than lead sulfide minerals. Mineralization in surface outcrops, and drill core is generally associated with iron and manganese oxides, barite, and silica. Silicification is both pervasive and structurally controlled in veins. Figure 7-4 and Figure 7-5 illustrate grade x thickness product maps of lead and zinc, respectively. These are provided to indicate the general location of these metals relative to the location of silver.

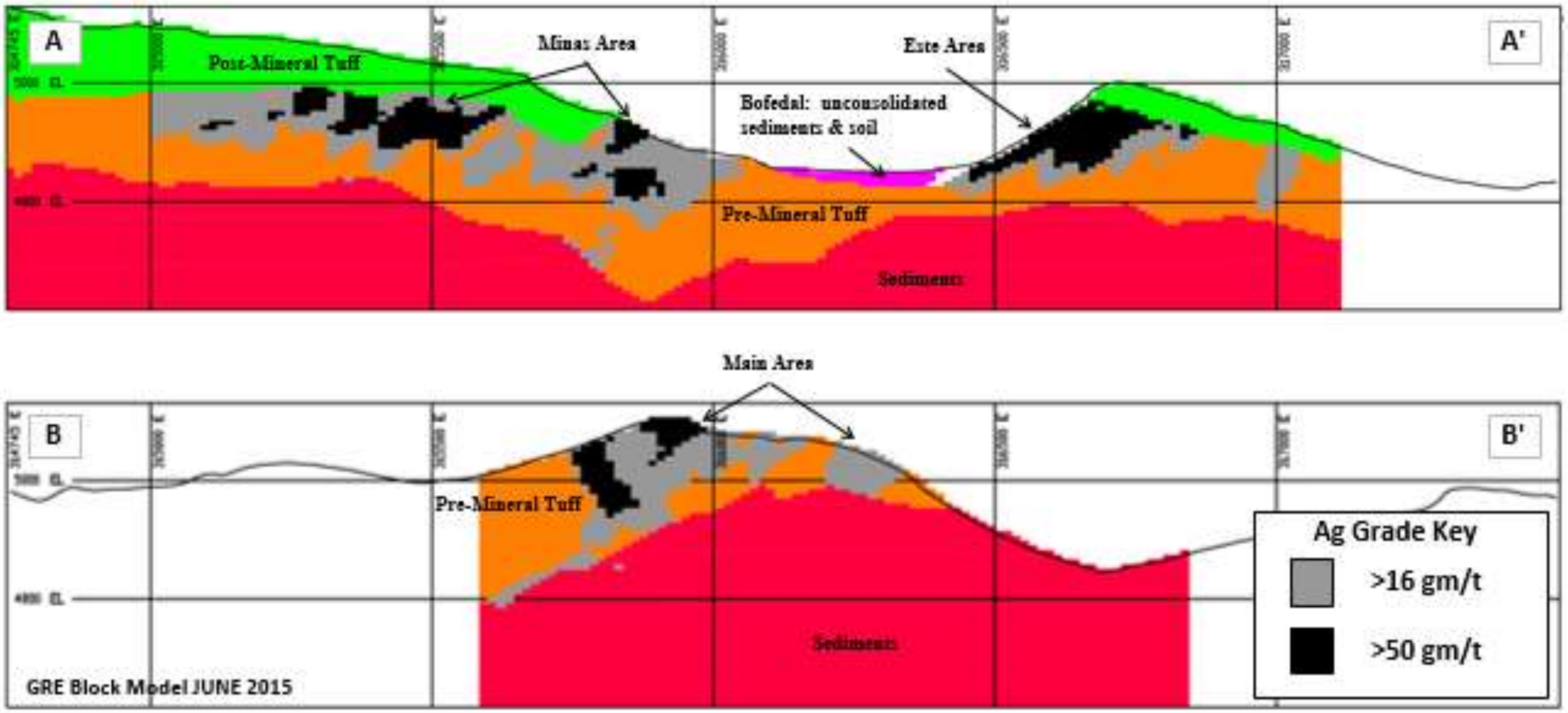


Figure 7-3: East-West Cross Sections Looking North

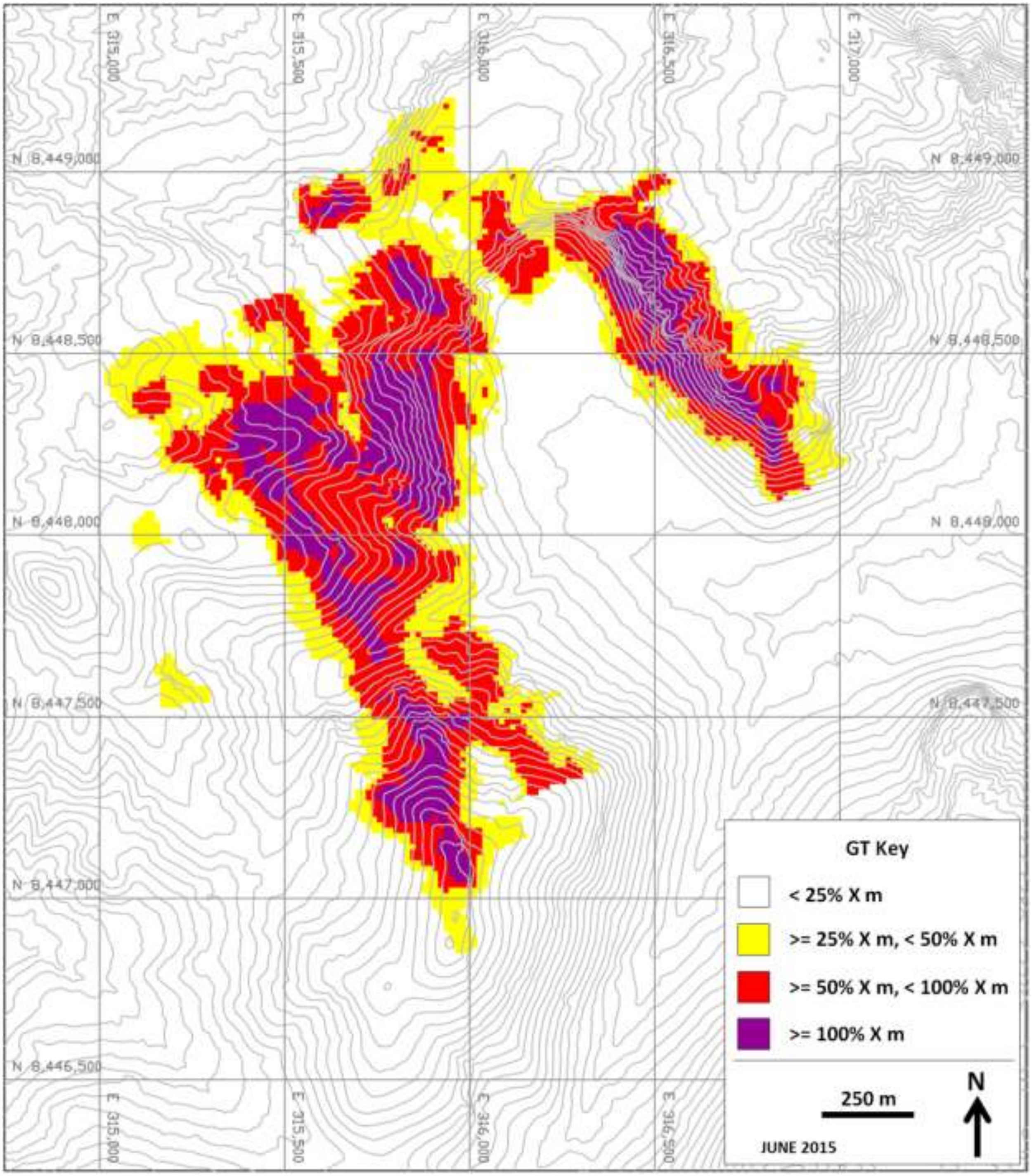


Figure 7-4: Grade Thickness Contours for Lead at 25,50, 100% X Meters

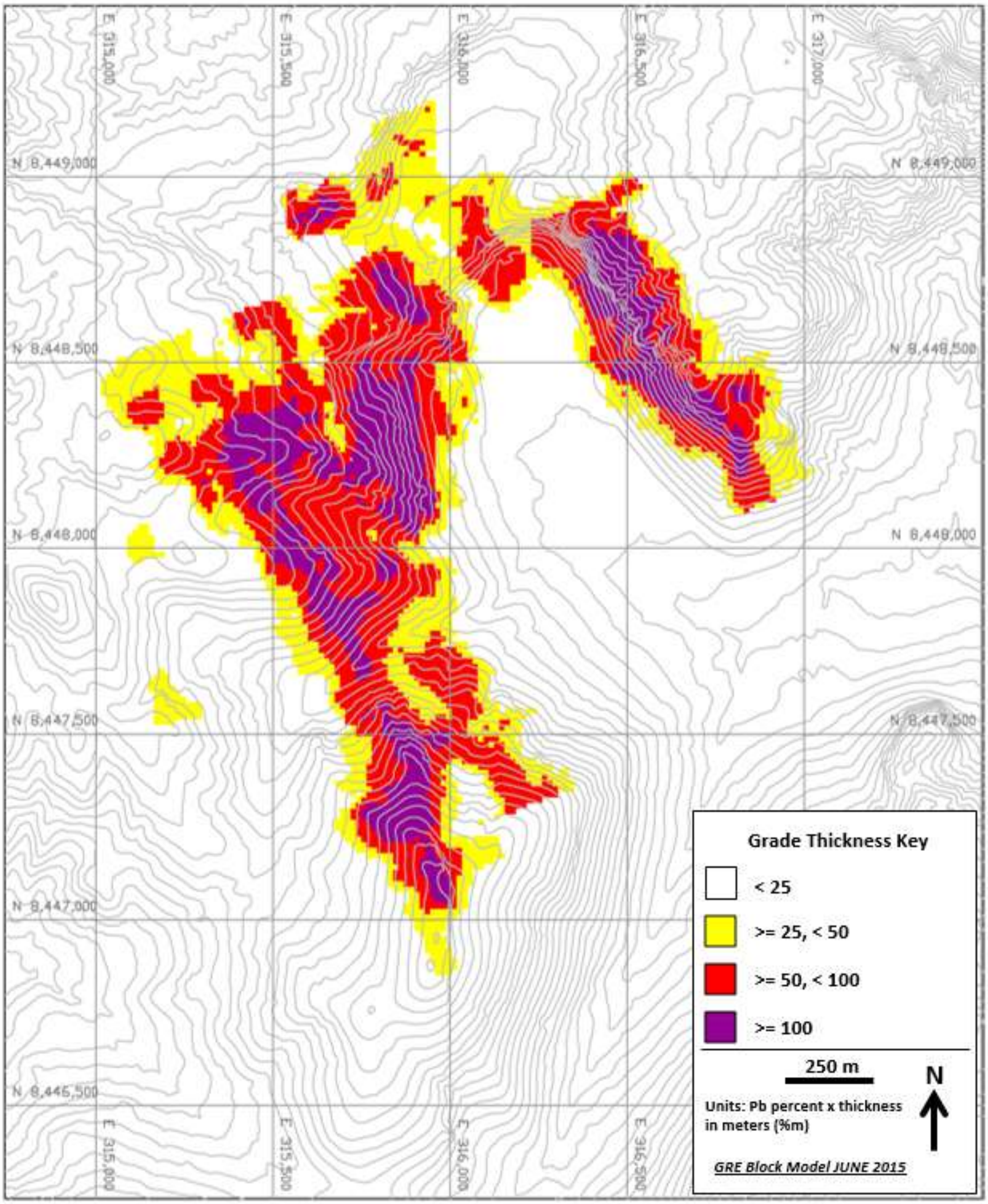


Figure 7-5: Grade Thickness Contours for Zinc at 10, 30, 100% X Meters

7.3.2 Geometallurgy

Discrete mineral domains were used to assign metallurgical performance in the previous feasibility study (M3, 2011). These domains were defined by BCM geologists based on review of the drill core and development of cross-sections of the deposit using professional judgement. From the cross-sections, solids of the domains were created and used to assign metallurgical parameters to the block model. While the current geometallurgical approach supercedes this methodolgy for predicting metal recoveries; a discussion of these domains is still relevant to the understanding of the Corani deposit mineral zonation and metallurgical responses.

Previously, nine mineral types were identified for the Corani deposit. These mineral types were grouped into four metallurgical categories, and each category was assigned an average recovery or a grade-dependent recovery. Description of the mineralization types applied by the BCM geologists that were previously used to predict metallurgical recovery are as follows:

- CSC – Coarse-grained silica-sulfide-celadonite characterized by readily discernible sulfides (galena-sphalerite-chalcopyrite ± tetrahedrite) with celadonite in crystalline to locally opaline quartz with good Ag-Pb-Zn recoveries.
- CS - A subset of CSC that contains coarse galena-sphalerite-chalcopyrite ± tetrahedrite without green celadonite clay.
- TET – Ag-bearing tetrahedrite characterized by recognizable late-stage, coarse-grained tetrahedrite cutting earlier sulfides and displaying the highest Ag contents and best Ag recoveries by flotation or leach: typically with low Pb-Zn content.
- PM – Pyrite-marcasite ± quartz typical of low temperature early-stage mineralization with little polymetallic mineralization, mainly Zn.
- FBS – Fine-grained black silica-sulfides characterized by very fine-grained mineralogy deposited from quenched ore fluids with highly variable metal content and generally poor leach recoveries and good flotation recovery with some challenges in separation.
- QSB - Crystalline quartz-sulfide-barite interpreted as early fault fill or late-stage breccia fill.
- PG – Plumbogummite, identified as a pale-green, waxy, Pb-phosphate mineral that in metallurgical test results shows diminished lead flotation and difficulties in separation of base metals.
- FeO – Iron-oxide mineralization with locally elevated Ag and generally low Pb-Zn. This is a gradational zone with mixtures of FeO and FBS and the most strongly oxidized areas shows high Ag leach recovery results; and

MnO – Manganese-oxide mineralization hosting mainly Ag with lesser Pb-Zn with very poor response to flotation and leach tests.

Though these mineralization types generally indicated metallurgical performance, the behavior for samples within each metallurgical type was highly variable, particularly within the mixed oxide/sulfide types (FBS).

Detailed analysis of the mineralogical test data suggested that these mineralogical domains did not fully capture the transitional nature of the Corani deposit. New analysis of the metallurgical, QEMSCAN, and geologic data (presented in Section 13) indicated that much of the Corani ore is transitional material that ranges from slightly to strongly oxidized.

Since metallurgical recovery is strongly related to the degree of oxidation, it was necessary to develop a new geometallurgical model that could represent variations in oxidation throughout the ore body, which are highly variable and inconsistent both vertically and horizontally. A new geometallurgical model was developed that utilizes mineralogy and other factors to provide more accurate predictions of metallurgical recovery. Those factors include elevation, zinc grade, and logged geologic data, especially key minerals such as galena, pyrite, goethite, and manganese oxides. These parameters serve as indicators for the degree of oxidation that has occurred and allow for a continuous estimate of metallurgical parameters. The new model provides a more accurate representation of the deposit's metallurgical response, particularly for the transitional material because it has the ability to represent the gradational boundary between sulfide and oxide material indicated by mineralogical data.

Figure 7-6 and Figure 7-7 are example cross sections of anticipated recovery based on the previous "ore type" geometallurgical model and the new continuous geometallurgical model. The new model greatly improves the ability to resolve variations in recoveries throughout the deposit volume and is a more realistic representation of the transitional material. The detailed development of the new geometallurgical model is presented in Section 13.

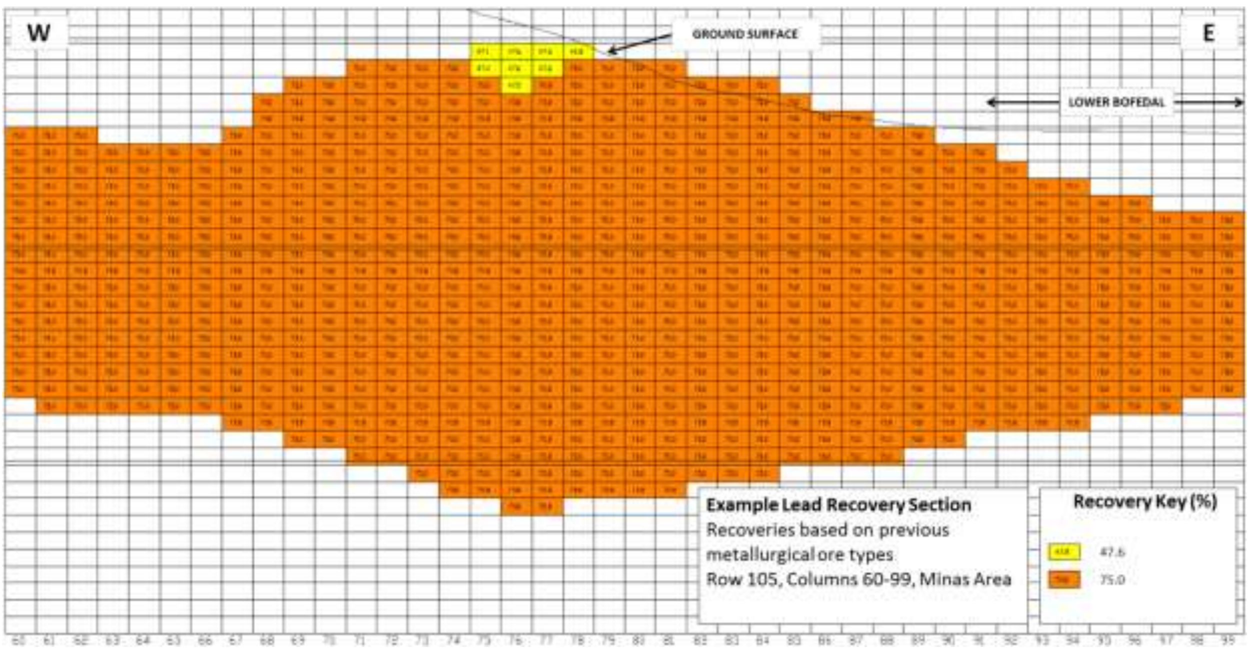


Figure 7-6: Lead Recovery, Previous Metallurgical Ore Types

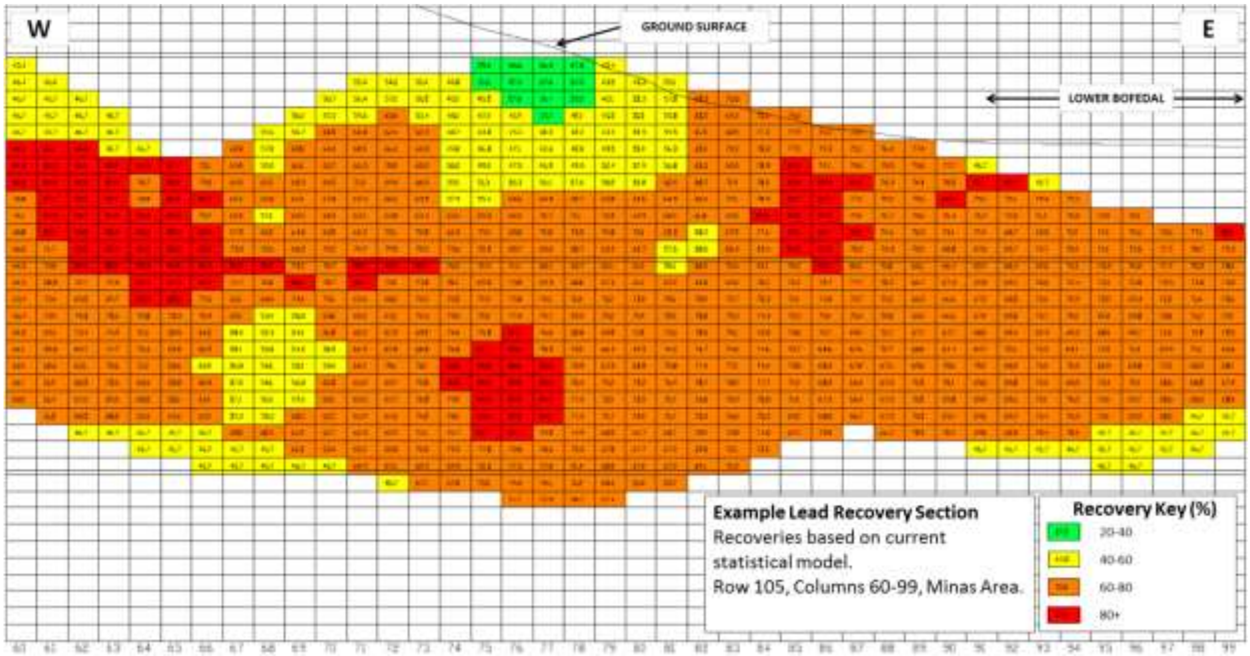


Figure 7-7: Lead Recovery, New Geometallurgical Model

8 DEPOSIT TYPES

The Project's mineralization is a low to intermediate sulfidation epithermal system with silver, lead, and zinc deposits hosted in stock works, breccias veins, and fractures. The Gold zone to the south is a low sulfidation epithermal gold occurrence in association with silica. The Antimony zone is comprised of stibnite-pyrite veins with silica. There is also sulfide mineralization in the sediments that are essentially barren of silver, and lead.

The above combinations are indicative of the epithermal mineralization that is sometimes associated with distal zoning around a porphyritic intrusion.

The Project resource within the Main, Minas, and Este zones are comprised of the low to intermediate sulfidation, silver, lead, and zinc mineralization. The Main and Minas areas are more associated with vein structures and the Este zone appears to be a broader zone of veinlets and stock works.

The polymetallic Ag-Pb-Zn mineralization is typical of that developed at an elevated crustal setting by rapid cooling of a hot hydrothermal fluid, derived dominantly from an intrusion source, which has come in contact with cool wall rocks and remains at an unknown depth and uncertain metal grade. The important aspect of Corani is that the dilational listric faults (discussed in Section 7) focused substantial intrusion-derived sulfide ore fluids which were rapidly cooled to provide economic Ag - polymetallic grades.

9 EXPLORATION

BCM has been conducting exploration within the Project area since early 2005. The BCM work has included detailed mapping, hand trenching with channel sampling, and core drilling.

9.1 EXPLORATION AND DRILLING PROGRAM

There are a total of 25 trenches completed within the Project resource area with 1,295 assayed intervals totaling 2,924 meters of trench data. There are sixteen additional trenches in the Gold Zone to the south. Generally, trenches results have economic or highly anomalous grade in both areas.

Diamond core drilling started in June 2005 and continued through February 2012. The total Corani District diamond drilling completed by BCM has been 556 holes representing 100,494.5 meters of drilling. Samples were sent to ALS-Chemex assay lab in Lima for preparation and assay. The database has been maintained by the BCM staff in the Lima office. Core rejects and pulps are being kept at Lima warehouse.

9.2 ACTIVITIES PLANNED TO EXPAND MINERALIZED ZONES AND EXPLORE PROSPECTS

The area immediately surrounding the FS mine plan Pit has been explored and tested using many drill holes and the extent and grade of the mineralization is well known. As such, BCM plans to drill more infill holes in the area of the early operation years to convert more of the reserves from probable to the proven category. Additionally, the future plan calls for recovering more samples for detailed metallurgical testing which will concentrate on developing specific composites of the material to be mined and processed during the early years of operation. Additionally, the metallurgical test work will focus on the correlation of mineralogy and metallurgical recovery.

The areas away from the principal Project ore zones is still open for exploration and BCM plans on engaging in a broader regional exploration program that will utilize geological prospecting and some geophysical and geochemical techniques in the hopes of defining future targets. In particular, the Gold Zone has received favorable initial drilling; however, additional drill holes and metallurgical testing is required in order to calculate a resource.

10 DRILLING

Drilling on the Project was started in June of 2005 and has been under the control of BCM since that time. All drilling to date has been by diamond core methods producing HQ core 6.36 cm (2.5 inches) and NQ core 4.76 cm (1.9 inches) in diameter. The block model for this study was based on the drill hole data that was complete by February of 2012.

The typical drill pattern consists of a series of drill fans on section lines spaced 50 m apart. The fans are arranged perpendicular to the strike of the individual deposits. Angled holes are used in an attempt to drill perpendicular to the general structural orientation of the deposits. Multiple holes are often drilled from one site in order to reduce surface impact and obtain the necessary drill coverage (sample spacing) at depth. Some zones have been in-filled to 25 m spacing and other lower grade portions of the deposits are still on 100 m spacing.

Since the initial resource estimate (2006), BCM has undertaken to in-fill the areas of mineralization and increase the confidence in the resource estimate. This has been accomplished by in-fill drilling the wider zones to the 50 m spacing, with a focus on areas of higher grade mineralization. A brief review of the grade and thickness maps in Section 7 indicates that each of the three deposits are still open in some areas.

Figure 10-1 presents a drill hole location map of the available Project data as of February 2012. This illustrates the information used in the development of this estimate of mineral reserves and resources.

The diamond drill hole data within the Project database as of February 2012was as follows:

- Number of drill holes 531
- Number of sample intervals 38,111
- Total meters of drilling 92,741.7 meters
- Number of silver assays 36,996
- Number of lead assays 36,996
- Number of zinc assays 36,996
- Number of copper assays 36,996

In addition to the diamond drilling data, the following trench data at the Project was used in the model.

- Number of trenches 25
- Number of sample intervals 1,297
- Total meters of trench data 2,923.8 meters
- Number of silver assays 1,295
- Number of lead assays 1,295
- Number of zinc assays Not assayed
- Number of copper assays Not assayed

Not all of the 556 drill holes are in the immediate area of the Project deposit. The number of drill holes that penetrate the block model are:

Drill Hole Data in Block Model

- Number of drill holes 470
- Number of sample intervals 34,443
- Total meters of drilling 85,211.8 meters
- Number of silver assays 33,442
- Number of lead assays 33,442

- Number of zinc assays 33,442
- Number of copper assays 33,442

Drill hole collars were originally surveyed by hand held GPS with a reported accuracy of ±3.0 meters. All of the drill hole collars resurveyed by conventional survey with substantially higher precision since 2008. Comparison of the new collar elevations with the surface topography map indicates that about 15% of the collar elevations differ from the topographic elevation by more than 5 m. About two-thirds of these occur in steep topography where the collar elevation is below topography resulting from the cut required to establish the drill pad.

Prior to late 2007, downhole surveys were not conducted on the drill holes at the Project. Since late 2007, a series of 12 relatively deep holes that targeted deep high grade zones within the Pit were drilled to confirm the presence and location of the grade intercepts. Downhole surveys were conducted on these 12 drill holes. The average depth of the 12 surveyed holes was 220 meters. Comparison of the downhole survey location of the bottom of the drill hole with the location projected from the collar survey, the average error for all 12 holes was 4.89 meters. The maximum of all errors was 11.13 meters. In the worst case, the drillhole location without survey would have been within one model block.

The indication from the 12 surveyed holes is that the lack of downhole surveys for holes of 200 m or less would have no major impact on the development of the block model. However, as additional drilling is completed precision surface surveys and downhole surveys should be implemented for all future drilling.

The last twelve exploration holes drilled between December 2011 and January 2012 were also used for grindability testing. Results show that the ore appears to be of medium hardness with respect to SAG and ball milling (see more details in Section 13.4).

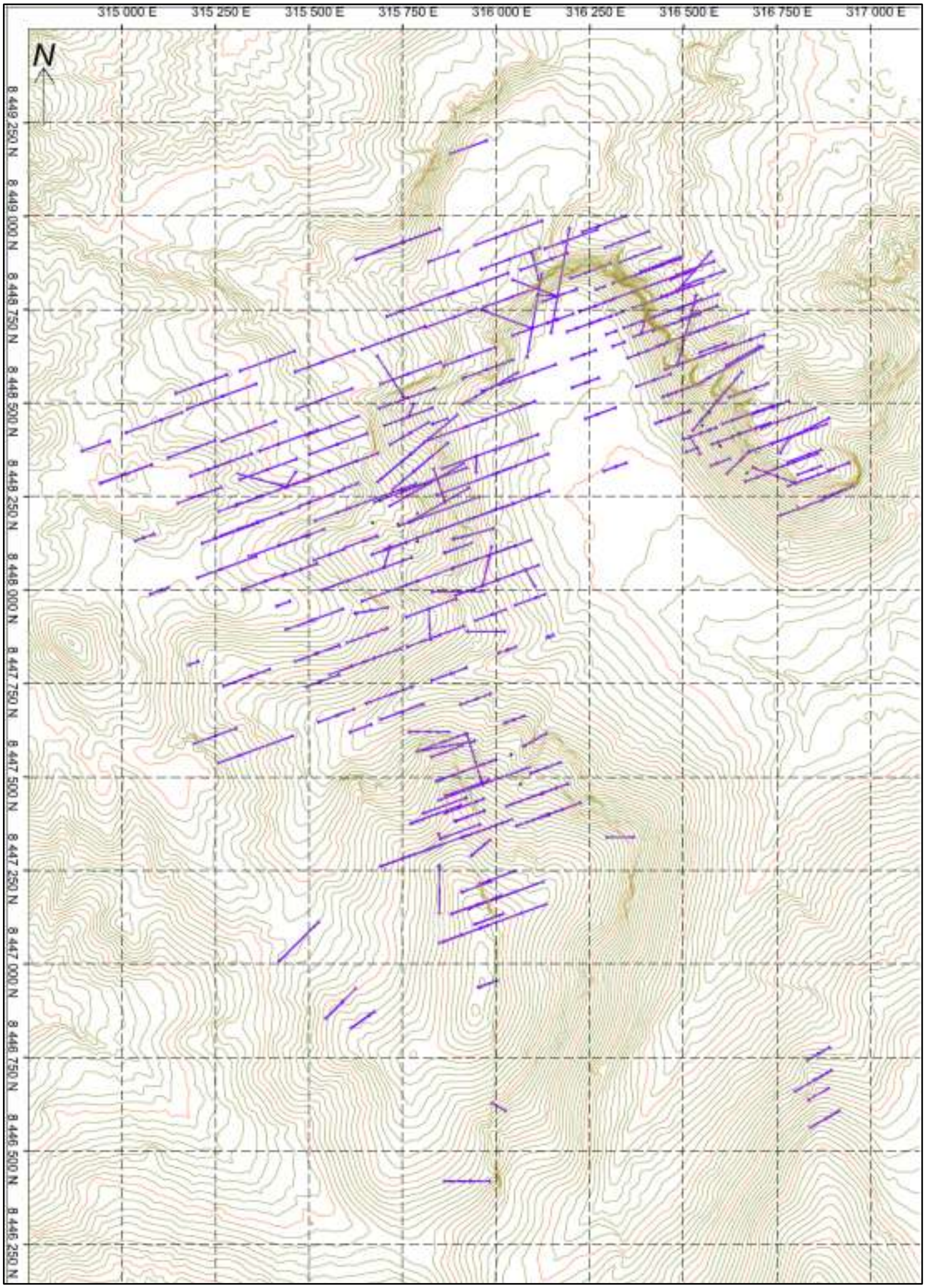


Figure 10-1: Drill Hole Location Map

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 SAMPLING METHOD AND APPROACH

Two types of samples are collected at the Project: 1) diamond core samples, and 2) trench channel samples. The sample preparation of each data type is summarized in this section.

11.1.1 Diamond Core Sample Collection

Diamond drilling is completed using conventional wireline practices and HQ / NQ core. The whole core is boxed into plastic and weatherproof cardboard boxes at the drill rig by the drill crew. Christian Rios, acting as qualified person observed the procedures outlined below between 2005 and 2012. The following list summarizes the drill core handling process:

- Whole HQ or NQ core is boxed at the drill site;
- Core is transported by vehicle to the Project camp where the core preparation facilities are located;
- Digital photos are taken of all the core as received from the drill;
- Basic geotechnical logging of RQD and core recovery is also completed on whole core. Pre-logs and logs are recorded by hand on paper for later input to a database;
- Assay sample intervals are generally 2.0 meters long. Some long intervals of barren units (post-mineral volcanic and basil sedimentary rocks) are not sampled;
- Core is split on site using conventional core splitters. Half core is returned to the box;
- Geologic logging is completed on the half core in the core boxes;
- The other half of core is bagged and tagged with a sample number. A sample number tag is included in the sample bag. Sample numbers are assigned by BCM personnel. All future handling and tracking of the sample is by the blind sample number; and
- Bagged samples are transported via truck to Cusco or Juliaca by BCM employees for bus shipment to ALS-Chemex labs in Arequipa, Peru where the samples are crushed and a pulp is prepared and shipped to the ALS-Chemex lab in Lima, Peru, where the chemical analyses are performed. Throughout this transportation process, proper chain-of-custody procedures are maintained.

ALS-Chemex picks up the samples at the bus station in Arequipa, Peru and logs them into their system by sample number for preparation and analysis.

11.1.2 Trench Sample Collection

The following procedures for trench sample collection were observed in the field:

- The trench is hand dug by local workers to remove surface alluvium and establish a clean bedrock surface. Trenches are typically 0.5 to 1 m wide and between 0.5 and 1 m deep;
- The floor of the trench is cleaned;

- A channel sample is collected from the trench using a hammer and moil point chisel;
- Sample intervals are 2 m horizontal intervals, based on GPS survey of the start and end point and a measuring tape;
- The bearing and dip of the trench are also recorded; and
- The 2 m sample intervals are "draped" on topography later in the data recording process. Consequently, some intervals on steep hill sides are actually longer than 2 m.

11.1.3 Density Data

Over 1,100 density determination were performed on rock using a waxed core method. Samples were chosen out of every 5th core box which resulted in a sample spacing of approximately 15 m.

GRE anyalized the density data to determine if a relationship could be found between density results and various parameters including assay results, mineralization type, and deposit area. Statistical analysis indicated that the best predictor of density was combined silver, lead, and zinc grade. The silver was converted to a percent assay to get common units with lead and zinc. The analysis showed that a combined grade of 0.9381% metal best defined high and low densities. Sampes with a combined grade under 0.9381% had an average density of 2.31 t/m^3, and samples with a combined grade greater than or equal to 0.9381% averaged 2.43 t/m^3. Post-mineral tuff with no metal grades had an average density of 2.3 t/m^3 and non-tuff material had an average density of 2.53 t/m^3. Table 11-1 shows the new density values.

This data was reviewed by the QP and the in situ densities used in the resource estimate.

Table 11-1: 2015 Updated Densities

Rock Type	Grade	Density
Pre-Mineral Tuff	< 0.9381	2.31
Pre-Mineral Tuff	>= 0.9381	2.43
Post-Mineral Tuff	Not Applicable	2.3
Other Materials	Not Applicable	2.53

11.2 SAMPLE PREPARATION, ANALYSES AND SECURITY

Core samples are received at ALS-Chemex labs where they are logged into their system for preparation and analysis. Each drill core sample is about 5 to 8 kg of half core as sent directly from the Project site.

11.2.1 Sample Preparation

The sample preparation procedures as applied by ALS-Chemex in Lima were as follows:

- The sample is dried at 110-120 degrees C;
- The entire sample is crushed by jaw and roll crusher to 70% passing 2 mm (about 10mesh);
- A 250 g subsample is obtained using a riffle splitter. Coarse rejects are returned to BCM; and

- The split is pulverized using a ring-and-puck pulverizer to 85% passing 75 micron.

The above methods correlate with the published ALS-Chemex preparation code of PREP-31.

11.2.2 Assay Procedures

The assay procedure for silver for the Project is as follows:

- A sample of the pulp is digested with three acids: hydrofluoric, nitric, and perchloric. This results in a cake;
- The remaining cake is leached with hydrochloric acid; and
- The hydrochloric acid solution is subjected to atomic absorption spectrophotometry (AA) to determine the concentration of dissolved silver.

The ALS-Chemex method code is AA62 for the above assay. The procedure is reported to be robust over the reported range of 1 to 1,500 g/t silver. The samples are also assayed for lead, zinc, and copper by three acid digestion followed by AA analysis. A few gold assays have been completed but they were not utilized or incorporated into the block model or resource calculations.

Sample chain of custody and security is handled by BCM in that drilling, logging, splitting and transport to a private transport carrier in Cusco or Juliaca, are all under the control of BCM personnel. Once the samples have been sent to ALS-Chemex, the sample is tracked by a blind sample number assigned by and recorded by BCM personnel.

Additional multi-element inductively coupled plasma (ICP) analysis is conducted on selected intervals at the Project site. This information is used for both exploration guidance and for process trace element information. The multi-element ICP analytical values are not used in the calculation of the block model, or mine plan. They are used to assist in the interpretation of the mineral ore types and to guide the interpretation of metallurgical process response.

11.3 CONCLUSION

Christian Rios acting as qualified person has concluded that the data collection, preparation, and analysis procedures are adequate for the preparation of mineral resources and mineral reserves. This opinion is also based on the work outlined in Section 12 regarding Data Verification.

12 DATA VERIFICATION

This section reports on the results of QA/QC procedures in place at Corani, as well as independent checks completed by Independent Mining Consultants (IMC) in 2011. The procedures that are addressed in this section are:

1. Standards Assays: Standards are inserted in the assay stream on a 1 in 20 basis.

2. Check Assays: Pulps are sent to a second lab for silver check assay on about a 1 in 25 basis.

3. IMC check on the database by comparing the certificates of assay with the values stored in the electronic database.

4. Trenches versus Diamond Drilling: IMC completed a nearest neighbor comparison in 2011.

The Bear Creek data verification process was originally focused primarily on silver. During 2005 changes were made to add check assays for lead and zinc to the silver check assays. Each of the data verification process will be discussed in the following sub-sections.

As a result of this verification work, Christian Rios (the qualified person) finds that this database is acceptable for the estimation of mineral resources and mineral reserves.

12.1 STANDARDS

Bear Creek geologists insert standards on 1 in 20 basis into the stream of samples being sent to the ALS-Chemex assay lab in Lima. The standards are prepared by outside commercial labs (ROCKLABS, others). The standards are pulp samples and the incoming core samples are half core, so the lab knows which samples are standards or blanks. However, the lab does not know which of the 8 different lab standards that have been used by Corani have been inserted.

Bear Creek personnel review the standards results periodically. If the lab reported value of the standard is more than 10% different from the certified value of the standard, the entire assay run associated with that standard is submitted for reassay.

As of March 2008, there had been 20 sample batches submitted for reassay due to out of tolerance response of the associated standard. The author uses the standards to check on assay bias, and sample handling procedures.

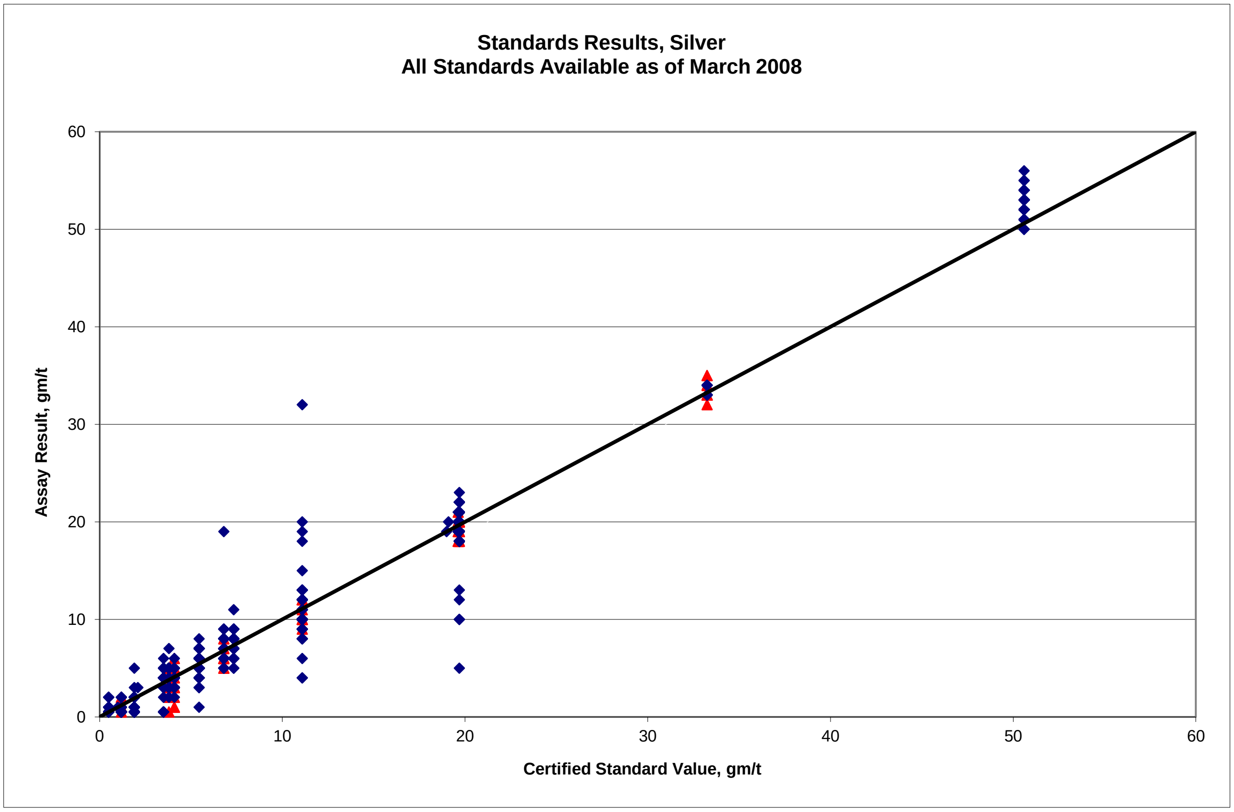
Figure 12-1 is an X-Y plot of the certified value of each standard versus the multiple reports of each standard available from ALS-Chemex. The line on the graph represents a 45 degree line or the ideal result from all values. The statistical results of the ALS-Chemex silver assays are summarized on the bottom of the table compared with the accepted value of the standard.

The table at the bottom of the figure indicates that the ALS-Chemex lab tends to undervalue the low grade silver standards. A number of standards in the 1.2 to 1.9 gm/t range are reported back as trace by ALS. This is of no impact on reserves as the cutoff grades are significantly above these values.

It is also interesting to note that the only 3 or 4 of the standards are in grade ranges that are of economic interest to the project: SG_14 at 11.1 gm/t, SI_15 at 19.7 gm/t, SI_25 at 33.25 gm/t, and 51714 at 50.6 gm/t. The average head grade of mill feed during the project life is around 52 gm/tonne with cutoffs in the range of 10 to 20 gm/t depending on the associated base metal credits. One could make an argument that a higher percentage of inserted standards should be in the range of interesting silver grades at Corani.

Figure 12-1 indicates several points that are outside of the cluster of the data for that standard. A few such points are indicated by arrow on the graph. These points are typically indications of sample swaps. For example, the low value for the SI_15 sample that averages 19.7 gm/tonne is likely a member of the standard SG_14 where it would report in the middle of the cluster.

A quick count from the graph indicates that there are probably about 0.5% sample swapping in the standards database. It is not known if this is a function of improper sample insertion, (the likely cause), or assay and database reporting errors elsewhere in the system.



Silver Standards Results,					
Standard Label	Number of Assays	Standard Value gm/t	Average Assay gm/t	Stan Dev of Assay gm/t	
51714	89	50.600	52.333	1.199	Starndards 2005 through Sept 2007
ROCK LABS SI_15	356	19.700	19.542	1.569	
ROCK LABS SG_14	370	11.100	10.800	1.696	
31811	140	7.350	7.525	0.743	
31812	158	5.450	5.491	0.958	
CIMM-2	110	3.800	3.802	0.717	
Unknown	47	3.500	3.500	1.452	
10687	92	1.900	1.054	0.709	
ROCK LABS SF_12	66	0.500	0.890	0.333	
ROCK LABS SI_25	12	33.250	33.500	0.522	
ROCK LABS SN_26	10	19.630	19.500	0.707	
CIMM-1	116	6.800	7.034	1.426	
CIMM_24	47	4.100	3.915	0.855	
CIMM_23	92	1.200	0.602	0.326	
CIMM_23	8	1.200	1.188	0.530	Standards Nov 2007 Through Mar 2008
CIMM_24	10	4.100	3.500	1.780	
CIMM-1	10	6.800	6.400	1.075	
CIMM-2	8	3.800	3.438	1.545	
ROCK LABS SG_14	8	11.100	10.750	1.165	
ROCK LABS SI_15	13	19.700	19.462	0.776	
ROCK LABS SI_25	10	33.250	33.600	0.966	
ROCK LABS SN_26	9	19.630	19.556	1.014	

june09/qaqc/JMM_Mod_11Sep-9_Standards-IMC28Dec07.xls

Figure 12-1: Corani Project Standards Results for Silver

12.2 CHECK ASSAYS

Check assay pulps are submitted to a second lab on a roughly 1 in 25 basis. The initial check assay protocols were established during 2005 with silver assay checks only. The procedures were amended in late 2005 to include check assays of silver, lead, and zinc.

The initial checks during 2005 are summarized on Figure 12-2. During this period, check assay pulps were submitted to SGS labs in Lima. There is a substantial variability in the check assay results during this period. There is no indication of bias in the data set, but there is substantial variability between the original and the check assays. The cause of the variability is not known. There is some potential that the check pulps have been swapped or mislabeled when shipped to the check lab.

Figure 12-3 through Figure 12-5 summarize the results of outside checks from late 2005 through March 2008. . The same trend is apparent regarding the high degree of scatter with the Inspectorate checks as with the SGS checks before them. The variability occurs in all three metals, silver, lead, and zinc. Although a degree of scatter is typical for the precious metal assays, the variability of lead and zinc are unusual for base metal check assays on pulps.

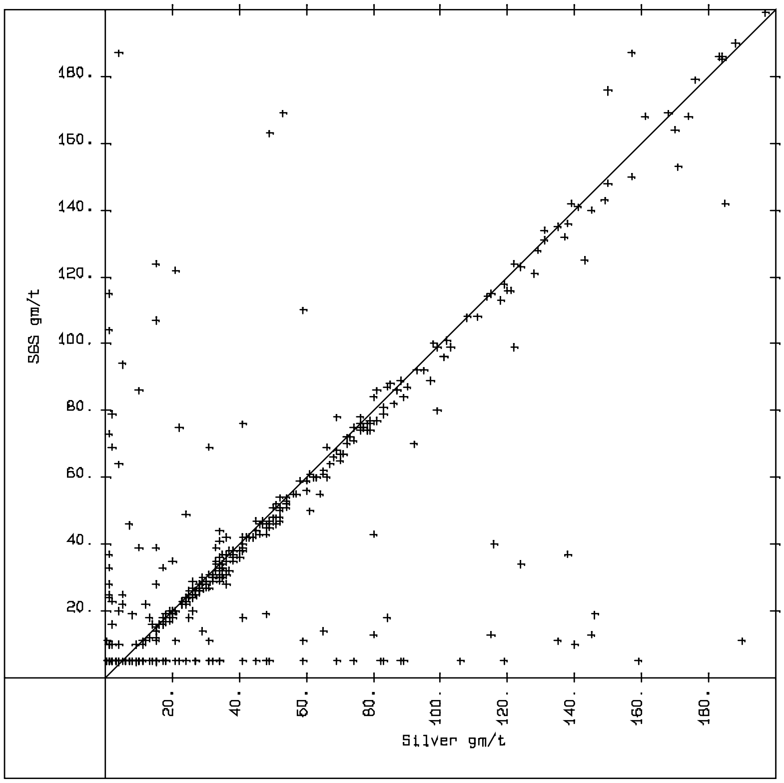
Additional plots were scanned for the period of late 2007 through March 2008. There is effectively no scatter in the plots for the most recent checks. It is not certain if the issue has been corrected or that it is not apparent with only 115 samples in the most recent check set.

The variability in check assays for the period of 2005 through 2007 can be summarized by a quick scan of the percentage of checks that were more than 25% different than the original assay.

Silver	1978 checks	18.2% are more than 25% different
Lead	1983 checks	4.2% are more than 25% different
Zinc	1984 checks	7.3% are more than 25% different

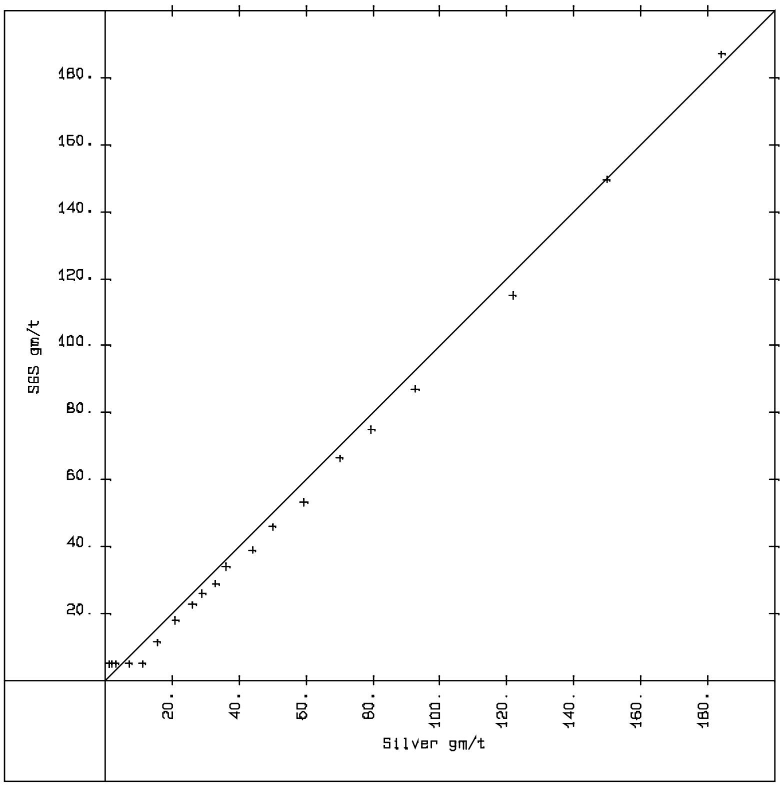
Hypothesis tests for each set of check assays do indicate that they can be accepted with 95% confidence and there is effectively no bias in the check assay result. However, the variability issue should be understood. It could simply be a function of miss-assignment of batch results to the working spreadsheet, or potential miss-labeling of pulps prior to shipment for check assay.

Many of the scattered silver results are the same samples with scattered lead and zinc results. The implication is that the entire check assay has been miss-labeled, or miss-located when inserted into the master spreadsheet.



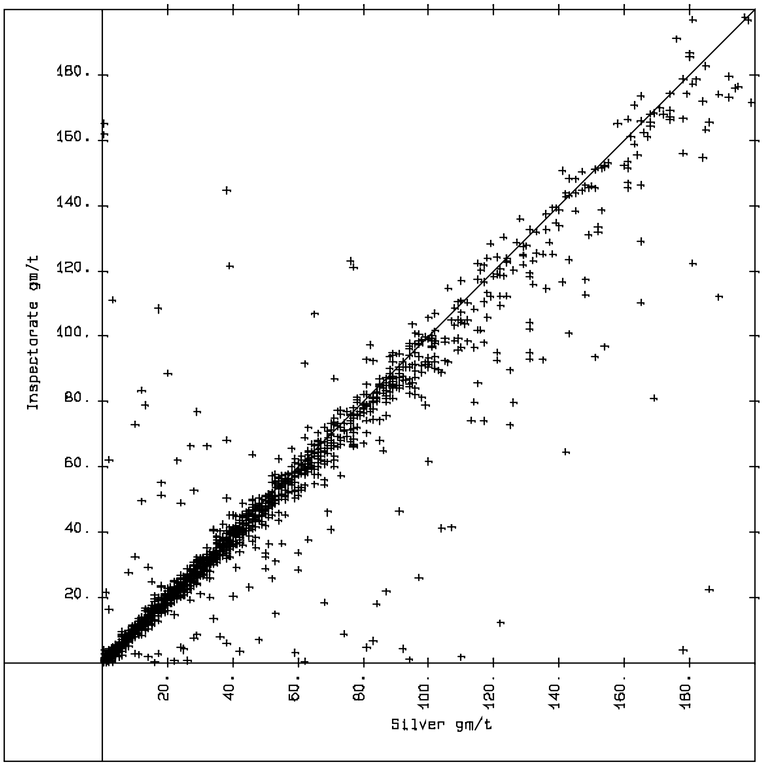
Number = 442
Original = 65.2 gm/t
SGS Check = 63.7 gm/t
T-Test = Pass at 95%
Paired T = Pass at 95%

XY Plot



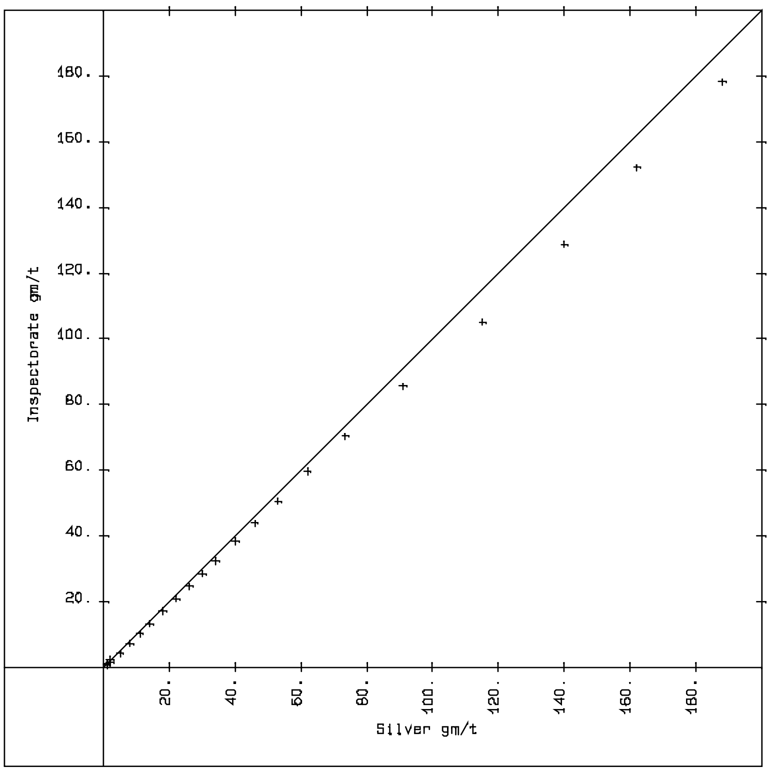
QQ_Plot
5% Cell

Figure 12-2: SGS Check Assays (Silver)



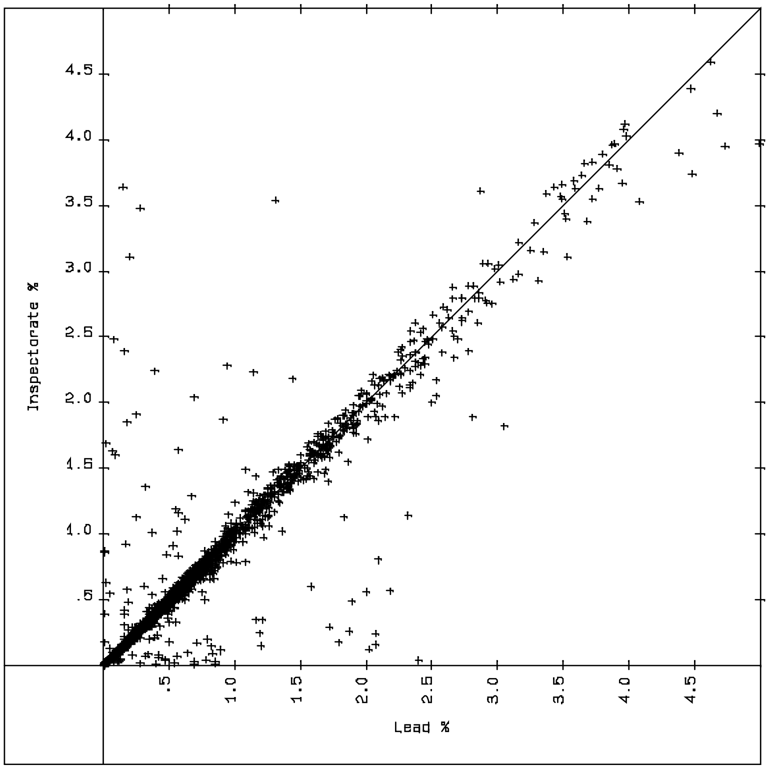
Number = 2093
Original = 48.8 gm/t
Ins Check = 47.1 gm/t
T-Test = Pass at 95%
Paired T = Fail at 95%

XY Plot



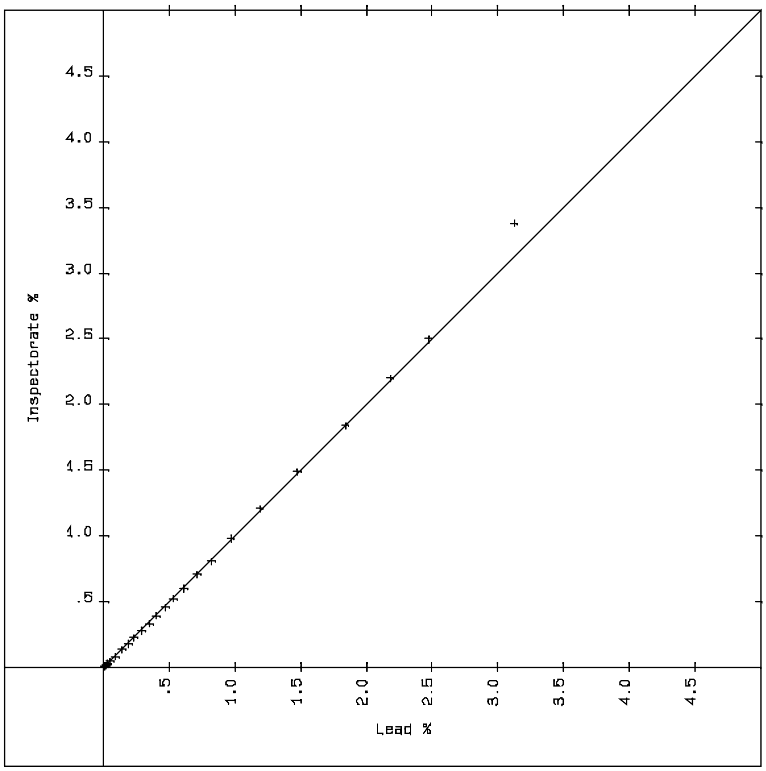
QQ Plot
5% Cells

Figure 12-3: Inspectorate Check Assays 2005-2008 (Silver)



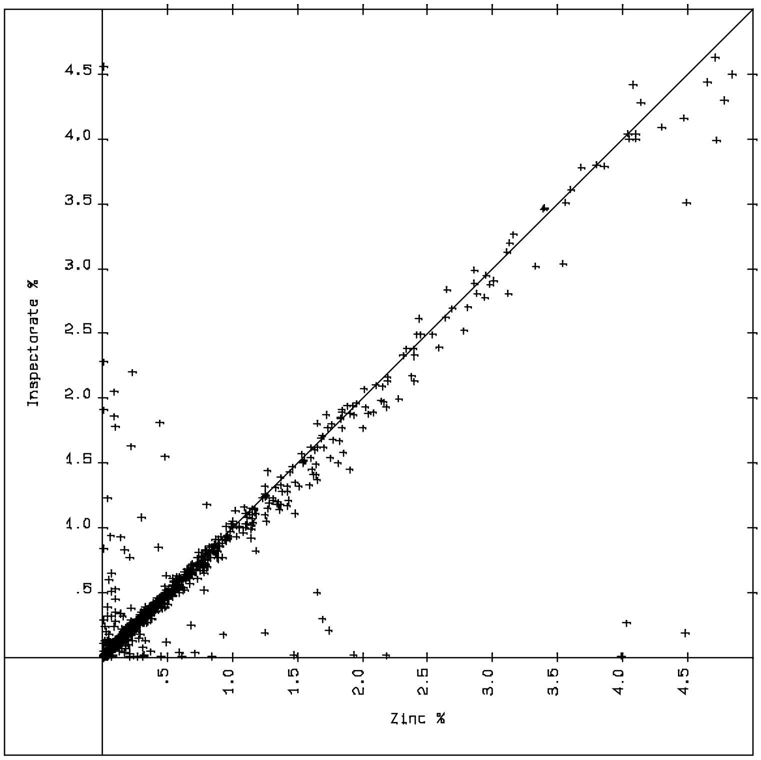
Number = 2098
Original = 0.777 gm/t
Ins Check = 0.793 gm/t
T-Test = Pass at 95%
Paired T = Pass at 95%

XY Plot



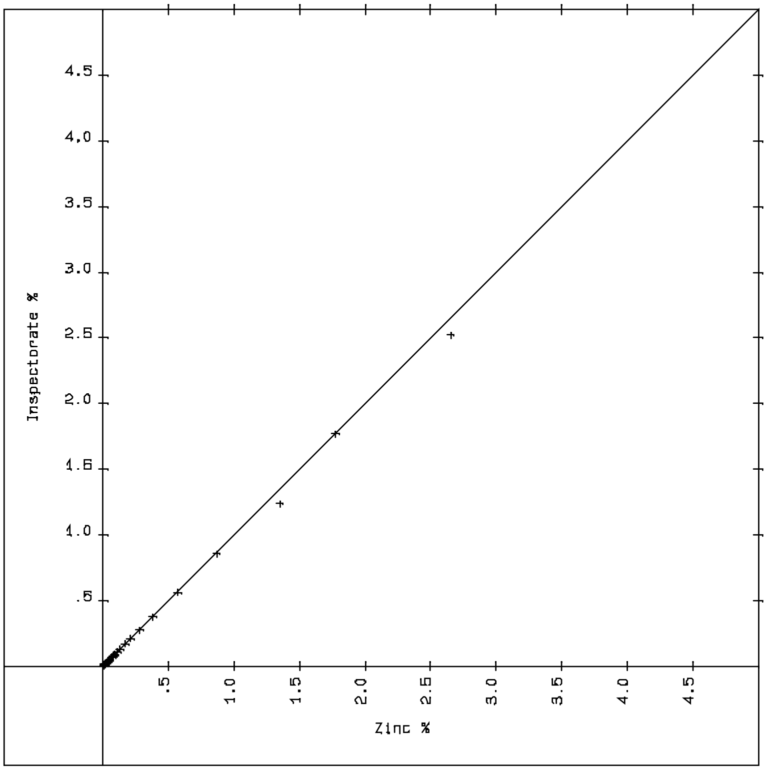
QQ Plot
5% Cells

Figure 12-4: Inspectorate Check Assays 2005-2008 (Lead)



Number = 2099
Original = 0.385 gm/t
Ins Check = 0.374 gm/t
T-Test = Pass at 95%
Paired T = Pass at 95%

XY Plot



QQ Plot
5% Cells

Figure 12-5: Inspectorate Check Assays 2005-2008 (Zinc)

12.3 CERTIFICATES OF ASSAY VERSUS DATABASE

A check of the digital database was completed by IMC in 2011 by comparing a selection of the certificates of assay versus the database provided by Bear Creek. A list of drill holes was selected by IMC at random and delivered to Bear Creek. The following summarizes the procedures used and the results of the comparison.

IMC requested assay certificates from Bear Creek Mining for the following 21 drill holes:

DDH-C3-A	DDH-C7	DDH-C12-A	DDH-C16-B	DDH-C18-A	DDH-C20-A
DDH-C29-B	DDH-C32-A	DDH-C34-A	DDH-C41	DDH-C42-A	DDH-C43-B
DDH-C46-A	DDH-C58-B	DDH-C66-A	DDH-C70-A	DDH-C74-B	DDH-C79-A
DDH-C84-A	DDH-C86-A	DDH-C92			

The assay certificate data was entered into an excel spreadsheet and then added to the IMC database containing the Corani data.

These 21 drill holes had 1,524 silver, copper, lead, and zinc assay intervals. Certificate data was received for 1,310 of the assay intervals. The following drill holes did not have an assay certificate for any of their intervals:

DDH-C3-A	1,988 to 2,052	62 intervals
DDH-C12-A	2,210 to 2,287	74 intervals
DDH-C16-B	9,388 to 9,447	57 intervals
DDH-C20-A	4,669 to 4,690	21 intervals

Thirty-one silver assays with a value less than 1.0 gm/t were entered into the database with a value of 1.0. Otherwise, there were 2 assay intervals where silver data that did not match the assay certificates, they are presented as follows:

Table 12-1: Certificate Check Errors for Silver

Hole Number	FROM (meters)	TO (meters)	Sample Number	Database Grade gm/t	Certificate Grade gm/t
DDH-C41	98	100	8842	8	108
DDH-C70-A	104	106	15521	29	11

Eight lead assays with a value less than 0.01 were entered into the database with a value of 0.01. There were 2 assay intervals where lead data that did not match the assay certificates, they are as follows in units of:

Table 12-2: Certificate Check Errors for Lead

Hole Number	FROM (meters)	TO (meters)	Sample Number	Database Grade %	Certificate Grade gm/t
DDH-C41	98	100	8842	0.04	5.90
DDH-C70-A	104	106	15521	0.02	0.01

Two zinc assays with a value less than 0.01 were entered into the database with a value of 0.01. There were 2 assay intervals where the zinc data did not match the assay certificates, they are as follows:

Table 12-3: Certificate Check Errors for Zinc

Hole Number	FROM (meters)	TO (meters)	Sample Number	Database Grade %	Certificate Grade %
DDH-C41	98	100	8842	0.01	1.59
DDH-C70-A	104	106	15521	0.04	0.01

Copper assays with a value of less than 0.01 for 383 intervals were entered into the database with a value of 0.01. There was 1 assay interval where copper data that did not match the assay certificate, it is follows in units of % copper:

Table 12-4: Certificate Check Errors for Copper

Hole Number	FROM (meters)	TO (meters)	Sample Number	Database Grade %	Certificate Grade %
DDH-C41	98	100	8842	0.01	0.13

The results for the individual metals all appear on the same record, indicating that two records out of 1310 records were entered in error. This sampling shows an error rate of 0.15% which is an acceptable level for the determination of mineral resources and mineral reserves.

The issues of trace assay entries for all metals are likely a function of continuity between data entry personnel. This issue has no material impact on the determination of reserves or resources, but should be addressed for consistency. The stated procedure by Bear Creek personnel is to enter the less than trace results at half of the value of the trace assay. For example: <1 silver should be entered as 0.5 silver within the database according to Bear Creek procedure.

This issue is minor and can be quickly corrected by Bear Creek staff.

12.4 TRENCHES VERSUS DIAMOND DRILLING

The trench assay results were compared against the nearby diamond drilling to determine if the two data sets provided similar results. The procedure used by IMC in 2011 was to composite the data into 8 m down hole (or down trench) length composites. The composites were then paired on a nearest neighbor basis.

The nearest neighbor procedure finds pairs of trench and diamond drilling composites that are within a specified distance of each other. A statistical comparison of the two data sets is then completed.

The specified distances for this test were 8 m, 16 m, and 24 m spacing between data pairs which correspond to the unit size of the 8 m length composites. There were only 20 to 25 pairs at the 8 m spacing, but there were over 80 pairs at the 16 m spacing which is sufficient to provide a robust statistical estimate.

Table 12-5 summarizes the results of the work for silver and lead. The trenches were not assayed for zinc.

Statistical hypothesis tests have been completed on the two closely located sample sets. The pass versus fail analyses on the table are based on the application of a 95% confidence band.

The T-test is a comparison of the population means. The Paired T calculates the differences between individual pairs and confirms that the differences are sufficiently small. The binomial test is a check of how many times one population is greater than the other, and the KS test (Komologorov-Smirnoff) is a comparison of the overall shape of each distribution.

In all cases, the tests indicate that the trench and diamond data can be commingled for the process of developing a block model and estimating resources.

Table 12-5: Nearest Neighbor Comparison – Trench vs. Diamond Drill Samples

Metal	Maximum Spacing Between Composites	Number of Pairs	Diamond		Trench		T Test on Means	Paired T on Pairs	Binomial Test	KS Test
			Mean	Variance	Mean	Variance				
Silver gm/t	16 meters	84	82.10	3534.6	82.94	3622.2	Pass	Pass	Pass	Pass
Lead %	16 meters	87	1.00	0.439	1.09	0.621	Pass	Pass	Pass	Pass

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 INTRODUCTION

The reader should note that Sections 13.2 through 13.6.3 describe the work relating to the 2011 Feasibility Study and Sections 13.64 through 13.72 describes the current work associated with the current Feasibility Study.

Metallurgical testing of the Corani deposit has been extensive. Over one hundred samples have been tested using a variety of testing protocols including bulk flotation, cyanidation and sequential flotation. Metallurgical response to these conventional metal recovery methods was, however, widely variable. Some samples did not respond to conventional flotation recovery, with bulk flotation being necessary. Others produced clean concentrates at relatively good recoveries. The purpose of this current study is to refine these results and provide clarity for a predictable metallurgical recovery model which can be used in conjunction with the block model for mine planning.

The high degree variability was attributed to the mineralogy of the deposit. Many of the poorer performing samples had high degrees of non-sulfide lead, resulting in poor lead recovery and zinc selectivity issues during lead flotation. In addition, the mineral grain size of many of the samples was very fine and interlocking with other sulfides would make recovery of high grade concentrates at high recovery, very challenging.

While geological classification has allowed definition of some of the zones by metallurgical response, accurate prediction of FBS mineralization metallurgical performance had remained elusive. However, based on the current study,much of the necessary data for predicting recoveries in a detailed block model are contained in the existing data base. The following describes subsequent metallurgical work performed by GRE and TS leading to the current understanding of metallurgy and recoveries.

13.2 SAMPLES

A plethora of testing, on a range of samples, has been conducted across the Corani deposit. In summary, about 66 discrete sections of drill core were batch tested using a sequential lead-zinc flotation flowsheet. Closed circuit metallurgical performance was determined via completion of locked cycle tests on a total of 16 samples. Bulk flotation response, evaluated via batch flotation tests, was determined for an additional 70 samples, approximately. Notably, for a portion of these samples, zinc contents were low and a lead concentrate was produced. The response of 46 of the samples to cyanidation leaching was tested on whole ore. In a single test, cyanidation of a flotation tailing was evaluated. The majority of the samples tested in the flotation evaluations, bulk and sequential, were subjected to mineralogical assessment, primarily using QEMSCAN. A summary of tests conducted, by year and testing facility is summarized in Table 13-1.

Table 13-1: Number of Discrete Samples for Metallurgical Testing

Test Type	Dawson 2006	G&T 2007	SGS 2007/2008	SGS 2009	SGS 2011	Total
Sequential Batch	5	6	31	-	25	67
Sequential Cycle	1	-	12	2	2	17
Bulk	8	71	-	-	-	79
Whole Ore CN	12	32	2	-	-	46

Note: The samples tested by SGS in 2009 were composites of samples tested in 2008. Thus, these samples are not truly discrete.

A description of the geological classification, used by BCM to delineate the samples, is provided here.

- CSC - Coarse-grained silica-sulfide-celadonite characterized by readily discernible sulfides (galena-sphalerite-chalcopyrite+-tetrahedrite) with celadonite in crystalline to locally opaline quartz;
- CS - A subset of CSC that contains coarse galena-sphalerite-chalcopyrite +- tetrahedrite without green celadonite clay;
- TET – Ag-bearing tetrahedrite characterized by recognizable late-stage, coarse-grained tetrahedrite cutting earlier sulfides and displaying the highest Ag contents: normally ores with low Pb-Zn contents;
- PM – Pyrite-marcasite +- quartz typical of early-stage mineralization with little polymetallic mineralization;
- FBS – Fine-grained black silica-sulfides characterized by very fine-grained mineralogy deposited from quenched ore fluids with highly variable metal content;
- QSB - Crystalline quartz-sulfide-barite interpreted as early fault fill or late-stage breccia fill;
- PG – Plumbogummite, identified as a pale-green, waxy, Pb-phosphate mineral that in metallurgical test results showed diminished lead flotation and difficulties in separation of base metals;
- FeOx – Iron-oxide mineralization with locally elevated Ag and generally low Pb-Zn. This is a gradation zone with mixtures of FeOx and FBS; and
- MnO – Manganese-oxide mineralization hosting mainly Ag with lesser Pb-Zn.

A summary of the number of samples tested for each geological classification is provided in Table 13-2 FBS is understood to be the main geological lithology by tonnage in the Corani deposit, and thus, the samples were also dominated by this geological classification.

Table 13-2: Number of Samples Tested by Oretype

Oretype	Dawson 2006	G&T 2007 Bulk Tests	G&T 2007 CN Tests	SGS 2007-2008	SGS 2009	SGS 2011
CSC	2	6	-	2	-	-
FBS	2	26	8	17	-	11
PM	-	15	3	6	-	4
QSB	-	7	3	2	-	6
FeOx	1	4	5	1	-	2
MnO	-	6	6	2	-	1
PG	-	7	7	-	-	1
Mixed	5	-	-	-	2	8

Notes: a) Samples that contained two oretypes were categories into the dominant oretype for that sample.
b) Samples from which samples across multiple holes were combined are referred to as 'mixed'.

In total, 36 samples were subjected to ore hardness testing using different grindability testing protocols. A summary of the number of samples tested using the different comminution tests is summarized below.

Table 13-3: Number of Comminution Tests Conducted on Discrete Samples

Test / Deposit	SGS Vancouver 2008	SGS Chile 2010	SGS Chile 2012
SPI	10	6	20
Bond Ball	10	6	20
Bond Abrasion	2	6	20
SMC Test	-	6	20
Bond Rod	-	6	20
LEIT	-	6	20
PLT	-	-	17
UCS	-	-	13
Este	2	2	11
Minas	6	2	7
Main	2	2	2

The location of the metallurgical testing and grindability samples are shown in Figure 13-1.

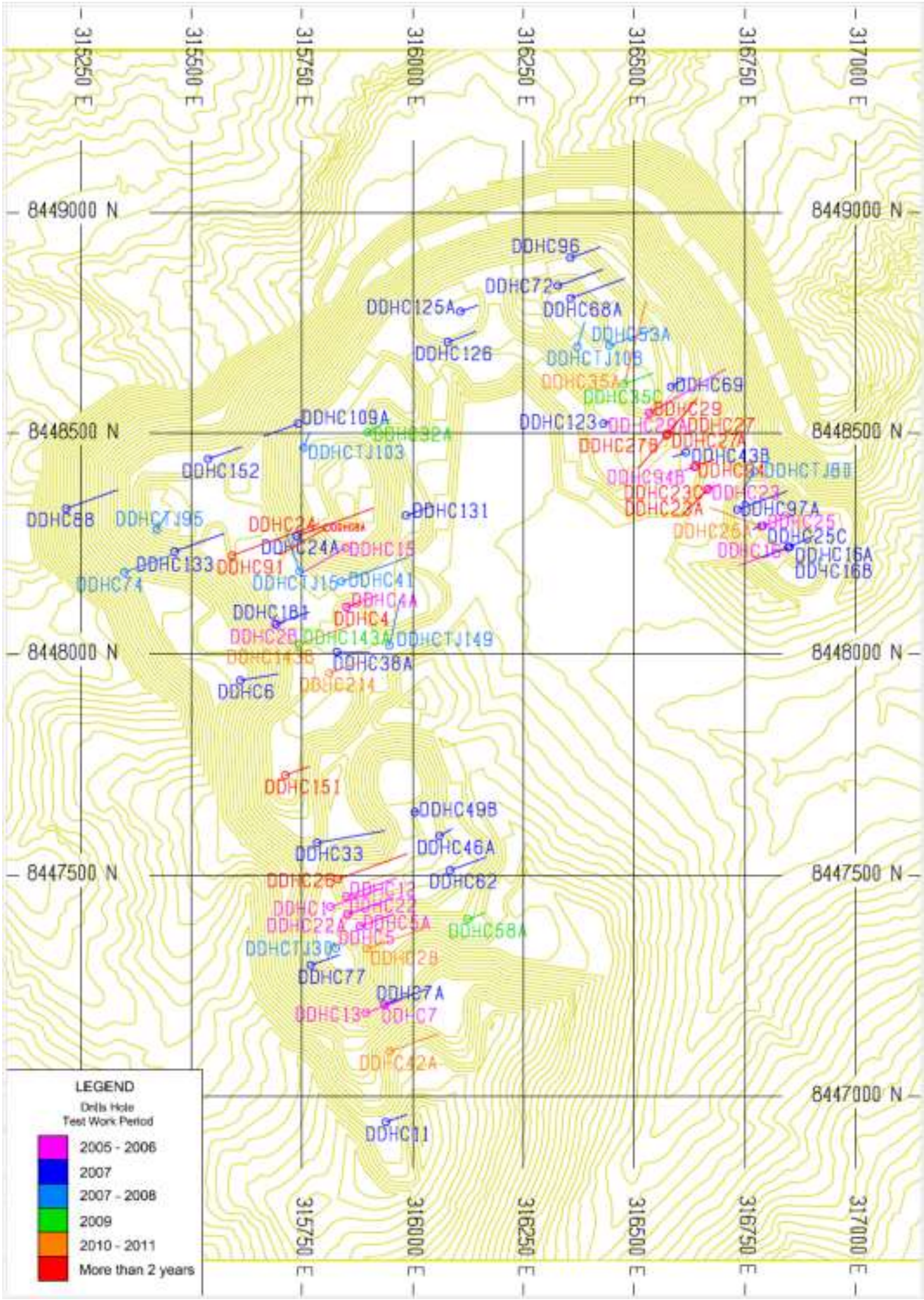


Figure 13-1: Location of Drill Holes from which Metallurgical Test Samples were sourced.

13.3 MINERALOGY

Lead occurs not only as Galena, but also in non-sulfide forms such as plumbogummite (PbAl3(PO4)2(OH)5•(H2O)). Non-sulfide forms of lead would not be expected to be recoverable via flotation. According to the SGS dataset, generated using QEMSCAN technology, galena and sphalerite are both very finely grained. Fine primary and regrind sizes would be anticipated to facilitate separation of the two minerals into clean flotation concentrate.

The relationship between these mineralogical characteristics and metallurgical response was demonstrated in these Figures, prepared by SGS, relating the mineralogical parameters to bulk flotation response. In these figures, SGS related the QEMSCAN mineralogical data that they generated against the metallurgical performance data generated by G&T using a bulk circuit for the same samples. The first Figure 13-2 displays the relationship between galena grain size and bulk circuit lead recovery. This data indicates galena grain size is an important driver of metallurgical response. Also of note, is that the grain sizes noted are very fine, measuring less than 17µm in all cases.

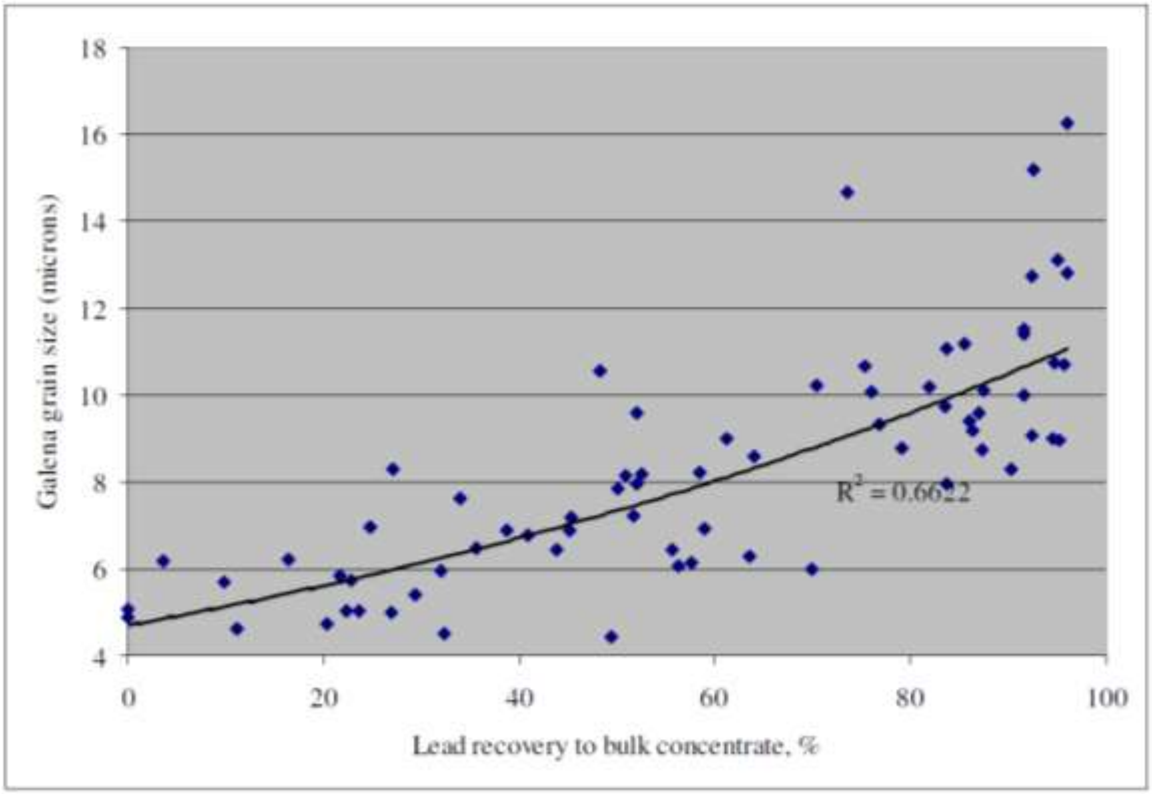


Figure 13-2: Relationship between Lead Recovery and Galena Grain Size (graph from SGS)

Figure 13-2 displays the correlation between the deportment of lead to galena and lead recovery to the bulk circuit for the same samples as Figure 13-3 The relationship is less clear than for galena grain size but indicates that this deportment plays a role in lead recovery. This is not surprising as non-sulfide forms of lead are not generally recoverable via flotation.

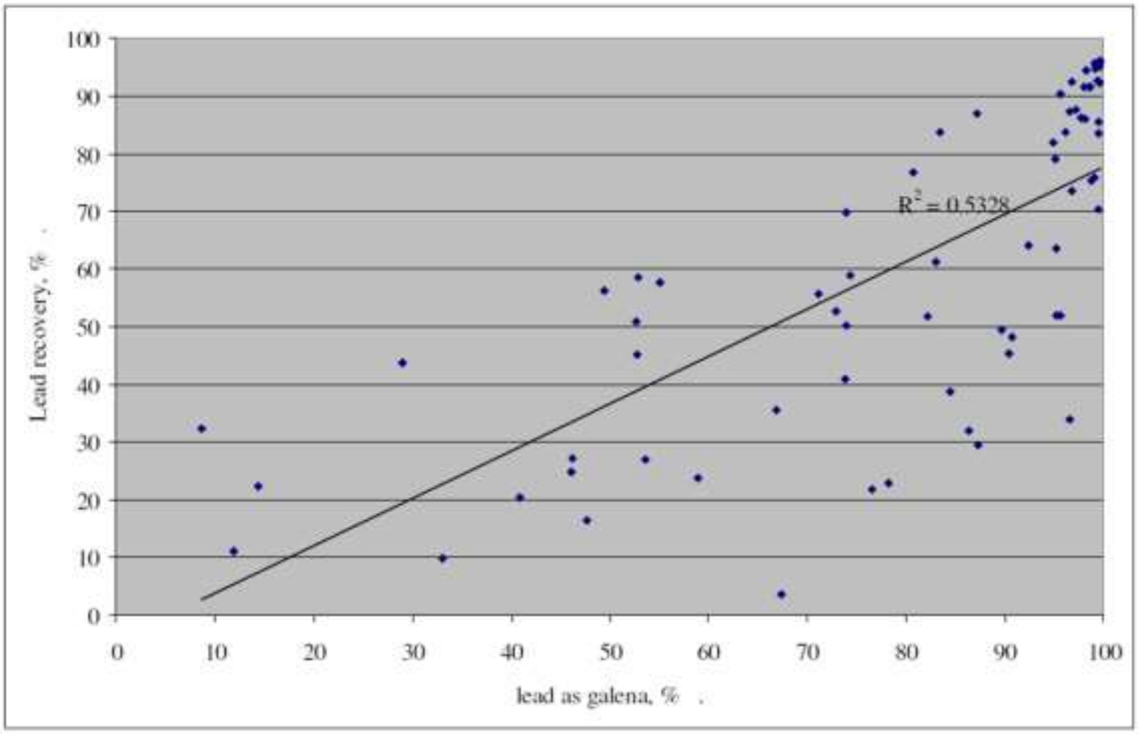


Figure 13-3: Relationship between Lead Recovery and Deportment of Lead to Galena (graph from SGS)

Zinc present in the samples was identified as sphalerite. Zinc performance was found to be mainly driven by sphalerite grain size. The relationship between zinc recovery and sphalerite grain size is shown in Figure 13-3. The grain sizes were similarly fine as for galena, showing the need for fine primary and / or regrinding.

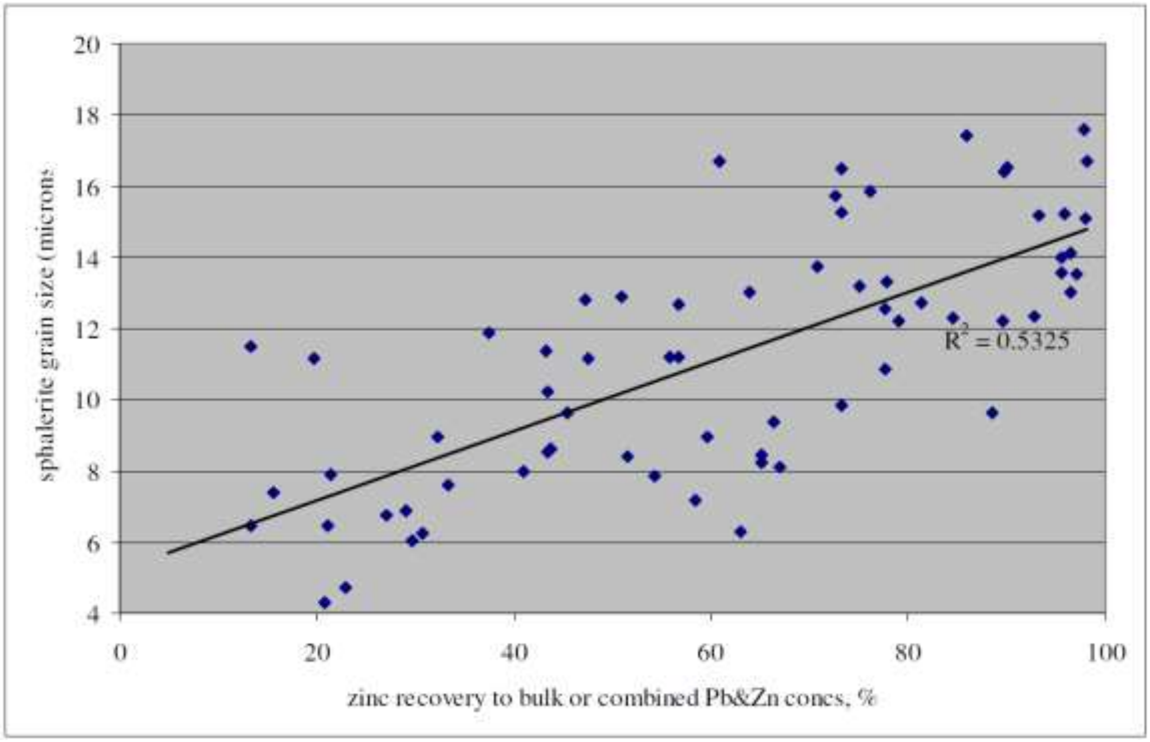


Figure 13-4: Relationship between Zinc Recovery and Sphalerite Grain Size (graph from SGS)

Silver mineralogy was briefly investigated by Hazen research then again by SGS in 2008. The data indicated that silver was present mainly as tetrahedrite. Other sulfosalts were identified as minor carriers and included: myargyrite, pyrargyrite-proustite, boulangerite acanthite and native silver. The Hazen mineralogy indicated that silver was also in solid solution with pyrite and possible sphalerite. Most of the silver bearing minerals mentioned responded well to flotation when properly liberated. In a sequential lead zinc flotation circuit, most of the silver minerals should report to the lead concentrate.

Photomicrographs and back-scatter images from a Scanning Electron Microscope (SEM) were taken by Hazen Research and confirmed this complex, fine grained mineralogy. However, coarse galena grains were also noted in some samples. An example showing both complex and coarse liberated particles from Hazen Research on the Dawson composites is shown in Figure 13-5. Friebergite (f) is intergrown with pyrite – marcasite (yellow) and coarse, liberated galena (g) cleavage fragments.

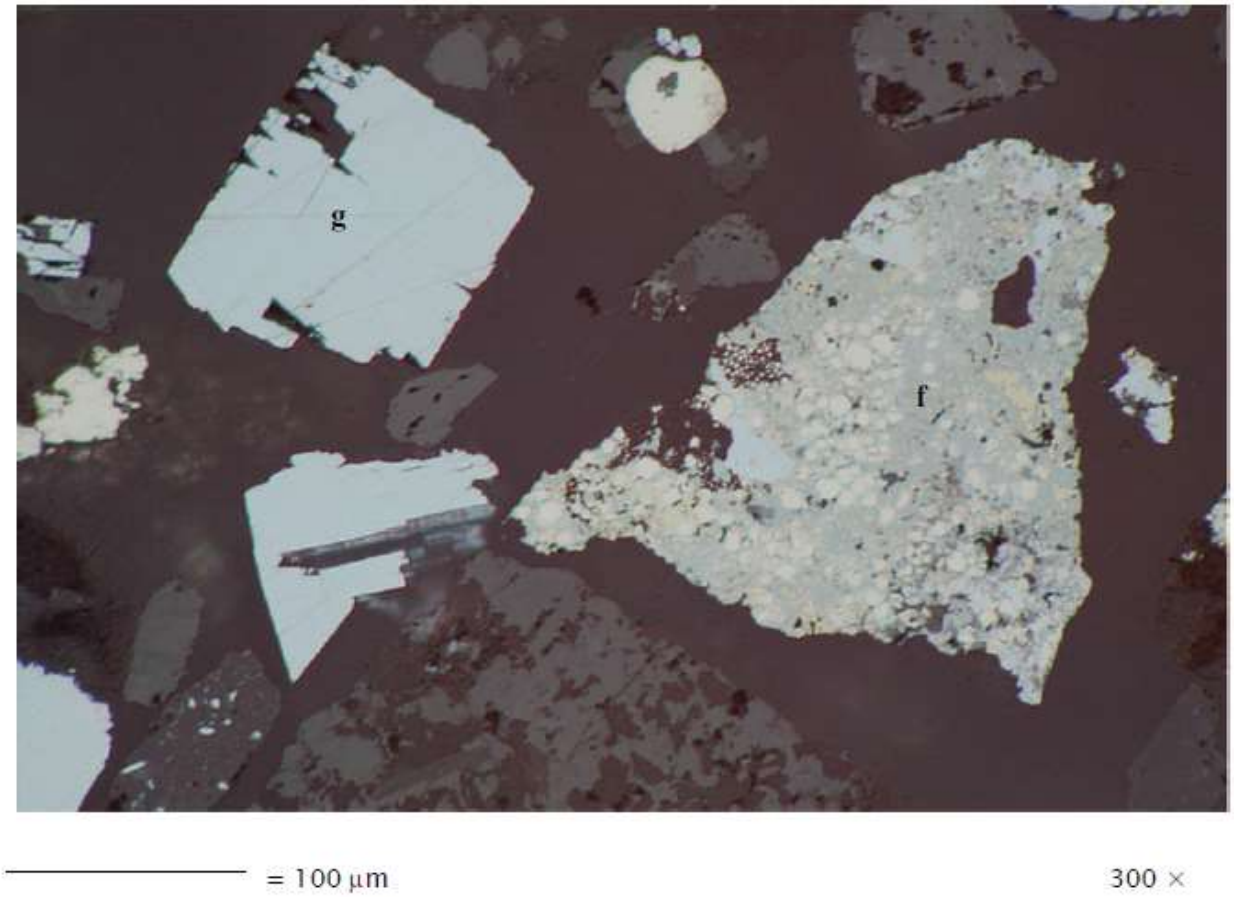


Figure 13-5: Dawson Composite D, minus 500µm sinks (Hazen Research)

13.4 GRINDABILITY TESTS

A summary of the grindability results is provided in Figure 13-4. The number of samples tested per zone is also provided. The following tests were performed on Corani samples,

- Low Energy Impact Test, LEIT
- SAG Mill Comminution, SMC
- SAG Power Index, SPI
- Bond Rod Mill Work Index, BRWI
- Bond Ball Mill Work Index, BBWI
- Abrasion Index
- Point Load Test, PLT
- Unconfined Compressive Strength, UCS

Table 13-4: Comminution Test Results Summary

Parameter	LEIT Kw-h/mt	Dwi kWh/m^3	SMC Axb	SPI min	Crusher Index	BRWI kWh/t	BBWI 75 kWh/t	BBWI 106 kWh/t	Abrasion Index	PLT Is 50 Mpa	UCS Mpa
No. Tests	26	26	26	36	36	26	10	26	28	17	13
No. Este	13	13	13	15	15	13	2	13	13.5	8	6
No. Minas	9	9	9	15	15	9	6	9	10	7	7
No. Main	4	4	4	6	6	4	2	4	4.5	2	-
Average	6.30	2.35	111	35.6	26.9	10.2	15.5	14.6	0.17	1.47	31.8
Max	10.2	3.98	210	90.6	49.8	13.4	19.2	18.6	0.49	2.71	52.7
Min	2.68	1.17	60.8	15.3	14.1	6.53	12.3	10.1	0.03	0.67	23.5

Note: The number after BBWI refers to the closing screen size used for the test, in µm.

Based on these results, the ore appears to be of medium hardness with respect to SAG and ball milling.

13.5 SEQUENTIAL FLOTATION TESTS

In various programs, both bulk and sequential flowsheets were evaluated. Much of the flotation testing, conducted by SGS, focused on the production of a marketable lead concentrate; from use of what would be referred to as a sequential flotation flowsheet. The separation of clean marketable lead and / or zinc concentrates was successful on only a portion of samples. For some samples, zinc or pyrite was recovered uncontrollably to the lead concentrate resulting in lower grade products. For other samples, flotation response was limited leading to very low recoveries.

G&T only conducted sequential flotation tests on a small sub-set of samples with favorable flotation response and suitable head grades.

Due to the improved marketability of the separate lead and / or zinc concentrates over a bulk concentrate; a sequential flowsheet was used for the design of the plant. The following subsections refer to the development of that sequential flowsheet.

13.5.1 Effect of Particle Size

The effect of primary grind was not studied in great detail for many of the programs. Much of the testing was devoted to various depression and collector regimes used to control sphalerite (zinc) flotation in the lead circuit. However, there were series of tests conducted in the SGS 2008 and 2009 campaigns that examined the effect of primary grind size.

Rougher test data for the 2008 tests data was taken from composites A and B, which represented only the FBS geological classification. The test data for the 2009 test program was from the Ag-Pb composite, which was a mix of geological classifications.

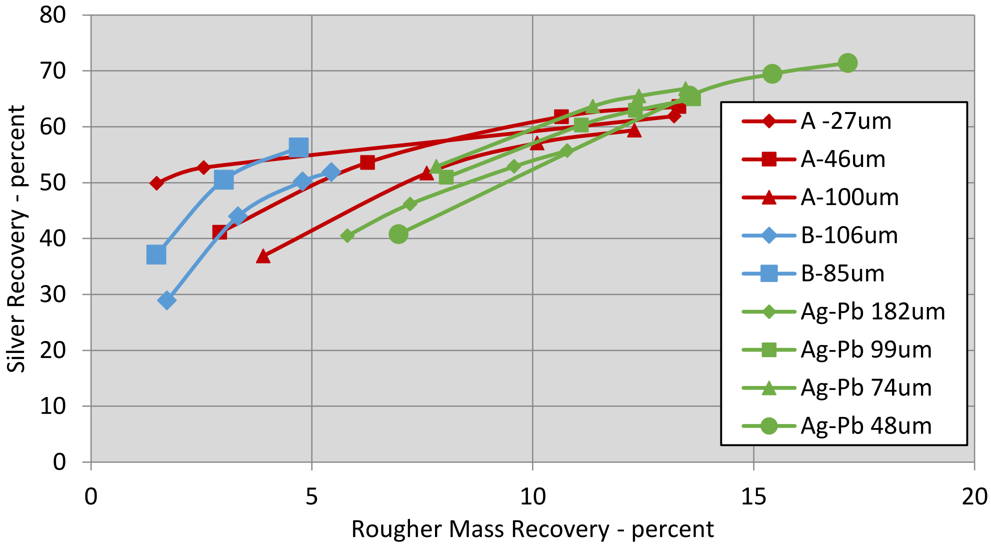


Figure 13-6: Relationship between Grind Size and Silver Rougher Recovery

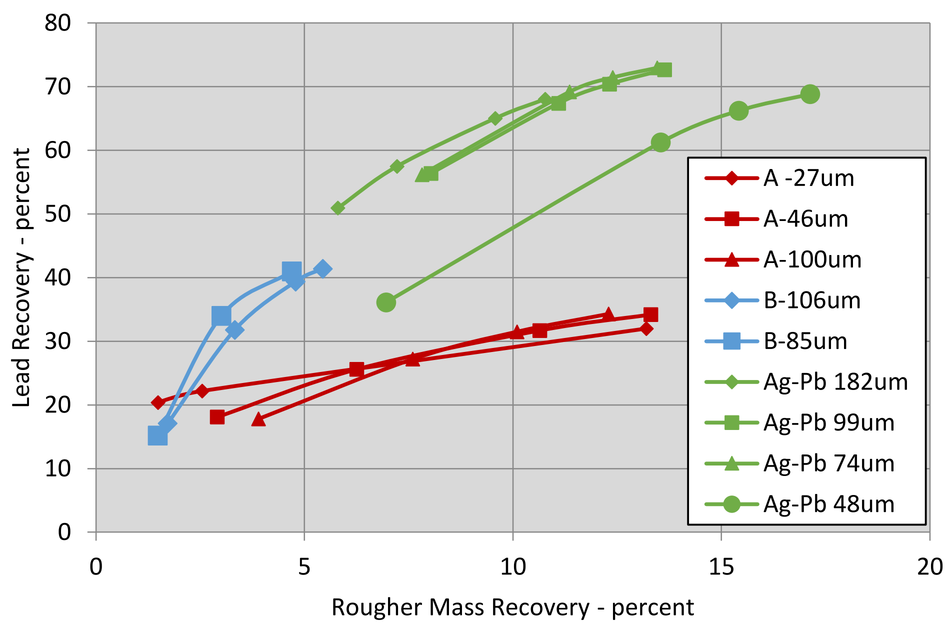


Figure 13-7: Relationship between Grind Size and Lead Rougher Recovery

As shown in the figures, the effect of primary grind was more pronounced for silver when compared to lead. The expected trend of better metallurgical performance was often observed for the finer grind sizes early during the flotation stages, equating to the low rougher mass recovery. As rougher mass recovery was increased, the effect of grind size was minimal. The data would suggest that if sufficient rougher mass recovery is achieved, primary grind sizes of 100 micron K80 or coarser could be used. There may be potential to further increase primary grind size.

13.5.2 Effect of Depressants

A challenge with many of the Corani deposit samples was the poor selectivity between lead and zinc in the lead flotation circuit. The majority of the testing conducted was devoted to improving the selectivity of the lead flotation circuit against the flotation of sphalerite (zinc).

To tackle the activation of sphalerite, several depression schemes were employed. The depression schemes could be classified into general categories as follows:

- Zinc sulphate (ZnSO4)
  - Used in combination with selective collectors
- Zinc sulphate/cyanide (NaCN) or Zinc oxide/cyanide (ZnO / NaCN)
  - Most common depressant scheme for zinc depression during lead flotation. Very effective to combat copper ion activation of sphalerite and pyrite.
- Sulfite/Sulfide includes reagents Na2SO3, Na2S, SO2.
  - Less common, but effective to sphalerite, pyrite depression when cyanide is not permissible. High dosages of these reagents will also depress galena (lead).

The development of depressants used for most of the locked cycle tests occurred while testing of composites A, B, B2, and G during the SGS program in 2008-2009. The most complete matrix of testing was completed on sample G. Results of zinc depression are displayed in the following figures. Unfortunately, there were no comparable tests without depressant.

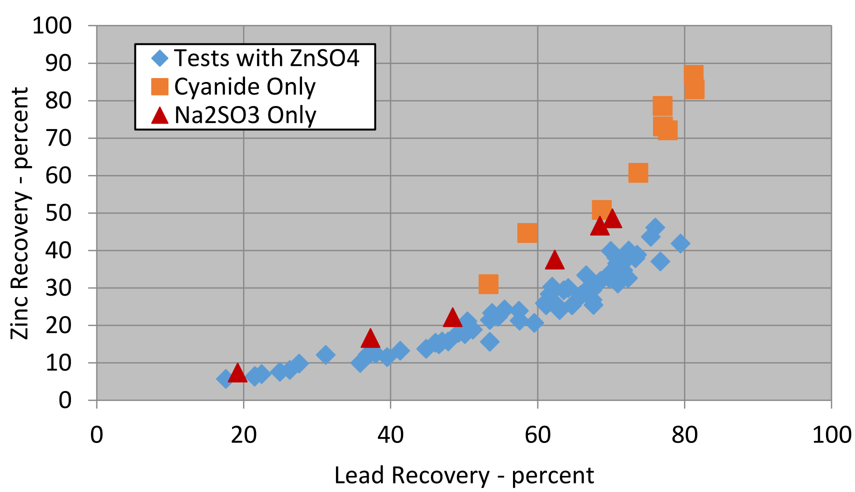


Figure 13-8: Effect of Depressants on Lead Rougher Flotation – G Composite (FBS)

As shown in the figure, the best selectivity was achieved when zinc sulphate (ZNSO4) was added. When cyanide was added without zinc sulphate, the activity of zinc was actually increased. It should be noted that one test was conducted with both zinc sulphate and cyanide and the results mirrored the zinc sulphate only conditions. Sodium sulphite (Na2SO3) alone did not appear to be effective for suppressing zinc in this sample.

To further investigate the effects of zinc sulphate, there were a number of tests that utilized variable dosages of zinc sulphate with the other variables remaining constant. The results of these tests are displayed in Figure 13-9.

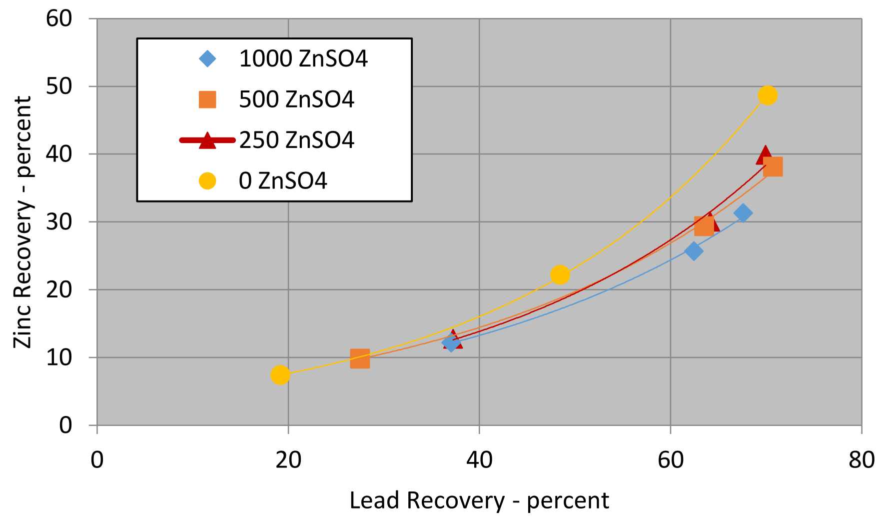


Figure 13-9: The Effect of Zinc Sulphate Dosage on Zinc Depression from the Lead Rougher Flotation, Composite G (FBS)

The curves indicated that the addition of zinc sulphate improved lead flotation selectivity against sphalerite. There was a slight improvement with increased dosage. As the project continued, a depression regime was established using zinc sulphate, cyanide and sodium sulfite. The data generated on composite G was not verified on any other composite, however, the inclusion of zinc sulphate in the depression scheme may have been the most important aspect of controlling zinc in lead flotation in all of the following tests. There may be an opportunity to reduce or eliminate the use of sodium sulfite and cyanide.

13.5.3 Collectors

Again, much of the focus of flowsheet development program was dedicated to developing selectivity between lead and zinc in the lead flotation circuit. Several collectors were used, cytec products 3418A, 242 and 404. Xanthates were also included in combination with cytec collectors. Potassium xanthate (strong) and sodium isopropyl xanthate (weaker) were primary xanthate collectors used. In the zinc circuit, xanthates were also used.

Unique to this project, a theoretical application of emulsified diesel oil (EDO) was also considered for this project. EDO was added after an initial low dosage application of a lead selective collector. The EDO acts as an extender, enhancing the hydrophobicity of the selective collector. This collector extender was widely used for much of the testing. Composite G had several combinations of collector tested and isolated (all other conditions constant). Figure 13-10 displays the selectivity differential between the different collectors.

As shown, the use of different collectors did not have a significant impact on selectivity between lead and zinc in the lead rougher flotation. The dosage of collector did shift the results up the curve (higher dosages) or down the curve (lower dosages).

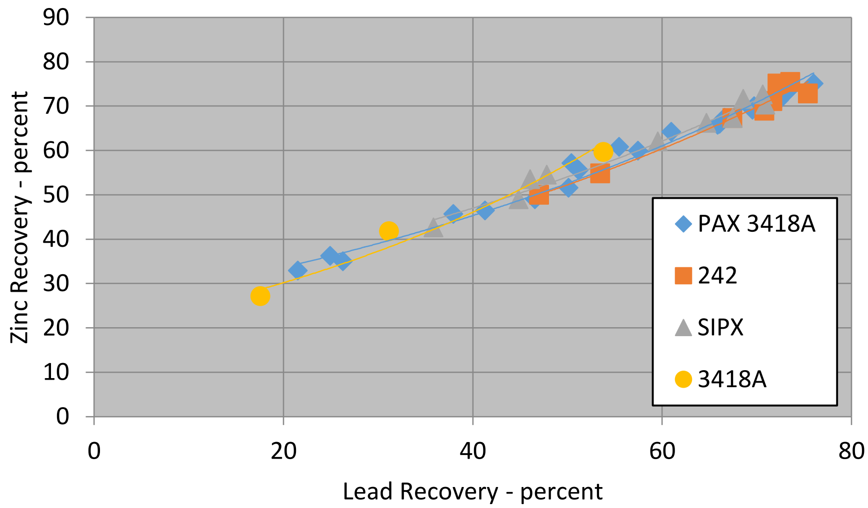


Figure 13-10: The Effect of Collector Type on Lead Rougher Flotation Selectivity against Zinc, Composite G (FBS)

Using the same series of rougher tests, the effects of EDO were isolated. As shown in Figure 13-11, EDO did not impart any gains in selectivity. This conclusion was reached later in testing at SGS in 2011.

Based on these results, it would appear that the use of lower cost xanthates should be considered. As with all collectors the dosage rate is an important factor to controlling selectivity in the lead circuit.

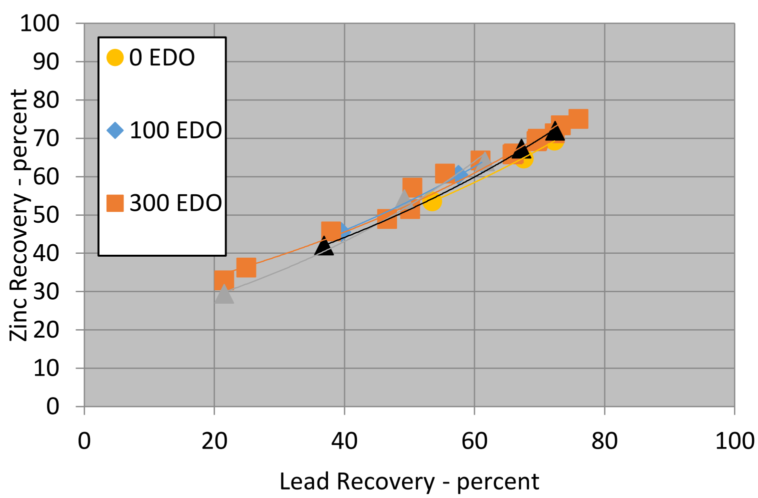


Figure 13-11: The Effect of EDO on Lead Rougher Flotation Selectivity against Zinc, Composite G (FBS)

13.5.4 Activators

In the lead circuit, activators such as sodium hydrosulfide and hydroximate collectors were briefly tested as a means of activating and recovering non-sulphide lead. The test results were discouraging, indicating that the plumbogummite, the dominate non-sulphide lead mineral, did not respond to these reagents.

In the zinc circuit, activation of zinc was accomplished with copper sulphate (CuSO4). This is standard practice in industry. Dosages of copper sulphate required ranged between 100 and 350 g/tonne.

13.5.5 pH Regulators

The use of pH regulators can be effective in controlling pyrite when used in combination with selective collectors and other depressants. In the lead circuit, controlling the pH to 8 to 9 is a common to help depress pyrite and other iron sulphides. Above pH 9, lead depression occurs. The natural pH of many of the samples tested was well below 7; most samples averaged 5.5. Increasing the pH with lime in the lead circuit was ineffective and often resulted in increased activity of sphalerite flotation. Decreasing the pH with sulphuric acid was also investigated but showed no advantage.

In 2011, a global composite (Ag-Pb-Zn) used for detailed flowsheet development indicated issues with pyrite activation as well as zinc activation in the lead flotation circuit. To combat issues of pyrite activation, sodium carbonate (NaCO3) was used effectively to reject pyrite and increase the grade of lead in the concentrate for this composite. Many other composites did not require the use of sodium carbonate to produce high grade lead concentrates.

For the zinc flotation circuit, the addition of lime to pH 11 or 11.5 was used for most of the composites tested.

13.5.6 Surface Modifiers

There were may surface modifiers tested mostly to improve the quality of the lead concentrates. These included sodium silicate (silicate mineral depressant and dispersant), starches (CMC and quar gum for silicate depressants). None of these reagents were useful in improving the metallurgical response of the flotation circuits.

13.5.7 Rougher Concentrate Regrind

Due to the focus pulp on chemistry, very little coordinated testing was devoted to investigating the effects of regrind. The quantitative mineralogy suggested that some of the mineralization (FBS) would require very fine primary and rougher regrind targets to achieve acceptable concentrate grades. Most of the testing conducted in 2008-2009 by SGS did only semi-quantitative sizing analysis. In these programs the target regrind size for both the lead and zinc regrinds appeared to be between 20 and 30 micron K80.

Based on the grain size data from the mineralogy, this target for the majority of the mineralization would be reasonable. The mineralization identified as CSC, could utilize a coarser regrind and primary grind size target.

13.5.8 Summary of Sequential Locked Cycle Test Results

While many of the locked tests had different conditions applied, the average locked cycle test performance for all tests performed is displayed in Table 13-5.

Table 13-5: Summary of Sequential Locked Cycle Test Data

Zone	No. Samples	Pb Con Gr - % or g/t		Rec to Pb Con - %		Zn Con Gr - % or g/t		Rec to Zn Conc. - %	
		Pb	Ag	Pb	Ag	Zn	Ag	Zn	Ag
CSC	2	68	1620	79	70	52	729	72	12
FBS	17	50	2569	50	49	52	462	77	23
PM	6	50	1269	62	46	56	251	59	12
QSB	2	47	3386	64	64	-	-	-	-
FeOx	1	waste							
MnO	2	waste							

Note: Ag grades are denoted in g/tonne, all other assays are in percent.

No samples representing the PG (plumbogummite) zone only were tested in the 2007-2008 SGS programs. Such a sample would need to be obtained and tested to generate metallurgical performance predictions. However, given the poor metallurgical response of PG samples to a bulk flowsheet, it is unlikely such mineralization would respond well to a sequential flowsheet.

13.5.9 Concentrate Quality

A suite of the lead and zinc concentrates produced by SGS from locked cycle testing was assayed for a series of minor elements. Results are summarized, for the lead and zinc concentrates, respectively, in Table 13-6 and Table 13-7.

The lead concentrates that SGS assayed, on average, graded 59 percent lead and 1815 g/tonne silver. The lead grade, particularly, is on the higher end of what is achievable for Corani mineralization. The concentrates, on average, graded about 0.8 percent antimony which would be expected to result in smelting penalties. Arsenic, mercury and cadmium were also elevated and there may also be penalties applied for these elements.

Table 13-6: Minor Elements in Lead Concentrates

Element	Units	Lead Concentrate from Locked Cycle Test on Composite											
		U	M	G	R	Main 1	3 zone AgPbZn Ave	1-5yr	Q	T	Minas 1	Minas 3	**Average**
Ag	g/t	2448	1776	1678	465	810	1576	2100	1752	3696	1493	2173	**1815**
Pb	%	55.5	55.5	51.2	58.9	53.6	54.8	51.0	62.6	51.1	73.0	81.1	**58.9**
Zn	%	7.18	4.11	9.03	2.60	6.10	8.15	8.10	0.98	5.40	0.82	2.90	5.03
Cu	%	4.50	1.70	1.90	0.38	1.90	1.25	1.62	0.52	2.00	1.50	0.35	1.60
Au	g/t	-	-	-	-	-	0.29	0.37	-	-	-	-	0.33
S	%	-	-	-	-	-	20.3	21.3	-	-	-	-	20.8
C(t)	%	-	-	-	-	-	0.27	1.77	-	-	-	-	1.02
Cl	g/t	-	-	-	-	-	15	< 10	-	-	-	-	<10
F	%	-	-	-	-	-	0.020	0.014	-	-	-	-	0.013
Hg	g/t	6.2	4.4	23.9	4.9	30.2	22.1	16.8	47.0	15.6	10.0	1.6	16.6
Al2O3	g/t	7366	4156	8122	14545	4911	3950	0.61	11333	3400	8878	1889	6232
As	g/t	510	1500	3000	2500	440	525	2750	740	910	470	<40	**1215**
Ba	g/t	2800	360	360	34	200	225	396	23	2800	3400	190	981
Bi	g/t	<200	<30	<30	<20	<20	62	74	<20	130	120	<20	<200
Ca	g/t	190	420	400	200	<40	240	232	2100	480	540	240	460
Cd	g/t	450	620	2800	2000	310	720	1390	5000	450	46	190	**1271**
Fe	%	5.6	11.0	8.6	7.6	10	9.45	8.1	5.1	13.0	3.0	1.3	7.5
Mg	g/t	600	190	79	190	78	115	117	220	940	480	150	287
Mn	g/t	370	410	160	130	320	540	278	170	1000	1400	240	456
Na	g/t	23	<30	<30	280	72	58.5	72	170	62	150	130	95
P	g/t	370	360	570	<150	<200	<200	266	<150	890	1600	<200	446
Sb	%	1.70	0.62	1.90	0.38	0.70	0.58	1.21	0.20	0.90	0.48	0.34	**0.82**
Se	g/t	<30	<30	<30	<30	<30	<30	< 30	<30	<40	<30	<30	<30
Sn	g/t	<25	<30	<30	<20	<40	<30	< 20	110	<50	<20	<20	<110
SiO2	%	-	-	-	-	-	2.95	4.15	-	-	-	-	3.55

Note: The 3 zone mixed sulphide (Ag-Pb-Zn) Composite ("3 zone AgPbZn Ave" in the table above) result is an average of two suites of assays from two locked cycle tests conducted on this composite.

The zinc concentrates, assayed by SGS, on average graded 53.8 percent zinc and 331 g/tonne silver. Mercury was elevated in these concentrates, grading on average 56 g/tonne in the zinc concentrates. At this level, penalties, if not marketability issues, would be anticipated. Antimony and cadmium were also elevated in these concentrates and potential penalties may be applicable.

Table 13-7: Minor Elements in Zinc Concentrates

Element	Units	Zinc Concentrate from Locked Cycle Test on Composite							
		U	M	G	R	Main 1	3 zone AgPbZn Ave	1-5yr	**Average**
Ag	g/t	286	272	371	288	250	385	463	**331**
Pb	%	1.27	5.76	4.86	6.63	2.16	4.04	4.80	4.22
Zn	%	56.4	55.8	53.0	49.3	58.0	51.9	52.3	**53.8**
Cu	%	0.50	0.30	0.37	0.52	0.28	0.29	0.63	0.41
Au	g/t	-	-	-	-	-	0.10	0.39	0.25
S	%	-	-	-	-	-	28.6	31.7	30.1
C(t)	%	-	-	-	-	-	0.20	0.16	0.18
Cl	g/t	-	-	-	-	-	62.5	< 10	34
F	%	-	-	-	-	-	0.021	0.008	0.014
Hg	g/t	23.3	38.8	67.3	47.0	55.1	89.1	69.4	**55.7**
Al2O3	g/t	4700	2078	4345	11334	8122	13750	2500	6690
As	g/t	600	890	170	740	230	285	722	520
Ba	g/t	970	140	200	23	390	465	179	338
Bi	g/t	<200	<30	<30	<20	<20	<50	< 20	<50
Ca	g/t	1400	800	920	2100	1100	1835	1090	1321
Cd	g/t	1800	3300	5200	5000	1600	2650	4940	**3499**
Fe	%	4.6	3.8	3.1	5.1	5.3	4.6	5.2	4.5
Mg	g/t	960	1100	70	220	140	305	71	409
Mn	g/t	540	370	130	170	190	555	287	320
Na	g/t	<10	<30	<30	170	100	190	49	78
P	g/t	240	230	370	<150	<200	370	< 200	212
Sb	%	0.36	0.08	0.16	0.20	0.36	0.145	0.21	**0.22**
Se	g/t	<30	<30	<30	<30	<30	<30	< 30	<30
Sn	g/t	62	<30	110	110	<40	<30	57	56
SiO2	%	-	-	-	-	-	7.65	2.66	5.16

Note: The 3 zone mixed sulphide (Ag-Pb-Zn) Composite ("3 zone AgPbZn Ave" in the table above) result is an average of two suites of assays from two locked cycle tests conducted on this composite.

13.6 ALTERNATIVE FLOWSHEET ARRANGEMENTS

Bulk flotation testing was conducted, for various samples, in the Dawson, G&T and SGS programs. For the SGS program, the sample set that was tested for bulk flotation may be biased towards samples with poor flotation response. But Dawson and G&T both seem to test a full range of samples and some conclusions as to the variability in metallurgical response to flotation can be drawn from that work.

In the 2006 test program, conducted by Dawson Metallurgical Laboratories, Ltd., a suite of 12 samples was tested for cyanidation leaching response. This sample selection appears to be unbiased whereas in subsequent programs, only a selection of samples was subjected to cyanidation bottle roll tests, presumably due to poor flotation response. As such, only limited information can be drawn from those datasets. Dawson also conducted testing on two samples evaluating a combination of flotation and cyanidation.

The following sub-sections discuss the result of tests conducted using these alternative flowsheets.

13.6.1 Bulk Flotation

In the program of testing by G&T, a test was conducted on each of 71 samples using the same reagent regime. In these tests, the objective was to recover all the sulphides into a bulk concentrate. A nominal primary grind sizing of 75µm K80 was used. Only collectors, SIPX and 3418A, were added to the bulk roughing stage with lime also added in the regrinding stage. Since no depressants were added, beyond the elevated pH in the cleaner, higher recoveries but poorer concentrate grades were obtained as compared to the sequential flowsheet.

Upon analysis of the G&T results, it is clear that the different ore-types designated by Bear Creek Mining were useful to delineate metallurgical performance based on these results. A summary of average batch cleaner test performance, obtained from each ore-type, is provided in Table 13-8.

Table 13-8: Average Bulk Circuit Performance by Ore-type (based on G&T Data)

Oretype	No.	Ave Head Grade - % or g/t				Conc. Grade - % or g/t			Recovery - %		
	Samples	Pb	Zn	Ag	S	Pb	Zn	Ag	Pb	Zn	Ag
CSC	6	3.70	0.94	137	3.55	27.7	7.3	1110	82	68	78
FBS	26	1.83	1.59	96	3.32	15.8	12.5	1200	64	57	71
PM	15	1.37	2.32	62	5.87	10.4	11.8	444	76	49	75
QSB	7	1.02	0.18	75	1.67	13.9	0.9	12723	11	3	46
FeOx	4	0.52	0.09	71	1.20	9.8	2.9	2665	34	36	55
MnO	6	1.75	0.23	64	3.04	10.6	1.5	3564	5	5	31
PG	7	1.23	0.11	27	0.69	9.9	2.1	1277	5	10	25

Note: Ag grades are denoted in g/tonne, all other assays are in percent.

13.6.2 Whole Ore Cyanidation Tests

Dawson utilized a nominal primary grind sizing of 75µm K80 to conduct the cyanidation bottle roll tests. The 96 hour silver extraction and cyanide consumption data for the tests conducted by Dawson are summarized in Table 13-9

Table 13-9: Dawson Cyanidation Leaching Results

Composite	Ag Extraction	Ag Grade - g/tonne		NaCN
	96 hours - %	Residue	Calc. Hd	kg/t
1	56.9	99.0	229.0	4.5
2	58.8	110.0	266.0	4.6
3	54.4	105.0	230.0	3.4
4	70.5	41.0	139.0	3.0
5	64.6	53.0	150.0	4.6
6	79.1	30.0	143.0	5.7
A	48.8	63.0	122.0	4.0
B	39.9	61.0	100.0	7.4
C	56.9	36.0	93.0	4.9
D	50.1	60.0	110.0	4.3
E	45.1	52.0	94.0	2.8
F	25.5	140.0	188.0	2.9
Wt. Ave.	**54.1**			

On average, just over half the silver was extracted using cyanidation bottle roll tests. Silver extraction kinetics tended to be slow after 8 hours.

Cyanidation testing was only conducted on selected samples, generally of poor flotation response, in the G&T and SGS testing programs. However, results showed, that under conditions almost the same as used by Dawson, similar silver extractions were obtained by G&T and SGS on these samples. Using a weighted average, about 51 percent of the silver was extracted from the samples tested by G&T and SGS.

Average performance for each deposit and oretype is summarized in Table 13-10. Results show that samples that have poor bulk flotation response (FeOx, QSB, MnO and PG) tended to respond more favorably to cyanidation. It may be possible to take advantage of this by pursuing a flowsheet involving cyanidation of flotation tailings.

Table 13-10: Average Cyanidation Extraction by Deposit and Classification (G&T Data)

Zone	No. Samples	Wt. Ave. Ag Extraction - %
Deposit		
Este	9	38
Main	11	73
Minas	12	54
Oretype		
FBS	8	35
PM	3	26
FeOx	3	70
QSB	5	81
MnO	6	59
PG	7	79

Note: Silver extraction averages were weighted using silver head grades.

13.6.3 Cyanidation Leaching of Flotation Tailings

The opportunity to process Corani mineralization using a combination of flotation and cyanidation was only evaluated by Dawson Metallurgical Laboratories in 2006. In that program, Dawson tested only two samples using this methodology. The silver recovery was superior to that obtained by the use of either flotation or cyanidation alone. Further testing would be required to determine if this flowsheet option has merit across a broad range of samples.

13.7 OPTIMIZED AND CURRENT CONTINUOUS PREDICTIVE METALLURGICAL MODEL

Corani is a silver-lead-zinc deposit that exhibits a high degree of variability with respect to metallurgical test results. Most of the metallurgical testing was based on composites constructed by geological classification. Some of the classifications were suitable for categorizing metallurgical response, for example, composites of CSC (Coarse grained silica-sulfide-celadonite) consistently performed very well. However, one of the categories which represented nearly half of the resource tonnage (FBS) had large ranges in metallurgical performance.

Global Resource Engineering (GRE) and Tom Shouldice (TS) conducted an evaluation of the geometallurgy for the Corani Project to address recommendations made by Blue Coast Metallurgy (Blue Coast Metallurgy, 2011) in conjunction with the test results described above regarding metal recovery projections. At the time of the 2011 Feasibility Study, 4 geometallurgical types had been identified and average recoveries were assigned to these material types for the purpose of reserve calculations and mine planning, as described above. These geometallurgical types represented varying degrees of oxidation and as a result, distinct differences in metallurgical behavior were observed between types. However, within each assigned met type, large ranges in recovery were observed. In order to further refine metallurgical recovery projections, the available metallurgical, mineralogical, geological and spatial data were extensively reviewed.

A statistical analysis was completed by GRE of Denver indicating that several measurable geological parameters could be used to make metallurgical predictions.

A number of advanced statistical methods were used in order to better understand the drivers of metallurgical behavior and identify data within the existing drillhole database that could be used to predict recovery for the deposit. These methods included cluster analysis and recursive partitioning. Since the conceptual understanding of the Corani deposit relies on various types of mineralization, it is expected that characteristics that drive recovery within one mineralization may be different or have a different magnitude in another mineralization. This was previously recognized from a qualitative standpoint (i.e. the qualitative assignment of Met types I through IV). The advanced statistical methods allow for a quantitative assessment of this behavior.

Using these methods, zinc grade, elevation, oxide minerals, pyrite and the form of lead (galena vs. phosphate) were identified as the best predictors of lead recovery; zinc grade, elevation, copper grade, and pyrite were identified as the best predictors of zinc recovery; and lead recovery was identified as the best predictor of silver recovery in the lead concentrate. In order to model the complex relationship between these parameters and recovery, multivariate adaptive regression splines were used to develop a model capable of predicting recoveries based on this data. Multivariate adaptive regression splines are a form of multiple regression that has the flexibility to model non-linear relationships between variables. The method also utilizes data partitioning which allows the model to identify characteristics within the dataset that potentially lead to different outcomes.

Though this method appears complex, it combines the benefits of a model based on ore type characterization (Met Types I –Met Type IV) and a model using regression. It partitions data based on similarities (similar to ore type characterization) and develops a regression model for predicting a continuous recovery result (similar to regression modelling). The result is a single model equation that can be used to predict recovery across all ore types.

In order to select a robust model, subsets of the dataset were selected randomly and used to develop a calibrated model; the remaining data was used as a “blind” dataset to validate the model. This was done 1000 times and the best performing model (based on calibration and validation results) was selected as the final model.

To create an overall balance for lead, silver and zinc; the concentrates were assumed to be constant grade. The grades of the concentrates were determined by the averages of locked cycle testing.

The sections below give a brief summary of each model used for recovery predictions. Additional details regarding the complete statistical analysis were reported by GRE in a separate document.

13.7.1 Recovery of Lead to Lead Concentrate

The model for predicting for lead recovery to the lead rougher concentrate included the following variables:

- Zinc feed grade
- Mine elevation
- Geological logging estimates of galena, goethite, manganese oxide and pyrite.

To ensure the robustness of the selected regression parameters, 70% of the data was randomly selected (the training dataset) and used to estimate the regression parameters; the remaining 30% (the validation dataset) was used to validate the model. This process was repeated 1000 times. The model with the most stable regression parameters and best calibration-validation performance was selected. The R^2 of the training dataset for the selected model was 0.77, and the R^2 of the validation dataset was also 0.77. The R^2 considering all of the data (both validation and calibration data) was 0.77. The performance of the model for the validation dataset and across all data is shown in Figure 13-12Figure 13-12. The colors in the figure represent the mineralization types assigned to samples using the prior geometallurgical approach.

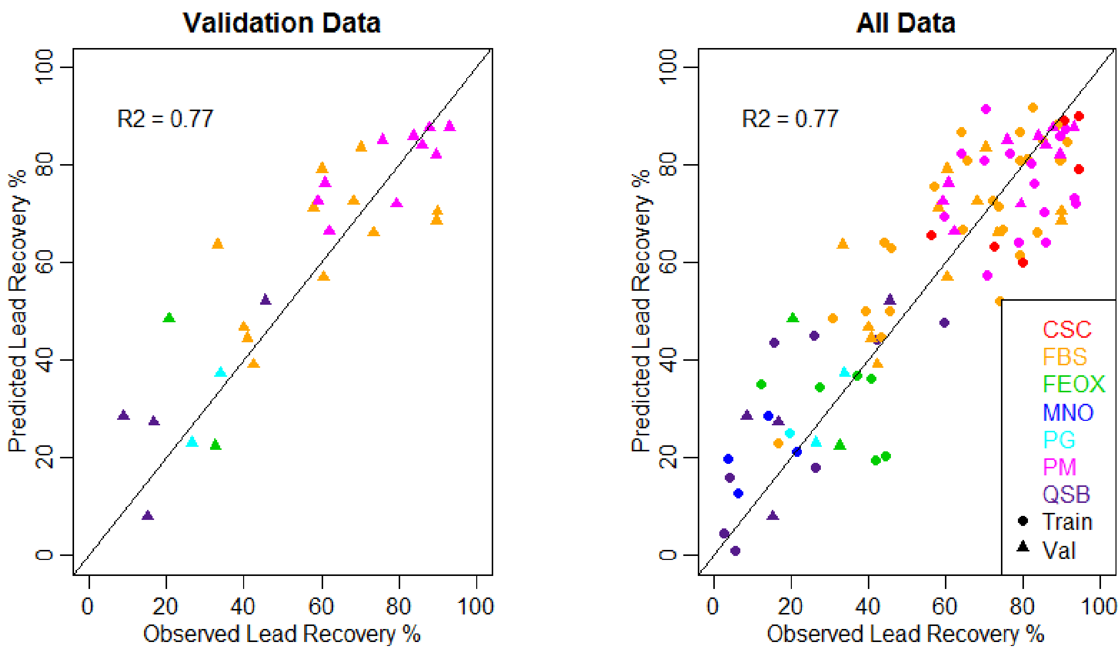


Figure 13-12: Selected Model with Training Data and Validation Data

Predicted lead rougher recoveries were compared to the final lead concentrate recoveries observed during lock cycle tests for the 12 samples in which both lock cycle test and batch test results were available. The results indicate that the model developed using lead rougher results from batch testing is a good predictor of final lead concentrate recovery. A comparison of observed lead rougher recoveries from batch testing, predicted lead rougher recoveries from batch testing, and observed final lead concentrate recoveries is shown in Figure 13-13.

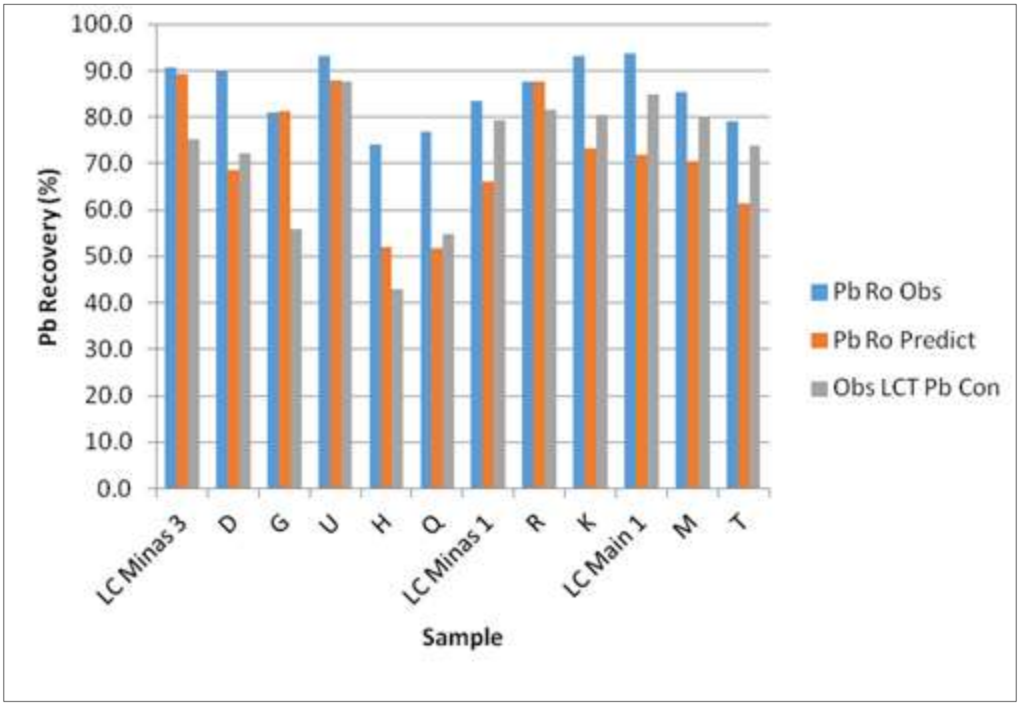


Figure 13-13: Comparison of Predicted and Observed Lead Recovery from LCT

Based on the LCT analysis, the final lead recovery to the lead/silver concentrate was calculated using the following equation:

$$Pb\ Recovery\ to\ Final\ Lead/Silver\ Concentrate = 61.9 - 40.9 * max(0,0.57 - zinc) + 7.7 * max(0, galena - 0.38) + 45.4 * max(0,0.37 - goethite) - 0.12 * max(0, elevation - 4891) + 32.9 * max(0,0.27 - MnOxi) - 6.21 * max(0, Pyrite - 1.07) - 16.4 * max(0,1.07 - Pyrite)$$

13.7.2 Silver Recovery

A cursory analysis of silver revealed that the strongest predictor of silver recovery was lead recovery. The relationship between silver and lead recovery to the lead rougher is shown below. Though one major outlier does exist, due to the large number of samples, the equation is not strongly influenced by this observation. This data is displayed in Figure 13-14 along with the resulting model results.

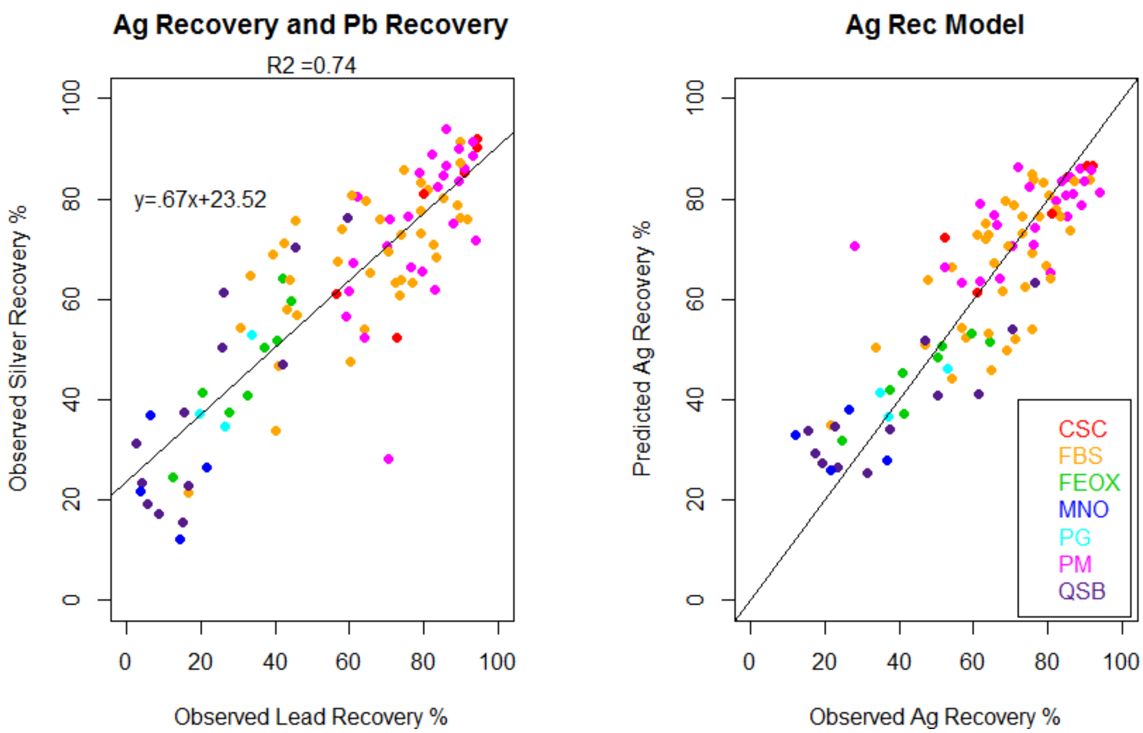


Figure 13-14: **Silver Recovery to Lead Rougher**

The predicted silver recoveries were compared to LCT test results for samples where batch results and LCT results were available. Silver recovery to the lead rougher was estimated using predicted lead recovery to the lead rougher. The results show that, on average, the difference between the predicted silver recovery to the lead rougher and the observed silver recovery to the final silver/lead concentrate is on average, 11.5%. This difference is assumed to represent the loss of silver during the lead cleaning stage. The final predicted silver recovery to the lead concentrate was estimated by subtracting 11.5% from the predicted silver recovery to the lead rougher. This yields an estimate of final silver recovery that closely matches the observed LCT test results (Figure 13-15).

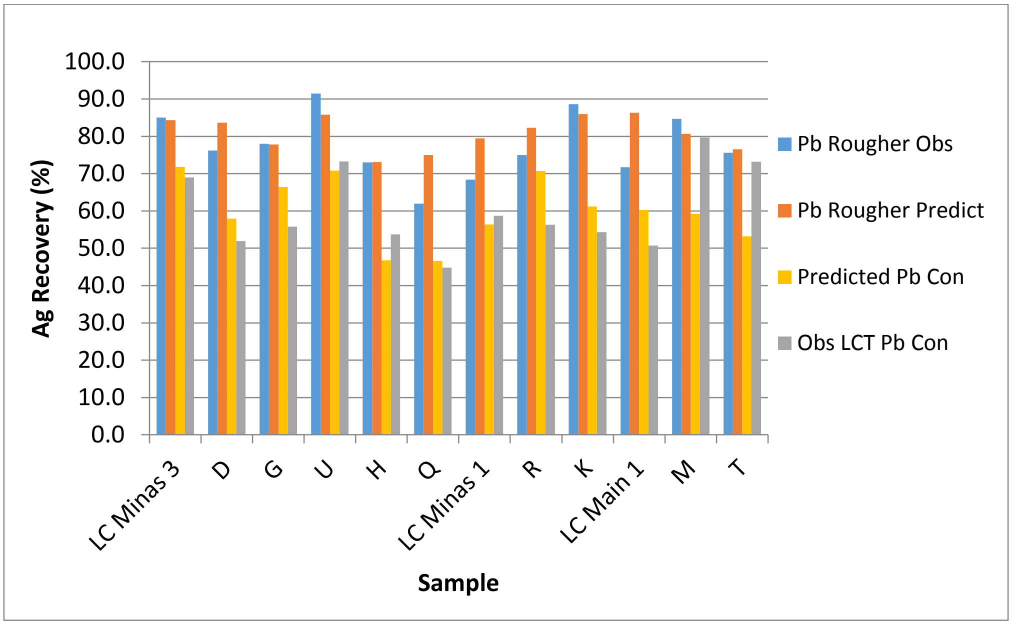


Figure 13-15: **Comparison of Predicted and Observed Silver Recovery to Lead Concentrate**

Based on the LCT analysis, the final silver recovery to the lead/silver concentrate was calculated using the following equation:

$$Ag\ Recovery\ to\ Final\ Lead/Silver\ Concentrate = 0.67 * Predicted\ Lead\ Recovery + 12$$

This is the same equation presented above (see Figure 13-14), for predicting silver recovery to the lead rougher, with the intercept adjusted to account for the loss of silver during the cleaning stage. This equation was used to assign final silver recoveries for the lead/silver concentrate to the Corani block model.

13.7.3 Zinc Recovery

Regression analysis for zinc was performed using“total flotable zinc”, which is the sum of the zinc that reports to the lead rougher and the zinc that reports to the zinc rougher during batch flotation testing. Non-flotable zinc which is zinc that does not float in the lead rougher or zinc rougher, reports to the final zinc rougher tail. The difference between total flotable zinc and zinc recovered to the final zinc concentrate is the sum of the zinc that is recovered in the final lead/silver concentrate and the zinc removed during the zinc cleaning stage (cleaner tail). A schematic indicating the modeled recovery (“total flotable zinc”) is shown in Figure 13-16.

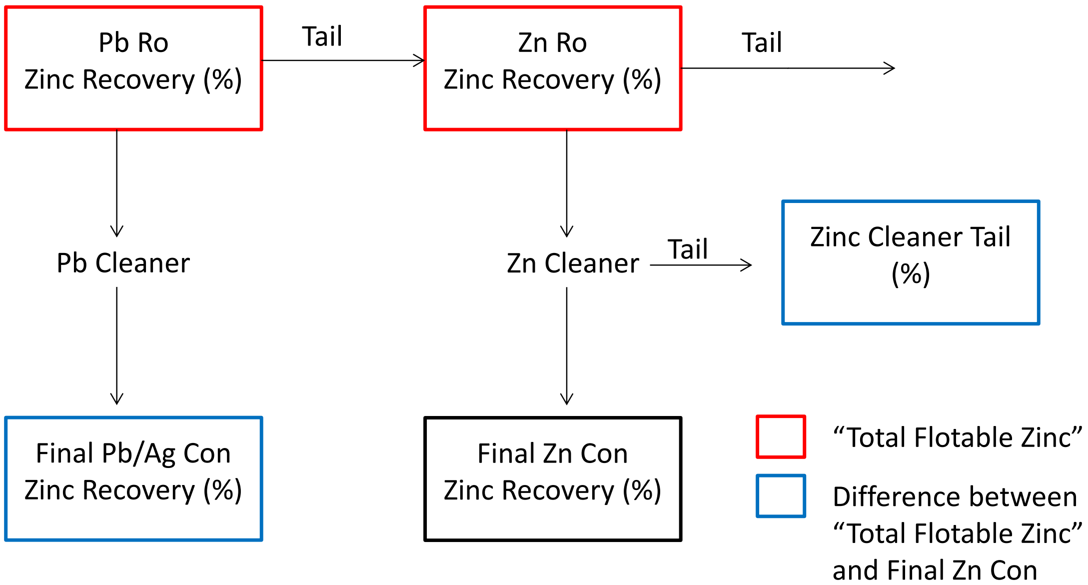


Figure 13-16: Schematic Indicating “Total Flotable Zinc” for Batch Tests

Based on the results from the above analysis, tests with lime additions to the lead rougher greater than 2,000 g/t were removed from the model dataset. From this test database, the test representing the best total zinc recovery result was selected for each sample. Also, as was done for the lead analysis, samples representing composites from multiple drillholes were removed from the potential training dataset. The resulting dataset included 58 of the original 72 samples considered for analysis.

In order to select the most robust model, a cross-validation process similar to the process employed for lead was used to develop the model for total flotable zinc. From the dataset, 70% of the samples were randomly selected to fit the model and the remaining 30% were used to validate the model. This process was completed 1000 times with 1000 different permutations of the dataset.

The independent variable combinations selected most frequently for the 1000 permutations were used for the final recovery equation. This combination had the highest validation and training set R^2, indicating the most robust model. The final total zinc rougher recovery model included zinc grade, elevation, copper, and pyrite in the following form:

$$
\begin{aligned}
&Total\ Flotable\ Zinc \\
&\qquad = 93.9 - 50.6 * max(0{,}1.02 - zinc) - 0.15 * max(0, elevation - 4901) - 5.4 \\
&\qquad * max(0, pyrite - 1.9) - 11.2 * max(0{,}1.9 - Pyrite) + 104.1 \\
&\qquad * max(0, copper - 0.03) + 1620.2 * max(0{,}0.03 - copper)
\end{aligned}
$$

The R^2 of the training dataset was 0.92, and the R^2 of the validation dataset was 0.96. The R^2 considering all of the data was 0.92. Once the model was limited to predicted zinc recoveries between 0 and 100%, the training R^2 improved to 0.93; the overall R^2 remained the same.

The performance of the model for the validation dataset and across all data is shown in Figure 13-17.

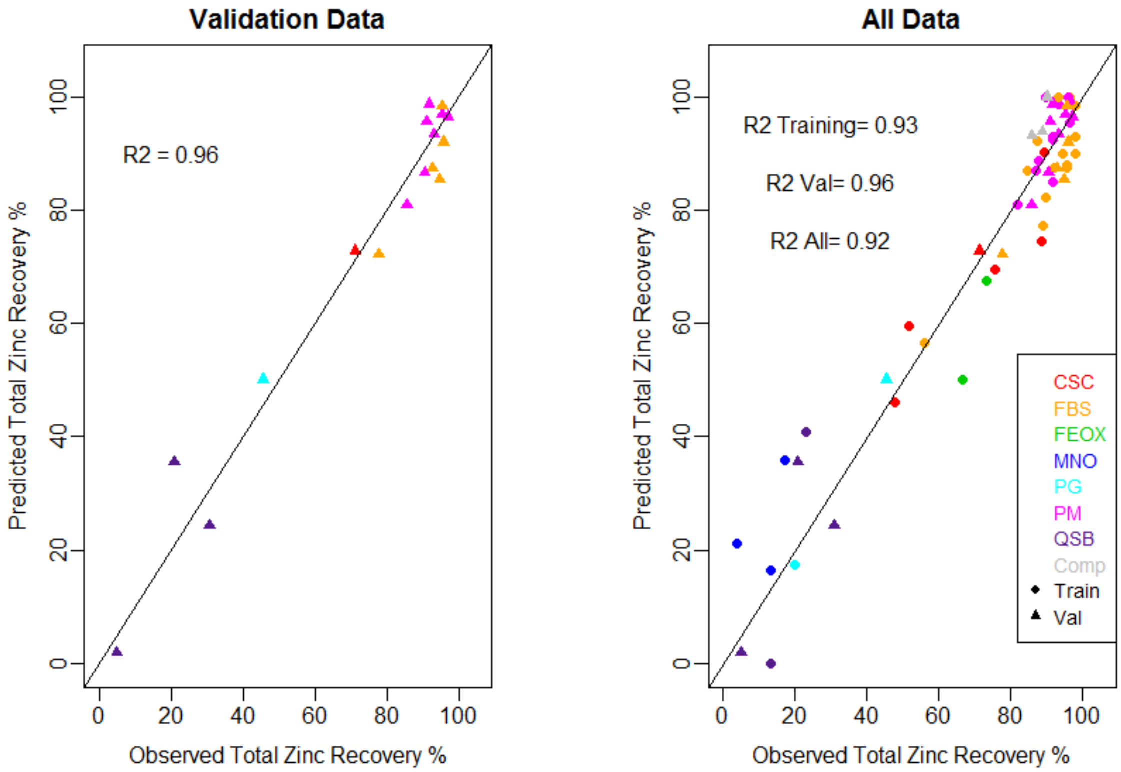


Figure 13-17: Selected Model with Training Data and Validation Data

The results from the model were compared to the lock cycle test results for the 8 samples where zinc recovery was reported. The average difference between the lock cycle tests and the predicted total flotable zinc was 22.5%. This difference is anticipated to represent the zinc that reports to the final lead concentrate and the zinc that is lost during the zinc cleaning stage.

Three LCT test results were notable for lower-than-expected zinc recovery. The difference between the predicted total zinc recovery and the observed LCT recovery for samples K, R, and LC Main 1 were on average 34.5%, while the difference between the LCT zinc recovery and predicted total zinc recovery for the remaining samples was only 15.2%.

A comparison between the calculated final zinc concentrate recovery from batch tests is shown in Figure 13-18. The LCT result for these samples is much lower than what was achieved in batch testing. The remaining samples show a good match between the LCT result and batch testing. Considering this, the adjustment of 22.5% to the predicted total flotable zinc may be an over-estimate of the zinc lost to the lead rougher and the zinc lost to the zinc cleaner tail.

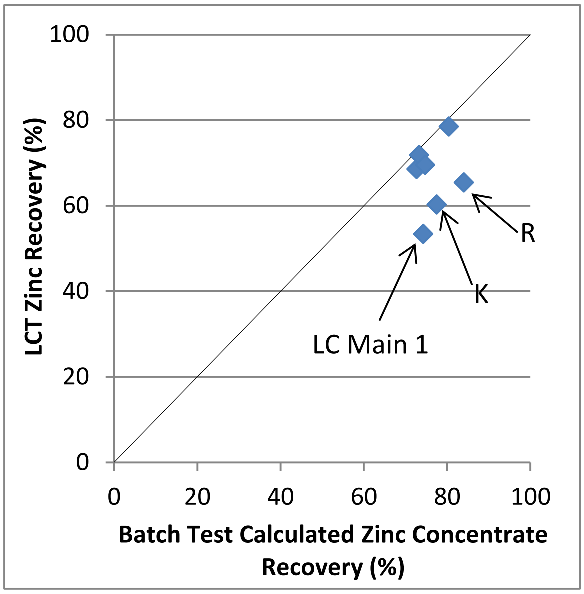


Figure 13-18: Comparison between Batch and LCT Results

A final zinc concentrate recovery was predicted based on the total predicted zinc recovery (Zn Ro+ Pb Ro) minus 15.2%, which is the average difference between the predicted recovery and the LCT zinc concentrate recovery result (not including samples R, K, and LC Main 1). The LCT zinc concentrate recovery, predicted total recovery, and predicted recovery to zinc concentrate are shown in Figure 13-19.

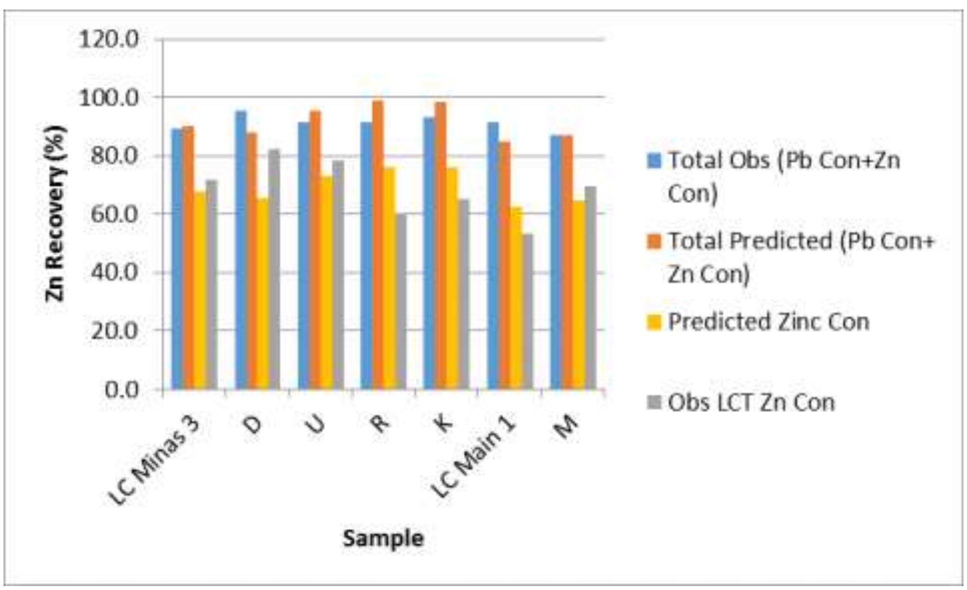


Figure 13-19: LCT/Predicted Zinc Recovery Comparison

13.7.4 Zinc Recovery to Final Zinc Concentrate

Based on the LCT analysis, the final zinc recovery to the zinc concentrate was calculated using the following equation:

$$\begin{aligned} Zinc\ Recovery\ to\ Final\ Zinc\ Concentrate \\ &= 78.7 - 50.6 * max(0{,}1.02 - zinc) - 0.15 * max(0, elevation - 4901) - 5.4 \\ &* max(0, pyrite - 1.9) - 11.2 * max(0{,}1.9 - Pyrite) + 104.1 \\ &* max(0, copper - 0.03) + 1620.2 * max(0{,}0.03 - copper) \end{aligned}$$

This is the same equation already presented for predicting total flotable zinc, with the intercept adjusted to account for the difference between total flotable zinc and final zinc recovery to the zinc concentrate (~15.2%). This equation was used to assign final zinc recoveries for the zinc concentrate to the Corani block model.

13.7.5 Zinc Displacement to Lead/Silver Concentrate

Based on the LCT results, the average displacement of zinc in the final lead/silver concentrate is 9%. Batch test results revealed that in general, the lead cleaning stage was relatively effective in removing zinc from the final lead concentrate. Of the 97 batch tests used to create the lead rougher recovery model, 21 reported displacements for zinc in the lead concentrate following the cleaning stage. The average zinc displacement among these tests was 9.3%. With the exception of 2 outliers, most samples have zinc recoveries below 15%. Over 80% of the samples have zinc recoveries to the lead cleaner of less than 10%.

From these results, a recovery of zinc to the final lead concentrate was estimated to be roughly 9%.

13.7.6 Silver Recovery to Zinc Concentrate

Based on the tests used in the total flotable zinc model analysis, there does not appear to be a strong relationship between silver recovery to the zinc concentrate and zinc grades. Therefore, the zinc concentrate was assumed to contain an average silver grade. The grade was an average of the values determined in the locked cycle testing. Silver recovery was then calculated once the zinc recovery and zinc concentrate mass was estimated from the zinc model.

13.7.7 Model Output Results

The models for recovery, along with the concentrate grade estimates, were input into the block modelling process to optimize the mining process. For the purposes of this study, the lead concentrate grade was assumed to be constant at 56.6% (average of all locked cycle tests). Similarly the zinc concentrate grade was assumed to be 52.9% and contained 385 g/tonne silver (averages of all locked cycle tests).

The block model simulation with the metallurgical models in place resulted in several pit shells at different revenue factors. Using the mine plan with revenue factor of 100, the estimates of metal recoveries were determined by mine operating year. These values are displayed in for the lead concentrate and zinc concentrate in Table 13-11 and Table 13-12, respectively. Compared to the previous study (M3, 2011), the average lead recovery was reduced by 8% while the silver and zinc recovery both increased by 8%.

Table 13-11: Lead Concentrate Grades and Recoveries by Mine Schedule

Production Year	Tonnes (000)	Feed Grade (% or g/t)		Grade (g/at or %)		Recovery (%)	
		Ag	Pb	Ag	Pb	Ag	Pb
Year 1	5,675	96	1.17	5772	56.6	67	69
Year 2	7,744	84	1.43	3382	56.6	70	69
Year 3	7,897	73	1.20	3390	56.6	70	71
Year 4 to 5	15,745	80	1.12	3863	56.6	71	74
Year 6 to 10	39,393	55	0.98	4380	56.6	62	45
Year 11 to 18	69,120	27	0.61	2423	56.6	69	72
LOM	137,698	52	0.91	3417	56.6	67	63

Note: Ag grades are denoted in g/tonne, all other assays are in percent. Mine plan revenue factor of 100.

Table 13-12: Zinc Concentrate Grades and Recoveries by Mine Schedule

Production Year	Tonnes (000)	Feed Grade (% or g/t)	Grade (g/at or %)		Recovery (%)	
		Zn	Ag	Pb	Ag	Pb
Year 1	5,675	0.48	385	52.9	2	54
Year 2	7,744	0.88	385	52.9	5	63
Year 3	7,897	0.85	385	52.9	6	69
Year 4 to 5	15,745	0.97	385	52.9	7	76
Year 6 to 10	39,393	0.38	385	52.9	2	49
Year 11 to 18	69,120	0.50	365	52.9	7	56
LOM	137,698	0.59	377	52.9	5	60

Note: Ag grades are denoted in g/tonne, all other assays are in percent. Mine plan revenue factor of 100.

The result of this work is a reliable predictive geometallurgical model that has significantly improved predictions of metallurgical recovery which can be used in mine planning and grade control (Figure 13-20). The previous geometallurgical model relied on assigning average recovery values to large volumes of material based on ore type. Anticipated recoveries based on the new model closely fit recoveries observed during metallurgical testing. The recovery projections (based on the new model) for over half the samples tested is within +/-9% of the observed test recovery for lead, within +/-7% of the observed test recovery for silver, and within 5% of the observed test recovery for zinc.

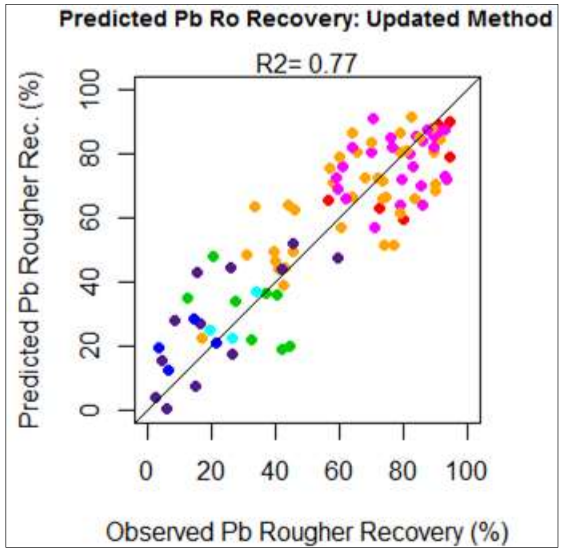


Figure 13-20: Predicted vs. Observed Recoveries for New Predictive Geometallurgical Model

14 MINERAL RESOURCE ESTIMATES

A block model of the Corani deposit was developed as the basis for determination of the Mineral Reserves and Mineral Resources. This section summarizes the development of the block model as well as the development of the mineral resource. BCM's block model was updated by GRE to include drillhole data gathered since the last model was created. The updated data also included new geologic cross sections and updates to the mineral domains. Also, per previous authors' recommendations, the density model was updated with a statistical analysis of the relationship between metal grades and specific gravity. The drillhole database was updated with geologic logs and assays of primary recovery indicators: copper, goethite, manganese oxide, pyrite, and galena. These new grade labels were modeled along with the economically viable metals in the block model.

14.1 BLOCK MODEL

The block model was developed using blocks sized 15 x 15 m in plan and 8 m high. The selection of the 8 m bench height was based on a study of dilution versus bench height and consideration of the mining equipment that might be used in the future for open pit production.

The model area contains all three areas: Main, Minas, and Este. The model is large enough to contain all reasonable open pit configurations for the three resources areas. The total model size and block size are summarized in Table 14-1 below.

Table 14-1: Block Model Information

Corani Block Model, Size and Location			
Block Size	X	15	meters
	Y	15	meters
	Z	8	meters
Number of Blocks	X	184	meters
	Y	214	meters
	Z	64	meters
Model Limits	X	314,745	317,505
	Y	8,446,250	8,449,460
	Z	4,618	5,130

The model was assembled in the Universal Transverse Mercator (UTM) coordinate system and is parallel to the UTM grid. Topographic information was assigned to the model based on topographic maps provided by BCM. The topography maps were consistent with field observations and the elevations of the drill holes at their coordinates.

14.1.1 Rock Type Boundaries

The Corani deposit lithology is comprised of Tertiary pre-mineral tuff and post-mineral tuff overlying Paleozoic sedimentary units.

The mineralized volcanic tuff was emplaced prior to mineralizing events (pre-mineral), and contains the economic minerals of interest. Post-mineral volcanic tuff is barren, and was emplaced subsequent to the mineralizing events. The mineral reserves and resources are completely contained within the pre-mineral tuff.

The basement sediments at Corani are generally barren, although there are local occurrences of mineralization within those units. Potential exists for additional mineralization to be found in the sediments. However, for this block model they were assumed to be barren and grades were not estimated.

Blocks were coded as: 11 = post-mineral volcanics, 20 = pre-mineral volcanics, 31 = sediments. Unassigned blocks do exist outside of the interpreted area. The blocks that did not receive a rock type assignment were assigned a default average density in case they were mined as waste within the Whittle optimized pit shell or mine plan.

As part of the resource estimation process, BCM updated cross sections of the drill hole data in each of the deposits and redrew the mineralization boundaries for each of the mineralization domains within the Corani pre-mineral rock units, with the intent of using the solids as domains for grade and recovery estimation.

GRE used the drill hole data, cross sections with geologic mineralization polygons, geologic logs, and a previously developed and coded block model containing codes for sediments, pre- and post-mineral tuff, and overburden/bofedal provided by BCM. This data and the previous block model codes were verified, and solids were created from mineralization polygons. Geologic log data were also entered in electronic form. The solids were extrapolated a sufficient distance beyond the outside drill holes so as not to limit the ability to assign rock type or grade when grade estimation was completed.

The mineralization solids were checked by generating cross sections through each of the three areas. The solids were initially used to assign mineralization type to the metallurgical test data. The geometallurgical data analysis is described in more detail in Sections 7 and 13 of this report. Results of that work demonstrated that the mineralization solids were not good predictors of recovery, and that a continuous statistical model could be used with better results. Geostatistics of the assay data from the mineralization domains were evaluated, and it was determined that they could be lumped together to form larger, more-contiguous solids, which were used during the estimation of silver, lead, and zinc grades. These groupings generally represent weathered and oxidized rocks, transition or supergene-enriched mineralization, and unaltered sulfide mineralization. The geometallurgical analysis resulted in recovery equations for silver, lead, and zinc, which were estimated for each block of the block model.

Figure 14-1 is an example of the geologic assignment to the model blocks on cross section.

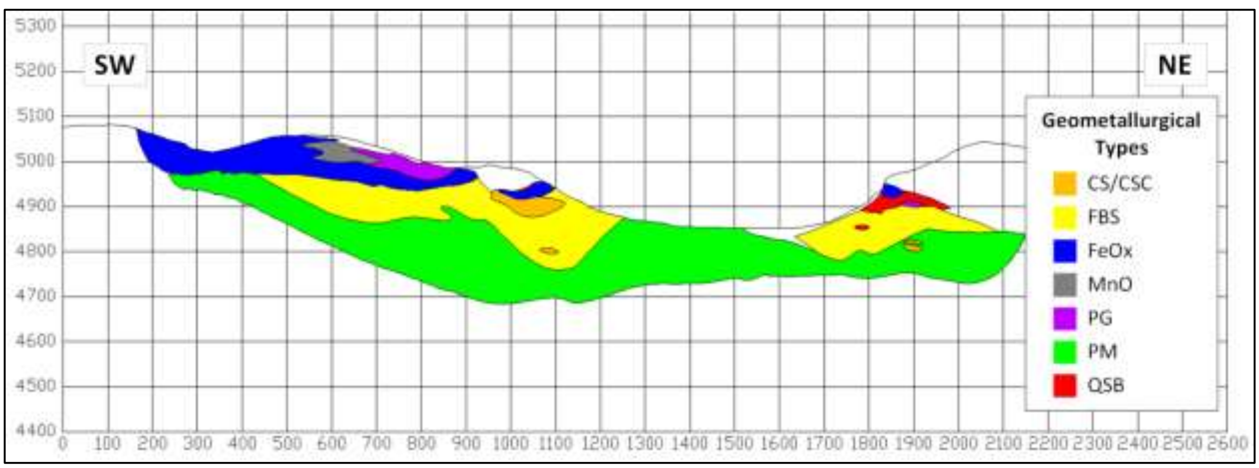


Figure 14-1: Geologic Assignment to Model Blocks

14.1.2 Density Assignment

Density was based on the combined grade of silver, lead, and zinc. The silver was converted to a percent assay to get common units with lead and zinc. The combined grade and density showed a break at 0.9381%. Blocks with a combined grade under 0.9381% were assigned a density of 2.31 t/m^3, and blocks with a combined grade greater than or equal to 0.9381% were assigned a density of 2.43 t/m^3. Post-mineral tuff with no metal grades to analyze was assigned an average density of 2.3 t/m^3 and non-tuff material was assigned an average density of 2.53 t/m^3.

Table 14-2: Assigned Block Densities

Rock Type	Grade	Density
Pre-Mineral Tuff	< 0.9381	2.31
Pre-Mineral Tuff	>= 0.9381	2.43
Post-Mineral Tuff	Not Applicable	2.3
Other Materials	Not Applicable	2.53

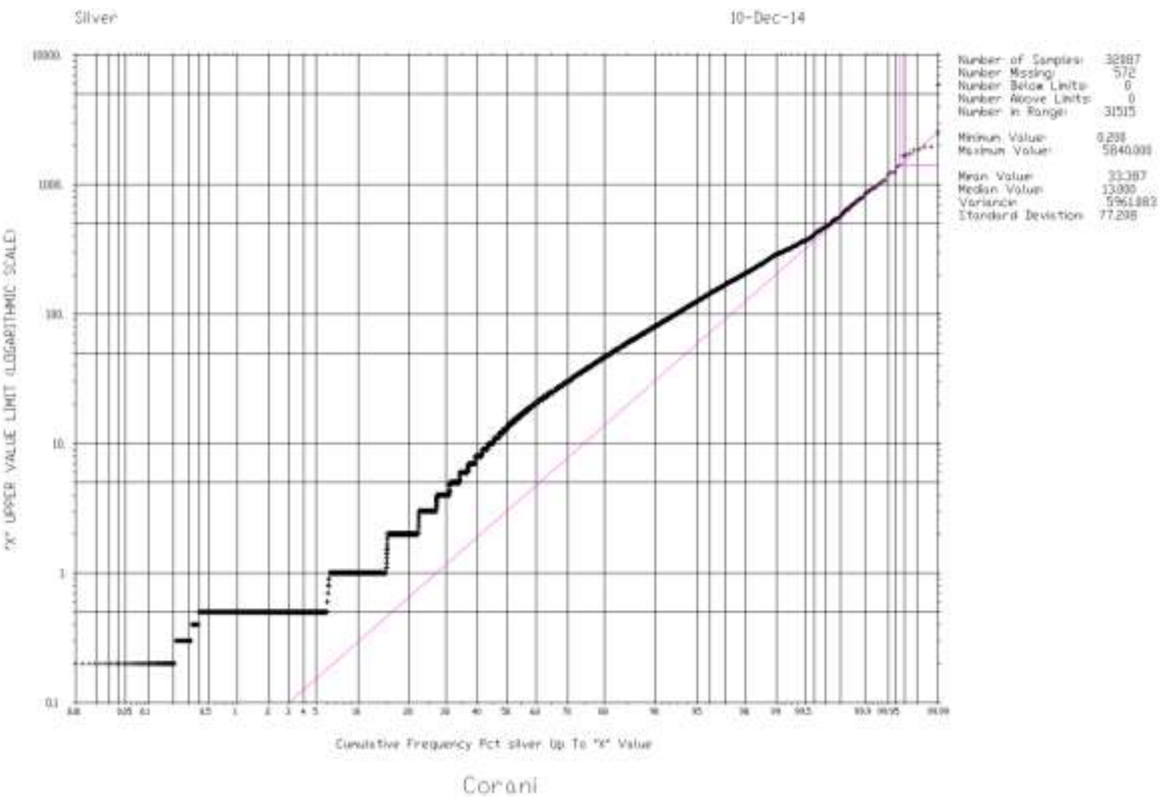
14.1.3 Block Grade Estimation

Block grades were estimated for silver, lead, zinc, copper, goethite, manganese oxide (MnOx), pyrite, and galena. The estimation zones were based on groups of the mineral domains. Coarse sulfides (CS/CSC), fine black sulfides (FBS), pyrite-marcasite (PM), and tetrahedrite (TET) were the first group. Iron oxides (FeOx), manganese oxide (MnOx), and plumbogummite (PG) were the second group. Quartz-sulfide-barite (QSB) was in its own, third group. These members of the groups were selected on the basis of their similar characteristics and location relative to each other. Although QSB had similarities to the first group, it was geographically distinct from the sulfide-rich zones.

Prior to compositing, individual assay values were cut in order to limit the influence of high-grade outliners on the block grade distribution. The cumulative frequency plot (CFP) of silver has a break at 1410 ppm and ten silver samples were capped at 1410 ppm (Figure 14-2). Although there was a definite break on the copper CFP, it was only used as a recovery indicator, and the break was well above the indicator threshold. Thus, there was no need to cap copper assays. There was no cumulative frequency analysis on the goethite, MnOx, pyrite, and galena labels since there was no assay on those labels.

The cumulative frequency plot of lead manifests itself as a continuous population with no discernable break. For this reason, lead was not capped before compositing. The cumulative frequency plot of zinc shows a similar continuous population similar to lead, and therefore it was not capped before compositing.

Figure 14-2: Cumulative Frequency Plot of Silver Grade (g/tonne) with Break and Capping



The Corani drill holes were composited to nominal 8 m down-hole or length composites respecting rock type. A technique was used that changed the composite length slightly within each rock type, in order to have composites of equal length that respected lithologic boundaries.

The pre-mineral tuff is the unit of importance for potentially economic mineralization, and can be used as the example of the compositing process. Within each drill hole, the length of the intercept of pre-mineral tuff was determined and that length was divided into equal-length composites of approximately 8 m in length. All composites within a rock type in the hole have the same length. That length could be something slightly more or less than the 8 m target value, in order to define the rock type into an integer number of composites. This process eliminates the existence of a short or partial composite at the rock boundary.

The 8 m value was selected based on a grade-versus-count evaluation of alternative composite lengths, to determine whether there was an improvement in ore selectivity with smaller composites versus the cost of production, compared with more cost-efficient use of longer composites and corresponding higher bench heights. The final selection of 8 m was also guided by the expected production rates and mine loading equipment that might be employed for production.

Composite samples for silver, lead, zinc, copper, goethite, MnOx, pyrite, and galena were loaded into SAGE 2001 for variogram analysis and modeling. The composites were divided into groups for each deposit (Main, Minas, and Este) and subdivided into mineral groups. Group 1 consists of the following mineral domains: 1 – CS – Coarse Sulfide; 2 – CSC – Coarse Sulfide and Celadonite; 3 – FBS – Fine Black Sulfide; 8 – PM – Pyrite Marcasite; and 10 – TET – Tetrahedrite. Group 2 consists of these mineral domains: 4 – FeOX – Iron Oxide; 6 – MnO – Manganese Oxide; and 7 – PG – Plumbogummite. Group 3 consists of 9 – QSB – Crystalline quartz sulfide barite. These groups were created based on the relative locations of the mineral domains to each other. This created 9 composite groups for the 8 grade labels, resulting in 72 total variograms to model.

It was determined that a correlogram created the best fit for this set of data. Downhole correlograms were used to determine the nugget for each composite sample group. Experimental correlograms had search parameters set for 30-degree increments for azimuth and 15-degree increments for dip. Omnidirectional correlograms were created for grade-zones that did not have a distinct directional trend. These particular grade-zones were modeled with isotropic parameters. The rest were modeled directionally to create anisotropic parameters for block modeling.

Typically, the general trend of all the estimation zones followed the trend of the contact between pre-mineral tuff and post-mineral tuff, which is 300 to 330 degrees azimuth and dipping 15 to 30 degrees down. This is in agreement with previous authors and BCM geologists. Silver, with a mobility different from lead and zinc, was estimated across mineral domain boundaries. The evidence of high silver grades in areas of expected low sulfidization and a smooth gradient of grades across mineral boundaries points to this as an appropriate modeling method. The model chosen for all grades was IDP, specifically ID3 for silver and ID2.5 for all others. This model was picked to model the high local variability of the drillhole data. By adjusting the power, a satisfactory level of smoothness was applied to the grade model. The IDP model was compared to a kriged model, and IDP was chosen due to the kriged model's failure to highlight the highest and lowest grades in the population.

Figure 14-3 through Figure 14-25 present the correlograms for the three economically viable metals in each deposit zone and mineral domain (where applicable). Table 14-3 through Table 14-12 show the parameters for input to the modeling software for all eight grade labels in the block model. These were taken from the variography of the composite data. Notably, the search range was increased from the previous study. The range from the variography analysis showed that the model has a range of 180 m in two zones and 170 m in another zone for the silver models. Since silver is the primary metal in this project, a search range of 180 m was used for all grade labels in order to preserve uniformity of the number of blocks modeled.

Figure 14-3: Silver Correlogram - Este

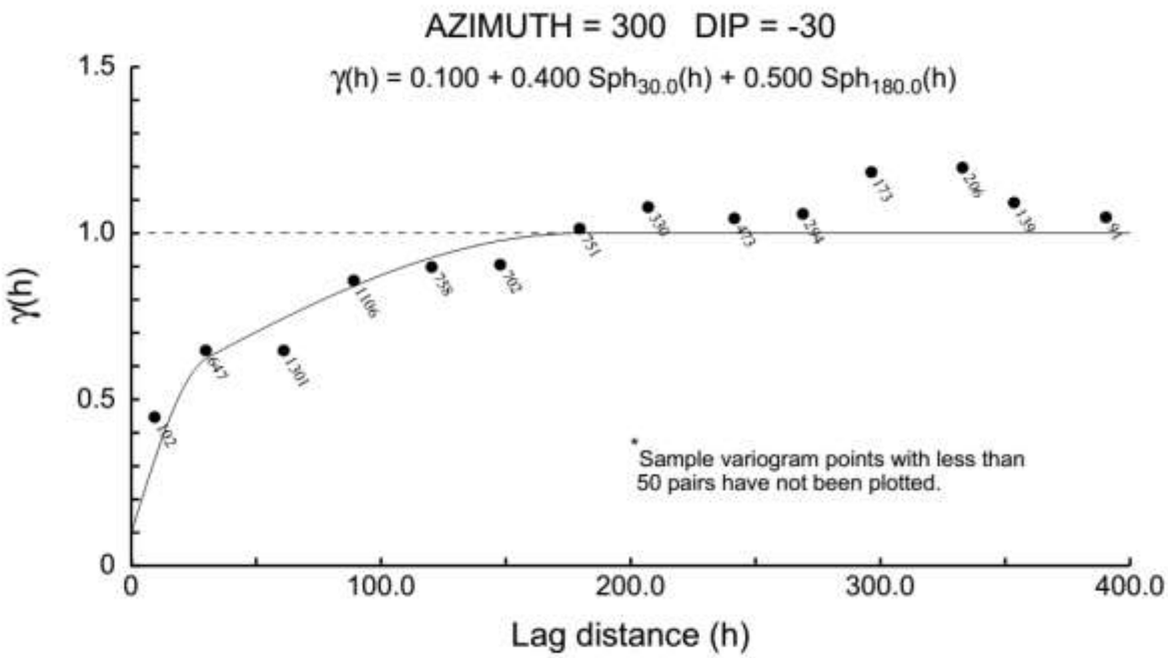


Figure 14-4: Silver Correlogram - Main

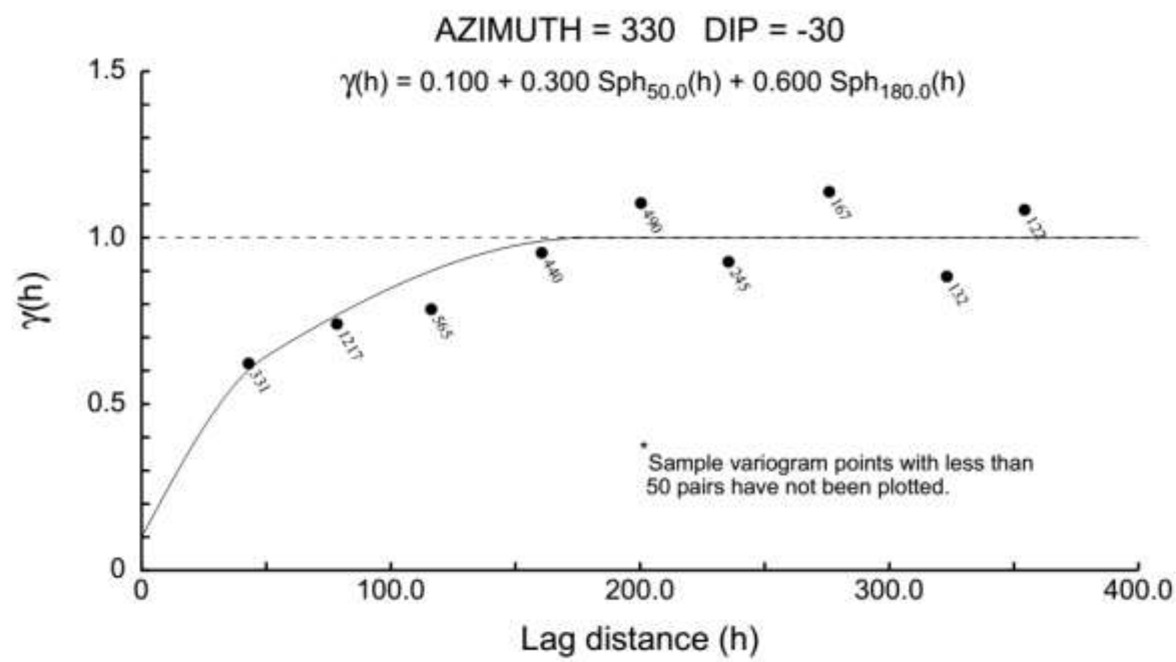


Figure 14-5: Silver Correlogram - Minas

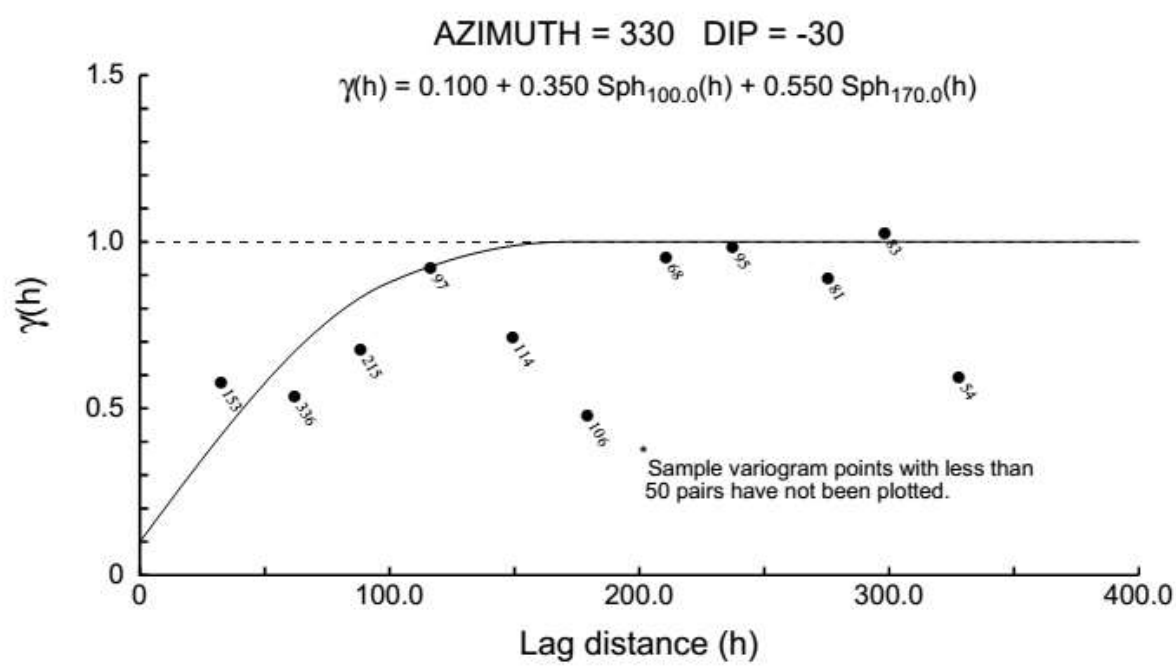


Figure 14-6: Lead Correlogram – Main, Mineral Group 1

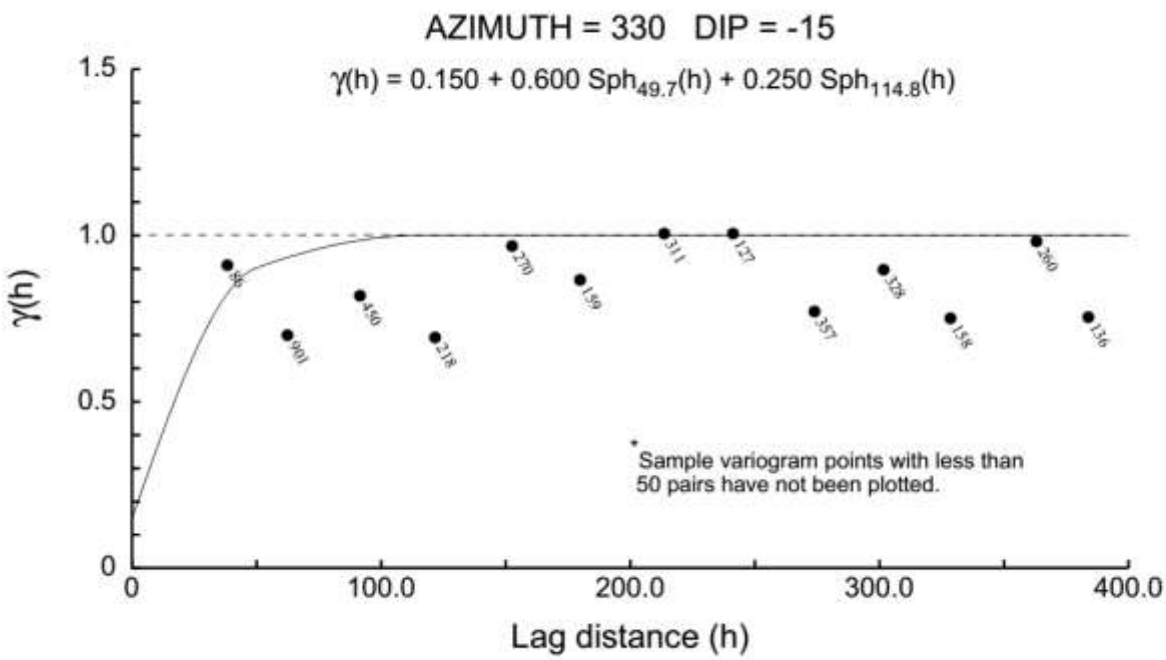


Figure 14-7: Lead Correlogram – Main, Mineral Group 2

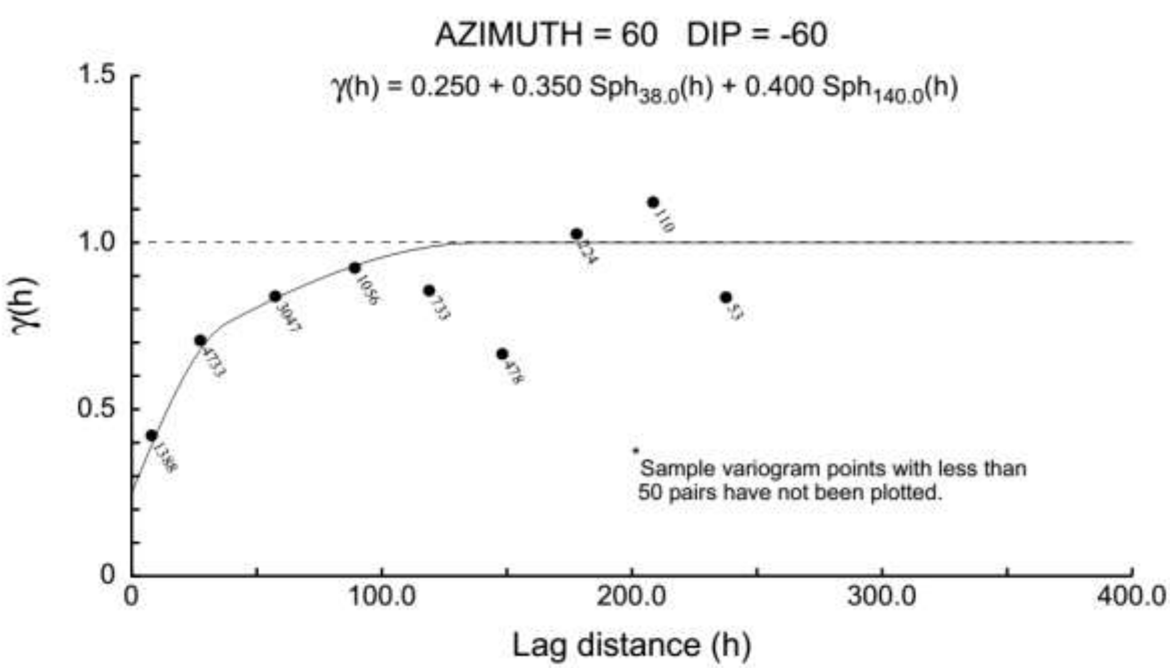


Figure 14-8: Lead Correlogram – Main, Mineral Group 3

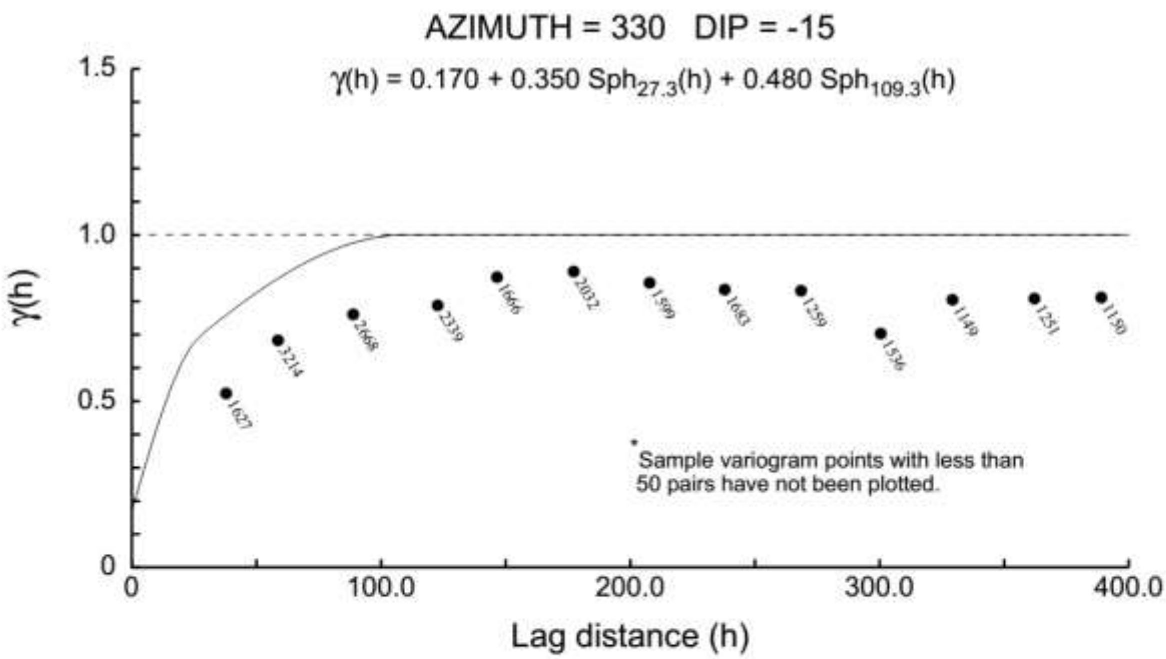


Figure 14-9: Lead Correlogram – Main, Mineral Group 3, Strong Model Direction

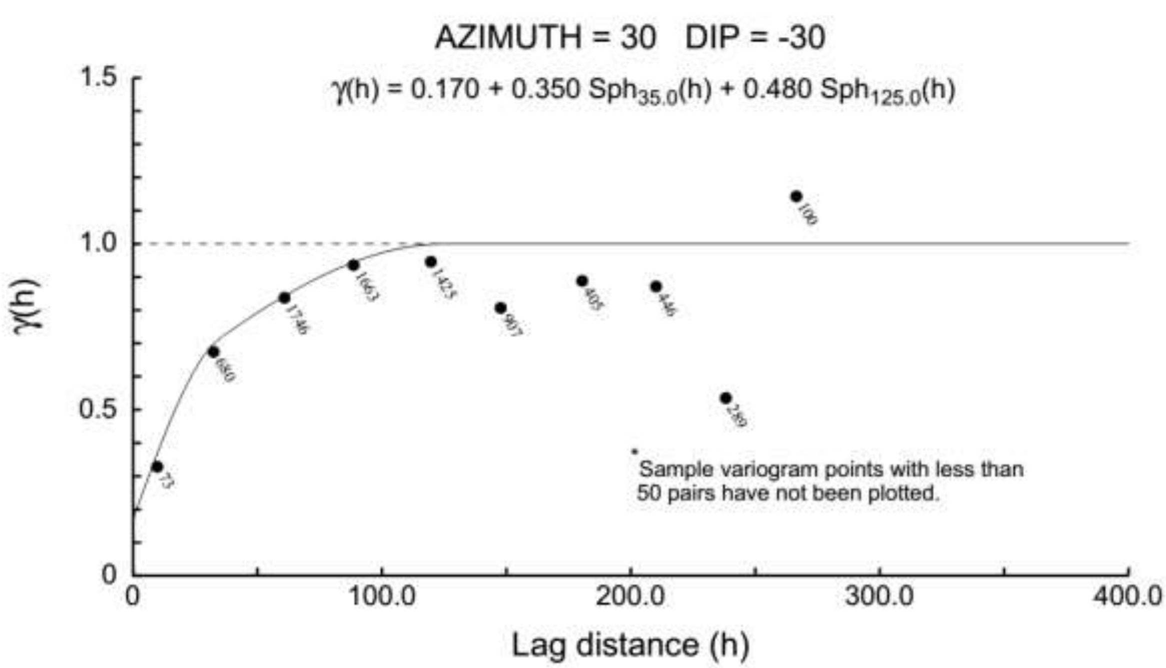


Figure 14-10: Lead Correlogram – Minas, Mineral Group 1

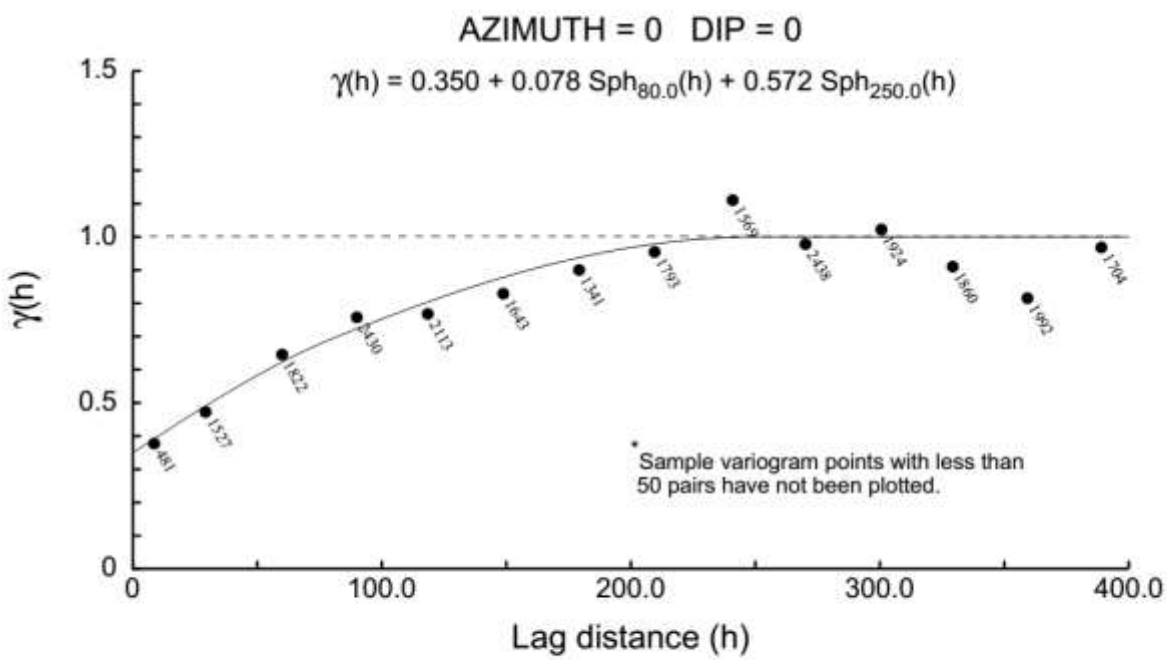


Figure 14-11: Lead Correlogram – Minas, Mineral Group 2

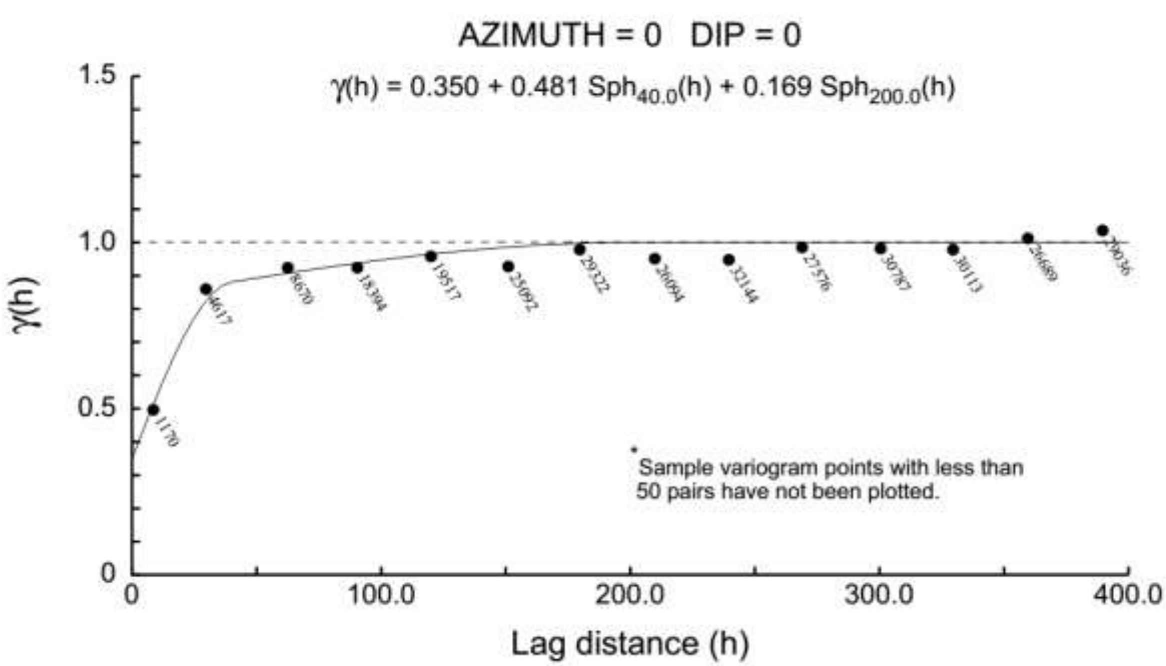


Figure 14-12: Lead Correlogram – Minas, Mineral Group 3

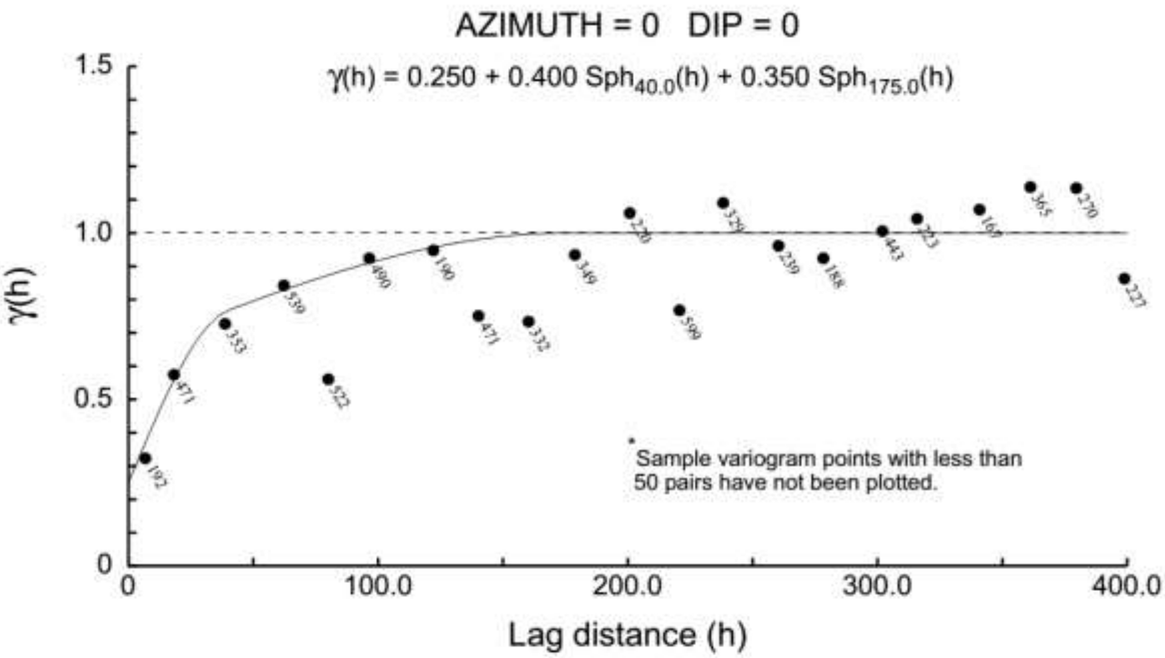


Figure 14-13: Lead Correlogram – Este, Mineral Group 1

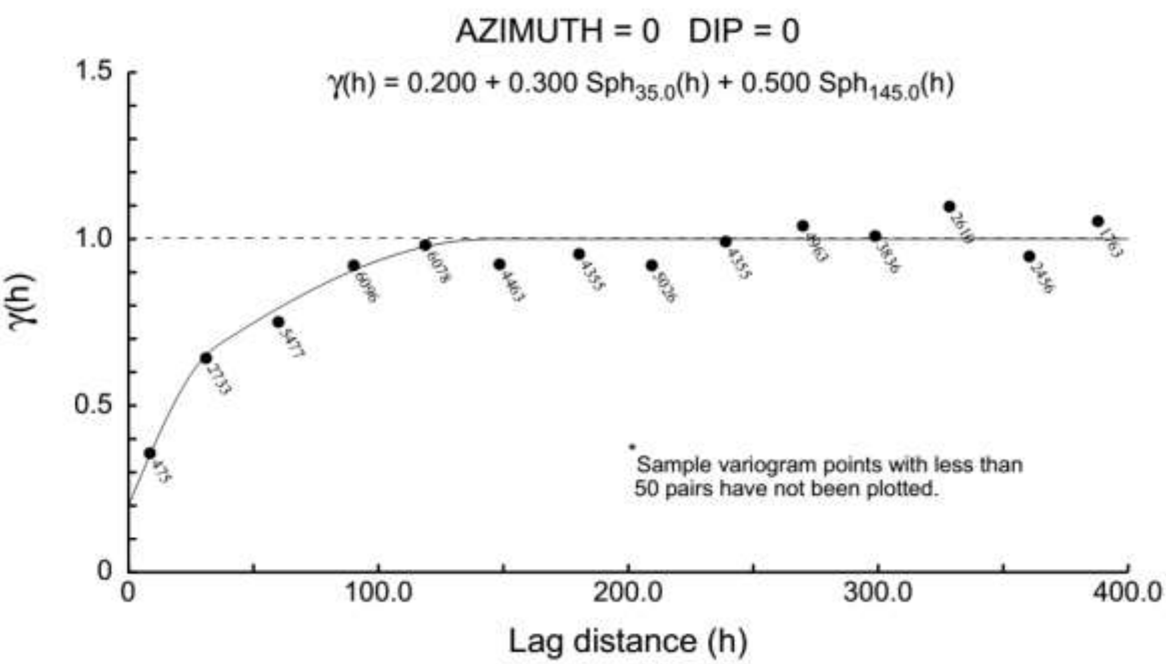


Figure 14-14: Lead Correlogram – Este, Mineral Group 2

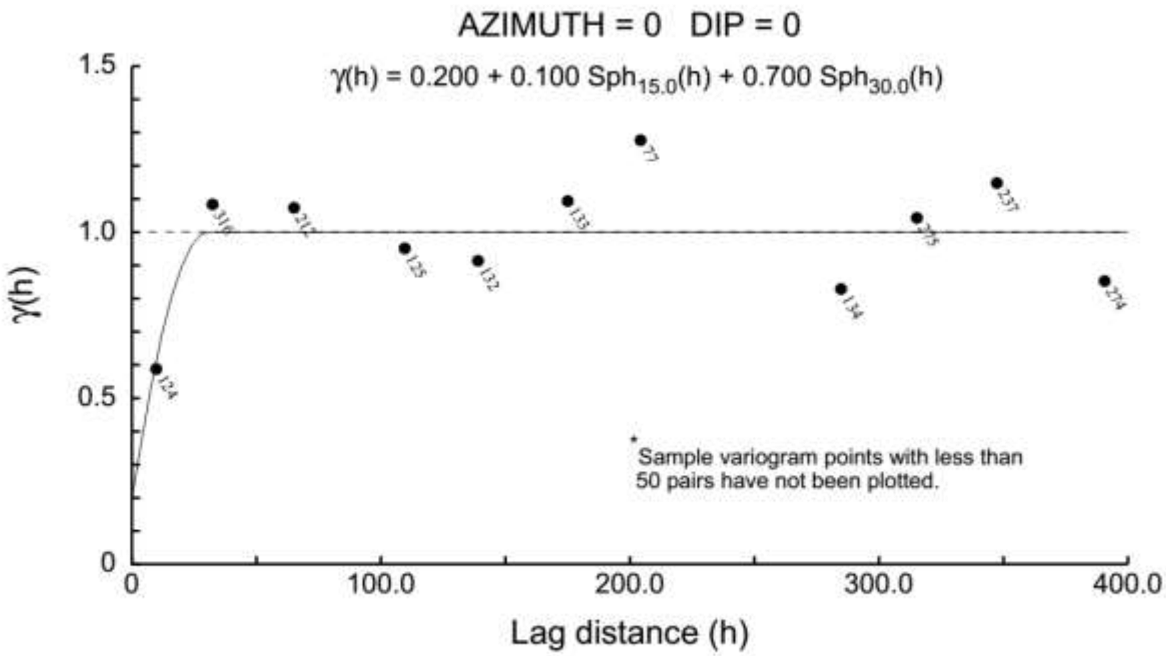


Figure 14-15: Lead Correlogram – Este, Mineral Group 3

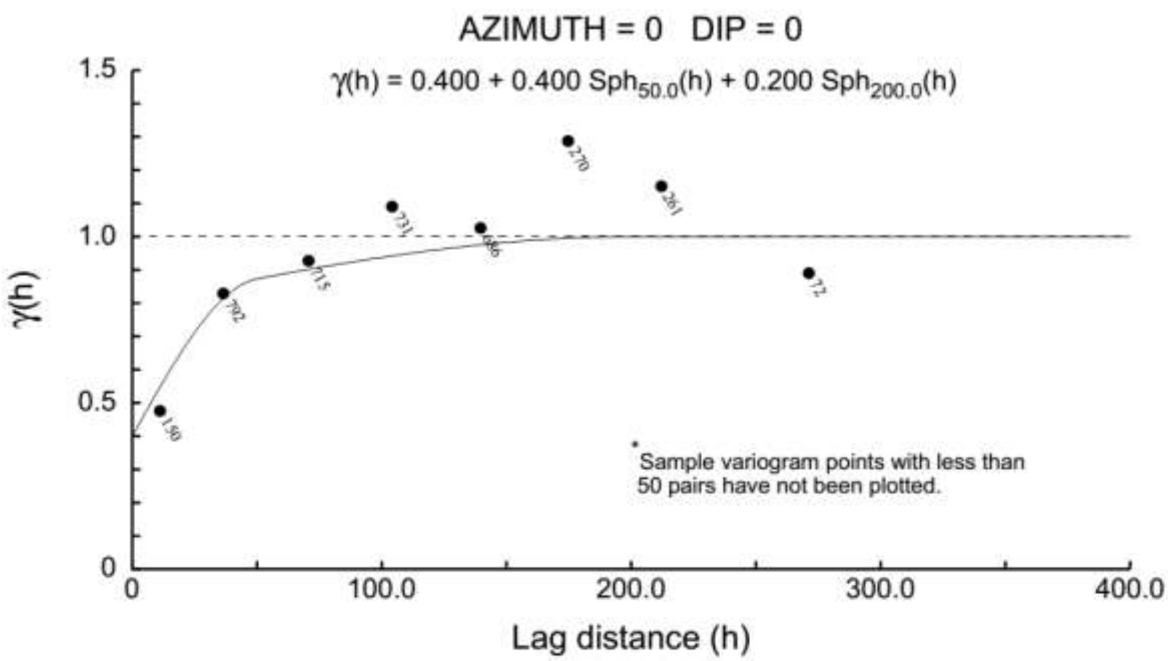


Figure 14-16: Zinc Correlogram – Main, Mineral Group 1

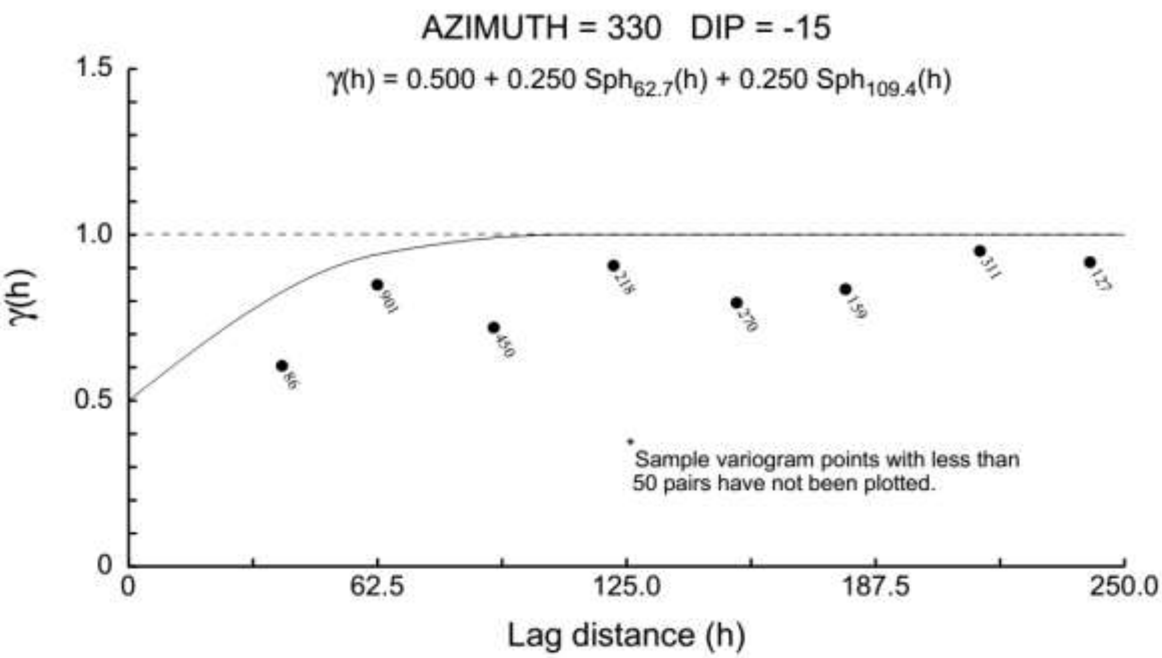


Figure 14-17: Zinc Correlogram – Main, Mineral Group 2

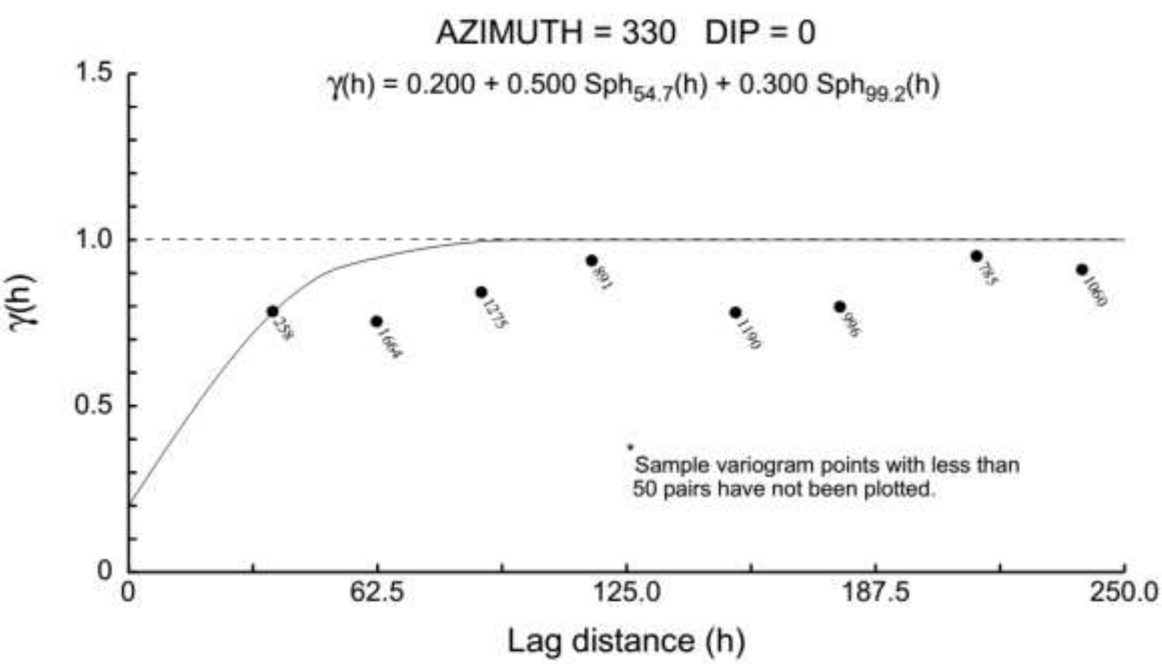


Figure 14-18: Zinc Correlogram – Main, Mineral Group 3

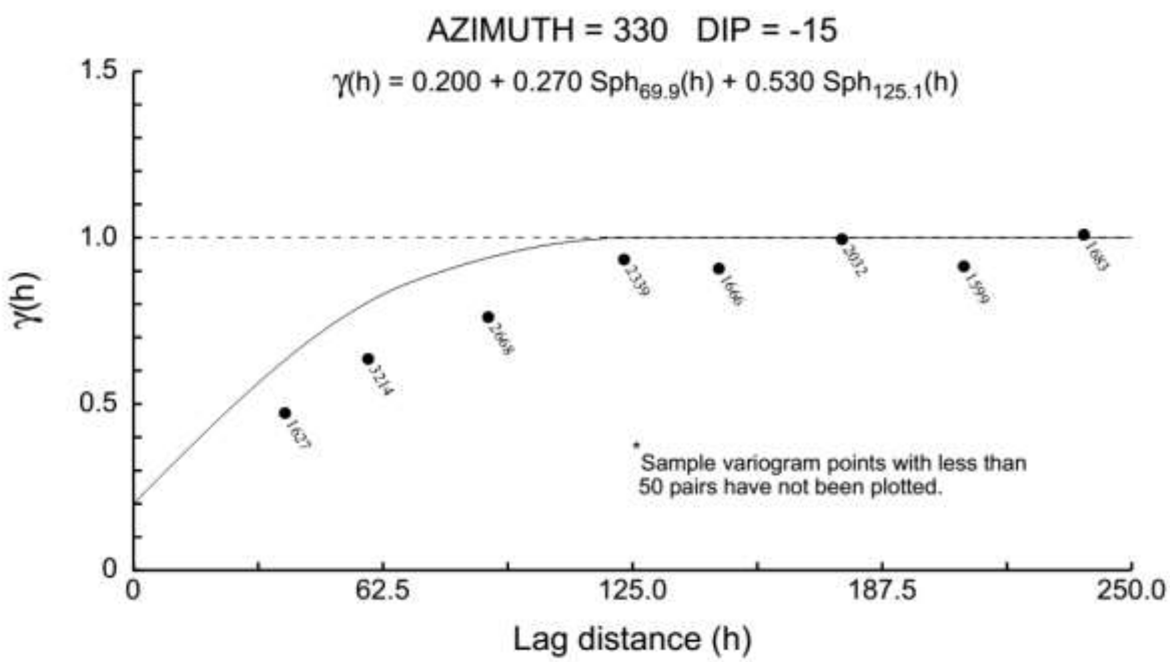


Figure 14-19: Zinc Correlogram – Main, Mineral Group 3, Strong Model Direction

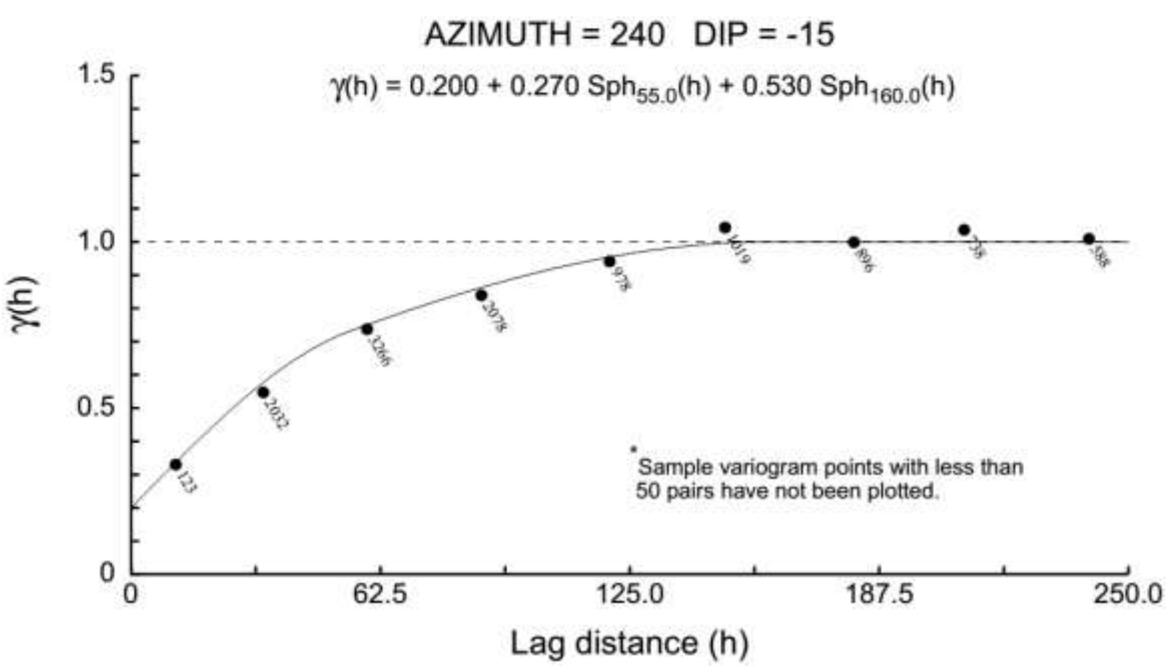


Figure 14-20: Zinc Correlogram – Minas, Mineral Group 1

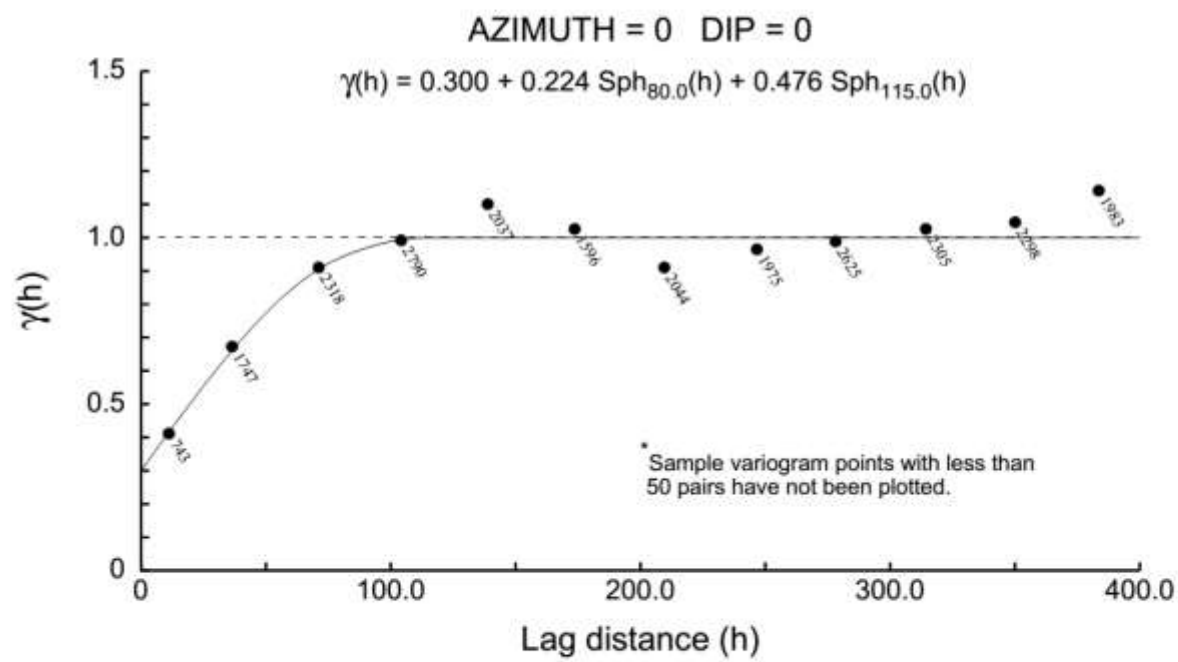


Figure 14-21: Zinc Correlogram – Minas, Mineral Group 2

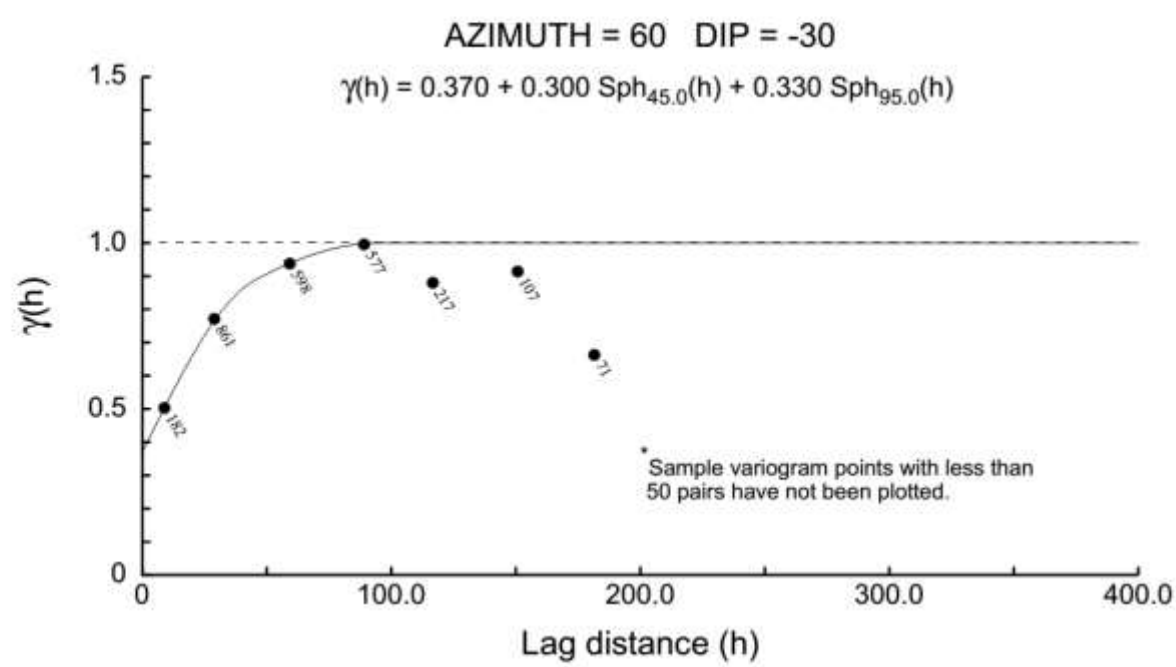


Figure 14-22: Zinc Correlogram – Minas, Mineral Group 3

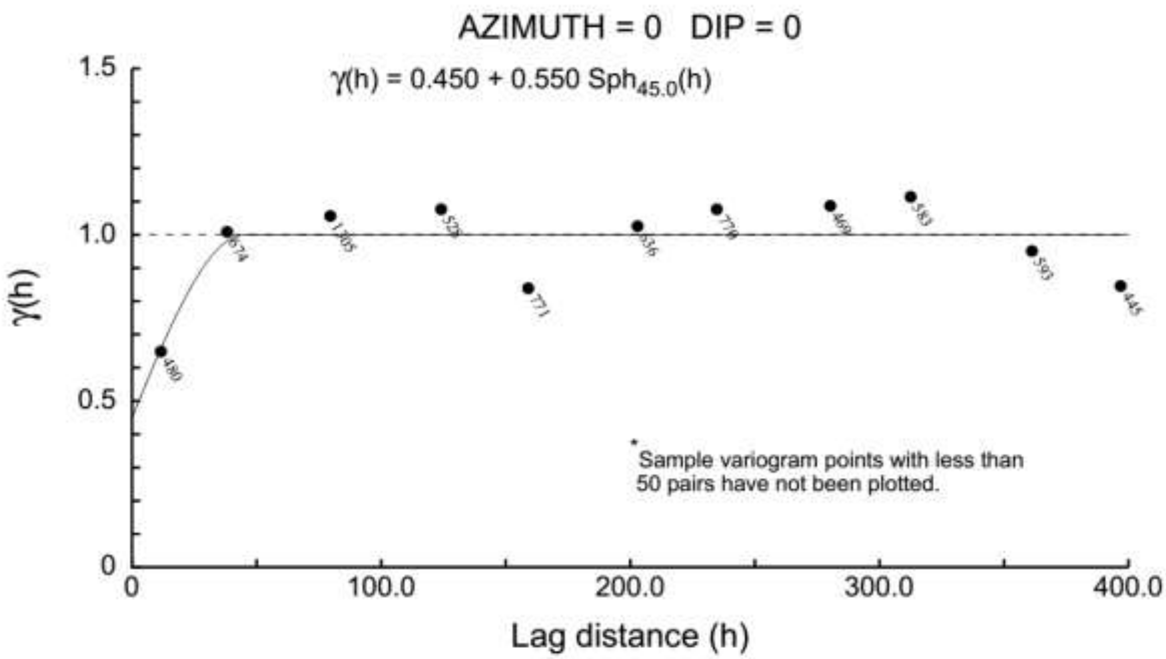


Figure 14-23: Zinc Correlogram – Este, Mineral Group 1

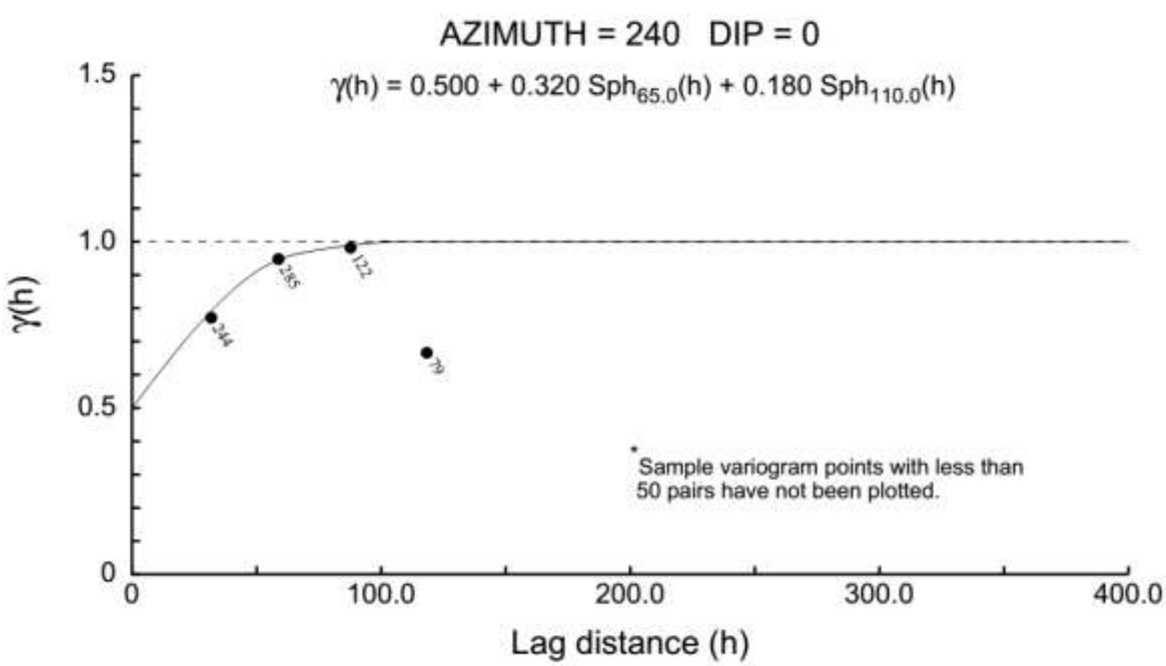


Figure 14-24: Zinc Correlogram – Este, Mineral Group 2

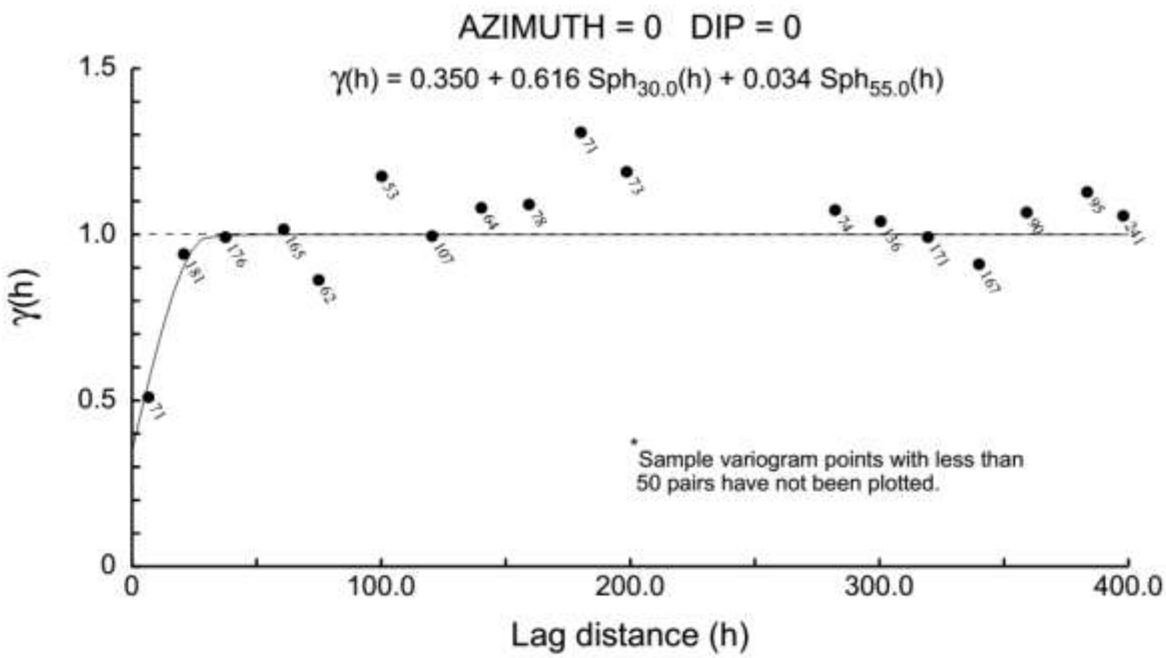


Figure 14-25: Zinc Correlogram – Este, Mineral Group 3

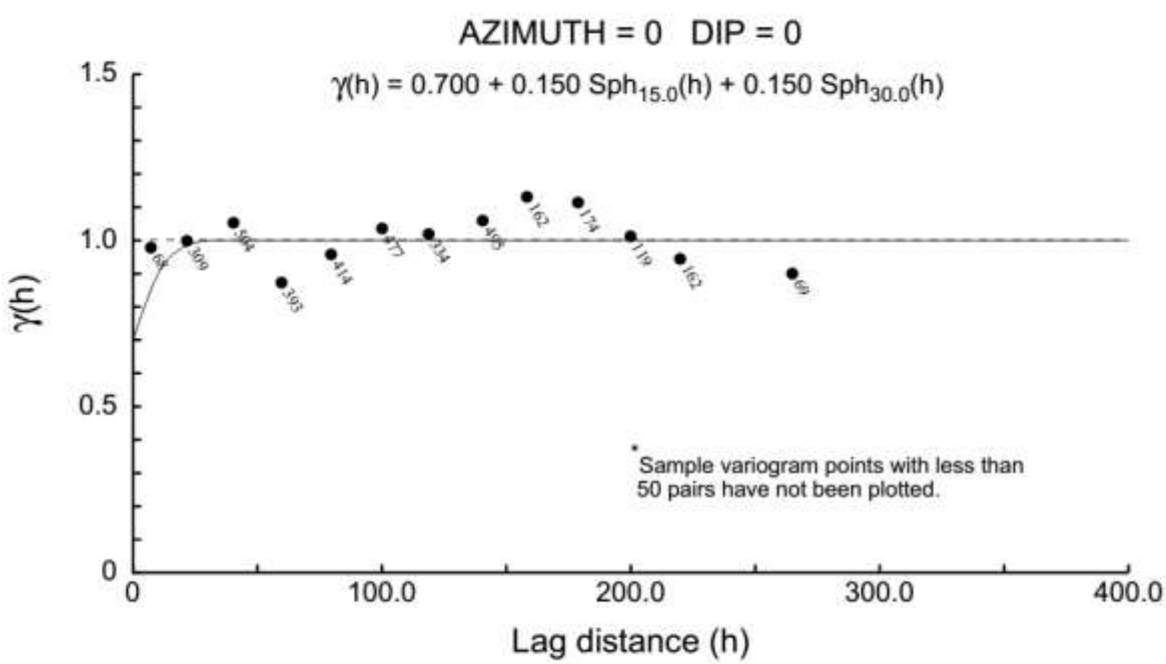


Table 14-3: Modeling Parameters for Silver, Lead, and Zinc

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Silver	Main	ALL	0.1	0.35	0.55	100	170	330	-30	15	170	140	100	180
Silver	Minas	ALL	0.1	0.3	0.6	50	180	330	-30	15	180	150	90	180
Silver	Este	ALL	0.1	0.4	0.5	30	180	330	-30	15	180	150	95	180
Lead	Main	1 2 3 8 10	0.15	0.6	0.25	50	125	338	-14	30	125	120	55	180
Lead	Main	4 6 7	0.25	0.35	0.4	38	145	330	0	-60	145	140	100	180
Lead	Main	9	0.17	0.35	0.48	35	130	334	-14	15	130	125	70	180
Lead	Minas	1 2 3 8 10	0.35	0.08	0.57	80	250	0	0	0	250	250	250	180
Lead	Minas	4 6 7	0.35	0.48	0.17	40	200	0	0	0	200	200	200	180
Lead	Minas	9	0.25	0.35	0.4	40	175	0	0	0	175	175	175	180
Lead	Este	1 2 3 8 10	0.2	0.3	0.5	35	145	0	0	0	145	145	145	180
Lead	Este	4 6 7	0.2	0.1	0.7	15	30	0	0	0	30	30	30	180
Lead	Este	9	0.4	0.4	0.2	50	200	0	0	0	200	200	200	180
Zinc	Main	1 2 3 8 10	0.5	0.25	0.25	50	140	338	-14	30	140	135	100	180
Zinc	Main	4 6 7	0.2	0.5	0.3	45	125	330	0	-60	125	120	90	180
Zinc	Main	9	0.2	0.27	0.53	55	165	334	-14	15	165	160	60	180
Zinc	Minas	1 2 3 8 10	0.3	0.22	0.48	80	115	0	0	0	115	115	115	180
Zinc	Minas	4 6 7	0.37	0.3	0.33	45	95	60	-30	0	95	80	70	180
Zinc	Minas	9	0.45	0.55	0	45	45	0	0	0	45	45	45	180
Zinc	Este	1 2 3 8 10	0.5	0.32	0.18	65	110	240	0	0	110	60	60	180
Zinc	Este	4 6 7	0.35	0.62	0.03	30	55	0	0	0	55	55	55	180
Zinc	Este	9	0.7	0.15	0.15	15	30	0	0	0	30	30	30	180

Table 14-4: Modeling Parameters for Copper, Geothite, and MnOx

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Copper	Main	1 2 3 8 10	0.6	0.05	0.35	100	180	330	-15	15	180	150	45	180
Copper	Main	4 6 7	0.4	0.45	0.15	75	150	330	-30	30	150	125	75	180
Copper	Main	9	0.5	0.1	0.4	75	220	330	-15	15	220	215	60	180
Copper	Minas	1 2 3 8 10	0.4	0.32	0.28	30	275	0	0	0	275	275	275	180
Copper	Minas	4 6 7	0.4	0.1	0.5	75	200	330	-15	15	200	90	50	180
Copper	Minas	9	0.3	0.24	0.46	40	250	0	0	0	250	250	250	180
Copper	Este	1 2 3 8 10	0.5	0.2	0.3	70	220	330	-15	30	220	125	40	180
Copper	Este	4 6 7	0.3	0.42	0.28	30	140	0	0	0	140	140	140	180
Copper	Este	9	0.3	0.38	0.32	35	135	0	0	0	135	135	135	180
Goethite	Main	1 2 3 8 10	0.45	0.25	0.3	55	120	0	0	0	120	120	120	180
Goethite	Main	4 6 7	0.35	0.55	0.1	50	125	240	-45	-85	125	120	80	180
Goethite	Main	9	0.45	0.25	0.3	50	110	270	-30	-10	110	100	60	180
Goethite	Minas	1 2 3 8 10	0.25	0.6	0.15	40	100	270	-60	30	100	50	90	180
Goethite	Minas	4 6 7	0.1	0.6	0.3	60	130	240	-30	0	130	55	120	180
Goethite	Minas	9	0.65	0.16	0.19	80	160	0	0	0	160	160	160	180
Goethite	Este	1 2 3 8 10	0.25	0.37	0.38	30	125	90	-15	0	125	90	75	180
Goethite	Este	4 6 7	0.4	0.37	0.23	10	65	0	0	0	65	65	65	180
Goethite	Este	9	0.15	0.85	0	40	40	0	0	0	40	40	40	180

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
MnOx	Main	1 2 3 8 10	0.03	0.17	0.8	40	250	30	-60	-70	250	165	120	180
MnOx	Main	4 6 7	0.2	0.7	0.1	40	125	60	0	0	125	85	110	180
MnOx	Main	9	0.2	0.3	0.5	110	210	240	-15	0	210	60	200	180
MnOx	Minas	1 2 3 8 10	0.1	0.55	0.35	60	175	0	0	0	175	175	175	180
MnOx	Minas	4 6 7	0.1	0.8	0.1	40	125	240	-15	0	125	100	100	180
MnOx	Minas	9	0.35	0.05	0.6	20	225	0	0	0	225	225	225	180
MnOx	Este	1 2 3 8 10	0.1	0.2	0.7	75	135	300	-60	30	135	75	125	180
MnOx	Este	4 6 7	0.35	0.65	0	66	66	0	0	0	66	66	66	180
MnOx	Este	9	0.6	0.4	0	50	50	0	0	0	50	50	50	180

Table 14-5: Modeling Parameters for Pyrite and Galena

Grade	Zone	Mineral Group	C0	C1	C2	h1	h2	Azimuth	Dip	Tilt (LH)	Primary Axis	Secondary Axis	Tertiary Axis	Search Range
Pyrite	Main	1 2 3 8 10	0.3	0.15	0.55	30	150	240	-45	0	150	55	120	180
Pyrite	Main	4 6 7	0.3	0.3	0.4	20	95	240	-45	-25	95	95	75	180
Pyrite	Main	9	0.2	0.45	0.35	45	90	240	-45	-65	90	90	75	180
Pyrite	Minas	1 2 3 8 10	0.1	0.32	0.58	30	350	0	0	0	350	350	350	180
Pyrite	Minas	4 6 7	0.65	0.24	0.11	31	150	0	0	0	150	150	150	180
Pyrite	Minas	9	0.72	0.12	0.16	98	100	0	0	0	100	100	100	180
Pyrite	Este	1 2 3 8 10	0.15	0.64	0.21	42	125	0	0	0	125	125	125	180
Pyrite	Este	4 6 7	0.4	0.6	0	40	40	0	0	0	40	40	40	180
Pyrite	Este	9	0.05	0.75	0.2	35	40	0	0	0	40	40	40	180
Galena	Main	1 2 3 8 10	0.3	0.5	0.2	33	125	240	-30	-15	125	100	110	180
Galena	Main	4 6 7	0.3	0.42	0.28	45	170	60	-45	-60	170	75	50	180
Galena	Main	9	0.2	0.45	0.35	55	135	150	-15	-75	135	65	105	180
Galena	Minas	1 2 3 8 10	0.75	0.14	0.11	35	80	0	0	0	80	80	80	180
Galena	Minas	4 6 7	0.25	0.58	0.17	25	100	0	0	0	100	100	100	180
Galena	Minas	9	0.7	0.25	0.05	8	80	0	0	0	80	80	80	180
Galena	Este	1 2 3 8 10	0.4	0.6	0	60	60	0	0	0	60	60	60	180
Galena	Este	4 6 7	0.8	0.14	0.06	5	50	0	0	0	50	50	50	180
Galena	Este	9	0.2	0.25	0.55	20	50	0	0	0	50	50	50	180

Table 14-6: Borehole Statistics for Silver, Lead, and Zinc

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	1	1	4	70.0	187.0	125.5	2,140.9	46.3	0.4
Ag	1	2	376	0.3	1,410.0	86.9	23,167.0	152.2	1.8
Ag	1	3	11,791	0.2	2,580.0	42.1	6,599.1	81.2	1.9
Ag	1	8	11,452	0.2	5,840.0	13.8	4,276.0	65.4	4.7
Ag	1	10	5	80.0	364.0	149.4	13,122.0	114.6	0.8
Ag	1	ALL	23,628	0.2	5,840.0	29.1	5,991.7	77.4	2.7
Ag	2	4	3,341	0.3	1,240.0	36.3	4,945.9	70.3	1.9
Ag	2	6	814	0.5	243.0	44.5	1,777.2	42.2	0.9
Ag	2	7	1,154	1.0	420.0	40.4	1,286.8	35.9	0.9
Ag	2	ALL	5,309	0.3	1,240.0	38.5	3,680.2	60.7	1.6
Ag	3	9	2,019	0.5	1,750.0	63.2	5,810.8	76.2	1.2

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	0	ALL	34,649	0.2	5,840.0	33.2	5,660.6	75.2	2.3
Pb	1	1	4	1.72	3.54	2.87	0.54	0.73	0.26
Pb	1	2	376	0.01	16.65	2.20	4.93	2.22	1.01
Pb	1	3	11,722	0.00	15.55	0.71	1.00	1.00	1.40
Pb	1	8	11,278	0.00	15.30	0.31	0.47	0.68	2.22
Pb	1	10	5	1.25	2.17	1.75	0.11	0.33	0.19
Pb	1	ALL	23,385	0.00	16.65	0.54	0.89	0.94	1.74
Pb	2	4	3,324	0.00	5.93	0.42	0.38	0.62	1.46
Pb	2	6	814	0.01	10.85	0.77	1.03	1.02	1.32
Pb	2	7	1,154	0.01	5.92	0.67	0.55	0.74	1.11
Pb	2	ALL	5,292	0.00	10.85	0.53	0.54	0.73	1.38
Pb	3	9	2,019	0.01	8.92	0.94	0.93	0.97	1.02
Pb	0	ALL	34,190	0.00	16.65	0.56	0.82	0.90	1.62
Zn	1	1	4	0.78	2.17	1.80	0.40	0.63	0.35
Zn	1	2	376	0.04	9.00	1.03	1.91	1.38	1.34
Zn	1	3	11,677	0.00	16.15	0.42	1.01	1.00	2.38
Zn	1	8	11,430	0.00	23.91	0.48	0.73	0.86	1.80
Zn	1	10	5	0.11	0.20	0.16	0.00	0.04	0.24
Zn	1	ALL	23,492	0.00	23.91	0.46	0.89	0.95	2.07
Zn	2	4	3,211	0.01	3.45	0.10	0.02	0.15	1.49
Zn	2	6	808	0.01	5.03	0.18	0.06	0.24	1.32
Zn	2	7	1,137	0.01	1.14	0.13	0.01	0.11	0.82
Zn	2	ALL	5,156	0.01	5.03	0.12	0.03	0.17	1.35
Zn	3	9	1,904	0.01	3.24	0.15	0.05	0.23	1.49
Zn	0	ALL	33,313	0.00	23.91	0.37	0.67	0.82	2.24

Table 14-7: Composite Statistics for Silver, Lead, and Zinc

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	1	1	1	125.5	125.5	125.5	0.0	0.0	0.0
Ag	1	2	101	2.6	461.8	83.4	7,517.5	86.7	1.0
Ag	1	3	2,993	0.2	730.9	41.7	3,061.4	55.3	1.3
Ag	1	8	2,917	0.4	704.0	13.3	791.9	28.1	2.1
Ag	1	10	1	181.4	181.4	181.4	0.0	0.0	0.0
Ag	1	ALL	6,013	0.2	730.9	28.6	2,287.8	47.8	1.7

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	2	4	857	0.5	534.1	36.7	3,400.2	58.3	1.6
Ag	2	6	212	1.0	166.4	43.9	1,216.5	34.9	0.8
Ag	2	7	298	2.0	205.0	40.2	761.3	27.6	0.7
Ag	2	ALL	1,367	0.5	534.1	38.6	2,491.4	49.9	1.3
Ag	3	9	525	1.2	472.6	63.0	3,041.4	55.1	0.9
Ag	0	ALL	8,872	0.2	730.9	33.4	2,598.8	51.0	1.5
Pb	1	1	1	2.87	2.87	2.87	0.00	0.00	0.00
Pb	1	2	101	0.07	8.22	2.15	2.34	1.53	0.71
Pb	1	3	2,993	0.00	8.51	0.71	0.62	0.79	1.11
Pb	1	8	2,908	0.00	6.97	0.30	0.26	0.51	1.71
Pb	1	10	1	2.17	2.17	2.17	0.00	0.00	0.00
Pb	1	ALL	6,004	0.00	8.51	0.53	0.56	0.75	1.40
Pb	2	4	857	0.00	3.44	0.42	0.28	0.53	1.26
Pb	2	6	212	0.01	5.75	0.77	0.76	0.87	1.14
Pb	2	7	298	0.01	3.46	0.69	0.39	0.62	0.91
Pb	2	ALL	1,367	0.00	5.75	0.53	0.40	0.63	1.19
Pb	3	9	525	0.02	7.05	0.94	0.60	0.77	0.83
Pb	0	ALL	8,841	0.00	8.51	0.56	0.54	0.73	1.32
Zn	1	1	1	1.80	1.80	1.80	0.00	0.00	0.00
Zn	1	2	101	0.12	7.89	1.05	1.24	1.11	1.06
Zn	1	3	2,965	0.00	7.48	0.42	0.65	0.80	1.91
Zn	1	8	2,912	0.01	9.75	0.47	0.43	0.66	1.39
Zn	1	10	1	0.15	0.15	0.15	0.00	0.00	0.00
Zn	1	ALL	5,980	0.00	9.75	0.46	0.56	0.75	1.64
Zn	2	4	816	0.01	2.24	0.10	0.02	0.13	1.25
Zn	2	6	210	0.02	1.26	0.17	0.02	0.15	0.89
Zn	2	7	293	0.01	0.80	0.13	0.01	0.09	0.69
Zn	2	ALL	1,319	0.01	2.24	0.12	0.02	0.13	1.06
Zn	3	9	493	0.01	1.82	0.15	0.03	0.17	1.13
Zn	0	ALL	8,500	0.00	9.75	0.36	0.43	0.66	1.80

Table 14-8: Composite Statistics for Copper, MnOx, and Goethite

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Cu	1	1	1	0.05	0.05	0.05	0.00	0.00	0.00
Cu	1	2	101	0.00	0.27	0.05	0.00	0.05	1.01
Cu	1	3	2,724	0.00	1.05	0.03	0.00	0.05	1.49
Cu	1	8	1,752	0.00	0.54	0.03	0.00	0.05	1.61
Cu	1	10	1	0.06	0.06	0.06	0.00	0.00	0.00
Cu	1	ALL	4,579	0.00	1.05	0.03	0.00	0.05	1.52
Cu	2	4	694	0.00	0.59	0.02	0.00	0.03	1.68
Cu	2	6	210	0.00	0.15	0.03	0.00	0.02	0.81
Cu	2	7	288	0.00	0.08	0.02	0.00	0.01	0.71
Cu	2	ALL	1,192	0.00	0.59	0.02	0.00	0.03	1.30
Cu	3	9	486	0.00	0.22	0.04	0.00	0.03	0.68
Cu	0	ALL	6,800	0.00	1.05	0.03	0.00	0.04	1.45
MnOx	1	1	0	0.00	0.00	0.00	0.00	0.00	0.00
MnOx	1	2	32	0.01	2.00	0.82	0.50	0.71	0.86
MnOx	1	3	806	0.00	5.48	0.72	0.69	0.83	1.15
MnOx	1	8	344	0.00	6.00	0.63	0.59	0.77	1.21
MnOx	1	10	0	0.00	0.00	0.00	0.00	0.00	0.00
MnOx	1	ALL	1,182	0.00	6.00	0.70	0.66	0.81	1.16
MnOx	2	4	260	0.00	4.97	0.62	0.53	0.73	1.17
MnOx	2	6	124	0.00	5.25	0.81	0.96	0.98	1.22
MnOx	2	7	112	0.01	2.67	0.48	0.45	0.67	1.40
MnOx	2	ALL	496	0.00	5.25	0.64	0.63	0.80	1.25
MnOx	3	9	312	0.00	5.39	1.26	0.80	0.89	0.71
MnOx	0	ALL	2,338	0.00	6.00	0.70	0.69	0.83	1.19
Goeth	1	1	0	0.00	0.00	0.00	0.00	0.00	0.00
Goeth	1	2	17	0.01	1.00	0.29	0.10	0.31	1.07
Goeth	1	3	508	0.00	3.00	0.38	0.26	0.51	1.35
Goeth	1	8	95	0.00	5.15	0.41	0.64	0.80	1.95
Goeth	1	10	0	0.00	0.00	0.00	0.00	0.00	0.00
Goeth	1	ALL	620	0.00	5.15	0.38	0.31	0.56	1.47
Goeth	2	4	563	0.00	5.39	0.66	0.63	0.79	1.19
Goeth	2	6	180	0.01	4.37	1.14	0.75	0.86	0.76
Goeth	2	7	253	0.01	7.39	1.35	1.50	1.23	0.90
Goeth	2	ALL	996	0.00	7.39	0.92	0.96	0.98	1.06
Goeth	3	9	328	0.00	6.42	1.21	1.36	1.17	0.96
Goeth	0	ALL	2,714	0.00	7.39	0.75	1.02	1.01	1.34

Table 14-9: Composite Statistics for Pyrite and Galena

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
FeS	1	1	1	1.50	1.50	1.50	0.00	0.00	0.00
FeS	1	2	86	0.06	6.73	1.28	1.73	1.31	1.02
FeS	1	3	2,647	0.00	10.00	1.26	1.35	1.16	0.92
FeS	1	8	2,786	0.00	12.68	1.29	1.15	1.07	0.83
FeS	1	10	1	2.15	2.15	2.15	0.00	0.00	0.00
FeS	1	ALL	5,521	0.00	12.68	1.28	1.25	1.12	0.88
FeS	2	4	394	0.00	10.21	0.46	0.56	0.75	1.61
FeS	2	6	46	0.01	2.63	0.74	0.64	0.80	1.09
FeS	2	7	73	0.00	7.25	0.62	1.17	1.08	1.74
FeS	2	ALL	513	0.00	10.21	0.51	0.66	0.81	1.59
FeS	3	9	197	0.00	17.27	0.75	2.24	1.50	2.01
FeS	0	ALL	7,306	0.00	17.27	1.09	1.25	1.12	1.03
PbS	1	1	1	0.50	0.50	0.50	0.00	0.00	0.00
PbS	1	2	91	0.02	12.27	1.27	3.49	1.87	1.47
PbS	1	3	1,793	0.00	10.00	0.23	0.30	0.55	2.39
PbS	1	8	1,778	0.00	6.34	0.18	0.16	0.39	2.14
PbS	1	10	1	0.22	0.22	0.22	0.00	0.00	0.00
PbS	1	ALL	3,664	0.00	12.27	0.23	0.34	0.58	2.48
PbS	2	4	153	0.00	1.30	0.07	0.02	0.15	2.24
PbS	2	6	32	0.00	0.28	0.07	0.01	0.08	1.09
PbS	2	7	25	0.00	1.00	0.07	0.04	0.20	2.67
PbS	2	ALL	210	0.00	1.30	0.07	0.02	0.15	2.15
PbS	3	9	151	0.00	2.53	0.17	0.10	0.32	1.91
PbS	0	ALL	4,454	0.00	12.27	0.21	0.29	0.54	2.55

Table 14-10: Block Statistics for Silver, Lead, and Zinc

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Ag	1	1	4	58.8	105.7	85.3	411.6	20.3	0.2
Ag	1	2	822	4.5	294.7	81.2	2738.8	52.3	0.6
Ag	1	3	53,120	0.6	521.8	29.4	1005.7	31.7	1.1
Ag	1	8	102,322	0.5	479.6	10.6	203.0	14.2	1.3
Ag	1	10	13	59.4	158.6	103.1	786.8	28.1	0.3
Ag	1	ALL	156,281	0.5	521.8	17.4	590.3	24.3	1.4
Ag	2	4	21,514	0.6	285.4	20.2	641.0	25.3	1.3
Ag	2	6	3,433	0.5	216.0	37.3	887.1	29.8	0.8
Ag	2	7	6,595	0.5	192.1	33.9	527.5	23.0	0.7
Ag	2	ALL	31,542	0.5	285.4	24.9	692.9	26.3	1.1
Ag	3	9	5,901	4.0	308.2	45.8	1368.8	37.0	0.8
Ag	0	ALL	277,500	0.5	521.8	17.1	543.9	23.3	1.4
Pb	1	1	4	1.57	2.49	2.04	0.20	0.45	0.22
Pb	1	2	822	0.33	6.16	2.03	0.68	0.83	0.41
Pb	1	3	51,752	0.00	5.95	0.54	0.24	0.49	0.91
Pb	1	8	101,022	0.00	5.93	0.23	0.08	0.29	1.27
Pb	1	10	13	0.82	2.91	1.85	0.36	0.60	0.33
Pb	1	ALL	153,613	0.00	6.16	0.34	0.18	0.42	1.23
Pb	2	4	21,463	0.01	2.91	0.29	0.10	0.31	1.07
Pb	2	6	3,411	0.01	3.60	0.59	0.29	0.54	0.91
Pb	2	7	6,583	0.01	3.36	0.52	0.15	0.39	0.75
Pb	2	ALL	31,457	0.01	3.60	0.37	0.14	0.38	1.02
Pb	3	9	5,920	0.03	4.98	0.77	0.24	0.49	0.64
Pb	0	ALL	190,990	0.00	6.16	0.36	0.18	0.42	1.17
Zn	1	1	4	0.72	1.54	1.22	0.14	0.37	0.30
Zn	1	2	822	0.05	3.90	0.94	0.32	0.56	0.60
Zn	1	3	52,185	0.01	5.56	0.32	0.20	0.45	1.42
Zn	1	8	100,955	0.01	5.66	0.34	0.10	0.32	0.94
Zn	1	10	13	0.14	3.06	1.56	1.23	1.11	0.71
Zn	1	ALL	153,979	0.01	5.66	0.34	0.14	0.38	1.11
Zn	2	4	20,712	0.02	1.07	0.11	0.01	0.07	0.69
Zn	2	6	3,402	0.03	1.16	0.15	0.01	0.09	0.60
Zn	2	7	6,561	0.03	0.59	0.12	0.00	0.05	0.45
Zn	2	ALL	30,675	0.02	1.16	0.11	0.01	0.07	0.64
Zn	3	9	5,697	0.01	1.25	0.16	0.01	0.09	0.60

Table 14-11: Block Statistics for Copper, Goethite, and MnOx

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Cu	1	1	4	0.019	0.045	0.035	0.000	0.011	0.321
Cu	1	2	822	0.013	0.175	0.044	0.001	0.026	0.600
Cu	1	3	49958	0.000	0.839	0.028	0.001	0.031	1.109
Cu	1	8	86454	0.000	0.326	0.017	0.001	0.024	1.389
Cu	1	10	13	0.023	0.080	0.049	0.000	0.016	0.328
Cu	1	ALL	137251	0.000	0.839	0.021	0.001	0.027	1.282
Cu	2	4	19025	0.000	0.322	0.015	0.000	0.019	1.226
Cu	2	6	3255	0.001	0.122	0.023	0.000	0.015	0.678
Cu	2	7	6029	0.000	0.083	0.017	0.000	0.010	0.596
Cu	2	ALL	28309	0.000	0.322	0.016	0.000	0.017	1.030
Cu	3	9	5849	0.000	0.159	0.046	0.001	0.023	0.495
Cu	0	ALL	171409	0.000	0.839	0.021	0.001	0.026	1.228
Goethite	1	1	4	0.00	0.02	0.01	0.00	0.01	1.62
Goethite	1	2	520	0.00	0.63	0.07	0.01	0.10	1.31
Goethite	1	3	28423	0.00	2.93	0.11	0.05	0.21	1.92
Goethite	1	8	24314	0.00	3.74	0.06	0.03	0.16	2.93
Goethite	1	10	7	0.00	0.01	0.00	0.00	0.00	1.02
Goethite	1	ALL	53268	0.00	3.74	0.09	0.04	0.19	2.26
Goethite	2	4	19734	0.00	4.31	0.49	0.19	0.43	0.89
Goethite	2	6	3222	0.00	4.06	0.92	0.37	0.61	0.66
Goethite	2	7	6204	0.00	4.49	1.03	0.50	0.71	0.69
Goethite	2	ALL	29160	0.00	4.49	0.65	0.33	0.58	0.89
Goethite	3	9	5492	0.00	5.56	0.72	0.46	0.68	0.94
Goethite	0	ALL	87920	0.00	5.56	0.31	0.24	0.49	1.57
MnOx	1	1	2	0.07	0.33	0.20	0.03	0.19	0.94
MnOx	1	2	755	0.00	2.54	0.34	0.19	0.43	1.26
MnOx	1	3	29351	0.00	4.89	0.29	0.24	0.49	1.69
MnOx	1	8	33184	0.00	2.77	0.18	0.12	0.35	1.97
MnOx	1	10	9	0.05	0.51	0.27	0.02	0.15	0.57
MnOx	1	ALL	63301	0.00	4.89	0.23	0.18	0.43	1.83
MnOx	2	4	14842	0.00	2.29	0.21	0.11	0.34	1.62
MnOx	2	6	2942	0.00	3.61	0.41	0.25	0.50	1.19
MnOx	2	7	5141	0.00	2.47	0.19	0.09	0.29	1.55
MnOx	2	ALL	22925	0.00	3.61	0.23	0.13	0.36	1.56
MnOx	3	9	5510	0.00	3.56	0.77	0.51	0.71	0.92

Table 14-12: Block Statistics for Pyrite and Galena

Grade Label	Group	Mineral Type	Sample Count	Minimum	Maximum	Mean	Variance	Standard Deviation	Coefficient of Variation
Pyrite	1	1	4	0.89	1.29	1.13	0.03	0.17	0.15
Pyrite	1	2	822	0.07	4.84	1.10	0.56	0.75	0.68
Pyrite	1	3	49192	0.00	9.00	1.02	0.67	0.82	0.80
Pyrite	1	8	97831	0.00	10.75	1.09	0.55	0.74	0.68
Pyrite	1	10	13	1.29	4.17	2.94	0.85	0.92	0.31
Pyrite	1	ALL	147862	0.00	10.75	1.06	0.59	0.77	0.72
Pyrite	2	4	17645	0.00	5.13	0.22	0.11	0.34	1.53
Pyrite	2	6	2574	0.00	2.44	0.22	0.15	0.39	1.81
Pyrite	2	7	3736	0.00	2.54	0.14	0.05	0.23	1.68
Pyrite	2	ALL	23955	0.00	5.13	0.21	0.11	0.33	1.60
Pyrite	3	9	5021	0.00	9.94	0.27	0.31	0.56	2.05
Pyrite	0	ALL	176838	0.00	10.75	0.93	0.61	0.78	0.85
Galena	1	1	4	0.24	0.39	0.33	0.01	0.07	0.22
Galena	1	2	822	0.03	6.38	0.95	0.99	0.99	1.05
Galena	1	3	45352	0.00	8.85	0.12	0.08	0.28	2.22
Galena	1	8	82509	0.00	7.70	0.10	0.08	0.28	2.81
Galena	1	10	13	0.13	0.46	0.30	0.01	0.12	0.40
Galena	1	ALL	128700	0.00	8.85	0.11	0.09	0.30	2.61
Galena	2	4	11542	0.00	0.79	0.01	0.00	0.03	2.31
Galena	2	6	1934	0.00	0.23	0.01	0.00	0.03	2.38
Galena	2	7	3022	0.00	0.47	0.01	0.00	0.02	2.63
Galena	2	ALL	16498	0.00	0.79	0.01	0.00	0.03	2.39
Galena	3	9	4706	0.00	1.46	0.06	0.01	0.10	1.62
Galena	0	ALL	149904	0.00	8.85	0.10	0.08	0.28	2.75

14.1.4 Classification

Blocks were coded as measured, indicated or inferred based on the silver grade estimate. The number of samples and the distance to the nearest sample were the discriminating factors. If the number of composite samples is 6 (maximum used to estimate grade) and the distance to the nearest composite is less than 12 meters, the block is classified as measured. If the number of composite samples is at least 2, the distance to the nearest composite is less than 50 meters, and the block has not been classified as measured, the block is classified as indicated. All other blocks with an estimated silver grade are classified as inferred.

14.1.5 Mineral Codes

Prior estimates used mineralization codes that were assigned to the block model based on ore mineralogy and categorizing the metallurgical responses of the deposit. Following detailed geometallurgical analysis, this approach was modified, and was replaced with one using a statistical model with zinc grade, elevation, and geologic log data (i.e. pyrite %, galena %, MnO, etc) to create a continuous recovery model/formula for each of the metals. Application of this method resulted in generation of the following formulas for the prediction of metals recoveries.

$$\begin{aligned} Lead\ Recovery \\ &= 61.9 - 40.9 \times \max(\,0, 0.57 - Zn) + 7.7 \times \max(0, galena - 0.38) + 45.4 \\ &\times \max(0, 0.37 - goethite) - 0.12 \times \max(0, elevation - 4891) + 32.9 \\ &\times \max(0, 0.27 - MnOx) - 6.21 \times \max(0, pyrite - 1.07) - 16.4 \times \max(0, 1.07 \\ &- pyrite) \end{aligned}$$

Note: Maximum Lead Recovery is 98% and minimum Lead Recovery is 2%

$$\begin{aligned} Zinc\ Recovery \\ &= 78.7 - 50.6\ \times \max(0, 1.02 - Zn) - 0.15\ \times \max(0, elevation - 4901) - 5.4 \\ &\times \max(0, pyrite - 1.9) - 11.2\ \times \max(0, 1.9 - pyrite) + 104.1\ \times \max(0, Cu - 0.03) \\ &+ 1620.2\ \times \max(0, 0.03 - Cu) \end{aligned}$$

Note: Maximum Zinc Recovery is 83% and minimum Zinc Recovery is 0%

$$Silver\ Recovery\ in\ Lead\ Concentrate = 0.67 \times Lead\ Recovery + 12$$

$$Silver\ Recovery\ in\ Zinc\ Concentrate = 41 - 0.41 \times (Silver\ Recovery\ in\ Lead\ Concentrate)$$

Figure 14-26 and Figure 14-27 compare predicted silver recovery for the previous and current models along a portion of an E-W cross section through the block model that includes Corani Minas. Figure 14-28 and Figure 14-29 do the same for zinc. Equivalent comparison sections for lead were presented in Section 7 (Figure 7-9 and 7-10). The cross sections show how the updated model has the ability to capture spatial variability in recovery and provides improved resolution relative to the old model based on discrete geomet categories.

Figure 14-26: Silver Recovery for Previous Model

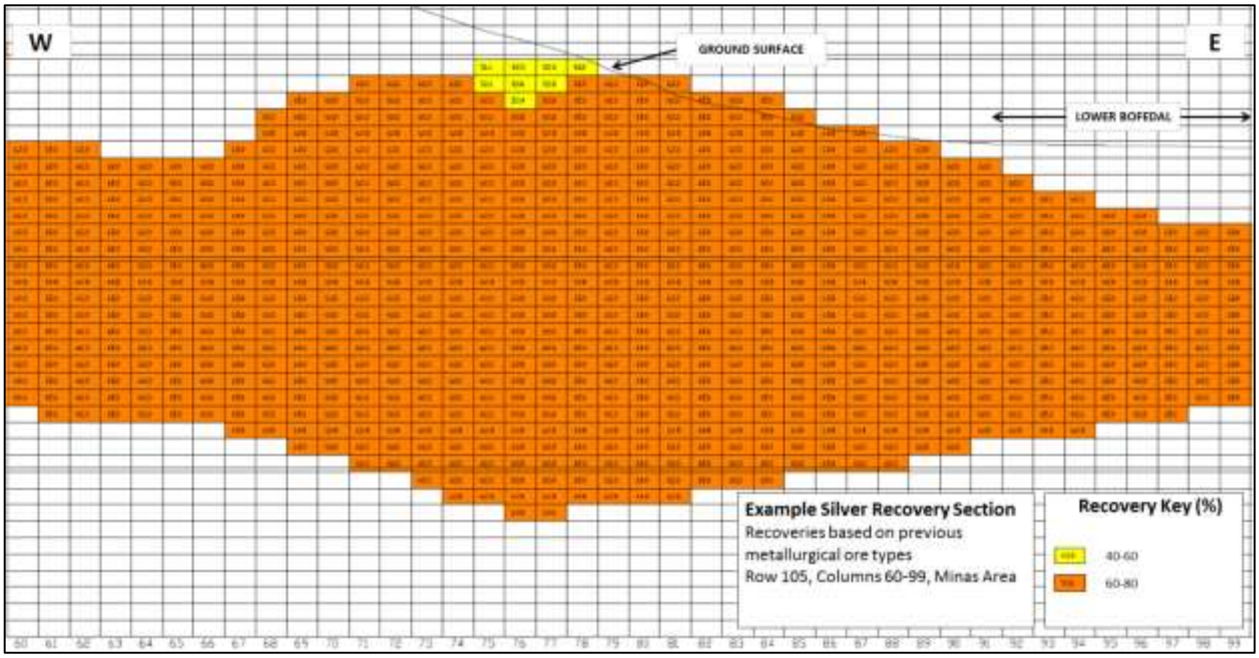


Figure 14-27: Silver Recovery for Current Model

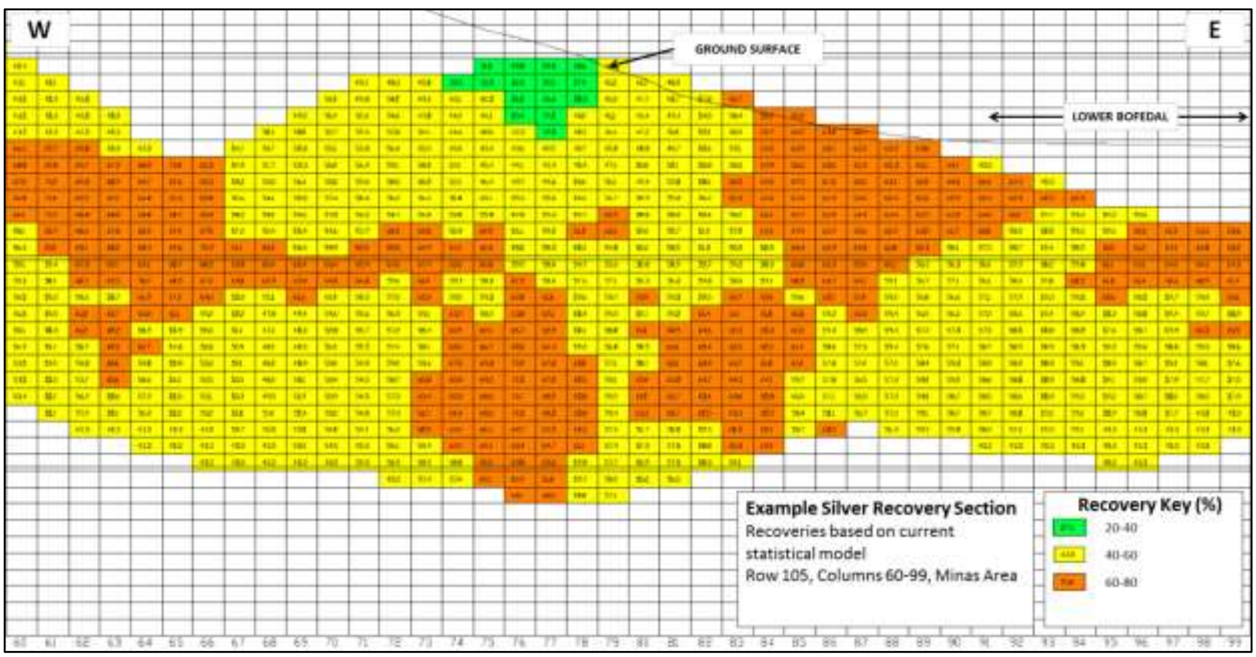


Figure 14-28: Zinc Recovery for Previous Model

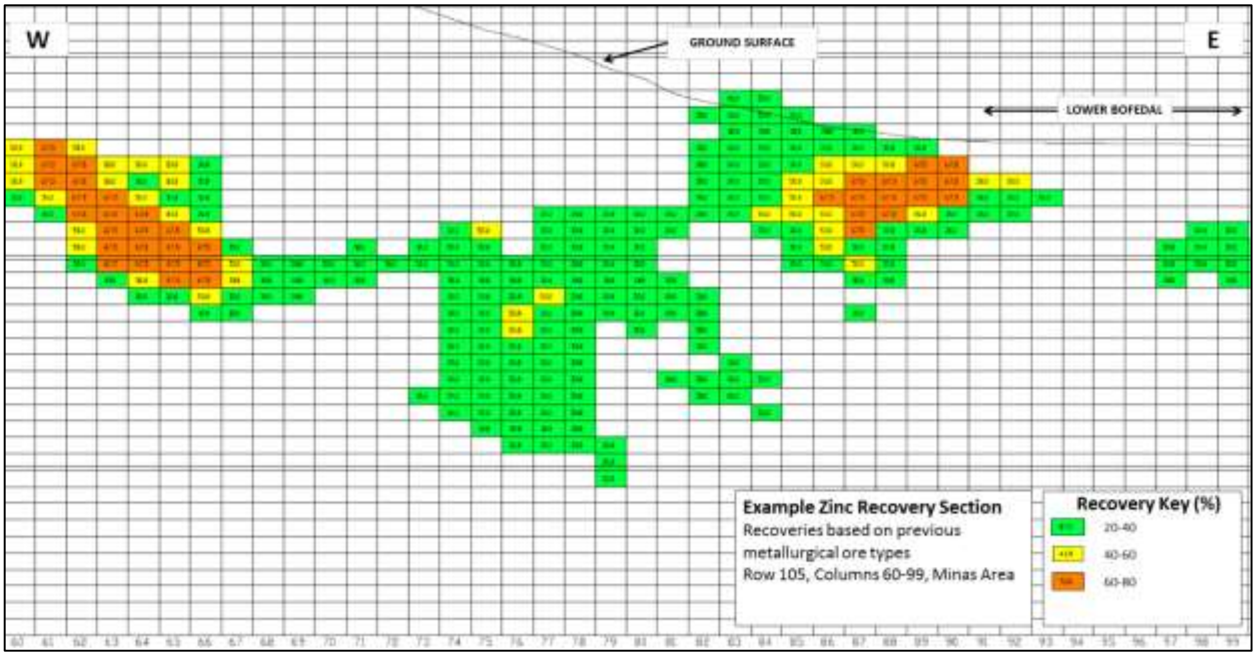
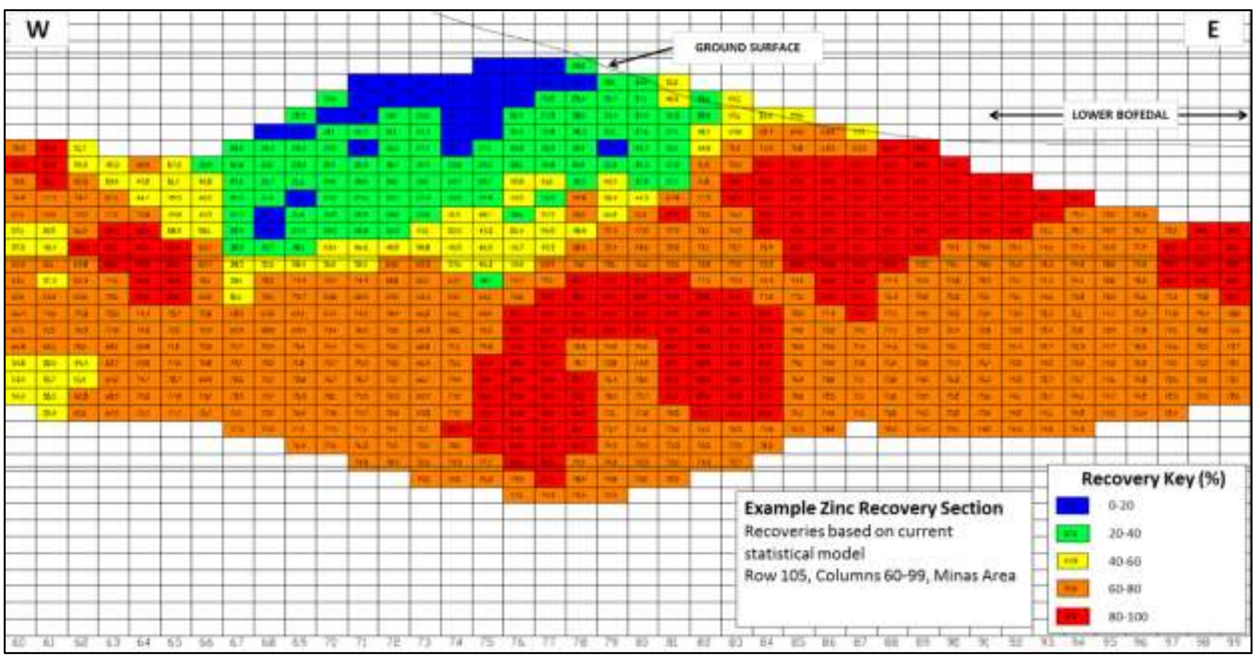


Figure 14-29: Zinc Recovery for Current Model



14.2 ACID ROCK DRAINAGE

To utilize waste rock for construction of the dump, backfill cap, embankment dams, etc., there needed to be a way to distinguish non-acid-generating rock (NAG) from potentially acid-generating rock (PAG). For this purpose, GRE developed an acid rock drainage (ARD) block model using available geochemical sample data collected by Vector in 2009, GRE in 2011, and AMEC in 2012.

Various analyses were performed on each sample group including full Acid-Base Accounting (ABA) analysis, whole rock analysis, metals by aqua regia digestion, and total carbon/total sulfur assay by LECO furnace. A summary of each sample group and analysis is given in Table 14-13.

Table 14-13: Geochemical Sample Summary

Collected By	Date	No. of Samples	Analyses			
			ABA	LECO	WRA	Metals
Vector	2009	23	23	23	7	7
GRE	2011	224	23**	201	23**	23**
AMEC	2012	397*	397	397	397	397
Total		**644**	**443**	**621**	**427**	**427**
*2 samples are reported as the same location and interval. The results for these samples were averaged **23 Samples from onsite kinetic cell program						

The parameters selected for inclusion in the model included AP/total sulfur, NP, pH, and metals of concern, which are described in the Geochemical Characterization Report (GRE, 2012) and include arsenic, cadmium, copper, lead, mercury, nickel, and zinc.

For the parameters considered, comparable analyses from different labs (i.e. total sulphur/total sulfur/ S-%) were matched, and results were converted to consistent units. GRE developed linear models relating ABA AP and total sulfur by LECO furnace by rock type for samples collected by Vector in 2009 and GRE in 2011. These relationships were used to predict AP for samples where only LECO furnace total sulfur/carbon was analyzed. AP was predicted only for samples from rock types PM, FeO, and FBS for which the relationship between total sulfur and AP had an R^2 greater than 0.97. If the model predicted an AP less than 0, an AP of 0 was assumed. This analysis is described in the Geochemical Characterization Report (GRE, 2012). Table 14-14Table 14-15 contain summary statistics for the raw data.

Table 14-14: Raw Data Summary Statistics

Parameter	AP	NP	As	Cd	Cu
Number	610	442	426	426	426
Mean	22.4	3.9	248.6	13.0	94.5
Std Dev	32.7	7.0	518.2	44.1	149.2
Variance	1066.4	48.5	268529.5	1942.5	22265.4
Maximum	206.0	97.0	5560.0	569.0	1200.0
Minimum	0.0	0.00	10.1	0.02	1.5
Range	206.0	97.0	5549.9	569.0	1198.5
Coef Var	145.7	178.0	208.4	339.8	157.9
Std Err	1.3	0.3	25.1	2.1	7.2

Table 14-15: Raw Data Summary Statistics

Parameter	Pb	Ni	Hg	Zn	pH
Number	426	426	426	426	442
Mean	2231.5	8.7	0.9	1518.3	6.2
Std Dev	4412.9	11	1.7	2596.2	1.5
Variance	19473957.0	120.7	3	6740476.8	2.3
Maximum	61900.0	74.8	18.8	32300.0	9.5
Minimum	10.0	0.6	0.01	59.0	2.3
Range	61890.0	74.2	18.8	32241.0	7.2
Coef Var	197.8	126.1	189.7	171.0	24.3
Std Err	213.8	0.5	0.08	125.8	0.07

The geochemical samples were loaded into TECHBASE and composited to 8-meter intervals along lithology boundaries. Inverse-distance-cubed weighting was used to estimate constituent concentrations in all model blocks contained within a wireframe boundary based on geologic boundaries of the project mineralized bodies. For each block in the block model, 2 to 8 samples were considered within a search ellipsoid, and a weighted average of the parameter of concern was calculated.

Using this model, the spatial variation of acid-generating waste rock within the pit and in the pit walls was determined. Material with an estimated net neutralizing potential less than -20 tonnes/kilotonne was designated as potentially-acid generating; material with an estimated net neutralizing potential greater than -20 tonnes/kilotonne was designated as non-acid generating.

14.3 MINERAL RESOURCES

The mineral resources were developed with the Whittle multi-algorithm software suite to determine the component of the deposit with reasonable prospects of economic extraction. For the resource pit shell, economic benefit was applied to inferred mineralization. However, no economic benefit was applied to inferred mineralization in the later determination of mineral reserves or in the economic analysis of the project.

The Whittle computer algorithm is a tool used to guide mine design. The algorithm applies approximate costs and recoveries along with approximate pit slope angles to establish theoretical economic breakeven pit wall locations.

The Whittle algorithm was also used as a guide to the design of the final mineral reserve pit. The detailed cost and process response input parameters for the mineral reserves are presented in Section 15. The majority of that information was also used as input for the determination of mineral resources.

The economic input for mineral resource determination was identical to that applied to the mineral reserve, with the following exceptions:

Differences Applied to Mineral Resource Whittle Pit Shell

1) The resource Whittle pit shell did receive economic credit for inferred-class material. Inferred was treated as waste for the mineral reserve.
2) The resource includes potentially leachable material processed at 4.82 $/tonne and above a 15 g/tonne silver cutoff. The reserve does not include any potentially leachable material.

The Corani mineral resource was contained in the Whittle pit and is summarized on Table 14-16. The resource statement on Table 14-16 includes the mineral reserves that are presented in Section 15.

The qualified person responsible for the estimation of the mineral resource was Terre Lane. Metal price changes could materially change the estimated mineral resources in either a positive or negative way.

At this time, there are no unique situations relative to environmental or socioeconomic conditions that would put the Corani mineral resource at a higher level of risk than any other Peruvian developing resource.

Table 14-16 Total Mineral Resources (Includes Both Resources and Reserves)

Total Mineral Resource, Includes Mineral Reserve							
Category	Ktonnes	Silver gpt	Lead %	Zinc %	Silver Million oz	Lead Million lb	Zinc Million lb
Measured	29,209	56.2	0.912	0.582	52.8	587	375
Indicated	181,902	40.7	0.741	0.495	238	2971.3	1983.5
Measured & Indicated	211,111	42.8	0.765	0.507	291	3,558	2,359
Inferred	31,231	40.6	0.742	0.512	40.8	510.6	352.4

Note: Cutoff Value : $11.00/tonne covers process and general and administrative costs.

Total Mineral Resource, Includes Mineral Reserve							
Category	Ktonnes	Silver gpt	Lead %	Zinc %	Silver Million oz	Lead Million lb	Zinc Million lb
Measured	29,209	56.2	0.912	0.582	52.8	587	375
Indicated	181,902	40.7	0.741	0.495	238	2971.3	1983.5
Measured & Indicated	211,111	42.8	0.765	0.507	291	3,558	2,359
Inferred	31,231	40.6	0.742	0.512	40.8	510.6	352.4

Note: Cutoff Value : $11.00/tonne covers process and general and administrative costs.

Table 14-17 Total Mineral Resource of Potentially Leachable Material

Total Mineral Resource, Includes the Mineral Reserve			
Category	Ktonnes	Silver gpt	Silver Million oz
Measured	5,006	38.0	6.12

Indicated	19,690	23.1	14.61
Measured & Indicated	24,697	26.1	20.72
Inferred	8,722	25.1	7.03

Total Mineral Resource, Includes the Mineral Reserve			
Category	Ktonnes	Silver gpt	Silver Million oz
Measured	5,006	38.0	6.12
Indicated	19,690	23.1	14.61
Measured & Indicated	24,697	26.1	20.72
Inferred	8,722	25.1	7.03

Note: Cutoff Grade: 15 g/tonne silver, assuming 50% recovery and leach processing of 4.80 $/tonne.

Figure 14-30, Figure 14-31, and Figure 14-32 compare cumulative frequencies of silver, lead, and zinc. Each plot shows the cumulative frequency of composite grades, nearest-neighbor block grades, inverse distance block grades, and kriged block grades. The plots show how a nearest-neighbor model estimate follows the trends of the composite assay data very closely in all of the metal models. The inverse distance and kriged models show how, once spatial variability is applied, the frequency plot of the population takes on the smoothed distribution that is expected of a grade model.

Figure 14-33 and Figure 14-34 are examples of a plan and cross section of the block model with drillhole intercepts plotted with composite assay data and the block grade model for silver. The visual inspection shows how the model agrees with the assay data and smooths the grade among input data points.

Figure 14-30: Cumulative Frequency of Silver Grades – Composites, Kriged, ID3, and Nearest Neighbor

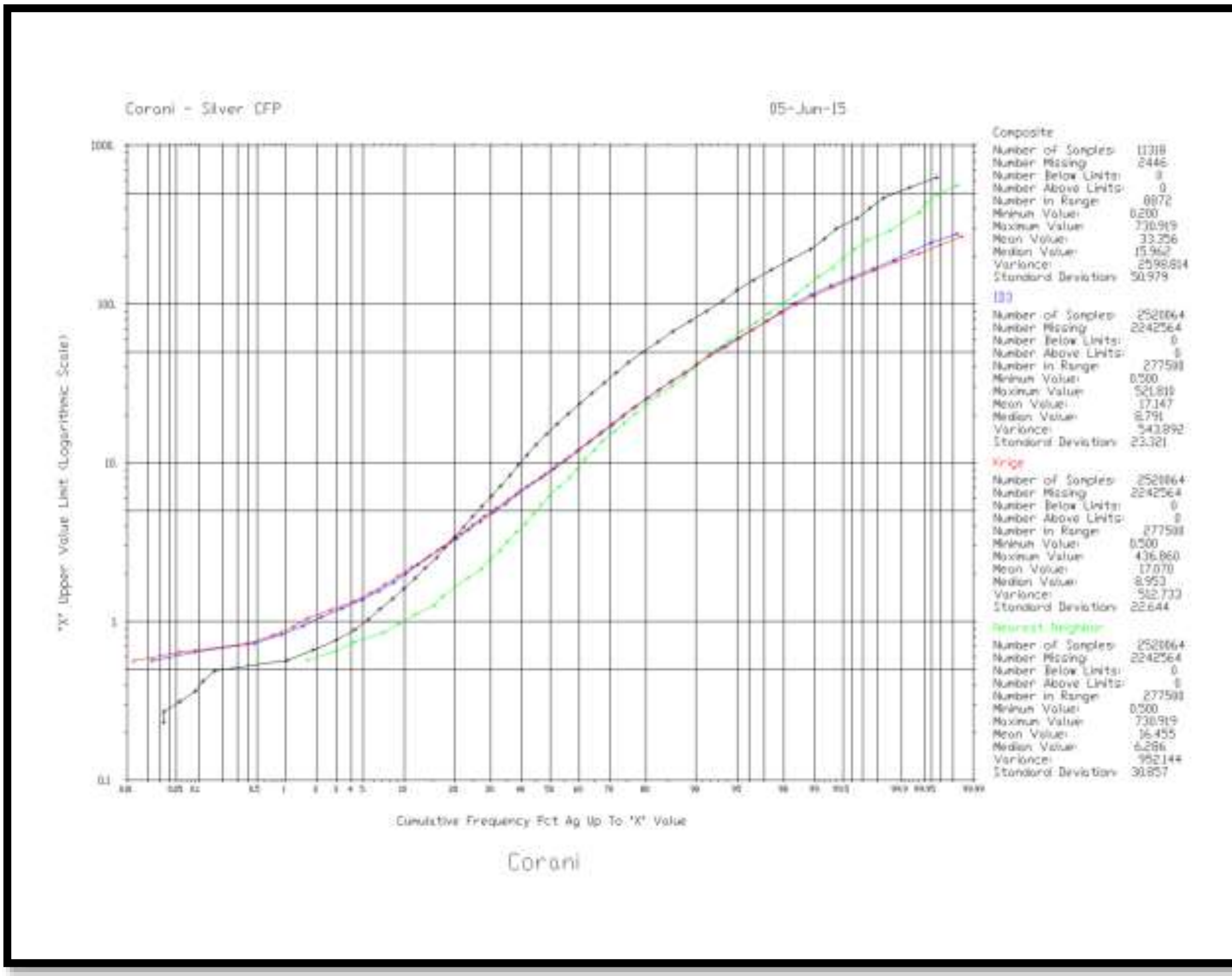


Figure 14-31: Cumulative Frequency of Lead Grades – Composites, Kriged, ID2.5, and Nearest Neighbor

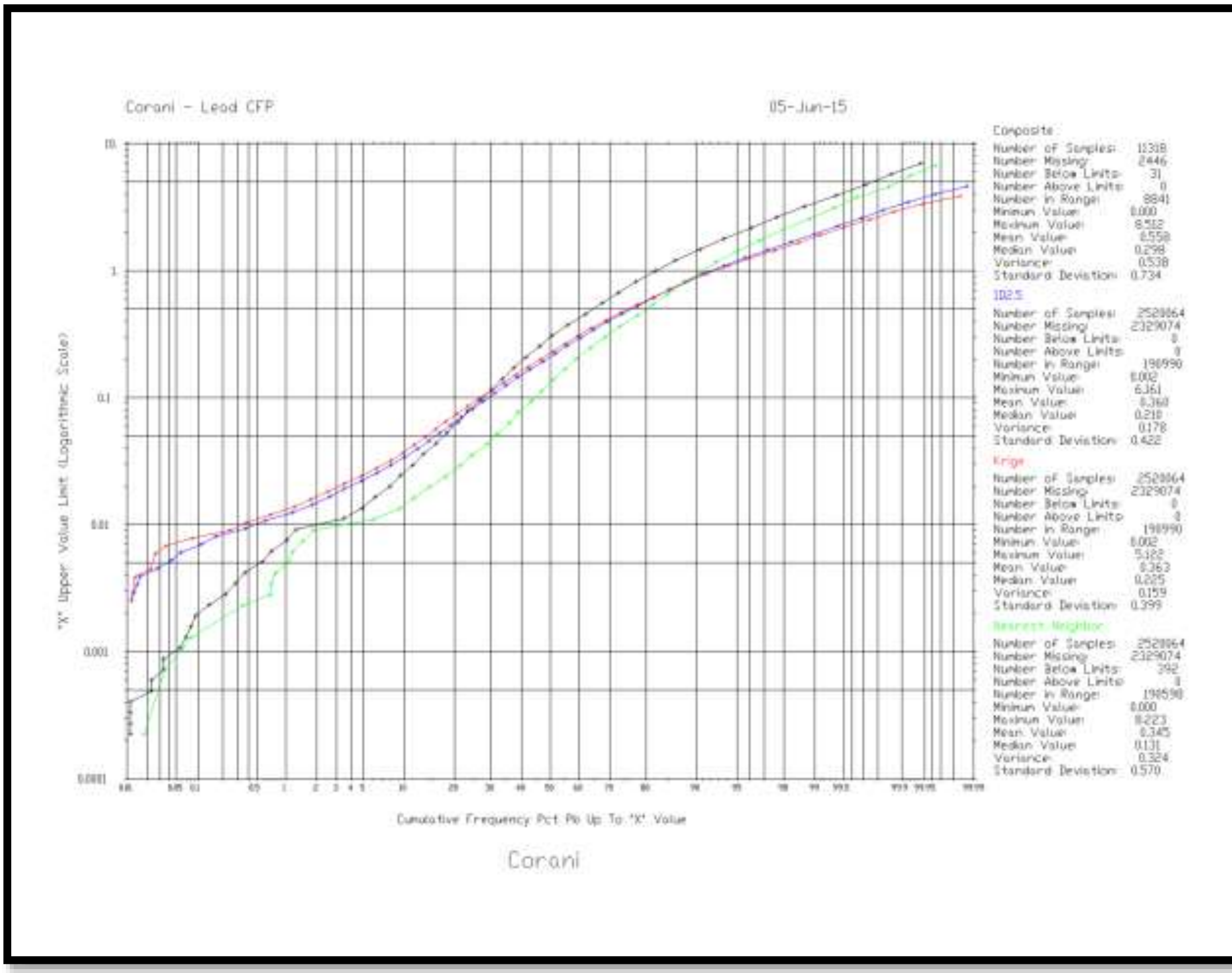


Figure 14-32: Cumulative Frequency of Zinc Grades – Composites, Kriged, ID2.5, and Nearest Neighbor

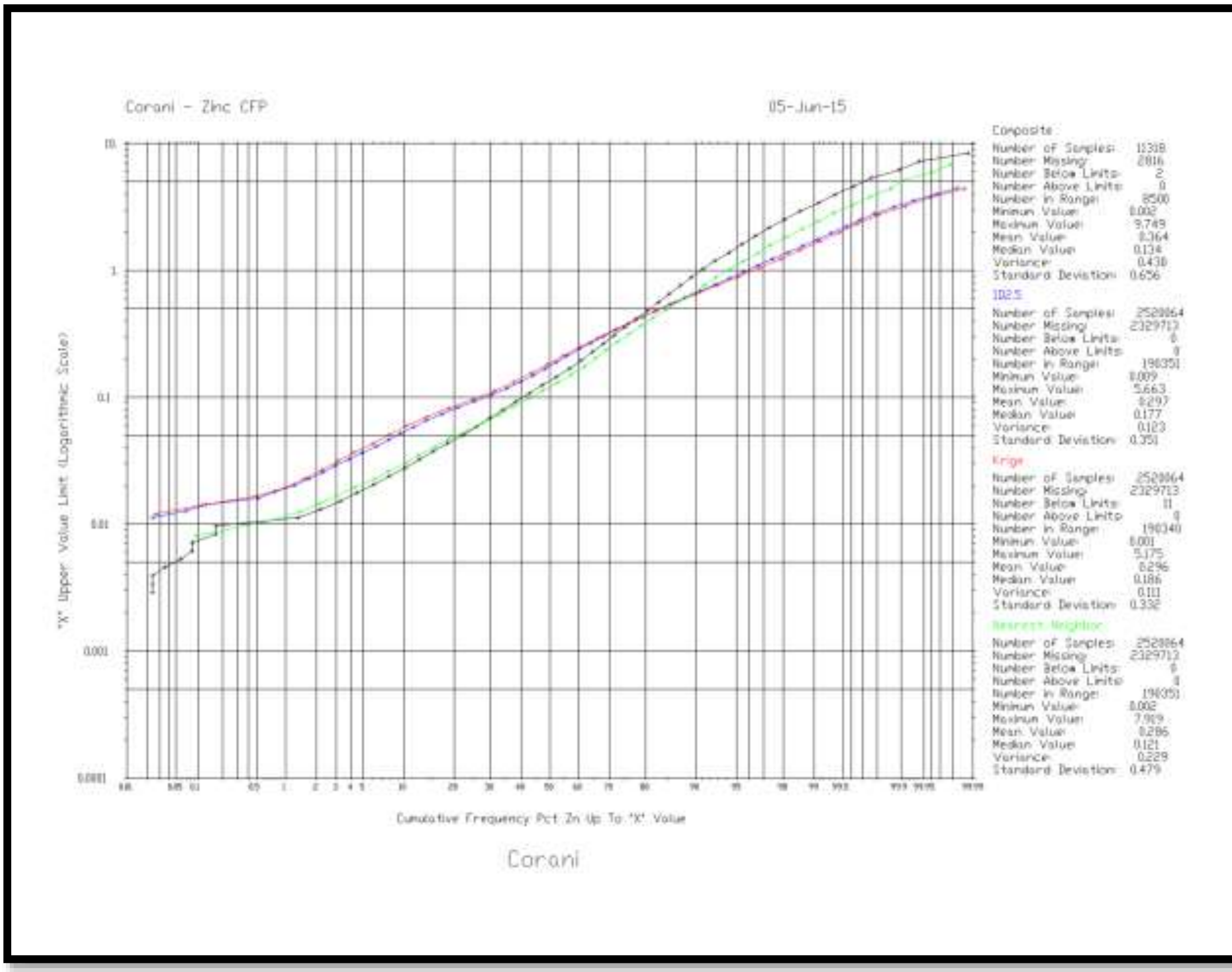


Figure 14-33: Silver Composite Assays and ID3 Block Grades on Bench 4914

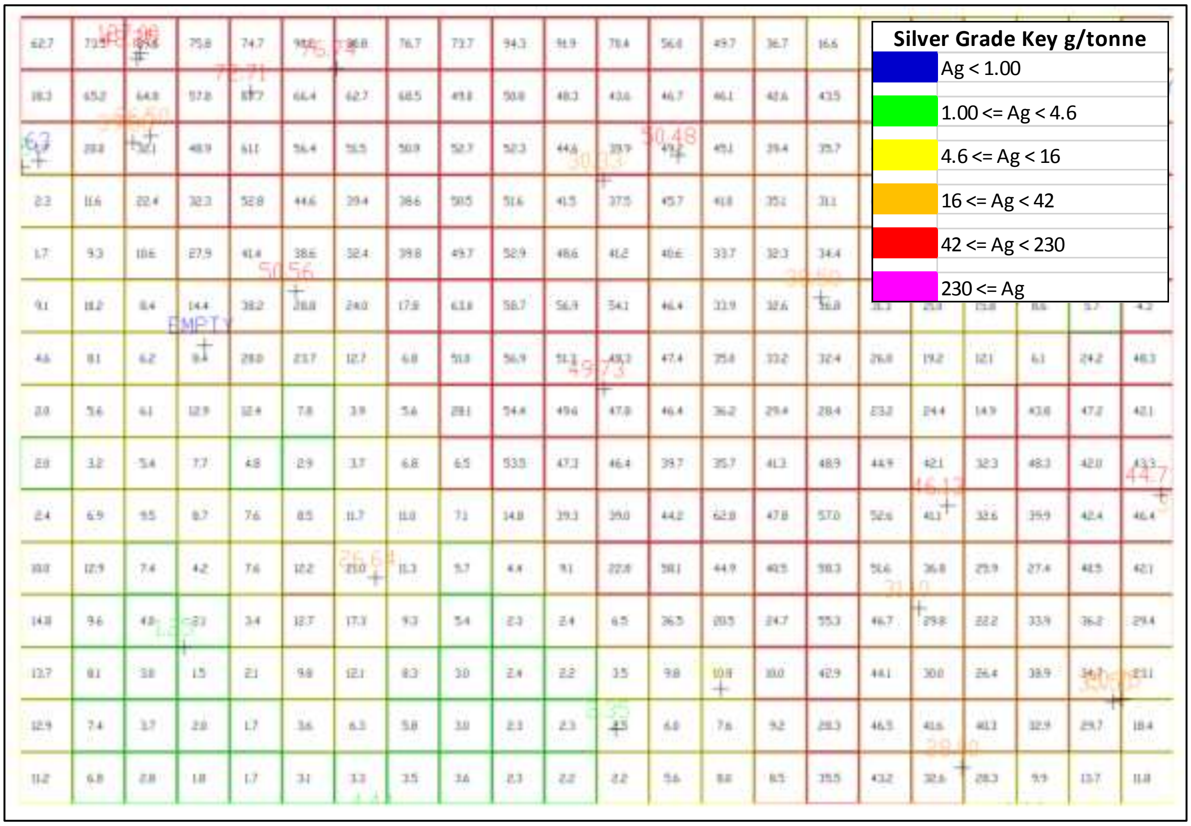
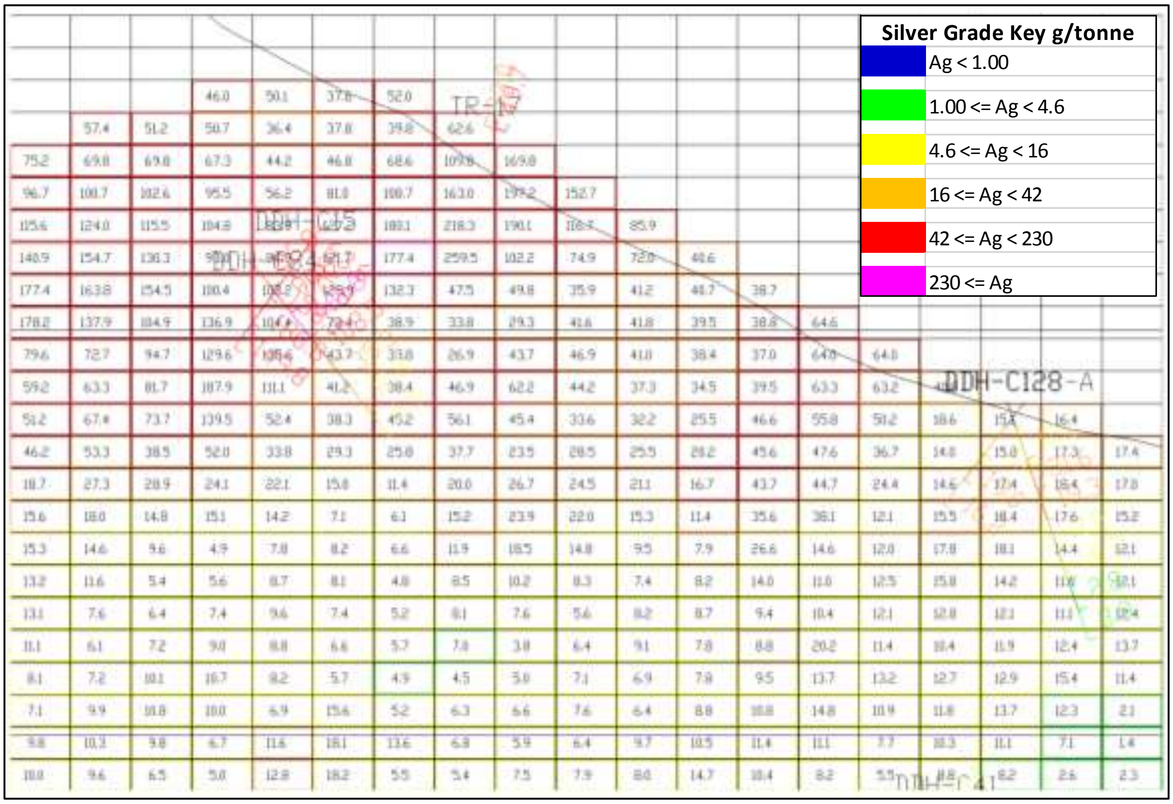


Figure 14-34: Silver Composite Assays and ID3 Block Grades Cross Section 8,448,252.5 N



15 MINERAL RESERVE ESTIMATES

The mineral reserve is the total of all proven- and probable-category ore that is planned for production. The mine plan that is presented in Section 16 details the production of that reserve. The mineral reserve was established by tabulating the contained tonnes of measured and indicated material (proven and probable) within the designed final pit geometry with a cutoff value greater than the breakeven value. The final pit design and the internal phase designs were guided by the results of Whittle pit optimization analyses.

15.1 WHITTLE PIT SHELLS

Whittle software uses multiple algorithms to create a near-optimum shell focusing mostly on the Lerchs-Grossman algorithm. It uses metal prices, average cost inputs, and a block-by-block recovery model with an approximate slope angle to produce a theoretical maximum pit containing the highest net economic value possible.

By multiplying the metal prices by a "revenue factor" while keeping other economic inputs the same, the Whittle software was used to produce a series of pit shells that are larger or smaller than the base pit relative to the change in metal prices. The pit shells produced using Whittle are theoretical pits, without consideration of vehicle access, minimum working area, catch benches, etc. All pit shells are intended to show approximate pit wall configurations, and the output is not meant to represent feasible designed pit walls. The smaller pits at lower metal prices are used as targets for high-yield starter pits and early phases in the mining sequence.

Economic inputs for Whittle are summarized in Table 15-1. The result from Whittle (Figure 15-1) was used to design the ultimate pit limits (Figure 15-2). The Whittle results for the adopted series of revenue factors were also calculated using these inputs, to produce a range of pit shells later used to guide phasing. The detailed development of the ultimate and phased pits is described in Section 16. The phased pits are then scheduled to create the production schedule, which in turn forms the basis from which the feasibility study capital and operating costs are estimated. The estimated capital and operating costs are then used in the financial model.

Figure 15-3 shows a plan view of elevation 4938 m, with nested Whittle pit shells representing revenue factors 1.00, 0.90, 0.80, 0.70, and 0.60. This illustrates Whittle's ability to target areas of successively higher net value with corresponding lower metals prices. The results are used to guide detailed layout, phasing, and scheduling of the mine plan.

Table 15-1: Whittle Inputs

Mining Cost per Tonne of Rock, Ore or Waste	$1.50
Mining Cost per Tonne of Bofedal	$6.38
General and Administrative Cost per Tonne of Ore Processed	$1.51
Process Cost per Tonne of Ore	$9.49
Slope Angles	
Este and Main Zones	42 degrees
Minas	46 degrees
Bofedal	15 degrees
Capital Replacement	
Mining, Cost per Tonne of Material Mined	$0.17
Milling, Cost per Tonne of Ore Processed	$0.11
Discount Rate	5%

Base Case Metal Price	
Silver = $20/oz	
Lead = $0.95/lb	
Zinc = $1.00/lb	
Lead Concentrate	**Zinc Concentrate**
Payable Metal	
Lead	Zinc
Pay 95% of contained metal	Pay the lesser of:
	85% of contained metal
	8% deduct from concentrate grade
Silver	Silver
Pay the lesser of:	Pay 70% of (contained metal - 3.5 ounces/tonne)
95% of the contained metal	
50 g/tonne deducted from concentrate grade	
Treatment Charge	
200 $/tonne	200 $/tonne
+4% for lead price over $2000 per tonne	+4% for zinc price over $2200 per tonne
Penalties	
$1/tonne for each 1% of zinc grade above 8%	$0
Refining Charge	
$1.50/payable oz. of silver	$0
Transportation & Sales Cost	
$120/tonne	$120/tonne

The metal recovery equations applied to the Whittle analysis (detailed in Section 13) are as follows:.

$$\begin{aligned} Lead\ Recovery &= 61.9 - 40.9 \times \max(0, 0.57 - Zn) + 7.7 \times \max(0, galena - 0.38) + 45.4 \\ &\times \max(0, 0.37 - goethite) - 0.12 \times \max(0, elevation - 4891) + 32.9 \\ &\times \max(0, 0.27 - MnOx) - 6.21 \times \max(0, pyrite - 1.07) - 16.4 \times \max(0, 1.07 \\ &- pyrite) \end{aligned}$$

Note: Maximum lead recovery is 98%; minimum lead recovery is 2%.

$$\begin{aligned} Zinc\ Recovery &= 78.7 - 50.6 \times \max(0, 1.02 - Zn) - 0.15 \times \max(0, elevation - 4901) - 5.4 \\ &\times \max(0, pyrite - 1.9) - 11.2 \times \max(0, 1.9 - pyrite) + 104.1 \times \max(0, Cu - 0.03) \\ &+ 1620.2 \times \max(0, 0.03 - Cu) \end{aligned}$$

Note: Maximum zinc recovery is 83%; minimum zinc recovery is 0%.

$$Silver\ Recovery\ in\ Lead\ Concentrate = 0.67 \times Lead\ Recovery + 12$$

$$Silver\ Recovery\ in\ Zinc\ Concentrate = 41 - 0.41 \times (Silver\ Recovery\ in\ Lead\ Concentrate)$$

To match the metallurgical testing results, lead concentrate grades were fixed at 56.6% lead and zinc concentrate grades were fixed at 52.9%. Silver grade in the zinc concentrate was limited to a maximum value of 385 grams per tonne in the production schedule.

Figure 15-1: Whittle Pit – RF100 ($20/oz Ag, $0.95/lb Pb, $1.00/lb Zn)

Figure 15-2: Final Pit Design Including Benches, Ramps, and Haul Roads

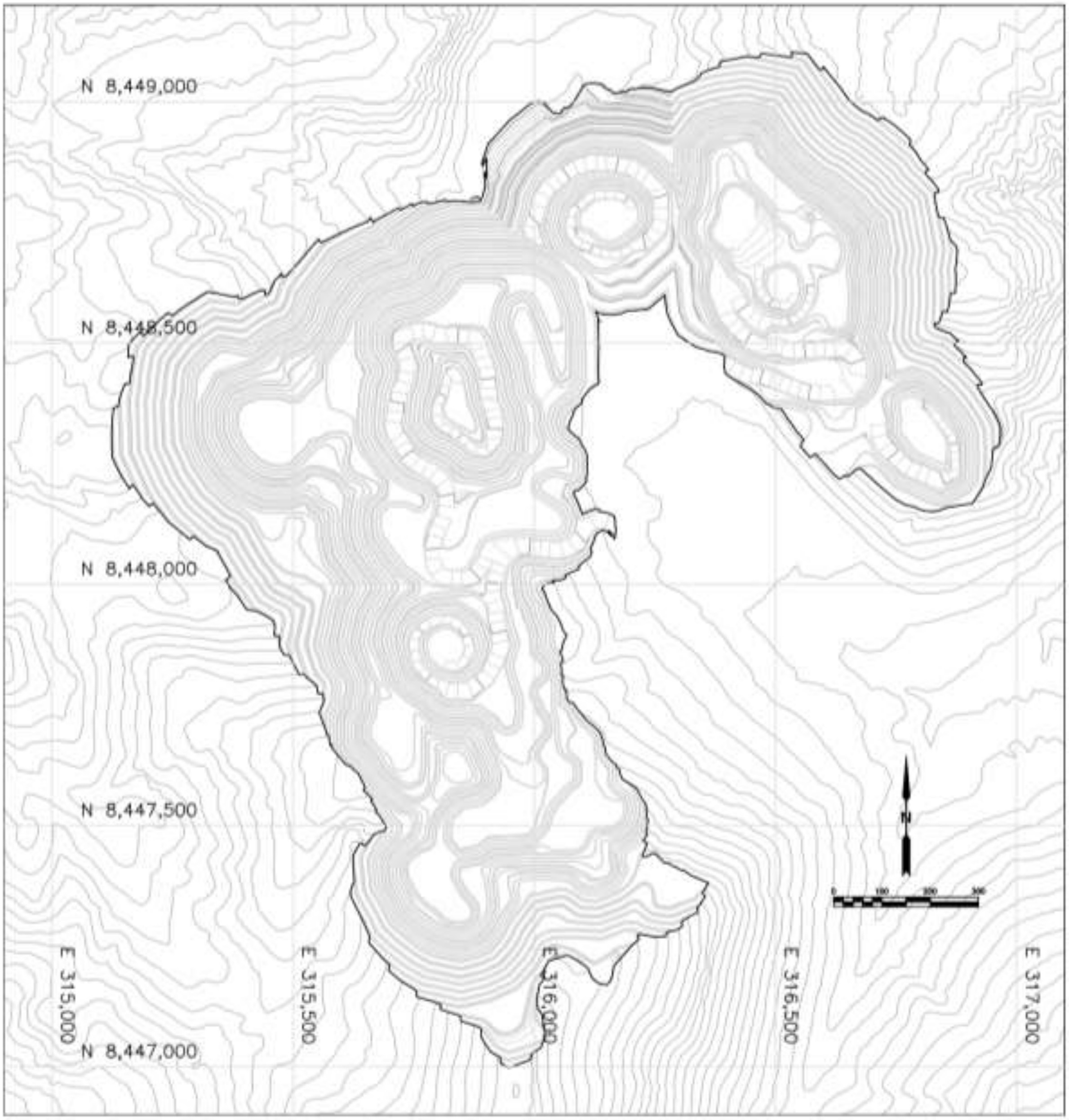
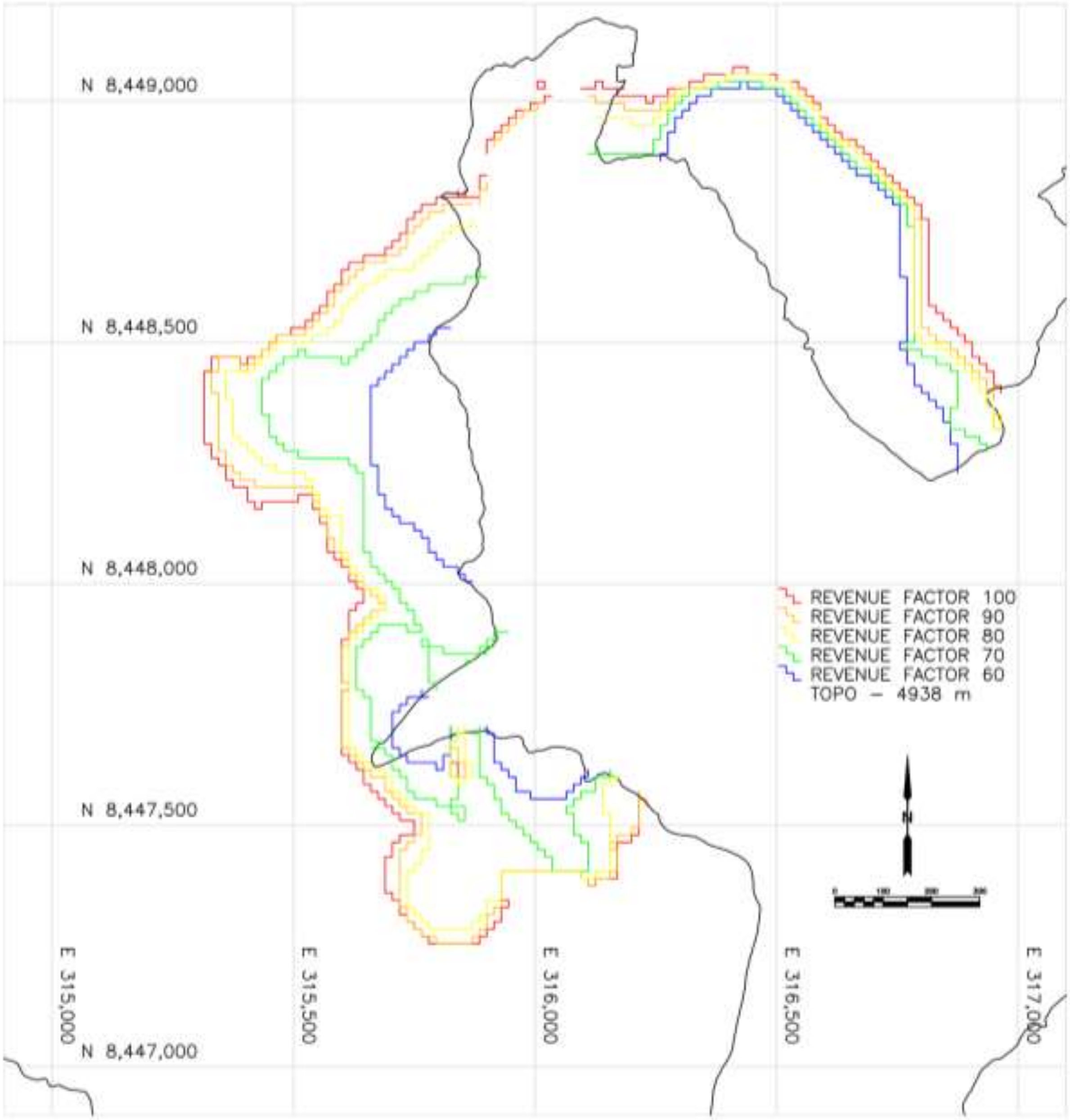


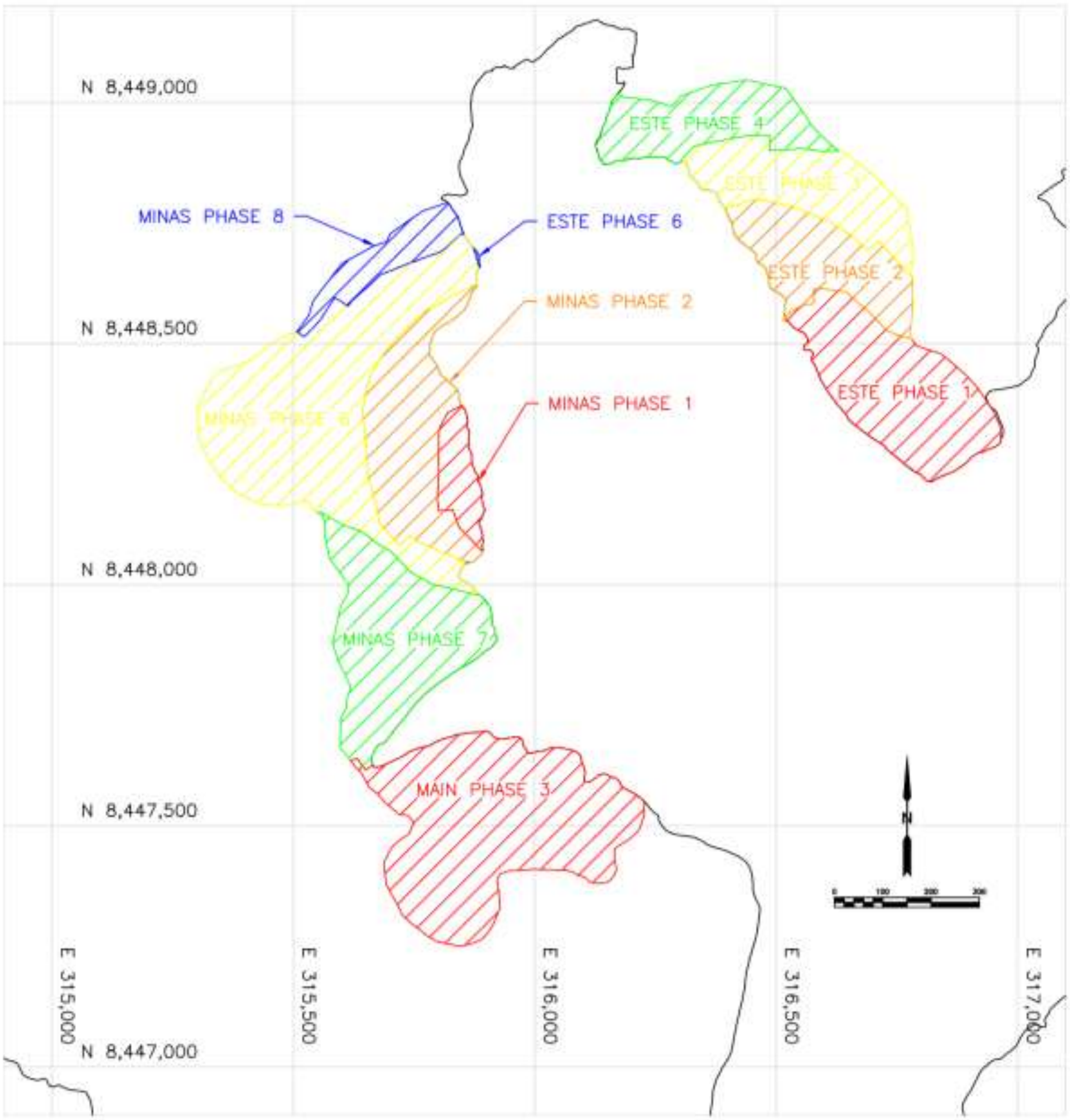
Figure 15-3: Whittle Pit Shells at 4938 m Elevation



15.2 PHASE DESIGN

After the ultimate pit limits (see Figure 15-2) had been determined, phases within the ultimate volume were designed. Design factors taken into account in the design for each pit phase included the following: road access; pit wall slope requirements by zone; distribution of the stripping ratio across phases; and, the maximum number of benches feasible to mine in a single year. Figure 15-4 shows how the phases fit together at elevation 4938 m.

Figure 15-4: Pit Phases at 4938 m Elevation



15.3 MINERAL RESERVE ESTIMATE

The Mineral Reserve is the sum of the Proven and Probable ore within the ultimate pit design that has a lead plus zinc concentrate NSR above the processing plus G&A costs. GRE used a cutoff value higher than the calculated value to improve cash flow and maximize project economics. The Mineral Reserve shown in Table 15-2 is the total ore that is processed in this feasibility study.

To calculate the NSR value, metal recovery, transportation, deductions, and smelter costs were derived as described previously. The NSR cutoff value used a processing cost of $9.49/tonne and a G&A cost of $1.51/tonne ($11.00 per tonne) for all material. GRE used Cutoff Values of $4, $8, and $12 per tonne above the $11.00 processing and G&A cost resulting in $11, $15, $19, and $23 per tonne NSR cutoffs for different phases of the pit schedule as shown in Table 15-3.

The Mineral Resources were generated within the $30.00 Silver, $1.425 Lead, and $1.50 Zinc price pit shell and the calculated $11/tonne NSR cutoff. Potentially leachable silver mineralization that falls within that pit shell are included if the recovered silver exceeds the estimated leach cost, which equates to a 15 gram per tonne cutoff. The Mineral Resources in addition to the Mineral Reserve are summarized in Table 15-2.

Table 15-2: Mineral Reserves and Resources

Mineral Reserves, variable $23.00-11.00 NSR cut-off							
Total	Ktonnes	Silver gpt	Lead %	Zinc %	Silver Million oz	Lead million lb	Zinc million lb
Proven	19,855	69.1	1.09	0.72	44.1	478.7	313.4
Probable	117,843	48.6	0.88	0.57	184.3	2289.2	1470.7
Proven & Probable	137,698	51.6	0.91	0.59	228	2,768	1,784

Mineral Resources in Addition to Reserves, $11.00 NSR cut-off, 15 g/tonne Ag cutoff (oxide)							
Total	Ktonnes	Silver gpt	Lead %	Zinc %	Silver million oz	Lead million lb	Zinc million lb
Measured	14,360	32.01	0.34	0.19	14.8	108.4	61.6
Indicated	83,749	25.37	0.37	0.28	68.3	682.2	512.8
M&I	98,109	26.34	0.37	0.27	83.1	790.6	574.4
Inferred	39,953	37.20	0.58	0.40	47.8	510.6	352.4

Notes:
The Mineral Reserve is within the 20 $/oz designed pit and utilizes variable NSR cutoff values to maximize early cash flows. This is the tonnage processed in the economic model.

The Mineral Resource is the tonnage contained within the 30$/oz silver, 1.425 $/lb lead, and 1.50 $/lb zinc prices Whittle pit using a 20 $/oz silver, 0.95 $/lb lead, and 1.00 $/lb zinc prices at a cutoff of 11 $/tonne NSR plus potentially leachable oxide at a 15g/t Ag cutoff ($4.80/tonne using 50% recovery in addition to ore already categorized within the Mineral Reserve.

Table 15-3 Variable NSR Cutoff for Pit Phases

Pit	Phase	NSR $ Cutoff
Este	1	23
Minas	1	23
Este	2	23
Minas	2	19
Este	3	19
Este	4	19
Este	5	19
Este	6	15
Minas	3	15
Minas	4	15
Minas	5	15
Minas	6	11
Minas	7	11
Minas	8	11
Minas	9	11
Minas	10	11
Main	1	11
Main	2	11
Main	3	11
Main	4	11
Main	5	11

The current Mineral Reserve Estimate is lower than the 2011 estimate. Each of the following items contributed:

- Metal Recovery Model, GRE's recovery model is continuous with some lower estimated recovery, and some higher estimated recovery. Though the total silver and zinc recovery went up slightly, some previous ore tonnes were lost because of the lower estimated recovery in some areas. The lower overall lead recovery also contributed.
- Density, GRE used a combination of lead, zinc, and silver grade to assign density, the previous method only considered silver grade. The revised method resulted in a slightly lower overall density.
- Estimation Algorithm, GRE use inverse distance to the 2.5 and 3 powers to estimate grade. The previous estimate used two stages of kriging resulting in more grade smoothing from oxidized, to transition, to sulfide ore types. GRE's estimate had slightly less smoothing and reduced ore tonnes slightly.
- Resource classification methodology, GRE's resource classification method was slightly more conservative than the previous estimate.
- Whittle Parameters (Processing and Mining Costs), GRE used slightly higher mining and processing costs in the Whittle pit optimization then the previous floating cone, in part from inflation and partially inclusion of capital replacement costs. The higher costs reduced the Mineral Reserve slightly. The cost used in 2011 and 2015 are compared below.

	2011	2015
Mining Cost	$1.34	$1.67
Processing Cost + G&A	$9.20	$11.00

- Variable cutoff grade without a low grade stockpile, GRE did not include processing of low grade ore at the end of the project life, which reduced the project life by a couple years. GRE preformed a quick calculation of the cost to prepare a storage area and encapsulate the ore to prevent oxidation, then estimated the cost of rehandle and process the ore at the end of the mine life and found it did not help the project economics. Future studies and test work should investigate the low grade stockpile concept in more detail.

GRE investigated the Corani pit optimization sensitivity to changes in mining, process, and G&A operating cost, as well as metal price, as part of the study. GRE considered changes of +/- 10%. The maximum change in ore tonnage mined by the Whittle optimizer was 4.2 million tonnes (3.2%). This small change indicates that the final Feasibility Study operating costs, which are slightly higher than the base case Whittle numbers, will not produce a significant change in pit design at the detailed design phase of project development.

Significant changes in metal price, operating costs, or recoveries could materially change the estimated mineral resources in either a positive or negative way.

At this time, GRE knows of no unique situations relative to environmental or socio-economic conditions that would put the Corani Mineral Reserves at a higher level of risk than any other Peruvian developing resource.

16 MINING METHODS

The Corani deposit is planned to be mined using conventional open pit mining methods. The mine design and planning are based on the resource model and reserve estimate as indicated in the previous sections.

16.1 SUMMARY

The mine plan considers extraction of the proven and probable ore material included in the mineral reserve presented in Section 15. The mine plan has been developed to deliver 7,875 ktonnes of ore per year (22,500 tpd x 350 processing days) to the crusher for processing by flotation to produce two concentrates: 1) lead-silver, and 2) zinc-silver concentrate.

The mine plan development included the following:

1) Ultimate pit design including benches, ramps, and haul roads;
2) Pit phase design based on the incremental Whittle Shells;
3) Detailed pit phase designs with benches, ramps, and haul roads;
4) Mine production scheduling;
5) Waste storage design and material allocation;
6) Time sequence mine plan drawing development; and
7) Equipment and manpower requirement calculations.

The following sections detail the development of the mine plan.

16.2 ULTIMATE PIT DESIGN

The Whittle analysis presented in Section 15 served as the basis for creating the final pit designs utilized for mine planning and for the statement of mineral reserves. The objective of the Whittle pit optimization was to maximize the economic extraction of the mineral resources contained in the block model. The optimal pit shell selected was based on the combined principles of incremental net present value (NPV) and cash flow, and included an analysis of best and worst case NPV, strip ratios, and smelter schedules on a shell-by-shell basis. The RF100 ($20/oz Ag, $0.95/lb Pb, $1.00/lb Zn) Whittle pit shell served as the basis for designing the ultimate pit to include benches, ramps, and haul roads, as well as access and tie-in to other project facilities. RF100 refers to the revenue factor for each block in the block resource model.

The ultimate pit design was developed using Maptek's Vulcan™ (Vulcan) mine design software. The final pit design is the result of multiple iterations in which ramp locations and configurations have been examined in an effort to maximize recovery of the resource and upfront cash flow, minimize waste stripping, create benches with dimensions to be operationally workable, and provide for efficient haulage routes for the mobile equipment. Permanent haul roads and ramps were located on the interior valley side of the pit wall where possible, minimizing waste rock addition.

The ultimate overall pit slope is based on analysis performed by McDonald Engineering Services (MES), who determined that the Corani pit can be excavated to an overall slope of 42° in mineralized tuff, and 46° in post-mineral tuff. The bofedal domain includes an approximate 30 m deep soil deposit located in the valley bottom between the Este and Main pit areas. Pit slopes within the soil deposit have not been studied in detail, and consequently conservative pit slope assumptions were applied to this domain.

In practical application, these structural domains were applied to overall pits as shown in Table 16-1. These slopes provide a reasonable factor of safety against slope failure. The overall slope angle is limited by the assumed strength of the rock mass and a conservative seismic coefficient. Table 16-1 shows an example of the Este/Main pit slope design. Double benches (two benches per catch bench) were used to create a catch bench with sufficient width to be amenable to cleaning with a D8-sized dozer. Single benches were used where more selectivity in design would be beneficial.

Table 16-1: Pit Design Structural Domains

Domain	Bench Face Angle (°)	Operating Bench Height (m)	Operating Inter-Ramp Angle (°)	Overall Slope Angle (°)
Este Pit	70	8	46	46
Minas Pit	70	8	46	46
Main Pit	65	8	42	42
Bofedal	30	8	15	15

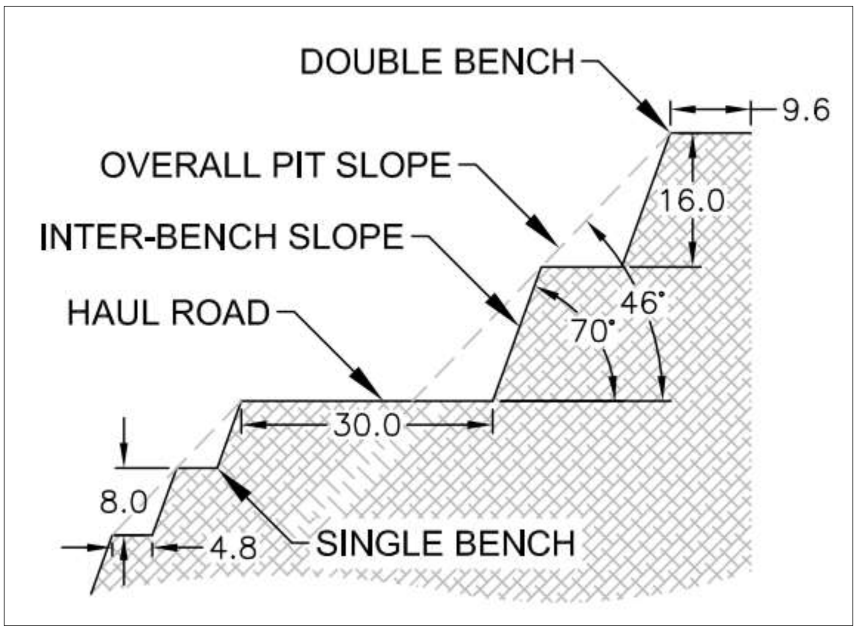


Figure 16-1: Pit Bench Design Example

Haul roads and ramps were designed to provide access to the pit phases and haulage routes to the ore crusher, waste dumps, construction areas, and truck shop. The roads have generally been laid out with a balance of cut-and-fill, or alternately utilizing waste rock fill that would otherwise have to be hauled to the waste rock dump. Internal haul roads are incorporated into the Main Waste Dump design. The haul road design is based on the following parameters:

- Total width two-way roads: 30 m
- Total width one-way roads: 15 m
- Running surface on two-way roads: 19.8 m
- Minimum road inside radius on corners: 8 m
- Berms and ditches: 5.1 m
- Maximum grade: 10%

16.3 PHASE DESIGN

A progression of Whittle shells based on incrementally decreasing metal prices was used to delimit the internal mining phases. The shells were created in Whittle by adjusting the prices of silver, lead, and zinc by a revenue factor, with 1.00 (100%) as the base case: 20 $/oz silver, 0.95 $/lb lead, and 1.00 $/lb zinc. The low metal price-based pits target the high-value blocks in smaller pits. The starting phases of the mine plan use these smaller pits as a guide. Each phase is based on successively larger pits, going from high-value blocks to low-value blocks until the phase design reaches the ultimate pit limits. Table 16-2 shows the progression of cutoff value as the pit phases advance. Figure 16-2, Figure 16-3, and Figure 16-4 show grades for estimated blocks, by block and by metal within the ultimate pit shell.

Table 16-2: Whittle Shell Cutoff Value Progression

Phase	NSR Cutoff ($/t)
este_ph1	23
minas_ph1	23
este_ph2	23
minas_rf1_ph2	19
este_ph3	19
este_ph4	19
este_ph5	19
este_ph6	15
minas_rf1_ph3	15
minas_rf1_ph4	15
minas_rf1_ph5	15

Phase	NSR Cutoff ($/t)
minas_rf1_ph6	11
minas_rf1_ph7	11
minas_rf1_ph8	11
minas_rf1_ph9	11
minas_rf1_ph10	11
main_rf1_ph1	11
main_rf1_ph2	11
main_rf1_ph3	11
main_rf1_ph4	11
main_rf1_ph5	11

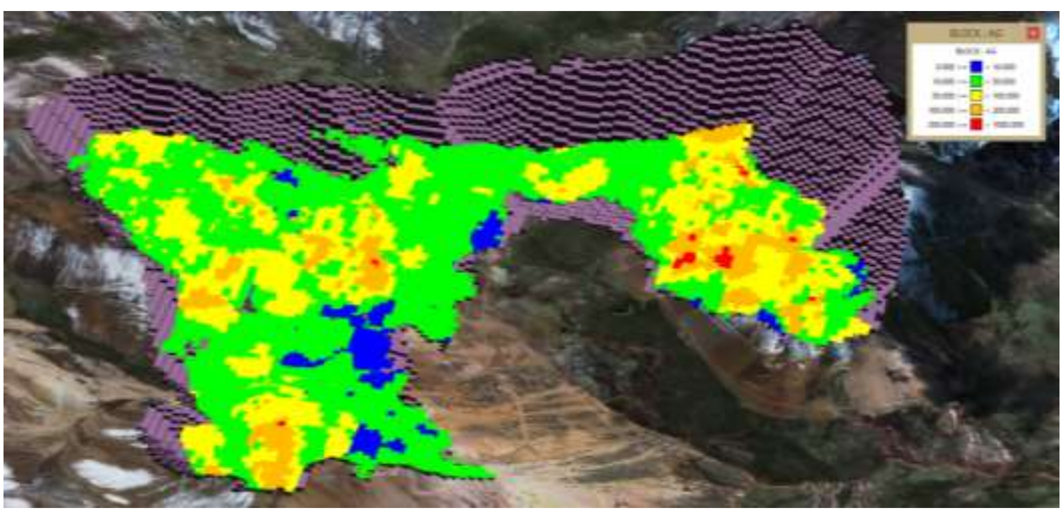


Figure 16-2: Whittle Shell – Silver Grade

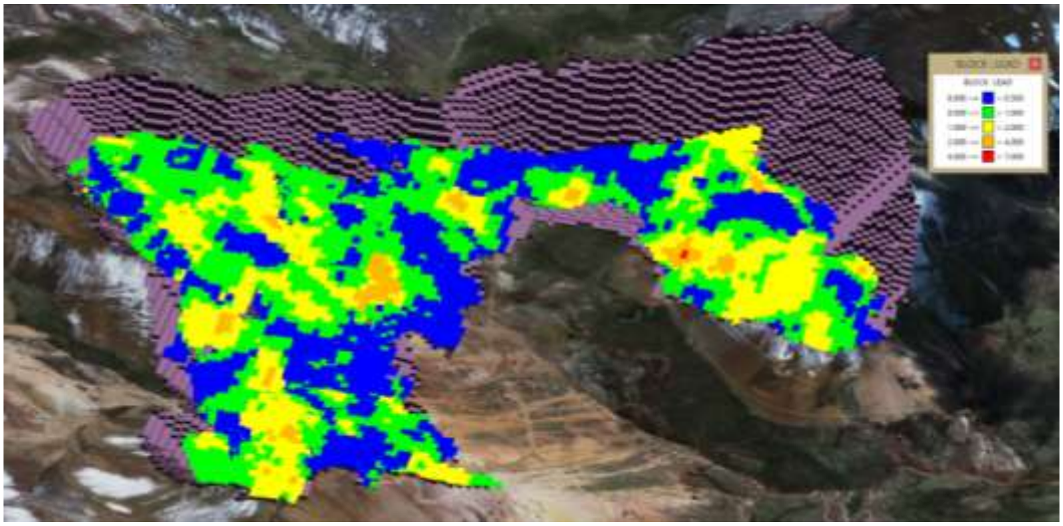


Figure 16-3: Whittle Shell – Lead Grade

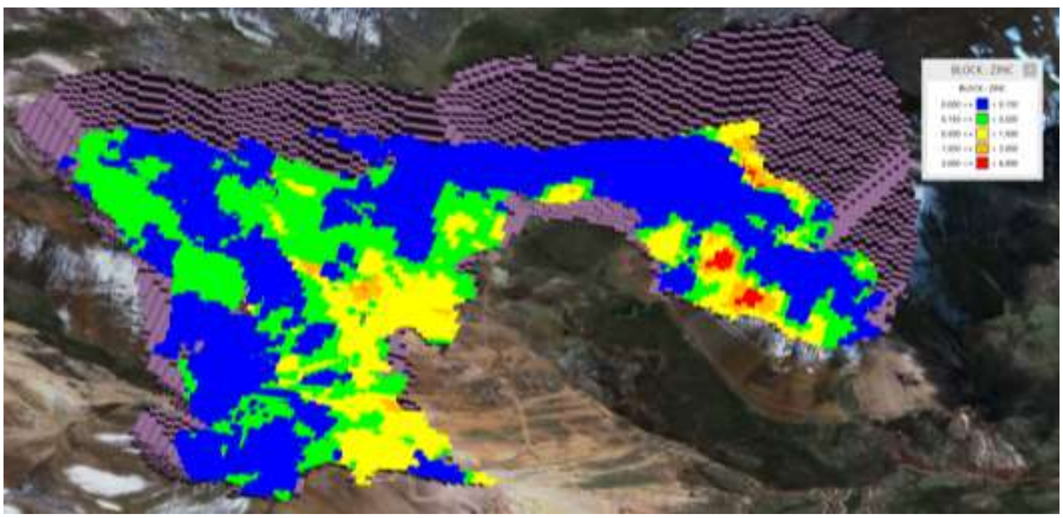


Figure 16-4: Whittle Shell – Zinc Grade

16.4 PIT PHASES

The series of Whittle pit shells that was generated at increasing revenue factors was used as a guide for the design of each mining phase and the pit development progression. Phases or pushbacks were laid out to provide access for the pre-stripping and mining of subsequent phases, culminating in the final designed pit. Phase designs were sized to provide practical operating room for mine equipment. The three distinct pit areas have a combined total of 21 mining phases as shown below:

- Este pit phases 1 through 6;
- Minas pit phases 1 through 10; and
- Main pit phases 1 through 5.

The phases are sequenced with the goal of producing the highest metal production and maximum cash flow early in the mine life, while completing mining in the Este pit early in the project life to facilitate backfilling that pit with filtered tailings and waste rock. It was necessary to schedule production from multiple phases simultaneously to smooth the strip ratio and balance mill feed grade. The key features of the planned mine schedule are as follows:

- The production plan is developed on a monthly basis and then summarized on a quarterly basis for the first three years, from year -1 to year 2, and annually thereafter.
- The mine will operate 365 d/a, producing an average of 21,575 tonnes of ore per day.
- The average productivity of the main loading fleet is 33 kt/d for ore and waste rock.
- The maximum material movement in the production schedule is 98 kt/d (365 daily basis).
- Mining from Este pit phase 6 is completed in year 6, allowing in-pit waste and tailings disposal to begin.
- Production from the Minas pit occurs over the entire project mine life.
- Production from Main pit begins in year 8 and ends just before the Minas pit at the end of the production life.

Pre-stripping requirements for each phase are determined by the geometric shape of said phase. Each phase was designed with a goal of maximizing ore value and minimizing pre-stripping requirements while maintaining access via haul road to the upper levels of the first phase of each mining area: Main, Minas, and Este. Each subsequent phase was designed to spread the remaining pre-stripping material roughly evenly through the remaining phases, thereby eliminating the need for an ever-changing mining fleet size.

Figure 16-5 through Figure 16-8 show the mine pit, Main Dump, and pit backfill limits at key points in the mine life. Figure 16-9 shows the pit phases at elevation 4938 m. The number in each area indicates the extraction sequence of the phases starting with Este phase 1 and finishing with the Main pit phase 5.

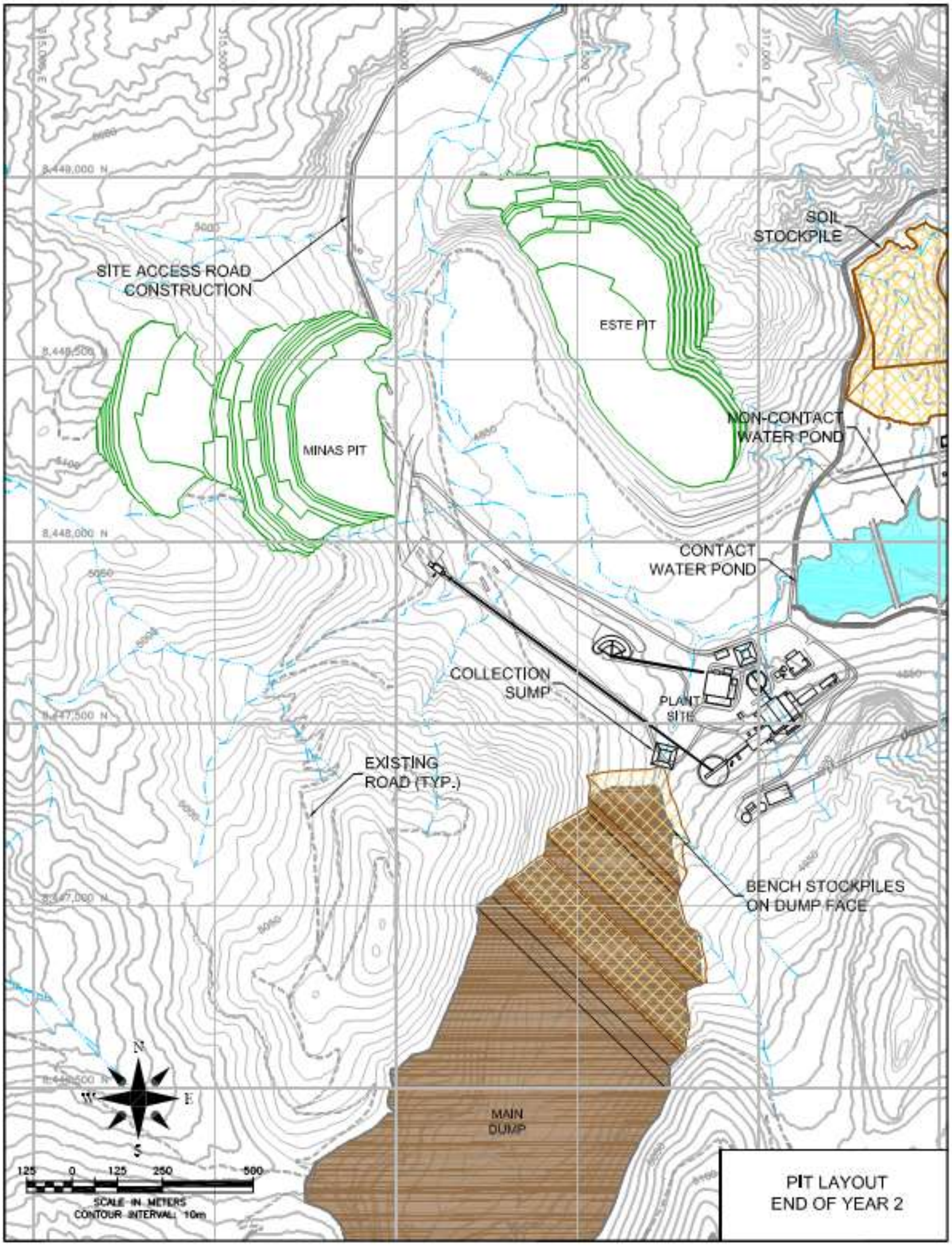


Figure 16-5: Corani Pit Phases in Plan at Year 2

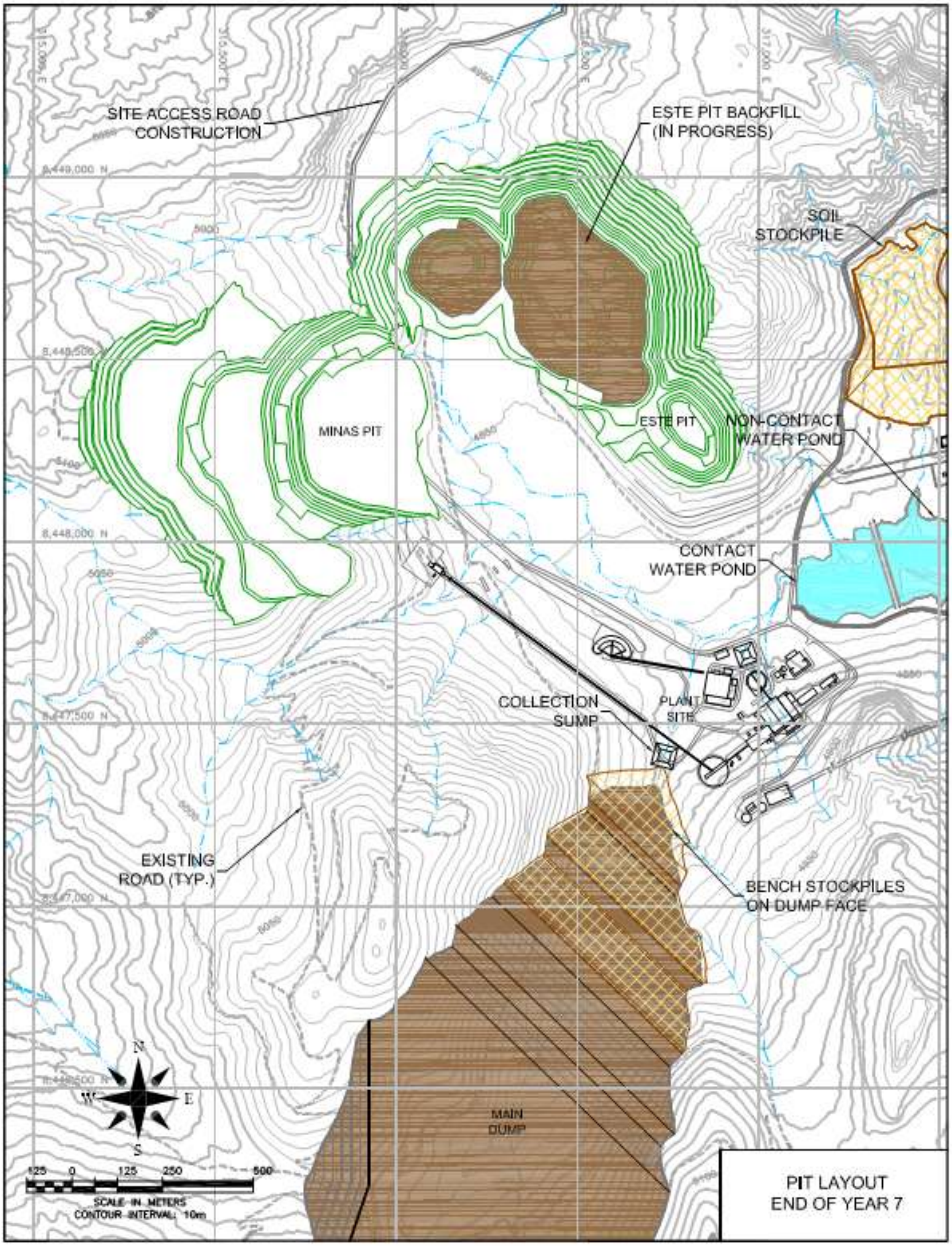


Figure 16-6: Corani Pit Phases in Plan at Year 7

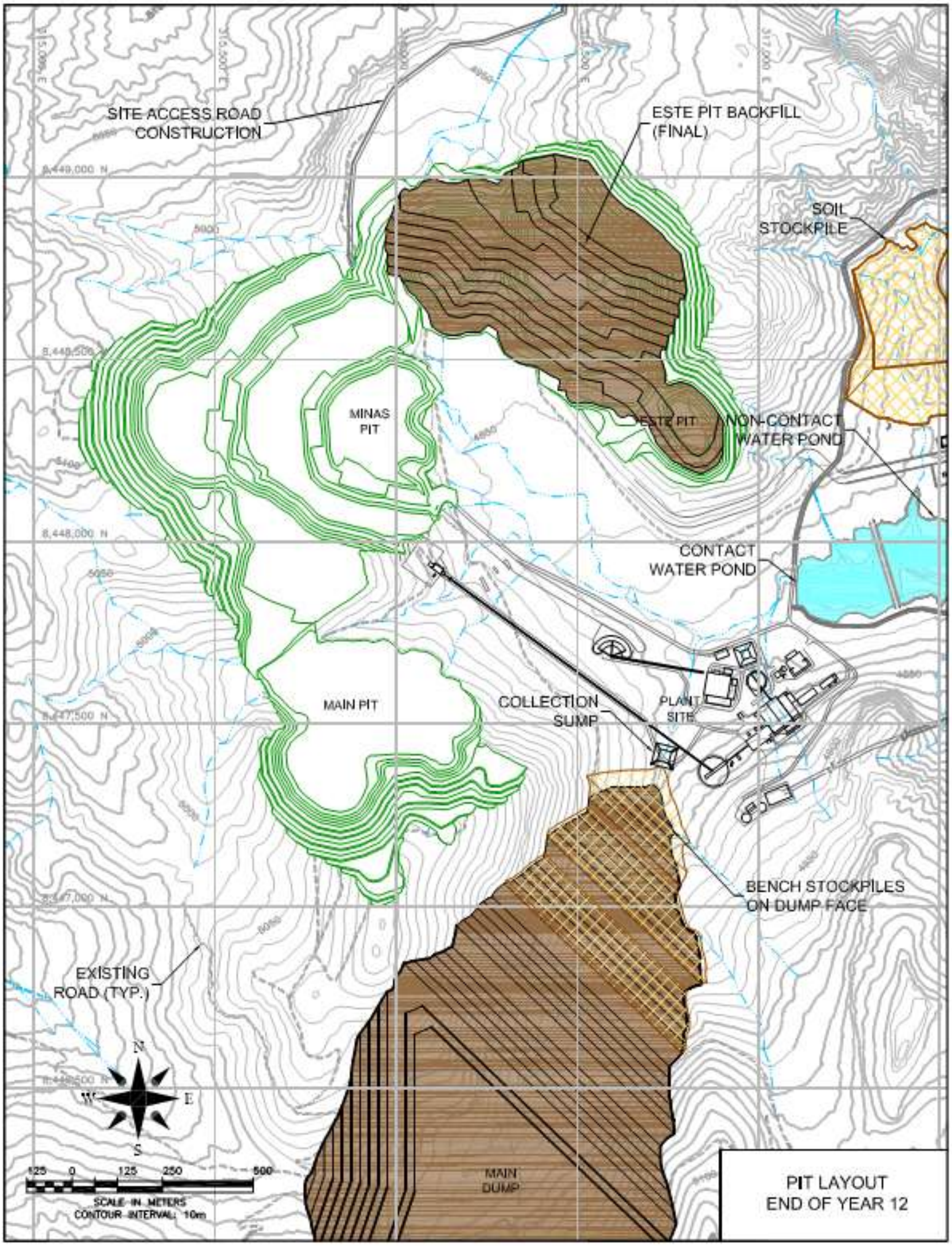


Figure 16-7: Corani Pit Phases in Plan at Year 12

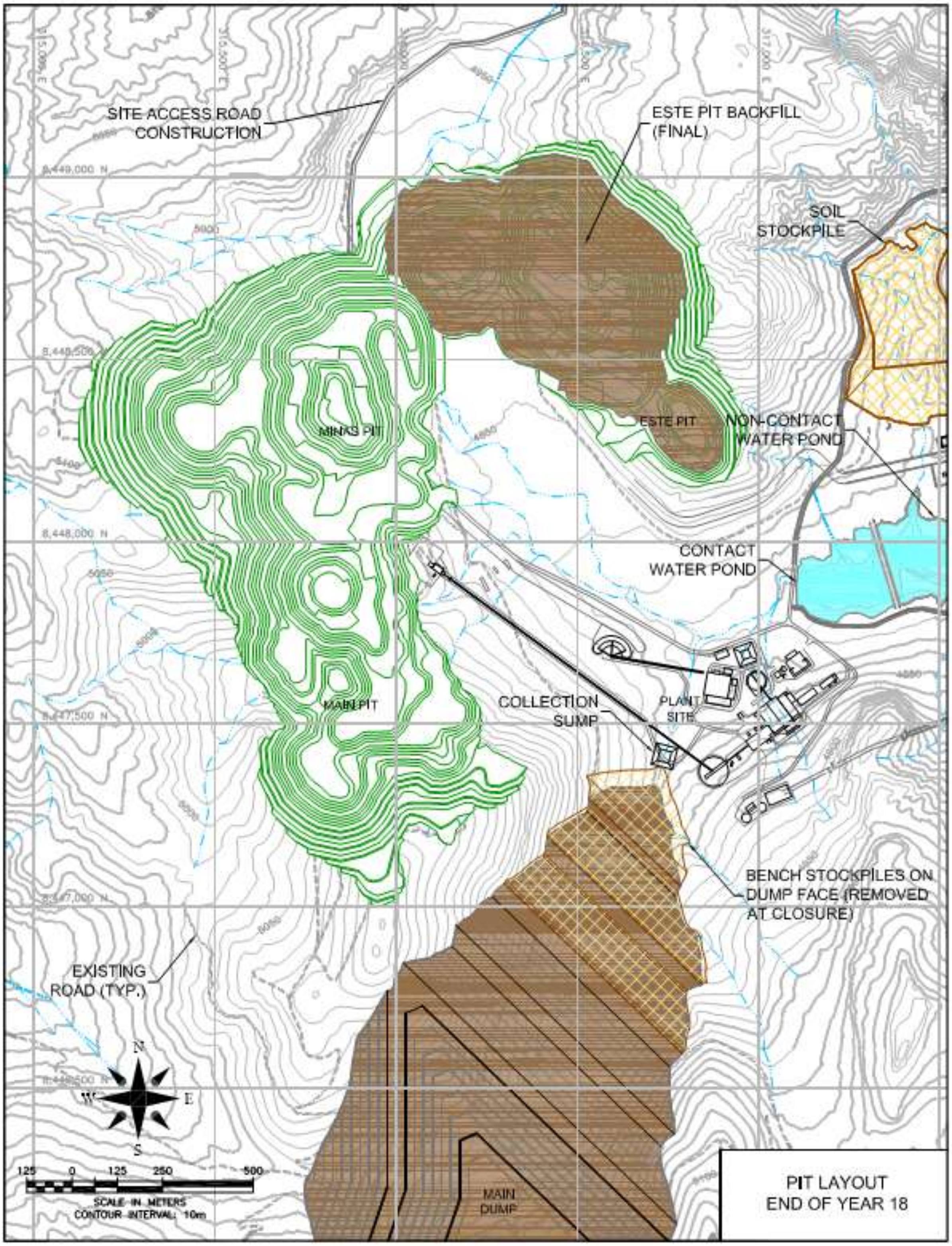


Figure 16-8: Corani Pit Phases in Plan at Year 18 (End of Mine Life)

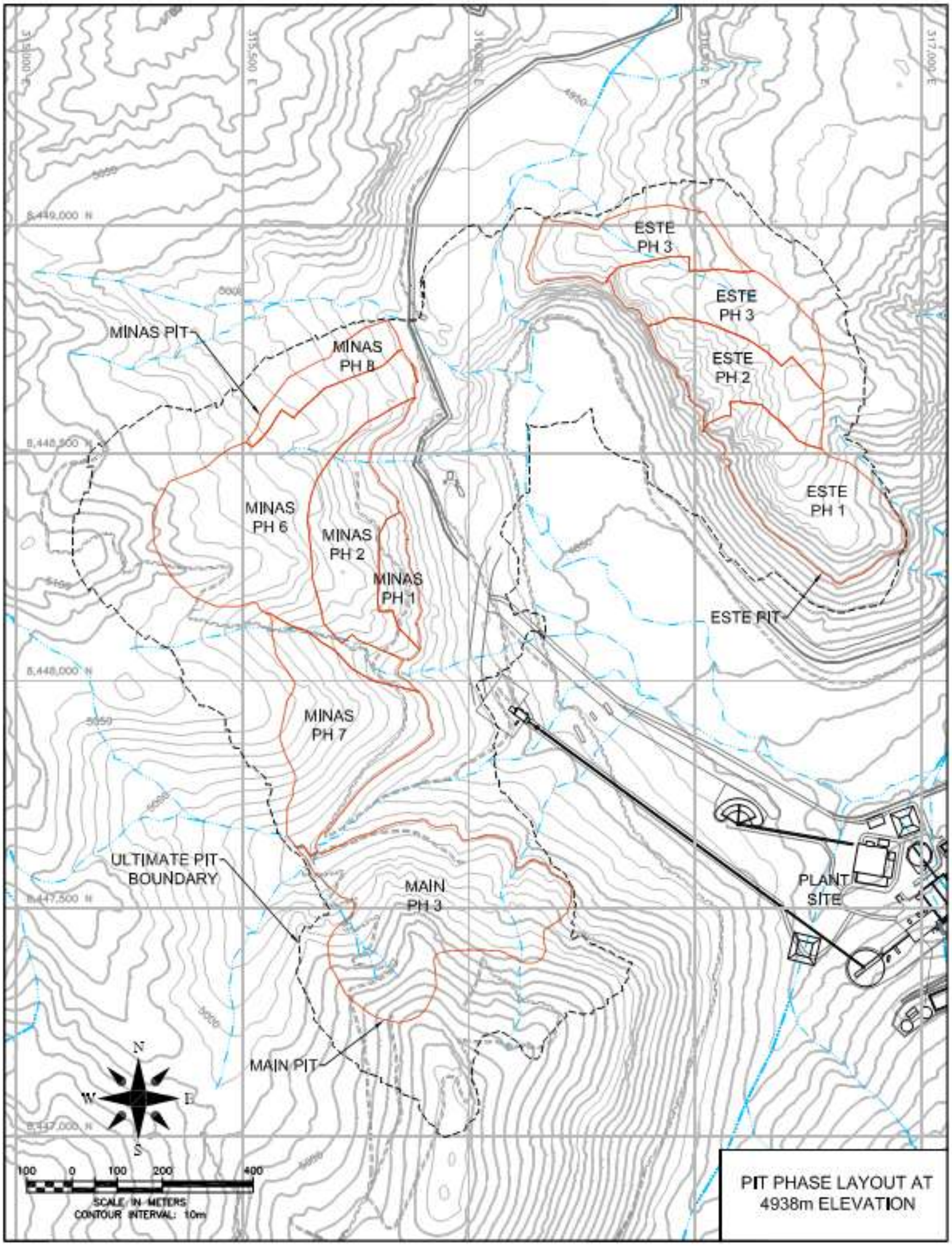


Figure 16-9: Pit Phase Map at Elevation 4938 m

16.5 WASTE ROCK AND TAILINGS MANAGEMENT

The Main Waste Rock Dump (Main Dump) is designed as a waste rock and filtered tailings co-disposal facility, and is located south of the Main pit. The facility has a total capacity of 318 million tonnes. In comparison to the previous plan, the current approach eliminates the wet tailings facility, reduces the number of independent waste dumps from two to one, simplifies surface water management, and reduces the project footprint and water requirements. Figure 16-10 shows the Main Dump configuration at the end of the mine life.

All waste rock and tailings produced from year -1 through year 6 is sent to the Main Dump. From year 7 to year 9, the majority of waste rock and tailings are hauled to the Este pit for backfill. After the Este backfill is completed in year 9, waste rock and tailings are sent to the Main Dump through to the end of the mine life. Bofedal soils in the area between the Este and Main pits are scheduled to be mined in years 4-6 for purposes of the current study. They will be hauled to the external dump and pit backfill areas for progressive reclamation where possible, and otherwise hauled to a soil stockpile located north of the Plant Water Pond. In practice the bofedal soils will be stripped over a period of several years ahead of the planned production to allow for trenching and drainage of the saturated soil material. Detailed scheduling of the bofedal soil material was considered beyond the scope of this study.

Post-mineral tuff waste rock is characterized as Non Acid-Generating (NAG) material, while pre-mineral tuff waste can be either NAG or Potentially Acid-Generating (PAG) material. All tailings are considered PAG. Both NAG, PAG, and tailings are placed selectively within designated zones within the dump and pit backfill areas. The placement zones are designed to maximize the geochemical and geotechnical stability of the facilities.

Roughly 62% of the waste material sent to the Main Dump is NAG material. The bulk of this material is planned to be dumped on the foundation and around the perimeter of the dump to provide a NAG buffer zone around the PAG waste. A 10-m-thick NAG layer will be placed along the foundation, and a 20-m-thick (vertical) layer of NAG material will be placed along the dump side slopes concurrently with mine operations. At closure, a minimum 1-m-thick NAG cap will be placed on the final lift, followed by an engineered soil cover. The external face of the dump and pit backfill areas will be reclaimed progressively throughout operations.

Mining operations continue in the Minas and Main pits until the end of the mine life; the final Minas and Main pits are backfilled with waste backhauled from the Main Dump to prevent the formation of pit lakes. Minas and Main pits are backfilled to approximately 4855 m elevation to achieve positive surface water drainage. Finally, all backfill not already reclaimed will be capped with a soil cover.

The waste material movement is shown in the mine schedule in Table 16-5. A more detailed discussion of mine waste management facility design is provided in Section 18.

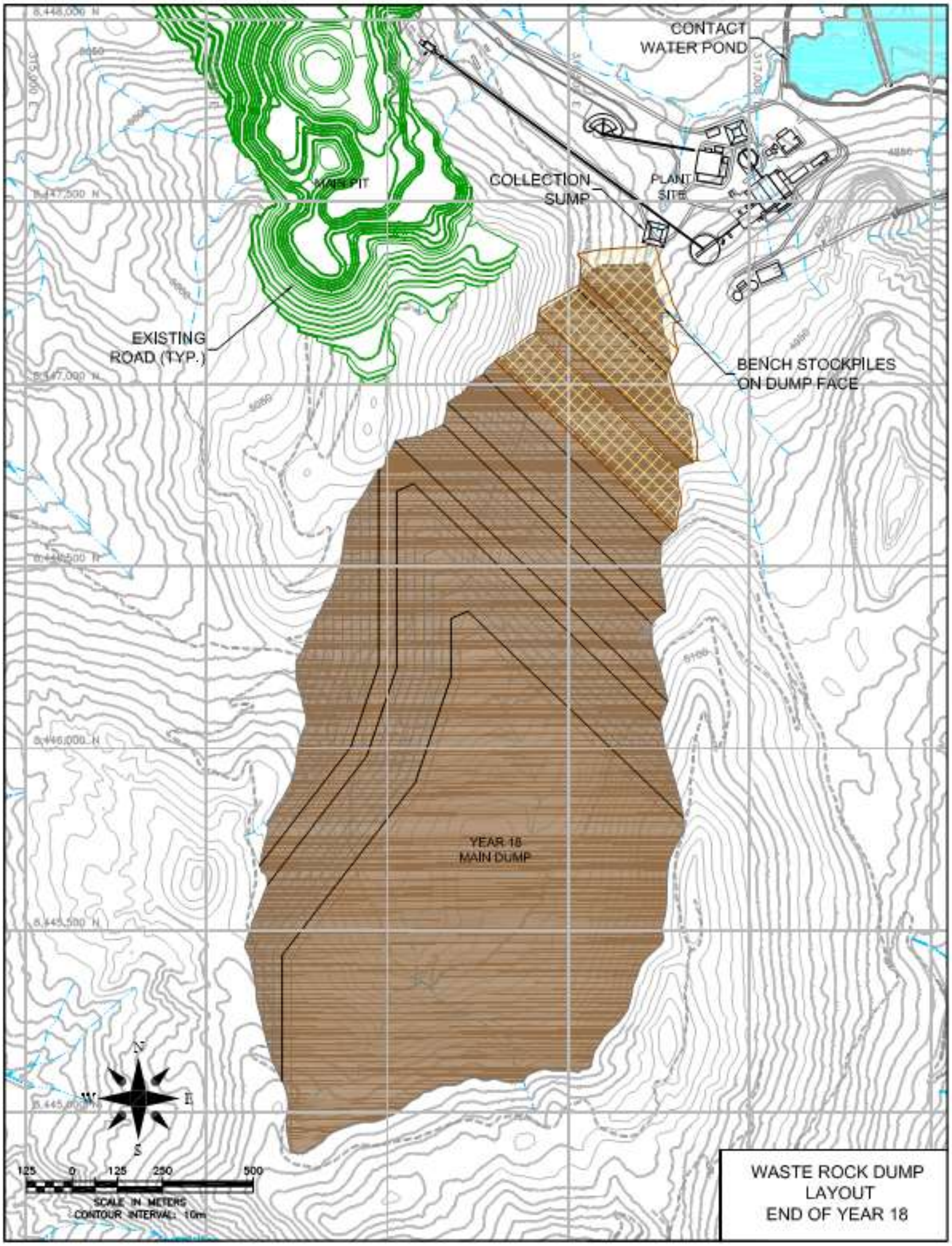


Figure 16-10: Waste Dump Location

16.5.1 Pre-Stripping and Pioneering

Before pre-stripping can begin in a phase, access roads will be pioneered into the starter pit areas. Overburden must be pre-stripped from within the pit before production mining begins and ahead of some internal phases to make ore available for blending purposes. The initial stripped materials will be sent to the Main Dump for disposal. Periodic pre-stripping continues through the first half of the mine life. Annual pre-stripping is described in detail in 16.6.4.

16.5.2 Soil Stockpiling

Soils will be removed from the mine pit and waste rock dump footprint areas for use in reclamation. The mine schedule considers the staged removal of soils as the footprints of the facilities expand. Excavated soil borrow materials will be used for concurrent reclamation as needed, with the excess soil materials stockpiled until they can be utilized.

Soil underneath the Main Dump foundation must be stripped and stockpiled prior to placement of pre-production waste rock. The same soil presently lining the valley floor will be used to cap the facility during reclamation and closure. Pre-production stripping requires that the soil be excavated and stockpiled in year -2 to clear the foundation for waste placement in year -1. Soil removed to accommodate pre-production stripping will be stockpiled north of the Plant Water Pond as shown in Figure 16-11. Additional material will be stockpiled at the toe of the Main Dump as needed (Figure 16-5). As the dump footprint grows, soil stripping and stockpiling activities will advance ahead of waste placement to minimize material handling in each time period. These activities will be prioritized during the dry season to facilitate handling and to minimize sediment load in runoff.

Excavation of bofedal soils within the mine pit area is scheduled within the mine plan as a pre-stripping activity, as described in the prior section.

16.5.3 Concurrent Reclamation

As the Main Dump rises in elevation, lower benches that have reached final limits will be re-graded and covered with the closure soil cover. The Main Dump will be covered with 1 m of cover material. Este pit backfill will also be reclaimed concurrently during operations and covered with 1.5 m of cover material. Minas and Main pits will be backfilled and covered at the end of mine life (closure phase).

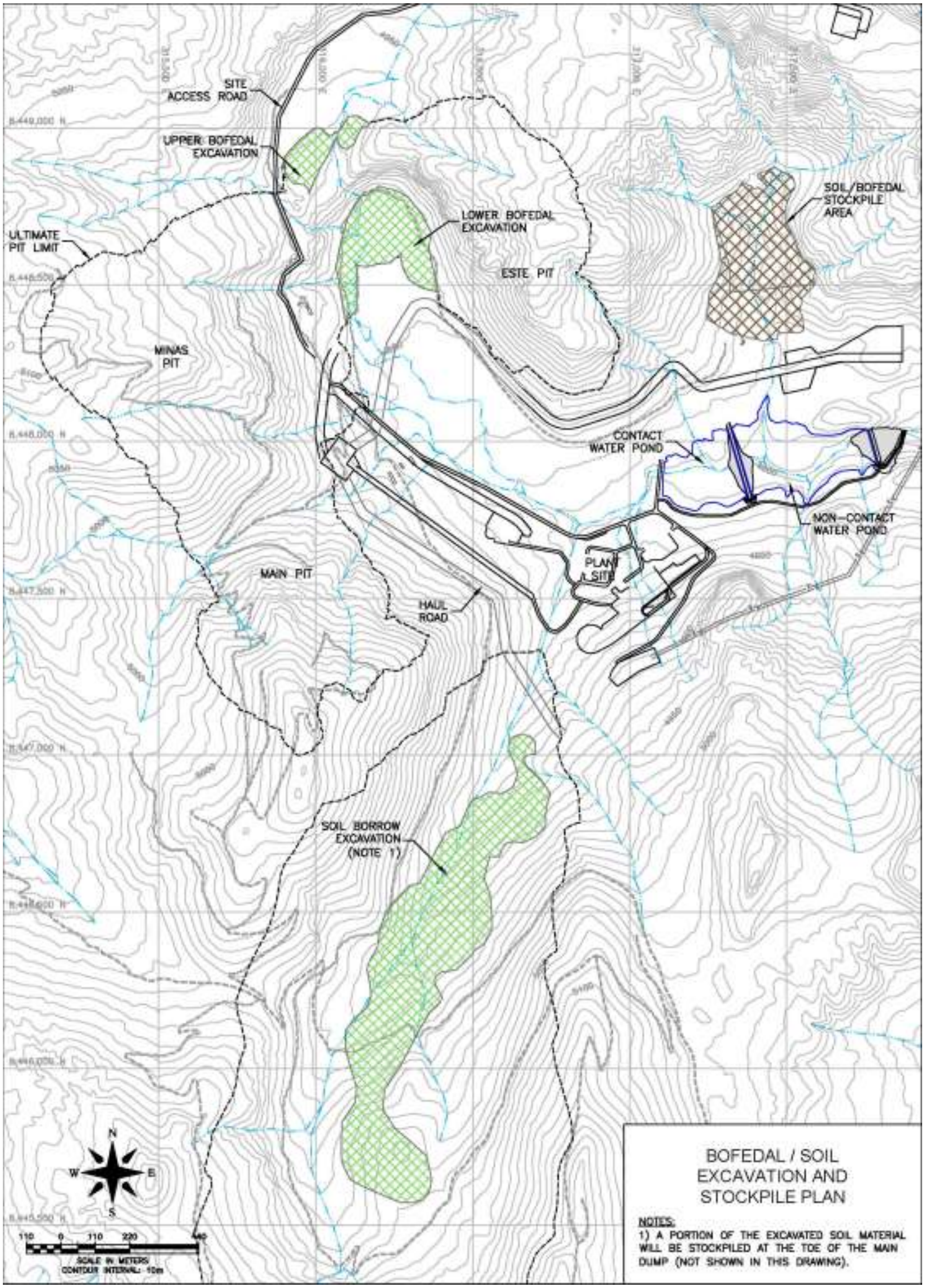


Figure 16-11: Borrow Area & Soil Stockpile

16.6 PRODUCTION SCHEDULE

The mine schedule was developed from the phase designs and the block model. The material contained within each pushback design was reported for multiple cutoff values so that the cutoff could be modified in the course of the mine scheduling process. Only measured and indicated category mineralization were included in the schedule; inferred resources were treated as waste.

16.6.1 Metal Recovery

Metal recovery for Corani has been difficult to predict on the basis of mineralization type and grade. The degree of oxidation is believed to underpin the observed variations in recovery, and oxidation is a process which by nature exhibits spatial variation. The 2011 FS approach applied average recoveries to broad mineral domains, with the value of each block in the model based on the average recovery assigned to each domain. However, the average for some mineral domains was applied to material that exhibited a wide range of recovery values during the metallurgical test program, in some cases ranging from quite low to quite high. The previous method would not allow low-recovery or high-recovery blocks to be identified on a block-by-block basis, which resulted in reduced NSR cutoff value selectivity.

The new recovery model applied in this study significantly increases resolution of NSR block values because it captures the spatial variability of recovery. This allows blocks which might have moderate to high grades, but low recovery due to oxidation, to be treated as waste if they fall below the NSR block-value cutoff. The new approach allows below cutoff material to be identified with a greater degree of confidence, and increases the ability to selectively schedule and apply variable cutoff values to different phases of the project. Recovered metal was calculated for each block in the block model using estimated grade, block elevation, and estimated values for geologic logged data. The recovery formulas are described in Section 13.7 of this report.

16.6.2 Mill Throughput

A tradeoff study was completed between the Pre-Feasibility Study (PFS) (2009) and the Feasibility Studies (FS) (2011) so that the optimum ore production rate would be used for the FS. Using the pre-feasibility study phases, alternative schedules were generated at 15,000; 22,500; and 30,000 tonnes per day. An ore rate of 22,500 tonnes per day was chosen by Bear Creek personnel as a balance between moving profits forward and increasing capital costs. The selected throughput was the maximum quantity of ore that could be processed in a single train comminution circuit and resulted in the lowest capital and operating cost per ton processed. The mine schedule was developed to produce 22,500 tonnes per mill operating day, or 7,875 ktonnes of ore per year to the crusher and flotation concentrator. Based on the base case mill throughput, the operating mine life is estimated to be 18 years.

In 2013, BCM and M3 undertook a capital cost throughput rationalization to see if there was a way to reduce the Corani Capex by reducing the throughput to 10,000 tonnes per day or 15,000 tonnes per day. The outcome of the study was that the capital cost decreases at a lower rate than the revenue from decreased ore production.

At 15,000 tonnes per day, the throughput drops 33%, the after tax NPV@5% drops 50%. At 10,000 tonnes per day, the throughput drops 56%, the after tax NPV@5% drops 75%. The conclusion of the evaluation is that the mine would be most profitable using the highest throughput possible through a single line of grinding equipment, using the largest equipment available. By adding a second grinding line, there is a step function in capital cost that reduces the IRR and NPV.

16.6.3 Cutoff Value

Cutoff values were based on NSR values at the base case prices. G&A costs were estimated to be $1.51/tonne, and processing costs were estimated to be $9.49. While the costs used to dig the Whittle pits are slightly different than the final numbers generated in this FS, a sensitivity analysis of operating costs during the Whittle optimization showed that the pit size is not sensitive to changes in operating cost. A 10% change in process operating cost had a 3.2% change in the Whittle pit ore tonnage. The breakeven cutoff value on this basis would be $11/tonne (NSR [Net Smelter Return] – Process Cost - G&A = 0). NSR is calculated on a block-by-block basis in the block model for determination of block cutoff value. The mine plan uses various NSR cutoffs from $23/tonne to $11/tonne to increase cash flow, improve payback, and maximize project economics. Table 16-3 displays cutoff value by pit phase.

Table 16-3: Cutoff Value by Pit Phase

Phase	NSR Cutoff RF100
este_ph1	23
minas_ph1	23
este_ph2	23
minas_rf1_ph2	19
este_ph3	19
este_ph4	19
este_ph5	19
este_ph6	15
minas_rf1_ph3	15
minas_rf1_ph4	15
minas_rf1_ph5	15

Phase	NSR Cutoff RF100
minas_rf1_ph6	11
minas_rf1_ph7	11
minas_rf1_ph8	11
minas_rf1_ph9	11
minas_rf1_ph10	11
main_rf1_ph1	11
main_rf1_ph2	11
main_rf1_ph3	11
main_rf1_ph4	11
main_rf1_ph5	11

The 2011 FS considered stockpiling of low grade ore material. Stockpiling of low-grade material was not considered in the current study.

Table 16-4 shows life of mine results for the designed pit. Although below cutoff material is treated as waste for this study, incremental economic evaluation of stockpiling scenarios may be considered during detailed design.

Table 16-4: Variable NSR Cutoff Designed Pit

Parameter	Designed Variable NSR Pit
Process Life	18
Mill Feed Tonnage	137.7
Average Silver Head Grade (g/t)	51.6
Contained Ag (Mozs)	228.4
Contained Pb (Mt)	1.26
Contained Zn (Mt)	0.81
Waste (t)	231.5
Strip Ratio (w:o)	1.68

16.6.4 Ore and Waste Production Schedule

Pre-stripping of waste from the post-mineral tuff cap has been scheduled ahead of mining of various phases to allow adequate uncovered ore inventory and to smooth the waste production profile. Waste from benches that contain ore are mined concurrent with the ore production from that bench. Many iterations of the phase design and production schedule were completed to establish a sound overall approach to the mine operating strategy and to maximize project economics.

Preproduction mining begins 9 months prior to production. First-year ore production ramps up from zero to design capacity over 9 months. Production was scheduled on a monthly basis to ensure that head grade met flotation process needs maintaining a zinc grade exceeding 0.20%, and is summarized quarterly for the first three years and annually thereafter. Table 16-5 summarizes the Corani production schedule; additionally Table 16-6 shows pit phase ore production by year.

Figure 16-12 shows the mining sequence by pit area phase over the mine life. Many of the phases overlap to optimize the mining rate and to meet the minimum zinc grade of 0.20%. This type of overlap is possible with the mobile truck and loader fleet specified in Section 21. In general, the zinc grade within the upper benches of each area is below the 0.20% Zn requirement and increases to over 1.0% Zn at depth. Therefore, the ore mining fleet was scheduled to mine the upper benches of one area with the lower benches of another to meet the blending requirement. The ore production from each area is envisioned to occur on separate dedicated campaigns in each area that last a fixed duration of multiple shift or days. The blending of the mill feed is performed by a dedicated loader at the plant run-of-mine stockpile area. The detailed blending of the mill feed on a shift-by-shift basis was considered beyond the scope of this study. Similarly, pre-stripping areas were spread over multiple phases so that the maximum mining rate rarely exceeded 3 fleets (or a maximum stripping ratio of 3H:1V during full production years). This strategy maximizes the utilization of the mobile mining fleets and smooths out the strip ratio during the mine life. Bofedal material is scheduled as pre-strip when it exists above all the ore benches within a phase or as in-bench waste when it exists on an ore bench. Bofedal removal is accomplished with a dedicated fleet consisting of a small excavator and articulated trucks. Figure 16-13 shows the breakout of ore, waste, and pre-strip material on a monthly basis over the mine life. Figure 16-14 provides a further breakout of the waste based on acid generation potential. Figure 16-15 shows the blended head grade to the mill for silver, lead, and zinc on a monthly basis over the mine life.

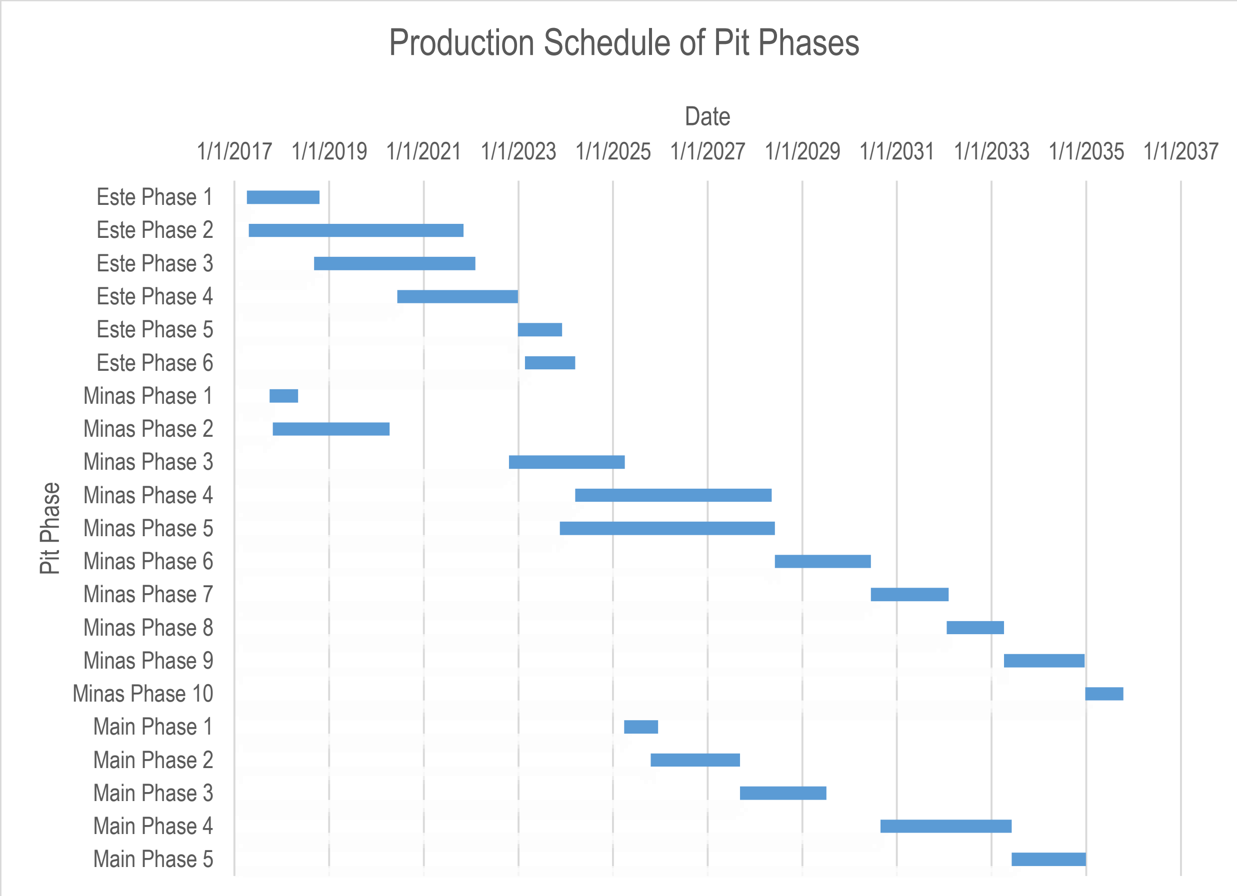


Figure 16-12: Pit Phase Schedule through Life of Mine

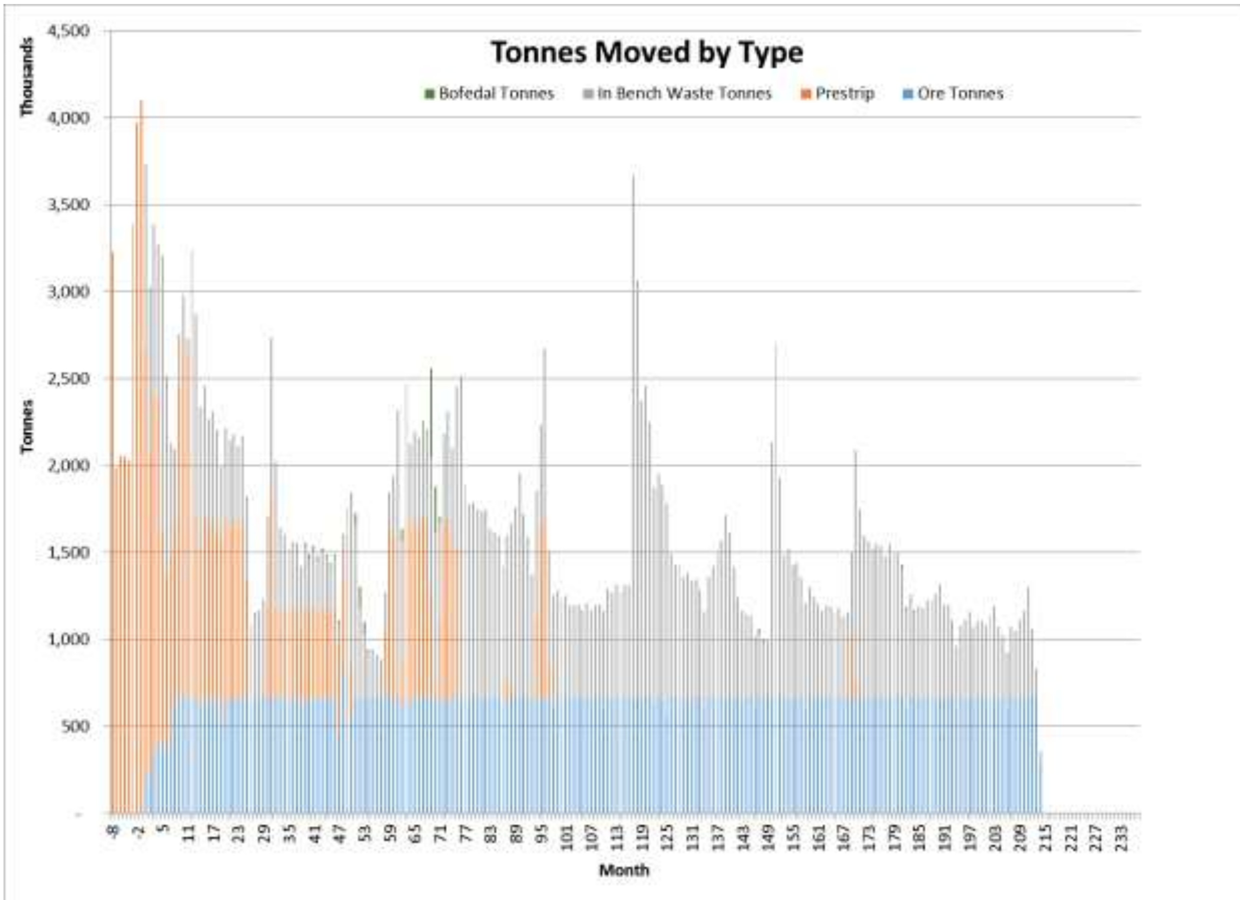


Figure 16-13: Monthly Tonnes Moved by Type

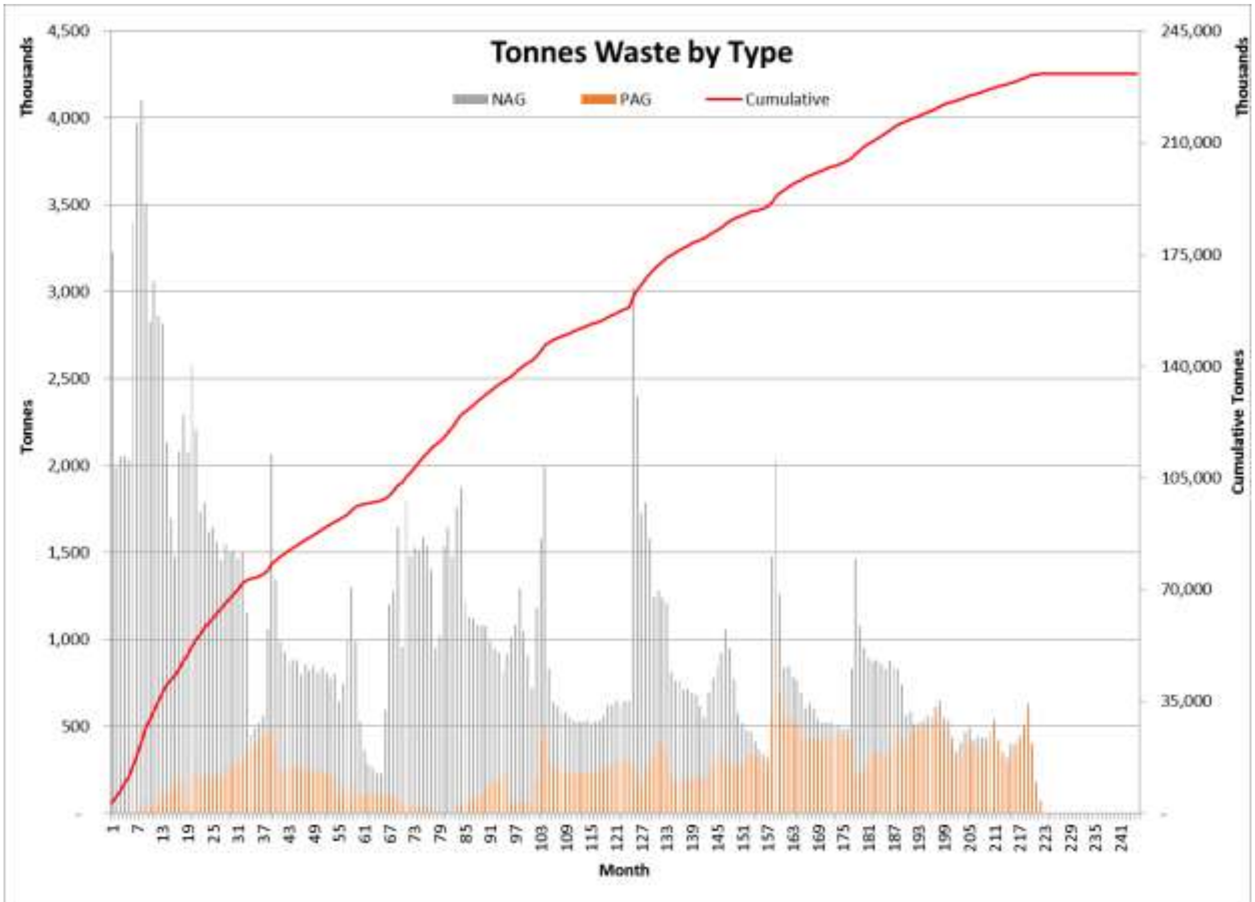


Figure 16-14: Monthly Breakout of Waste by Acid Generation Potential

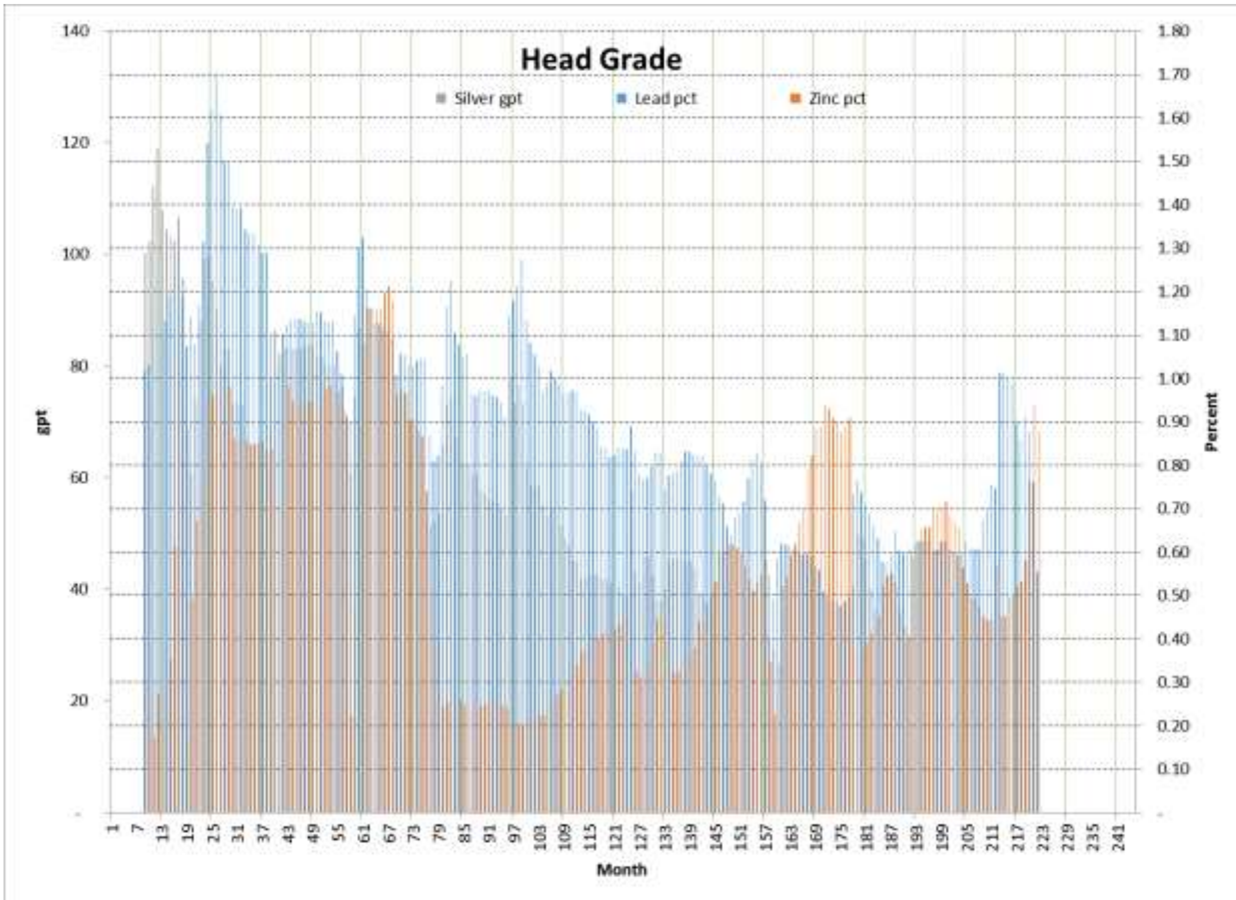


Figure 16-15: Monthly Head Grade by Metal Type

Table 16-5: Production Schedule

		Contained Resource in Thousands			Head Grade			Lead Con		Zinc Con		Recovered Resource in Thousands			All Waste Thousands of Tonnes		
Year	Ore kTonnes	Silver Ounces	Lead lbs	Zinc lbs	Silver gpt	Lead pct	Zinc pct	Silver gpt	Lead pct	Silver gpt	Zinc pct	Rec Ag oz	Rec Pb lb	Rec Zn lb	NAG	PAG	PAG+NAG Cumulative
yr-1 Q1	-	-	-	-	-	0.00%	0.00%	-	-	-	-	-	-	-	-	-	-
yr-1 Q2	-	-	-	-	-	0.00%	0.00%	-	-	-	-	-	-	-	5,218	-	5,218
yr-1 Q3	-	-	-	-	-	0.00%	0.00%	-	-	-	-	-	-	-	6,128	8	11,353
yr-1 Q4	-	-	-	-	-	0.00%	0.00%	-	-	-	-	-	-	-	11,401	68	22,822
yr1 Q1	752	2,563	17,672	3,754	106.0	1.07%	0.23%	8,243	56.60	385	52.90	1,677	7,819	1,476	9,271	127	32,220
yr1 Q2	1,182	4,199	30,117	6,491	110.5	1.16%	0.25%	7,268	56.60	385	52.90	2,845	15,063	2,179	7,497	316	40,033
yr1 Q3	1,736	5,823	48,326	27,057	104.3	1.26%	0.71%	5,045	56.60	385	52.90	4,309	31,796	16,393	4,751	489	45,273
yr1 Q4	2,005	4,855	50,179	22,879	75.3	1.14%	0.52%	5,000	56.60	385	52.90	3,265	24,293	12,589	6,494	449	52,216
yr2 Q1	1,942	5,434	50,875	31,396	87.0	1.19%	0.73%	4,781	56.60	385	52.90	3,884	30,145	15,855	5,047	675	57,938
yr2 Q2	1,963	6,005	70,123	41,578	95.1	1.62%	0.96%	3,414	56.60	385	52.90	4,488	47,670	27,671	4,193	627	62,757
yr2 Q3	1,854	4,893	62,579	38,979	82.1	1.53%	0.95%	2,927	56.60	385	52.90	3,741	45,894	26,220	3,793	704	67,255
yr2 Q4	1,985	4,671	60,922	37,708	73.2	1.39%	0.86%	2,871	56.60	385	52.90	3,558	44,530	24,710	3,615	857	71,727
yr3	7,897	18,603	208,415	147,565	73.3	1.20%	0.85%	3,390	56.60	385	52.90	14,030	148,240	101,162	6,992	4,304	83,023
yr4	7,856	20,473	193,597	164,783	81.1	1.12%	0.95%	3,940	56.60	385	52.90	15,795	142,426	125,132	6,971	2,638	92,632
yr5	7,890	20,270	196,328	173,035	79.9	1.13%	0.99%	3,789	56.60	385	52.90	15,742	146,944	130,908	6,884	1,360	100,876
yr6	7,847	14,988	169,875	125,375	59.4	0.98%	0.72%	3,572	56.60	385	52.90	10,911	108,284	88,629	16,557	388	117,821
yr7	7,921	16,006	181,376	43,970	62.9	1.04%	0.25%	5,431	56.60	385	52.90	9,774	68,817	13,177	14,353	1,033	133,208
yr8	7,875	15,956	186,602	38,254	63.0	1.07%	0.22%	7,108	56.60	385	52.90	9,007	48,488	11,538	11,393	2,076	146,676
yr9	7,875	11,978	166,843	55,841	47.3	0.96%	0.32%	3,910	56.60	385	52.90	7,682	74,220	18,984	4,042	2,915	153,632
yr10	7,875	10,846	144,581	67,380	42.8	0.83%	0.39%	3,369	56.60	385	52.90	7,377	81,371	29,181	10,657	3,165	167,455
yr11	7,826	10,896	138,265	64,415	43.3	0.80%	0.37%	3,765	56.60	385	52.90	7,404	73,107	29,035	8,527	3,151	179,132
yr12	7,867	8,131	127,363	95,375	32.1	0.73%	0.55%	2,442	56.60	385	52.90	6,186	90,617	45,372	5,349	3,385	187,866
yr13	7,875	7,215	115,841	85,249	28.5	0.67%	0.49%	2,283	56.60	385	52.90	5,586	87,798	39,566	4,002	5,890	197,758
yr14	7,875	6,888	93,338	147,709	27.2	0.54%	0.85%	2,390	56.60	321	52.90	5,501	74,693	101,486	1,416	5,186	204,360
yr15	7,897	9,162	113,354	85,117	36.1	0.65%	0.49%	3,134	56.60	361	52.90	6,860	78,988	48,045	7,179	4,024	215,563
yr16	7,875	7,152	106,332	105,815	28.2	0.61%	0.61%	2,315	56.60	385	52.90	5,662	83,973	61,350	1,059	5,853	222,475
yr17	7,858	6,635	111,394	93,330	26.3	0.64%	0.54%	2,108	56.60	385	52.90	5,256	87,239	48,418	597	4,723	227,795
yr18	6,172	4,731	123,523	81,085	23.8	0.91%	0.60%	1,294	56.60	365	52.90	3,731	96,015	52,402	200	3,526	231,521
Sum	**137,698**	**228,374**	**2,767,819**	**1,784,141**	**51.6**	**0.91%**	**0.59%**					**164,273**	**1,738,432**	**1,071,476**	**173,586**	**57,936**	

Table 16-6: Pit Phase - Ore Production by Year

		Year																		
Pit Phase	**Ore Tonnes (000's)**	**-1**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**
este_ph1	3,250	-	3,250	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
minas_ph1	541	-	541	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
este_ph2	7,813	-	384	1,615	3,159	2,655	-	-	-	-	-	-	-	-	-	-	-	-	-	-
minas_rf1_ph2	10,156	-	1,500	6,129	2,527	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
este_ph3	8,127	-	-	-	2,211	5,200	716	-	-	-	-	-	-	-	-	-	-	-	-	-
este_ph4	7,104	-	-	-	-	-	7,104	-	-	-	-	-	-	-	-	-	-	-	-	-
este_ph5	5,141	-	-	-	-	-	56	5,085	-	-	-	-	-	-	-	-	-	-	-	-
este_ph6	2,122	-	-	-	-	-	-	1,168	955	-	-	-	-	-	-	-	-	-	-	-
minas_rf1_ph3	7,227	-	-	-	-	-	14	1,594	4,444	1,175	-	-	-	-	-	-	-	-	-	-
minas_rf1_ph4	6,542	-	-	-	-	-	-	-	1,261	1,575	1,575	1,575	555	-	-	-	-	-	-	-
minas_rf1_ph5	6,653	-	-	-	-	-	-	-	1,261	1,575	1,575	1,575	667	-	-	-	-	-	-	-
minas_rf1_ph6	12,543	-	-	-	-	-	-	-	-	-	-	-	2,738	6,259	3,546	-	-	-	-	-
minas_rf1_ph7	8,383	-	-	-	-	-	-	-	-	-	-	-	-	-	3,226	4,725	432	-	-	-
minas_rf1_ph8	5,548	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	4,306	1,242	-	-
minas_rf1_ph9	8,064	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	3,483	4,581	-
minas_rf1_ph10	6,304	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	132	6,172
main_rf1_ph1	3,337	-	-	-	-	-	-	-	-	3,337	-	-	-	-	-	-	-	-	-	-
main_rf1_ph2	8,189	-	-	-	-	-	-	-	-	213	4,725	3,251	-	-	-	-	-	-	-	-
main_rf1_ph3	6,948	-	-	-	-	-	-	-	-	-	-	1,474	3,866	1,608	-	-	-	-	-	-
main_rf1_ph4	8,754	-	-	-	-	-	-	-	-	-	-	-	-	-	1,103	3,150	3,159	1,343	-	-
main_rf1_ph5	4,952	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	1,807	3,145	-
Totals	**137,698**	**-**	**5,675**	**7,744**	**7,897**	**7,856**	**7,890**	**7,847**	**7,921**	**7,875**	**7,875**	**7,875**	**7,826**	**7,867**	**7,875**	**7,875**	**7,897**	**7,875**	**7,858**	**6,172**

16.6.5 Annual Production Schedule

The following narrative of the annual mine operations and material movement and is intended to help the reader follow the evolution of the mine plan presented graphically in other sections. Yearly ore tonnes, waste tonnes, and head grades are reported in Table 16-5: Production Schedule. Haul truck productivity is summarized by year and pit phase in Table 16-6 (7).

16.6.5.1 Year -1

Pre-production begins the second quarter of year -1 (pre-production) and continues through the year during which a total of 22.8 million tonnes of waste is stripped in Este pit Phase 1 & 2 and Minas pit Phase 1 & 2 in preparation for the start of mining operations. During this period ore is not mined, if ore is encountered it will be stockpiled until the plant is ready to begin processing (year 1). Haulage access roads to the pits are opened on the upper east side of the Este pit (1515 m) for access to phase 1 and 2 and the upper south side of the Minas pit (4950 m) for access to phase 1. During pre-production the average waste haul is 130 meters downhill from the pit to the valley and 40 meters uphill from the valley to the waste dump.

16.6.5.2 Year 1

In year one (start of production) a total of 15.9 million tonnes of waste is stripped in Este pit Phase 1 & 2 and Minas pit Phase 1 & 2. Pre-stripping in Este phase 3 begins with a total of 3.8 million tonnes of waste material removed. Mining operations begin in Este pit phase 1 & 2 and Minas pit phase 1 & 2 with a total of 5.6 million tonnes of ore mined. The overall strip ratio averages 6.2. During year one the average Ag head grade to the plant is 99.0 gpt. Haulage access shifts to the east side of Este pit and haulage access for Minas moves from the pit rim to the bottom of the pit near the crusher, shortening the haul distance significantly. The average ore haul from the pit to the crusher for all phases is 90 meters downhill. The average waste haul from the pit to the valley is 90 meters downhill and from the valley to the dump is 100 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 130 meters uphill.

16.6.5.3 Year 2

In year two prestripping continues in Este phase 3 with a total of 12.1 million tonnes of waste material moved. Mining operations continue is Este pit phase 2 and Minas pit phase 2 with 7.7 million tonnes of ore mined. The overall strip ratio averages 3.5. During year two the average Ag head grade to the plant is 84.4 gpt Haulage access for both pits remains in the same locations as year one. The average ore haul from the pit to the crusher is 50 meters downhill. The average waste haul from the pit to the valley is 50 meters downhill and from the valley to the dump is 130 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 160 meters uphill.

16.6.5.4 Year 3

In year three prestripping finishes in Este phase 3 with the bulk of prestrip tonnage coming from Este phase 4 for a total of 4.9 million tonnes of waste removed. Mining operations continue in Este pit phase 2 and Minas pit phase 2 and begin in Este pit phase 3 with 7.9 million tonnes of ore mined. The overall strip ratio averages 1.4. During year three the average Ag head grade to the plant is 73.3 gpt. Haulage access in Este pit advances downwards towards the valley bottom. Access in Minas remains in the east side near the crusher. The average ore haul from the pit to the crusher is 30 meters downhill. The average waste haul from the pit to the valley is 30 meters downhill and from the valley to the dump is 150 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 180 meters uphill.

16.6.5.5 Year 4

In year four 6 million tonnes of waste is removed from Este pit phase 4. Mining operations focus of Este pit phase 2 and 3 with 7.9 million tonnes of ore mined. The overall strip ratio averages 1.22. During year four the average Ag head grade to the plant is 81.1 gpt. Haulage access in Este relocates to the south west pit ramp, and access in Minas remains the same. The average ore haul from the pit to the crusher is 0 meters. The average waste haul from the pit to the valley is 0 meters and from the valley to the dump is 170 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 200 meters uphill.

16.6.5.6 Year 5

In year five a small amount of prestrip material is removed from Este phase 4 with the bulk of prestripping waste removed from Minas phase 3 for a total of 3.4 million tonnes removed. Mining operations conclude in Este pit phase 3 and begin in Este pit phase 4, Este pit phase 5, and Minas pit phase 3 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 1.04. During year five the average Ag head grade to the plant is 79.9 gpt. Haulage access remains the same as the previous year. The average ore haul from the pit to the crusher is 50 meters downhill. The average waste haul from the pit to the valley is 50 meters downhill and from the valley to the dump is 180 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 210 meters uphill.

16.6.5.7 Year 6

In year six stripping begins and ends in Este phase 6 and begins in Minas pit phase 5 for a total of 8.5 million tonnes of waste removed. Mining operations continue in Este pit phase 5, Minas pit phase 3, and begin in Este pit phase 6 for a total of 7.8 million tonnes of ore mined. The overall strip ratio averages 2.16. During year six the average Ag head grade to the plant is 59.4 gpt. Haulage access remains the same for Minas, and new production in Este phase 6 creates an access point along the west edge of the phase near the Minas pit boundary. The average waste haul from the pit to the valley is 10 meters downhill and from the valley to the dump is 190 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 220 meters uphill.

16.6.5.8 Year 7

In year seven a very small amount of waste is stripped from Este phase 4 (92 tonnes) with the majority of waste coming from Minas pit phase 5 for a total of 2.8 million tonnes removed. Mining operations take place in Este pit phase 6, Minas pit phase 3, Minas pit phase 4, and Minas pit phase 5 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 1.94. The average Ag head grade to the plant is 62.9 gpt. Backfill of the Este pit begins in year 7 using waste rock and tailings. Este pit backfill continues through year 9. Haulage access remains the same in the Minas pit, and Este backfill access uses the same pit access points that were previously used for the final phases in Este. The average ore haul from the pit to the crusher is 60 meters downhill. The average waste haul from the pit to the Este backfill for all phases is 120 meters downhill. The average tailings haul from the tailings load out area to the Este backfill is 40 meters downhill.

16.6.5.9 Year 8

In year eight 2.6 million tonnes is stripped from Main pit phase 1 and 2. Mining operations take place in Minas pit phase 3, Minas pit phase 4, Minas pit phase 5, Main pit phase 1, and Main pit phase 2 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 1.71. During year eight the average Ag head grade to the plant is 63.0 gpt. Haulage access remains the same in the Minas pit, and the Main pit access starts on the north side of the pit bottom. The average ore haul from the pit to the crusher is 160 meters downhill. The average waste haul from the pit to the Este backfill for all phases is 170 meters downhill. The average tailings haul from the tailings load out area to the Este backfill is 20 meters uphill.

16.6.5.10 Year 9

In year nine 183,008 tonnes of waste is stripped from Main pit phase 2. Mining operations take place in Minas pit phase 4, Minas pit phase 5, and Main pit phase 2 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 0.88. During year nine the average Ag head grade to the plant is 47.3 gpt. Haulage access relocates to a new ultimate pit ramp in Minas near the crusher and access to Main pit advances down several benches on the north side of the pit. The average ore haul from the pit to the crusher is 80 meters downhill. The average waste haul from the pit to the valley is 80 meters downhill and from the valley to the east backfill is 40 meters uphill for all phases. The average tailings haul from the tailings load out area to the Este backfill is 70 meters uphill.

16.6.5.11 Year 10

In year ten mining operations take place in Minas pit phase 4, Minas pit phase 5, Main pit phase 2, and Main pit phase 3 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 1.76. During year ten the average Ag head grade to the plant is 42.8 gpt. Haulage access in the Minas pit will continue to use the ultimate pit ramp for the remainder of the mine life, and the access for the Main pit has advances down several levels on the northwest side of the pit. The average ore haul from the pit to the crusher is 70 meters downhill. The average waste haul from the pit to the valley is 70 meters downhill and from the valley to the dump is 210 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 240 meters uphill.

16.6.5.12 Year 11

In year eleven mining operations take place in Minas pit phase 4, Minas pit phase 5, Minas pit phase 6, Main pit phase 2, and Main pit phase 3 for a total of 7.8 million tonnes of ore mined. The overall strip ratio averages 1.49. During year eleven the average Ag head grade to the plant is 43.3 gpt. Haulage access in the Main pit has moved down several levels on the northwest side of the pit bottom. The average ore haul from the pit to the crusher is 30 meters downhill. The average waste haul from the pit to the valley is 30 meters downhill and from the valley to the dump is 220 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 250 meters uphill.

16.6.5.13 Year 12

In year twelve mining operations take place in Minas pit phase 6 and Main pit phase 3 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 1.11. During year twelve the average Ag head grade to the plant is 32.1 gpt. Haulage access remains the same as the previous year. The average ore haul from the pit to the crusher is 20 meters downhill. The average waste haul from the pit to the valley is 20 meters downhill and from the valley to the dump is 230 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 260 meters uphill.

16.6.5.14 Year 13

In year thirteen mining operations take place in Minas pit phase 6, Minas pit phase 7, and Main pit phase 4 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 1.26. During year thirteen the average Ag head grade to the plant is 28.5 gpt. Haulage access for the Main pit has moved further down the valley near the southeastern edge of the Minas pit near the crusher. The average ore haul from the pit to the crusher is 10 meters downhill. The average waste haul from the pit to the valley is 90 meters downhill and from the valley to the dump is 100 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 270 meters uphill. In year thirteen PAG waste exceeds NAG waste for the first time in the project lifespan.

16.6.5.15 Year 14

In year fourteen mining operations take place in Minas pit phase 7 and Main pit phase 4 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 0.84. During year fourteen the average Ag head grade to the plant is 27.2 gpt. Haulage access for the Main pit remains the same as the previous year. The average ore haul from the pit to the crusher is 0 meters. The average waste haul from the pit to the valley is 0 meters and from the valley to the dump is 250 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 280 meters uphill. In year fourteen PAG waste exceeds NAG waste 4 to 1.

16.6.5.16 Year 15

In year fifteen a total of 482,831 tonnes is stripped from Minas pit phase 8. Mining operations take place in Minas pit phase 7, Minas pit phase 8, and Main pit phase 4 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 1.42. During year fifteen the average Ag head grade to the plant is 36.1 gpt. The average ore haul from the pit to the crusher is 10 meters downhill. The average waste haul from the pit to the valley is 10 meters downhill and from the valley to the dump is 260 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 290 meters uphill.

16.6.5.17 Year 16

In year sixteen mining operations take place in Minas pit phase 8, Minas pit phase 9, Main pit phase 4, and Main pit phase 5 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 0.88. During year sixteen the average Ag head grade to the plant is 28.2 gpt. The average ore haul from the pit to the crusher is 20 meters uphill. The average waste haul from the pit to the valley is 20 meters uphill and from the valley to the dump is 270 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 300 meters uphill. In year sixteen PAG waste exceeds NAG waste 6 to 1.

16.6.5.18 Year 17

In year seventeen mining operations take place in Minas pit phase 9 and Main pit phase 5 for a total of 7.9 million tonnes of ore mined. The overall strip ratio averages 0.68. During year seventeen the average Ag head grade to the plant is 26.3 gpt. Haulage access for the Main pit has advanced down to the same location as the haulage access of the Minas pit and will remain there for the rest of the mine life. The average ore haul from the pit to the crusher is 70 meters uphill. The average waste haul from the pit to the valley is 70 meters uphill and from the valley to the dump is 280 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 310 meters uphill. In year seventeen PAG waste exceeds NAG waste 8 to 1.

16.6.5.19 Year 18

In year eighteen mining operations take place in Minas pit phase 10 for a total of 6.2 million tonnes of ore mined. The overall strip ratio averages 0.60. During year eighteen the average Ag head grade to the plant is 23.8 gpt. The average ore haul from the pit to the crusher is 120 meters uphill. The average waste haul from the pit to the valley is 120 meters uphill and from the valley to the dump is 290 meters uphill for all phases. The average tailings haul from the tailings load out area to the dump is 320 meters uphill. In year eighteen PAG waste exceeds NAG waste 18 to 1. At the end of year eighteen Minas and Main pit backfill closure and reclamation operations begin.

16.7 WATER MANAGEMENT AND TREATMENT

There are three potential sources of water in the pits: groundwater inflow to the pit, precipitation runoff generated in the pit, and surface runoff/stream flow originating from the catchment above the pit. Varying inflow rates over the life of the mine were predicted with the help of a MODFLOW model based on site-specific geologic and monitoring data. Groundwater inflow to the pit is anticipated to be relatively low since the pits primarily form side-hill cuts. For most of the mine life, inflow rates range from 600 to 1,000 m^3/day. The primary source of water requiring management in the pits is from precipitation runoff. Inflowing groundwater, and water from direct precipitation in the pits, will be collected in sumps. Sump locations will change as the pits develop, moving to the deepest portions of the pits as they develop. Water from the pits will be pumped to the process plant for consumption during operations. In an effort to minimize the volume of water entering the pits from the catchment upstream of the pits, runoff and stream flow will be diverted around the pits or to drop structures that will convey water across the pits to natural drainages. This diversion system will change over time as needed to accommodate the changing pit configuration, while continuously protecting the diverted water from contact with disturbed sulfide-bearing rock. Seepage that is produced from the Este backfill will be handled like other contact water, and pumped to the plant for consumption in the process.

16.8 MINING EQUIPMENT

Mine mobile equipment was selected to meet the production schedule as outlined in Table 16-5. All pieces of mine equipment within this study are standard off-the-shelf units.

The productivity of each piece of equipment was estimated, using first principles and data from the CAT Performance Handbook 45, including high-altitude deration adjustments. Although CAT specifications have been utilized as convenient reference standard, equivalent equipment from alternate suppliers may be utilized and would be selected during detailed design. Equipment productivity and material tonnage moved was then used to determine the total operating hours required to satisfy each unit operation. Truck hauling productivities were calculated in Vulcan. Loading, dozing, and other unit operation productivities were calculated from first-principles. For ore and waste rock a bank density of 2.35 tonnes/m^3 with a swell factor of 30% was assumed. For bofedal soils a wet bulk density of 1.49 tonnes/m^3 and a swell factor of 10% was used. A mechanical availability of 85% and use of availability of 95% was assigned to the fleet. Figure 16-16 summarizes the number of pieces of major mobile equipment required.

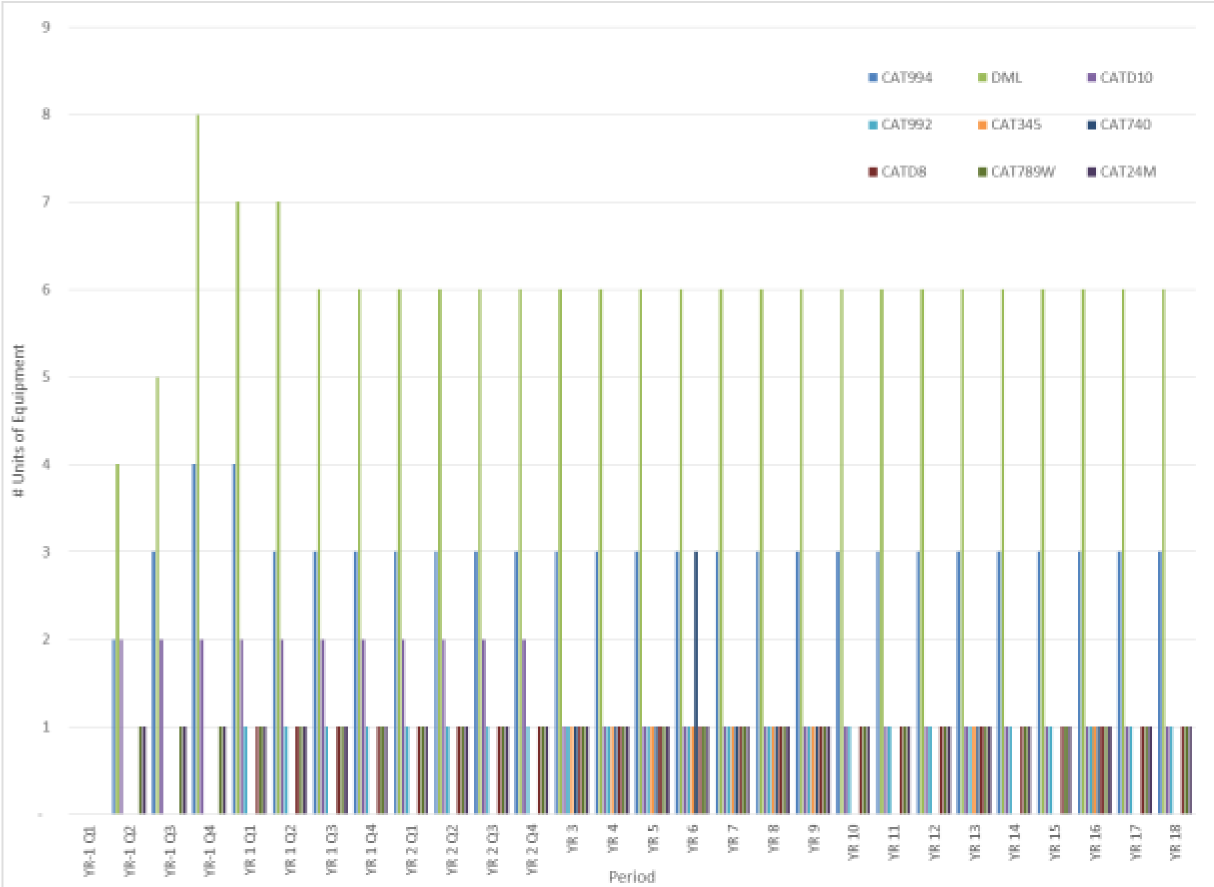


Figure 16-16: Number Units of Equipment

16.8.1 Drilling

Production drilling will be performed with Atlas Copco DML drills using 6 ¾" (171 mm) diameter holes for ore and waste in 8 m benches, using a square pattern with a 4.6 m nominal burden and spacing. Of the 8 required drills, 6 will be purchased by the mine and two drills will be leased under a short-term lease during the high pre-stripping requirement in year -1 and year 1. Drill holes will be sampled and assayed for ore control. Drills are scheduled to be overhauled every 15,000 hours and replaced every 30,000 hours.

16.8.2 Loading

Primary loading will be performed by CAT 994 high lift loaders with 18 m^3 bucket capacity. A maximum of 4 loaders will be required. Of the 4 required loaders, 3 will be lease to purchased by the mine and the fourth 994 loader will be leased under a short-term lease during the high pre-stripping requirement in year -1 and year 1. Wheel loaders were included in the loading fleet for the advantage of having more maneuverability between mining areas (up to 3 pit phases mined simultaneously) and a lower capital cost than shovels. Loading of filtered tailings will be performed by a CAT 992 high lift loader with a 12 m^3 bucket capacity. Loaders are scheduled to be overhauled every 15,000 hours and replaced every 30,000 hours.

16.8.3 Hauling

Rear-dump 181-tonne-payload CAT 789 haul trucks will be used for primary mine production. A maximum of 21 CAT 789 trucks are required. Of the 21 required trucks, the mine will lease-to-purchase only 16. The remaining 5 trucks that will be required during the high stripping requirement in year -1 and year 1 will be leased under a short-term lease. The 16 mine trucks provide excess haulage capacity during the later mine life phases, which allows the trucks to be used for spares during breakdowns and overhauls. Trucks are scheduled to be overhauled every 15,000 hours and replaced at every 60,000 hours. Table 16-7 summarizes the truck haul productivities for ore and waste by year and phase.

Table 16-7: Productivity per Haul Truck by Year and Pit Phase

Year	Pit	Phase	Average Ore Productivity (tph)	Average Waste Productivity (tph)
-1	Este	1	-	360.5
		2	-	365.1
	Minas	1	-	437.7
		2	-	375.6
1	Este	1	661.1	403.8
		2	744.0	437.2
		3	-	349.7
	Minas	1	423.5	355.6
		2	1026.1	539.8
2	Este	2	799.4	437.0
		3	683.0	383.3
	Minas	2	1256.9	611.8
3	Este	2	850.8	433.9
		3	860.8	423.9
		4	-	309.0
	Minas	2	964.1	483.3
4	Este	2	897.3	439.3
		3	828.1	422.9
		4	670.9	353.9
5	Este	3	760.9	403.9
		4	776.4	392.0
		5	709.4	370.6
	Minas	3	451.9	265.9
6	Este	5	623.2	339.5
		6	748.5	402.7
	Minas	3	512.9	303.0
		5	490.7	292.5
7	Este	6	632.9	343.8
	Minas	3	591.6	659.4
		4	1084.9	1349.4

Year	Pit	Phase	Average Ore Productivity (tph)	Average Waste Productivity (tph)
		5	547.2	586.2
8	Main	1	636.1	436.5
		2	550.0	366.0
	Minas	3	689.5	751.7
		4	1056.9	770.4
		5	589.3	540.2
9	Main	2	762.1	382.8
	Minas	4	967.0	495.9
		5	609.7	320.8
10	Main	2	1020.1	423.2
		3	787.7	360.3
	Minas	4	829.1	364.4
		5	630.2	323.4
11	Main	3	998.5	398.4
	Minas	4	762.7	343.0
		5	700.5	329.9
		6	760.4	337.7
12	Main	3	998.5	398.4
	Minas	6	931.2	356.1
13	Main	4	559.9	279.3
	Minas	6	950.2	360.3
		7	1637.5	406.5
14	Main	4	592.6	279.4
	Minas	7	1207.7	366.2
15	Main	4	640.3	282.0
	Minas	7	1207.7	366.2
		8	623.1	278.7
16	Main	4	666.3	286.8
		5	628.1	274.6
	Minas	8	712.7	294.4
		9	826.7	248.6
17	Main	5	691.5	284.1
	Minas	9	780.0	184.9
		10	691.6	180.3
18	Minas	10	595.8	173.6

16.8.4 Support Equipment

The major tasks to be completed by the support equipment include the following:

- Waste rock dump support
- Bench and road maintenance
- Reclamation support
- Stockpile construction
- General maintenance
- Ditch preparation and maintenance
- Loader support/cleanup

574-hp (D10 class) and 310-hp (D8 class) track dozers will be utilized for waste dump and filtered tailings construction and maintenance. Three CAT 740-class articulated trucks and one CAT 345-class excavator are provided for soil excavation and stockpiling. One CAT 24M class motor grader and one CAT 789 size water truck will run during the daytime shift only to maintain the haul roads. Magnesium chloride will be applied to all haul roads once per year.

Additional auxiliary equipment will serve and support the mine operating and maintenance groups, and is shown in Table 16-8 below.

Table 16-8: Support and Auxiliary Equipment

Support Equipment	% Utilization	# Units of Equipment
CAT 336 W/Rock Breaker	25%	1
CAT 950 Loader	25%	1
CAT 450 Backhoe	25%	1
CAT 236 Skid Steer	25%	1
Forklift	25%	1
Telehandler (TL642)	10%	1
Crane	10%	1
4x4 Pickup	10%	10
Employee Bus	100%	2
Fuel/Lube Truck	10%	1
Mechanic/Service Truck	10%	1
Explosives Truck	10%	1
Heavy Duty Pumps	10%	3
Sump Pump	10%	2
Portable Lights	100%	5

16.9 MANPOWER

Mine manpower is comprised of salary supervision and engineering staff, operations hourly labor, and maintenance hourly labor. Manpower requirements were estimated based on the equipment list and mine schedule. The mine operates two 12 hours shifts per day, utilizing 4 crews with a week on week off schedule. For most of the mine life there will be 31 salaried staff for supervision, engineering, geology, training, and ore control as follows:

- 1 mine manager to supervise all mining activity
- 2 mine superintendents (1 per operating day)
- 4 mine foremen (1 per crew)
- 2 maintenance superintendents (1 per operating day)
- 4 maintenance foremen (1 per crew)
- 4 chief mine engineers (1 per crew)
- 4 chief geologists (1 per crew)
- 4 geologists (1 per crew)
- 4 engineers (1 per crew)
- 2 surveyors (1 per operating day)

Operations manpower was estimated using required equipment operating hours and 2,100 working hours per person per year for all mining equipment. Mine operating labor increases to 165 persons during pre-production, and declines until another spike in operating labor in year 6. Between year 7 and 18, operating labor varies between 99 and 124 persons. Maintenance manpower includes mechanics, electricians, welders, and laborers (helpers). The maintenance staff was sized based on the number of pieces of heavy equipment. Mine maintenance labor increases to 112 persons during pre-production, and stays between 94 and 101 persons until the end of mine life in year 18. Operation and maintenance manpower requirements are displayed in Table 16-9. Figure 16-17 and Figure 16-18 below illustrate the operations and maintenance labor requirement over the mine life.

Table 16-9: Operations & Maintenance – Manpower

Manpower	yr-1 Q1	yr-1 Q2	yr-1 Q3	yr-1 Q4	yr1 Q1	yr1 Q2	yr1 Q3	yr1 Q4	yr2 Q1	yr2 Q2	yr2 Q3	yr2 Q4	yr3	yr4
Operator 1	0	22	25	43	44	40	33	41	37	34	32	33	29	27
Operator 2	4	45	52	88	82	66	52	68	59	53	54	55	52	45
Operator 3	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Driver	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Mechanic	5	35	41	60	62	53	51	51	51	51	51	51	53	53
Electrician	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Welder	2	9	11	15	16	14	13	13	13	13	13	13	14	14
Laborer	2	17	20	30	31	26	26	26	26	26	26	26	26	26
Sum	51	166	187	274	273	237	213	237	224	215	214	216	212	203

Manpower	yr5	yr6	yr7	yr8	yr9	yr10	yr11	yr12	yr13	yr14	yr15	yr16	yr17	yr18
Operator 1	28	37	31	31	26	30	28	25	28	24	27	26	22	18
Operator 2	56	69	57	54	46	53	49	41	45	43	63	50	46	47
Operator 3	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Driver	30	30	30	30	30	30	30	30	30	30	30	30	30	30
Mechanic	53	56	53	53	53	50	50	50	53	50	50	53	50	50
Electrician	4	4	4	4	4	4	4	4	4	4	5	6	7	8
Welder	14	14	14	14	14	13	13	13	14	13	13	14	13	13
Laborer	26	28	26	26	26	25	25	25	26	25	25	26	25	25
Sum	215	242	219	216	203	209	203	192	204	193	217	209	197	195

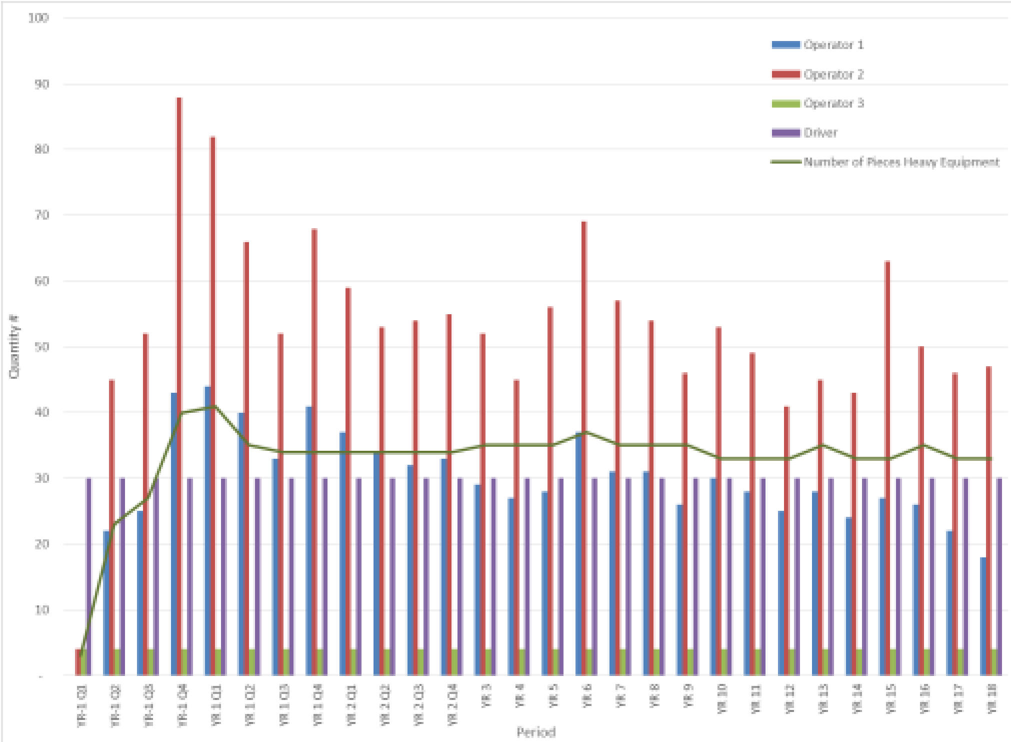


Figure 16-17: Operations Manpower Summary

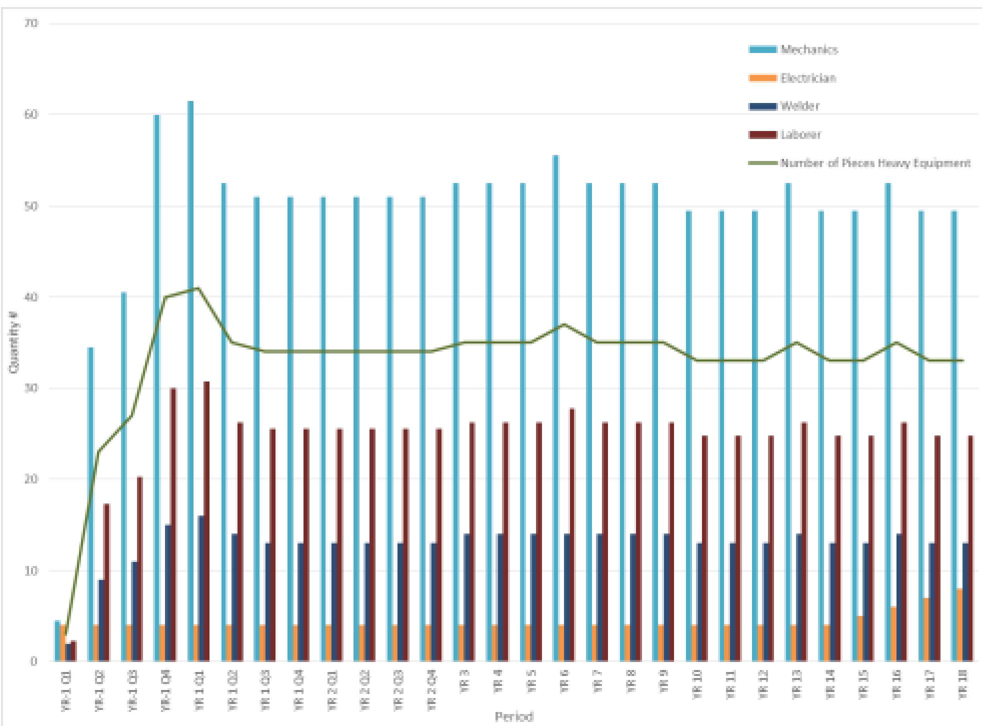


Figure 16-18: Maintenance Manpower Summary

16.10 BLASTING AND EXPLOSIVES

Bulk explosives and explosive accessories were determined from the 6 ¾ " drill hole diameter, 8 meter bench height, powder factor of 0.59 kg/m^3 (1 lb/cy), and tonnage of rock mined. A 10% allowance for emulsion was added to the bulk explosive cost estimate to account for wet holes (assumes 10% of the holes are wet and are loaded with emulsion). A six-row (six holes per row) blasting pattern was assumed to estimate the explosive accessories. Table 16-10 summarizes the results of the explosives calculations and the consumable quantities required for blasting on a per tonne basis.

Table 16-10: Explosives Calculations

Description	Value	Units
Powder Factor	0.59	kg/m^3 (1lb/cy)
Stemming Length	3.50	m
Sub-drilling	1.03	m
Bench Height	8.00	m
Powder Column	5.53	m
Loading Factor	18.44	kg/m
Weight of Explosive per Borehole	101.9	kg
Burden	4.6	m
Spacing	4.6	m
ANFO per tonne blasted	0.2514	kg/tonne
Caps per tonne blasted	0.0029	Caps/tonne
Boosters (1 per hole)	0.0025	Booster/tonne
Snap Lines	0.0025	Snap Line/tonne

16.11 WORK SCHEDULE

The work schedule assumes mine production will operate 24 hr/d, 7 d/wk, 350 d/yr. Operations and mining personnel will work on two 12 hr/d shifts. All hourly and salaried personnel will work a 1-week-on/1-week-off rotation. Each employee works 2,100 hours per year for a total of 25 work weeks and 2 additional weeks for vacation and training.

17 RECOVERY METHODS

17.1 SITE LAYOUT CONSIDERATIONS

The Project site is steep, high-altitude terrain that has limited flat space. Due to these considerations particular attention is required to develop acceptable sites for the facilities.

The development of the site layout was based on maximizing the ease of the operation and minimizing both the capital and operating costs.

17.2 PROCESS DESCRIPTION

A processing plant with a capacity of 22,500 MTPD has been selected as the optimum size.

The Corani process plant will be a conventional lead-zinc flotation plant that will produce separate lead and zinc concentrates. A number of metallurgical studies have been conducted on the project and formed the basis of the design (Blue Coast, 2011; DJB Consultants, 2011; SGS, 2007; 2008a; 2008b; 2009a; 2009b; and 2010).

Figure 17-1 is a simplified flow sheet for the overall process, which is described as follows:

Size reduction of the ore by a primary jaw crusher to reduce the screened run-of-mine (ROM) material, which is 100% passing 800 mm (F_{100}), to a product size of 80% passing (P_{80}) 150 mm.

Stockpiling the primary crushed ore and then reclaiming by feeders and conveying to the grinding circuit.

The grinding circuit consists of a semi-autogenous (SAG) mill and ball mill operated in closed circuit with hydrocyclones to produce a product with a P_{80} of 106 microns prior to processing in a flotation circuit.

The flotation plant will consist of selective lead-zinc flotation. The flotation circuits will each consist of rougher flotation and three stages of cleaning, to produce a high value lead concentrate and a lower value zinc concentrate with payable silver values. Both lead and zinc circuits will include a regrind mill to grind the rougher concentrate to a P_{80} of 25 microns.

Final lead and zinc concentrate will be thickened, filtered and loaded in super sacks for shipment.

The flotation tailings will be thickened, filtered and conveyed to the dry tailing stack area. The tailings will be stacked using conveyors and a radial stacker to a stockpile which will then be reclaimed by a front end loader and hauled away to be spread and compacted as required.

Filtrate from the tailing filters will be reused in the process plant water system.

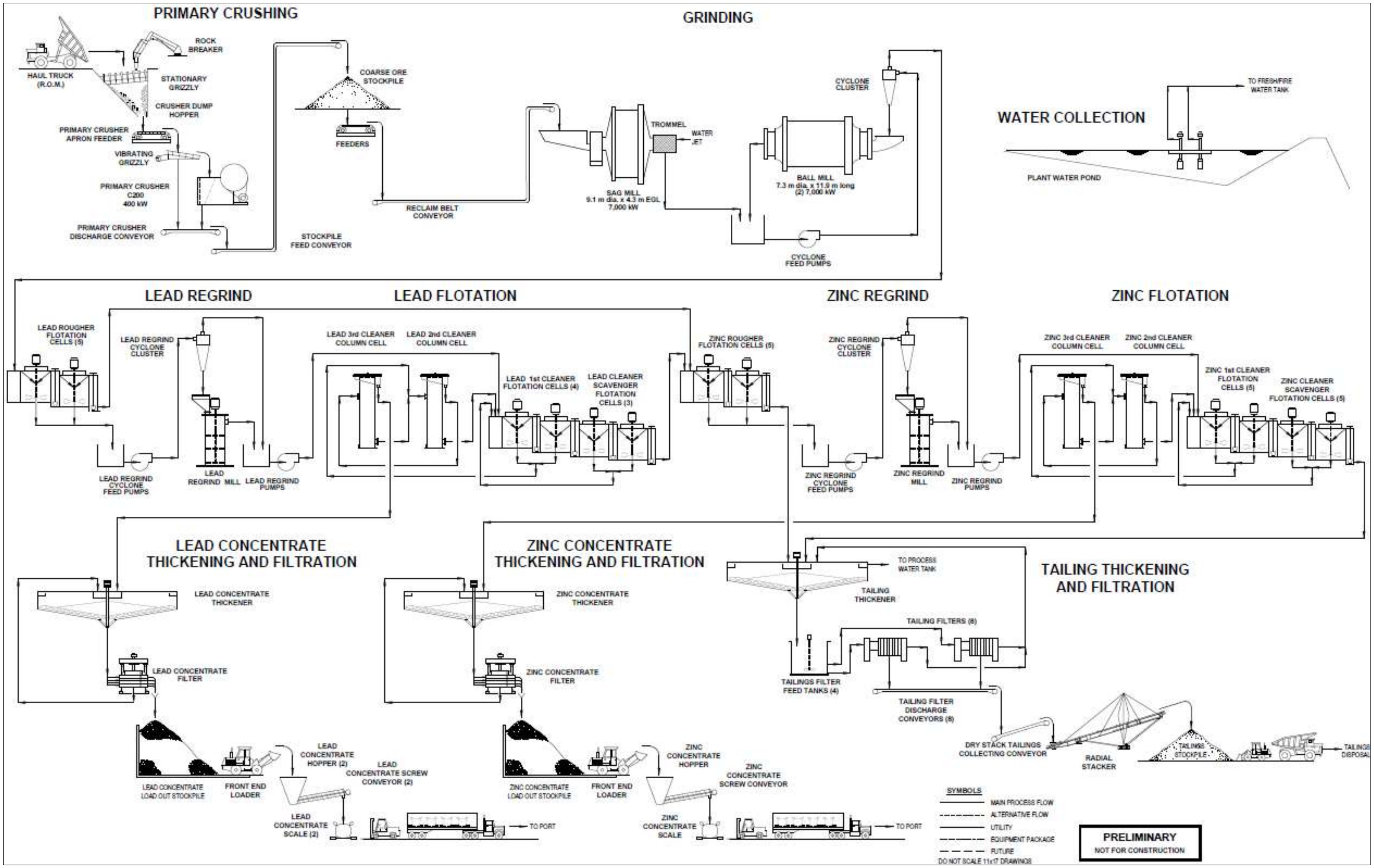


Figure 17-1: Simplified Process Flow Diagram for the Corani Project

17.2.1 Process Design Criteria

BCM tasked M3 to design a process plant for the Corani project with a nameplate capacity of 22,500 mtpd. For the design, M3 used an availability factor (run time) of 92%, except for the crushing area, which was designed using an availability of 75%. These design availabilities are typical for recent projects of this type.

The process plant has been designed for 8,218,000 metric tons in a 365-day operating year for an average of 22,500 mtpd. At 92% availability, the processing design rate is 24,457 mtpd or 1,019 mtph to reach the target average of 22,500 mtpd.

Table 17-1 is a summary of the main components of the process design criteria used for the study. More detailed design criteria have been produced for internal use.

Table 17-1: Process Design Criteria

DESCRIPTION	DESIGN
Primary Crushing	
Feed Rate, mtph	1,250
Availability, %	75
Feed F_{100}, mm	800
Product P_{80}, mm	150
Crushing work index (LEIT), kWh/t, 80th percentile	8.06
, average	6.30
, design	6.77
SAG Mill Grinding	
Feed F_{80}, mm	150
Transfer T_{80}, microns	1,180
SMC, A	69.7
, b	1.23
, t_a	0.89
Ball Mill Grinding	
Feed F_{80}, microns	1,180
Product P_{80}, microns	106
Ball Mill Work Index, kWh/t , 80th percentile (design)	15.88
, average	14.6
Lead Flotation	
Rougher Conditioning Time, min	10
Plant Rougher Flotation Time, min	35
Plant First Cleaner Flotation Time, min	12
Plant Cleaner Scavenger Flotation Time, min	12
Plant Second Cleaner Flotation Time, min	10
Plant Third Cleaner Flotation Time, min	10
Zinc Flotation	
Rougher Conditioning Time, min	10
Plant Rougher Flotation Time, min	24
Plant First Cleaner Flotation Time, min	8
Plant Cleaner Scavenger Flotation Time, min	8
Plant Second Cleaner Flotation Time, min	6
Plant Third Cleaner Flotation Time, min	6

The process mass balance was developed for the Corani process using MetSim. The process simulation assumed the following overall recoveries for lead, zinc and silver in each concentrate (lead and zinc concentrates).

Table 17-2: Metal Recoveries Used for Mass Balance Simulation

Concentrate	Lead Recovery, %	Zinc Recovery, %	Silver Recovery, %
Lead Concentrate	75%	7.3	70%
Zinc Concentrate	18%	69%	30%

These recoveries were used only for the mass balance and not in the calculation of metal recoveries in the financial model. The grades used as the basis for the simulation, were 1.17% Pb, 0.7% Zn, and 57 g/t Ag. M3 assumed that at these grades, the feed to the mill will be predominantly sulfide and will follow the recovery projections for the mixed sulfide ores.

17.3 CRUSHING AND CRUSHED ORE STOCKPILE

ROM ore will be trucked from the mine by dump haul trucks to a crusher feed hopper or to a stockpile ahead of the primary crusher. Ore will pass through a stationary grizzly located over the primary crusher dump pocket. ROM ore from the primary crusher dump pocket will dump onto an apron feeder that will discharge onto a vibrating grizzly. The vibrating screen will discharge oversize ore directly into the jaw crusher. The primary crusher will be a C200 jaw crusher with an opening of 2000 mm x 1500 mm. The vibrating grizzly undersize will combine with the primary crusher product onto the primary crusher discharge conveyor which will transfer the material to the stockpile feed conveyor and onto the coarse ore stockpile.

The discharge conveyor will feed ore onto the stacking conveyor, which will deliver the coarse ore to the stockpile. The coarse ore stockpile will have a total capacity of 94,000 tons and a live capacity of 22,000 tons, which is equivalent to about 24 hours of SAG mill feed. Three belt feeders (two operating and one standby) will reclaim crushed ore from the stockpile and transfer it onto the reclaim/SAG feed conveyor.

17.4 GRINDING

The grinding circuit for the Corani Project will be a conventional SAG Mill-Ball Mill system, where the SAG mill will operate in closed circuit with a water jet return system, and the ball mill will operate in closed circuit with hydrocyclones. A future pebble crusher is planned to crush the pebble-screen oversize before returning it to the SAG mill, converting the circuit to a standard SAG-ball-crush (SABC) arrangement.

17.4.1 SAG Mill

The SAG feed conveyor feeds 1,019 mtph to the SAG mill, which is 9.1 m diameter by 4.3 m long. The SAG mill discharges to the cyclone feed pump box with a transfer size (T_{80}) of approximately 1,180 microns. A water jet return system is designed to return oversized particles to the SAG for additional size reduction.

17.4.2 Ball Mill

The ball mill, which is 7.32 m diameter by 11.9 m EGL operates in closed circuit with a hydrocyclone classification. A circulating load of 250% of feed is pumped to the cyclone cluster using a 30x26 cyclone feed pump with a 1,120 kW variable frequency drive (VFD) motor. The cyclone underflow is returned to the ball mill. The cyclone overflow discharges to the flotation circuit with a P_{80} of 106 microns.

17.5 FLOTATION

The Corani process plant will produce separate lead and zinc concentrates using a conventional froth flotation process in two rougher stages. Lead will be floated first from the primary cyclone overflow slurry, followed by zinc flotation from the lead rougher tails. Lead flotation will be conducted at a pH of 8 which will be achieved by the addition of milk of lime in the grinding circuit. This pH adjustment was found to be necessary during the laboratory tests to improve pyrite rejection from the lead concentrates. For zinc flotation in the second stage, the pH will be increased to 11 by adding milk of lime to promote the collection of zinc sulfide. The laboratory results predict that a larger portion of the silver will report to the lead concentrate.

17.5.1 Lead Flotation

The cyclone overflow from the grinding circuit will first report to a series of two conditioning tanks. The conditioning reagents zinc cyanide and sodium sulfite will be added to the first conditioning tank. Sodium isopropyl xanthate (SIPX) and AP404, both collectors, and methyl isobutyl carbinol (MIBC), or equivalent, as frother, will be added to the second conditioning tank. Each conditioning tank is designed to provide a residence time of 5 minutes.

The conditioned slurry will then flow by gravity to the rougher flotation bank. Two flow schemes have been developed for lead regrinding. In the standard flow scheme design, the concentrate from all the cells in the rougher bank is sent to the regrind circuit. An alternate design scheme is to forward the concentrate from the first two cells to the third cleaner column cell via the second cleaner concentrate pump box. The alternate scheme is designed to be used when the mill feed type and grade result in high-grade concentrates from the first two rougher cells. Concentrate from the remainder of the lead rougher cells will follow the standard flow.

Lead rougher concentrate will be pumped to eight 102-mm diameter hydrocyclones. Cyclone underflow will flow by gravity to the lead regrind vertical mill which will operate in open circuit. Lead regrind cyclone overflow and lead regrind discharge will combine in the lead regrind pump box and then be pumped to the lead first cleaner circuit. The target particle size distribution for the lead first cleaner is a P_{80} of 25 microns.

Three stages of cleaning are designed to upgrade the reground lead concentrate to meet smelter specifications. The lead first cleaner concentrate will be transferred to lead second cleaner column cell. Tailing from the lead first cleaner circuit will be processed in the lead cleaner scavenger flotation circuit to produce tailing that can be forwarded to the zinc flotation circuit without significant lead loss. Concentrate from the cleaner scavenger flotation circuit will be returned to the first lead cleaner flotation circuit. Tailing from the cleaner scavenger circuit will be pumped to the zinc flotation circuit where it will combine with the rougher flotation tailing.

The concentrate from the lead second cleaner column cell will be pumped to the lead third cleaner column cell, while the tailing from the lead second cleaner flotation circuit will be recycled to the first lead cleaner flotation circuit. Concentrate from the lead third cleaner column will be pumped to the lead concentrate thickener as final lead concentrate and the tailing will be returned to the lead second cleaner column.

The sizes and numbers of the flotation cells that will be installed in the lead flotation circuits are shown in Table 17-3.

Table 17-3: Lead Flotation Cells

STAGE	NUMBER OF CELLS	SIZE OF CELLS m^3
Rougher	5	300
First Cleaner	4	50
Cleaner-Scavenger	3	50
2nd Cleaner Column	1	4.5-m dia.
3rd Cleaner Column	1	3.2-m dia.

17.5.2 Zinc Flotation

The zinc flotation flow design is similar to the lead flotation circuit, except it does not have the flexibility to advance part of the rougher concentrate to the third cleaner column. Laboratory test results show that this flexibility is not required for the zinc circuit.

Tailing from the lead rougher flotation circuit, along with the tailing from the lead cleaner scavenger flotation circuit, will report to the zinc rougher flotation circuit. The feed to the zinc flotation circuit will be conditioned in two conditioning tanks that will operate in series. Copper sulfate will be added to the first conditioning tank to activate zinc sulfide. SIPX, MIBC and milk of lime will be added to the second conditioning tank. Each conditioning tank will have a residence time of 5 minutes.

Zinc rougher concentrate will be pumped to eight 102-milimeter diameter hydrocyclones. Cyclone underflow will flow by gravity to the zinc regrind vertical mill which will operate in open circuit. Zinc cyclone overflow and zinc regrind discharge will combine in the zinc regrind pump box and then be pumped to the zinc first cleaner circuit. The target P_{80} is 25 microns. The cyclone underflow will be sent to the vertical mill for size reduction.

Three stages of zinc cleaner flotation and a zinc cleaner scavenger flotation circuit will upgrade the reground zinc concentrate. The first cleaner concentrate will be transferred to the zinc second cleaner flotation circuit. Tailing from the zinc first cleaner circuit will be processed in the zinc cleaner scavenger flotation circuit. Concentrate from the cleaner scavenger flotation circuit will be returned to the feed of the zinc first cleaner flotation circuit. Tailing from the cleaner scavenger circuit will be pumped to the final tailing thickener.

The concentrate from the zinc second cleaner column will be pumped to the zinc third cleaner flotation column. Concentrate from the zinc third cleaner flotation column will be pumped to the zinc concentrate thickener as final zinc concentrate. The tailing from the zinc second cleaner flotation circuit will be recycled to the zinc first cleaner flotation circuit.

Tailing from the zinc rougher flotation circuit, along with the tailing from the zinc cleaner scavenger flotation circuit, will report to the final mill tailing thickener.

The sizes and numbers of the flotation cells designed for the zinc flotation circuits are shown in Table 17-4.

Table 17-4: Zinc Flotation

STAGE	NUMBER OF CELLS	SIZE OF CELLS m^3
Rougher	5	300
First Cleaner	5	20
Cleaner-Scavenger	5	20
2nd Cleaner	1	3.6-m dia.
3rd Cleaner Column	1	2.6-m dia.

17.6 CONCENTRATE THICKENING, FILTRATION, STORAGE

Lead concentrate from the third lead cleaner flotation circuit will be dewatered in the 10-m diameter lead concentrate thickener. The thickened lead concentrate will be pumped to an automatic tower filter press using 20 - 900 x 1,750 mm plates. The filtered concentrate will be conveyed to the lead concentrate surge hopper at approximately 8 % moisture. From the hopper, the lead concentrate will be loaded into super sacks and transported from the Project via trucks.

Zinc concentrate from the third zinc cleaner flotation circuit will be dewatered in the 8-m diameter zinc concentrate thickener. The thickened zinc concentrate will be pumped to an automatic tower filter press using 16 - 900 x 1,750 mm plates. The filtered concentrate will be conveyed to the zinc concentrate surge hopper at approximately 8 % moisture. From the hopper, the zinc concentrate will be loaded into super sacks and transported from the Project via trucks.

Concentrate will be loaded onto trucks and shipped to a port where it will be stored and loaded into a ship to be taken to the smelter for further processing.

17.7 TAILING THICKENING

The tailing from the zinc rougher flotation circuit and zinc cleaner scavenger flotation circuit will flow by gravity to the tailing thickener. The slurry will be dewatered from approximately 20 percent solids by weight in the feed to the thickener to approximately 45 to 55 percent solids by weight in the thickener underflow. The thickened tailings will be pumped to four agitated tailing filter feed tanks, which will provide slurry to eight 2.5 x 3.5 m vertical plate filter presses with 74 plates each. The filtered tailing containing approximately 17% moisture by weight will be conveyed to a radial stacker and stockpiled. A front end loader will be used to load haul trucks which will transport the tailing to the waste rock storage facility for spreading and compaction. Filtrate from the tailing filters will be reused in the process plant.

17.8 REAGENTS AND CONSUMABLES

Reagent storage, mixing and pumping facilities will be provided for all of the reagents used in the processing circuits. Table 17-5 below is a summary of consumables used in the process plant.

Table 17-5: Process Consumables and Consumption Rates

Consumable	Consumption g/t
Reagents	
Sodium Isopropyl Xanthate (SIPX)	40
Cytec AP 404	15
Lime	3500
Zinc Sulfate	620
Sodium Cyanide	210
Copper Sulfate	290
Methyl Isobutyl Carbinol (MIBC)	50
Sodium Sulfite	505
Sodium Hydroxide	10
Flocculant	20
Antiscalant	5
Grinding Media & Wear Steel	
SAG Mill Balls	500
Ball Mill Balls	500
Regrind Balls	20

Consumable	Consumption g/t
Primary Crusher Liners	8
SAG Mill Liners	50
Ball Mill Liners	30
Regrind Mill Liners	10

17.9 WATER REQUIREMENT

Water resources for supporting a mining and mineral processing operation in the high Andes is typically problematic. Careful consideration has been given to the water demands of the project and the potential sources of water to support the operation. Surface water captured from runoff during the wet season has been identified as the source for water used to support the operation. Project water demand is dominated by the makeup requirement for the process plant. Process water demand was used as the foundation for determining the sitewide water balance after estimating the additional water demands, such as dust control.

17.9.1 Process Water Balance

A water balance for the process plant was developed for the Corani project using MetSim modeling (Figure 17-2). The Corani process plant is projected to require 225 m^3/h of fresh water makeup to sustain its operation. In addition, an average of 147 m^3/h of fresh water is estimated for mine dust control. The total fresh water requirement will then be 372 m^3/h. This is equivalent to 0.4 m^3 of water per tonne of ore processed, which is within typical operating ranges.

The fresh water makeup for the Corani project considers the water loses in the following: lead concentrate, zinc concentrate and the filtered tails. An allowable moisture of 17% (wt%) was considered in the final tails. The water balance might slightly change once the allowable moisture from the geotechnical report is received. Fresh water for reagent mixing and potable consumption is also considered in the overall water balance. The use of recycled process water in various points of the plant is reflected in the overall water balance. Evaporation loses were not considered in the overall water balance; this will be updated once information is received.

17.9.2 Sitewide Water Balance

A project-wide water balance was developed for the operational mine life as part of project design. That model was developed to allow the estimation of seasonal plant makeup water requirements as well as estimate the seasonal quantities of natural, non-contact water that would be released to the downstream environment at the project boundaries.

The water balance considers the inflows and outflows to each Project area including precipitation, evaporation, base flow, process plant water consumption and recycle, and water releases to the community. These flows are varied over the life of the project on a monthly basis to determine the process makeup water and non-contact, storm water capture and release requirements. This variation also occurs as facilities are operated over time. For example, as the pit increases in size the amount of contact water increases. Following the ramp-up period, the annual process consumptive use of water remains steady over the life of the mine.

The intent of the model is to determine the probable ranges of conditions that may exist during operations. The number of multiple input parameters, their probable variations, and resultant possible combinations inhibit the creation of a deterministic model for operational conditions. The model however, will form the basis for additional operational water balance calculations prior to commissioning of the project and during operation and closure of the project facilities. Updating of the model parameters will be performed as additional input data and operational data become available.

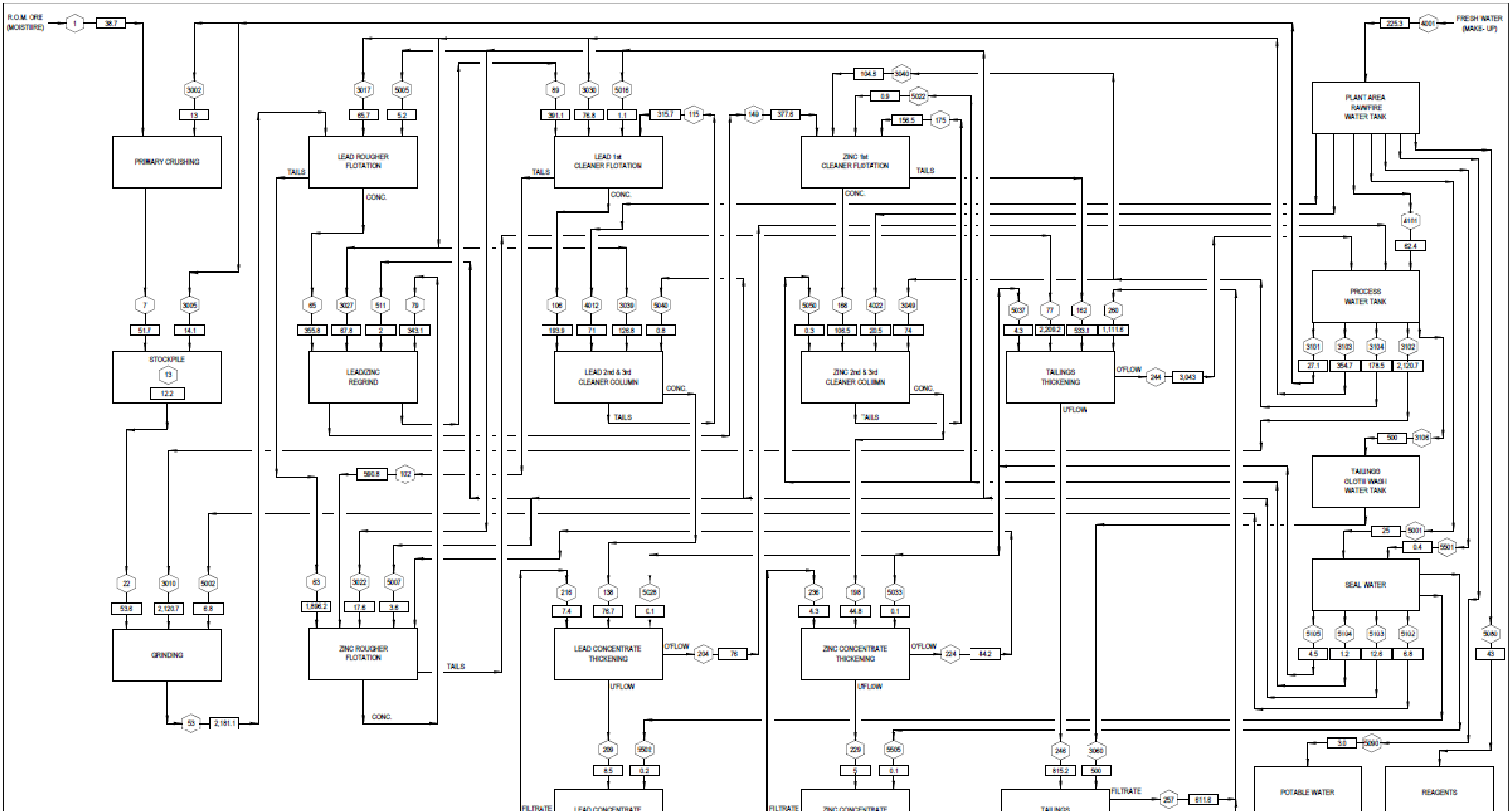
The water balance model is based on climate parameters adopted for the project ESIA (Amec, 2013). Precipitation inflows are based on a synthetic precipitation series that includes two dry years at the start of operations. The synthetic series also includes some years that are wetter than average. It is intended to simulate the natural variability that would be expected over the project life. A site weather station has been recording hourly data since December 2008, and flume stream gauging stations have been recording hourly data at the site since 2012. Site-specific runoff coefficients were calculated using the combined data record available from 2012 to 2015. Incorporation of these site data measurements offers a significant improvement over previous studies, which relied on literature based assumptions for runoff coefficients. Groundwater inflows to the mine pits were estimated based on a transient MODFLOW groundwater model reflecting the mine plan and pit backfilling schedule described in Section 16.

The total process water demand is a function of the plant tailings production rate and the water content of the tailings leaving the plant. Process plant makeup water demand is satisfied based on water available in the watershed, with the following water supply priority order: contact water (runoff and baseflow from disturbed areas and pit inflow), non-contact water (runoff and baseflow from undisturbed areas in the catchment), water from the upstream cell of the Plant Water Pond (contact water and stored non-contact water), and finally, water from the downstream cell of the Plant Water Pond (stored non-contact water). Once the plant demand is met, the excess non-contact water from the remaining area(s) is either stored in the Plant Water Pond or released downstream. Under normal expected operating conditions all contact water is consumed by the plant or temporarily stored in the upstream cell of the Plant Water Pond and is not discharged to the environment.

The Plant Water Pond consists of two adjoining ponds or cells. The downstream cell (approximately 550,000 cubic meters) is for the storage of non-contact water, the Non-Contact Water Pond. The upstream cell (approximately 200,000 cubic meters) is primarily for the storage of contact water, the Contact Water Pond, but when the supply of contact water is low, it may be necessary to supplement the supply with non-contact water. The two pond system is intended to minimize the amount of contact water produced at the project while still ensuring a dependable water source is available during the dry season to maintain operation of the process plant.

If the makeup water demand is greater than the contact water supply, the water is sourced first from the Contact Water Pond (mixed contact/non-contact water), and second from the Non-Contact Water Pond (only stores non-contact water). During years 2 through 5 of operation, the plant demand is sufficient to consume all contact water.

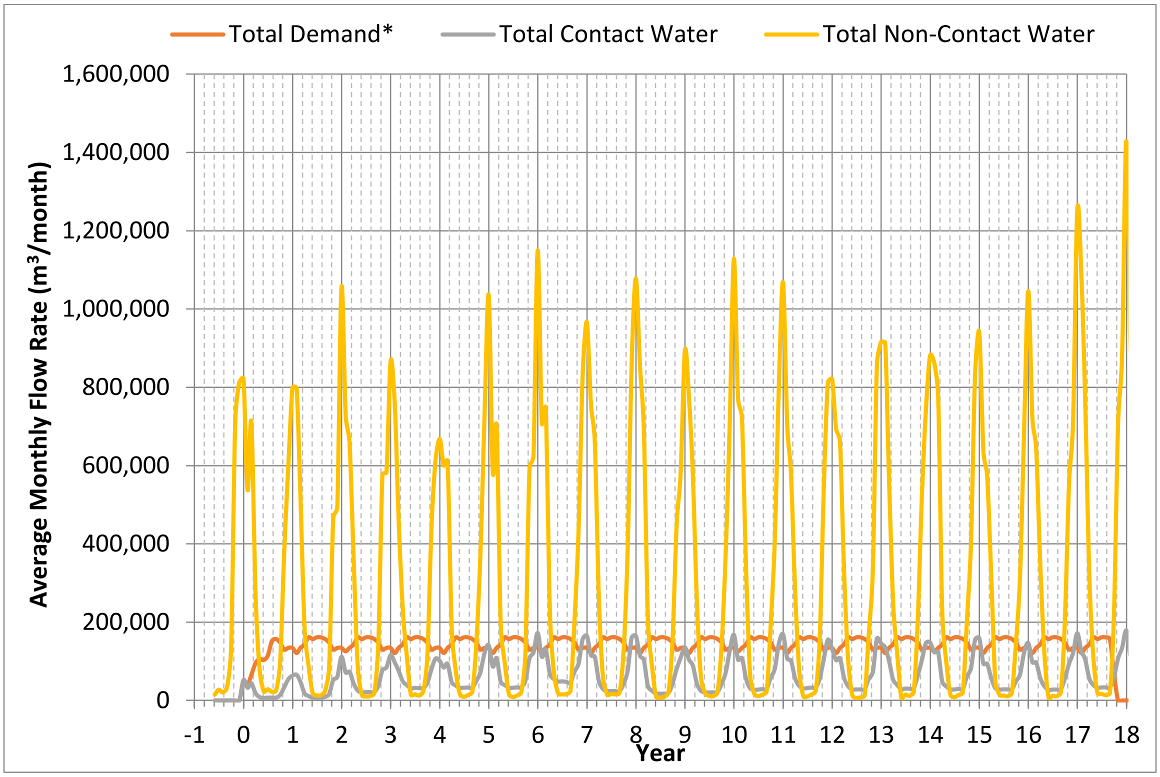
The storage capacity of the downstream cell represents nearly 4 months of plant demand and is necessary to provide makeup water to the plant during the dry season when very little makeup water is naturally available from the watershed. The average dry season storage requirement over mine life is around 550,000 cubic meters and ranges from 475,000 cubic meters to 715,000 cubic meters. The dry season storage requirement is dependent on the amount of contact water available to meet the immediate demand of the plant. Years when the storage requirement exceeds the storage capacity of the non-contact pond occur early in mine life when the amount of contact water is at a minimum (see Figure 17-3) due to the small footprint of the mine pits. During these times it will be necessary to store non-contact water in the Contact Water Pond (see Section 18). These years correspond to years when there is little to no contact water in the Contact Water Pond (Figure 17-3), and the Contact Water Pond will primarily operate as an extension of the Non-Contact Water Pond. However, some excess storage capacity will be maintained in the contact water pond to capture contact water in the case of an extreme event.



NOTE:
1. EVAPORATION LOSSES ARE NOT CONSIDERED IN THE WATER BALANCE.

Metsim No.	Description	In (MTPH Soln.)		Metsim No.	Description	Out (MTPH Soln.)
1	Ore Moisture	38.7		13	Stockpile	12.2
4001	Fresh Water	225.3		211	Lead Concentrate	1.4
				231	Zinc Concentrate	0.8
				258	Filtered Tails	203.6
				804	Reagents Mixing	42.2
				5091	Potable Water	3.0
Total		264.0				263.2

Figure 17-2: Process Water Balance



*Note: "Total Demand" includes process plant demand and water required for dust suppression on mine roads

Figure 17-3: Total Monthly Inflows and Consumption

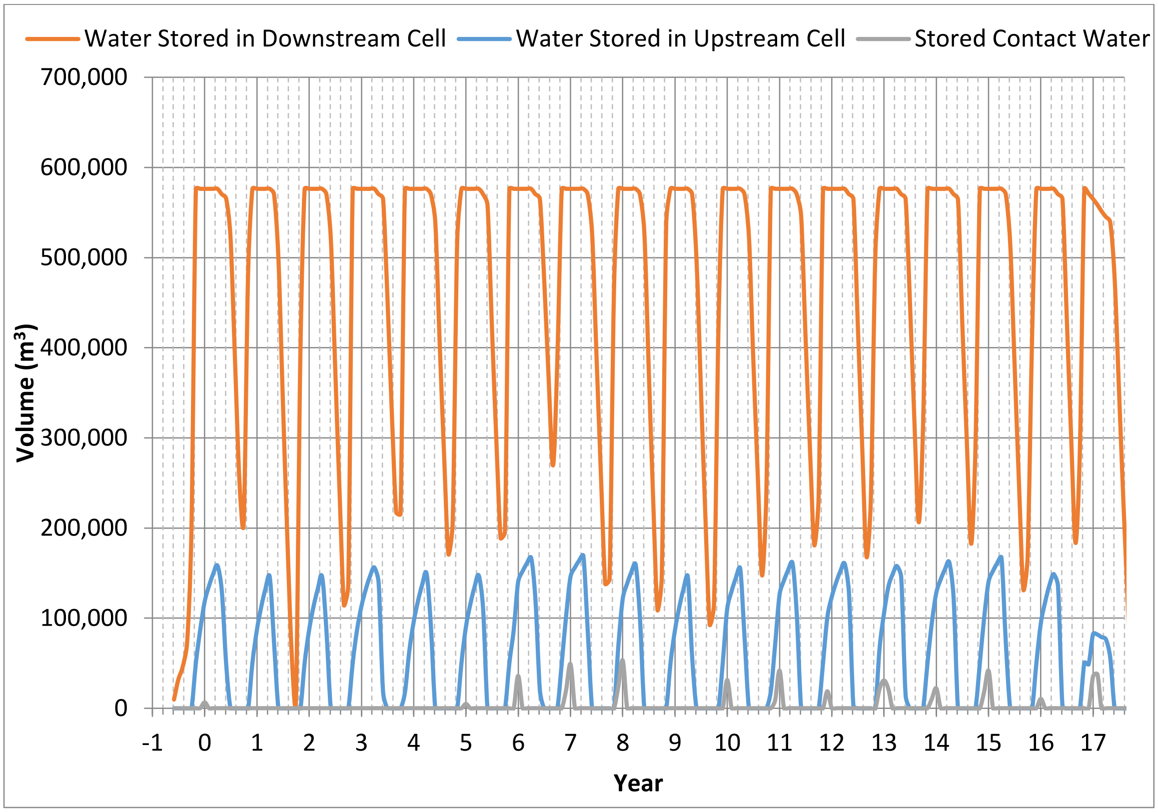


Figure 17-4: Plant Water Pond Storage

Due to the reduction in the makeup water demand relative to the previous study, the water available in the watershed greatly exceeds what is required for operations on an annual basis. Based on available flume data, the annual flow generated in the catchment in a year with average total precipitation is more than twice the annual plant makeup demand. In addition, progressive reclamation measures incorporated into the optimized project design significantly limit actively disturbed areas that produce contact water, minimizing the volume of contact water requiring storage.

17.10 MILL POWER CONSUMPTION

The power consumption in the process plant for a typical year is tabulated in Table 17-6, with a total consumption of 258.5 billion watt-hours. This translates to about 38.2 kWh per tonne of ore processed.

Table 17-6: Summary of Mill Power Consumption in a Typical Year

Cost Item	Connected kW	Total (kW hr/yr)
Concentrator		
Primary Crushing & Conveying	838	3,973,301
Grinding	23,437	152,587,405
Flotation	11,528	66,212,612
Concentrate Thickening/Filtration	984	5,753,610
Reagents Storage	222	1,468,600
Tailing Management & Reclaim Systems	3,348	23,958,123
Water Supply System	835	3,746,138
Ancillary	245	839,592
Total Connected (kW)	41,438	
Total Consumption (kW-hr)		258,539,382

17.11 CONTROL PHILOSOPHY

17.11.1 Process Control Philosophy

The plant will include standard process control systems that may be operated from a central control room. Two x-ray analyzers and two particle size monitors will provide process control for the performance of the grinding circuit and the flotation circuits. Standard process control equipment such as flow meters, level detectors, and density gauges will provide input to the process control system.

17.11.2 Control Systems

A crusher control room, located in the primary crusher area at the mine will be the operating control center for the crusher and coarse ore transport conveyors.

A central control room (CCR) will be provided in the concentrator facility core, which will be the main operating control center for the complex. From the CCR control consoles, primary crushing, material handling systems, grinding and flotation, reagents, tailing, and utility systems will be monitored and/or controlled.

A computer room, located adjacent to the CCR will contain engineering workstations (EWS), a supervisory computer, historical trend system, management information systems (MIS) server, programming terminal, network and communications equipment, and documentation printers. This will be primarily used for Distributed Control System (DCS) development and support activities by plant and control systems engineers.

Although the facilities will normally be controlled from the CCR, local video display terminal will be selectively provided on the plant floor for occasional monitoring and control of certain process areas. Any local control panels that are supplied by equipment vendors will be interfaced with the DCS for remote monitoring and/or control from the related control room.

18 PROJECT INFRASTRUCTURE

The infrastructure for the Corani Project requires significant development and planning. The site is remote, at high elevation, and a considerable distance from major urban areas. The infrastructure developed for the Project includes transportation, process buildings and related facilities, water supply and management, power supply, communications, and material storage stockpiles.

Several project components were optimized subsequent to the December 2011 Technical Report (M3, 2011). Detailed engineering studies, site investigation work, and laboratory testing programs were ongoing at the time of the December 2011 report. The optimizations presented in December 2011 have been advanced through additional fieldwork and detailed engineering to support the optimization concepts presented in this study. A brief summary of the work performed subsequent to the 2011 report for the present 2015 study is presented below.

- Additional metallurgical and geotechnical drilling within the Corani Project area
- Re-logging and re-interpolation of the mineralogy
- An updated mineralogical database and block model
- Geometallurgical model for predicting recovery within the block model
- Updated capital and operating cost estimation
- Mine plan optimization
- Geotechnical site investigations and waste characterization test work and studies
- Detailed process review
- Project-wide water balance studies
- Engineering and design to advance the project to feasibility study level
- Updated quantity and cost estimation

The current components and arrangement of facilities is described in this section of the Report. Several of these project components are described in more detail in other sections of this report, and only a general description of the relevant aspects of the project infrastructure-related components is given here. The following components of the project are described as follows.

- Rescheduled Mine Plan incorporating the ability to partially backfill the open pits
- Reconfigured waste rock management facilities
- Elimination of the Tailings Storage Facility and inclusion of Dry Stack Filtered Tailings System
- Elimination of East Waste Rock Dump
- Co-deposition of waste rock and tailings
- Elimination of low-grade ore stockpile
- Updating of project-wide surface water management plan
- Removal of one fresh water pond, retention of a one-pond system with two compartments (contact water and non-contact water)
- Updated surface water management plan for the new facilities
- Peruvian contractor estimates for the power line, camp and access road
- Re-sizing of the large equipment shop and location
- General site arrangement modifications

18.1 TRANSPORTATION

Transportation to and around the site is by roadways that have to be developed or improved to accommodate the demands of the project. An access road will have to be constructed to link the project site to the Interoceanic Highway that provides access to the town of Macusani and to the rest of the country for receiving supplies and delivering products. The lead and zinc concentrate produced by the mining and mineral processing operations will be delivered to the Port of Matarani or other destination via trucks using the access road and public highway system of Peru.

18.1.1 Access Road

The access road to the Corani Project will be a 46 km roadway consisting of established roads, new roadway built on an existing alignment, and roadways built on a new alignments (Figure 18-1) The route to the mine site follows Interoceanic Highway 34B (Carretera Interoceánica) approximately 15.8 km from Macusani. The Interoceanic Highway is a two-lane, paved highway that connects Macusani to the Peruvian port cities of Matarani and Ilo. Three segments of new roadway from its intersection with the Interoceanic Highway at the Huiquisa bridge to the residential camp to be constructed for mine construction and operations personnel were designed by Anddes Asociados SAC (Anddes,). The final section of the roadway from the residential camp was designed by HC & Asociados (HC&A, 2012).

The first segment of the Anddes alignment is 8.3 km long, starting at the Huiguisa bridge on the intercontinental highway and ending in the village of Tantamaco (Figure18-2). This segment contains five tunnels and four bridges. The second segment is 2.8 km from Tantamaco to the village of Isivilla. This segment follows and existing road that will be upgraded to the roadway width and minimum radius of the design configuration. The third segment is 22.55 km long from Isivilla to Jarapampa in the valley of the Chacaconiza River.

The fourth segment of the access road is 12 km and follows the alignment designed by HC&A (2012) from Jarapampa to the residential (Figure 18-1). This section of road climbs out of the Chacaconiza valley, passes the site of the residential camp, and follows a topographic ridge south-southwest to the mine entrance.

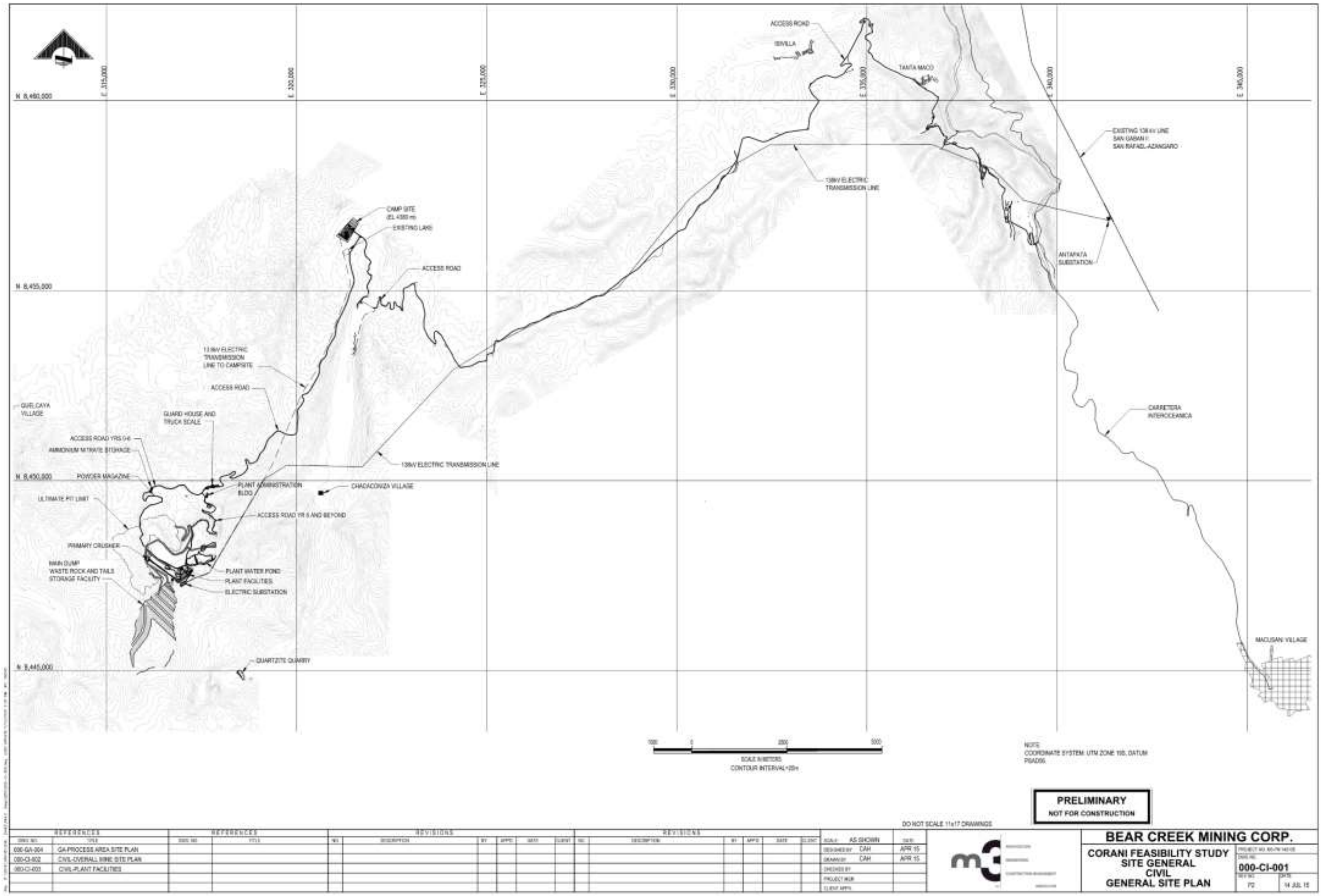


Figure 18-1: General Site Plan

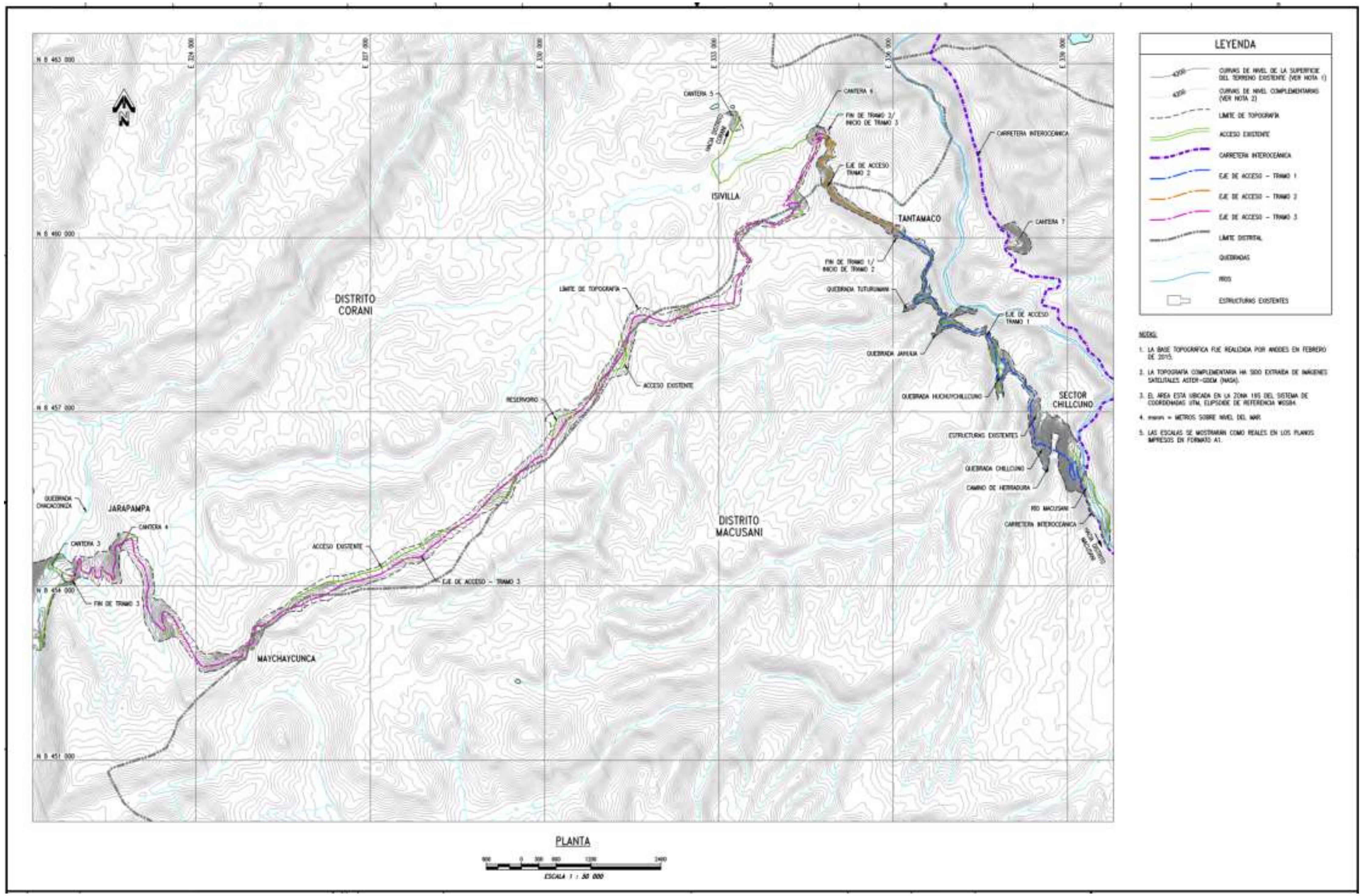


Figure 18-2: Access Road Segments 1, 2, and 3

(after Anddes, 2015)

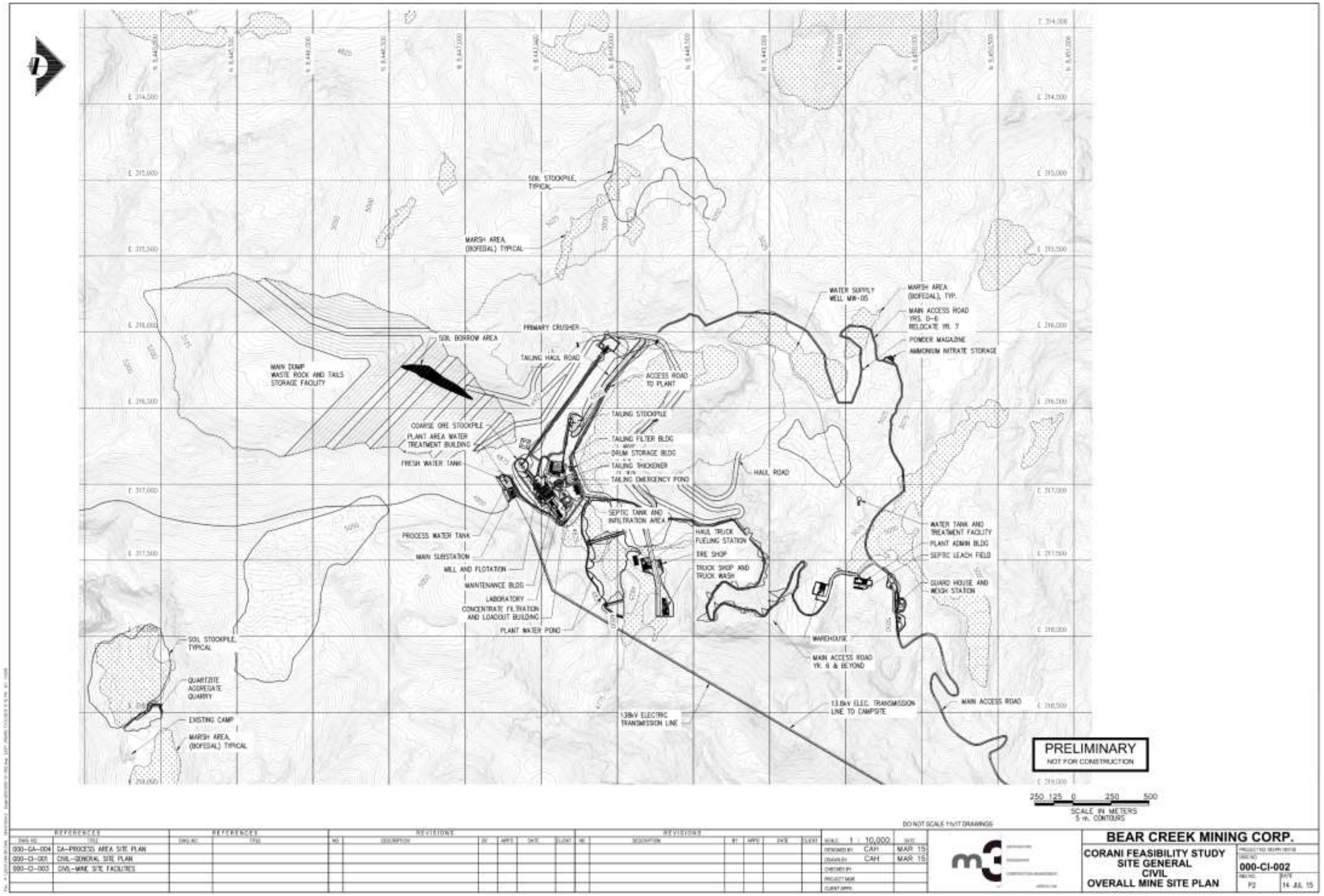


Figure 18-3: Overall Mine Site Plan

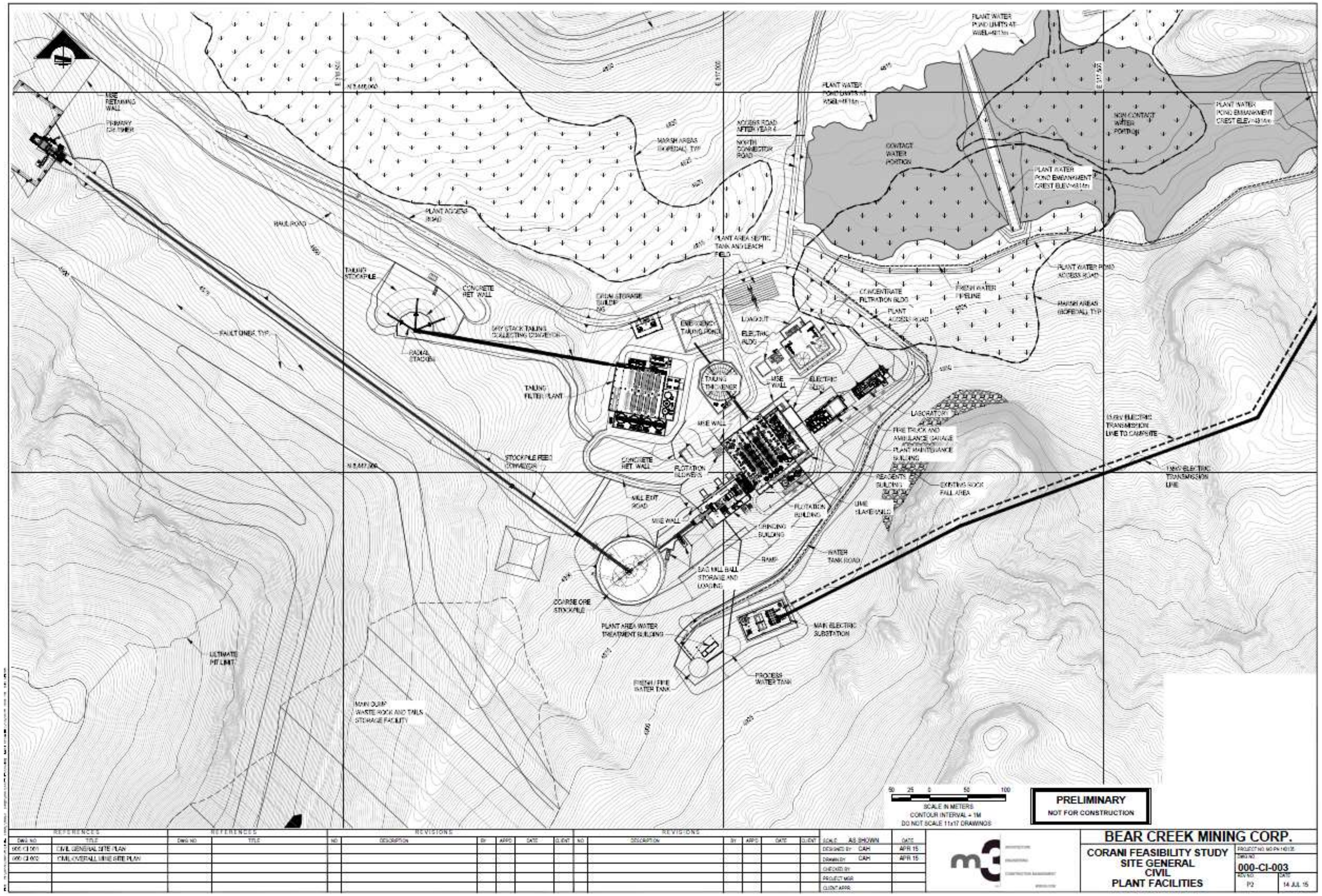


Figure 18-4: Mine Site Facilities Drawing

18.1.2 Site Roads

A network of roads has been designed to connect the facilities within the mine and plant site (Figure 18-3). The final section of the mine access road connects the residential camp to the mine entrances, as presented in Section 18.1.1. (Figure 18-1). The mine entrance is located at approximately 5,000 masl on a ridge overlooking the mine and processing plant site. During construction and the first several years of mining, traffic from the mine entrance to the plant site will be on a road that is routed west around the Este Pit location, past the primary crusher, and down to the plant, which is at an elevation of approximately 4,850 masl (Figure 18-3) In Year 7, a new road will be established east of the Este Pit to connect the mine entrance and administrative complex to the plant site. Additional roads for access to the various components of the processing plant, the Plant Water Pond, and other project facilities have be laid out and included in the initial capital cost (Figure 18-4).

18.2 SITE BUILDINGS AND FACILITIES

Corani Project buildings and facilities are divided into four functional areas: Administration, Mine Facilities, Process Facilities, and Residential Facilities.

The mine facilities include the following.

- Powder magazine
- Ammonium nitrate storage
- Yard storage
- Truck wash
- Truck fuel storage and fueling station
- Tire shop

The administrative facilities are located near the main entrance and include the following (Figure 18-3).

- Guard house and weigh station
- Administration building
- Warehouse

The process facility and ancillary buildings include the following (see Figure 18-4):

- Primary crusher
- Crushed Ore Storage and Reclaim
- Grinding, flotation, and reagents buildings
- Concentrate handling and load out building
- Tailings thicker
- Tailings filtration building
- Tailings stacker and stockpile
- Plant maintenance and emergency services
- Analytical laboratory
- Plant water storage and treatment area
- Electrical substation
- Plant water supply pond
- Plant waste water treatment facility

- Tailings emergency containment pond
- Drum storage building

The residential camp facilities are located approximately 12 km northeast of the mine entrance along the access road and include the following.

- Dining and food preparation
- Residential accommodations
- Medical services
- Recreational facilities
- Water and wastewater management
- Security services

18.2.1 Mine Service Facilities

Mine service facilities area will be northeast of the main plant facilities and east of the Este pit. This area includes the Truck Shop, Tire Shop, and Fuel Facility. This area will provide services and support to the mining operation and equipment fleet.

The Truck Shop building will house offices, men's and women's dry, break/lunch room, warehouse, tool cribs, electrical and mechanical rooms, lube room, electrical repair shop, welding/repair area, three large equipment repair bays, two light vehicle repair bays, light vehicle parts storage, and a tank farm with containment for lubricants, fluids, and waste products. The building will be a pre-engineered structure with insulated metal roofing and siding. The foundation will be concrete spread footing, piers and grade beams. Concrete floor slabs will be provided over the entire area of the building, with thicknesses suitable for the offices, warehousing, and mine truck support and aprons. An uncovered Truck Wash bay will adjacent to the Truck Shop.

A Tire Shop with reinforced concrete slab, air compressor, tire handler, and small office structure will be constructed near the truck shop. Other mine facilities include ready-line and fuel storage and dispensing facilities, a mine control tower and explosives storage facilities, including a powder magazine and ammonium nitrate storage bin.

18.2.2 Administration Facilities

The administration facilities are located near the main entrance partially because of space constraints and partially to keep suppliers and non-essential personnel out of the mining and process areas. The administration facilities include the main gate and guard house, an administration office building, and a warehouse to receive parts and supplies necessary for operation and maintenance.

18.2.2.1 Security

The main mine guard house and security office will be located at the entrance to the mine site on the access road. The guard house will contain security offices, restrooms, and a small reception area. A Peruvian security firm will be contracted to provide on-site security services to the Project starting at the time of preproduction. A truck scale is located nearby to weigh loads of supplies coming into the site and concentrate trucks leaving the site.

18.2.2.2 Warehouse Building

The warehouse building will be a pre-engineered structure with metal roofing and siding. An enclosed storage area is included, a dock for truck unloading and an external fenced warehouse yard adjacent to the building is also provided. Offices in the building and workshop areas will be ventilated.

18.2.2.3 Administration Building

There are two administration buildings, one that is located close to the process facilities that will include facilities such as This facility will be an air-conditioned and heated pre-engineered metal building with insulated roofing and siding, and installed on a concrete mat foundation. A gravel-surfaced visitors parking area and employee parking area is also provided.

The administration building is located close to main entrance and access road and includes offices, a reception area, a conference room, a training room, and a first-aid room. The offices will be used for senior personnel, records and archives, accounting, and engineering. Other areas of the building contain conference rooms, restrooms, and a reception area. This facility will be a heated/air conditioned pre-engineered metal building with insulated roofing and siding, and installed on a concrete mat foundation. A gravel surfaced visitors and employee parking area is also provided.

18.2.3 Process Facilities

The process plant facilities include a primary crushing facility located close to the mine, crushed ore stockpile area located close to the concentrator area, a grinding building, a flotation area, a concentrate dewatering and load-out facility, a tailing storage facility, a plant maintenance building, administration building, and laboratory building.

18.2.3.1 Primary Crusher

The primary jaw crusher structure will be of reinforced concrete and structural steel construction. The dump pocket is designed to receive two CAT 785 haul trucks at the same time. A hydraulic rock breaker for reducing oversize mined material is provided. Crusher maintenance will be performed with mobile cranes.

18.2.3.2 Crushed Ore Storage and Reclaim

From the primary crusher, overland conveyors will transport the crushed ore to a crushed ore storage facility which will be an open stockpile with no superstructure. A concrete tunnel will be provided beneath the stockpile to house reclaim feeders. The feeders discharge onto the SAG mill feed conveyor that connects the stockpile to the mill building. A dust extraction system will be provided to control the emission of fugitive dust. All transfer points between belt feeders and the SAG mill feed conveyor will be connected to the dust extraction systems through ducting. Belt conveyors outside the tunnel and building will have belt covers to minimize dust emissions.

18.2.3.3 Grinding Area

The Grinding Building will house a SAG grinding mill, ball mill, primary cyclones, mill discharge screen and primary cyclone feed pumps in an engineered structure with metal roofing and siding. Internal structural steel platforms that provide maintenance access are supported independent of the main building structure. The building foundations will be concrete spread footing, piers and grade beams. Concrete floor slabs will be provided over the entire area of the building with curbed containment for spillage control and thicknesses suitable for the maintenance traffic where appropriate. Overhead travelling cranes are provided for maintenance of the grinding equipment. Controls rooms and offices within the structure will be air conditioned/heated, with high-bay lighting provided throughout.

18.2.3.4 Flotation Area

The flotation circuit has an engineered structure with metal roofing and siding. This building will house four phases of (air-induced) flotation; pump systems, lead and zinc regrind ball mills, as well as zinc, and lead concentrate thickeners.

Concrete floor slabs with curbed containment are provided. Flotation tanks will be elevated on concrete and structural steel supports and cell covers and structural steel access platforms will be included. Maintenance on the flotation circuit will be facilitated by two cranes to serve the lead and zinc sections separately. High-bay area lighting will be provided to illuminate the area.

18.2.3.5 Concentrate Dewatering and Loadout

Concentrate filtering and load-out facilities will be located within an engineered structure with metal roofing and siding. The filter area includes two Larox-type filters: one filter for lead, and one for zinc. Filtered concentrates will be stored in segregated piles underneath the filters. Internal structural steel platforms that provide maintenance access are supported independent of the main building structure. The building foundations will be concrete spread footing, piers and grade beams. Concrete floor slabs will be provided over the entire area of the building with curbed containment for spillage control and thicknesses suitable for the maintenance and concentrate transport truck traffic where appropriate.

Filtered concentrates from the stockpiles will be transferred to bagging machines to prepare the concentrates for loadout and shipment to market. The load-out facility has a covered area and loading docks to load highway transport trucks with a fork lift.

18.2.3.6 Tailing Filtering Facility

The tailing filtering facility includes a thickener, agitated filter feed tanks, and recessed plate filters for dewatering the tailings. Support equipment includes compressors to blow air through the compressed filter cake at the end of the cycle and conveyors with a radial stacker to transport the filtered product to a stockpile area. Trucks will be used to haul the filtered product to the Main Dump area for co-deposition with waste rock from mining operations.

18.2.3.7 Reagents Area

A pre-engineered reagent mixing building with metal roofing and siding will be provided to store, prepare and supply reagents to the process. The reagent mixing and storage facility will be located adjacent to the flotation area to receive and store reagent supplies. The reagent building includes a truck unloading area, inside storage, mixing tanks, distribution tanks, an overhead crane, and concrete with containment and floor sumps. Cyanide is kept in a separate area in accordance with Peruvian regulations.

18.2.3.8 Plant Maintenance Building

The maintenance workshop building will be a pre-engineered structure with insulated roofing, siding, and a reinforced concrete mat foundation. Offices, a conference room, a copy room, a break room, restrooms, showers, a tool crib, work areas including mechanical maintenance area with an overhead crane, and an electrical/instrumentation repair area are provided.

18.2.3.9 Analytical Laboratory

The laboratory building will be a pre-engineered structure with insulated roofing and siding, and a reinforced concrete mat foundation. Laboratory facilities include sample receiving, sample drying, sample preparation, metallurgical lab, wet lab, fire assay, electrical and mechanical rooms, a computer room, men's and women's restrooms and locker facilities, a lunch room, a loading dock, and various offices. The building will be heated/air conditioned, and fume extraction and dust collection equipment is provided.

18.2.4 Residential Facilities

The distance of the Project site from any significant urban area that could provide lodging and services to mine personnel requires that the Project include a residential camp. The camp site is located at an elevation of approximately 4,400 masl at a distance of about 10 km to the northeast of the mine facilities. The UTM, Zone 19S, Datum PSAD56 coordinates of the site are: 8,456,510 m North; 321,312 m East.

The camp has been organized in such a way that the facilities are located in a central core. The core/circle design is to offer higher protection against cold wind and other inclement weather.

Parking areas are centrally located close to the dormitories. Vehicle traffic is limited in dormitory areas, and only design for pedestrians is considered. The inner circle distributes traffic for facilities centrally located for delivery of supplies.

The external circle is for ease of pick-up of workers to transport them to the mine and return to the camp.

The camp includes the following buildings:

- Cafeteria and kitchen;
- Laundry;
- Recreation facility;
- Store;
- Medical facility;
- Camp administration;
- Camp sewage treatment;
- Camp water treatment;
- Camp maintenance;
- Dormitory Type 1, 12 men;
- Dormitory Type 2, 24 men
- Dormitory Type 3, 64 men
- Dormitory Type 4, 36 men
- Camp guard/security house; and
- Recreation area (open area).

In this camp concept, areas have been set aside for construction by contractors of housing facilities. The camp design will provide utilities throughout the entire site. Food and recreational facilities constructed will be available for use by owner and contractor personnel.

18.3 WATER SUPPLY AND MANAGEMENT

Mineral processing operations require large volumes of water, which is often problematic in arid, high altitude terrain. Water management is also a significant concern with regard to potential environmental impacts, especially to people and lands downstream from an industrial operation. Surface water will be used to supply almost all of the water needs for the Corani Project. The exception is that a groundwater well will be used to supply water to the Administration area near the main entrance. Natural surface water runoff from the areas of disturbance related to the project is sufficient under average climatic conditions to provide the makeup water needs for the Corani Project. This water can be obtained through the collection of stormwater during the wet season. However, during the dry season, baseline stream flows and stormwater runoff are insufficient for process needs. In order to meet this requirement, a pond system with an approximate capacity of 750,000 m^3 is planned. The issues of surface water management water supply, fire protection, sanitary waste management, and are presented in the following sections.

18.3.1 Surface Water Management Plan

The site surface water management plan was formulated to provide a conceptual management template for management of storm water runoff and stream flows in the Project area. The effective management of surface water resources at the Project site is critical to the protection of water resources downstream of the Project area. In general, surface water will be managed to separate water that has not been in contact with potentially acid-generating material (non-contact water) from water that has been in contact with material exposed during mine development (contact water). Non-contact water will generally be diverted around the project facilities, except when mineral processing needs dictate the use of some or all of the non-contact water (freshwater). Major surface water conveyance and storage structures have been designed using hydrology and hydraulics software. Data used in the models was derived from previous and ongoing site investigations. Where input criteria required assumptions, those assumptions were based upon conservative estimates.

Development of the mine pits will be completed sequentially, with site conditions changing frequently. To effectively demonstrate evolving site conditions, surface water management features and designs have been presented in “snapshots” for various years during the mine-life, including the preproduction and closure periods. In practice, the surface water management features will be reevaluated and expanded as necessary each year during the dry season in order to accommodate the expanding project footprint.

During construction, contact water will be directed to settling ponds for recycling and treatment, as necessary. During production, contact water will be used in mineral processing. At closure, contact water will be minimized by reclamation activities. However, portions of the mineral processing plant will be converted to a water treatment plant to treat any seepage from the main dump and mine pits during the initial closure period, until such time as water quality reaches a condition that allows release without treatment.

Diversion ditches and associated culvert systems, as well as ponds, sumps and pipelines, have been designed to address the majority of surface water flow at the project site. In addition, best management practices (BMPs) will be employed to minimize erosion and sediment transport as well as deposition of sediment at the project site. BMPs will include the use of temporary diversion ditches, check dams, rock-containment berms, straw wattles/coir logs, silt fencing, terraced and bermed slopes, ditch linings, riprap, erosion matting/blankets, rock or geotextile cover,

18.3.2 Water Supply

Storage ponds are necessary to handle excess contact water runoff and to meet plant makeup water demand during the dry season (typically May through September), when insufficient runoff is available for mineral processing and dust suppression. The Freshwater Pond and Contact Water Pond are located northeast of the plant complex (Figure 18-4). The ponds have approximately 750,000 m^3 of storage, equivalent to approximately 5 months of makeup demand.

The Freshwater Pond and Contact Water Pond are adjacent to one another (Figure 18-5). The Contact Water Pond is upstream and is capable of storing approximately 200,000 m^3 of contact water. The Freshwater Pond is downstream and is capable of storing approximately 550,000 m^3 of non-contact water. Zoned embankment dams will partition the basin into two ponds for the purpose of managing contact and non-contact water separately. The two pond system is intended to minimize the amount of contact water produced at the project while still ensuring a dependable water source is available during the dry season. The dams will have a low-permeability core, compacted rockfill shells, a seepage cutoff, and internal drainage and filter control. The dam separating the Contact Water Pond and Freshwater Pond will be 15 m high, with a 1 m freeboard allowance. The Freshwater Pond embankment dam will be 35 m high, with a 2 m freeboard allowance. A spillway will allow non-contact water to be released downstream. Water will be pumped from the contact water pond to the plant for use in the process circuit.

The pond system capacity was based on the dry season storage requirement to supply the plant through the dry season after considering water immediately available in the watershed. The relative size of the ponds was determined by assessing the wet season accumulation of excess contact water which is the contact water generated that is in excess of the plant demand. For the base case, the maximum wet season storage requirement for contact water occurs around Year 5 and is approximately 50,000 m^3, according to the water balance. The maximum contact water produced by the 24-hour, 100-year storm event is approximately 100,000 m^3 (occurring in Year 6). The contact water pond has enough capacity to store both the maximum accumulated wet season contact water and the 24-hour, 100 year storm event. The water balance indicates that the average dry season storage requirement is 550,000 m^3. However, early in mine life when a relatively small amount of contact water is generated by the project, the dry season storage requirement exceeds the storage capacity of the Freshwater pond. During that time, it will be necessary to store some freshwater in the Contact Water Pond. The Contact Water Pond has been sized conservatively to ensure that storage capacity is available for contact water stormwater flow events during these times.

Site investigation of the planned water storage ponds will require additional field work. Geotechnical work will include test pits and drilling of the overlying unconsolidated material and bedrock. Permeability testing of the underlying bedrock will be undertaken to determine foundation grouting requirements. Geologic and structural mapping of the area will be used to guide the drilling program.

The pond system will be constructed prior to other facilities in order to serve a secondary purpose in reducing sediment load in runoff from construction areas during the construction phase. The system will include a low-level outlet to release water during the construction phase, and a spillway to release water during operations. Water will not be discharged from the contact pond during the operations phase.

Precipitation runoff internal to the plant facilities is a minor percentage of the total water supply requirement. Runoff in plant areas will be collected in small sumps and recycled to the Process Water Tank.

Precipitation and groundwater reporting to the mine pits is a minor source of water supply early in the mine life, but increases as the pits become wider and deeper. Pit water will be collected in sumps excavated into the pit floor. Sumps are configured as a simple box-cut into the un-mined bench below the active pit floor. Water will be pumped from the sumps to the plant via skid-mounted pumps and high density polyethylene (HDPE) pipelines.

Seepage, groundwater entering the floor of the valley, and precipitation runoff reporting to the toe of the Main Dump is expected to be minimal except following storm events. A sump will be located at the toe of the Main Dump, and water will be pumped from there to the plant via skid-mounted pumps and HDPE pipelines.

Water may be released from the non-contact water pond during operations when necessary. The pond system includes a bypass to allow stream flows in excess of water supply demands to be routed around the pond and released to the natural drainage downstream.

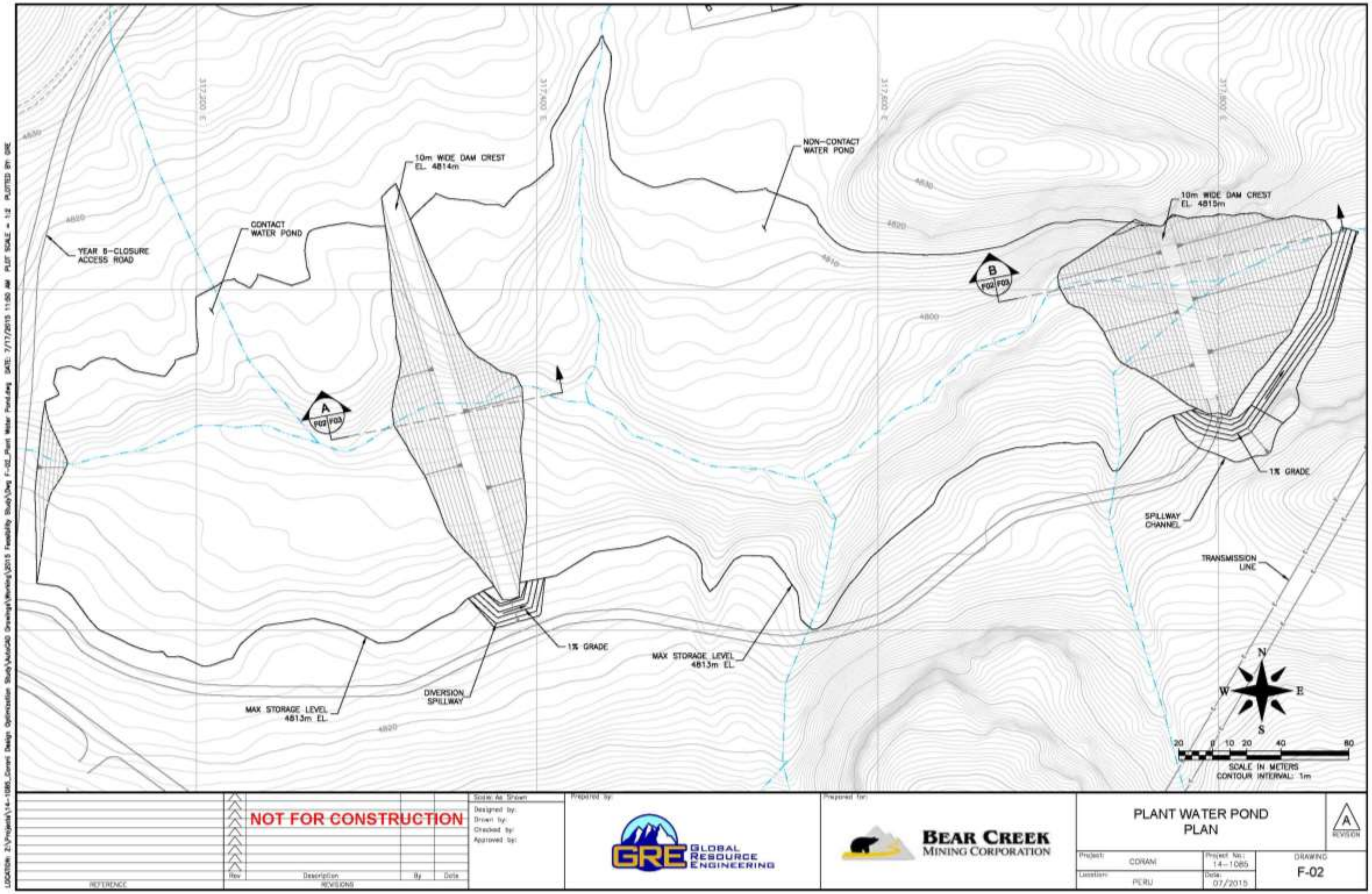


Figure 18-5: Contact Water Pond and Freshwater Pond

The Non-Contact Pond will receive and accumulate freshwater (non-contact) during the wet months to be used as makeup water when the supply of contact water is insufficient to meet makeup demand. Non-contact water from the Non-ContactPond will be pumped to the project Fresh/Fire Water Tank at an elevation of approximately 4,900 masl. Freshwater for plant use would be drawn from the upper nozzle on the tank to ensure an adequate reserve for a fire water supply. Freshwater will be also be drawn from the upper nozzle for the water treatment plant to supply potable water to sinks, eye-wash stations, and drinking fountains throughout the facilities. A nozzle at the bottom of the tank provides water for fire suppression (Section 18.3.2).

Fresh water for the Administration area near the main entrance will be provided by a water well to be installed in the valley west of the entrance and north of the mine pits. Water will be pumped to a fresh/fire water tank located at approximately 5,110 masl. Water from this tank will provide freshwater and fire suppression for the Administration area. A water treatment plant will be used to ensure that freshwater provided from this tank is potable and meets drinking water criteria.

The water supply for the residential camp site will come from the Imaginamayo River. Fresh water will be pumped to a storage tank that will be treated in the camp water treatment plant on a demand basis. Water for camp use will be drawn from the upper nozzle on the tank to maintain sufficient capacity in the lower part for fire suppression at the camp. Treated, potable water will be distributed by gravity to the various camp facilities: kitchen, laundry, recreation center, and dormitories. Water for fire suppression will be drawn from the lower nozzle.

18.3.3 Fire Protection

A freshwater pipeline will deliver up to 40 m^3/hr of freshwater from the Plant Water Pond to the project Fresh/Fire Water Tank, located at the project site at an elevation of approximately 4,875 masl. The Primary Crusher and mine services area (Truck Shop/Fuel Facility) are located elevations that do not permit gravity flow to provide sufficient pressure for fire suppression in these areas. A fire pump will be required to provide water at sufficient volume and pressure for fire protection to those areas.

Fresh water for plant use would be drawn from the upper of two nozzles located on the tank at an elevation high enough to ensure an adequate reserve for a fire water supply. From the lower nozzle, a buried firewater loop would supply fire water to hydrants located throughout the concentrator area, the ancillary buildings, and the laboratory.

Individual hand-held fire extinguishers will also be located throughout the offices and work areas. In addition a fire truck will be available to service the camp site as well as the main site.

The off-site Administration Building, Gatehouse and Security Office, located close to the Mine Access Road and entrance to the mine, will be provided with hand-held fire extinguishers. These extinguishers will be distributed throughout the offices and common areas in accordance with North American and Peruvian Fire Codes.

18.3.4 Sanitary Sewage

A packaged, activated-sludge wastewater treatment plant will treat sewage from the process facilities, laboratory, warehouse, plant maintenance building, truck shop, and tire shop. Treated effluent goes to an infiltration area next to the wastewater treatment plan.

The camp facilities, the administration building, and the security office and gatehouse located close to the Mine Access Road will also be served by a package, activated-sludge sewage treatment plant, and the treated water will be disposed through an adjacent leach field.

18.4 POWER SUPPLY AND POWER DISTRIBUTION

A new 138 kV power transmission line is necessary to provide power to the Corani project. A new power substation will connect with Power Transmission Line L-1013 (San Gabán II – San Rafael – Azángaro) as the power source. A new 138 kV power transmission line will be built to connect the Antapata substation to the Main Corani substation to be built near the Project's main process buildings. The proposed alignment for the 138 kV line (Figure 18-1) was provided by Promotora (2015). The transmission line route was selected based on using the route already provided by the Project's Mine Access Road.

Power will be distributed from the Main Corani Substation after a pair of transformers (one in use, one, standby) step down to the distribution voltage of 13.8 kV. Both overhead and underground power lines will be used to distribute power throughout the plant site and administration area. A 13.8 kV transmission line with a length of approximately 13 km will connect the mine site to the residential camp site.

18.4.1.1 New Antapata Substation

This substation will be connected to the San Gabán – Azángaro 138 kV Transmission Line at approximately 55 Km from the San Gabán Substation. The new Macusani substation will include the installation of three (3) switch yards at 138 kV.

18.4.1.2 Transmission Line

The link between the Macusani and Corani substations is made through a 138 kV – 60 Hz transmission line, approximately 37 km in length. The line includes:

- A metallic self-supporting lattice;
- Aluminum alloy conductors (AAAC) with a 300 mm^2 nominal section;
- Two ground wires (one steel galvanized wire and another OPGW-type wire with 16 optical fibers);
- Glass insulators with a 146 mm spacing and a 280 mm diameter;
- 380 mm leakage distance; and
- A 160 kN electromechanical rupture load.

The insulator chains in the suspension are made up of fifteen (15) insulators and the insulator chains in the grounding have sixteen (16) insulators and a reinforced concrete foundation, forming a set of four separate columns. Each column will made of corrugated steel-reinforced concrete with a formulation of f'c=210 kg/cm^2.

18.4.1.3 New Corani Substation

This substation will require the installation of a 138/13.8 kV transformer. The 138kV side will have a power switch and the power transformer, for an installation altitude of 5000 masl, will have an automatic regulation load of:

138 ±10x1%/13.8/4.16 kV - 48/48/16 MVA (ONAN) - 60/60//20 MVA (ONAF)

It will also have a YNyn0(d1) connection (or vector) group as well as a regulation board. It also includes six cells at 13.8 kV and an automatic capacitive reactive compensation bank of 5 MVAR.

Power will be distributed from the receiving substation at the Project site through underground duct banks to nearby major loads via local substations. Power distribution to all other areas, such as tailing water reclaim, the administrative services building located close to the concentrator, laboratory, fresh water pumping, crushing station and truck shop, and to the mine, will be via overhead 13.8 kV power lines.

18.5 COMMUNICATIONS SYSTEM

The project off-site telecommunications will be served by a fiber optic telecommunications link to a data center in Lima that will be underbuilt on the 138 kV transmission line from the Antapata substation. Telephone and data communications including voice, data and internet communications will be provided for the mine site and Macusani. The communications system will connect to a central communications center, which will include a telephone/fax PBX and network servers for email, internet and data services. Other network servers to manage site operations and for data storage will also be located in the central communications center, with the exception of the process servers which will be located at the processing facility. The mine site telephone system will link all essential areas of the site together, and through the satellite system, to outside of the project site.

The mine radio system will include one base station and a control-tower station at the mine from which all mining equipment and haul trucks will be dispatched and controlled, and a number of repeater stations will be installed. One station will provide coverage to the tailing area, and others are required to extend coverage throughout the mine site and to Macusani.

All vehicles will be equipped with radios and essential personnel will have hand-held radios. Key personnel will also be equipped with mobile telephones. Cellular phones will have coverage to Macusani as a safety precaution.

18.6 MINE WASTE DISPOSAL FACILITIES

The Mine Plan, described in detail in other sections of this report, was updated to incorporate updated core logging and mineralogy definition, and geometallurgical modeling better defining metal recoveries for each block. The Mine Plan also incorporated an additional objective of allowing mined portions of the open pits to be completed such that continued waste rock production from mining could be used as backfill for portions of the pits. The resulting Mine Plan reflects the completion of the Corani Este portion of the open pits prior to completion of the Main Corani and Corani Minas pit areas. Backfilling of Corani Este will occur relatively early in the Mine Plan, leaving only portions of the Corani Minas and Main pits requiring backfill at the end of active mining. Final backfilling of the Corani Minas and Main pits would be carried out as a continuation of mine operations using the mine truck-shovel fleet immediately following completion of ore mining. The objective of the backfilling of the pits is to develop a post-mining configuration in which all pit areas are backfilled below the pit rim, including the pit floor and the lower pit walls, in order to eliminate post-closure pit lakes. This change in concept from previous studies is expected to significantly reduce post-closure liability and potential water treatment requirements. In addition to environmental benefits, this more rigorous sequencing of the mine plan to incorporate pit backfilling as part of active mine operations also significantly reduces the previously proposed haul distances and uphill elevation change for haulage to the external dumps. Figure 18-6 shows the final backfilled pit and external dump configurations.

The Mine Plan produced a production schedule for waste rock based upon non-acid-generating (NAG) material versus potentially acid-generating (PAG) material. The geochemical characterization of the material suggests that the NAG will include all post-mineral tuff (PMT) and portions of the pre-mineral tuff. This production schedule was incorporated into the optimized design of the Main Dump.

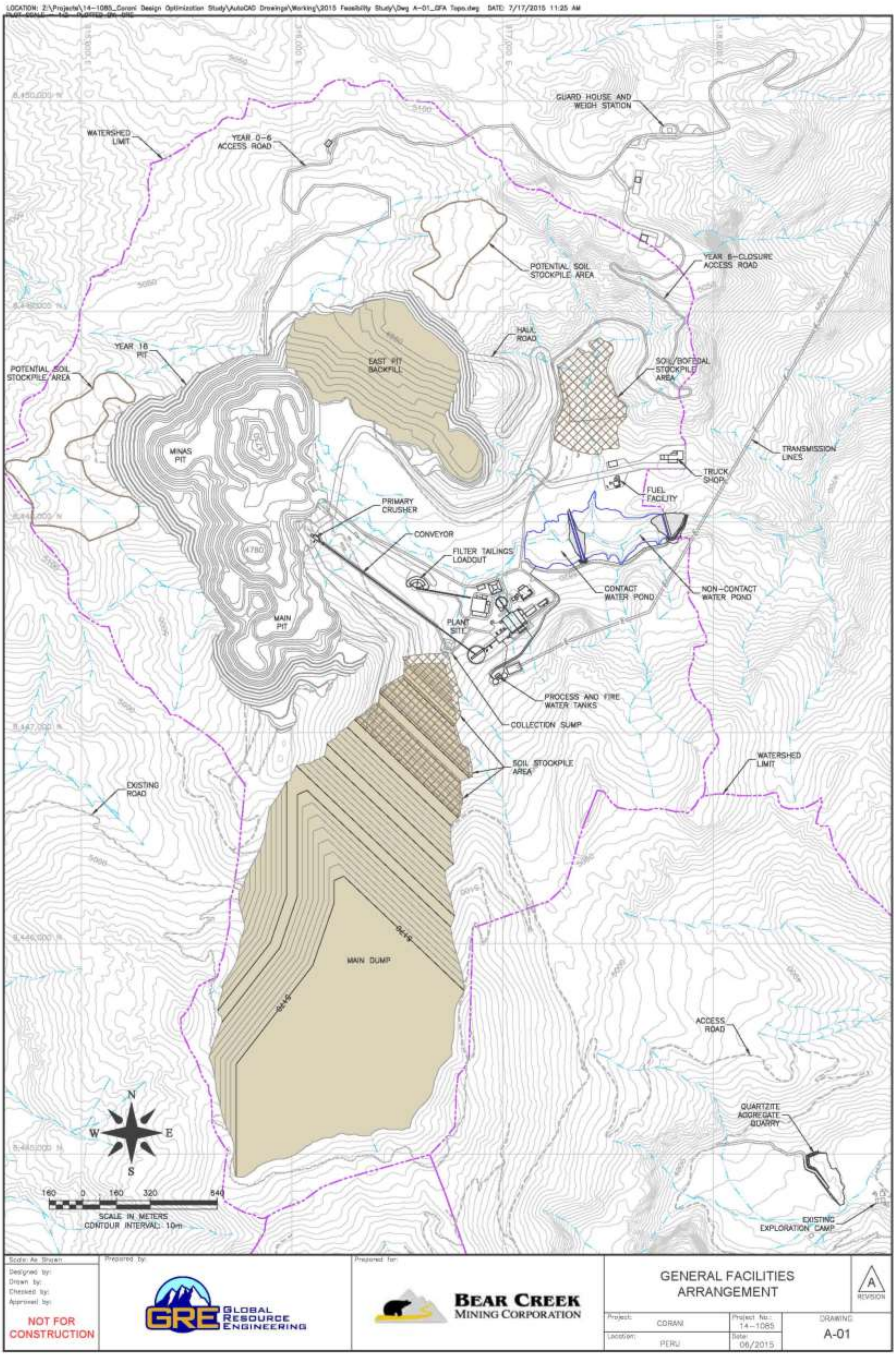


Figure 18-6: Mine and Process Area General Arrangement

18.6.1 Waste Rock and Tailings Management Facilities

With the modification of the Mine Plan to facilitate pit backfilling, the size and number of external waste rock dump facilities and tailings storage facilities was reduced compared to previous studies. The Main Dump has been redesigned to hold both waste rock and tailings as a co-disposal facility. In addition, the waste rock production schedule was utilized to advance the individual waste rock dump designs such that only NAG material would be placed in the valley floor of the Main Dump and a combination of PAG and NAG waste and tailings would be co-disposed within the remainder of the dump. The previously envisioned East Dump is no longer required under the optimized plan.

The Main Dump lies in a quebrada (gorge or valley) that has been influenced by the natural oxidation of sulfide materials exposed in the walls of the valley. This valley has limited development of natural soils and vegetation and, with the exception of some soft and/or weathered soils/rock types within the dump footprint, is well suited to dump development. The weathered and soft soils are planned to be stripped and utilized for later reclamation and some construction materials. The proposed Main Dump will cover a large percentage of the exposed natural sulfide-bearing rocks in the valley walls, and will reduce long-term oxidation of these materials. The scheduled production of NAG, PAG and co-deposition of tailings material allows the placement of NAG material on the external portions of the dump and to cover the PAG material within the center of the dump, thus alleviating potential long-term oxidation of the PAG materials.

With the intent of reducing the number of waste dumps and the costs associated with the more-distant previously proposed tailings facility, the ultimate capacity of the Main Dump will be approximately 180 million cubic meters of combined filtered tailings and waste rock.

Initial construction of the Main Dump will require removal of a portion of the available soils and unconsolidated materials within the facility's planned footprint. Some of the borrow materials will be used for construction, while the main use for this material will be for cover materials during reclamation and closure at the end of the project life. As part of the development process, an underdrain will be constructed at the base of the dump. This will allow for any seepage to flow to a collection sump at the toe of the dump. Seepage collected at the toe will be recycled to the plant during operations, and treated as necessary following closure.

The first lift of material over the underdrain will consist of NAG material. Subsequent placement of materials will be in lifts of waste rock alternating with thin lifts of filtered tailings. Rock and filtered tailings will be placed in alternating layers up to the top of the dump. Alternating PAG waste rock and tailings lifts will be positioned in the central portion of the dump, and these will be encapsulated by NAG waste rock on all sides. Only NAG waste rock will be placed on the top, bottom, and external face of the dump.

The Main Dump will be used from year -1 to year 7. Waste and tailings will then be placed in Este as pit backfill for two years, followed by the re-activation of the Main Dump, which will then be used through the end of the mine life. At the end of active mining, a portion of the material from the Main Dump will be used to backfill the bottom of the Corani Main and Corani Minas pits to an elevation adequate to prevent the formation of a pit lake.

18.6.2 Low-Grade Ore Stockpile

Previous mining schedules included provision for a low-grade ore stockpile. The current optimized plan does not require a low-grade stockpile, further reducing the project footprint. A stockpile plan can be reintroduced given more favorable metal pricing.

19 MARKET STUDIES AND CONTRACTS

19.1 CONCENTRATE MARKETING

19.1.1 Markets

The project will produce a lead concentrate, containing the majority of the recovered silver as well as a separate zinc concentrate. The high silver grade of the lead concentrate will make it a desirable concentrate for smelters. The concentrates will be sold and shipped to Asian smelters.

19.1.2 Contracts

There are no established contracts for the sale of concentrate currently in place for this project.

19.1.3 Concentrate Transport Logistics

Concentrate will be bagged in two-tonne supersacks via intermodal containers or flatbed trailers from the mine site to the port of Matarani at an estimated charge of US$40.00/wmt.

19.1.3.1 Concentrate Transport Insurance

Insurance will be applied to the provisional invoice value of the concentrate to cover transport from the mine site to the smelter.

19.1.3.2 Owner's Representation

An Owner's representation will be applied to the provisional invoice value of the concentrate to cover attendance during unloading at the smelter, supervising the taking of samples for assaying, and determining moisture content.

19.1.3.3 Transportation Options

The decision was made to transport concentrate by container trucks to the Peruvian Port of Matarini (See Figure 19-1). The container trucks will meet all required environmental regulations and are fully enclosed. The first five years of lead and zinc concentrate production will be approximately 110,000 mtpy and 83,000 mtpy, respectively dropping down to an average of 65,000 mtpy for lead concentrate and 39,000 mtpy for zinc concentrate for the remaining 13 years of the mine life.

For this study, it is assumed that the concentrate will be sold to the Asian market. The transportation cost is estimated at US$90.00 per ton for shipping to the smelter by ocean freight.

The total freight cost including overland freight plus ocean freight and handling fees for concentrates shipped from the Corani plant to the smelter is US$130.00/ wet metric ton.

The map in Figure 19-1 shows the project location, which also demonstrates the distance from the project site to the Matarani port.

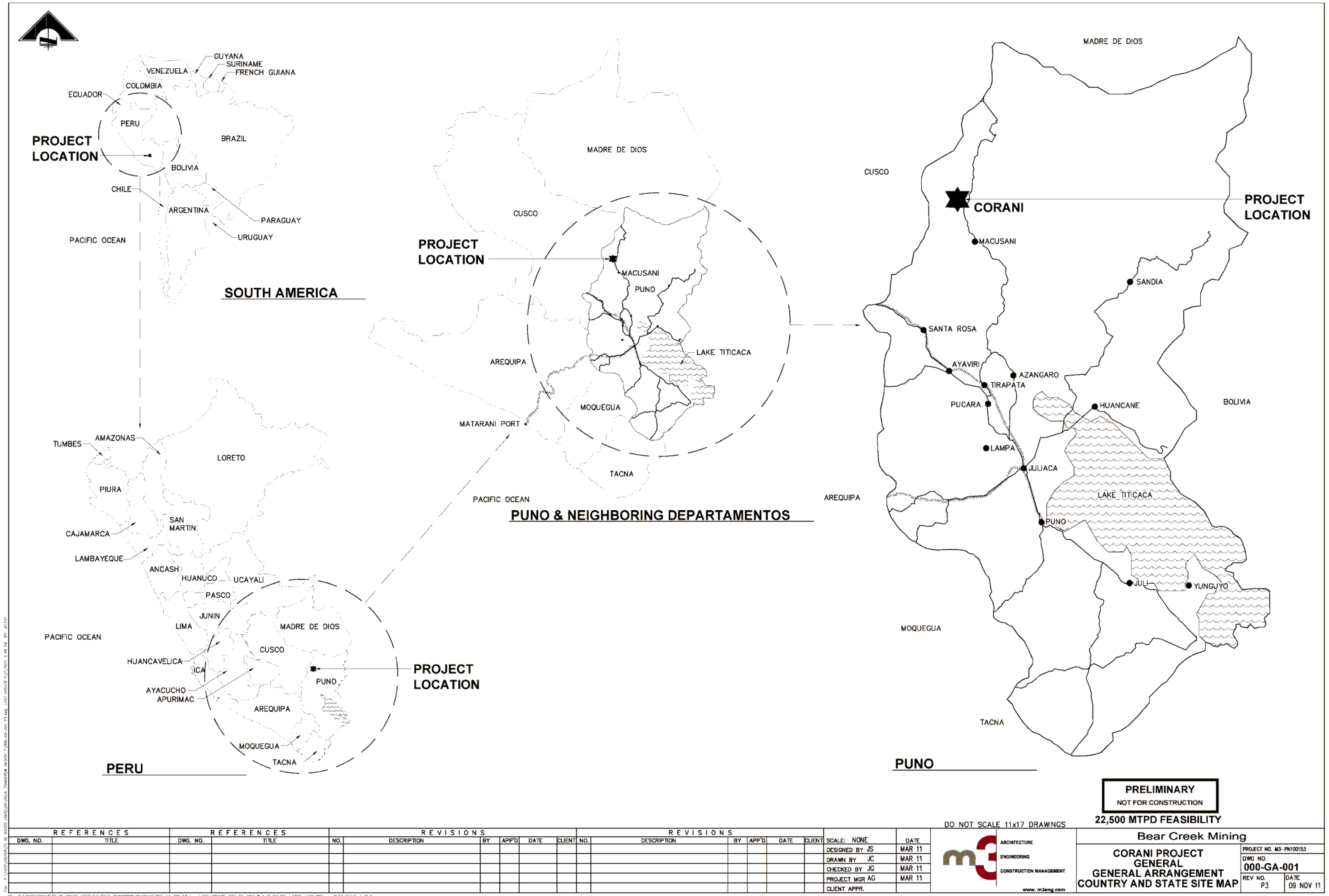


Figure 19-1: Corani Project Country and State Site Map

19.1.4 Smelter Terms

Smelter terms and penalties were supported by an independent market analysis (Andes Mining Reseach, 2014). The data from this study were used to evaluate the revenue, charges, premiums, and penalties that are presented in detail below.

19.1.4.1 Sale of Lead and Zinc Concentrates

Every smelter has different rates for impurities depending upon the normal feed. Higher levels of impurities will decrease the value of concentrates delivered to the smelter.

19.1.4.2 Zinc Treatment Cost and Premiums

Spot zinc treatment charges are half annual terms. Zinc metal premiums are being maintained despite the recent drop in zinc prices.

19.1.4.3 Lead Treatment Cost and Premiums

The price of lead has been fairly constant over the last several years and appears to be stable for the next few years. Table 19-1 shows the assays for materials found in each type of concentrate for this project.

Table 19-1: Concentrate Assays

Material	Units	Amount in Zinc Concentrate	Amount in Lead Concentrate
Ag	g/t	331.0	1815
Pb	%	4.22	58.9
Zn	%	53.8	5.0
Cu	%	0.41	1.60
Au	g/t	0.25	0.33
S	%	30.1	20.8
C(t)	%	0.18	1.02
Cl	g/t	34	<10
Hg	g/t	55.7	16.6
As	g/t	520	1215
Ba	g/t	338	981
Ca	g/t	1321	460
Cd	g/t	3499	1271
Fe	%	4.5	7.5
Sb	%	0.22	0.82
SiO_2	%	5.2	3.6

Additional test work will be performed to investigate lowering the quantities of some penalty elements; for example, reducing the amount of SiO_2 in the Zinc concentrate as well as As and Sb in both concentrates. Lowering these impurities in the final concentrate will increase the value of the concentrate to potential smelters.

19.1.5 Sale of Concentrates

19.1.5.1 Zinc Concentrates

The Pacific Rim Benchmark Terms for zinc concentrates are as follows:

- **Payable Metals**
  - Zinc: 85 percent (minimum deduction of 8.0 units)
  - Silver: Deduct 3.50 ounces per Dry Metric Tonne (DMT) and pay for 70% of the balance

- **Treatment Charge**
  - US$229/DMT Cost Insurance & Freight Free Out (CIF FO) Main Asian Ports basis a Zinc price of US$2,500 per MT and shall be increased / decreased for each US$1.00/MT off variance above or below US$2,500 per MT as follows:
    - Base T/C-$229.00 @$2,500-
    - Scale US$/MT

- **Zinc Price**
  - above $3,500 + 0
  - $3,000 - 3,500 + 3.00 US cents for each US$/MT
  - $2,500 - 3,000 + 6.00 US cents for each US$/MT
  - $2,500 - 2,000 - 4.00 US cents for each US$/MT
  - $2,000 - 1,500 - 2.00 US cents for each US$/MT
  - below $1,500- 0

- **Penalties**
  - Fe: US$1.50 for each 1% over 8%
  - As: US$2.00 for each 0.1% over 0.1%
  - SiO_2: US$0.50 for each 1% over 0.5% >4.0% may be unacceptable
  - Hg: US$0.30 for each 10 ppm > 30 ppm < 100 ppm plus US$0.50 for each 10 ppm > 100 ppm
  - Cd: US$ $1.00 for each 0.1% > 0.4%

19.1.5.2 Lead Concentrates

The Pacific Rim Benchmark Terms for lead concentrates are as follows:

- **Payable Metals**
  - Lead: 95 percent
  - Silver: 95 percent (minimum deduction 50 grams per DMT)

- **Treatment Charge**
  - US$175 per DMT CIF FO
  - Refining Charge – Silver
  - $0.50 per payable oz.

- **Penalties**
  - As – 0.50% free; US$2.00 per DMT for every 0.10% above 0.50%
  - Sb – 0.50% free; US$2.00 per DMT for every 0.10% above 0.50%
  - Bi – 0.50% free; US$1.00 per DMT for every 0.01% above 0.50%
  - Hg – 100 ppm free; US$1.00 per DMT for every 10 ppm above 100 ppm
  - Zn – 8% free, US$1.00 for every 1% above 8%.

19.1.5.3 Metal Prices for Study

The metals prices used for this study were as listed in Table 19-2.

Table 19-2: Metals Prices Used for Study

Material	
Zinc	US$1.00/lb
Lead	US$0.95/lb
Silver	US$20/troy oz

Penalty element assay values were based on chemical analyzes of concentrates produced during the composite locked-cycle test performed by SGS.

20 ENVIROMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Principal environmental risks associated with this type of project fall into these categories:

- Potential risks to air quality from dust;
- Potential degradation of surface and groundwater quality;
- Potential changes to the volume of surface and groundwater;
- Visual impacts due to the creation of pits, mine waste disposal facilities, roads, and other mine workings; and
- Permanent changes to land use resulting from mining activities.

The existing Environmental and Social Impact Assessment (ESIA) quantifies the magnitude, extent, and mitigation of risks and potential impacts related to the old project configuration which included a conventional wet tailings facility and related water supply infrastructure. The new project configuration developed for the current study will be incorporated into a modification of the ESIA to be completed subsequent to the publication of this report. Project optimization performed for the current study resulted in a reduced project footprint, a reduction in water consumption, and other changes which are anticipated to reduce environmental impacts associated with project development relative to the previous study. In a number of cases, the development of the Project is anticipated to improve existing environmental conditions.

Peruvian Law 28090 regulates the obligations and procedures mine owners must follow in relation to mine closure, and requires that a mine closure plan be approved and financial guarantee for the cost of implementation be established. The plan must describe the rehabilitation methods and their costs for the operation, closure, and post-closure phases. The plan must allow for progressive closure, with mine owners reporting semiannually to the Ministry of Energy and Mines on progress with the implementation of the approved plan.

As required under Peruvian regulations, a preliminary closure plan was submitted and later approved in September 2014 (Walsh, 2014). A revised closure plan based on the current project configuration will be submitted to Peruvian regulators following modification of the ESIA .

The development of closure concepts for this study has considered International Finance (IFC) guidelines and industry standards in addition to Peruvian regulatory requirements. The general approach to mine reclamation and closure developed at this time is described below. The estimated cost (based on the Preliminary Closure Plan) has been incorporated into the overall cost estimate for the Project.

The sections below present the following:

- A summary of existing studies;
- Known permitting considerations;
- A brief description of the strategies for environmental management during operations, reclamation and closure;
- Socioeconomic and community considerations.

20.1 ENVIRONMENTAL BASELINE STUDIES

Environmental sampling has been ongoing since July 2009. Extensive additional site characterization was conducted in 2011 and the first half of 2012; the results of these studies were presented in the ESIA (AMEC, 2012). The following sections summarize the existing environmental data.

20.1.1 Summary of Air, Noise, Groundwater and Surface Water Studies

Below is a summary of the results from the currently-published baseline studies for air, noise, and water:

- The air quality results were below the maximum permissible limits set by the national environmental standard for air, reflecting the absence of significant airborne pollutant-generating activities in the zone.
- Noise measurements were taken near populated areas. The results were below the maximum thresholds specified in the national environmental standard for residential zones.
- Groundwater appears to be located in shallow aquifers comprised of alluvial, glacial, and aeolian deposits that lie over low-permeability basement rocks. Results from the site investigation indicated that the shallow aquifers have high storage but moderate to low hydraulic conductivity. Little evidence exists for a conductive and extensive hard-rock aquifer. In general, it appears that groundwater resources in the Project area are not sufficiently large to be useful for agricultural, domestic, or industrial use. However, the protection of the groundwater resources from impact is part of the focus of the ESIA and future environmental planning.
- Surface water samples exhibited highly to slightly acidic characteristics during sampling events conducted throughout the year. These acidic conditions are related to naturally-occurring oxidation of mineralized rocks exposed at the site and from areas previously disturbed by historic mining activities. Several metal concentrations exceeded the national environmental water standards. Similarly, some metal concentrations measured in sediments exceeded Canadian Environmental Quality Guidelines. As a result, the major drainages leaving the Project area do not currently meet national environmental water standards.

20.1.2 Summary of Biological Studies

The biological baseline study describes the ecosystem of the site and the species abundance, richness, biodiversity, and endemism. The species present on the site have been cross-referenced with threatened species lists (international and national). Of particular note is that:

- A number of flora are included on the national threatened species list but not on international conservation lists, including umbellifer, certain daisy species, and *Valerina nivalis*.
- Three vegetation species found are considered endemic to Peru, of which *Nototriche pelicea* is considered endemic to the Department of Puno.
- A number of mammal species identified in the biological baseline study are included in both national and/or international threatened species lists, including *Hippocamelus antisensis* (north Andean deer), *Leopardus colocolo* (pampus cat), *Puma concolor* (puma) and *Vicugna vicugna* (vicuña). However only *Vicugna* was observed in the field. The remaining species were recorded based on sightings from members of the local community; therefore, their presence on the site is not confirmed.

20.1.3 Summary of Geochemical Studies

A geochemical waste characterization program was developed to assess the acid rock drainage (ARD) behavior of, and potential leaching of contained metals from, all mine wastes associated with the Project. This program included static tests, LECO Furnace total sulfur and total carbon assays, and onsite and laboratory kinetic cell tests, as presented in the ESIA (AMEC, 2012). These were combined with geologic and metallurgical characterization of lithologies and material types. The conclusions from this work included the following:

- The geochemistry of the waste rock will be dominated by certain mineralization types; in particular, mineralized lithic tuff with fine black sulphides (FBS) and mineralized tuff with pyrite and marcasite (PM).
- Whole rock analysis of waste rock samples indicated that several metals of environmental and processing concern exist at high levels; synthetic precipitation leaching procedure testing suggested that many of these metals are readily leachable.
- The kinetic tests showed that many waste types were acid generating, though the behavior among certain mineralization types was mixed.
- The ABA indicator testing of the tailings suggested that the tailings would be potentially acid generating (PAG) material due to the presence of residual sulfides.

Additional geochemical humidity cell kinetic tests were inititated in 2014 by Amphos 21 (Amphos 21 Consulting Peru, SAC, 2014). Initial results from these tests confirmed previous kinetic cell testing by demonstrating that the PMT is non-acid generating, and that sulfidized waste often produces acidic leachate.

20.2 PERMITTING CONSIDERATIONS

Refer to Section 4.4 for an explanation of the permits required to execute the Project.

20.3 ENVIRONMENTAL MANAGEMENT DURING OPERATIONS, RECLAMATION, AND CLOSURE

The project design approach considers implementation of best management practices and a sustainable approach to project development. The project development approach has incorporated the ability to close or reclaim project facilities progressively throughout the project life. A more detailed description of the potential impacts associated with the project development can be found in the ESIA and Closure Plan studies.

20.3.1 Environmental Management Objectives

The operational and preliminary closure plan environmental management objectives are to identify potential and viable measures to mitigate environmental impacts that can be implemented during the operational, reclamation, closure, and post-closure periods. These measures are intended to alleviate potential long-term impacts from mining operations and to minimize long-term liability. Current conditions at the Project site show a degree of degradation of water quality in the general vicinity of the mineralized zones. This is primarily due to naturally occurring oxidation of sulfide-bearing materials, which has resulted in a depression of pH and an increase in dissolved metals and salts in surface waters leaving the Project area. In addition, historic mining activity within the main Corani basin has resulted in the presence of underground mine workings and historic mine tailings which are a significant source of ARD.

The measures to be applied for reclamation and closure of the Project are intended to return the receiving environment to a condition that is, at worst, equal to the measured baseline conditions, and to the extent practical, better than these conditions. The closure measures are not intended to change the naturally occurring conditions at the site, nor are they intended to mitigate the effects of historic mining within the project area, apart from where future mining activity can be used to implement mitigation measures in the course of the proposed mining, processing, and mine waste management defined for the Project. Historic mine areas will be closed/mined out in the pit areas and areas directly in the footprint of the Main Dump. Historic mine workings that are not directly affected by mining activity, but are still within the catchment area of the mine's water management facilities and are contributing ARD to the facilities will also be closed. Moreover, it should be noted that although a large number of the historic mining facilities contributing to ARD will be removed during Corani mine operations and closure, a significant portion of the undisturbed bedrock at the site is naturally acid-generating, and will continue to produce ARD during operations and closure.

The following activities performed during operations, reclamation and closure are intended to mitigate and minimize generation of ARD:

- Encapsulation of PAG tailings and waste rock within the Main Dump;
- Extraction of legacy mine waste and disposal within PAG encapsulation cells inside the Main Dump;
- Saturation and encapsulation of PAG tailings and waste rock within the pit backfill areas;
- Concurrent placement of reclamation covers over the Main Dump and the Este backfill during operations;
- Segregation of contact waters from non-contact waters and consumption of contact water, and a portion of naturally acidified runoff from the watershed, in the flotation plant during operations;
- Backfilling of the Minas and Main pits to prevent formation of pit lakes and placement of an ET cover.

In the following sections, the overall site conditions that affect the selection and implementation of the reclamation and closure measures are identified. The reclamation and closure considerations for individual project components and facilities are then presented. The general items considered in estimation of costs for the reclamation, closure, and post-closure periods are also identified. A large proportion of the reclamation and closure measures will be completed as part of operation of the mine. Cost estimation for reclamation and closure includes only those activities required following the end of mining and mineral processing. For example, the cost for pit backfilling does not include the cost for backfilling that occurs during the course of normal operations which is covered under mining costs. Additional information regarding the cost estimation for specific items is provided in the relevant sections of this document.

20.3.2 General Site Conditions Relevant to Environmental Management

As described in Section 7 of this document, mineralization at Corani consisted of multiple phases of epithermal emplacement of sulfide minerals, and the emplacement of salts related to the hydrothermal fluids associated with the mineralizing events. The sulfide minerals have the potential to become oxidized in response to exposure to oxygen and moisture. Glacial erosion has removed surficial weathered zones and unconsolidated material, in many places exposing fresh surfaces of sulfide-bearing bedrock. Consequently, naturally occurring growth media are limited or absent within the project area. Key areas without soils capable of sustaining vegetation include:

- The valley that will contain the Main Dump and portions of the plant complex;
- Large portions of the mountain slopes in and around orebody; and
- Higher elevations of the project site and areas with exposed bedrock.

The areas not capable of sustaining vegetation are readily apparent from the color aerial photo shown in Figure 20-1.

The characteristics of the natural environment of the Project area have limited the impact of sulfide mineral oxidation and metal leaching from the mineralized zones. The climate and altitude of the Project area are not compatible with the rapid establishment of vegetation as a possible reclamation measure, and the ability to establish vegetation has been limited by the acidic soil conditions that naturally exist in the majority of the Project area. An extensive blanket of PMT overlies portions of the mineralized zone. This material is classified as being geochemically inert, due to the absence of sulfide mineralization in the rock. A large volume of PMT will be mined during the development of the open pits and used as inert cover material and engineered fill.

The reclamation and closure measures identified in the following subsections have been developed to minimize post-closure management requirements for the areas of disturbance related to mining activities. However, as described further below, it is anticipated that some long-term post-closure management and monitoring will be required for an extended period following mine reclamation and closure.

Figure 20-1: Aerial Photograph of the Corani Site, Current Conditions

20.3.3 Project Components and Facilities

The following sections describe the management plans for specific project components and facilities. Additional design details can be found in Section 18. The main project components and facilities with environmental management and/or reclamation considerations include the following:

- The Este, Minas and Main Corani open pit areas;
- Waste rock and tailings co-disposal areas including the Main Dump and in-pit backfills;
- The plant facilities, crushing facilities, tailings dewatering plant, and related infrastructure;
- The surface water management infrastructure and Plant Water Pond;
- Soil stockpiles areas;

It is not the intent to identify all aspects of environmental management, reclamation and closure measures at this time. Rather, only the major aspects with significant cost implications are considered.

The project configuration at the end of mining, just prior to closure, is shown in Figure 18-6. Figure 20-2 shows the preliminary closure plan concepts described in the following sections.

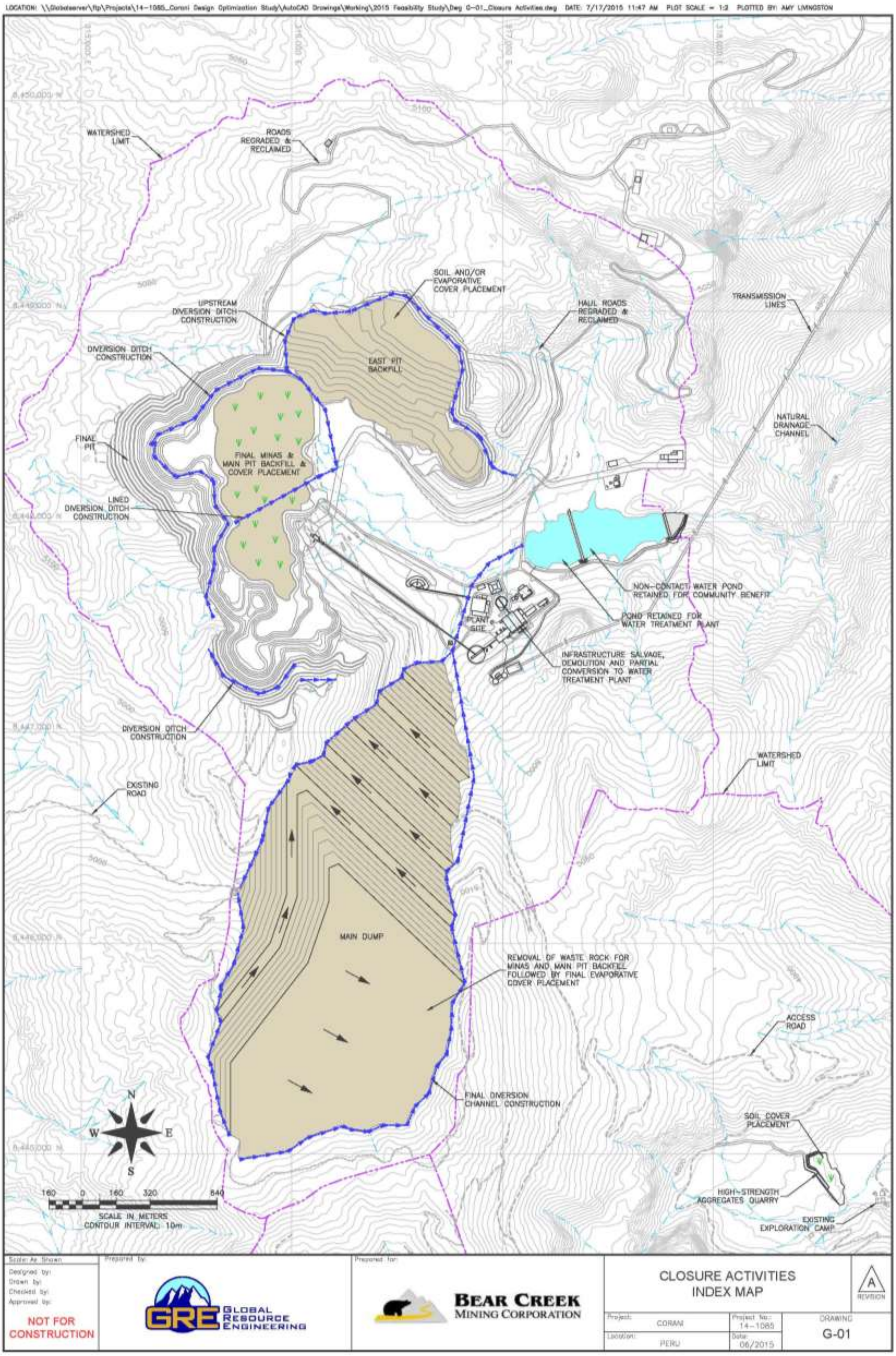


Figure 20-2: Preliminary Closure Plan Concepts

20.3.4 Open Pits

Mining of the open pit will occur over a period of approximately 18 years. Upon completion of mining, the Este, Minas, and Main pit areas will form a contiguous horseshoe-shaped excavation around the east, north and west sides of the Corani valley. Mining will be performed by developing a sequence of high-angle pit walls offset by benches to form an overall pit slope of approximately 1H:1V. This pit wall configuration is anticipated to be stable based upon geotechnical work carried out to support design of the pit walls (McDonald, 2012).

20.3.4.1 Bodedal Soils

The southern pit wall will intersect the bofedal to the south (the "lower bofedal"), and a portion of the bordering unconsolidated bofedal sediments will be removed in stages as the pit footprint grows. Excavation of the lower bofedal will result in removal of a total thickness of approximately 30 m of unconsolidated bofedal soil material. It will be necessary to dewater the bofedal soils prior to their excavation. Dewatering will be done in a progressive, staged manner during excavation, using a network of successively deeper dewatering trenches.

A significant part of the surface of the bofedal adjacent to the mine area contains a deposit of historic, ARD-generating tailings. This material will be run through the process plant, and the resulting tailings product will be disposed in the Main Dump or pit backfill along with tailings from virgin ore. The area where the historic tailings are excavated will be reclaimed using bofedal soils removed from the mine pit area. Additionally, the historic underground workings located in the Corani Minas pit area will be mined out. Removal of the historic workings and the tailings material will eliminate a major existing source of ARD and dissolved metals in the pit area.

A small region of the northern pit wall approaches the bofedal to the north of the pit (the “upper bofedal”). The currently planned pit limit extends approximately 20 m into the upper bofedal area. It is assume that during detailed design the pit would be optimized to avoid intersecting the upper bofedal, or that detailed evaluation of the pit slopes in this area would otherwise be performed in subsequent studies.

Localized stockpiles of bofedal material will be established for use in reclamation and closure activities. Segregation of the materials will be performed to separate organic material from the rest of the excavated soils and unconsolidated materials. Collection and storage of these materials will be maximized to the extent practical.

20.3.4.2 Pit Water Management

Several natural drainages will be tributary to the fully developed pit area, and these drainages will continue to contribute surface runoff toward the pit. During operations, this water will be diverted around the pit or routed through the pit by pipeline to natural drainages downstream of the pit. Runoff generated within the footprint of the pits and groundwater inflow to the pit will be collected in sumps and pumped to the process plant for use as make-up water.

The goal of the surface water management system at closure will be to reduce the volume of water requiring treatment by separating water that has been in contact with potentially acid generating material from water that has been in contact with only non-acid generating material. Water that has been in contact with potentially acid generating material (runoff from exposed PAG pit slopes and seepage emerging from the pit backfill) will be piped to a water treatment plant located at the former plant site. This water will be treated as necessary before being released downstream. Water from the areas up-gradient of the pit will be diverted to drop structures that will convey the water over the exposed pit wall, and then routed over the pit backfill areas in lined channels before release to natural drainages downstream. Diversions will be built to minimize runoff from non-acid generating portions of the pit wall. It is anticipated that this water will not require treatment and therefore will be routed around acid-generating portions of the pit wall and released downstream.

20.3.4.3 Backfill

The Corani pits will be backfilled for the purpose of storing waste rock and tailings, and also to prevent formation of pit lakes which could contain acidic water with elevated dissolved metal and salt constituents. Mine tailings and portions of the mineralized waste rock are Potentially Acid Generating (PAG). Similarly, the floor of the pit terminates in mineralized rock and therefore has potential for sulfide mineral oxidation and acid generation. Concurrent reclamation and closure of the mine pit will be performed as feasible during mine operations and completed at closure.

Mining of the Este pit area will be complete by the end of year 6. Backfilling of the Este pit will take place over the following three years using tailings and waste rock generated by ongoing mining of the Minas and Main pits during that time. Tailings and waste rock will be placed in designated backfill zones. After backfilling has been completed, an engineered soil cover will be installed over the backfilled area. The cover will consist of a layer of non-acid generating (NAG) material overlain by natural bofedal soils and will limit the amount of precipitation water that can enter the backfill thereafter.

Mining in both Minas and Main pits prevents backfilling of this pit area until the end of the mine life. In the current mining sequence, the Main and Minas pits begin as distinct pit areas but merge to form a mostly contiguous pit area late in the mine life. Backfilling of the Minas/Main pit area will occur over the course of several months in year 19 directly following completion of mining. The area will be backfilled by importing waste rock and tailings temporarily placed in the Main Dump. Based upon the current mine plan, an estimated 14 million m^3 of material will be required to backfill the final mining area sufficiently to satisfy reclamation and closure needs. Backfill will be placed to a level high enough to ensure that the final backfilled surface remains above the water table after stabilization occurs during the post-closure period. After this material has been placed, available water will be diverted to artificially recharge the backfill, saturating the material as quickly as possible, and limiting the time during which the backfill and the pit walls are exposed to atmospheric oxygen. Artificial recharge of the Este backfill is not required due to the low hydraulic conductivity of the tailings backfill.

The final Minas/Main backfill surface will be graded to resemble the topography of the adjoining lower bofedal area, and to merge into that topography. Stockpiled bofedal sediments will be used to cover the backfill, with the intention of promoting gradual natural establishment of a healthy bofedal environment that is an extension of the lower bofedal. A preliminary model of the cover was developed using a numerical model to evaluate the performance of various engineered cover configurations. Based upon the model results, direct infiltration through the cover is expected to be reduced to a minimal level following cover installation.

For the exposed backfill areas, and pit floor areas not otherwise covered by backfill material, a layer of inert NAG material will be placed to form a cover over the PAG material. The NAG material will be selected to have a grain size distribution engineered to function as a store-and-release evapotranspiration cover. In addition, a layer of bofedal material will be placed over the NAG layer. For the Minas/Main cover, this will allow gradual establishment of a wetland condition similar to the existing natural bofedal in the lower bofedal valley bottom.

A hydrogeological model was developed to simulate the groundwater flow through the backfill after flooding of the Minas/Main backfill and installation of the surface water management and cover systems. The model results indicate a relatively small seepage rate emerging from the backfill. Seepage emerging from the backfill will be collected and routed to the treatment plant as necessary.

The effectiveness of the progressive reclamation and closure activities will be monitored and optimized over the course of the project. At the same time, the need for supplemental reclamation and closure activities will be assessed, and any corrective measures determined to be appropriate will be implemented.

20.3.5 Main Dump

The Main Dump will be developed in the valley southwest of the pits. This dump will be the repository for all waste rock and tailings not backfilled in the pits. Waste rock will be segregated throughout the time of mining activity based on the potential-acid-generating (PAG) and non-acid-generating (NAG) nature of the material. Non-acid-generating material will be placed as the foundation and outer shell of the Main Dump. PAG waste rock and filtered tailings will be encapsulated within the central core of the dump.

As the Main Dump is developed, soil material, unsuitable as a foundation and necessary for the post-mining dump cover, will be stripped prior to placement of mine waste. Most of this material will be stockpiled north of the Plant Water Pond, or directly placed on final dump surfaces as part of concurrent reclamation.

During operations, dump slopes will be progressively reclaimed with the placement of previously stockpiled natural soils from the base of the dump. Runoff from these areas will be routed around the active dump areas and released downstream to natural drainages. Following the placement of the reclamation cover, runoff from the reclaimed surface will be directed to toe ditches and routed beyond the facility. At closure any seepage from beneath the toe of the dump will be collected and treated as necessary prior to release.

The current condition of the ground in Quebrada Muerta beneath the Main Dump is un-vegetated and sterile. The goal of the closure plan is to establish conditions similar to those that existed prior to mining activities, and therefore, does not include the establishment of vegetation on the Main Dump. The closure plan includes erosion control methods in place of vegetation.

20.3.6 Plant Facilities & Related Infrastructure

During operations, runoff generated in the area of the plant footprint is considered contact water and will be diverted to a sump and pumped to the process plant for use as makeup water. As part of reclamation and closure activities, a portion of the mineral process plant will be converted to a water treatment plant. Water from the surface water management system will be routed to the plant, and treated as necessary prior to release to the receiving environment with a high-density sludge (HDS) treatment system.

At the end of the operating life of the plant, with the exception of those components to be utilized in the post-closure water treatment system, the plant will be decommissioned and demolished. Where viable, equipment will be salvaged, sold, or transported offsite as scrap material. A small amount of infrastructure will be left in place for post-closure use. This would include a water treatment and equipment storage area, the office building, and equipment storage required for the maintenance and monitoring of post-closure site conditions. All other structures will be demolished, and the demolition debris buried in an on-site landfill. This would include the majority of the plant buildings, truck shop, explosives magazine, crushing system, warehouses, reagent storage areas, etc. The main power line utilized during operations is retained for the ARD treatment plant. All major pipelines will be either salvaged or, if buried, left in place.

Concrete foundations for the plant will be broken up and buried in place in most areas. The degree to which they are broken up will be determined by the ability to create a natural-appearing post-mining topography with a natural drainage pattern. Leaving foundations in place may be considered where this can be accomplished without breakage. A suitable soil cover will be placed over these areas to facilitate the function and appearance of natural ground.

Once the off-site man camp is no longer required, the camp will be converted to an alternative beneficial use by a future custodian and/or local community representative. The main access road and haul roads left at the end of mining operations that will be required to provide access to the site, and access to locations necessary for maintenance and monitoring of the project areas, will be reduced in size to that of similar local roads in the project region. A portion of the internal roads will be removed by ripping, and reclaimed to conditions similar to the surrounding area; the remainder of the roads will be converted to community use.

20.3.6.1 Surface Water Management & Plant Water Supply Pond

During Project development and operations, a surface water management system will be developed to route runoff from areas of disturbance, groundwater collected in the pit, and seepage from the Main Dump to the process plant for use as makeup water. Contact water exceeding the immediate consumptive capacity, and water required for dry season operation of the plant, will be stored in separate compartments of the Plant Water Pond, located approximately one kilometer east of the plant. The capacity of the collection pond is approximately 750,000 cubic meters, and will consist of two separate cells: a non-contact water cell (in the downstream portion, with a volume of 550,000 cubic meters) and a contact water cell (in the upstream portion, with a volume of 200,000 cubic meters). Environmental baseflow to the community will be maintained during Project development subject to the natural availability of flow in the catchment area. Concepts related to the surface water management plan for operations are presented in more detail in Section 18.

The contact water cell of the plant water pond will be used during closure as a flow equalization pond for the water treatment plant. Use of the pond may also be necessary in the first year of closure to store ARD-impacted water while the process plant is converted into an ARD treatment plant.

At closure, the non-contact pond could constitute a potential resource for the downstream communities. Transfer of ownership and operation of the pond to a group of the local communities or government organizations would be performed following the implementation of reclamation and closure measures.

Upon closure, permanent water conveyance structures will be retrofitted as required for minimum maintenance. This may include the installation of new riprap channels and passive sediment control structures.

20.3.6.2 Solid Stockpile Areas and Regrading Plan

Due to the fact that soil stockpile areas cover existing vegetation and ground surface, they are a form of low-impact land disturbance. Once the stockpiled material has been removed for use in closure covers, the foundation of the soil stockpiles will be ripped to relieve compaction, and then revegetated. Sediment-control Best Management Practices (BMPs) will be utilized to minimize erosion.

After demolition, the mine will have areas of impacted soil for the following reasons:

- Compacted soil from vehicle traffic;
- Potentially contaminated soil from operations or closure activities; and
- Areas that were beneath prior structures that require reclamation.

Elements of the generalized regrading and re-vegetation plan include:

- All visibly contaminated soils will be excavated and treated in an on-site soil land farm;
- All compacted soils will be tilled with a disc to relieve soil compaction and improve drainage;
- Impacted areas will receive a soil cover, where cover existed prior to development;

- Impacted areas will be revegetated with native species, where capable of supporting vegetation; and
- Erosion-control BMPs will be applied to minimize erosion until revegetation is established.

If required, impacted soil will be disposed of in a hazardous waste facility, but experience with other mining operations suggests that the potential quantity of soil that cannot be land-farmed will be small.

20.3.7 Monitoring and Maintenance

Monitoring and maintenance during operations will ensure that the control measures used to prevent environmental impacts are effective. This will include climate monitoring, water quality sampling, groundwater and surface water monitoring, air monitoring, and geotechnical monitoring of the Main Dump, pits, and pit backfill. The monitoring program for operations is described in more detail in the ESIA.

Monitoring and maintenance of the mine reclamation and closure activities will begin while these activities are underway, and will continue in the post-closure period. Monitoring and maintenance will be done primarily by a team of onsite personnel. Equipment will be made available for the monitoring of reclaimed areas of the project and for the performance of regular maintenance. The monitoring will include water quality sampling and analysis, in addition to regular inspection of the Main Dump, process area, and the pit areas. The required duration of the monitoring and maintenance is undefined at this time, but a minimum period of 25 years has been considered in the cost estimate.

Many monitoring activities carried out during operations will be continued in closure, including:

- Automated weather station monitoring;
- Manual water level measurement in monitoring wells;
- Manual water quality sampling in monitoring wells;
- Automated stream flow monitoring via permanent flumes;
- Manual surface water quality sampling;
- Automated geotechnical monitoring at the Main Dump and the Este Pit Backfill; and
- Site-wide inspections for revegetation success, erosion control, surface water management, and general maintenance.

Maintenance will also focus on the surface water channels around the pit, over the pit backfill, and the Main Dump. Sediment removal and repair of any damage to channel lining will be specific areas of focus for the maintenance program.

20.3.8 Closure Schedule

The objectives specifically related to closure include:

- Protection of human health and safety, and of the environment;
- Physical stability of all post-mining landforms and topography;

- Geochemical stability of all post-mining landforms and all deposits of mine residuals;
- To the extent possible, restoration of pre-mining land use and/or creation of beneficial alternative land use.

The schedule for performing reclamation and closure activities can be summarized as follows:

- Este pit backfill - Concurrent reclamation of completed surfaces begins Year 8; completion of reclamation midway through Year 10.
- Minas/Main pit backfill - Completion of backfill reclamation midway through Years 18-21.
- Main Dump - Concurrent reclamation of completed surfaces as they become available, beginning in Year 2; completion of activities in tandem with the Minas/Main pit backfill in Years 18-21.
- Surface water management system – Years 18-21.
- Water treatment plant conversion - Year 18.

20.4 SOCIOECONOMICS AND COMMUNITY

The Project site is located within the jurisdiction of two rural communities (comunidades campesinas), Chacacuniza and Quelcaya. These communities, situated at over 4,000 masl, are small isolated rural settlements, with a population of around 120 individuals each. The majority of the population speaks Quechua as their first language.

The geographical isolation and lack of infrastructure in the zone result in reduced economic and social development opportunities, and high levels of poverty. However, both communities have access to water, electricity, and rudimentary wastewater services. The main housing construction material is adobe, with thatched roofs and earth floors. The most common illness is respiratory infections which children and the elderly are susceptible to during periods of snow and cold weather.

BCM has undertaken community relations activities as part of previous permitting activities related to exploration campaigns. Further activities were undertaken as part of the ESIA that was approved by the Peruvian Government in 2013. The ESIA process included a thorough description of the current social and economic status of the communities, and the analysis of possible and anticipated, positive and negative impacts on these communities. Following analysis of the impacts, methods were developed to avoid, remedy or mitigate the identified negative community impacts, including the development of social programs aimed at providing enhanced economic and social development opportunities. These are presented in the ESIA.

Further community engagement occurred concurrent with the impact assessment process. This included two initial workshops with each of the two communities (Chacacuniza and Quelcaya) affected by the project. The first workshops were undertaken before the ESIA studies began, and the ESIA process was explained to the community. Additional workshops were undertaken during the elaboration of the ESIA. Finally, a public meeting was held. At this meeting, the ESIA and Community Participation Plan were presented to the approving authorities. Since the mine plan described in this document results in a reduced Project footprint and a reduction in water consumption, it is anticipated that impacts to the community will be reduced relative to those described in the current ESIA. For this reason, the ESIA modification is expected to be considered a technical modification which does not require additional public hearings.

Additionally, Bear Creek completed a Life of Mine ("LOM") Investment Agreement in June 2013. This agreement was entered into with the District of Carabaya, five surrounding communities, and relevant, ancillary organizations specifying investment commitments over the 23 year project life, including the preproduction period. Under the agreement, annual payments are to be made into a trust designed to fund community projects totaling 4 million nuevos soles per year (approximately $1.6 million per year), beginning with the first installments payable in 2013. Payments will remain constant throughout the pre-development phase and during production. Cessation or interruptions of operations will cause a pro-rata decrease in the annual disbursements. As an integral part of the LOM agreement, a trust or foundation structure is established for approval of investments and disbursement of funds. Each of the five communities (Corani (Aconsaya), Chacacuniza, Quelcaya, Isivilla, and Aymana) has agreed to the formation of committees which will consider and approve investment projects for the benefit of the communities, such as schools, medical facilities, roads, or other infrastructure. The amounts of the total annual investment to be directed towards each community is agreed to and defined in the agreement. Bear Creek is an oversight member of the trust and will assist towards the success of the projects; however, the Company will have no voting powers. In this structure, Bear Creek's intent is to appoint independent members with community social responsibility experience and credibility in order to provide oversight of the foundation's functions in meeting its commitments to the communities and all of its members.

21 CAPITAL AND OPERATING COSTS

Capital and operating costs for the Corani Project were based on the optimized feasibility mine plan and process plant design. The capital costs were based on estimates for the equipment, materials, labor, and services required to implement the design. Operating costs were based on estimates of labor, materials, power, supplies, fuel, and estimates from consultants and potential suppliers to operate the mine and plant as designed.

21.1 CAPITAL COST SUMMARY

The capital cost estimate prepared for this Report addresses a silver-lead-zinc mine and concentrator capable of producing and processing an average of 22,500 tpd of ore (dry basis). The total estimated cost to design, procure, construct and start-up the facilities described in this section is $625.1 million.

The estimated capital expenditure or capital costs (**CAPEX**) for the Corani Project consists of four components:

(1) The initial CAPEX to design, permit, pre-strip, construct, and commission the mine, plant facilities, ancillary facilities, utilities, and operations camp, and complete onsite and offsite environmental mitigation and remediation. The initial CAPEX also includes indirect costs for engineering, construction management, and Owner's costs.

(2) The sustaining CAPEX for facilities expansions, mining equipment replacements, expected replacements of process equipment and ongoing environmental mitigation activities;

(3) The closure and reclamation CAPEX to close and rehabilitate on and off-site components of the Project, which includes post-closure water treatment; and

(4) Working capital to cover delays in the receipts from sales and payments for accounts payable and financial resources tied up in inventory. Initial and working CAPEX are the two main categories that need to be available to construct a mining project.

Initial CAPEX also includes an estimate of contingency based on the accuracy and level of detail of the cost estimate. The purpose of the contingency provision is to make allowance for uncertain cost elements which are predicted to occur, but are not included in the cost estimate. These cost elements include uncertainties concerning completeness and accuracy of material takeoffs, accuracy of labor and material rates, accuracy of labor productivity expectations, and accuracy of equipment pricing.

Table 21-1: Initial Capital Cost Summary

AREA	2015 TOTAL ($000)
Direct Cost	**$352,062**
General Site	11,021
Mine Capital + Preproduction	63,066
Primary Crushing	24,335
Reclaim Stockpile	8,413
Grinding	48,921
Flotation and Regrind	49,200
Concentrate Thickening & Filtration	16,174
Tailing Thickening & Tailings Pond (See Note 1)	9,443
Tailings Filtration (See Note 1)	55,263
Fresh Water/ Plant Water	11,491
Power Supply Infrastructure	8,551
Reagents	12,881
Ancillaries	33,302
Indirect Cost	**$104,735**
Contractor Indirects	20,436
EPCM Services	45,732
Commissioning and Vendor Reps	1,976
Capital & Commissioning Spare Parts & Initial Fills	11,913
Freight, Duties	24,679
Owners Costs	**103,180**
General Owner's Cost Items	26,718
Operating and Construction Camp	27,918
Mine Access Road (All sections)	32,623
Power Transmission Line	15,921
Contingency	**65,150**
Contingency (Process Plant)	59,746
Contingency (Mine)	5,404
Total	**$625,127**

1) No tailings pond in 2015 plant design as tailings will be filtered.

Table 21-2: Capital Cost Summary

Area	Detail	Initial CAPEX ($000s)	Sustaining CAPEX ($000s)	Total CAPEX ($000s)
Direct Costs	Mine Costs	63,066	18,814	81,880
	Processing Plant	288,966	20,178	309,144
Indirect Costs		104,765	0	104,765
Owner's Costs		103,180	0	103,180
Total CAPEX without Contingency		**$559,977**	**$38,992**	**$598,969**
Contingency		65,150	-	-
Total CAPEX with Contingency		**$625,127**	**$38,992**	**$664,119**

The primary assumptions used to develop the CAPEX are provided below:

- All cost estimates were developed and are reported in United States of America dollars (USD).
- Contingency during the pre-production period is applied at a rate of 15% for prestripping and all construction costs.
- Qualified and experienced construction contractors will be available at the time of project execution.
- Borrow sources are available within the Project boundary.
- Weather related delays in construction are not accounted for in the estimate.

21.1.1 Currency

The estimate is expressed in 2nd quarter 2015 US dollars. No provision has been included to offset future escalation. No funds have been allocated in the estimate to offset potential currency fluctuations. No provision has been made for currency fluctuations.

21.1.2 Estimate Exclusions

Items not included in the M3 capital estimate are as follows:

- Sunk costs;
- Allowance for special incentives (schedule, safety, etc.);
- Reclamation costs (included in financial analysis);
- Escalation beyond 2nd Quarter 2015;
- Foreign currency exchange rate fluctuations;
- Interest and financing cost.

Risk due to political upheaval, government policy changes, labor disputes, permitting delays, weather delays or any other force majeure occurrences are also excluded.

Accuracy

The estimate has been developed to a level sufficient to assess/evaluate the project concept, various development options and the overall project viability. After inclusion of the recommended contingency, the capital cost estimate is considered to have a level of accuracy in the range of ±15%.

Capex Responsibility

Several consultants participated in the development of the initial CAPEX for the Corani Project. Table 21-3 lists the main areas of scope and the responsible parties for each section of the capital cost estimate.

Table 21-3: Capital Cost Estimate Areas and Responsibility Parties

Area	Responsible Party
Mine	GRE
Main Waste Dump/Tailing Co-disposal Facility	GRE
Plant and Ancillary Facilities	M3
Transmission Line	PROMOTORA
Mine Access Road	Anddes, HC&A
Camp Costs	EMSA
Indirect Cost	All
Owner's Cost	BCM
Contingency	All

21.2 MINE CAPITAL COST

Mine CAPEX includes direct mining equipment and pre-stripping costs, process plant costs, infrastructure costs such as the permanent operations camp the power transmission line, and the mine access road and reclamation and closure costs.

Upfront capital costs were minimized by leasing the required mining equipment for a 5-year term. The preproduction leasing and operating costs were capitalized. The total Year 1 capital is $ 56.9M and includes the initial capital leasing payments, inventory and preproduction operating costs. Figure 21-1 shows the capital expenditures over the mine life. Note that the X-axis in the figure is partially in quarterly periods and partially in annual periods, corresponding to the periods in the cost estimate.

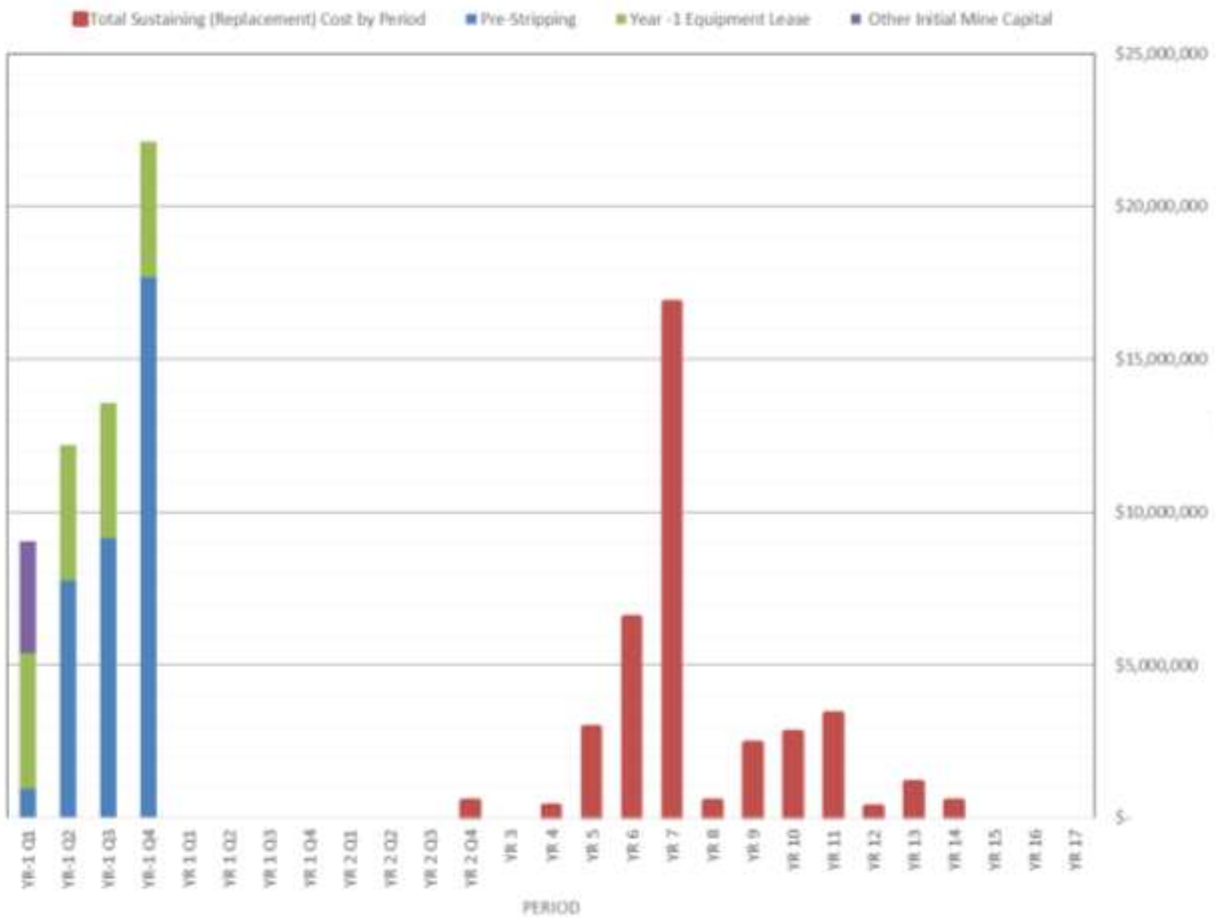


Figure 21-1: Mine Capital Cost Summary

Table 21-4: Corani Project Summary of Mine Capital and Operating Costs

	Total	YR -1	YR 1	YR 2	YR 3	YR 4	YR 5
OPEX - Total	**$ 714,528,400**	**$ -**	**$ 67,975,489**	**$ 57,514,661**	**$ 50,852,129**	**$ 47,580,246**	**$ 33,434,668**
OPEX - Direct Op	$ 580,997,182	$ -	$ 48,287,730	$ 38,105,553	$ 31,730,084	$ 28,535,784	$ 31,382,631
OPEX - Overhaul	$ 23,438,493	$ -	$ 1,955,918	$ 1,565,697	$ 1,278,634	$ 1,127,835	$ 1,269,183
OPEX - Equip. Leasing	$ 110,092,726	$ -	$ 17,731,841	$ 17,843,411	$ 17,843,411	$ 17,916,627	$ 782,855
Tonnes (Ore + Waste)	$ 369,219,550	22,822,468	35,068,685	27,254,649	19,192,936	17,464,821	16,133,470
Op Cost per Tonne	$ 1.94	$ -	$ 1.94	$ 2.11	$ 2.65	$ 2.72	$ 2.07
CAPEX - Total	**$ 75,742,090**	**$ 56,928,551**	**$ -**	**$ -**	**$ -**	**$ 13,049,083**	**$ -**
CAPEX - Initial Lease	$ 17,731,841	$ 17,731,841	$ -	$ -	$ -	$ -	$ -
CAPEX - Prestripping	$ 35,521,863	$ 35,521,863	$ -	$ -	$ -	$ -	$ -
CAPEX - Initial (Other)	$ 3,674,847	$ 3,674,847	$ -	$ -	$ -	$ -	$ -
CAPEX - Sustaining (Balloon payments on leased equipment)	$ 18,813,539	$ -	$ -	$ -	$ -	$ 13,049,083	$ -

	YR 6	YR 7	YR 8	YR 9	YR 10	YR 11	YR 12
OPEX - Total	**$ 44,785,931**	**$ 43,868,807**	**$ 40,670,474**	**$ 34,217,611**	**$ 41,613,879**	**$ 38,036,272**	**$ 29,954,480**
OPEX - Direct Op	$ 40,969,496	$ 36,904,181	$ 33,747,389	$ 27,175,150	$ 34,267,348	$ 31,461,954	$ 26,957,370
OPEX - Overhaul	$ 1,701,153	$ 1,525,637	$ 1,372,526	$ 1,067,627	$ 1,404,210	$ 1,271,431	$ 1,059,388
OPEX - Equip. Leasing	$ 2,115,282	$ 5,438,988	$ 5,550,559	$ 5,974,835	$ 5,942,320	$ 5,302,888	$ 1,937,722
Tonnes (Ore + Waste)	24,792,058	23,307,190	21,343,423	14,831,190	21,697,278	19,503,455	16,601,176
Op Cost per Tonne	$ 1.81	$ 1.88	$ 1.91	$ 2.31	$ 1.92	$ 1.95	$ 1.80
CAPEX - Total	**$ -**	**$ 82,106**	**$ -**	**$ 53,880**	**$ 440,126**	**$ 980,550**	**$ 2,528,063**
CAPEX - Initial Lease	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Prestripping	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Initial (Other)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Sustaining (Balloon payments on leased equipment)	$ -	$ 82,106	$ -	$ 53,880	$ 440,126	$ 980,550	$ 2,528,063

	YR 13	YR 14	YR 15	YR 16	YR 17	YR 18	YR 19
OPEX - Total	**$ 31,577,934**	**$ 29,187,854**	**$ 38,934,841**	**$ 29,989,209**	**$ 28,172,586**	**$ 26,161,328**	**$ -**
OPEX - Direct Op	$ 28,399,213	$ 26,473,242	$ 36,315,041	$ 28,446,309	$ 26,770,161	$ 25,068,547	$ -
OPEX - Overhaul	$ 1,119,349	$ 1,041,161	$ 1,511,902	$ 1,127,997	$ 1,057,634	$ 981,211	$ -
OPEX - Equip. Leasing	$ 2,059,372	$ 1,673,452	$ 1,107,898	$ 414,902	$ 344,791	$ 111,571	$ -
Tonnes (Ore + Waste)	17,766,344	14,477,051	19,099,531	14,787,056	13,178,581	9,898,189	-
Op Cost per Tonne	$ 1.78	$ 2.02	$ 2.04	$ 2.03	$ 2.14	$ 2.64	$ -
CAPEX - Total	**$ 82,106**	**$ 366,110**	**$ 416,198**	**$ 509,984**	**$ 51,596**	**$ 171,630**	**$ 82,106**
CAPEX - Initial Lease	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Prestripping	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Initial (Other)	$ -	$ -	$ -	$ -	$ -	$ -	$ -
CAPEX - Sustaining (Balloon payments on leased equipment)	$ 82,106	$ 366,110	$ 416,198	$ 509,984	$ 51,596	$ 171,630	$ 82,106

21.2.1 Initial Equipment Listing

GRE assumed that the initial fleet would be purchased through a 5 year lease. Leased equipment will be repaid in five annual payments at 5% interest followed by a 15% balloon payment (buyout) in year 5, after which time the equipment is wholly owned by the mine. The initial capital cost includes the first year of the lease payments for a total of $17.7 million USD. It does not include the remainder of the lease or the equipment buyout. The initial fleet satisfies the life of mine fleet requirements however, short term leasing will be required to satisfy periodic spikes in the number of required equipment due to pre-stripping and bofedal removal. Short term leasing reduces the purchased fleet size and is discussed in Section 21.2.4. Equipment operating costs were taken from InfoMine cost services. Capital costs for major equipment were quoted “delivered duty paid” to the Port of Callao (Lima) from the Ferreyros Caterpillar dealer in Lima with lease interest rates and buyout terms provided by caterpillar finance. Initial capital cost estimates for ancillary and support equipment were estimated from InfoMine Mine Cost Services.

21.2.2 Preproduction Operating Expenses

Operating expenses prior to ore production are capitalized. These expenses accounting for the first 3 quarters of mine operations and total $35.5 million USD.

21.2.3 Other Initial Mine Capital

Year -1 (preproduction) capital includes other initial expenditures for spare parts, consumables (explosives), and engineering support equipment such as computers and surveying equipment. The cost of the initial spare parts inventory is assumed be 2.5% of the equipment capital purchase price. This assumes that large items such as engines and transmissions are not held in inventory, but are available in-stock from a local CAT dealer. $1,000,000 in consumables are assumed to be held in initial inventory and engineering support equipment is assumed to cost $500,000 in the first year.

21.2.4 Short-Term Leasing Schedule

For short periods of time in the mine life, the required number of equipment units for waste rock and bofedal soil mining operations exceeds the leased equipment fleet capacity. Short term leasing of equipment has been modeled to handle spikes in the number of required equipment units in an effort to reduce CAPEX. Short term leasing of equipment assumes an operational hourly lease rate equivalent to a 24 month buyback. In other words, the equipment capital cost is divided by 24 months and then converted to an hourly operating cost. Short term leasing occurs for two periods of the mine life. Extra equipment (one CAT 994, five CAT 789s, and two DML Drills) is leased in Years 1-2 to handle pre-stripping requirements and one truck (CAT 740) is leased in Year 7 to handle bofedal removal operations.

21.2.5 Capital Replacement Schedule

The capital replacement schedule summarizes capital costs for replacement of mine equipment at the end of its service life. Each piece of mine equipment has a specified useful service life. The useful service life is based on manufacturer recommendations and industry experience. Most pieces of heavy equipment are scheduled to be replaced at 30,000 hour intervals. CAT 789 haul trucks are scheduled to be replaced at 60,000 hours. Light equipment and support equipment are replaced on fixed yearly intervals, every 3-5 years, depending on the unit.

Equipment replacement is calculated in three steps within the model. First the number of equipment required per period is tabulated. From the tabulated list of equipment, the operating hours for each piece of equipment by period is accumulated. When a piece of equipment reaches is determined end of service life (30 or 60 thousand operating hours), the equipment is replaced and begins to accumulate hours for a new piece of equipment starting from zero. Replacement equipment is leased under the same criteria as the initial equipment lease; a five year lease payment at 5% interest followed by a 15% balloon payment. Finally, the equipment replacement cost by period is summarized and inserted in the replacement schedule at the appropriate time. The total lease cost is the individual equipment lease payment times the number of units being replaced.

Trucks that would require replacement during the last 4 years of the mine life and any other equipment requiring replacement in the last 2 years of the mine life were not replaced. The equipment lives were assumed to be extended to avoid purchasing new equipment at the end of the mine life.

21.3 PLANT CAPEX

M3's battery limits include the process plant from primary crusher to filtered tailing plant discharge, associated process and mine support infrastructure such as administration buildings, truck shop, truck wash, plant maintenance building, warehouse, laboratory, maintenance shops, medical facility, main Corani substation including emergency power generation and distribution to support the site, fresh water supply and reclaim water pipelines and pumping costs, an non-haulage site roads from the main entrance. M3's scope does not include the access road, power supply transmission line, power transmission line to the operations camp, and the operations camp site with dormitories, cafeteria, dining, laundry, recreational center. The operations camp cost estimate prepared by EMSA is included in Owner's cost.

21.3.1 Plant Equipment

All major plant equipment was reinvestigated for this feasibility study update. New flowsheets and Metsim mass balance reports in conjunction with a new grinding power study, led to a revised process design criteria (PDC) for the Project. New pricing were solicited from qualified vendors for the following equipment:

- Jaw crusher
- Conveyors and feeders
- SAG and ball mills
- Regrind ball and vertical stirred mills
- Hydrocyclones
- Mechanical flotation cells
- Column flotation cells
- Thickeners
- Concentrate filters
- Tailings filters
- Lime slaker system
- Slurry and process pumps
- Major electrical equipment (transformers, MCC's, and switchgear)
- Field erected and shop fabricated tankage
- Samples and onstream analyzers

Commodity pricing was also updated for the following:

- Structural steel pricing
- Concrete
- Pipe materials

21.3.2 Material Quantities

Discipline engineers developed material take-off quantities (MTOs) for earthworks, concrete, steel, piping, electrical disciplines and instrumentation based on general arrangement drawings, P&ID drawings, civil site plans and plot plans developed for the Project.

21.3.3 Pricing Methodology

The capital estimate is built up by cost centers as defined by the project Work Breakdown Structure (WBS) and by prime commodity accounts, which include earthwork, concentrate, structural steel, mechanical equipment (including plate work), piping, electrical and instrumentation.

The estimate is based on the assumption that equipment and materials will be purchased on a competitive basis and installation contracts will be awarded in defined packages on either a time and materials basis or as lump sum contracts.

Below is a discussion of how the estimating methodologies have been applied within the commodity groups.

21.3.4 Labor Productivity

Installation hours are based on United States standard rates for the lower 48 states and have been adjusted with productivity factors for working in the Peruvian Andes at high altitude. The productivity factors were developed using historical data from similar projects in the region, as well as comparing man-hours provided by local contractors with the U.S. standards.

Overall, the labor man-hours reflect a 2.5 times decrease in productivity from U.S. standards to account for the altitude, longer workday/workweek, general workforce skill level, the extent of manual production and the remoteness of the site.

21.3.5 Labor Rates

Labor rates were derived for various trades and skill levels using information from recent historical projects in Peru.

The wage rates used reflect a 14-day work period of 12 hours shift and 7 days off. Labor rates do not cover contractor field indirect costs including: mobilization and demobilization, temporary facilities, temporary utilities, testing services, and construction equipment. These items are included with the construction indirect cost.

Average construction crew rates have been developed for each commodity type from the labor information by blending appropriate labor and skill levels to derive reasonable crew mixes.

21.3.6 Buildings

The structural components (civil, concrete & steel) for the process ancillary buildings have been based on MTOs. Architectural finishes, plumbing, and electrical additions were factored on a square meter basis.

Process plant building costs are included in the overall cost of the plant area in which they fall. Table 21-5 lists the capital costs of onsite ancillary facilities.

Table 21-5: Onsite Ancillary Facilities CAPEX

Onsite Auxiliary Facilities	Direct Cost ($000s)
Ancillaries - General	3,957
Administration Building	3,154
Security Building	711
Truck Scale	308
Assay Lab	5,912
Warehouse	3,836
Truck Shop/Truck Wash/Truck Warehouse	7,428
Tire Change Facility	620
Plant Maintenance Building	4,616
Drum Storage	602
Fuel Station	2,305

Onsite Auxiliary Facilities	Direct Cost ($000s)
Explosives Storage	124
Total Onsite Auxiliary Facilities	**$33,302**

The operations and construction camp were costed by EMSA, a camp facilities specialist based in Peru. The camp cost is included as part of the Owners cost is not subjected to project indirects in M3's capital cost estimate. The camp includes site includes the following:

Table 21-6: Camp Facilities Costs

Camp Facilities	Direct Cost ($000s)
Civil Works and Pad	2,931
Dormitories – Supervisors (24 beds)	571
Dormitories – Management (120 beds)	1,433
Dormitories – Staff & Operators (236 beds)	2,085
Dormitories – Support Staff (36 beds)	300
Kitchen & Cafeteria	1,433
Laundry	874
Medical Building	407
Recreation Building	526
Camp Store	314
Camp Admin Office	829
Security Building	45
Other Small Facilities	111
Utilities (Potable Water, Sewage Treatment & Waste Water Disposal, Internet, Fencing, Site Lighting, etc.	4,531
Total Onsite Auxiliary Facilities	**$16,390**

21.3.7 Power Transmission Capital Costs

Promotora provided the costs for the 138 kV transmission line and its interconnection from the Antapata substation to the Corani substation. The cost for the 13.8KV transmission line from the Corani substation to the Camp facilities was also provided. Their cost estimate less cost of the Corani substation is presented in Table 21-7 below.

Table 21-7: Power Transmission Capital Cost Estimate

ITEM	COST ($000s)
Antapata substation interconnection	5,770
138 kV transmission line, Antapata - Corani	8,092
13.8 kV transmission line, Corani - Camp	476
Freight - Equipment & materials	220
Detailed Engineering, CM, Environmental Monitoring, Right-of-Way, Connection Fee	1,362
Total Cost	**$15,920**

21.3.8 Mine Access Road

The new mine access road design requires 56.4 kilometers of road building including four bridge crossings and a major tunnel. A detailed cost estimate for the mine access road design was prepared by Peruvian contractor, Anddes, and is described in Section 18 (Anddes, 2015). The Anddes design originates from the Interoceanic Highway at Huiquisa and continues to the crossing of the Chacoconiza River at Jarapampa. The Anddes cost estimate is detailed and includes earthwork, MSE walls, paving where necessary, the Huiquisa tunnel, civil structures (bridges), civil drainage structures (culverts), signage and environmental work plans.

The end of the Anddes mine access road design at the river crossing at Jarapampa is in between the Corani Mine site and the Corani operations camp. The two road segments, Jarapampa to the mine and Jarapampa to the operations camp, equals 17.45 kilometers total. To develop costs for these road segments, M3 used the cost estimate prepared by HC&A for the 2011 Corani feasibility study. M3 proportioned length of road needed, 17.45 km, against the cost of the entire HC&A mine access road estimate measuring 56.38 km giving a factor of 30.95% of the total HC&A estimate. The mine access road subcontracts are handled as Owner managed construction contracts and included in Owners costs.

Table 21-8: Mine Access Road

ITEM	COST ($000s)
Mine Access Road –Section 1 Huiquisa –Tantamaco-Isavilla 8.3 km (Anddes, 2015)	8,493
Mine Access Road – Section 2 Isivilla Bypass 2.9 km (Anddes, 2015)	990
Mine Access Road – Section 3 Isivilla–Machaycunca-Jarapampa 22.5 km (Anddes 2015)	5,706
Anddes Construction Indirects	706
Anddes EPCM	3,618
Anddes Contingency (15%)	2,927
HC & Associates Mine Access 17.5 km Road Segment – Jarapampa to Corani Mine Site & Jarapampa to Operations Camp	10,183
Total Cost	**$32,623**

21.3.9 Indirect Costs

Indirect costs are those costs that can generally not be tied to a specific work area, as summarized in Table 21-9. This category includes "other direct costs" that are related to construction that can't be assigned directly to a work area including the following:

- Quality assurance testing is included at 2% of total direct costs for civil, concrete, piping, steel, and electrical costs;
- survey is included at 1% of total direct costs for civil, concrete, and steel costs;
- mobilization of contractors is 0.5% of total direct cost without mine & mobile equipment and including quality assurance;
- pipe spooling detail is included at 3% of piping materials; and
- Programming included at 0.2% of direct costs.

Table 21-9: Indirect Capital Cost Summary

Indirect Cost Items	Cost ($000s)
Quality Assurance Testing	3,433
Surveying	1,075
Pipe Spooling	172
Programming	931
Mobilization	1,501
Construction Camp Operating & Bussing	12,816
Freight + Customs + Export Packing	24,678
EPCM Costs	45,732
Vendor Erection Supervision, Start-up, and Commissioning	1,976
Capital and Commissioning Spares, First Fills	11,912
Other Indirect Costs (Fuel Station Subcontract)	509
Total Indirect Costs	**$104,735**

21.4 EPCM COSTS

EPCM cost estimates break down into various categories that total approximately 15.2% of direct constructed field cost excluding mining pre-strip and mine equipment costs, as shown in Table 21-10.

Table 21-10: EPCM Capital Cost Summary

EPCM Components	Percentage of Total Direct Field Cost	Cost ($000s)
Project Services	1.0%	3,004
Project Control	0.75%	2,523
Management & Accounting	0.75%	2,523
EPCM Fee Fixed	1.5% of EPCM cost	676
Engineering	6.0%	19,524
Construction Management	6.5%	18,023
EPCM Total	**15.2%**	**$45,732**

21.5 OTHER CAPITAL COSTS

21.5.1 Surface Water Management

Capital costs for the initial surface water management system include diversions around the plant facilities, initial pit areas, initial dump areas, and initial stockpile areas. The costs include excavation of diversion ditches and lining of ditches with riprap where they cross erodible soils. Initial pumping and piping infrastructure for routing contact water from the mine pit to the plant is also included. Initial capital costs for the surface water management system are estimated at approximately $5,000,000.

A sustaining capital allowance of $500,000 per year has been included to allow expansion of the surface water management system as the project footprint expands and as the mine pits become deeper.

21.5.2 Plant Water Pond

Capital costs for construction of the plant water pond include foundation preparation and earthworks for the two dams, as well as construction of spillways and outlet works. Because field studies for this facility have not been performed, these costs are conceptual. A cost allowance of $4,000,000 has been included, based on what are believed to be conservative assumptions.

21.5.3 Main Dump Foundation Preparation

Soils will be removed from the footprint of the Main Dump for use in reclamation to provide a bedrock foundation for the dump fill. The methods used to estimate costs for stripping and stockpiling soils are based on equipment productivity calculations similar to those developed for the mine plan (See Section 21.8.3). The dump includes an underdrain to be constructed of crushed quartzite quarry rock. Initial capital costs for foundation preparation and underdrain construction are estimated at $6,000,000. A $3,000,000 sustaining capital cost allows for expansion midway through the project life.

21.6 OWNER'S COSTS

The current CAPEX includes an estimate for Owner's Costs. These costs include estimates for Owner's staffing during preproduction, site communications, Owner's camp operating costs, administrative and construction offices, operator training, Owner's commissioning, insurance, environmental compliance, community development, land acquisitions, consultants, and legal expenses.

Table 21-11: Owner's Cost Table

Owner's Cost Build-up		
Item	**Sub Section**	**2015 Total ($000s)**
Staff Build-up	G&A	500
Communications	Phone	200
	Radio	150
	Internet Service - Servers	425
Camp Costs	Food	1,440
	Temp Housing	183
	Power and Water	1,080
Temp Sanitation	Porta-johns	125
	Other	1,200
Offices	Temp on-site (months)	48
	Off-site (months)	36
Admin Equipment	Office furniture, computer hardware, office software	550
Mine & Plant Shop Equipment & Warehouse Fit-out	Small Tools	75
	Warehouse First Fills	200
Medical, Security, & Safety	Medical Station Supplies	50
	Safety Supplies	75
Pre-production Employment & Training	Community Training	300
	Job Specific Training	2,500

Owner's Cost Build-up		
Item	**Sub Section**	**2015 Total ($000s)**
Owner Management	During Build (1.5 years)	840
Owner Commissioning Team		480
Insurance		1,080
Corporate Services		250
Environmental Compliance		600
Community Development		1,500
ROW & Land Acquisition		400
Addition Consultants		400
Legal, Permits & Fees		500
Total		**$15,186**
Additional Engineering		
Detailed Metallurgical Testing		1,000
Total		**$1,000**
Owner Directed Contracts - Offsite Infrastructure		
Permanent Camp - EMSA		16,390
Construction Camp - based on EMSA		11,528
Mine Access Road - Anddes Macusani to Camp		19,513
Mine Access Road - HC&A Camp to Plant		10,183
Promotora Power line & Substation Upgrade		15,921
Total		**$73,535**
Subtotal		**89,721**
Contingency	**15% of subtotal**	**13,458**
Total Owner's Costs		**$103,180**

21.7 MINE OPERATING COST

21.7.1 Mine Operating Cost Summary

Mining operating cost per tonne is calculated as total operating cost divided by total ore and waste tonnes. Total operating cost is the sum of direct operating costs and overhaul costs to mine ore, waste, and bofedal soils, and to load and haul the filtered tailings to the dry stack tailings disposal area. The average operating cost per tonne over the life of the project is $1.93.

Cost per tonne rises from $1.85 in year 1 of production to $2.72 in year 4 when the final lease payments for mining equipment are realized. After year 4, cost per tonne varies from $1.78 to $2.64 in the last year of mine life. Figure 21-2 illustrates the unit operating cost and total breakdown of operating costs by period. Note that the X-axis in the figure is partially in quarterly periods and partially in annual periods, corresponding to the periods in the cost estimate.

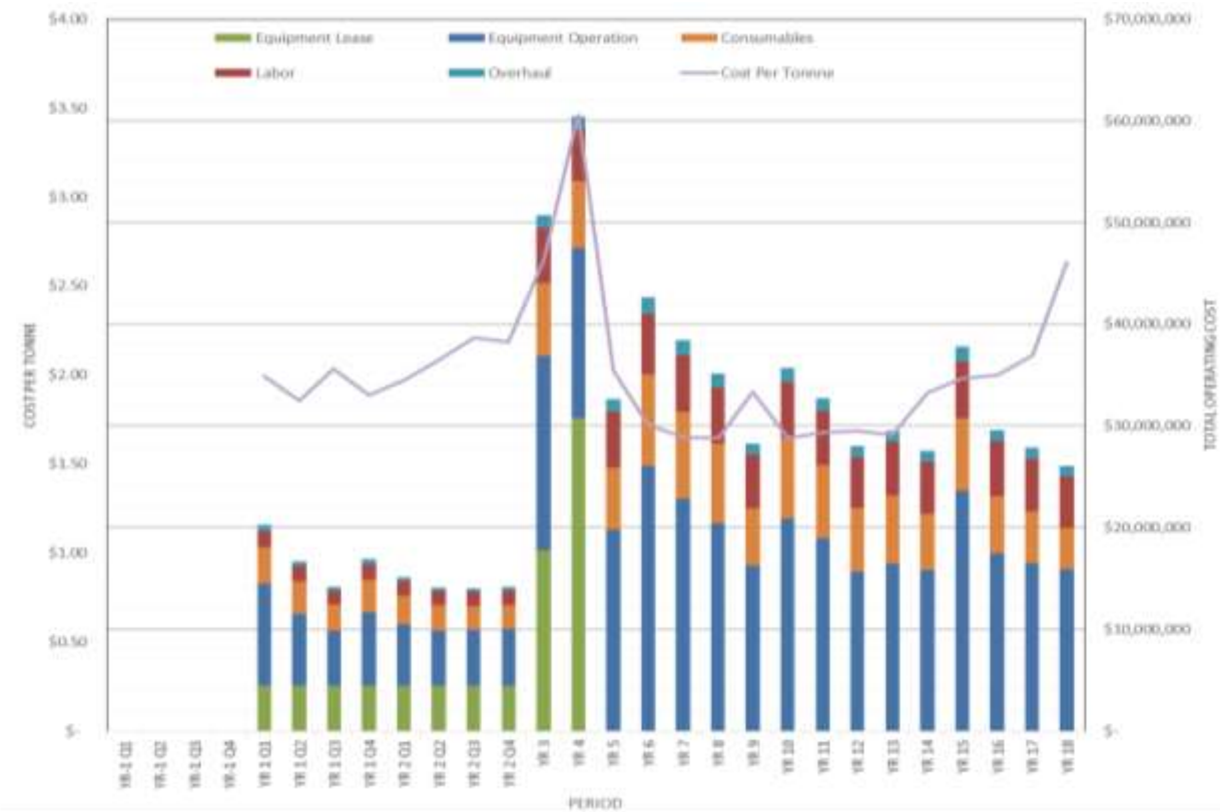


Figure 21-2: Unit and Total Operating Cost Summary

21.8 MINE PLAN & MATERIAL DEFINITIONS

21.8.1 Mine Plan Summary

The general mine plan begins with the Este and Minas pits, followed by Main. Initial prestripping is required to access the ore benches within the pits. The bulk of prestripping occurs early in the mine life. For the purposes of this study, prestripping was increased in order to ensure that sufficient volume was available in the Corani Este pit to begin accepting co-disposed waste and filtered tailings in Year 6 of the mine life. From the pits, ore is hauled to the plant, and waste (pre-strip and in-bench waste) is hauled to the waste rock dump. Once ore processing begins, filtered tailings will be generated and hauled to the dump for co-disposal with the waste rock. Later in the mine life, bofedal soil excavation is required to access specific benches in the Minas and Este pits. This material will be excavated and stockpiled for use as cover material during closure. Figure 21-3 summarizes the required material movement over the mine life. As the mine plan progresses the Este, Minas, and Main pits grow deeper and the waste rock dump grows higher causing material haulage and equipment productivity to vary over time. Note that the X-axis in the figure is partially in quarterly periods and partially in annual periods which corresponds to the periods in the cost estimate.

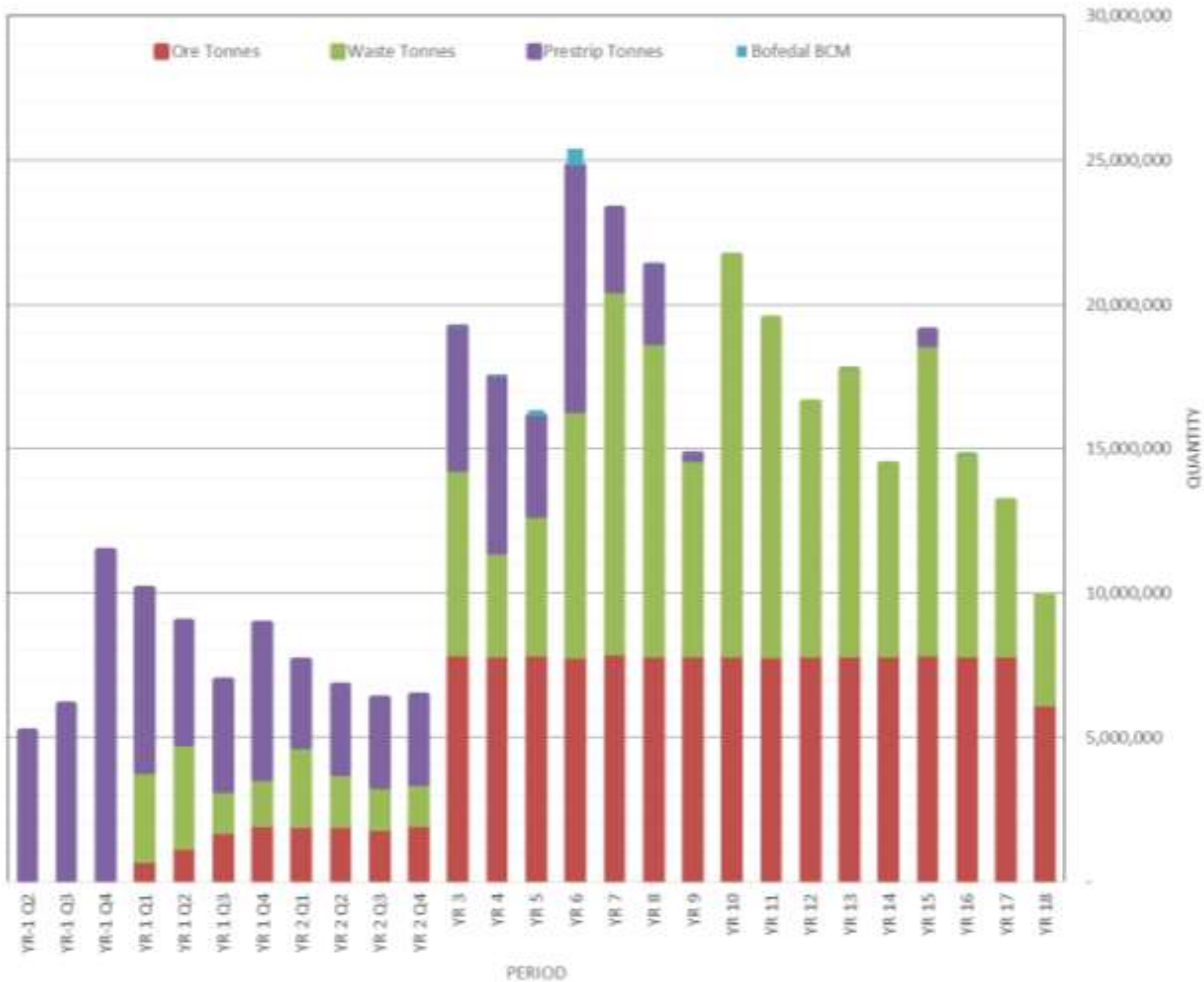


Figure 21-3: Material Movement by Period

21.8.2 Mine Design and General Sequence

Whittle Software was used to develop the Corani pit shell based on metal prices ($20/oz Ag, $0.95/lb Pb, $1.00/lb Zn). The Whittle pit shell was imported into Vulcan software as the basis for creating the detailed mine design and then generally sequenced by phase in Vulcan. Each phase was further sequenced by bench into a monthly mine production schedule showing tonnages of ore and waste by bench and period.

21.8.2.1 Ore

Ore blocks within the mine plan are defined by a net-of-process cut-off value. Cut-off values were determined by net smelter return. An initial cut-off $23/tonne transitioned to a cut-off of $11/tonne by the end of mine life.

Ore production ramps up over the course of 9 months, from zero to full production of approximately 22,500 tonnes per operating day in Year 1. The schedule assumed 365 mine operating days per year.

21.8.2.2 Pre-Strip

Pre-strip material is waste rock of a mining phase that must be mined prior to ore production of each phase.

21.8.2.3 In-Bench Waste

In-bench waste is defined as any block below the defined cut-off grade that occupies a bench that also contains ore. In-bench waste is mined concurrently with the ore on a bench.

21.8.2.4 Bofedal Removal

Bofedal soil is a peat-like low-strength, organic rich, sandy soil material common in the valley floor. Bofedal soil removal is required within the mine plan to access Este pit Phases 3-6 and Minas pit Phases 3-8. The bofedal soil mining schedule is timed to remove the material to expose the bedrock surface ahead of ore and mining of subsequent phases.

21.8.2.5 Filtered Tailings

Filtered tailings are the residual ground rock remaining after mineral processing and water removal. The filtered tailings production quantity is the ore production tonnage less the lead and zinc concentrate tonnage.

21.8.3 Equipment Productivity

Productivity for each piece of equipment was estimted in detail, based on computer modeling with data from the CAT Performance Handbook 45, including high altitude deration adjustments. Equipment productivity and material tonnage moved was then used to determine the total operating hours required to satisfy each unit operation. Truck hauling productivities were calculated in Vulcan. Loading, dozing and other unit operation productivities were calculated individually in detail. For ore, waste, and pre-strip a bank density of 2.35 tonnes/m3 with a swell factor of 30% was assumed. For bofedal a wet bulk density of 1.49 tonnes/m^3 and a swell factor of 10% was used. The total number of units of equipment required during the mine life is summarized in Figure 21-4 (To show resolution in scale, the number of 789 trucks is shown separately in Figure 21-5). All effective equipment productivity includes 85% mechanical availability and 95% use of availability. Fixed delays were assumed to be 1.25 hours per shift.

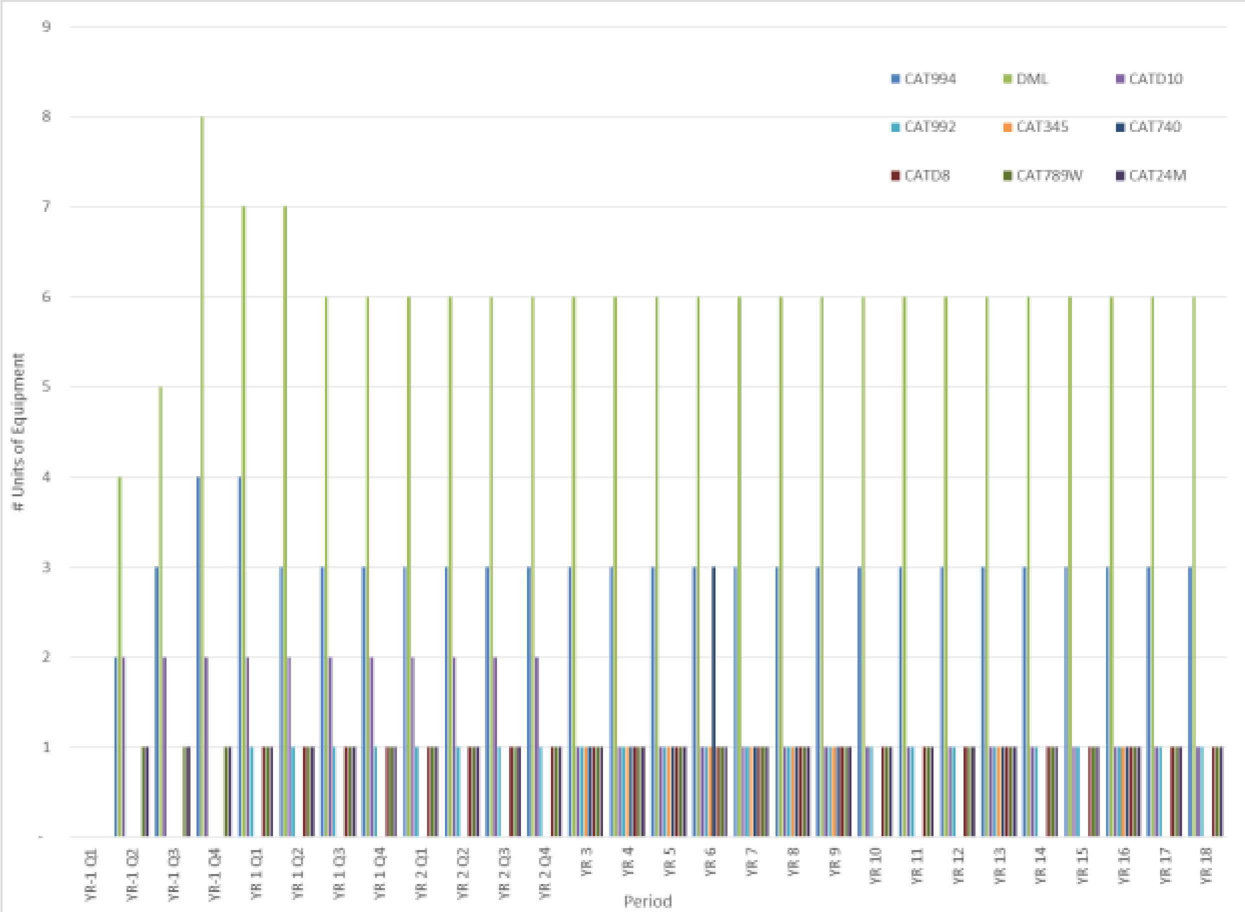


Figure 21-4: Number of Units of Mining Equipment

21.8.3.1 CAT 994 Loader – All Ore & Waste

The CAT 994 wheel loader handles loading operations for ore and waste (including pre-stripping) within the pit. Loading operations are based on filling a CAT 789 haul truck. From this loading operation and cycle time, the effective hourly productivity is calculated. Effective hourly productivity accounts for machine availability, fixed delays, and operating efficiency. Efficiency also accounts for altitude deration. The CAT 994 loader productivity is summarized in Table 21-12. Based on total ore and waste tonnes, the total number of required loader operating hours is calculated. The total number of loader operating hours is then used to determine the total number of equipment units by period and operating cost.

Table 21-12: CAT 994 Effective Productivity

Equipment Specification	Units	CAT 994
Nominal Bucket Capacity	lcm	18
Fill Factor	%	90%
Elevation correction (5000m AMSL)	%	93%
Effective Bucket Capacity	bcm	12.5
Effective Bucket Capacity	tonnes	29.3

Equipment Specification	Units	CAT 994
Truck Size/Capacity (weight)	tonnes	180.7
Truck Size/Capacity (volume)	cm	112.5
Number of Passes to fill truck	passes	6
Truck Spot Time	minutes	0.50
Loader Cycle Time	minutes	0.67
Mechanical Availability	%	85%
Use of Availability	%	95%
Total Fixed Delays per shift	hrs/shift	1.25
Operating efficiency	%	95%
Effective productivity	bcm/hr	876
Effective productivity	tonne/hr	2059

21.8.3.2 Haul Truck Productivity (CAT 789 & CAT 740)

For truck haulage productivity, Vulcan Software was used to determine the travel time between mass centroids at each period in the mine life. For each time period, the outline of the active pit areas was used to determine source centroids and outlines of the active dump areas to determine destination centroids. Haul cycle times were calculated using 3D polylines following designed roads. Cycle times in Vulcan include travel time, spot time, load time, acceleration/deceleration, dump time, and estimated average delays. Cycle times were used to estimate single truck productivity, which was then used to calculate total truck operating time required per period. Separate truck cycle times were estimated for ore, waste, bofedal, and filtered tailings. The following ranges of productivities occur during the mine life Figure 21-5):

- Ore haul productivities range from 400 to 1500 tonnes per hour;
- Tailings haul productivities ranges from 350 to 800 tonnes per hour;
- Waste haul productivities ranges from 130 to 600 tonnes per hour.

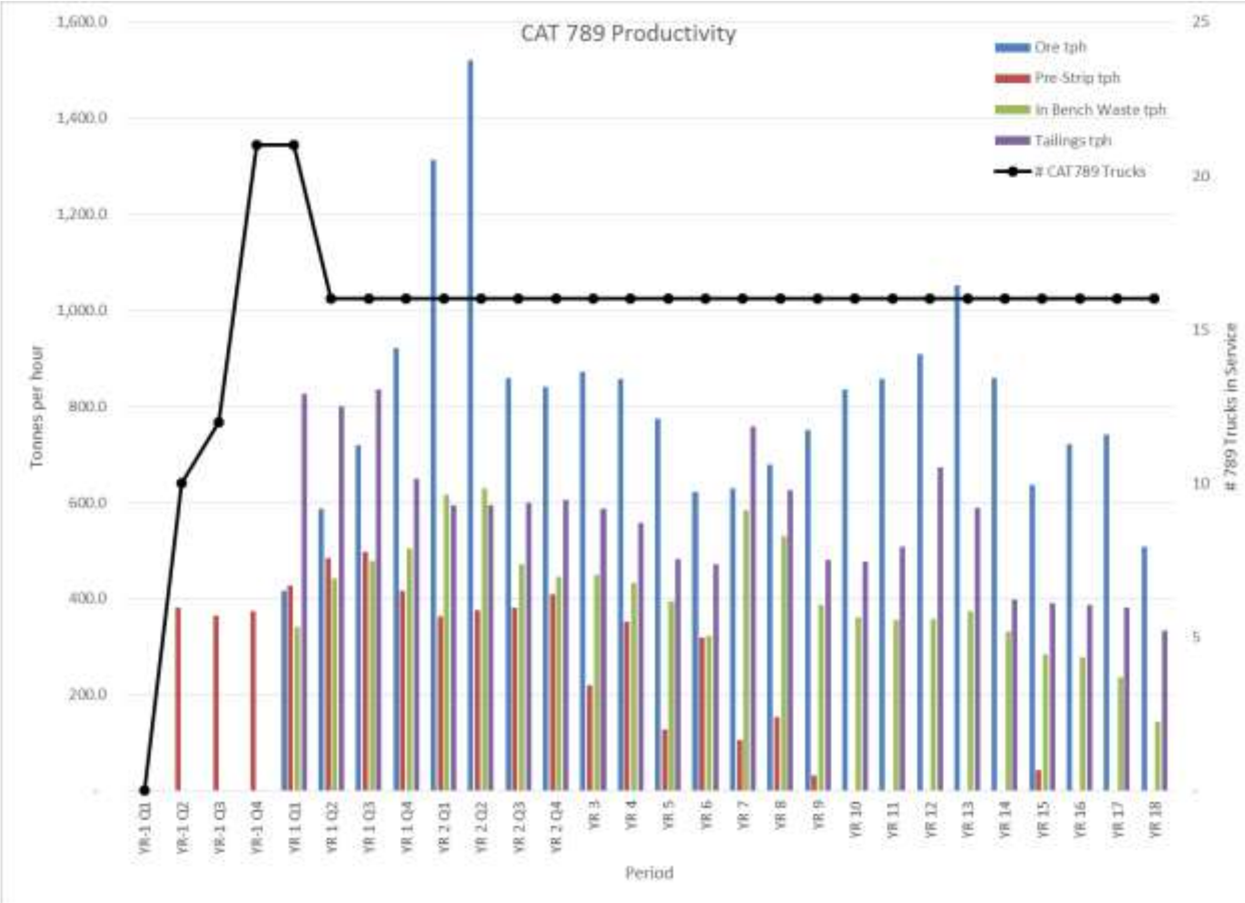


Figure 21-5: CAT 789 Truck Haul Productivities

21.8.3.3 Atlas Copco DML Drill – Ore & Waste

To calculate single drill productivity, the mass of shot rock per borehole was calculated using the Mining Reference Handbook - Method Based on Known Powder Factor (SME, 2002). The volume of shot rock per borehole was based on a Powder Factor of 1 lb/cubic yard, 8 meter high benches, a drilling penetration rate of 100 feet/hr, and 6¾-inch diameter boreholes. From the volume of shot rock per borehole and the drilling rate a theoretical productivity was calculated. Based on the theoretical single drill productivity, availability and delay factors were applied to arrive at an effective productivity (Table 21-13). Based on the total tonnes of ore and waste, the single drill effective productivity was used to calculate the total operating hours required. The total number of required operating hours was used to estimate the total number of equipment required.

Table 21-13: Effective DML Drill Productivity

Equipment Specification	Units	DML
Drilling Rate	m/hr	30.5
Drill Diameter	in	6.75
Bench height	m	8
Mass of Shot Rock per borehole	tonnes	405.5

Equipment Specification	Units	DML
Volume of Rock Broken per borehole	m^3	171.8
Total Drill Time per Hole	hr	0.38
Single Drill Productivity	tph	1069
Mechanical Availability	%	85
Use of Availability	%	95
Total Fixed Delays per shift	hrs/shift	1.25
Operating efficiency	%	95
Effective productivity	tonnes/hr	1015

21.8.3.4 CAT D10 Dozer

The CAT D10 dozer is used to spread waste rock at the waste dumps after being dumped from haul trucks. The CAT D10 productivity was calculated from the guidelines provided in the Caterpillar Performance Handbook 45 (CAT, 2015). Based on a universal blade type and a 15m level push, the theoretical maximum dozer productivity was pulled from the CAT productivity chart. From the theoretical productivity a variety of correction factors were applied to account for availability, delays, inefficiencies, and material to arrive at an effective dozer productivity. Productivities were calculated for dozing flat terrain (waste dump), dozing steep terrain (road construction), and ripping (steep terrain). The productivity for dozing on flat terrain was used to estimate the total required operating hours for waste rock per period. Waste rock includes in-bench waste and pre-stripping. A summary of dozer productivity is shown in Table 21-14.

Table 21-14: Effective Dozer Production

Equipment Specification	Units	D10 Flat
Max Dozer Production	Lm3/hr	2300
Blasted rock material correction factor	unitless	1
Grade correction factor (1 for Level working surface)	unitless	1
Material weight correction	unitless	0.76
Elevation correction (5000m AMSL)	unitless	1
Mechanical Availability	%	85
Use of Availability	%	95
Total Fixed Delays per shift	hrs/shift	1.25
Effective Dozer Production	tonne/hr	2993

21.8.3.5 CAT 992 Loader

The CAT 992 loader is used to load filtered tailings at the plant into 789 haul trucks. From the guidelines provided in the Caterpillar Performance Handbook 45 (CAT, 2015) the bucket capacity and cycle time was used to determine the machine theoretical productivity. Correction factors for availability, elevation, and efficiency were applied to the CAT 992 theoretical productivity to estimate the effective loader productivity. Based on the loader productivity and the tonnage of filtered tailings produced, the total operating hours per period were estimated. The total operating hours were used to determine the total number of loaders required. The CAT 992 productivity is summarized in Table 21-15.

Table 21-15: Effective CAT 992 Productivity Summary

Equipment Specification	Units	CAT 992 HL
Nominal Bucket Capacity	lcm	12
Fill Factor	%	90
Elevation correction (5000m AMSL)	%	90
Effective Bucket Capacity	bcm	8.3
Effective Bucket Capacity	tonnes	19.5
Truck Size/Capacity (weight)	tonnes	180.7
Truck Size/Capacity (volume)	cm	105
Number of Passes to fill truck	passes	9
Truck Spot Time	minutes	0.50
Loader Cycle Time	minutes	0.67
Mechanical Availability	%	85
Use of Availability	%	95
Total Fixed Delays per shift	hrs/shift	1.25
Operating efficiency	%	95
Effective productivity	bcm/hr	587
Effective productivity	tonne/hr	1379

21.8.3.6 CAT 345 Excavator

The CAT 345 excavator is used to load a CAT 740 haul truck during bofedal excavation activities. From the information provided in the Caterpillar Performance Handbook 45 (CAT, 2015) the bucket capacity and cycle time were used to determine the machine theoretical productivity. Correction factors for availability, elevation, and efficiency were applied to the CAT 345 theoretical productivity to estimate the effective excavator productivity. Based on the effective productivity and the volume of bofedal produced, the total operating hours per period were estimated. The total operating hours were used to determine the total number of excavators required. The CAT 345 productivity is summarized in Table 21-16.

Table 21-16: Effective CAT 345 Productivity

Equipment Specification	Units	CAT 345
Effective Bucket Capacity Volume	bcm	2.54
Effective Bucket Payload Weight	tonnes	3.78
Bucket Cycle Time	minutes	0.33
Truck Size/Capacity (weight)	tonnes	38
Truck Size/Capacity (volume)	cm	23.1
Number of Passes to fill truck	passes	9
Truck Spot Time	minutes	0.5
Cycles per Hour	cycles/hr	180
Effective BCM / hr	bcm/hr	260

21.8.3.7 CAT D8 Dozer – Filtered Tailings

The CAT D8 dozer is used to push filtered tailings at the waste dumps after they dumped from the haul trucks. Based on a universal blade type and a 15m level push, the CAT D8 theoretical productivity was calculated from the guidelines provided in the Caterpillar Performance Handbook 45 (CAT, 2015). From the theoretical productivity a variety of correction factors were applied to account for availability, delays, inefficiencies, and material to arrive at the effective dozer productivity. Productivity was calculated for dozing on flat terrain. The productivity for dozing on flat terrain was used to estimate the total required operating hours for filtered tailings per period. A summary of dozer productivity is shown in Table 21-17 .

Table 21-17: Effect CAT D8 Productivity

Equipment Specification	Units	D8
Max Dozer Production	Lm3/hr	1050
Loose tailings material correction factor	unitless	1.2
Material weight correction	unitless	0.76
Elevation correction (5000m AMSL)	unitless	1.0
Mechanical Availability	%	85
Use of Availability	%	95
Total Fixed Delays per shift	hrs/shift	1.25
Operating efficiency	%	95
Effective Dozer Productivity	tonne/hr	1640

21.8.3.8 CAT 789 Water Truck

The 789 water truck is assumed to operate continuously during one 12-hr day shift. Hours for the water truck are manually reduced during the preproduction periods and ramp up to full operation in the first year of ore production. Water trucks were not scheduled to run at night due to low temperatures which could result in the formation of ice on roads. The number of operating hours was then used to determine the required number of trucks.

21.8.3.9 CAT 24 Motor Grader

The CAT 24 grader is assumed to operate continuously during one 12-hr day shift in support of haul road maintenance operations. In the model the grader is scheduled to have the same number of operations hours as the water truck. The number of operating hours was then used to determine the number of required graders.

21.8.3.10 Support Equipment

Support equipment operating hours are estimated independently of ore and waste production. Equipment that supports larger unit operations is calculated as a percentage of the full operating time. Ancillary support equipment such as light trucks and personnel buses was estimated based on engineering judgement. The number of units of support equipment estimated in the mine cost model is shown in Table 21-18.

Table 21-18: Units of Support Equipment

Support Equipment	% Utilization	# Units of Equipment
CAT 336 W/Rock Breaker	25%	1
CAT 950 Loader	25%	1
CAT 450 Backhoe	25%	1
CAT 236 Skid Steer	25%	1
Forklift	25%	1
Telehandler (TL642)	10%	1
Crane	10%	1
4x4 pickup	10%	10
Employee Bus	100%	2
Fuel/Lube Truck	10%	1
Mechanic/Service Truck	10%	1
Explosives Truck	10%	1
Heavy Duty Pumps	10%	2
Sump Pump	10%	2
Portable Lights	100%	5

The CAT 336 will support the 994 loader in the pit, using a rock breaker attachment to reduce oversized material and maintain smooth and efficient loading operations. Since the 336's role is to assist the 994, it is assumed that the 336 will operate 25% of the 994 hours and during those operational hours the excavator will be 100% efficient. Based on the operations hours, equipment unit was required.

The 950 loader will work in support of ore dumping into the crusher hopper, cleaning up any loose debris falling out of the trucks. The 950 hours are therefore based on 25% of the 789 ore haulage hours, of that time the 950 is assumed to be 100% efficient. Based on the operating hours, one piece of equipment was required.

Other support equipment was assumed to work a fixed percentage of two 12hr shifts per day according to the estimated utilization of each piece of equipment. The estimate utilization is based on engineering judgment. The model assumes that during operating hours, the equipment is 100% efficient and that all delays and maintenance occur during non-operational hours.

21.8.4 Operating Hours

Operating hours for primary heavy equipment are based on equipment productivity and material tonnage. Productivity for haulage varies over the life of the mine. As productivity and ore/waste tonnage change, operating hours fluctuate over the life of mine. Operating hours for support equipment are based on fixed hours per shift. The total operating hours per equipment type are used later in the model to calculate hourly operating cost. Figure 21-6 summarizes the operating hours for the following heavy equipment units:

- CAT 994
- CAT 789
- Atlas Copco DML
- CAT D10
- CAT 992

- CAT 345
- CAT 740
- CAT D8
- CAT 789 Water Truck
- CT 24M

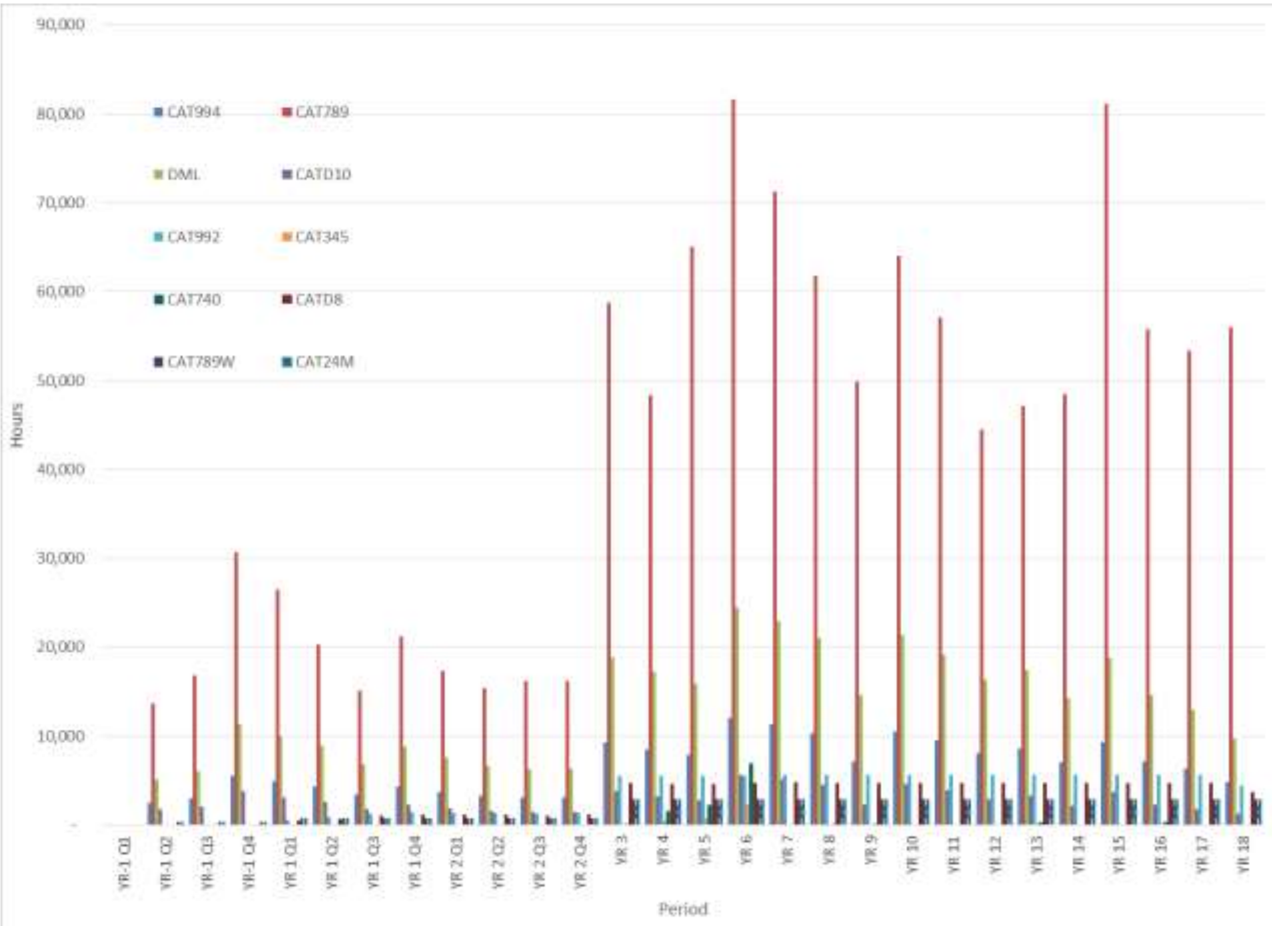


Figure 21-6: Operating Hours by Equipment

21.8.5 Direct Operating Cost

Total equipment operating cost is comprised of direct operating costs and overhaul costs. Direct operating costs are discussed in the following section and are comprised of the following sub categories:

- Equipment operating cost
- Manpower labor (Includes supervisors, operators, and maintenance)
- Consumables

Equipment hourly operating cost is based on total equipment operating hours per period times the hourly operating cost estimate from InfoMine. Total equipment operating hours are discussed above in Section 21.8.4. Manpower is comprised of mine supervision staff, machine operators and maintenance staff. Mine supervision staff is a fixed number based on past project experience. Operators and maintenance staff are based on the total number of pieces of heavy equipment. Consumables are calculated on an individual basis to meet staff and production requirements using price quotations from Peruvian suppliers for major items.

21.8.5.1 Equipment Operating Cost

Hourly operating costs are derived from Mine and Mill Equipment Costs (InfoMine, 2014). Hourly operating costs include maintenance parts, fuel/electricity, lube, tires, and wear parts. Local prices for fuel and electricity were used in the cost estimate. Hourly operating costs do not include overhaul parts & labor or maintenance labor. These items where estimated separately. The hourly operating cost for major equipment is summarized in Figure 21-7. From hourly operating cost and total operating hours, the operating cost sub-total for heavy machinery and support equipment by period was calculated.

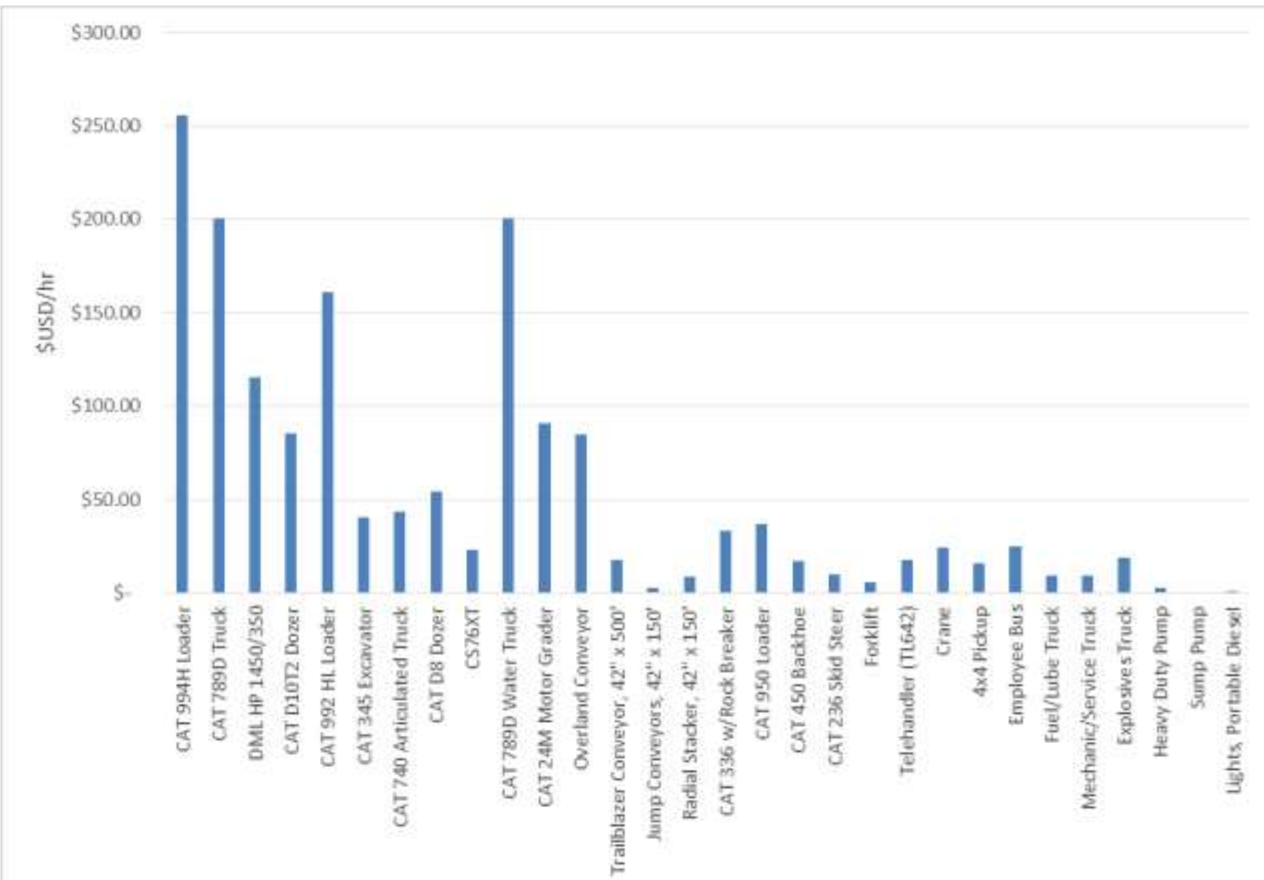


Figure 21-7: Equipment Operating Cost per Hour

21.8.5.2 Mine Supervision

Mine supervision was included in the cost estimate as an operating cost. Supervision staff salaries were provided by Bear Creek. For supervision manpower two 12-hr shifts per day and 4 crews (two on duty, the other two off duty) were assumed. The manpower to handle supervision assumed:

- 1 Mine manager to supervise all mining activity

- 2 Mine superintendents (1 per operating day)
- 4 Mine foreman (1 per crew)
- 2 Maintenance superintendents (1 per operating day)
- 4 Maintenance Foreman (1 per crew)
- 4 Chief Mine Engineers (1 per crew)
- 4 Chief Geologists (1 per crew)
- 4 Geologists (1 per crew)
- 4 Engineers (1 per crew)
- 2 Surveyors (1 per operating day)

Table 21-19 summarizes the total supervision manpower, salary by position, and total salary by position for the 2 required crews. The salary used is a fully burdened salary.

Table 21-19: Supervision Manpower & Salaries

Position	Total Personnel	Annual Salary, USD	Crew Salary, USD
Mine Manager	1	$ 130,800	$ 130,800
Mine Superintendent	2	$ 97,600	$ 195,200
Mine Foreman	4	$ 72,800	$ 291,200
Maintenance Superintendent	2	$ 97,600	$ 195,200
Maintenance Foreman	4	$ 72,800	$ 291,200
Chief Mine Engineer	4	$ 84,800	$ 339,200
Chief Geologist	4	$ 84,800	$ 339,200
Geologist	4	$ 54,400	$ 217,600
Engineer	4	$ 54,400	$ 217,600
Surveyor	2	$ 16,500	$ 33,000

21.8.5.3 Equipment Operators

Based on the equipment operating hours, the number of operators required assumed 2,100 working hours per person per year for each of the following pieces of equipment: CAT 994, CAT 789, DML, CAT D10, CAT 992, CAT 345, CAT 740, CAT D8, CAT 789W, and CAT 24M. For the remainder of the equipment, the number of operators required was based on engineering judgement. For each type of equipment, an operator of an appropriate skill level was assigned (one through three). Three tiers of operator pay grade, based on skill level were created. Salaries for the three operator pay grades were correlated from salary data sourced from M3's project database. From the number of operators and operator salary, total operations labor cost by period was calculated. The salary used is a fully burdened salary (Table 21-20). Figure 21-8 summarizes the total number of operators required during the mine life.

Table 21-20: Operator Salary

Operator	Annual Salary	Units
Operator 1	$ 18,800	$/yr
Operator 2	$ 17,800	$/yr
Operator 3	$ 16,800	$/yr
Driver	$ 12,000	$/hr

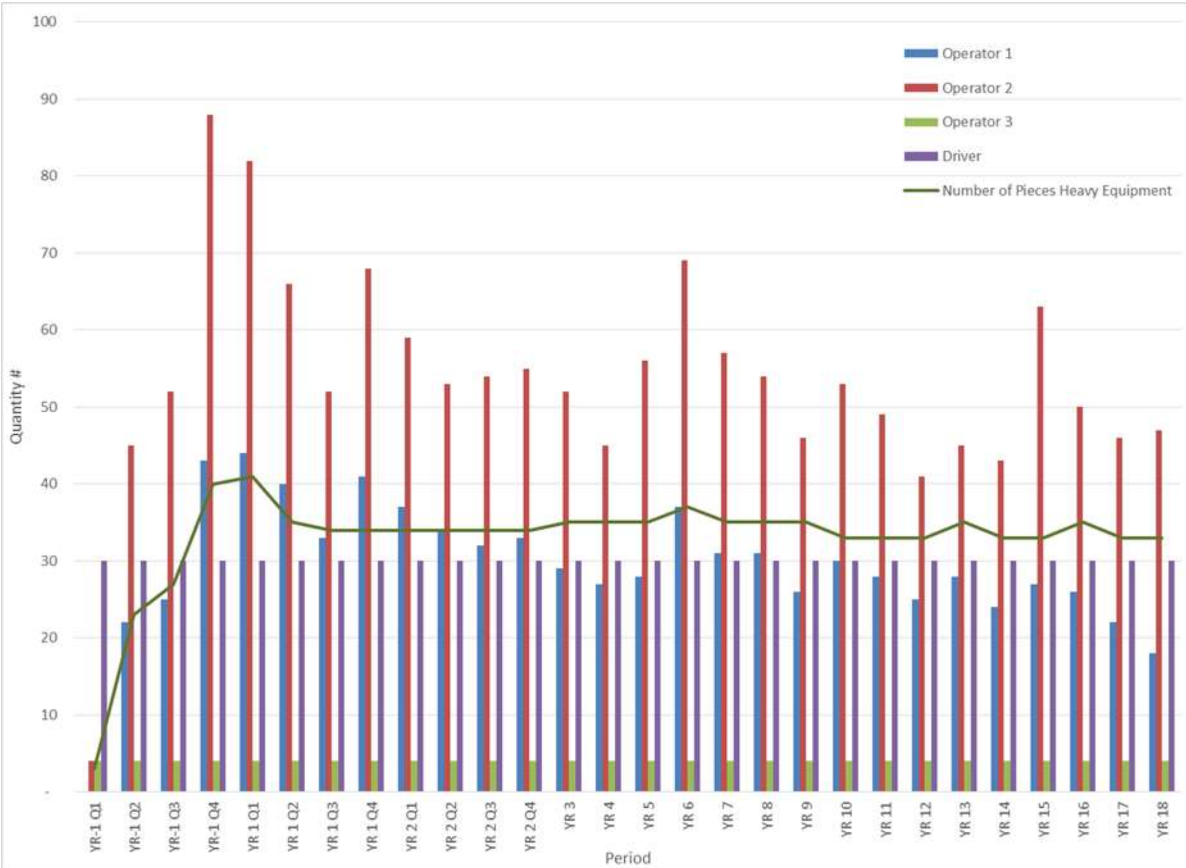


Figure 21-8: Operator Manpower

21.8.5.4 Consumables

21.8.5.4.1 Explosives

Explosives quantities were based off of a 100 ft/hr drilling penetration rate, 6¾ " borehole diameter, 8-m bench height, and a powder factor of 1 lb/cubic yard. Based on these assumptions the weight of explosives per borehole and the mass of shot rock per borehole were calculated using the Mining Reference Handbook - Method Based on Known Powder Factor (SME, 2002). By dividing the mass of ammonium nitrate/fuel oil (ANFO) per hole by the tonnage of shot rock per hole, ANFO per tonne of rock blasted was calculated. For a combined ore, waste, and pre-strip tonnage, total required ANFO per period was estimated. A 10% allowance for emulsion was added to the bulk explosive cost estimate to account for wet holes. A six row (six holes per row) blasting pattern was assumed in order to estimate the explosive accessories. Table 21-21 summarizes the results of the explosives calculations and the consumable quantities required for blasting. Total explosives quantities per period were multiplied by unit cost (InfoMine, 2014) for a total explosives cost per period. The average blasting cost is 0.33 $/tonne.

Table 21-21: Explosives Calculations

Description	Value	Units
Powder Factor	0.59	kg/m^3 (1lb/cy)
Stemming Length	3.50	m
Sub-drilling	1.03	m
Bench Height	8.00	m
Powder Column	5.53	m
Loading Factor	18.44	kg/m
Weight of Explosive per Borehole	101.9	kg
Burden	4.6	m
Spacing	4.6	m
ANFO per tonne blasted	0.2514	kg/tonne
Blasting Caps (1 per hole)	0.0025	Caps/tonne
Boosters (1 per hole)	0.0025	Booster/tonne
Snap Lines (1 per row)	0.0004	Snap Line/tonne

Table 21-22: Explosives Cost Table

Item	InfoMine Cost ($)		Unit Cost ($)	
Boosters	5.32	/ea	5.32	/ea
Blasting Caps	216.40	/100	2.16	/ea
ANFO	55.60	/100lbs	1.22	/kg
Emulsion	69.50	/100lbs	1.53	/kg
Snap Lines	8.64	/ea	8.64	/ea

21.8.5.4.2 Dust Suppression

Magnesium chloride will be applied on the haul roads to reduce the amount of water required for dust suppression. Vendor guidelines suggest an application rate of ½ gallon (product sold and measured in imperial units) per square yard (2.3 L/m^2). For one application of the mine haul roads, this equals $629,240 for 180,816 gallons (408,644 L) at $3.48 per gallon. A single application per year was assumed.

21.8.5.4.3 Personal Protective Equipment and Hand Tools

A small daily allowance for the purchase and replacement of hand tools and personal protective equipment was assumed.

21.8.6 Equipment Maintenance & Overhaul

Maintenance labor was estimated based on the total units of heavy equipment. The maintenance manpower is assumed to include the capacity to handle overhaul as well as normal maintenance duties. Overhaul parts are accounted for separately to show the cost breakout from labor costs.

21.8.6.1 Maintenance Manpower

Manpower includes mechanics, electricians, welders, and laborers (helpers). The maintenance staff was sized to accommodate both maintenance and overhaul duties together. Manpower for maintenance and overhaul was based on the number of pieces of heavy equipment. One and a half mechanics were estimated for each piece of heavy equipment (CAT 994, CAT 789, DML, CAT D10, CAT 992, CAT 345, CAT 740, CAT D8, CAT 789W, CAT 24M, CAT 336, CAT 950, and CAT 450). One mine electrician was assigned per shift. One welder is included for every four mechanics, and one laborer for every 2 mechanics. The total number of required maintenance staff is summarized in Figure 21-9. This maintenance workforce will also be able to support the light equipment such as trucks, buses, and miscellaneous equipment.

Salaries for the maintenance manpower were sourced from M3's project database. The salary used is a fully burdened salary (Table 21-23).

Table 21-23: Maintenance Salary

Labor	Annual Salary	Units
Mechanic	$ 17,800	$/yr
Electrician	$ 17,800	$/yr
Welder	$ 17,800	$/yr
Laborer/Helper	$ 12,600	$/yr

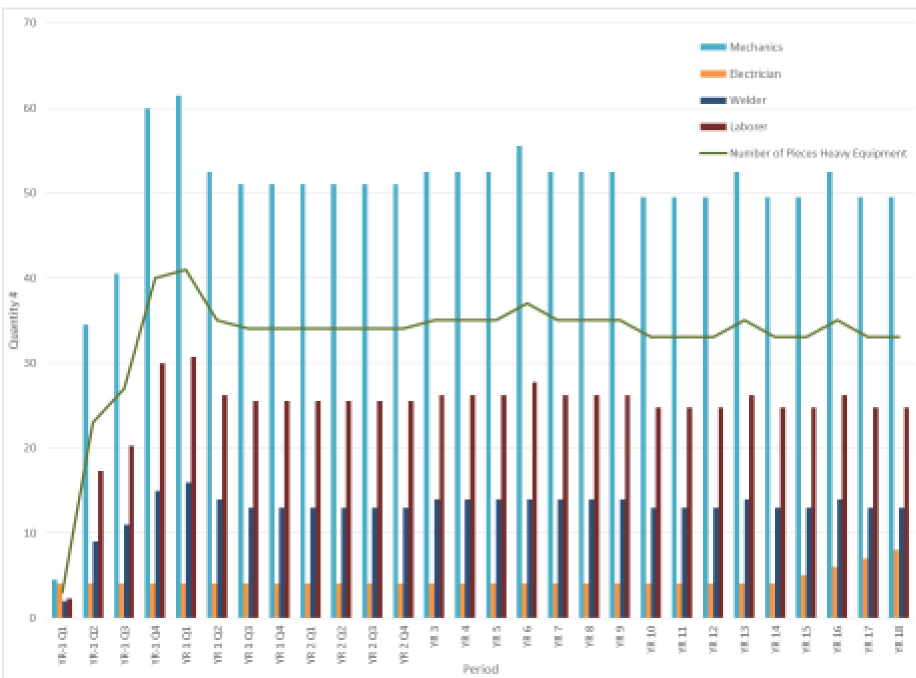


Figure 21-9: Maintenance Manpower

21.8.6.2 Overhaul Hourly Cost

For the purpose of this cost estimate equipment overhaul is considered the replacement of major equipment components, such as engines and transmissions, at regularly scheduled intervals. For the heavy equipment in this cost estimate all units are scheduled to undergo overhaul approximately every 15,000 hours. Overhaul parts costs are accounted for on a cost per hour basis. The hourly overhaul parts costs are multiplied by operating hours to arrive at a total overhaul cost per period. Overhaul labor is included in "Maintenance Manpower". Hourly overhaul costs are not applied to light vehicles and or small equipment such as sump pumps. These items are assumed to undergo routine maintenance and then be fully replaced at fixed intervals.

21.9 PROCESS PLANT OPERATING & MAINTENANCE COSTS

The process plant operating costs are summarized by areas of the plant and then by cost elements of labor, power, reagents, grinding media, wear items, maintenance parts and supplies and services. Average LOM process plant operating cost is $8.76 per tonne of ore processed as shown in Table 21-24.

Table 21-24: Process Plant Operating Cost

Cost Item	Annual - $	$/Ton Ore
Primary Crushing		
Operating Labor and Fringes	375,160	0.048
Power	475,568	0.060
Liners	270,900	0.034
Maintenance	677,070	0.086
Supplies & Services	140,000	0.018
Subtotal Primary Crushing	**$1,938,699**	**$0.246**
Grinding		
Operating Labor and Fringes	358,888	0.046
Power	7,817,143	0.993
Grinding Media	10,158,750	1.290
Liners	1,477,350	0.188
Maintenance	1,686,822	0.214
Supplies and Services	390,000	0.050
Subtotal Grinding	**$21,888,954**	**$2.780**
Flotation		
Operating Labor and Fringes	734,048	0.093
Power	3,532,859	0.449
Reagents	24,692,850	3.136
Grinding Media	201,600	0.026
Liners	325,238	0.041
Maintenance	1,540,283	0.196
Supplies and Services	165,000	0.021
Subtotal Flotation	**$31,191,878**	**$3.961**
Concentrate Thickening, Filtration and Tailings		
Operating Labor and Fringes	928,408	0.118
Reagents	637,875	0.081
Power	4,185,273	0.531
Maintenance	2,632,112	0.334
Supplies and Services	2,350,331	0.298
Subtotal Concentrate Thickening, Filtration and Tailings	**$10,733,999**	**$1.363**
Ancillary		
Operating Labor and Fringes	706,702	0.090
Power	194,747	0.025
Maintenance	1,328,345	0.169
Supplies and Services	763,611	0.097
Subtotal Ancillary	**$2,993,405**	**$0.380**
Total Process Plant	**$68,746,934**	**$8.730**

21.9.1 Process Labor & Fringes

Process labor costs were derived from a staffing plan and based on prevailing annual labor rates in the area which were provided by Bear Creek Mining. Labor rates and fringe benefits for employees include all applicable social security benefits as well as all applicable payroll taxes. A total of 159 employees will be employed at the process plant with 105 in operations and 54 in maintenance. A summary of the staffing plan and gross annual labor costs are shown in Table 21-25.

Table 21-25: Process Plant Staffing Summary

Department	Number of Personnel	Total Labor ($)/year
Mill Operations	105	$3,103,206
Mill Maintenance	54	$1,430,128
Total Labor Cost	**159**	**$4,533,334**

21.9.2 Power

Power costs were based on the purchasing power from a local utility company and associated rates were applied. Power consumption was based on the connected kW derived from the equipment list, discounted for operating time per day and anticipated operating load level. The overall power rate is estimated at $0.051 per kWh with a consumption of 32.8 kWh per ore tonne. A summary of the power consumption and cost are shown in Table 21-26.

Table 21-26: Summary of Electric Power

Cost Item	Area	Connected kW	Average (kW hr/yr.)	Average Cost ($/yr)
Primary Crushing & Conveying	100 & 200	1,762	9,012,383	461,975
Grinding	300	24,670	148,140,844	7,593,700
Flotation	400	11,078	66,950,379	3,431,876
Concentrate Thickening & Filtration	500	1,360	8,582,851	439,957
Tailings Thickening &Tailing Filtration	600 & 620	11,845	70,731,272	3,625,685
Water Supply System	650	1,133	5,456,845	279,718
Reagents	800	350	1,879,372	96,337
Ancillary	900’s	453.	1,511,321	77,470
Total Connected (kW)	-	52,653	-	-
Total Consumption (kW - hr)	**-**	**-**	**312,265,268**	**$16,006,718**
Cost Per kW-hr				**$0.0513**

Note: Errors in totals may be due to rounding.

21.9.3 Reagents

Consumption rates were determined from the metallurgical test data or industry practice. Reagents prices were supplied by Bear Creek Mining from local sources in the area with an allowance for freight to site.

A summary of process reagent consumption and costs are shown in Table 21-27.

Table 21-27: Summary of Reagents

Process Reagent (Consumption Basis)	Consumption		Unit Rate	
	kg/tonne ore	Ave. kg/year	$/kg	Ave. Ann. Cost
Sodium Isopropyl Xanthate (SIPX)	0.04	305,996	2.25	$688,491
Lime	3.5	26,774,660	0.17	$4,551,692
Methyl Isobutyl Carbinol (MIBC)	0.05	382,495	3.05	$1,166,610
AP 404 Promoter	0.015	114,749	2.40	$275,397
Sodium Cyanide	0.21	1,606,480	2.53	$4,064,393
Copper Sulfate	0.29	2,218,472	2.55	$5,657,103
Sodium Hydroxide	0.01	76,499	0.89	$68,084
Sodium Sulfite	0.505	3,863,201	0.85	$3,283,721
Zinc Sulfate	0.62	4,742,940	0.87	$4,126,358
Antiscalant	0.005	38,250	2.75	$105,186
Flocculent (thickening)	0.02	152,998	4.05	$619,642

21.9.4 Maintenance Wear Parts and Consumables

Grinding media consumption and wear items (liners) were based on industry practice for the crusher and grinding operations. These consumption rates and unit prices are shown in Table 21-28.

Table 21-28: Grinding Media and Wear Parts

Process Plant Ore Tonnes (Annual Production) 7,875,000				
	Consumption		Unit Rate	
Grinding Media & Wear Parts	**kg/tonne ore**	**Ave. kg/year**	**$/kg**	**Ave. Ann. Cost**
Primary Crusher Liners	0.008	61,199	4.30	$263,157
SAG Mill Liners	0.050	382,495	2.18	$833,839
Ball Mill Liners	0.030	229,497	2.62	$601,282
Lead Regrind Vertimill Liners	0.005	38,250	4.13	$157,970
Zinc Regrind Vertimill Liners	0.005	38,250	4.13	$157,970
SAG Mill - Balls	0.500	3,824,951	1.34	$5,125,435
Ball Mill - Balls	0.500	3,824,951	1.24	$4,742,940
Lead Regrind Vertimill - Balls	0.010	76,499	1.28	$97,919
Zinc Regrind Vertimill - Balls	0.010	76,499	1.28	$97,919

Allowances were made to cover the cost of maintenance of all items that were not specifically identified and to cover the cost of maintenance of the facilities. The allowance was calculated using the direct capital cost of equipment multiplied by five percent for each area, which totaled approximately $3.2 million for year.

21.9.5 Process Supplies & Services

Allowances were provided in process plant for outside consultants, outside contractors, vehicle maintenance, and miscellaneous supplies. The allowances were estimated using M3's information from other operations and projects. The estimated cost per year is $1.8 million.

Laboratory

Laboratory costs estimates are based on labor and fringe benefits, power, reagents, assay consumables, and supplies and services. The laboratory costs are summarized in Table 21-29. The laboratory labor cost is based on a staff of 20 for an annual cost of $943,325.

All other laboratory costs were developed as allowances based on M3's information from other projects and guidance from Bear Creek Mining.

Table 21-29: Laboratory Cost

Cost Item	LOM Cost	$/tonne
Labor & Fringes	$10,977,498	0.080
Power Allocation Ancillary Facilities	$76,256	0.001
Reagents & Fuel	$288,000	0.002
Assay Consumables	$4,140,000	0.030
Wear & Maintenance Parts	$891,000	0.006
Maintenance Labor, Fringes, and Allocations (2%)	$514,846	0.004
Supplies and Services	$90,000	0.001
Total Laboratory Cost	**$16,977,600**	**$0.123**

21.10 CONCENTRATE HANDLING, TRANSPORTATION AND STORAGE

Concentrate will be loaded into supersacks and then loaded onto trucks, which will haul the supersacks to the port of Maturani where it will be stored and loaded into a ship to be taken to the smelter for further processing. The truck transportation and port costs are estimated to be $40.00 per wet metric tonne. The shipping cost is estimated to $90.00 per wet metric tonne. A total of $195.7 million is estimated to be paid for concentrate transportation for the life of the mine.

21.11 GENERAL SERVICES AND ADMINISTRATION (G&A)

The operating cost for the General Administration areas were determined and summarized by cost element. The cost elements include labor (136 employees), supplies, support infrastructure, services, and other expenses. In addition to these cost a contingency was added in the amount of $1.0 million. The departments included are as follows:

- Administration
- Controller's
- Human Resources
- Purchasing
- Safety & Environmental

Table 21-30 shows a detail schedule of the G&A area showing the annual cost by category.

Table 21-30: G&A Costs by Area

Cost Item	LOM Cost	$/tonne
Labor & Fringes	$55,601,424	0.404
Power	$4,681,949	0.034
Vehicle Operating & Maintenance	$13,050,000	0.095
Communications	$2,727,000	0.020
Safety Supplies / Incentives	$3,240,000	0.024
Offsite Training & Conferences	$648,000	0.005
Insurance	$19,332,000	0.140
Corporate Services and Travel	$16,092,000	0.117
Environmental	$1,944,000	0.014
Security & Medical	$4,455,000	0.032
Professional Membership Costs	$108,000	0.001
Community Development	$5,400,000	0.039
Bussing (150 weekly and 40 per day)	$4,032,000	0.029
Staff Living Expenses (250 people at the camp)	$29,565,000	0.215
Consultants	$1,215,000	0.009
Computer Equipment/Software	$675,000	0.005
Misc. Office Supplies	$324,000	0.002
Misc. Freight & Couriers	$324,000	0.002
Recruiting and Relocation	$3,564,000	0.026
Mine Access Road Maintenance	$7,042,140	0.051
Legal, Permits, Fees	$4,950,000	0.036
Contingency (10%)	$17,902,460	0.130
Total General & Administrative Cost	**$196,872,972**	**$1.430**

21.12 PROGRESSIVE RECLAMATION AND CLOSURE COST

The closure activities are summarized in Section 20. The following section outlines the cost estimation methods. All closure costs have been integrated into the economic model.

21.12.1 Methods

Various methods have been employed for assessing mining closure and reclamation costs. The methods fall into categories based on activity type.

- Earthmoving;
- Demolition and Salvage;
- Erosion-Control and Reclamation; and
- Water management and water treatment.

Each of the methods will be discussed in subsections below.

21.12.1.1 Earthmoving

63% of gross (undiscounted) closure and reclamation costs are related to earthmoving. The methods to assess earthmoving reclamation costs are based on the same direct engineering calculations of vehicle productivity utilized in the mine plan with the existing mine fleet (See Section 21.8.3).

21.12.1.2 Demolition and Salvage

Demolition costs for metal buildings are assumed to be offset by the salvage value of the material taken from the buildings. This is consistent with current experience at other mine sites. The demotion of concrete pads involves the destruction of the pads in-place and their burial in roughly the same location. The degree pad break-up is assumed to establish a natural shallow groundwater flow path. The demolition costs were taken from engineering cost estimation handbooks (RS Means, 2014) adjusted by M3 for Peruvian labor rates. Salvage value is assessed at 5% of the plant Capex.

21.12.1.3 Erosion Control and Reclamation

Hydro seeding costs were taken from the approved closure plan (Walsh, 2012). Cost to relieve compaction (disking) was taken from the Arizona mining reclamation cost guidelines (Brown and Caldwell, 2005), and erosion control costs were estimated from professional experience.

21.12.1.4 Water Management and ARD Treatment

Water management structures upon closure were based on the same rates for excavation, riprap, etc. as used in the operations costs. The ARD treatment costs were estimated by the direct engineering calculation of the costs to convert the flotation plant to an ARD treatment plant. The change-over utilizes many of the same tanks, clarifiers, mixers, etc. ARD treatment reagent costs and sludge management costs were based on the same cost assumptions for hydrated lime within the plant. Labor rates for the treatment plant were taken from the labor costs for the operating mine.

21.12.2 Reclamation Cost Timetable

The reclamation costs can be divided into three chronological categories: concurrent reclamation, final reclamation, and post closure costs.

21.12.2.1 Concurrent Reclamation

Concurrent reclamation occurs during the life of the mine. Concurrent closure costs are assigned in the cost model to the year in which they occur. For example, the placement of the Este backfill cover occurs from Years 7-9.

21.12.2.2 Final Reclamation

Final reclamation involves the intensive effort after mineral processing ceases. The largest expenses are backfilling the minas and main pits followed by the placement of closure covers over the Main Dump, the pit backfill, and disturbed ground. At this time, the concrete pads are broken and buried and portions of the process plant are converted for water treatment.

21.12.2.3 Post-closure

The post closure phase includes all tasks related to long-term surveillance and monitoring and post-closure water treatment. Costs are higher in the first years after mine closure due to the need to ensure that revegetation goals and water quality goals are met. After five years, the costs decrease assuming that the post-closure landforms are stabilized and revegetated. The largest post-closure effort is the water treatment plant, which is assumed to operate long-term.

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

The financial evaluation presents the determination of the net present value (NPV), payback period (time in years to recapture the initial capital investment), and the internal rate of return (IRR) for the project. Annual cash flow projections were estimated over the life of the mine based on the estimates of capital expenditures, production cost and sales revenue. Revenues are based on the production of a zinc concentrate and a lead-silver concentrate, although the zinc concentrate also contains a significant amount of silver. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

22.2 MINE PRODUCTION STATISTICS

Mine production is reported as ore and waste from the mining operation. The annual production figures were obtained from the mine plan as reported earlier in this report. A total of 137.7 million tonnes of ore are mined at an average grade of 51.6 g/t silver, 0.91% lead, and 0.59% zinc. A total of 231.5 million tonnes of waste are mined for a stripping ratio of 1.68:1.

22.3 PLANT PRODUCTION STATISTICS

The design basis for the process plant is 22,500 tonnes per day at 92% mill availability. The metal recoveries which are variable by ore characteristics are projected to average 60.1% for zinc, 62.8% for lead and 71.9% for silver.

The estimated life of mine metal production is presented in Table 22-1.

Table 22-1: Metal Production

	Life of Mine
Zinc (million lbs)	1,071.7
Lead (million lbs)	1,738.8
Silver (million ozs)	164.3

Note: production values are prior to smelter deductions

22.2 SMELTER AND REFINERY RETURN FACTORS

The lead and zinc concentrates will be transported to a holding facility at the port of Matarani, and consolidated for shipment to a smelter for final processing. Smelter treatment and refining charges will be negotiated at the time of the finalization of the sales agreements.

The smelter charges used in the financial model are presented in Table 22-2 and Table 22-3.

Table 22-2: Smelter Treatment Factors (Lead Concentrate)

Lead Concentrate	
Payable lead	95.0%
Minimum Deduction (%)	0%
Payable silver	95.0%
Ag Minimum Deduction (oz/dmt)	1.61
Treatment charge ($/dmt)	$175.00

Lead Concentrate	
Refining charge – Ag ($/payable oz.)	$0.50
Lead Concentrate Transportation	
Concentrate Trucking and port ($/wmt)	$40.00
Concentrate Shipping ($/wmt)	$90.00
Moisture	8.0%
Penalties	
Antimony $/dmt per 0.10% over 0.50%	$2.00
Arsenic $/dmt per 0.10% over 0.50%	$2.00
Bismuth $/dmt per 0.01% over 0.5%	$1.00
Mercury – $/dmt per 10 ppm over 100 ppm	$1.00
Zinc $/dmt per 1% over 8%	$1.00

Table 22-3: Smelter Treatment Factors (Zinc Concentrate)

Zinc Concentrate	
Payable zinc	85.0 %
Minimum Deduction (%)	8.0 %
Price Participation Basis ($/tonne metal)	%2,205
Price Participation between $3,000 - $3,500 - $/dmt	$0.03
Price Participation between $2,500 - $3,000 - $/dmt	$0.06
Price Participation between $2,500 - $2,000 – $/dmt	-$0.04
Price Participation between $2,000 - $1,500 - $/dmt	-$0.02
Payable silver (% of balance)	70.0 %
Silver Minimum Deduction (oz/dmt)	3.5
Treatment charge ($/dmt)	$229.00
Refining charge – Ag (% of metal price)	0.0 %
Zinc Concentrate Transportation	
Concentrate Trucking and port ($/wmt)	$40.00
Concentrate Shipping ($/wmt)	$90.00
Moisture	8.0%
Penalties	
Arsenic $/dmt per 0.10% over 0.1%	$2.00
Cadmium $/dmt per 0.10% over 0.4%	$1.00
Iron $/dmt per 1.00% over 8.00%	$1.50
Mercury – $/dmt per 10 ppm over 30 ppm less than 100 ppm	$0.30
Mercury – $/dmt per 10 ppm over 100 ppm	$0.50
Silica $/dmt per 1% over 0.5%, >4% may be unacceptable	$0.50

22.3 CAPITAL EXPENDITURE

22.3.1 Initial Capital

The base case financial indicators have been determined using the assumption of 100% equity financing of the initial capital. The total initial capital estimate for the project, which includes pre-production mine development, construction, owners' costs and contingency is $625.1 million. Approximately 88% of these expenditures will be incurred over a two-year period.

The initial capital is presented in Table 22-4.

Table 22-4: Initial Capital

Item	$ in millions
Mining	$60.7
Process Plant	$461.3
Owner's Cost	$103.2
Total	**$625.1**

22.3.2 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 $38.99 million, and is expected to be spent over the 18-year mine life as presented in Table 22-5.

Table 22-5: Sustaining Capital ($ million)

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Mining Equipment	$ -	$ -	$ -	$13.05	$ -	$ -	$0.08	$ -	$0.05	$0.44
Process Plant	$ -	$0.50	$0.50	$0.50	$0.65	$4.83	$7.40	$0.50	$0.65	$0.50
Total	**$ -**	**$0.50**	**$0.50**	**$13.55**	**$0.65**	**$4.83**	**$7.48**	**$0.50**	**$0.70**	**$0.94**

	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Total
Mining Equipment	$0.98	$2.53	$0.08	$0.37	$0.42	$0.51	$0.52	$0.17	$0.82	$18.81
Process Plant	$0.65	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$0.50	$ -	$20.18
Total	**$1.63**	**$3.03**	**$0.58**	**$0.87**	**$0.92**	**$1.01**	**$0.55**	**$0.67**	**0.82**	**$38.99**

22.3.3 Working Capital

A 60-day delay of revenue recognition until receipt of cash has been used for accounts receivables. A delay of payment for accounts payable of 30 days is also incorporated into the financial model. In addition, a working capital allowance of $3.5 million for plant parts and supplies inventory is estimated for year -1. Parts and supplies inventory is calculated using total equipment cost for processing/surface equipment at a 5% factor.

The financial model assumes that the project will qualify for early recovery of Peruvian value added taxes (IGV) during the development and construction phase of the project. Therefore, the financial model reflects a 90-day delay between initial payment and subsequent recovery of 18% Peruvian value added (IGV) taxes. Also included in the working accounts is a stockpile inventory adjustment. All the working capital is recaptured at the end of the mine life and the final value of these accounts is $0.

22.3.4 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. Revenue is the gross value of payable metals sold before treatment charges and transportation charges. The metal price assumptions used in the economic model are as follows:

Zinc	$1.00/lb
Lead	$0.95/lb
Silver	$20.00/oz

22.3.5 Total Operating Cost

The average Total Operating Cost over the life of the mine is estimated to be $21.90 per tonne of ore processed, excluding the cost of the capitalized pre-stripping. Total Operating Cost includes mine operations, process plant operations, general and administrative cost, smelting and refining charges and shipping charges. Table 22-6 shows the estimated operating cost by area per tonne of ore processed.

Table 22-6: Life of Mine Operating Cost

Operating Cost	$/ore tonne
Mine	$5.19
Process Plant	$8.76
General Administration & Laboratoy	$1.55
Smelting/Refining Treatment & Concentrate Transport	$6.40
Total Operating Cost	$21.90

Note: Total shown in table is inconsistent because of rounding of the inputs

22.4 TOTAL CASH COST

The average Total Cash Cost over the life of the mine is estimated to be $3.80 per ounce of payable silver, including reclamation and net of lead and zinc credits. Total Cash Cost for the project is summarized in Table 22-7 below.

Table 22-7: Life of Mine Total Cash Cost

Direct Mining Costs	$2,135,293
Transportation and Refining Charges	$881,731
Subtotal	**$3,017,024**
Lead Payable Revenue	($1,569,256)
Zinc Payable Revenue	($909,579)
Total Cash Cost, Net of Lead and Zinc Revenues	**$538,189**
Reclamation	$36,292
Total Cash Cost, Including Reclamation	**$574,481**
Payable Silver Ounces	151,048
Total Cash Cost per Ounce of Payable Silver, Net of Lead and Zinc	**$3.56**
Total Cash Cost per Ounce of Payable Silver, Including Reclamation and Net of Lead and Zinc Revenues, and Reclamation	**$3.80**

22.4.1 Salvage Value

A $6.9 million allowance for salvage value has been included in the cash flow analysis.

22.4.2 Reclamation & Closure

An allowance of $36.3 million for the cost of reclamation and closure of the property has been included in the cash flow projection.

22.5 DEPRECIATION

Depreciation was calculated using the following assumptions for both initial and sustaining capital.

- Mine capital 5 year straight line method
- Process Plant and Infrastructure 10 year straight line method
- Last year of production is the catch up year if assets are not fully depreciated

22.6 TAXATION

22.6.1 Royalty Tax

The royalty tax is applied to operating profit at progressive rates from 1% to 12% based on operating margin (operating profit divided by sales), subject to a minimum tax of 1% of sales which is applicable regardless of the Company's operating margin. It is estimated that $46.2 million of royalty tax will be paid during the life of the mine.

22.6.2 Special Tax (IEM)

A special tax (IEM) is applied to operating profit at progressive rates from 2% to 8.4% based on operating margin (operating profit divided by sales). It is estimated that $28.1 million of special tax will be paid during the life of the mine.

22.6.3 Worker's Participation Tax

A labor profit sharing tax is generally based on pre-tax profits, after deduction for the royalty tax and special tax (IEM), and is assessed at an 8% rate. It is estimated that $79.0 million of labor profit sharing tax will be paid during the life of the mine.

22.6.4 Income Tax

Income taxes are assessed on pre-tax profits, after deduction for the special tax (IEM), royalty tax and worker's participation tax, at a rate of 26%. It is estimated that $573.4 million of income taxes will be paid during the life of the mine.

22.7 NET INCOME AFTER TAX

Net income after taxes for the project amounts to $1,188.3 million.

22.8 PROJECT FINANCING

The financial model has been prepared on the assumption that the project will be financed 100% with equity.

22.9 NET PRESENT VALUE, INTERNAL RATE OF RETURN, PAYBACK

The economic analyses for the project are summarized below in Table 22-8.

Table 22-8: Financial Analysis Results

($millions)	Pre-Tax	After Tax
NPV @ 0% ($000)	$1,785	$1,212
NPV @ 5% ($000)	$1,015	$644
IRR	27.90%	20.60%
Payback (Years)	2.9	3.6

22.10 SENSITIVITY ANALYSIS

The results of the sensitivity analysis for the project both before taxes and after taxes are shown on Table 22-9 to Table 22-12 and Figure 22-1 to Figure 22-4.

Table 22-9: NPV Sensitivity Analysis @ 5% - Before Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	$1,707,042	$759,748	$899,973	$1,598,087
10%	$1,361,281	$887,633	$957,746	$1,306,803
0%	$1,015,519	$1,015,519	$1,015,519	$1,015,519
-10%	$669,757	$1,143,404	$1,073,292	$724,235
-20%	$323,995	$1,271,290	$1,131,065	$432,950

Table 22-10: IRR% Sensitivity Analysis – Before Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	38.7%	23.8%	22.7%	37.2%
10%	33.5%	25.9%	25.1%	32.7%
0%	27.9%	27.9%	27.9%	27.9%
-10%	21.6%	29.8%	31.1%	22.7%
-20%	14.3%	31.6%	35.0%	16.7%

Table 22-11: NPV Sensitivity Analysis @ 5% - After Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	$1,119,086	$462,823	$535,329	$1,045,112
10%	$882,180	$554,177	$592,897	$844,712
0%	$643,643	$643,643	$643,643	$643,643
-10%	$403,557	$733,003	$693,687	$441,929
-20%	$151,349	$822,087	$742,962	$230,522

Table 22-12: IRR% Sensitivity Analysis – After Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	29.1%	17.3%	16.2%	27.9%
10%	25.0%	19.0%	18.3%	24.4%
0%	20.6%	20.6%	20.6%	20.6%
-10%	15.7%	22.2%	23.4%	16.5%
-20%	9.6%	23.6%	26.7%	11.7%

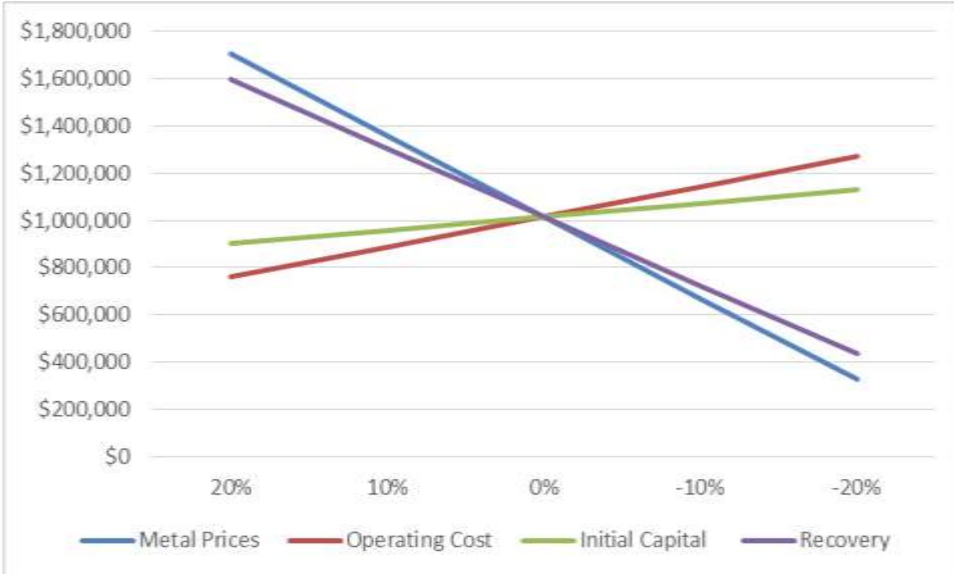


Figure 22-1: Sensitivity Analysis on NPV (before tax) at 5%

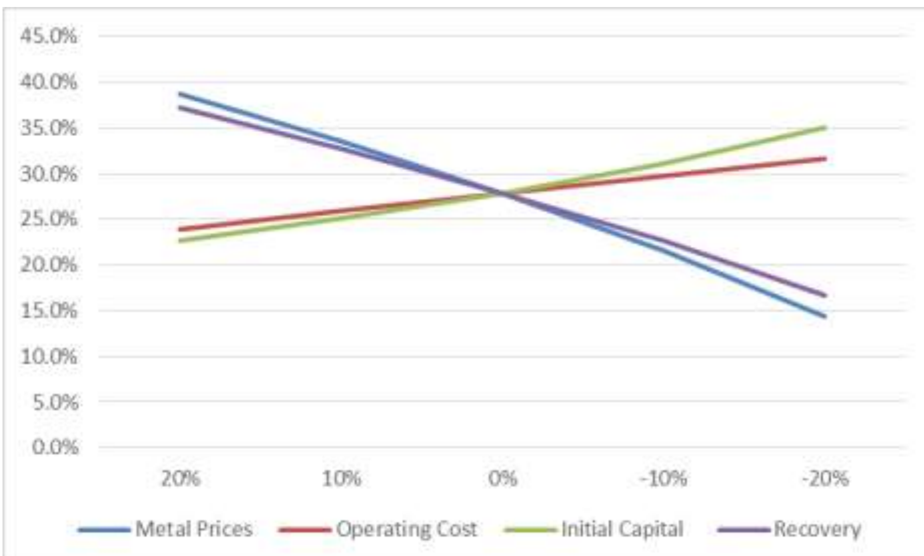


Figure 22-2: Sensitivity Analysis on IRR% (before tax)

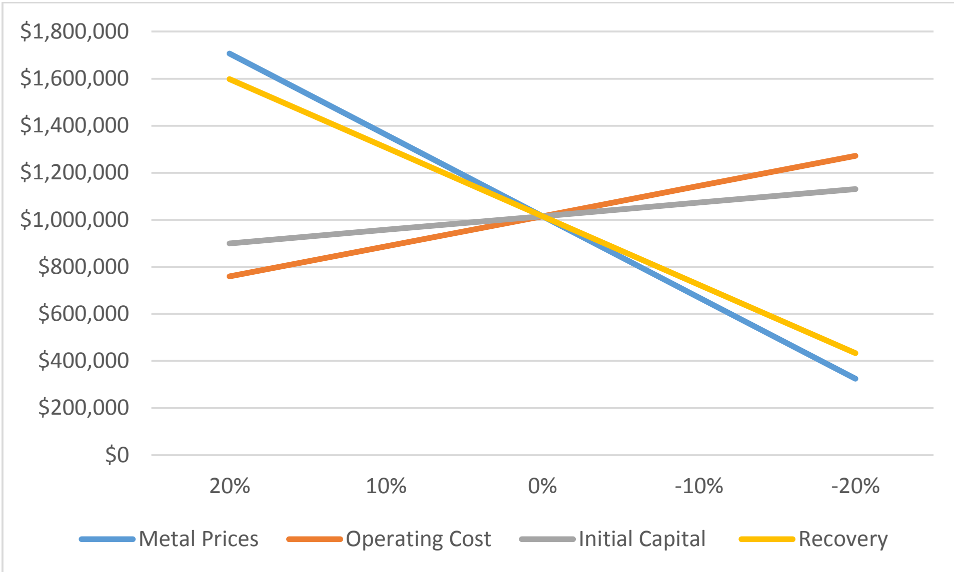


Figure 22-3: Sensitivity Analysis on NPV (after tax) at 5%

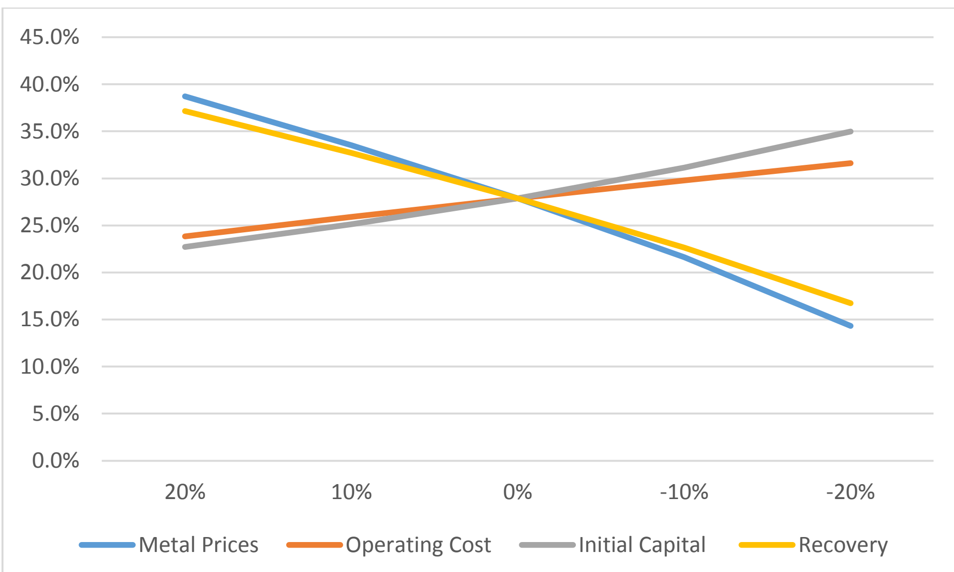


Figure 22-4: Sensitivity on IRR% (after tax)

Table 22-13: Detail Financial Model

22,500 tpd	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Mining Operations																						
Ore to Mill																						
Beginning Inventory (kt)	137,698	137,698	137,698	137,698	137,698	132,023	124,279	116,383	108,527	100,637	92,791	84,870	76,995	69,120	61,245	53,459	45,584	37,709	29,834	21,893	14,018	6,143
Mined (kt)	137,698	-	-	-	5,675	7,744	7,897	7,856	7,890	7,847	7,921	7,875	7,875	7,875	7,786	7,875	7,875	7,875	7,941	7,875	7,875	6,143
Ending Inventory (kt)	-	137,698	137,698	137,698	132,023	124,279	116,383	108,527	100,637	92,791	84,870	76,995	69,120	61,245	53,459	45,584	37,709	29,834	21,893	14,018	6,143	0
Silver Grade (g/t)	51.59	-	-	-	95.58	84.36	73.27	81.06	79.91	59.41	62.85	63.02	47.31	42.84	40.93	34.66	27.69	26.18	28.12	36.17	28.08	23.82
Zinc Grade (%)	0.588%	0.000%	0.000%	0.000%	0.461%	0.877%	0.846%	0.951%	0.995%	0.725%	0.252%	0.220%	0.322%	0.388%	0.425%	0.550%	0.654%	0.630%	0.632%	0.482%	0.529%	0.597%
Lead Grade (%)	0.912%	0.000%	0.000%	0.000%	1.169%	1.432%	1.197%	1.118%	1.129%	0.982%	1.039%	1.076%	0.961%	0.833%	0.761%	0.732%	0.637%	0.610%	0.579%	0.607%	0.723%	0.907%
Contained Silver (kozs)	228,373	-	-	-	17,440	21,003	18,603	20,473	20,270	14,988	16,006	15,956	11,978	10,846	10,244	8,776	7,010	6,627	7,180	9,158	7,110	4,705
Contained Zinc (klbs)	1,784,141	-	-	-	60,181	149,661	147,565	164,783	173,035	125,375	43,970	38,254	55,841	67,380	72,674	95,462	113,460	109,396	110,661	83,615	91,826	80,800
Contained Lead (klbs)	2,767,819	-	-	-	146,294	244,498	208,415	193,597	196,328	169,875	181,376	186,602	166,843	144,581	130,633	127,119	110,564	105,955	101,389	105,393	125,473	122,882
Waste																						
Beginning Inventory(kt)	231,521	231,521	231,521	231,521	208,699	179,305	159,795	148,498	138,889	130,645	113,700	98,314	84,845	77,889	64,067	52,761	45,048	36,351	28,227	17,341	6,854	3,711
Mined (kt)	231,521	-	-	22,822	29,394	19,511	11,296	9,609	8,244	16,945	15,386	13,468	6,956	13,822	11,305	7,714	8,697	8,124	10,885	10,488	3,143	3,711
Ending Inventory (kt)	-	231,521	231,521	208,699	179,305	159,795	148,498	138,889	130,645	113,700	98,314	84,845	77,889	64,067	52,761	45,048	36,351	28,227	17,341	6,854	3,711	(0)
Total Material Mined (kt)	369,220	-	-	22,822	35,069	27,255	19,193	17,465	16,133	24,792	23,307	21,343	14,831	21,697	19,091	15,589	16,572	15,999	18,826	18,363	11,018	9,854
Waste to Ore Ratio	1.68	-	-	-	5.18	2.52	1.43	1.22	1.04	2.16	1.94	1.71	0.88	1.76	1.45	0.98	1.10	1.03	1.37	1.33	0.40	0.60
Process Plant Operations																						
Mixed Sulfide Ore Milled																						
Ore Processed (kt)	137,698	-	-	-	5,675	7,744	7,897	7,856	7,890	7,847	7,921	7,875	7,875	7,875	7,786	7,875	7,875	7,875	7,941	7,875	7,875	6,143
Silver Grade (g/t)	51.59	-	-	-	95.58	84.36	73.27	81.06	79.91	59.41	62.85	63.02	47.31	42.84	40.93	34.66	27.69	26.18	28.12	36.17	28.08	23.82
Zinc Grade (%)	0.59%	0.00%	0.00%	0.00%	0.46%	0.88%	0.85%	0.95%	0.99%	0.72%	0.25%	0.22%	0.32%	0.39%	0.42%	0.55%	0.65%	0.63%	0.63%	0.48%	0.53%	0.60%
Lead Grade (%)	0.91%	0.00%	0.00%	0.00%	1.17%	1.43%	1.20%	1.12%	1.13%	0.98%	1.04%	1.07%	0.96%	0.83%	0.76%	0.73%	0.64%	0.61%	0.58%	0.61%	0.72%	0.91%
Contained Silver (kozs)	228,373	-	-	-	17,440	21,003	18,603	20,473	20,270	14,988	16,006	15,956	11,978	10,846	10,244	8,776	7,010	6,627	7,180	9,158	7,110	4,705
Contained Zinc (klbs)	1,784,141	-	-	-	60,181	149,661	147,565	164,783	173,035	125,375	43,970	38,254	55,841	67,380	72,874	95,462	113,460	109,396	110,661	83,615	91,826	80,800
Contained Lead (klbs)	2,767,819	-	-	-	146,294	244,498	208,415	193,597	196,328	169,875	181,376	186,602	166,843	144,581	130,633	127,119	110,564	105,955	101,389	105,393	125,473	122,882
Total Zinc Concentrate																						
Zinc Concentrate (kt)	919	-	-	-	28	81	87	107	112	76	11	10	16	25	30	43	54	53	57	45	39	45
Zinc Concentrate Grade (%)	52.89%	0.00%	0.00%	0.00%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%	52.89%
Lead Concentrate Silver Grade (oz/t)	12.0	-	-	-	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.4	12.3	10.0	8.4	12.4	11.7
Recovered Silver (kozs)	11,036	-	-	-	347	1,003	1,074	1,329	1,390	941	140	123	202	310	365	527	670	654	576	376	484	526
Recovered Zinc (klbs)	1,071,696	-	-	-	32,643	94,476	101,182	125,157	130,935	88,647	13,180	11,541	18,987	29,187	34,423	49,613	63,151	61,957	66,884	51,687	45,584	52,261
Total Lead Concentrate																						
Lead Concentrate (kt)	1,364	-	-	-	63	135	119	114	118	87	55	39	60	65	59	64	70	69	64	58	78	77
Lead Concentrate Grade (%)	56.58%	0.00%	0.00%	0.00%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%	56.58%
Lead Concentrate Silver Grade (oz/t)	109.9	-	-	-	185.6	108.7	109.0	126.7	121.8	114.8	174.6	228.5	125.7	108.3	113.7	91.7	69.7	67.4	79.7	111.1	65.6	41.6
Recovered Silver (kozs)	153,237	-	-	-	11,750	14,669	12,956	14,466	14,352	9,970	9,634	8,885	7,481	7,067	6,739	5,843	4,899	4,674	5,074	6,475	5,118	3,185
Recovered Lead (klbs)	1,738,788	-	-	-	78,988	168,273	148,271	142,455	146,974	108,306	68,832	48,498	74,236	81,388	73,945	79,469	87,738	86,479	79,379	72,727	97,286	95,545
Payable Metals																						
Zinc Concentrate																						
Payable Silver (kozs)	5,473	-	-	-	174	504	539	667	698	473	70	62	101	156	183	264	337	328	263	154	243	258
Payable Zinc (klbs)	909,579	-	-	-	27,705	80,184	85,876	106,225	111,128	75,238	11,186	9,795	16,115	24,772	29,216	42,108	53,598	52,585	56,766	44,038	38,688	44,355
Lead Concentrates																						
Payable Silver (kozs)	145,575	-	-	-	11,162	13,935	12,308	13,743	13,634	9,472	9,152	8,440	7,107	6,713	6,402	5,551	4,654	4,441	4,821	6,151	4,862	3,025
Payable Lead (klbs)	1,651,849	-	-	-	75,038	159,859	140,857	135,333	139,625	102,891	65,390	46,073	70,524	77,319	70,248	75,495	83,351	82,155	75,410	69,091	92,421	90,768
Income Statement ($000)																						
Zinc ($/lb)	$ 1.00	$ -	$ -	$ -	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00	$ 1.00
Lead ($/lb)	$ 0.95	$ -	$ -	$ -	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95	$ 0.95
Silver ($/oz)	$ 20.00	$ -	$ -	$ -	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00	$ 20.00
Revenues																						
Zinc Concentrates - Zn	909,579	-	-	-	27,705	80,184	85,876	106,225	111,128	75,238	11,186	9,795	16,115	24,772	29,216	42,106	53,598	52,585	56,766	44,038	38,688	44,355
Zinc Concentrates - Ag	109,463	-	-	-	3,480	10,072	10,787	13,342	13,958	9,450	1,405	1,230	2,024	3,111	3,670	5,288	6,732	6,553	5,257	3,077	4,859	5,166
Lead Concentrates - Pb	1,569,256	-	-	-	71,286	151,866	133,814	128,566	132,644	97,746	62,120	43,769	66,998	73,453	66,736	71,721	79,184	78,048	71,640	65,636	87,800	86,229
Lead Concentrates - Ag	2,911,496	-	-	-	223,242	278,706	246,169	274,861	272,682	189,437	183,047	168,810	142,135	134,270	128,049	111,013	93,078	88,814	96,414	123,017	97,244	60,507
Total Revenues	5,499,794	-	-	-	325,714	520,829	476,646	522,995	530,412	371,871	257,758	223,604	227,272	235,606	227,670	230,131	232,593	226,000	230,078	235,769	228,591	196,257
Operating Cost																						
Mining	714,528	-	-	-	67,975	57,515	50,852	47,580	33,435	44,786	43,869	40,670	34,218	41,614	38,036	29,954	31,578	29,188	38,935	29,989	28,173	26,161
Process Plant	1,206,915	-	-	-	52,871	67,816	68,900	68,609	68,850	68,547	69,073	68,747	68,747	68,747	68,111	68,747	68,747	68,747	68,840	68,372	68,372	56,071
General Administration	213,851	-	-	-	11,805	11,884	11,890	11,888	11,889	11,888	11,891	11,889	11,889	11,889	11,885	11,889	11,889	11,889	11,891	11,889	11,889	11,829
Treatment & Refining Charges																						
Zinc Concentrates																						
Treatment Charges	210,494	-	-	-	6,411	18,556	19,873	24,582	25,717	17,411	2,589	2,267	3,729	5,733	6,761	9,745	12,404	12,169	13,137	10,191	8,953	10,265
Price Participation	(1,669)	-	-	-	(51)	(147)	(158)	(195)	(204)	(138)	(21)	(18)	(30)	(45)	(54)	(77)	(98)	(96)	(104)	(81)	(71)	(81)
Penalty	22,520	-	-	-	686	1,985	2,126	2,630	2,751	1,863	277	243	399	613	723	1,043	1,327	1,302	1,405	1,090	958	1,098
Transportation	129,054	-	-	-	3,931	11,377	12,184	15,072	15,767	10,675	1,587	1,390	2,286	3,515	4,145	5,974	7,605	7,461	8,054	6,248	5,489	6,293
Lead Concentrate																						
Treatment Charges	243,825	-	-	-	11,081	23,606	20,800	19,984	20,018	15,194	9,656	6,803	10,414	11,417	10,373	11,146	12,308	12,132	11,136	10,203	13,648	13,403
Price Participation	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Penalty	8,921	-	-	-	405	863	761	731	754	556	353	249	381	418	379	408	450	444	407	373	499	490
Silver Refining Charges	72,787				5,581	6,968	6,154	6,872	6,817	4,736	4,576	4,220	3,553	3,357	3,201	2,775	2,327	2,220	2,410	3,075	2,431	1,513
Transportation	195,698	-	-	-	8,890	18,939	16,688	16,033	16,542	12,190	7,747	5,458	8,355	9,160	8,322	8,944	9,875	9,733	8,934	8,185	10,949	10,753
Total Operating Cost	3,017,024	-	-	-	169,587	219,362	210,071	213,787	202,937	187,707	151,596	141,918	143,942	156,417	151,885	150,550	158,411	155,188	165,046	149,535	151,290	137,796

Detail Financial Model Continued

22,500 tpd	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Salvage Value	(6,903)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(6,903)
Social Cost	4,208		356	254	362	290	251	251	365	297	251	251	365	272	280	225	136	-	-	-	-	-
Reclamation & Closure	36,292	-	-	-	760	1,130	673	433	216	631	13	-	172	184	323	58	1,149	13	-	-	-	-
Total Production Cost	3,050,622	-	356	254	170,709	220,781	210,995	214,471	203,518	188,636	151,861	142,170	144,479	156,873	152,489	150,833	159,696	155,201	165,046	149,535	151,290	130,893
Operating Income	2,449,173	-	(356)	(254)	155,005	300,048	265,650	308,524	326,895	183,235	105,897	81,435	82,794	78,733	75,181	79,298	72,897	70,799	65,032	86,233	77,302	65,364
Depreciation on BC Sucursal Cost	23,241				1,376	2,201	2,014	2,210	2,241	1,571	1,089	945	960	996	962	972	983	955	972	996	966	829
Depreciation	664,119				70,632	70,682	70,732	73,392	73,457	58,020	58,777	58,827	56,293	56,431	1,979	2,468	2,485	2,547	2,527	2,000	805	2,063
Total Depreciation	687,359	-	-	-	72,009	72,883	72,746	75,602	75,698	59,592	59,866	59,772	57,253	57,426	2,941	3,441	3,468	3,502	3,500	2,997	1,771	2,893
Net Income After Depreciation	1,761,813	-	(356)	(254)	82,996	227,165	192,904	232,922	251,196	123,643	46,031	21,663	25,541	21,306	72,240	75,857	69,429	67,297	61,532	83,237	75,531	62,471
Interest Expense	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Income After Interest	1,761,813	-	(356)	(254)	82,996	227,165	192,904	232,922	251,196	123,643	46,031	21,663	25,541	21,306	72,240	75,857	69,429	67,297	61,532	83,237	75,531	62,471
Taxes	573,472	-	-	-	17,962	71,653	62,343	74,187	80,161	41,407	16,470	8,251	9,079	7,907	22,852	24,492	22,596	21,855	20,137	26,613	24,494	20,325
Total Taxes	573,472	-	-	-	17,962	71,653	62,343	74,187	80,161	41,407	16,470	8,251	9,079	7,907	22,852	24,492	22,596	21,855	20,137	26,613	24,494	20,325
Net Income After Taxes	1,188,341	-	(356)	(254)	65,034	155,511	130,560	158,734	171,035	82,236	29,561	13,412	16,461	13,399	49,388	51,365	46,833	45,442	41,395	56,624	51,037	42,146
Cash Flow																						
Operating Income after Depreciation & Interest	1,761,813	-	(356)	(254)	82,996	227,165	192,904	232,922	251,196	123,643	46,031	21,663	25,541	21,306	72,240	75,857	69,429	67,297	61,532	83,237	75,531	62,471
Add back Depreciation	687,359	-	-	-	72,009	72,883	72,746	75,602	75,698	59,592	59,866	59,772	57,253	57,426	2,941	3,441	3,468	3,502	3,500	2,997	1,771	2,893
Working Capital																						
Account Recievable (60 days)	-	-	-	-	(53,542)	(32,074)	7,263	(7,619)	(1,219)	26,062	18,758	5,614	(603)	(1,370)	1,305	(405)	(405)	1,084	(670)	(936)	1,180	5,315
Accounts Payable (30 days)	-	-	-	-	13,939	4,091	(784)	305	(892)	(1,252)	(2,968)	(795)	186	1,025	(372)	(110)	646	(265)	810	(1,275)	144	(1,109)
IGV Payment	(120,941)	(1,017)	(9,184)	(15,333)	(13,522)	(5,603)	(5,380)	(5,235)	(4,672)	(5,119)	(5,104)	(4,961)	(4,699)	(4,999)	(4,827)	(4,527)	(4,592)	(4,496)	(4,894)	(4,512)	(4,438)	(3,829)
IGV Refund	120,941		1,017	9,184	23,470	5,385	5,603	5,380	5,235	4,672	5,119	5,104	4,961	4,699	4,999	4,827	4,527	4,592	4,496	4,894	4,512	4,438
Inventory - Parts, Supplies	-	-	-	(3,489)	-	-	-	-	-	-	-	-	-	-	-	-	872	2,616	-	-	-	-
Total Working Capital	0	(1,017)	(8,166)	(9,638)	(29,655)	(28,200)	6,723	(7,169)	(1,549)	24,363	15,805	4,962	(175)	(644)	1,104	(214)	1,048	3,532	(259)	(1,828)	1,398	4,815
Debt Financing	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Capital Expenditures																						
Initial Capital																						
Mine	60,664	-	-	60,664	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Process Plant	461,284	23,064	184,513	207,578	46,128	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Owners Cost	103,180	-	23,731	79,448	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital																						
Mining	16,814	-	-	-	-	-	-	13,049	-	-	82	-	54	440	981	2,528	82	366	416	510	52	172
Process Plant	20,178	-	-	-	-	500	500	500	650	4,828	7,400	500	650	500	650	500	500	500	500	500	500	500
Total Capital Expenditures	664,119	23,064	208,245	347,650	46,128	500	500	13,549	650	4,828	7,482	500	704	940	1,631	3,028	582	866	916	1,010	552	672
Cash Flow before Taxes	1,785,054	(24,081)	(216,768)	(357,582)	79,221	271,347	271,873	287,806	324,696	202,770	114,220	85,897	81,915	77,148	74,655	76,056	73,363	73,465	63,857	83,395	78,148	69,508
Cummulative Cash Flow before Taxes		(24,081)	(240,849)	(598,431)	(519,209)	(247,862)	24,011	311,817	636,513	839,283	953,503	1,039,400	1,121,314	1,198,463	1,273,117	1,349,173	1,422,536	1,496,001	1,559,858	1,643,253	1,721,401	1,790,909
					1.0	1.0	0.9	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Taxes																						
Income Taxes	573,472	-	-	-	17,962	71,653	62,343	74,187	80,161	41,407	16,470	8,251	9,079	7,907	22,852	24,492	22,596	21,855	20,137	26,613	24,494	20,325
Cash Flow after Taxes	1,211,582	(24,081)	(216,768)	(357,582)	61,259	199,694	209,529	213,619	244,535	161,363	97,750	77,646	72,835	69,241	51,803	51,563	50,766	51,610	43,720	56,782	53,654	49,183
Cummulative Cash Flow after Taxes		(24,081)	(240,849)	(598,431)	(537,172)	(337,478)	(127,948)	85,670	330,205	491,568	589,318	666,964	739,799	809,040	860,842	912,406	963,172	1,014,782	1,058,502	1,115,284	1,168,938	1,218,121
					1.0	1.0	1.0	0.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Economic Indicators before Taxes		
NPV @ 0%	0%	$ 1,785,054
NPV @ 5%	5%	$ 1,015,519
IRR %		27.9%
Payback	Years	2.9

Economic Indicators after Taxes		
NPV @ 0%	0%	$ 1,211,582
NPV @ 5%	5%	$ 643,643
IRR %		20.6%
Payback	Years	3.6

23 ADJACENT PROPERTIES

This report focuses on the areas of the Project concessions that contain reserves / resources. There are two additional zones of mineralization within the Project concession area that were briefly described in Section 7. The Gold zone and the Antimony zone are located south of the Project resource area and are contained within the Project mineral claims.

There are no adjacent mineral properties outside of the Project claim area that have any bearing upon the Project.

In a more regional context, several exploration companies are active in the areas looking for uranium mineralization in the post mineral tuff. These companies are primarily focused on occurrences east and northeast of the Project, closer to the city of Macusani.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT EXECUTION

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 proposed 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.

24.1.1 Objectives

The project execution plan has been established with the following objectives:

- To maintain the highest standard of safety so as to minimize 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.

The project would be executed in accordance with the Execution Plan which is designed to achieve the following objectives:

- Conformance to the budget
- On-schedule completion
- Compliance with Project quality standards
- Uncompromised safety
- Inclusion of Peruvian participation
- Environmental compliance

24.1.2 Plan of Approach

24.1.2.1 Philosophy

This section describes the execution plan for advancing the Corani 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 beginning with the submission of the Feasibility Study through permitting, project financing, basic engineering, long lead procurement, contracting, 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 a Health and Safety Plan that is comprehensive yet concise so that contractors, construction managers, and members of BCM's development team are safe during the field construction phase of the project;
- Develop and execute project control procedures and processes;
- Perform constructability reviews;
- Implement project accounting and cost control best practices;
- Issue a cost control plan and a control budget; and
- Oversee project accounting.

BCM intends to utilize an Engineering, Procurement and Construction Management (EPCM) approach utilizing multiple hard money and low unit cost prime contracts for CM, as the recommended method for executing the project. The capital cost estimate is based on this methodology. Mine development pre-production work activities as well as the mine access road construction, power transmission line, and operating and construction camp construction will be performed by contractors selected through a pre-qualification and pre-tending process. Because the project is located in an area with an abundance of qualified contractors, construction is to be performed by companies from within Peru, wherever possible.

Some items affecting the project are:

- Ability to start work that does not require engineering;
- Availability of construction and engineering resources;
- 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).

As previously mentioned, M3 utilized an EPCM approach as the basis for the capital cost estimate. This approach provides for multiple contracts that would include civil, concrete, structural steel, mechanical, piping, electrical and instrumentation.

The mechanical and electrical equipment required for the project will be procured worldwide from multinational suppliers. Concrete and building construction materials will be sourced locally, wherever possible. Structural and miscellaneous steel, piping, tanks, electrical and miscellaneous process equipment will be sourced within Peru, and to the extent practical, within the region.

24.1.3 Project Organization

An internationally experienced EPCM team would be assembled to manage the development of the project. This team would develop and implement the Project Procedures Manual that would include the following information:

- Project Management Plan;
- Engineering Management Plan;
- Procurement Plan;
- Logistics and Transportation Plan;
- Construction Plan;
- Commissioning and Start up Plan;
- Quality Assurance Plan;
- Environmental, Health and Safety Plan;
- Communication Plan;
- Project Controls Plan;
- Project Schedule; and
- Project Close-Out Plan.

The EPCM contractor will provide critical project execution guidance and oversight to ensure timely project completion. Key activities will include following:

- Ensure that project operating procedures reflect the requirements of the Quality Management System and are adequate and up-to-date;

- Manage engineering, procurement and construction activities to accomplish funded activities in accordance with approved budgets and schedules;

- Ensure timely processing and disposition of budget and schedule change requests and revisions to approved budgets;

- Ensure that engineering deliverables (e.g., drawings, specifications, requisitions) comply with applicable government regulations and sound engineering practices;

- Ensure that required reviews and approvals are provided and documented;

- Conduct the project kick-off meeting. Analyze financial and execution risks to project performance and develop and implement mitigation actions. Regularly review risk status and effectiveness of risk management activities;

- Develop an integrated EPCM schedule that will plan all the major activities in accordance with contractual requirements.

Figure 24-1 is a block diagram of the ECPM project organization that is envisioned for the Corani Project.

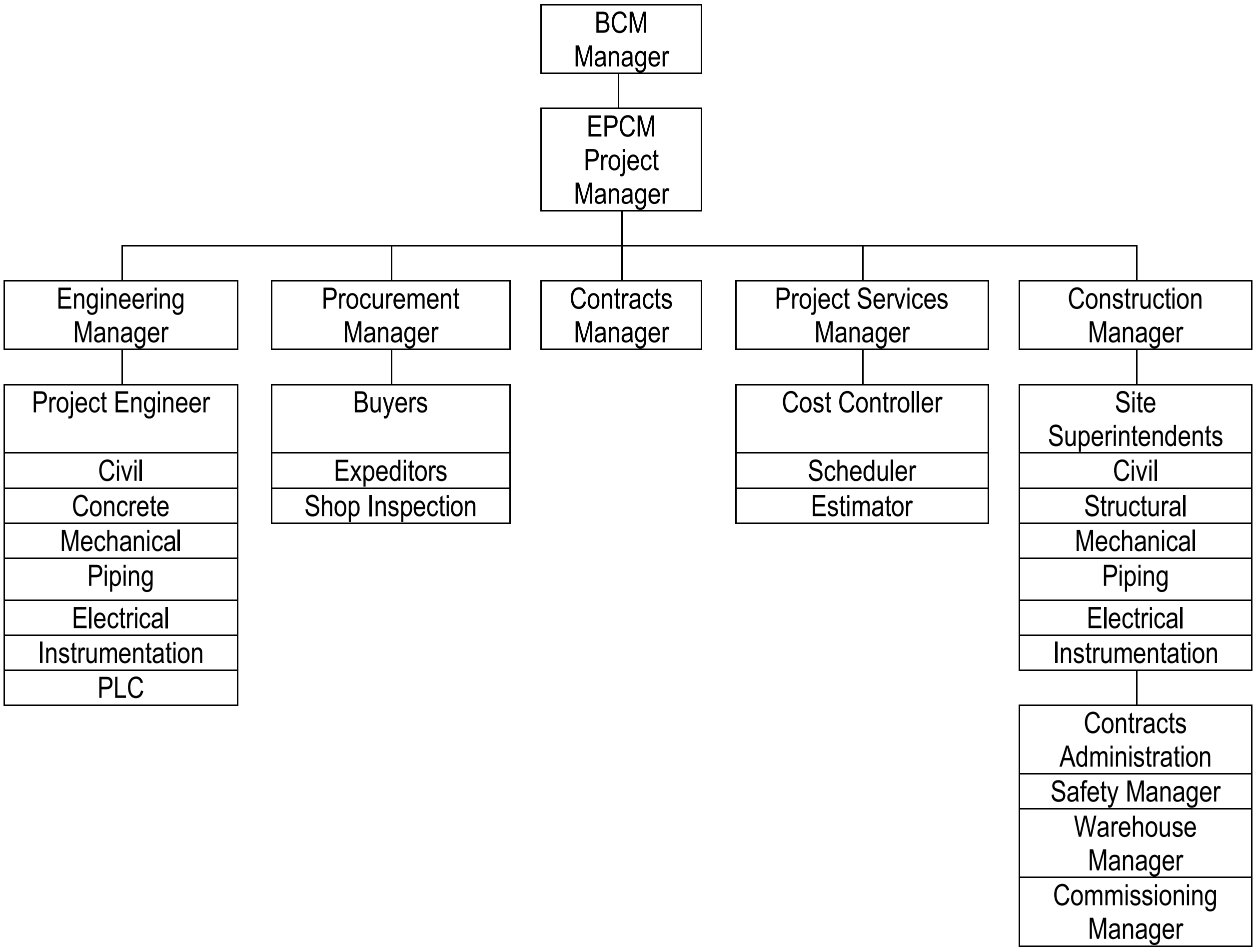


Figure 24-1: Project Organization Block Diagram

24.1.4 Project Schedule

The critical path for completion of the project begins with permitting and continues through the completion of construction activities. Concurrently, optimization of recoveries will continue through metallurgical testing through construction and during ramp-up phases. As is typical in polymetallic ore deposits, metallurgical procedures are continiously being optimized. The grinding mills, the longest-lead equipment items, are not critical if the purchase order can be placed at the end of basic engineering. If delays occur to the start of basic and detailed engineering, the mill procurement and construction will become critical. The construction of the Plant Water Supply Pond will commence immediately following the receipt of construction permits in order to capture and store a full season of precipitation for project commissioning and start up. Construction of the major earthworks for these facilities will need to be carried out primarily in the dry season, and they should be considered part of the critical path. Construction of the main Mine Access Road, haul roads, Construction Camp and Warehouse will also commence following receipt of permits.

Refer to the Gantt chart Project Schedule shown in Figure 24-2 for a breakdown of scheduled activities and milestones. A conceptual level EPC schedule was developed to identify critical project milestones. The following engineering, test work and permitting durations were developed based on consultants input, client input and historical project data. Construction durations were based on quantities and man-hours developed in the capital cost estimate:

- Basic Engineering – 6 months
- Detailed Engineering – 15 months
- Permitting – 16 months
- Major Offsite Contracts (Camp, Power Line, Access Road) – 13 months
- Mine Construction/Prestripping – 12 months
- Plant Construction – 17 months
- Commissioning and Start-Up – 4 months

The total time from receiving financing to start-up is estimated to be approximately 31 months.

Funding must be available during the basic engineering phase to purchase vendor engineering for major equipment in an effort to minimize delays in design during detailed engineering. Award for purchase of the grinding mills and tailings filters is also planned during detail engineering as their quoted lead times are approximately 52 weeks, plus six weeks added for delivery to site. Major emphasis will be placed on finalizing design criteria and finalization of major equipment performance specifications and purchase order placement. During this phase, fabrication of major equipment will likely be committed before the appropriate permits have been received. Delays in the award of purchase orders may result in delays during construction.

A sequence of effort has been developed from this study with a prospective schedule by which the project will likely proceed. The schedule includes Owner activities, engineering, contracts, long lead procurement, plant and ancillary construction, mine preparation activities, and plant commissioning activities and is presented as Figure 24-2.

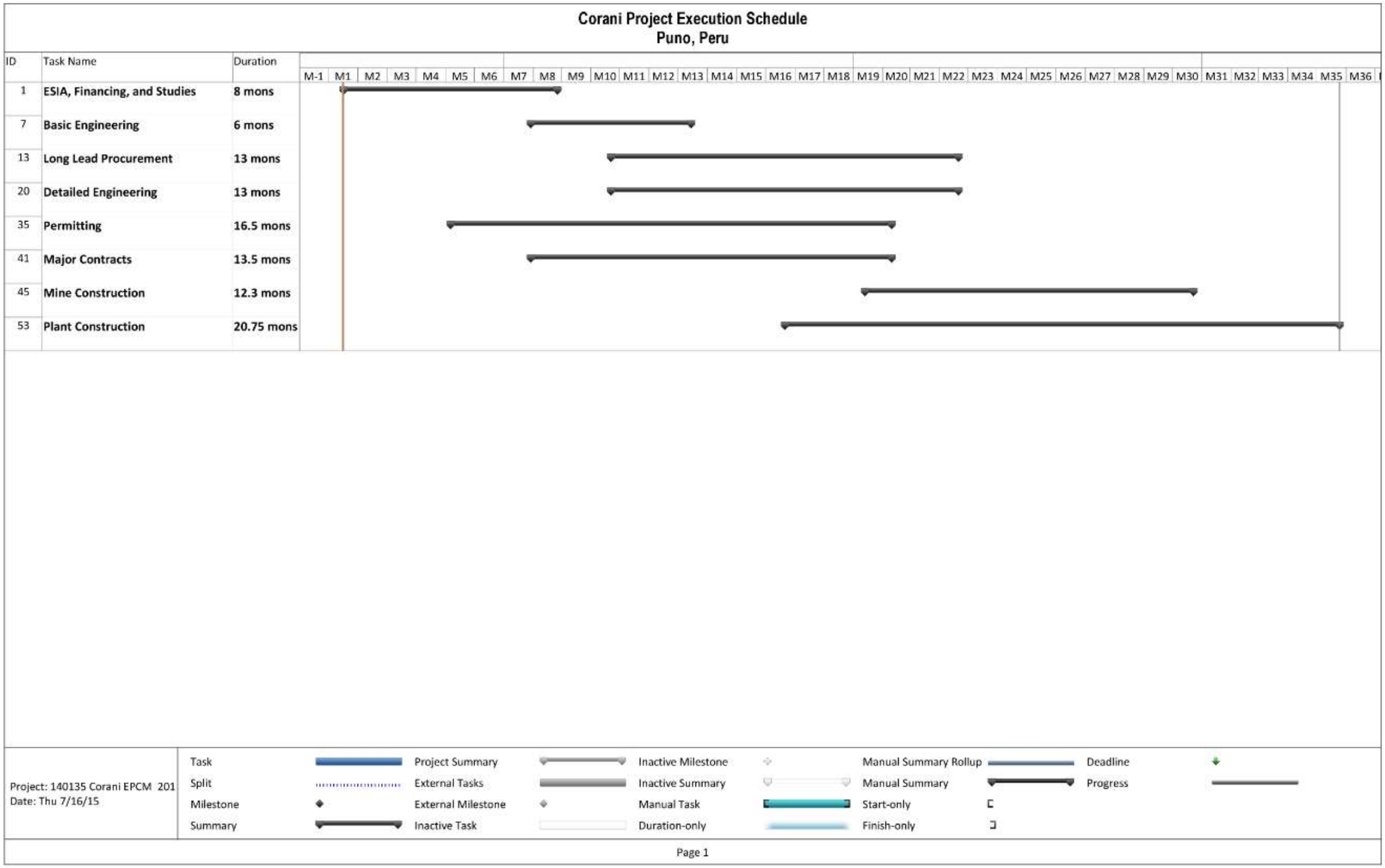


Figure 24-2: Project Schedule

24.1.5 Engineering

Project engineering would be developed in two-phases:

- A Basic Engineering phase that would confirm and expand on the feasibility designs and initiate the procurement of long-lead equipment items,

- A Detailed Engineering phase that would be carried out by a leading international engineering company following the completion of the Basic Engineering phase. As detail engineering designs and quantity take-offs are completed these would be transferred to the procurement and contracts groups for purchase and contracting and to the construction team at the project site.

24.1.5.1 Codes and Standards

Design of the Project processing facility and infrastructure requirements would be done in accordance with all applicable and acceptable Peruvian codes and standards. Where no applicable Peruvian codes or standards exist, North American codes and standards would be applied.

24.1.5.2 Electrical, Control Systems, and Instrumentation

To the greatest extent possible, major equipment components, i.e. Motor Control Centers, switchgear, etc. would be bundled into pre-engineered and prefabricated containerized units or Power Distribution Centers (PDCs) to reduce the number of on-site craft man hours. Other considerations include field instruments and controls installed and wired in I/O cabinets.

24.1.5.3 Mechanical and Piping

To the greatest extent possible, equipment and piping systems would be preassembled or modularized by the manufacturer to reduce onsite craft man hours.

24.1.5.4 Civil and Structural

All site specific conditions, such as soil conditions, subsurface conditions, surface water and other key design criteria outlined in geotechnical reports and provided in Site Design Criteria will be considered in the design packages. Structural steel will be detailed using TEKLA software. Mechanical steel will be dictated utilizing either Inventor or TEKLA. This will allow fabrication of steel prior to the award of steel installation contracts.

Engineering will be done to match the plant protocol for drawing titles, equipment numbers and area numbers. Design will produce drawings in the Metric System of Units (SI) format. Drawings and specifications will be prepared in Spanish.

A site conditions specification will be prepared to ensure that vendors are aware of the site conditions. Individual equipment specifications will be prepared.

Engineering control will be maintained through drawing lists, specification lists, equipment lists, pipeline lists, cable schedule, and instrument lists. Control of Engineering Requisitions for Quote (ERFQ) 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 Peru. Reinforcing bar will be fully detailed to allow either site or shop fabrication.

Owner review of engineering progress and design philosophy will be an ongoing process.

24.1.6 Procurement

Procurement activities include:

- Early award of purchase orders for supply of long-lead equipment, particularly crushing, grinding and mining equipment. This will enable detail engineering to proceed without delays due to lack of adequate vendor data;
- Other "semi" long lead equipment will require early issue of engineering requests for quote (ERFQs) such that return of bids and evaluations support the scheduled award designed to provide a smooth cash flow. This category includes the flotation equipment, thickeners, overland pipe line systems conveyors and large process pumps;
- A comprehensive expediting and inspection program to ensure timely delivery of vendor information and equipment. In particular, extra effort will be required in relation to supplier QA/QC, shop inspection and shop expediting. Global shop inspection service companies can be utilized; and
- Special attention would be given to the early award of purchase orders for classification equipment, electrical transformers, switchgear, and power distribution centers.

Sourcing of the majority of equipment and materials is expected to be from USA, Canada, Europe, Chile and China. Some major and minor mechanical equipment and material would be procured from Peruvian suppliers.

Due to the location and altitude of the site, pre-fabrication and skid-mounted packages would be considered to the greatest extent possible. Pre-fabricated modules would be equipped with piping and valves, wiring and instrumentation to reduce on site labor.

Procurement of long delivery equipment and materials is scheduled with their relevant engineering tasks. This will ensure that the applicable vendor information is incorporated into the design drawings and that the equipment will be delivered to site at the appropriate time 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.

24.1.7 Inspection

The EPCM contractor will be responsible for conducting 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, 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 may also be provided by the EPCM engineering design team.

24.1.8 Expediting

The EPCM contractor will also be responsible to expedite the receipt of vendor drawings to support the engineering effort as well as 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 logistics contractor will be responsible to coordinate and expedite the equipment and material shipments from point of manufacture to site, including international shipments through customs.

24.1.9 Contracting

Working with the project construction management team a detailed contracting plan indicating scope breakdown and contract type will be developed during the project detail engineering phase.

Contracting activities include:

- Development of standard contracts for construction;
- Issue of bid documents for construction contracts. The construction management team will provide input into the development of construction contracts;
- Review and verify construction completion so that progress payments can be made during execution of the contract; andExecute contract close-out activities to ensure compliance with contract scope of work and contract financial terms.

A combination of vertical, horizontal, and design-construct contracts may be employed as best suits the work to be performed, degree of engineering and scope definition available at the time of award. A concrete batch plant will be located on site that will use screened colluvial and alluvial materials native to the area. There will be a dedicated construction camp at the same site as the operations camp that will be located approximately one mile from the plant along the upper EFSFSR intersection with the mine access road.

The civil contract will cover all clearing, grubbing, bulk excavation, engineered fill, grading, and possibly, geomembrane lining of ponds and pipe trenches.

The concrete contract will include all concrete forming, rebar, placement and stripping. If possible, the batch plant will be tied to the concrete placement contract to leverage the economy of having one management for both functions.

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. Table 24-1 represents the Proposed Contract Work Package list:

Table 24-1: 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
3	Mine Access Road	Includes Roadway, Bridges, Tunnel, Drainage Culverts & Trenching
4	138 kV Power Transmission Line	Includes 13.8 kV Power Line from Corani Substation to Operations Camp
5	Construction Camp/Operations Camp Installation	Possibly by provider of modular construction camp
6	Main Substation	Includes Emergency Generator Installation & Testing
7	Mine Pre-Stripping Contract	Contractor
8	Field Electrical Distribution - Sub Station to Process Areas, Ancillary Buildings	Overhead lines and duct banks from switch gear
9	Water Supply System - Yard Water Piping	Includes Fire Suppression
10	Septic System - Sewer Piping, Plant & Leach Field	Two septic systems required: process plant area and camp area
11	Clearing, Grubbing, Site Excavation, Engineered Backfill, Grading, Trenching, - all Areas	Contractor
12	Concrete Work - All Areas	Contractor
13	Structural Steel Buildings & Platforms	From foundation bolts. Includes roofing and siding installation.
14	Architectural Finishes	In offices and larger frame structure buildings
15	Field Erected Tanks	Typically part of design-supply-erect contract.
16	Mechanical Equipment	Crusher, conveyors, reclaim feeders, grinding mills, flotation cells, thickeners, pumps, mechanical steel, etc.
17	Process Piping & Field Instrumentation	Contractor
18	Instrumentation & Controls Programming	PLC programming, HMI screen development; I/O & communications.

24.1.10 Project Services

The construction program begins with installation of essential infrastructure including the access road, power transmission line, and residential camp facilities. This preliminary construction work will be managed by BCM and is included in the Owner's Cost.

The EPCM contractor will be responsible for management and control of the various project activities and ensure that the team has appropriate resources to accomplish BCM's objectives. Initial work onsite includes clearing and grubbing of the plant site, mass earthwork for site development, project access road and in-plant roads. Concrete foundations for the process buildings and other support structures will be constructed beginning in Q4 of 2016. The grinding building, flotation building, concentrate filtration building, reagents building, and tailings filtration building are planned to be a bridge-frame metal, moment frame structures as are the truck shop and plant maintenance building. Other ancillary buildings will likely be pre-engineered metal buildings and modular buildings.

Construction work is scheduled for approximately 17months from mobilization to the commencement of commissioning. Earthworks for the mine access road will commence after the permits have been released as soon as the contractor can be mobilized to the field.

Construction Management will be done as Agent for BCM 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 team will be comprised of individuals capable of coordinating the construction effort, supervising and inspecting the work, performing field engineering functions, administering contracts, supervising warehouse and material management functions, and performing cost control and schedule control functions. These activities will be under the direction of a resident construction manager and a team of engineers, and locally hired supervisors, and technicians. There would also be a commissioning team to do final checkout of the project.

Construction progress will be measured by using quantity ledgers for construction quantities to develop percent completion and earned hours by contractors. Quantity surveyors will measure the amount of civil quantities, yards of concrete placed, tons of steel erected, and similar measures for architectural, piping and electrical quantities. Mechanical installations will be measured based on the estimated installation hours from the control estimate developed during detailed engineering.

Some site services will be contracted to third party specialists, working under the direction of the resident construction manager. Construction service contracts identified at this time include field survey and QA/QC testing services.

24.1.11 Quality Plan

A project specific, Quality Plan will be developed and implemented on the site. The Quality Plan is a management tool for the EPCM contractor, through the construction contractors, to maintain the quality of construction and installation on every aspect of a project. The plan, which consists of many different manuals and subcategories, will be developed during the engineering phase and available prior to the start of construction.

24.1.12 Commissioning Plan

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 performance.

In addition to these activities, the commissioning portion of the work will also include coordination of facilities operations training, maintenance training and turnover of all compiled commissioning documentation in an agreed form.

The commissioning and start up team is planned to be an integrated organization of plant start-up professionals.

Plant start-up would be initiated with the preparation of an overall plan for acceptance testing; safety; lock-out tag-out; compilation of instruction manuals; and supply of reagents, spare parts and supplies. Also included is process control system final check-out and training.

Commissioning includes those activities necessary for an effective transition between construction and mechanical completion when systems are turned over to the commissioning and start-up team. These activities include the following:

- Ensure that equipment is operationally ready for start-up (i.e. to accept feed);
- Sequence starting and running of tested logical groups of equipment;
- Wet and dry runs of systems;
- Demonstration of the suitability of the facilities to be ready for processing and production; and
- Coordinate with and assist the owner to achieve hand over of the completed facilities.

25 INTERPRETATION AND CONCLUSIONS

25.1 INTRODUCTION

This Report summarizes the finding of the FS and establishes the Project as a large silver, lead, and zinc deposit that should proceed with project development. Highlights of this study include:

- The study defines a significant undeveloped silver deposit containing proven and probable mineral reserves of 228.4 million ounces of silver, 2.8 billion pounds of lead and 1.8 billion pounds of zinc.
- The base case after-tax net present value ("NPV") is $643M at a 5% discount rate with an internal rate of return ("IRR") of 20.6% ($20/oz silver, $0.95/lb lead and $1.00/lb zinc). On a pre-tax basis, the base case NPV at a 5% discount rate is $1015.5M with an IRR of 27.9%.
- Average annual payable silver production is 13.5M ounces per year for the first five years and 8.4 million ounces per year over the life-of-mine ("LOM"). On a silver equivalent ounce basis, average annual payable production is 23.8 million ounces per year for the first five years and 15.3 million ounces per year over the LOM.
- Cash cost is a negative $(0.15) per ounce of silver for the first five years, with a LOM cash cost of $3.80 per ounce of silver (net of base metal credits at $0.95/lb lead and $1.00/lb zinc).
- Project produces lead and zinc concentrates. Extensive metallurgical testing and statistical interpretation of results has established improved confidence in flotation recoveries from the 2011 feasibility study.
- Initial capital cost is $625 million with capital payback of 3.6 years at base case metal prices.
- Mine life is 18 years.
- Mill capacity is 22,500 tonnes per day.
- Stripping ratio is 1.68:1 (waste:ore).
- Opportunities include conversion of all or part of the 83M measured and indicated silver resource ounces to a mineral reserve classification with additional drilling, metallurgical testing, and engineering.

25.2 CONCLUSIONS

The Project has an after-tax internal rate of return (IRR) of 20.6%, net present value of $643 million at a 5% discount rate based upon metals prices of $20 per ounce silver, $0.95 per pound for lead and $1.00 per pound for zinc.

Payable silver production averages 13.5 million ounces per year over the first five years. The project will produce an average of 8.4 million payable ounces of silver, 91.8 million pounds of lead and 50.5 million pounds of zinc annually over a 18-year mine life.

Total cash cost for the first five years is a negative $(0.15)/oz silver, with a mine-of-life cash cost of $3.80/oz silver, net of base metal credits. The initial capital investment on the project is estimated to be $625 million with sustaining capital expenditures during mine operations averaging $2.2 million per year over the 18-year mine life. The project achieves payback of capital in 3.6 years using base case metal prices.

The FS is based upon assumptions derived from mine planning sequences completed by GRE and metallurgical test work performed by SGS Laboratories in Vancouver, BC and other laboratories, and evaluated by TS Metallurgical Services who, in conjunction with GRE, developed a detailed, predictive model for recoveries on a block by block basis in the mine sequence. The mining sequence primarily derives ore from the higher-grade starter pits and uses elevated NSR cutoffs in the early years and moves to lower-grade areas with near break-even NSR cutoffs in the later years of production. Operations are for 18 years based on current reserves.

The Mineral Reserve includes 228.4 million ounces of silver contained within 137.7 million tonnes of reserves. An additional 98.1 million tonnes of measured and indicated resource (containing 83 million ounces of silver at 26.3 g/t) and 40.0 million tonnes of inferred resource (containing 47.8 million ounces of silver at 37.2g/t) remain in addition to the Mineral Reserve that could be included in future plans of operations with additional drilling, test work and engineering. These additional Mineral Resources include potentially leachable material.

Table 25-1: Key Assumptions for the Corani Project – Base Case

ITEM	
Annual ore production – Years 1 to end of life (tonnes)	7,875,000
Overall process recovery – silver – into both lead and zinc cons	71.9%
Overall process recovery – lead – into lead cons	62.8%
Overall process recovery – zinc – into zinc cons	60.1%
Total processed tonnes	137,698,000
Average silver grade (g/t)	51.6 g/t
Average lead grade (%)	0.91%
Average zinc grade (%)	0.59%
Payable ounces of silver net of smelter payment terms (total)	151.0 million
Payable pounds of lead net of smelter payment terms (total)	1.65 billion
Payable pounds of zinc net of smelter payment terms (total)	909.6 million
Overall stripping ratio	1.68 to 1
Life-of-mine years	18

The Feasibility Study recommends proceeding with project development based on:

- Good economics at the base case assumptions with good exposure to up-side silver and base metals prices,
- Well-defined resources open to expansion and conversion to reserves at higher metals prices,
- A well-designed, conventional metallurgical process producing highly marketable, separate lead and zinc concentrates,
- Favorable infrastructure for waste rock and tailing disposal, power and access,
- Available local water supply,
- Well-defined permitting process, and
- Local community acceptance and support.

Only measured and indicated resources were used to establish the operations plan when converting resources to reserves.

25.2.1 Mineral Resource

As a result of the work described in Section 15, Mineral Reserve Estimates, it is the conclusion that the mineral resource as stated on Table represents the Mineral Resources and mineral reserves.

25.2.2 Mineral Reserves

As a result of the work described in Section 15, Mineral Reserve Estimates, it is the conclusion that the mineral resource as stated on Table 25-2 represents the mineral reserves and resources.

Table 25-2: Mineral Reserves and Resources

Mineral Reserves, variable $23.00-11.00 NSR cut-off							
Total	Ktonnes	Silver gpt	Lead %	Zinc %	Silver Million oz	Lead million lb	Zinc million lb
Proven	19,855	69.1	1.09	0.72	44.1	478.7	313.4
Probable	117,843	48.6	0.88	0.57	184.3	2289.2	1470.7
Proven & Probable	137,698	51.6	0.91	0.59	228	2,768	1,784

Mineral Resources in Addition to Reserves, $11.00 NSR cut-off, 15 g/tonne Ag cutoff (oxide)							
Total	Ktonnes	Silver gpt	Lead %	Zinc %	Silver million oz	Lead million lb	Zinc million lb
Measured	14,360	32.01	0.34	0.19	14.8	108.4	61.6
Indicated	83,749	25.37	0.37	0.28	68.3	682.2	512.8
M&I	98,109	26.34	0.37	0.27	83.1	790.6	574.4
Inferred	39,953	37.20	0.58	0.40	47.8	510.6	352.4

Mineral Reserves are based on metal prices of $20/oz silver, $0.95 per pound for lead and $1.00 per pound for zinc using variable NSR cutoffs throughout the project life. The mineral resources uses a Whittle pit shell generated with metal prices of $30/oz for silver, $1.425/lb for lead and $1.50/lb zinc. The Mineral Resource NSR cut-off was $9.49/tonne processing cost, plus $1.51 G&A cost. The Mineral Resource includes potentially leachable mixed oxide material that fell within the Mineral Resource pit shell using a silver cut-off grade of 15g/tonne and block elevation above 4900 meters. **Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.**

Measured and indicated resources contained within the Feasibility Study design pit were used to determine final pit limits and thus converted into proven and probable reserves, respectively. The additional resource material is mostly measured and indicated resource that occurs outside of the Feasibility Study pit but which meets the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definition of mineral resource.

25.2.3 Mining

Mining of the Project deposit will be accomplished using straight forward, conventional open pit hard rock mining techniques. The high altitude will pose challenges that do not exist at some other projects. However, there are a number of other open pit operations in Peru and Chile at altitudes similar to those of the Project that are successful operations.

Mining will be performed using conventional open pit methods using 181-tonne payload trucks and a mixture of hydraulic excavators and wheel loaders mining on eight meter high benches. The mine requires minimal pre-production waste stripping of 22.8M tonnes.

25.2.4 Metallurgy & Process

Processing of the ore will be by conventional flotation recovery methods. The ore will be crushed close to the mine and the material conveyed to the processing plant which will be approximately 500 m from the mine. The ore will be ground to 80% passing 106 microns in a SAG/Ball mill circuit. The material will then be floated with the rougher concentrates being reground to 80% passing 25 microns prior to cleaning to produce high-value separate lead-silver and zinc concentrates. Concentrates will be trucked to the port of Matarani for ocean shipment to smelters.

25.2.5 Capital Costs

The project capital cost estimate has been prepared by three independent engineering companies. The mining costs were prepared by by Global Resource Engineering ("GRE") of Denver, Colorado, the process and portions of the infrastructure capital cost have been prepared by M3 Engineering & Technology Corporation (M3) of Tucson, Arizona and the Main Waste Dump/Filtered Tailing Co-Disposal site and remaining infrastructure costs have been prepared by GRE. The 138 kV power transmission line to the site has been designed by Promotora de Proyectos, S.A.C. The main portion of mine access road has been redesigned in 2015 by Anddes Associados SAC with the remaining portion of the mine access road designed in 2011 by HC & Associates. The operations and construction camp was designed and estimated by EMSA of Peru. The initial startup capital is estimated to be $625M and the sustaining capital cost is estimated to be $2.2M annually over the life of mine. The sustaining capital costs include buyout of the leased mine equipment, mine equipment rebuilds, surface water management, the construction of a new mine road in Year 7 of the mine plan, and an allowance of $500,000 per year for plant capital replacements.

25.2.6 Operating Costs

Mining costs were prepared on a year by year basis with costs varying mostly due to changing haulage distances. The life-of-mine average mining costs will be $1.42 per tonne of total waste and ore mined. The process costs are estimated to be $7.91 per tonne of processed ore and the G&A is estimated to be $1.40 per tonne of processed ore or $11M per year.

25.2.7 Infrastructure

The project has favorable infrastructure. Access will be via a new 63 km road to be built over flat topography resulting in low construction costs. The new road will connect to the Interoceanic Highway at the Huiquisa Bridge; a two-lane, paved highway connecting to the port of Matarani.

The mine is 30 km from a new high-voltage power line with abundant capacity to meet the project needs.

The project has an excellent low environmental impact site for containment of filtered tailings resulting in very low capital and operating costs, as the plant will be located immediately adjacent to the mine and the tailing will be co-disposed into the Main Waste Dump near the plant. The site is also located in the upper part of drainages with ample surface water supply and as such there are several surface and underground water source alternatives. The FS provides for the construction of a plant water supply pond to capture contact water and fresh water in separate partitions to provide make-up water to the plant areas.

25.2.8 Environmental and Social

The Company has maintained very good working relationships with the local communities and has continued to operate exploration and development activities at Corani without interruption. The Company owns all the land in the area of the mine and plant. The Company's commitment to the local communities has been further solidified with the recent agreement to provide $1M in aid over the next three years.

The project is designed to meet and, in many ways exceed, international standards of environmental compliance. The tailings co-disposal facility has been designed by GRE to the highest standards of containment and stability. Importantly, the latest design technology will facilitate the permitting process. A water treatment plant using many existing plant facilities will be installed at closure to improve water quality as necessary.

Furthermore, the waste rock storage facilities are designed to encapsulate potentially-acid-generating waste rock and tailings within a non-acid-generating buffer zone in order to minimize the acid producing potential of the facility. Finally, the closure plan provides for the covering of the co-disposal tailing and waste rock facilities in accordance with industry best management practices.

25.2.9 Key Project Results

The FS shows that the Project has strong economic results and is in a stable mining country with significant human and physical infrastructure.

The Feasibility Study recommends proceeding with project development. The permitting procedures should be carried out in accordance with the description explained in Section 4.4 on Permitting and the summary of permit requirements by phase provided in Table 4-3.

25.3 RISKS

25.3.1 Location

There is some risk associated with the operation and construction of the Project at high altitudes. These include diminished productivity of workers, reduced capacity of certain equipment (for example diesel powered equipment), special designs for electrical equipment and compressors and physiological issues for workers at high altitudes. Other projects exist at the altitude of Corani so considerations will need to be made with how personnel and equipment are utilized. Precautions and planning for human physiology limitations have been taken into account with the location of infrastructure, the camp and access from the local town. Also the capacity limitations on machinery have been taken into account in the design of the Project.

25.3.2 Land Ownership

100% of land is owned by BCM.

25.3.3 Permitting

Recently the permitting of mining projects in Peru has faced challenges due to social conflicts. There is a risk that the Project will face delays in the permitting process due to conflicts outside the normal permitting process.

25.3.4 Social Unrest

Recently, there have been several occurrences of social unrest pertaining to developing mining projects. This could cause delays in the development of the project.

25.3.5 ESIA

The main environmental approval required in order to begin mining activities is an Environmental and Social Impact Assessment (ESIA). In 2013, the Ministry of Energy and Mines approved the ESIA based on the Feasibility Study prepared in 2011.

The design and operating improvements incorporated in the 2015 Corani Feasibility Study are expected to require only a modification of the existing approved ESIA, without the necessity for additional public hearings, as they are entirely located within the previously approved project footprint. Furthermore, as the environmental impact of the proposed Corani operation has been reduced as a result of the modifications described above, the Company anticipates final permitting timelines will shorten and costs will be lower than previously anticipated.

Bear Creek's plans for the Corani project are to focus on preparing for development of the project starting with the preparation and submission of the amended ESIA in the third quarter of 2015.

25.3.6 Finance Risks

Significant fluctuation in the long-term financing conditions could affect the cost of capital needed for the start of operation. As with any large scale mine development, there is a risk that additional capital may be difficult to raise in the event that costs increase during the preproduction period. A currency exchange risk exists in the event that the Peruvian Nuevo Sol (PEN) weakend and lowers the cost of in-country expenses, it would raise the cost of imported materials which are priced in USD. However, during operations the sale of metals, priced in USD, would benefit from a weaker PEN.

25.3.7 Assumptions of Early Return of the IGV

The financial model assumes a prompt return of the Peruvian general sales tax known as the Impuesto General a las Ventas (IGV). This will be accomplished by BCM entering into a tax agreement with the Peruvian tax authorities. Obtaining this IGV agreement in a timely manner will affect the cash flow. In this regard, an agreement with the Peruvian Government is required and there could be delays in receiving this tax stability agreement.

25.3.8 Metal Prices

Uncertainty in metal prices cannot be disregarded. Metals can be subjected to the price fluctuation and volatility. Metal prices could suffer a downtrend which would affect the profitability of the project.

25.3.9 Acquisition of Water Rights

Acquisition of water rights falls under the jurisdiction of the department of Puno and although no problems are expected, water is becoming increasingly an issue of contention in Peru. Given the project's location on the eastern side of the continental divide, water rights permitting issues are expected to be manageable.

25.3.10 Inflation of Material and Labor Costs

A scarcity of materials and labor caused by the development of other large mining projects both in Peru and around the world could reduce the availability of trained labor for plant construction and operations. This may cause an increase in both labor and capital costs.

25.3.11 Exchange Rate

Fluctuations of world exchange rates, especially between the Peruvian Sole and the US dollar could impact the project.

25.4 OPPORTUNITIES

25.4.1 Mineral Resource and Reserves

The FS defines significant sulfide resources 73.4 million tonnes of measured and indicated containing 62.4 million ounces averaging 26.4 g/t Ag and 31.2 million tonnes of inferred resources containing 40.8 million ounces of silver averaging 40.6 g/t Ag) that are not included in the current mine plan. Depending upon future silver prices, these resources may be converted into reserves and incorporated into the mine plan.

The FS includes an oxide mineral resource (24.7 million tonnes of measured and indicated containing 20.7 million ounces averaging 26.4 g/t Ag and 8.7 million tonnes of inferred resources containing 7.0 million ounces of silver averaging 25.1 g/t Ag). Additional testwork will be needed to develop a process to handle the oxide mineralization, however it does present an opportunity for additional silver.

Additionally, numerous opportunities exist to discover new mineralization by continuing district exploration. Recent engineering and condemnation drilling has intercepted mineralization up to five kilometers from the current ore body in previously unexplored areas.

Mineral Process and Material Handling

The following additional opportunities are identified with regard to mineral processing and material handling.

- Flotation optimization may improve metal recoveries. Testing under optimized conditions could increase recovery over that predicted by the geometallurgical model.
- Data generated during additional geotechnical drilling may show that it is feasible to steepen pit slopes.
- Higher concentrate grades may be achievable, which would increase the net smelter returns.
- Operating cost improvements may be derived from using conveyor systems to transport tailings to the disposal sites.

25.4.2 Used Equipment

With recent worldwide financial events, the used equipment market is emerging again. An opportunity exists for purchase of used equipment to reduce capital costs.

25.4.3 Gold Zone

The Gold Zone is an area where additional resources could possibly be added to the project. Additional engineering and drilling may be needed to establish this as a resource.

26 RECOMMENDATIONS

Recommendations within this section relate to specific portions of this document's evaluations and studies. Identified items present opportunities to improve the understanding of technical issues associated with the project as well as reduce risk. Multiple identified options present opportunities to improve the project. The detailed scope of work will be developed as part of the front-end Detailed Engineering effort. A preliminary estimate of costs associated with this effort is included at the end of this section.

26.1 SITE GEOTECHNICAL

It is recommended that additional work be done to ensure that the currently planned site layout is feasible from a geotechnical standpoint. Some of the assumptions made in designing project facilities require field verification. Specific areas requiring additional field evaluation include:

- Building foundations;
- Primary crusher structure, conveyor supports;
- Project support facilities foundation requirement review;
- Roadways;
- Main Dump foundation;
- Pit slopes.

Several facilities were relocated during the project optimization, and foundation characterization has yet to be performed for some of the new locations. Standard geotechnical drilling and foundation characterization needs to be performed in order to allow detailed design of the facilities. These include buildings such as the new truck shop, the new primary crusher location, and supports for the new conveyor alignment. Several of the new roadways cross bofedal wetland areas. Geotechnical characterization of the roadways needs to be performed, and engineered crossings need to be designed. Foundation requirements for several support facilities also need to be evaluated.

The Main Dump footprint expands the previously proposed waste dump footprint. Construction and reclamation materials are available within the dump footprint. Characterization of these materials, and assessment of their ability to satisfy project requirements, should be carried out as part of the early detailed design work. Foundation strength characterization (including undisturbed sampling and strength testing), detailed design of the seepage collection pond and pumping system, and additional construction material characterization are required. Limited tailings filtering and geotechnical tests were completed as part of the Feasibility Study. Additional testing should be completed to confirm the allowable tailings moisture content, the ability to filter the tailings, and final drain-down and stability characteristics (including testing of filter tailings dynamic/seismic properties). Updated geotechnical modeling and a detailed placement sequence for co-disposal and PAG encapsulation must be performed based on new field and laboratory data. These analyses must be performed for the Main Dump and for the Este pit backfill area.

26.2 MINE GEOTECHNICAL

Additional geotechnical drilling should be completed within the planned pit. This will confirm the current pit slope design basis, and potentially allow an increase in the pit slope angles. The pit will intersect the unconsolidated sediments lining the floor of the upper bofedal and lower bofedal areas. Additional drilling, testing, and analysis are required to design the pit slopes within the bofedal soils and to develop a detailed plan for dewatering and mining the bofedal soil material. This will involve drilling several boreholes through the unconsolidated sediments, with production of detailed stratigraphic logs and undisturbed sampling for density and strength testing of the unconsolidated material. Boreholes would be completed as monitoring wells, and multiple-well aquifer testing will be performed to better asses the dewatering requirements for the material. Detailed pit slope design and soil mining plans must then be developed.

26.3 PLANT WATER POND

Site investigation and geotechnical design of the Plant Water Pond are required for permitting and for detailed design of the pond system. The process water ponds are required very early in the Project development schedule as a source of water and for sediment control during construction. Geotechnical drilling of the impoundment and embankment dam areas, as well as test pitting of shallow soil cover areas, must be performed. Permeability testing of the underlying bedrock must be undertaken to determine foundation grouting requirements, as well as for design of the seepage cutoff and seepage collection systems. All borings must be grouted to avoid compromising the potential dam foundation areas. Geologic and structural mapping should be performed ahead of the drilling program in order to provide guidance for drilling program design.

26.4 METALLURGY

It should be verified that smelters selected for the study have the capacity and ability to accept the proposed quantity and quality of produced lead and zinc concentrates. As part of the program, additional concentrate analysis should be completed to further define the concentrate qualitites. Letters of intent would be desirable for future project financing and associated due diligence. Transportation, treatment charges, and refinery charges should be confirmed. Under current market conditions there may be an opportunity to improve business terms beyond the assumptions used in this study.

Additional metallurgical testing should be completed to optimize the known flotation test conditions. The reagent scheme should be optimized and additional testing could explore the possibility of higher $ZnSO_4$ dosages on difficult samples in addition to removing and/or reducing other depressants.

The geometallurgical model was developed using all metallurgical testing data, including tests representing non-optimized conditions. Once additional testing has been performed, and samples representing optimized test conditions are available, the statistical model should be re-evaluated to ensure estimated recoveries represent optimal conditions.

The selected process flowsheet should be re-tested to confirm:

- Primary grind size
- Regrind size
- Residence time for each flotation stage
- Reagent selection and dosage

Additional lock cycle testing is recommended for each deposit, particularly material representing moderate to low zinc grades which is under-represented in the current test database. This will allow for validation of the final estimated recoveries and the selected concentrate grades. This testing should include analysis of minor elements; limited test data is available regarding the concentration of minor elements in the final concentrate.

The geotechnical testwork on filtered tailings mentioned in Section 26.1 may lead to the need for additional tailing filtration testing. The filtration equipment needed for mechanical placement of tailings is a significant capital and operating cost to the project. It will be important to determine the filtered tailing physical properties so that the equipment can be specified to achieve the required results.

Testing of an acid brine leaching process for oxide mineral resources should be considered, or testing of an alternative flowsheet, for the non-floatable resource areas of the deposit. If this were successful, it could potentially add value to the project.

26.5 CONSTRUCTABILITY

Generation of a site-wide cut-and-fill material balance is recommended, including specialized construction materials such as clays, concrete aggregate, drain rock, road base, and rip rap.

A detailed Project execution plan and schedule should be produced.

26.6 OPTIMIZATION

It may be possible to reduce the cost of delivering tailings to the Main Dump and pit backfill disposal sites by varying the proportion of tailings delivered by conveyor systems and by trucks during the period when tailings are being produced. It is recommended that an optimization study be carried out to determine this, and a detailed plan should be devised. Tailings will be co-disposed with waste rock in the Main Dump and pit backfills. In general, it is expected that it will be cheaper to use conveyors instead of trucks to deliver tailings to ultimate disposal destinations, but exclusive use of conveyors may be less practical for tailings destined for the pit backfill. During the period when pit backfilling will be taking place, the current schedule indicates that truck capacity will be available; therefore, an optimization study should specify the ideal mix of conveyor/truck transport of tailings over time, depending partly on truck availability.

Use of a maintenance-and repair-contract (MARC) for the mine fleet should be investigated. This could reduce the skilled-trade staff.

A study should be conducted to match operating equipment to the high-altitude conditions, potentially identifying equipment outfitted with pressurized cabs and other worker comfort and performance additions. Caterpillar equipment offers high-altitude arrangements (HAA), and these modifications allow their power ratings to be valid to 4,877 m.

26.7 ENVIRONMENTAL/SOCIAL

In 2013, the Ministry of Energy and Mines approved the Environmental Study (ESIA) prepared in 2012. It is highly recommended to start as soon as possible an Updated Environmental Study incorcoprating the results from the new engineering study. The design and operating improvements incorporated in the 2015 Corani Feasibility Study are expected to require only a modification of the existing approved ESIA, without the necessity for additional public hearings, as they are entirely located within the previously approved project footprint. Furthermore, as the environmental impact of the proposed Corani operation has been reduced as a result of the modifications.

In addition, it is recommended that Bear Creek commence the permitting process on water rights and mine plan approval; both of which are critical, early-stage permits. Bear Creek is also encouraged to utilize all efforts in maintaining its social license and ensuring the continued strong support from local communities, local and regional governments, and the central government Peruvian government.

26.8 MINING AND MODELLING

The 2011 Feasibility Study recommended review of the check assay and standards because significant scatter was observed in that data. This issue has not been fully resolved and future drill programs should work to address the issue.

The other Resource related recommendation in 2011 was with regard to rock density information. GRE believes the new analysis of density using the combine Zinc, Lead, and Silver grade solved the issue. The density calculation is no longer a concern.

26.9 ESTIMATED COST FOR RECOMMENDED WORK

During the course of the work on this Project, the contributors have developed several recommendations for future consideration and execution by BCM. The estimated costs to complete the recommended work are shown in Table 26-1.

Table 26-1 Estimated Costs for Recommended Work

ITEM	ESTIMATED COST
Site Geotechnical	$ 230,000
Mine Geotechnical	$ 1,100,000
Plant Water Pond	$ 230,000
Metallurgy	$ 1,310,000
Constructability	$ 160,000
Environmental / Social	$ 300,000
Optimization	$ 40,000
TOTAL	**$ 3,370,000**

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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 41 years. I have worked for engineering consulting companies for 12 years and for M3 Engineering & Technology Corporation for 29 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 Sections 1, 2, 3, 4, 5, 6, 18, 19, 21, 22, 24, 25, 26, and 27 of the technical report titled "Optimized and Final Feasibility Study, Corani Project, Puno, Peru, Form 43-101F1 Technical Report" dated effective May 30th, 2015.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have visited the Corani Project property on December 8 - 9, 2010.

9. 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 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 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 30th day of May, 2015

Signed "Daniel H. Neff"
Signature of Qualified Person

Daniel H. Neff
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Laurie M. Tahija, Q.P., do hereby certify that:

1. I am currently employed as Vice 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 Montana College of Mineral Science and Technology, in Butte, Montana and received a Bachelor of Science degree in Mineral Processing Engineering in 1981.

3. I am recognized as a Qualified Professional (QP) member (#01399QP) with special expertise in Metallurgy/Processing by the Mining and Metallurgical Society of America (MMSA):

4. I have practiced mineral processing for 32 years. I have over twenty (20) years of plant operations and project management experience. I have been involved in projects from construction to startup and continuing into operation. I have worked on scoping, pre-feasibility and feasibility studies for mining projects in the United States and Latin America, as well as worked on the design and construction phases of some of these projects.

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 17 of the technical report titled “Optimized and Final Feasibility Study, Corani Project, Puno, Peru, Form 43-101F1 Technical Report” dated effective May 30th, 2015.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have not visited the Corani Project property.

9. 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 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 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 30th day of May, 2015.

Signed "Laurie M. Tahija"
Signature of Qualified Person

Laurie M. Tahija
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Christopher K. Chapman do hereby certify that:

1. I am currently employed as Principal Mining and Geological Engineer by:

   Global Resource Engineering
   600 Grant Street, Suite 975
   Denver, CO 80203

2. I am a graduate of the Colorado School of Mines with a Bachelor of Science degree in Geological Engineering (2000).

3. I am a registered Professional Engineer in the State of Colorado (40679).

4. I have worked as a Mining & Geological Engineer for a total of 15 years since my graduation from university. I have been involved with review, design, engineering, construction, startup, operations, and reclamation for numerous mining projects.

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 have personally visited the project on seven occasions from November 2010 to May 2012.

7. I am responsible for the preparation of the following sections of the technical report titled "Optimized and Final Feasibility Study, Corani Project, Puno, Peru, Form 43-101F1 Technical Report", dated effective May 30, 2015 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
17.9.2	Project Water Balance
18.3, 18.6	Project Infrastructure
20	Environmental Management During Operations, Reclamation, and Closure
26.1, 26.2 and 26.3	Recommendations

8. I have not had prior involvement with the property that is the subject of the Technical Report.

9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.

10. I have read National Instrument 43-101 and Form 43-101F1, and confirm the sections of the Technical Report prepared under my supervision (as listed in Item 7 above) have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

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 30th day of May, 2015.

*Signed "Christopher K. Chapman"*________
Signature of Qualified Person

Christopher K. Chapman________________
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Terre A. Lane do hereby certify that:

1. I am currently employed as Principal Mining Engineer by:

   Global Resource Engineering
   600 Grant Street, Suite 975
   Denver, CO 80203

2. I am a graduate of the Michigan Technological University and received a Bachelor of Science degree in Mining Engineering in 1982.

3. I am a Qualified Professional with MMSA, member number 01407QP in Ore Reserves and Mining.

4. I have practiced my profession since 1982 in capacities from mining engineer to senior management positions for engineering, mine development, exploration, and mining companies. I have been involved in the estimation of resources and mine design for several hundred projects at locations in North America, Central America, South America, Africa, Australian/New Zealand, India, China, Russia and Europe. My relevant experience for the purpose of this technical report is as the resource estimator and mine engineer with over 30 years of experience.

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 have not visited the Corani property.

7. I am responsible for the preparation of the following sections of the technical report titled "Optimized and Final Feasibility Study, Corani Project, Puno, Peru, Form 43-101F1 Technical Report" dated effective May 30, 2015 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
14	Mineral Resource Estimate
15	Mineral Reserve Estimate
16	Mining Methods
21 (Mining Portions)	Capital and Operating Costs

8. I have not had prior involvement with the property that is the subject of the Technical Report.

9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.

10. I have read National Instrument 43-101 and Form 43-101F1, and confirm the sections of the Technical Report prepared under my supervision (as listed in Item 7 above) have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

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 30th day of May, 2015.

*"Signed Terre A. Lane"*__________________
Signature of Qualified Person

Terre A. Lane__________________________
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Richard D. Moritz do hereby certify that:

1. I am currently employed as Principal Process and Mining Engineer by:

   Global Resource Engineering
   600 Grant Street, Suite 975
   Denver, CO 80203

2. I am a graduate of the University of Nevada, Reno and received a Bachelor of Science degree in Mining Engineering in 1979 and a MBA in 1987.

3. I am a Qualified Professional with MMSA, member number 01256QP in Metallurgy/Processing, and Mining.

4. I have practiced mine and process engineering for over 30 years since my graduation. I have worked for several mine and process engineering companies, a royalty company, and several mining companies over the course of my career. I have been involved with review, design, engineering, construction, startup, and operations for numerous mines.

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 have not visited the Corani property.

7. I am responsible for the preparation of the following sections of the technical report titled "Optimized and Final Feasibility Study, Corani Project, Puno, Peru, Form 43-101F1 Technical Report", dated effective May 30, 2015 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
7.3.2	Mineral Ore Types for Process Metallurgy
13.7	Metallurgical Performance Projections

8. I have not had prior involvement with the property that is the subject of the Technical Report.

9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.

10. I have read National Instrument 43-101 and Form 43-101F1, and confirm the sections of the Technical Report prepared under my supervision (as listed in Item 7 above) have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

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 30th day of May, 2015

*Signed "Richard D. Moritz"*________________
Signature of Qualified Person

Richard D. Moritz_____________________
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Thomas Wylie Shouldice do hereby certify that:

1. I am currently employed as Consulting Metallurgist by:
   TS Technical Services Ltd.
   1215 Canyon Ridge Place,
   Kamloops, BC
   V2H 0A1

2. I am a graduate of Queen's University and received a Bachelor of Applied Science (Engineering) degree in Mining - Mineral Processing in 1993.

3. I am a Professional Engineer in British Columbia, Canada with APEGBC. My member licence number is 27489.

4. I have practiced metallurgy and mineral processing for 22 years. I have worked for metallurgical testing and consulting companies for 15 years and for TS Technical Service Ltd for 2 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 was present at the metallurgical testing facility during portions of the metallurgical testing in 2007. I have not made a visit to the mine site.

7. I am responsible for the preparation of the following sections of the technical report titled "Optimized and Final Feasibility Study, Corani Project, Puno, Peru, Form 43-101F1 Technical Report", dated effective May 30, 2015 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
13	Mineral Processing and Metallurgical Testing
26.4	Recommendations – Metallurgy

8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I am Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.
10. I have read National Instrument 43-101 and Form 43-101F1, and confirm the sections of the Technical Report prepared under my supervision (as listed in Item 7 above) have been prepared in compliance with that instrument and form.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.
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 30th day of May, 2015.

*Signed "Thomas W. Shouldice"*____
Signature of Qualified Person

Thomas W. Shouldice P.Eng.______
Print name of Qualified Person

CERTIFICATE of QUALIFIED PERSON

I, Christian Rios, do hereby certify that:

1. I am currently self-employed and am contracted as a Consulting Geologist by:

   Bear Creek Mining Company
   Av. Republica de Panamá 3505
   Piso 6
   Lima, 27
   Peru

2. I graduated with a Bachelors of Geological Engineering from La Universidad de San Marcos, Lima, Peru, in 1999, and obtained a Masters of Science, Economic Geology from the University of Arizona in 2008.

3. I am a Certified Professional Geologist through membership in the American Institute of Professional Geologists, CPG - 11274, and have been since June 2009.

4. I have been employed as a geologist in the mining and mineral exploration business, continuously, for the past 15 years, since my graduation from university. I worked for Bear Creek Mining Company for 13 years from 2002 to 2014.

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 visited the Corani property several times from December 2004 to March 2015.

7. I am responsible for the preparation of the following sections of the technical report titled "Optimized and Final Feasibility Study, Corani Project, Puno, Peru, Form 43-101F1 Technical Report", dated effective May 30, 2015 (the "Technical Report"), relating to the Corani Silver-Lead-Zinc project in Peru.

SECTION	SECTION NAME
7	Geological Setting and Mineralization
8	Deposit Types
9	Exploration
10	Drilling
11	Sample Preparation, Analyses and Security
12	Data Verification
14.1.1 and 14.1.2	Mineral Resource Estimates
23	Adjacent Properties

8. I was formerly the VP Exploration of Bear Creek Mining Corporation (a subsidiary of which I am currently a consultant to) and as such have had prior involvement with the property that is the subject of the Technical Report.

9. As I am a former employee of, and am currently a consultant to, a subsidiary of Bear Creek Mining Corporation, I am not Independent (as defined in NI 43-101) of Bear Creek Mining Corporation.

10. I have read National Instrument 43-101 and Form 43-101F1, and confirm the sections of the Technical Report prepared under my supervision (as listed in Item 7 above) have been prepared in compliance with that instrument and form.

11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

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 30th day of May, 2015.

*Signed "Christian Rios"*____________________
Signature of Qualified Person

Christian Rios____________________________
Print name of Qualified Person