M3-PN100153
December 2011

Corani Project

Form NI 43-101F1 Technical Report

Feasibility Study

Puno, Peru

REVISION 0
Prepared For:

BEAR CREEK
MINING CORPORATION

DATE AND SIGNATURES PAGE

See Appendix A, Feasibility Study Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this report in accordance with Form 43-101F1.

This report is current as of 22 December 2011.

CORANI PROJECT

FORM 43-101F1 TECHNICAL REPORT

FEASIBILITY STUDY

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1 SUMMARY

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 Region 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 63 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 tailing and waste rock 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 list of claims and a general location map is provided on Table 4-1 and Figure 4-2 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 (22.1Ma +/- 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 centre of the deposit and to the east.

Southeast of the principal areas of mineralization there are smaller areas of mineralization and 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; these are discussed in Section 4.3.

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.

BCM took operational control of the project 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 taking control of the project BCM has completed over 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 where the resource has grown and culminates with this Report.

1.4 DEVELOPMENT AND OPERATIONS

1.4.1 Production Schedule

The mine requires minimal pre-production waste stripping of 16.2 million tonnes. And after that, a mining sequence that directly feeds the mill with 7.875 million tonnes of ore per year was developed. In addition to the direct mill feed, the plan also calls for stockpiling of 16.6 million tonnes of lower grade material that will be fed into the mill at the end of the mine life.

The mining sequence calls for the waste stripping to average 1.28:1 (waste:ore) for the first 7 years and then the stripping ratio will grow to 1.97:1 for the following 5 years and then will drop to an average of 1.83:1 for the final 6 years.

1.4.2 Mine Equipment

Mining will be performed using conventional open pit methods using 135 t trucks and 15 m^3 hydraulic shovels mining on 8 meter-high benches.

1.5 METALLURGY

BCM has completed metallurgical optimization tests on several master composites in order to define recoveries for the purposes of the Feasibility Study (“FS”) reserve calculation. The composites represent several different types of ore and various grade ranges expected during the operation of the mine. The composites therefore approximate the expected life of mine concentrator feed material. The master composite test work establishes that when the two materials (mixed sulfide and transitional ore) are blended in accordance with the mine sequence that the recoveries and resulting concentrate grades are as outlined in Table 1-1 below.

Additionally, BCM has completed a re-logging of all drill cores to identify the different metallurgical material types. The re-logging data was then utilized to define each block in the resource model with a metallurgical rock type and assigning every block a specific recovery and concentrate grade for the purposes of determining its Net Smelter Return (“NSR”) value and reserve classification. The life-of-mine overall recoveries and concentrate grades are tabulated in Table 1-1 below.

Table 1-1: Average Recoveries and Concentrate Grades of the Life of the Project

	Lead Con		Zinc Con	
	Pb	Ag	Zn	Ag
Recovery	71.11%	60.29%	51.58%	3.90%
Average Con Grades	56.50%	2.9 kg/t	53.00%	437 g/t

1.6 PROCESS

The Project processing facility is designed to treat 22,500 t/d 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 a metallurgical testing program that was supervised by Blue Coast and BCM.

The ore will be crushed in a primary 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 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 values for further upgrading.

Tailing from the zinc flotation circuit will be thickened and then pumped to a conventional tailing storage facility. Water will be reclaimed from the tailing thickener overflow and from the supernatant of the tailing storage facility. The reclaimed water is recycled back to the process plant as process water. In the final half of the mine life, a pyrite flotation circuit will be added to separate the pyrite from the zinc flotation tailing. The purpose of this circuit is to reduce the acid rock generating potential of the tailing. The pyrite concentrate will be placed in the central part of the tailing storage facility where it will remain submerged and un-oxidized after the closure of the mine.

Lead and zinc concentrates will be thickened, filtered, and stockpiled separately. They will then be loaded into sealed-top trucks and trucked to the Port of Matarani for ocean shipment to smelters.

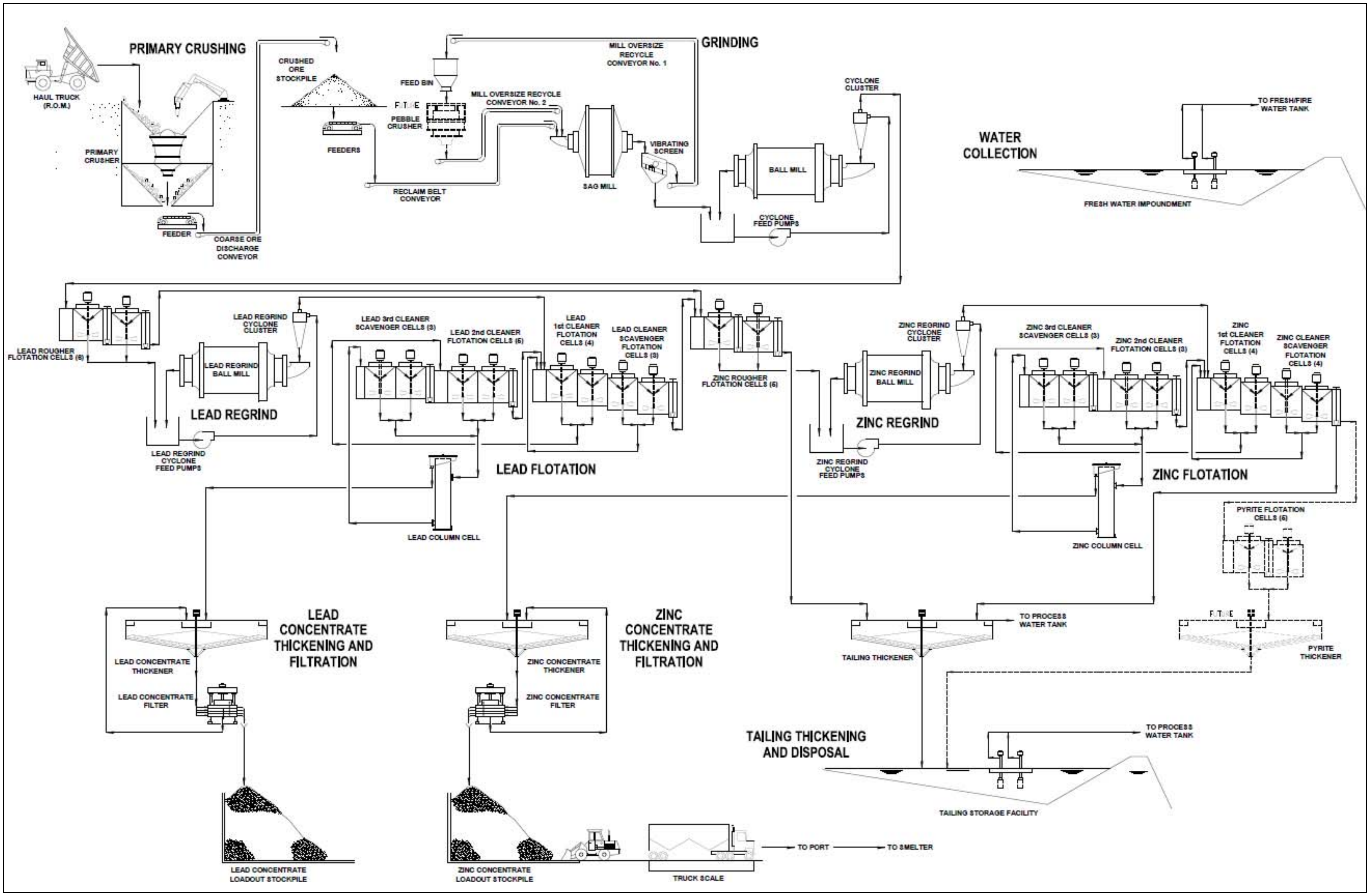


Figure 1-1: Simplified Process Flow Diagram for the Corani Project

1.7 ENVIRONMENTAL AND PERMITTING

Due to the mineralogy of the area and the nature of the activities to be carried out, the main environmental considerations associated with the Project include impacts to surface water and ground water, in relation to the tailing storage facility and waste rock dumps. In addition, as a result of historical mining activities, a number of historic environmental liabilities are present on the project site. These can be resolved, to the extent practical, as the site is developed.

BCM obtained the permits required for the previous field exploration activities and have identified the permits required for the construction, exploitation and closure phases.

In order to begin construction and mining activities, the main environmental approval required is an Environmental and Social Impact Assessment (ESIA). Work on this permit process is underway, including community consultation processes. The ESIA will need to address the potential environmental impacts of the tailing storage facility and waste rock dumps. A list of other permits required for each development phase can be found in Section 4.4.

1.8 RECLAMATION AND CLOSURE

1.8.1 Objective of Preliminary Closure Plan

A Preliminary Closure Plan has been developed to identify potential and viable measures which can be implemented during the operation, reclamation and closure period and the post-closure period at Corani. The measures should alleviate the potential long-term impacts from the mining operations and minimize long-term liability. It is the intent of the measures to be applied for reclamation and closure of the project to return the receiving environment to a condition, at worst, equal to the measured baseline conditions and, to the extent practical, to improve these conditions.

1.8.2 Project Components

The main project components considered in the preliminary closure plan and for cost estimation purposes consist of the following.

- The Este, Main and Corani open pit areas;
- Waste rock facilities consisting of the Este dump, Main dump and in-pit backfill areas;
- The surface water management and water collection pond systems;
- The plant facilities and related infrastructure;
- The fresh water dam located in the Quelcaya drainage; and
- The tailing storage facility (TSF).

1.8.2.1 Open Pits

As described in the mine plan, portions of the "U-shaped" pit are completed prior to the end of mining activities. From approximately year 12 through year 18, continued mining within the pit will produce waste rock which can be used to backfill the deeper portions of the pit and place material against the lower pit walls. Additional backfilling will be performed after mining activity has ended.

By backfilling the pit to the elevation of the southern pit rim, the backfill material will become flooded and sub-aqueous conditions will be established. Under these conditions, the absence of oxygen will inhibit long-term oxidation of the pit walls and the backfill material to the extent that long-term issues related to acid rock drainage from these areas will be alleviated. There will be no pit lakes left within the pit area.

For the exposed backfill areas and pit floor, a layer of inert PMT material will be placed to form a cover over the potentially acid generating (PAG) backfill. In addition, a layer of bofedal material will be placed over the pit floor. The mine plan includes the stockpiling of low grade ore that will be processed following the completion of mining. This will occur over a period of two to three years during which time the reclamation and closure of the pit area will have been performed. Water draining from the pit area will be collected and used as makeup water in the process circuit and pumped with the tailing slurry to the TSF. Following the period of processing low grade material, active management or treatment of groundwater originating from within the pit will be required.

1.8.2.2 Waste Rock Facilities

In addition to the waste rock backfilled into the pit, two WRF areas will be developed: the East dump and the Main dump. Segregation of the waste rock based on the mineralized (PAG) and non-mineralized, non-acid generating (NAG) nature of the material will be performed throughout mining activity. Non-acid generating post mineral tuff (PMT) material will be placed in the East dump. PAG waste rock will be placed in the Main dump and forms approximately 60 to 70 percent of the material to be placed in the Main dump. The remaining material will consist of PMT and managed placement of the two materials will be performed within the dump. Completion of placement of waste rock in both the East and Main waste dumps will occur prior to the completion of mining in the pit area. Once backfill areas are available in the pit (approximately year 12) the reclamation and closure of the dumps will be performed.

At the end of operations for the dumps, the upper surface will be graded to shed water to the sides of the dump and a permanent collection ditch connecting to the groin ditches will be constructed against the contact with natural ground. During project development the removal of the soft and organic materials will be performed and the material stockpiled for use during closure. A layer of these materials will be placed across the dump surface and, over the long-term, is anticipated to recreate a natural bofedal condition.

1.8.2.3 Surface Water Management & Water Collection Pond Systems

During project development and operations, a surface water management system will be developed to route runoff and potentially acidic runoff or waters from areas of disturbance to a collection pond system. As part of the reclamation and closure activities, a portion of the process plant previously utilized in the mineral processing will be converted to a water treatment plant. Water from the management system would be routed to the plant and treated, as necessary, prior to release to the receiving environment. During the reclamation and closure period, the collection ditches routing water from various locations around the property would be upgraded to create low-maintenance erosion resistant channels and ditches for long-term operation during the post closure period.

1.8.2.4 Plant Facilities & Related Infrastructure

At the end of the operating life of the processing plant, with the exception of those components to be utilized in the post-closure water treatment system, the plant would be decommissioned and demolished. A small amount of infrastructure would be left in place for post-closure use, including conversion of portions of the plant for water treatment. The main power line and ancillary equipment utilized during operations would be removed and the cable salvaged. Once the off-site man camp is no longer required, the camp would be converted to an alternative beneficial use by a future custodian 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 the necessary locations for maintenance and monitoring of the project areas will be reduced in size to that of similar local roads in the project region. All other haul roads would be removed by ripping and reclaimed to conditions similar to the surrounding area.

1.8.2.5 Fresh Water Dam

At reclamation and closure, a permanent spillway will be excavated through the west abutment of the dam and a stilling basin will be constructed for the Fresh Water Dam. The resulting dam consists of a potential resource for the downstream communities as a fresh water supply dam and irrigation water source. Transfer of ownership and operation of the dam to a group of the local communities or government organizations would be performed following the implementation of reclamation and closure measures. All operational pumping systems and infrastructure would be removed.

1.8.2.6 Tailing Storage Facility (TSF)

At reclamation and closure, a spillway will be constructed capable of passing the probable maximum flood (PMF) event with discharge through the spillway entering the valley downstream of the main dam to the impoundment surface will be configured such that water entering the impoundment during the post-closure period runs to the spillway without the impoundment of a significant volume of water upstream of the dam prior to closure.

At around Year 10 of operations, all tailing entering the impoundment will be de-pyritized in a supplemental flotation circuit to reduce the residual sulfide levels in the tailing material. The flotation circuit will produce a pyrite concentrate that will be deposited sub-aqueously within the impoundment.

Access will be possible for light equipment to the impoundment as the impoundment surface; remote from the spillway area dries and consolidates. Over a period of 2 to 3 years after the completion of the above steps, the impoundment surface will be progressively covered by a layer of bofedal material that is to be salvaged from the impoundment footprint and stockpiled prior to commissioning of the facilities. In the long-term it is anticipated that a natural bofedal will establish itself on the impoundment surface.

The primary seepage collection system, located at the toe of the main tailing impoundment will be maintained during the reclamation and closure and post-closure period in order to collect and monitor the seepage quantity and quality at that point. Similarly, seepage collection systems at the saddle berms will also be maintained during this period. Initially, water from the seepage collection system will be recycled into the impoundment or sent to the water treatment plant as required. It is anticipated, however, that the long-term water quality at the seepage collection system will develop a chemistry at or better than the baseline conditions.

1.8.3 Monitoring and Maintenance

Monitoring and maintenance of the mine reclamation and closure activities and in the post closure period will be performed. This will consist primarily of a team of onsite personnel. Equipment to monitor the reclaimed areas of the project and to perform regular maintenance will be available for this purpose. The monitoring will include water quality sampling and analyses in addition to regular inspection of the TSF, WRFs and the pit areas. The required duration of the monitoring and maintenance is undefined at this time however, a minimum period of 25 years has been considered.

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 consultants input, client input and historical project data. Construction durations were based on quantities and man-hours developed in the capital cost estimate:

- ESIA Preparation/Review and Permitting – 17 months

- Detailed Engineering – 18 months
- Construction – 23 months
- Commissioning and Start-Up – 6 months

The summary Project Schedule shown in Figure 24-1 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 Start up 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

Execution of engineering work would likely take place in North America. 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 on site 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.

Construction of the process plant and infrastructure facilities would be performed by contractors specialized in the scope of work as described in the Engineering Requests for Proposal (ERFP) documents. Contracts would be awarded to major construction contractors following competitive bidding based on bid documents prepared by the engineer.

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.

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.

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 $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. See Table 1-2.

Table 1-2: Life of Mine Operating Cost

Operating Cost	$/ore tonne
Mine	$3.82
Process Plant	$7.91
General Administration	$1.40
Smelting/Refining Treatment & Concentrate Transport	$5.85
Total Operating Cost	**$18.99**

Note: Total shown in table is inconsistent because of rounding of the inputs

1.11 CAPITAL COST ESTIMATE

The project capital cost estimate has been prepared by three independent engineering companies. The mining costs were prepared by Independent Mining Consultants of Tucson, Arizona, the process and portions of the infrastructure capital cost have been prepared by M3 Engineering of Tucson, Arizona and the Tailing Storage Facility ("TSF") and remaining infrastructure costs have been prepared by Global Resource Engineering ("GRE"). The initial startup capital is estimated to be $574M and the sustaining capital cost is estimated to be $7.2M annually over the life of mine. The capital costs include detailed long-term plans for tailing dam expansions as well as ongoing capital (i.e. mine fleet replacement) and mine closure. See Table 1-3.

Table 1-3: Capital Cost Summary

AREA	TOTAL	
General Site Including Access Road	$	47,046,501
Mine Capital + Preproduction	$	84,315,000
Primary Crushing	$	15,038,626
Reclaim Stockpile	$	6,381,502
Grinding	$	44,958,665
Flotation and Regrind	$	39,552,771
Concentrate Thickening	$	10,161,130
Tailing Disposal	$	62,288,241
Fresh Water/ Plant Water	$	17,900,934
Power Supply Infrastructure	$	13,873,238
Reagents	$	3,277,208
Ancillaries	$	21,894,455
Off Sites (Camp)	$	13,822,799
Direct Cost	**$**	**380,511,068**
Contractor Indirects	$	11,505,867
EPCM Services	$	47,161,819
Commissioning and Vendor Reps	$	2,489,369
Capital Spare Parts & Initial Fills	$	8,859,561
Owner’s Cost	$	30,771,413
Freight, Duties	$	27,192,086
Indirect Cost	**$**	**127,980,115**
Contingency (Process Plant)	$	59,010,715
Contingency (Mine)	$	6,872,450
Total	**$**	**574,374,347**

Note: Total shown in table is inconsistent because of rounding of the inputs

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 1-4 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 1-4: Key Assumptions for the Corani Project – Base Case

Annual ore production – years 1 to end of life (tonnes)	7,875,000
Overall process recovery – silver – into both lead and zinc cons	64.2%
Overall process recovery – lead – into lead cons	71.1%
Overall process recovery – zinc – into zinc cons	51.6%
Total processed tonnes	156,130,000
Average silver grade (g/t)	53.8 g/t
Average lead grade (%)	0.90%
Average zinc grade (%)	0.49%
Payable ounces of silver net of smelter payment terms (total)	160.2 million
Payable pounds of lead net of smelter payment terms (total)	2.1 billion
Payable pounds of zinc net of smelter payment terms (total)	744 million
Overall stripping ratio	1.69 to 1
Life-of-mine (mining only) years	18
Life-of-mine (processing) years	20

The results of the economic analysis for the project has an after-tax internal rate of return (IRR) of 17.6%, net present value of $463 million at 5% discount rate based upon metal prices of $18.00 per ounce silver, and $0.85 per pound for both lead and 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 $18.00/oz silver, $0.85/lb lead, and $0.85/lb zinc. The economic cutoff for mineral reserves with those prices is $10.54/tonne NSR (Net Smelter Return).

Mineral resources were developed with metal prices of $30.00/oz silver, $1.00/lb lead, and $1.00/lb zinc. The economic cutoff for mineral resources was $9.20/tonne NSR.

Table 1-5 summarizes the mineral reserves and the mineral resources in addition to the mineral reserves.

Table 1-5: Mineral Reserves and Mineral Resources

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

The mineral resources and mineral reserves were developed by Independent Mining Consultants (IMC) with John Marek P.E. 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.

1.14 CONCLUSIONS

The Project has an after-tax internal rate of return (IRR) of 17.6%, net present value of $463 million at a 5% discount rate based upon metals prices of $18 per ounce silver, and $0.85 per pound for lead.

Payable silver production averages 13.4 million ounces per year for the first 5 years. The project will produce an average of 8 million payable ounces of silver, 105 million pounds of lead and 37 million pounds of zinc annually over a 20 year mine life.

Total cash cost for the first five years is a negative $(0.45)/oz silver, with a mine-of-life cash cost of $3.66/oz silver, net of base metal credits. The initial capital investment on the project is estimated to be $574 million with sustaining capital expenditures during mine operations averaging $7.2 million per year over the 20 year mine life. The project achieves payback of capital in 3.8 years using base case metal prices.

The FS is based upon assumptions derived from mine planning sequences completed by IMC and metallurgical test work performed by SGS Laboratories in Vancouver, BC and reviewed by Blue Coast Metallurgy. The mining sequence primarily derives ore from the higher-grade starter pits in the early years and moves to lower-grade areas in the later years of production. Operations are for 20 years based on current reserves.

In the mine sequence, only 270 million ounces contained within 156 million tonnes have been used as reserves in this plan. An additional 134 million tonnes of measured and indicated resource (containing 88.7 million ounces of silver at 20.5 g/t) and 49.8 million tonnes of inferred resource (containing 48.0 million ounces of silver at 30g/t) remain that could be included in later plans of operations. About 89% of these resources are mixed sulfide and transition material peripheral to the reserve pit. About 11% are contained within oxide mineralization, which outcrops at surface.

1.15 RECOMMENDATIONS

1.15.1 Plant Relocation

More geotechnical tests are required to finalize the location of the mill and surrounding facilities. Geotechnical studies so far completed suggest that a minor relocation or repositioning of the plant may be required to minimize foundation work costs.

1.15.2 Gold Zone

For the Gold Zone, a program of metallurgical testing should be undertaken to establish an appropriate recovery method and costs and then a mineral resource should be calculated for this area.

1.15.3 Mining and Modeling

IMC recommends that the rock density data should be characterized by mineral ore type to determine if there is a better method for applying rock density in future model estimates. Also the scatter or variability in check assays and standards should be examined in more detail to better explain the variability in the data.

1.15.4 Metallurgy

The primary mesh of grind for the Corani ore needs to be studied further. While the mill design was based on a grind of 106 microns, recent metallurgical tests show that some ore types may require finer or coarser grinds. In addition, overgrinding of heavy minerals by grinding mills in closed circuit with cyclones may allow a coarser grind during actual operation.

Other metallurgical aspects that merit further investigation include benchmarking of flotation test results to actual operations, improvement of zinc and silver recoveries to the zinc concentrate, grindability tests to determine variability of the ore during the life of mine, and optimization of the cost and performance of the inert grinding media required for the process.

1.15.5 Process Plant Design

MetSim mass balance simulations will need to be repeated once benchmarked recoveries and concentrate grades are available. This will allow optimization of size selections for pumps, pump boxes, pipes and flotation cells.

A few pieces of equipment were designed based on typical industry parameters. Better design accuracy can be achieved if experimental measurements of these parameters are conducted. These include regrind work indices, better estimates of regrind feed size distribution in the regrind mills, thickener settling rates for both tailing and concentrates, and laboratory filtration rates for concentrates.

1.15.6 Environmental and Social

Given the mineralogy of the area, water management is a key environmental consideration. Potential impacts to surface water and ground water in relation to the tailing storage facility and waste rock dumps should be thoroughly assessed and mitigation and management programs developed.

An environmental liability closure plan will be developed and the environmental liabilities will be resolved as the mine is developed.

1.15.7 Other Recommendations

During the course of the work on this Project, the contributors have developed several recommendations for future consideration and execution by BCM, including:

- Further development of the project should proceed with basic engineering to better define and optimize the arrangement of facilities and to confirm equipment sizing and selection.
- M3 recommends that BCM start talks with prospective lead and zinc smelters to improve estimates for smelter treatment and refining charges, payable rates, transport costs, specifications, and others. The smelters may request BCM to submit samples of typical concentrates that the Corani plant will produce.
- The capital cost estimate is based on budgetary quotes for equipment. Capital cost reduction is possible with the negotiation of firm purchase orders.
- With recent worldwide financial events, the used equipment market is emerging again. Consideration should be given to the purchase of used process equipment.

2 INTRODUCTION

2.1 PURPOSE

Bear Creek Mining Corporation (BCM) commissioned M3 Engineering and Technology Corporation (M3) to produce this National Instrument 43-101 Technical Report (Report) for the Corani Project (Project) in the Region of Puno, Peru. 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 270 million ounces of silver, plus 3.1 billion pounds of lead and 1.7 billion pounds of zinc over a period of 20 years.

This Report was prepared in conjunction with an updated NI 43-101 resource estimate completed by Independent Mining Consultants, Inc. (IMC), a consulting company based in Tucson, Arizona. 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. Items covered included preliminary mine planning leading to an estimate of "potentially mineable mineral resources", site investigations, flow sheet development based on the available metallurgical test work, and cash flow projections.

Since the scoping studies BCM has completed further exploration and a number of studies have been undertaken to progress the understanding of the Project to a Feasibility Study standard. The Feasibility Study incorporates an updated resource estimation and mine design performed in October 2011 by IMC based upon 93,577 meters of drilling and sampling in 544 diamond drill holes and trenches completed through May of 2009. The studies have included revision of the resource and reserve estimates, open pit (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 November 2011 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., of M3 will be the principal Qualified Person ("QP") and author of the study.
- Art Ibrado, Ph.D., of M3 is the co-author of the study.

- John Marek, P.E., Consulting Mining Engineer, Independent Mining Consultants, Inc (IMC);
- Chris Martin, C.Eng., Blue Coast Metallurgy, Ltd. (Blue Coast);
- Chris Chapman, P.E., Global resource Engineering (GRE).
- Edmundo Laporte-Ramírez, ESCI, LLC as qualified person for Walsh
- Marc Leduc, P.Eng., Bear Creek Mining

Table 2-1: List of Contributing Authors

SECTION	SECTION NAME	COMPANY	RESPONSIBLE PARTY
1	Summary	M3, GRE & Walsh	Daniel H. Neff, P.E., Chris Chapman, P.E. & Edmundo Laporte-Ramírez, P.E.
2	Introduction	M3	Daniel H. Neff, P.E.
3	Reliance on Other Experts	M3	Daniel H. Neff, P.E.
4	Property Description and Location	M3 & Walsh	Daniel H. Neff, P.E. & Edmundo Laporte-Ramírez, P.E.
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	M3, GRE & Walsh	Daniel H. Neff, P.E., Chris Chapman, P.E. & Edmundo Laporte-Ramírez, P.E.
6	History	IMC	John Marek P.E.
7	Geological Setting and Mineralization	IMC	John Marek P.E.
8	Deposit Types	IMC	John Marek P.E.
9	Exploration	IMC	John Marek P.E.
10	Drilling	IMC	John Marek P.E.
11	Sample Preparation, Analyses and Security	IMC	John Marek, P.E.
12	Data Verification	IMC	John Marek, P.E.
13	Mineral Processing and Metallurgical Testing	Blue Coast	Chris Martin, C. Eng.
14	Mineral Resource Estimates	IMC	John Marek, P.E.
15	Mineral Reserve Estimates	IMC	John Marek, P.E.
16	Mining Methods	IMC	John Marek, P.E.
17	Recovery Methods	M3	Art S. Ibrado, Ph.D.
18	Project Infrastructure	M3 & GRE	Daniel H. Neff, P.E & Chris Chapman, P.E.
19	Market Studies and Contracts	M3	Daniel H. Neff, P.E.
20	Environmental Studies, Permitting and Social or Community Impact	Walsh & GRE	Edmundo Laporte-Ramírez, P.E. & Chris Chapman, P.E.
21	Capital and Operating Costs	M3, IMC & GRE	Daniel H. Neff, P.E., John Marek, P.E. & Chris Chapman, P.E.
22	Economic Analysis	M3	Daniel H. Neff, P.E.
23	Adjacent Properties	IMC	John Marek, P.E.
24	Other Relevant Data and Information	M3	Daniel H. Neff, P.E.
25	Interpretation and Conclusions	M3	Daniel H. Neff, P.E.
26	Recommendations	All	Daniel H. Neff, P.E.
27	References	M3	Daniel H. Neff, P.E.

Abbreviations: ALL – All QP Contributors; IMC – Independent Mining Consultants, M3 – M3 Engineering & Technology Corporation, Blue Coast – Blue Coast Metallurgy, Ltd.

Note: Where multiple authors are cited, refer to author certificate (Appendix A) for specific responsibilities.

This Report has been compiled for BCM by M3. The Report is based on information and data supplied to M3 by BCM and other parties. M3 has relied upon the data and information supplied by the various qualified persons listed above as being accurate and complete.

M3 and the other authors have relied on information provided by BCM and on information provided in a previous Technical Report prepared by Allan V. Moran, R.G., C.P.G. of SRK Consulting (U.S.) Inc. during March of 2006 and the Technical Report produced by Vector in 2009. Where possible, M3 has confirmed the information provided by comparison against other data sources or by field verification.

Where checks and confirmations were not possible, M3 has 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, M3 has 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 M3's 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. Chris Martin, C.Eng., of Blue Coast Metallurgy, Ltd., 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.

John Marek, of Independent Mining Consultants, Inc. visited the project site from during the week of 10 July, 2006 to observe firsthand the project site, observe drilling/sampling/logging practices, and to examine available drill core. In addition, available reports, cross sections, geologic interpretations and other relevant geologic data were reviewed and discussed with BCM geology personnel.

Chris Chapman, PE, of Global Resource Engineering Ltd. visited the project site on six occasions from November 2010 to November 2011. Mr. Chapman was responsible for management of the feasibility study geotechnical and environmental site investigation programs, which were ongoing at the time of this report.

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, Andy Gonzales, and Art Ibrado
- IMC
  - July 10, 2006 by John Marek
- Walsh
  - Aug 2010 by Omar Mendoza, a civil engineer (colegiado), that worked on environmental liabilities.
  - Aug 2010, Mar 2011 by Pedro Uipan, a biologist (colegiado), that worked on the biological baseline information.
  - Aug 2010, Oct 2011 by Carlos Huatuco, an agronomist (colegiado), who worked on the physical baseline
- GRE
  - Nov 2010, March 2011, June 2011, July 2011, Aug 2011, Nov 2011 by Chris Chapman. March, 2011 and August, 2011 by Chris Chapman and Rob Dorey.

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 4th Quarter 2011 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
CCR	Central Control Room
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
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)
IMC	Independent Mining Consultants, Inc. (Tucson, Arizona)
INACC	Instituto Nacional de Concesiones y Catastro Minera

Abbreviation	Definition
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
SAMPNO	Sample Number
SOW	Scope of Work
TSF	Tailing Storage Facility
Vector	Vector Peru S.A.C. (Lima, Peru)
WRF	Waste Rock Facility

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.
Decant pond	Body of supernatant water that has separated from the tailing solids (process water), plus any rainfall runoff collected on the tailing storage facility.
Quebrada	A term in Spanish, meaning gorge, valley or draw.
Supernatant water	Process water that has separated from the tailing solids in the tailing storage facility.
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)

Unit Abbreviation	Definition
gm/t	gram per tonne (metric); alternate spelling
h	hour
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 contributors to this report have, in the preparation of this report, relied upon certain reports, opinions and statements of other experts, i.e. persons that are not considered an independent "Qualified Person", and the technical data, information and reports listed in Section 27.

In cases where the M3 Feasibility Study author, Dan Neff, P.E., Qualified Person, has relied on contributions of the Qualified Persons listed in Appendix A, the conclusions and recommendations are exclusively the Qualified Persons' own. The results and opinions outlined in this report that are dependent on information provided by Qualified Persons outside the employ of M3 are assumed to be current, accurate and complete as of the date of this report.

Reports received from other experts have been reviewed for factual errors by BCM and M3. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statements and opinions expressed in these documents are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of these reports.

M3 has not reviewed the drilling results, the pit design, the status of the mining claims, or the land and incorporation documents.

The metal prices utilized for the Base Case financial analysis are set at the consensus price deck. Metal prices change rapidly. There is no guarantee that the costs and financials presented herein will be accurate.

Mining is a risky business. The risk must be borne by the Owner. M3 does not assume any liability other than performing this technical study to normal professional standards.

3.1 MINING CONCESSIONS

On March 15, 2007, BCM 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.

On February 3, 2011, BCM entered into an additional amendment agreement (the "2011 Amendment Agreement"); accordingly, BCM has paid the entire obligation for its 100% interest in the Corani project.

A title opinion dated 21 October 2011 was prepared by Estudio Grau Abogados (Estudio Grau, 2011) with respect to the mining concessions on the Project property. The opinion stated that the claims are in good standing and that BCM owns the title. M3 is relying on this opinion with regard to the status of the mining claims and has not performed any other review on the subject.

3.2 GEOLOGY AND RESOURCE DEFINITION

Geological characterization of the Corani deposit has been developed by IMC as described in sections prepared by IMC in this report.

3.3 RESERVES AND MINE ENGINEERING

The reserve and mine engineering section of the study was performed under the responsibility of IMC.

3.4 METALLURGY AND PROCESS ENGINEERING

The latest metallurgical program for the Corani Deposit was developed by Blue Coast Metallurgy after a thorough review of previous metallurgical test results. The latest flotation test results became the basis for the flotation plant design criteria. M3 developed the detailed flow sheets, designed the process plant layout, and sized mill equipment based on the metallurgical test results, parameters and criteria provided by Blue Coast.

3.5 POWER SUPPLY

Further information regarding power supply and costs (existing and purposed) can be found in the report titled "PFS, Corani Project, Power Supply for the Corani Project, Technical Report", (Pepsa, 2009) prepared by PEPSA Tecsult - Proyectos Especiales Pacífico S.A.

In addition, PROMOTORA and M3 updated the above mentioned report and developed in more detail the power supply infrastructure for this study.

3.6 ENVIRONMENTAL AND PERMITTING

The environmental permitting section of this report has been prepared by Walsh Peru S.A. The preliminary Closure Plan that identify potential and viable measures to be implemented during the operation, reclamation and closure phases, as well as the post-closure period have been developed by GRE.

3.7 PROJECT INFRASTRUCTURE

The plant site and related facilities were designed by M3. The tailing storage facility, waste rock facilities, and fresh water dam were developed by GRE. GRE also developed the concepts for waste handling and water management.

3.8 GEOTECHNICAL

Anddes Asociados S.A.C. carried out geotechnical site investigations and laboratory testing.

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 Claim Status

According to Estudio Grau (2011), BCM 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: Corani Project Mining Concessions

Name	Identification Code	Available Extent (Hectares)
Corani I	010289403	300.0000
Corani II	010289503	300.0000
Minazpata 1	010289203	1000.0000
Minazpata 2	010289303	300.0000
Minazpata 3	010038904	1000.0000
Minazpata 4	010357604	159.8808
Corani 100	010251005	5.0000
Corani 200	010251105	21.9730
Chaupitera	010250805	800.0000
Pacusani	010250905	900.0000
Corani 5	010068505	93.2601
Corani III	010021905	300.0074

Estudio Grau (2011) states that:

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

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

3. 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 rights.

4. 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 year 2028 is the current legal absolute limit as to when production needs to occur with respect to the mineral concessions comprising the Corani Project, failure to do so will cause their termination or expiry. 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.

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.

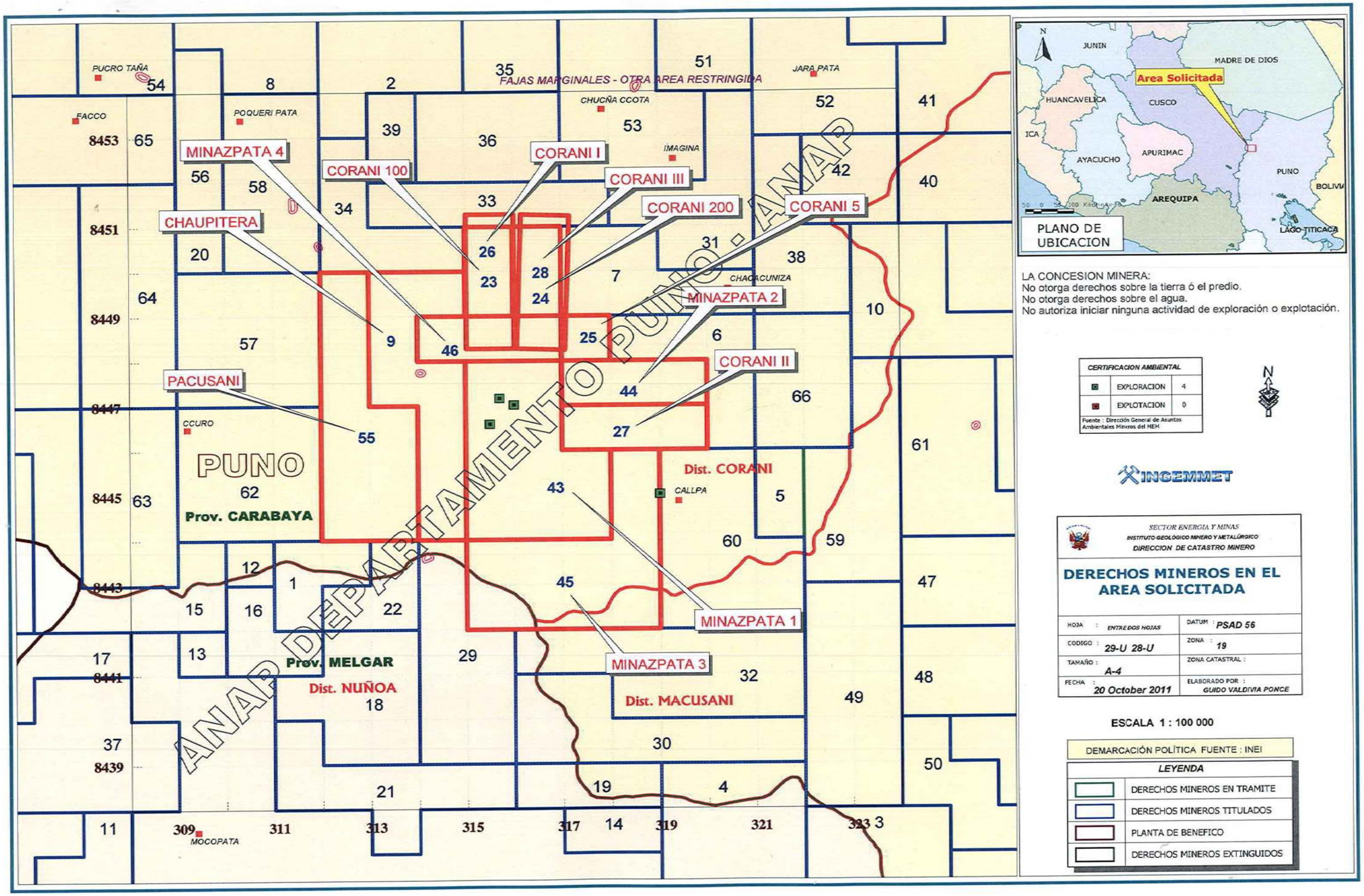


Figure 4-2: Map of Corani Mineral Concessions

4.2.4 Maintenance Obligations

According to Estudio Grau (2011), the twelve (12) mineral concessions comprising the Corani Project are subject to compliance with the 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 a 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 (3) maintenance obligations, together with timely payment of License Fees, are required to keep them in good standing.

Failure to comply with License Fee payments or Penalty payments for two (2) consecutive years causes the termination of the mineral concessions.

Table 4-2 below shows the projected annual amounts for each of the alternative maintenance obligations to be complied with to keep the Corani Project in good standing from 2011 through 2014:

Table 4-2: Corani Mineral Concession Maintenance Obligations

Mineral Concession	Annual License Fees Years 2011-2014	Alternative Annual Maintenance Obligations Years 2012-2014		
		Minimum Production in gross sales	Minimum Exploration Expenditures	Penalty Payment Payable from 2012 onwards
	(US$)	(US$)	(US$)	(US$)
Corani I	900.00	30,000.00	18,000.00	1,800.00
Corani II	900.00	30,000.00	18,000.00	1,800.00
Minazpata 1	3,000.00	100,000.00	60,000.00	6,000.00
Minazpata 2	900.00	30,000.00	18,000.00	1,800.00
Minazpata 3	3,000.00	100,000.00	60,000.00	6,000.00
Minazpata 4	479.64	16,000.00	9,592.85	959.28
Corani 100	15.00	500.00	300.00	30.00
Corani 200	65.92	2,197.30	1,318.38	131.84
Chaupitera	2,400.00	80,000.00	48,000.00	4,800.00
Pacusani	2,700.00	90,000.00	54,000.00	5,400.00
Corani 5	279.78	9,326.01	5,595.61	559.56
Corani III	900.02	30,000.74	18,000.44	1,800.04
Total	15,540.36	518,012.13	310,807.28	31,080.72

4.2.5 Other Encumbrances

According to Estudio Grau (2011), other registered liens and encumbrances for the property include the following:

1. The twelve (12) mineral concessions comprising the Corani Project are subject to a first ranking mining mortgage, in favor of Rio Tinto Mining and Exploration Limited. (See point 3 below, as this mortgage is in the process of being officially removed).

2. This first ranking mining mortgage was granted on 16 July 2008, and is duly registered with the Peruvian Public Registry.

3. The secured obligations under the above mentioned Mining Mortgage Agreement have been duly fulfilled and performed. Nevertheless, the cancellation of the mining mortgage is pending to be formalized, at which time the mortgage to Rio Tinto will be eliminated.

4. The mineral concessions comprising the Corani Project are not subject to any kind of contractual royalty. However, in accordance with Peruvian law, once in production, they will be subject to legal royalty now levied on the “operating profit” obtained by the mining agents in each quarter (calendar), through a sliding scale, which starts in 1% and reaches up to 12%, on the basis of the operating margin registered during the respective quarter. The amount effectively paid as royalty is deductible as an expense for the corporate Income Tax.

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 will need to be prepared in accordance with the guidelines established by the General Directorate of Mining Environmental Affairs (DGAAM) of the MINEM and then approved by the same agency. Failure to disclose liabilities and/or comply with approved plan can result in fines and charges.

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 (Vector Peru, 2009). 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.

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). This assessment process relies on a degree of certainty in relation to the project description and, therefore, it is usually undertaken when sufficient technical detail is available for the elaboration of a feasibility study, with the two being developed concurrently.

The first phase of the ESIA is the collection of appropriate baseline data from the project site and other potentially impacted areas. BCM has completed the physical and biological baselines in accordance with Peruvian ESIA regulations. The social baseline is currently under contract and the field collection phase will begin in the following months. The next phase in the development of the ESIA is the social and environmental impact analysis and development of a comprehensive environmental and social management plan. BCM has prepared a scope of work for the preparation of the ESIA documents required to obtain approval from the MINEM, the designated approving authority. The scope of work includes the following:

- The development of a detailed project description including provision for collaboration between environmental specialists and the mine design team.
- Alternatives analysis.
- Environmental and Social Baseline including revision of existing information.
- Identification and Analysis of potential and anticipated, environmental, social and archaeological impacts resulting from each Project phase.
- Development of impact mitigation and management methods presented as a comprehensive Environmental and Social Management Plan
- Environmental Cost-Benefit Analysis
- Provision of community consultation services including community workshops and public hearings and the development of a community relations plan

In addition to an approved ESIA, a certificate of the non-existence of Archaeological remains is required prior to ground breaking activities. This is usually applied for concurrently with the ESIA. The archaeological baseline report, developed in September 2011, indicates that no archaeological remains were discovered. However confirmation of this is required by the Ministry of Culture.

The statutory timeframe for approval of the ESIA is established as a maximum of 120 days from the time of submission. This accounts for time required for MINEM to request further clarification (observations) if required and for the applicant to respond. However, in practice the time for approval is highly variable and could take between 6 and 12 months.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

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.

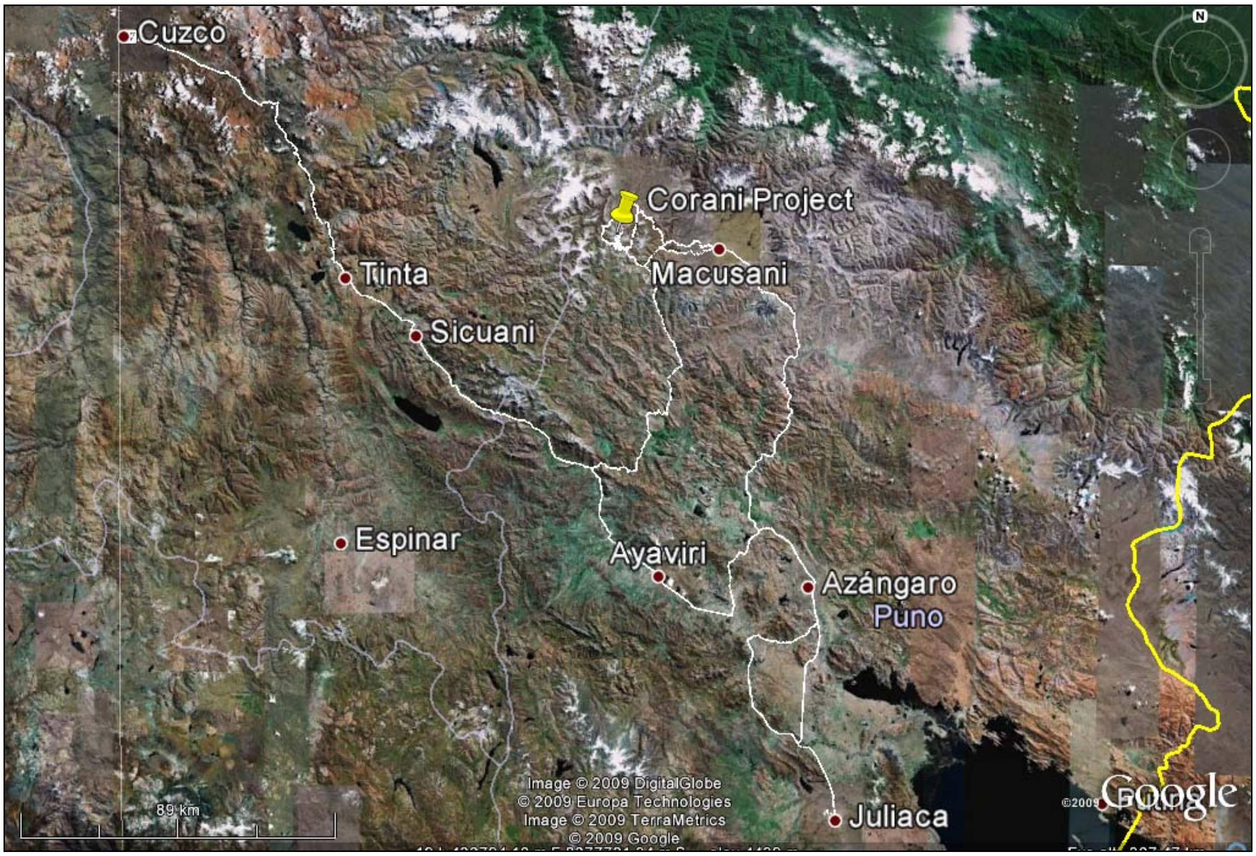


Figure 5-1: Map of Existing Access to the Project

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 below 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.

The proposed Mine Access Road is described in Section 18.1.1, and will connect the mine process plant to the town of Macusani.

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 length of the proposed Mine Access Road connecting the process Plant to the town is anticipated to be approximately 63 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 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 neighbouring 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.

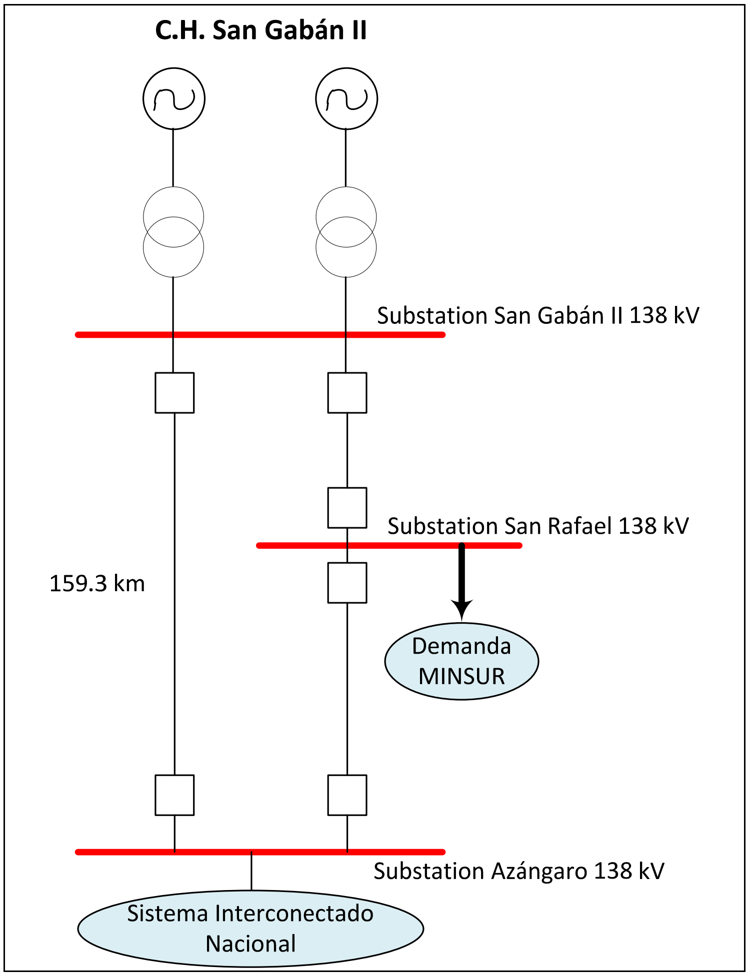


Figure 5-2: Existing Facilities Schematic

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

MINSUR, a mining company focused on the production of tin, has installed the transformer called Substation San Rafael of 24/30 MVA, ONAN/ONAF, 138/10kV between the transmission lines L-1013 and L-1009.

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.

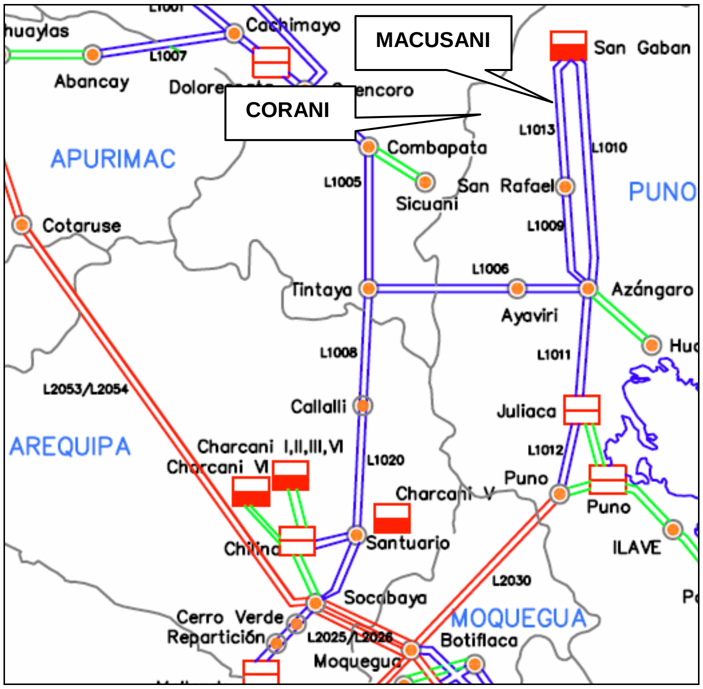


Figure 5-3: Southern Electric Ring Schematic

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. Three 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.

5.5 SURFACE RIGHTS

Land acquisition in Peru involves purchase of the title to the land, plus purchase of surface rights to use the land. The title and surface rights may not belong to the same entity. BCM owns over 80% of land and surface rights required for the project. Purchase of the remaining 20 percent is actively being pursued and is estimated to cost BCM about 1 million US dollars. This amount has not been included in the capital cost estimate for the Project. In addition, BCM has agreements with the two local communities for additional project development. Details of the project land positions are available at BCM's corporate office to serious inquiries.

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. 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 1965, Compañía Minera Korani increased 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. 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).

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 4.2.

6.2 HISTORICAL EXPLORATION AND ESTIMATES

There have been four 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 25, 2006, National Instrument 43-101 Technical Report, Initial Resource Estimate for Corani Silver-Gold Exploration Project. SRK Consulting;

2) October 4, 2006, Corani Project Mineral Resource Technical Report, Independent Mining Consultants, Inc.; and

3) May 12, 2008, Technical Report, Corani Resource Estimate and PEA, Independent Mining Consultants, Inc.

4) October 14, 2008, NI43-101 Technical Report, Prefeasibility Study Corani Project Puno Perú, Vector Perú S.A.C.

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.

Table 6-1: Mineral Resources on 31 March 2006 (As Published by SRK Consulting)

Category	Kilotonnes	SILVER g/t	LEAD %	ZINC %	COPPER %
Measured	7,759	65.12	1.081	0.162	0.035
Indicated	20,123	43.61	0.678	0.251	0.033
Measured + Indicated	27,882	49.60	0.790	0.230	0.034
Inferred	87,627	72.91	1.032	0.578	0.034

Table 6-2: Mineral Resource on 4 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 All Deposits	Measured	24,944	72.6	1.06	0.59	58.2	582.7	323.5
	Indicated	115,457	53.4	0.87	0.49	198.3	2,215.9	1,254.2
	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**

(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 on 12 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	Measured	36,716	55.9	0.90	0.56	66.0	731.3	453.8
All	Indicated	211,705	38.4	0.70	0.43	261.1	3,245.5	2,013.9
Deposits	**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

(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 on 22 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.

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 surface 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 Palaeozoic (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.

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

A sequence of Tertiary (22.1Ma +/- 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 identical to the lower tuffs, is effectively barren. The postmineral sequence forms prominent spires and thickens to the north (from 0 meters to over 200 meters according to project drilling).

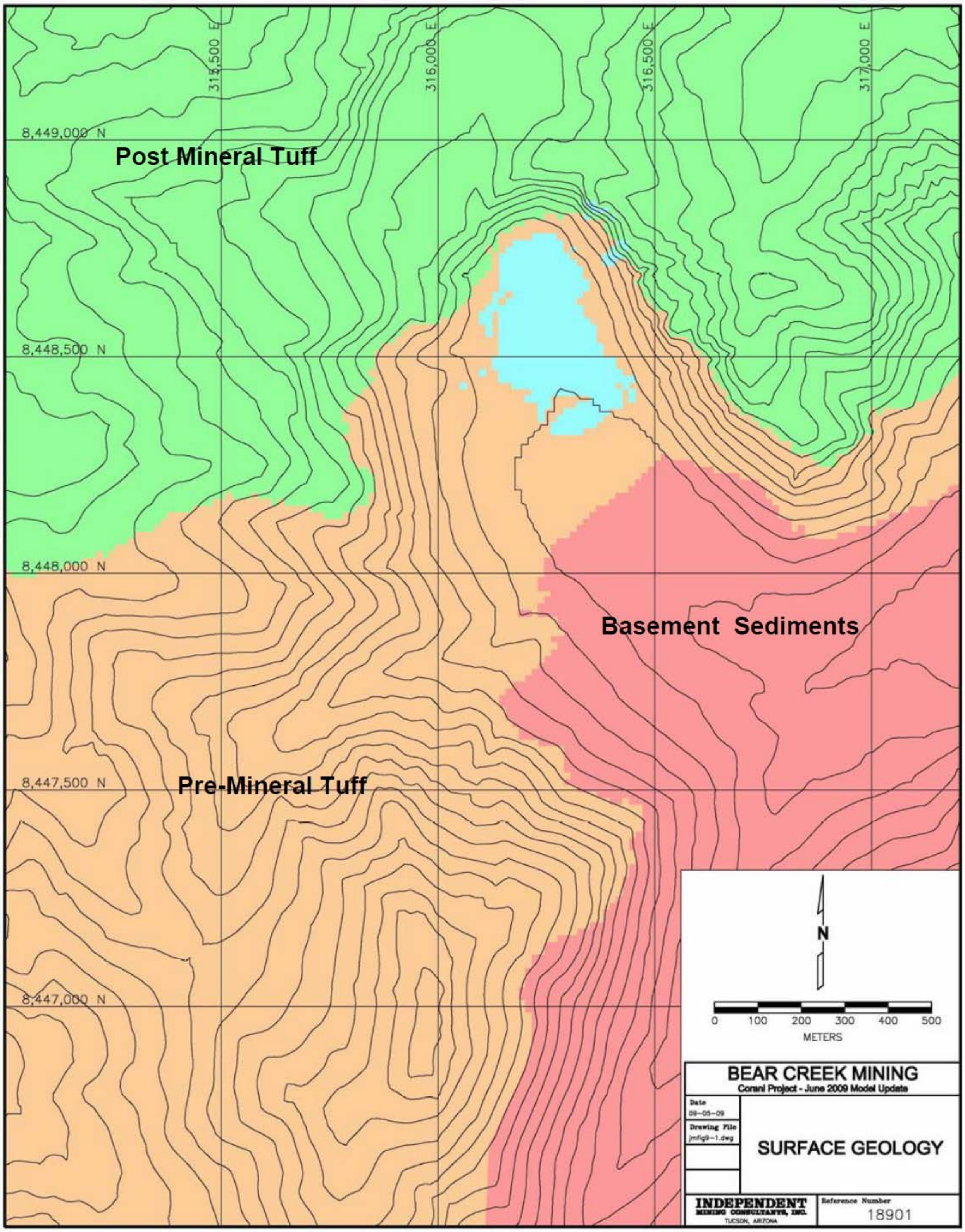


Figure 7-1: Surface Geology in Model Area

7.2.2 Alteration

Alteration in the Project area consists of a broad, 5 x 2 kilometers 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 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 as follows:

- Este;
- Minas; and
- Main.

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)), 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 (Ag-Tetrahedrite) 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 rapidly cooled (quenched) ore fluid with highly variable metal contents, this is mineral type forms the largest portion of the ore tonnes;

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

- Manganese oxides occur as mainly Ag but also Pb-Zn within Mn.

Approximately 3 kilometers south of the Project resource area, mineralization in the Gold Zone is hosted in the same 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 holes in the Gold zone with the best intervals as:

- 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 kilometres is an area of stibnite mineralization in discrete, narrow, veins and breccias zones referred to as the Antimony zone. Currently, BCM is drilling condemnation and additional exploration drill holes in the southern part (5 km from the current ore body) and it has encountered mineralization. BCM drilled 2 holes in this zone and neither hole had a significant mineralization. Previous Technical Reports have described the Gold and Antimony Zones in more detail. 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 22 million year old 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 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 listric faults and also contain significant lead-zinc-silver mineralization. The stockwork veins tend to be mineralized with the fine black-sulfide mineralization indicating that the sulfide minerals were emplaced rapidly into the rock and quenched quickly. The stockwork veins tend to be randomly oriented so it is felt that the vein formation was controlled by fluid movement into the broken or weakened rock mass which was caused by the regional tectonic activity which formed the listric faults.

7.2.5 Structure

The structure of the rock units varies greatly between the three principle 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. For 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 rock has principal listric faults that have been interpreted in drill core to occur approximately every 150 meters and are continuous through the deposit along strike.

In between the listric faults the rock mass has been broken and weakened and a random orientation of structures has formed. Since the principal vein forming faults are listric faults they have a curved presentation and as you travel down dip the fault dip decreases until it becomes horizontal. Because of this horizontal coalescing of the principle veins there is a well-defined bottom to the mineralization in the mineralized tuff unit. Above the “bottom” forming fault the rock is hydrothermally altered and strongly deformed. Below the “bottom” forming fault the rock appears very fresh and 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 higher 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 along the “bottom” fault so the mineralization is weak.

The 10 million year old 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 very vertical joint pattern which in turn 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 sulfidation epithermal silver, lead, and zinc mineralization within stock works, 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 features 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. during July of 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, and galena. Zinc mineralization in the form of sphalerite may or may not be collocated 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 the lead metal can also occur as Plumbogummite, a lead aluminum phosphate. This mineral has a lower flotation recovery than lead sulfide minerals. Roughly 14% of the reserve process ore tonnage and contained metal is categorized as Plumbogummite rich ore. About 9% of the recoverable lead is attributable to Plumbogummite rich ore group.

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. These are provided to indicate the general locations of these metals relative to the locations of silver.

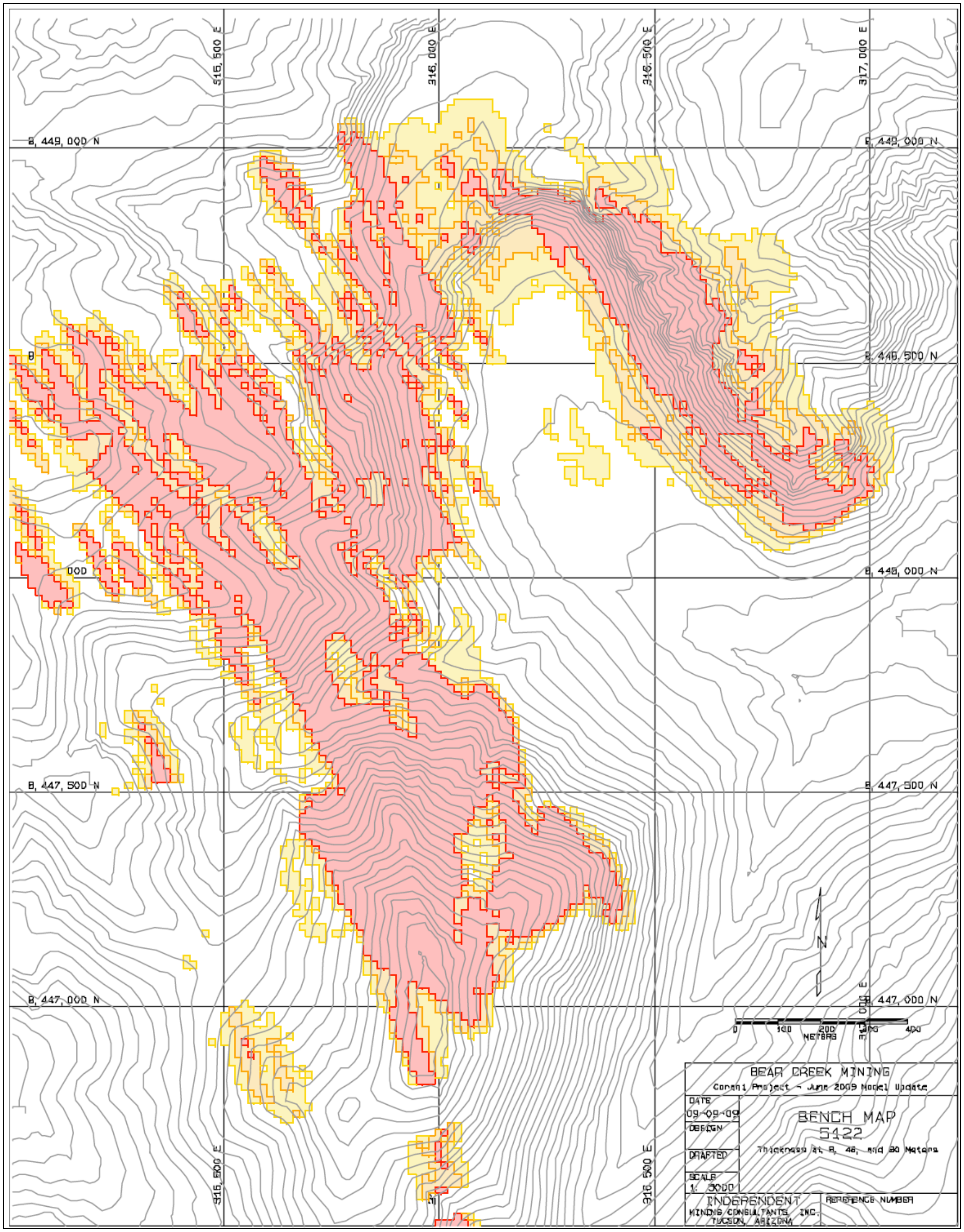
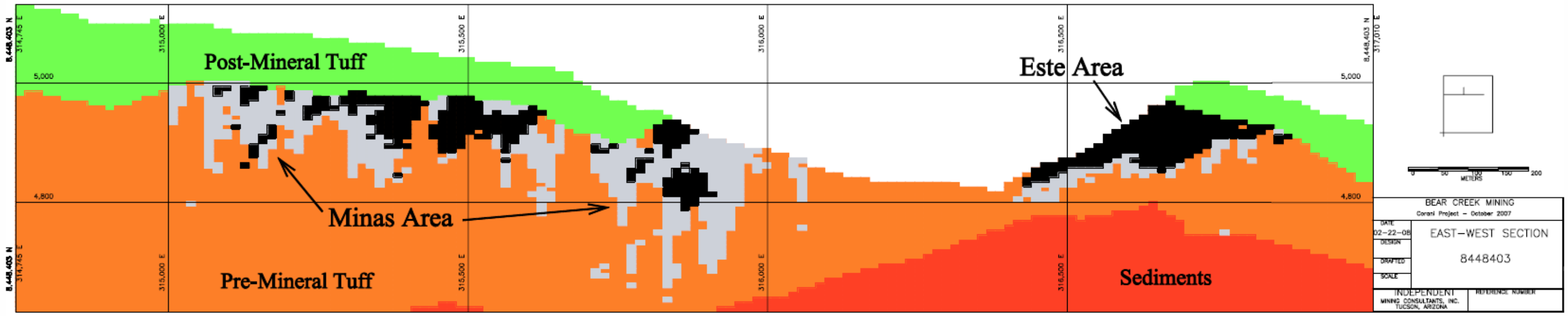


Figure 7-2: Thickness Map of Silver Mineralization (15 g/t)



Silver Grade Shading
Black = + 50 gm/t
Grey = + 16 gm/t

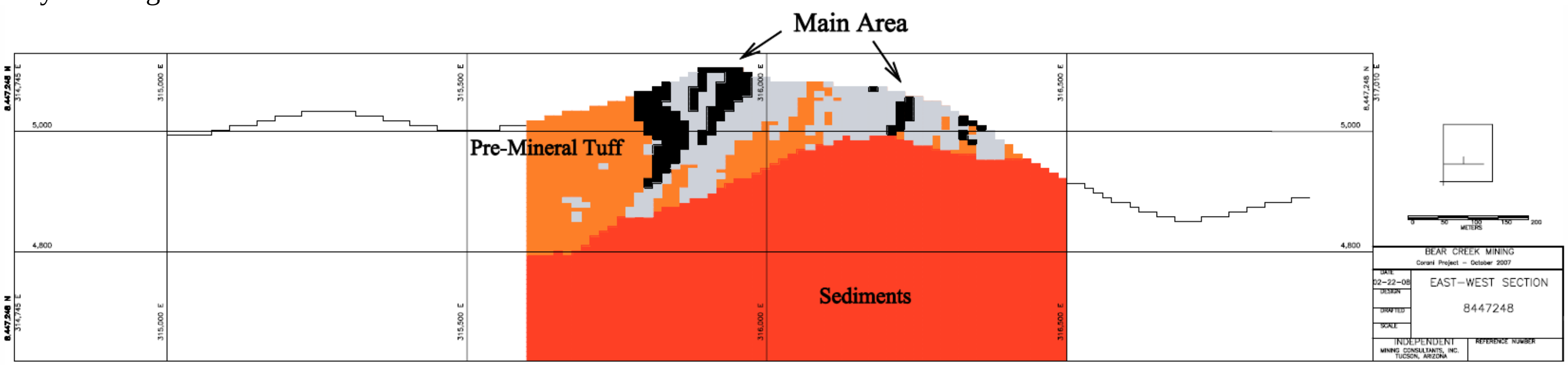


Figure 7-3: East-West Cross Sections Looking North

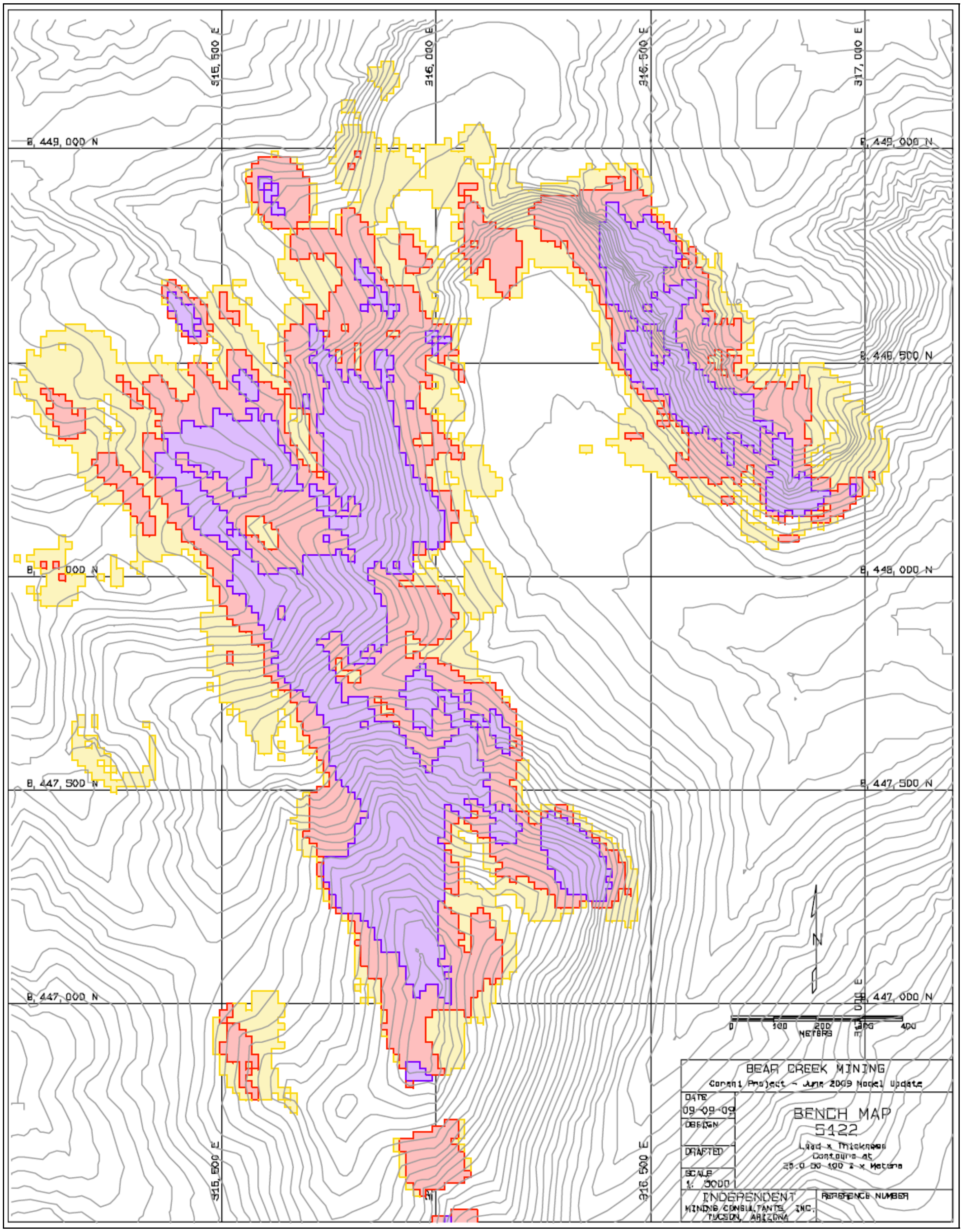


Figure 7-4: Grade Thickness Contours for Lead at 25, 50, 100% X Meters

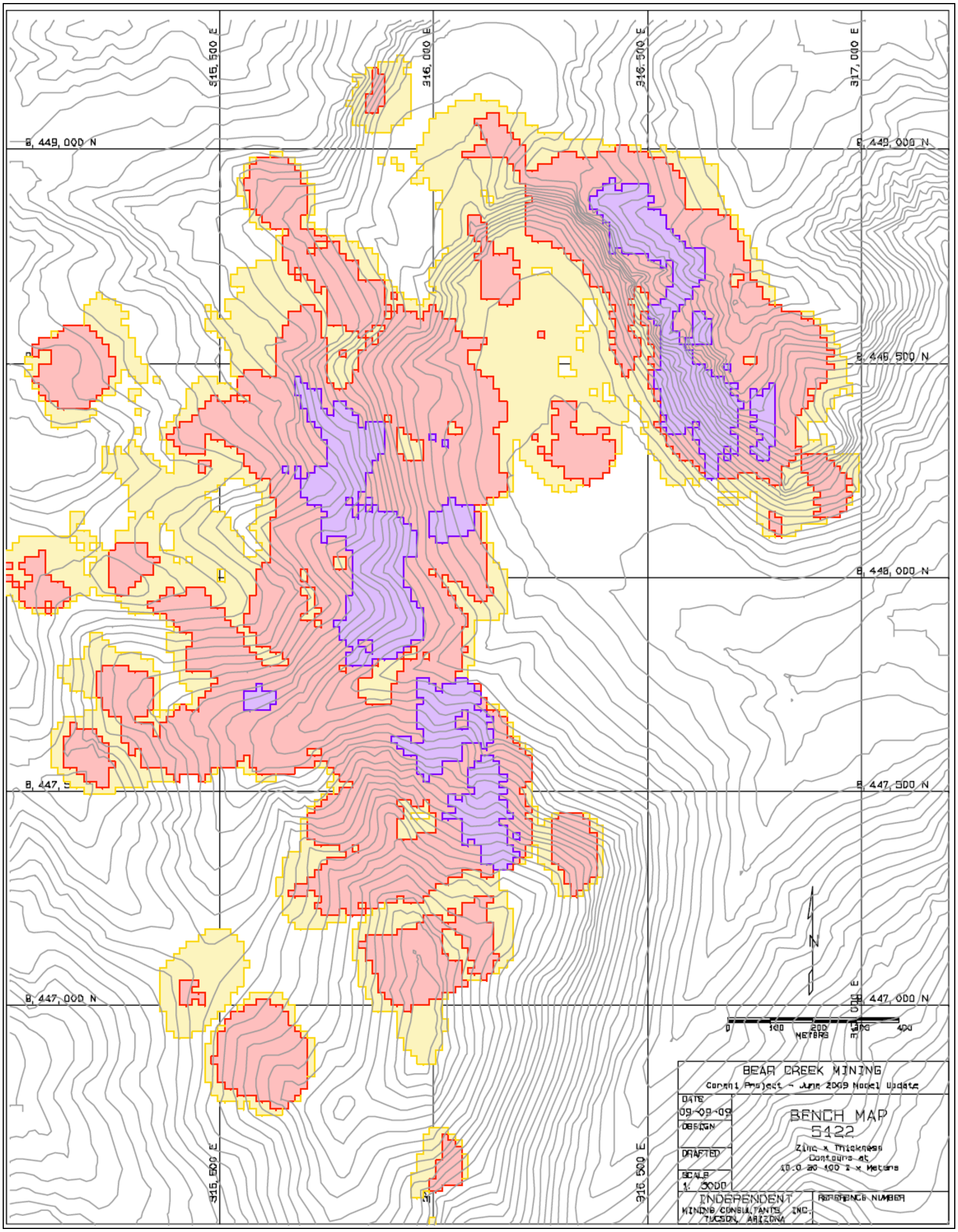


Figure 7-5: Grade Thickness Contours for Zinc at 10, 30, 100% X Meters

7.3.2 Mineral Ore Types for Process Metallurgy

The mineralogy types discussed previously were separated into mineral zones of like process response. Those zones were interpreted by BCM geologists and assigned to the block model by IMC for use in mine planning and process response estimation.

The mineralization types or geo-met type (short for geological-metallurgical types) are each described below. In addition a metallurgical type is given for each. The metallurgical types arise from studies using QEMSCAN analysis matched with diagnostic flotation tests. The metallurgical types are tabulated in Table 7-1 below with a brief description.

- 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. Met Type I;

- CS - A subset of CSC that contains coarse galena-sphalerite-chalcopyrite +- tetrahedrite without green celadonite clay. Met Type I;

- 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 floatation or leach: normally ores with low Pb-Zn contents. Met Type I;

- PM – Pyrite-marcasite +- quartz typical of early-stage mineralization with little polymetallic mineralization. Met Type II;

- 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. Met Type II;

- QSB - Crystalline quartz-sulfide-barite interpreted as early fault fill or late-stage breccia fill. Met Type II;

- PG – Plumbogummite, identified as a pale-green, waxy, Pb-phosphate mineral that in metallurgical test results shows diminished lead floatation and difficulties in separation of base metals. Met Type III;

- FeO – Iron-oxide mineralization with locally elevated Ag and generally low Pb-Zn. This is a gradation zone with mixtures of FeO and FBS exhibiting Type III Met Type response and the most strongly oxidized areas showing Type IV(a) high Ag leach recovery results; and

- MnO – Manganese-oxide mineralization hosting mainly Ag with lesser Pb-Zn with very poor response to floatation and leach tests. Met Type IV(b).

Table 7-1: Metallurgical Types

METALLURGICAL TYPE	FLOTATION RESPONSE	PB/ZN SEPARATION	CYANIDE LEACH RESPONSE	GENERAL MINERALOGY/PREDOMINANT ORE TYPE	OCCURRENCE
Type I	Very Good	Good	Poor	Coarse Sulfides – CSC, Tet, CS	Sulfide section of East Minas, Central Este and East Main
Type II	Good	Good only with flotation extenders (EDO)	Poor	Fine Sulfides – FBS, PM, QSB	Sulfide sections of all deposits
Type III	Diminished	Not relevant since generally low in zinc	Poor	Mixed Fe oxide and sulfide – Pg, FeO, with minor FBS, PM, QSB	Oxide sections of Este, East Minas and East Main
Type IV-Leach	Poor	Poor	Good	Mixed Fe oxide and sulfide – FeO and Pg predominant, with minor FBS	West Minas, West Main and Este Minas connection
Type IV-No Leach	Very Poor	Poor	Poor	Oxides with high amounts of Mn oxide – MnO, FeO	Limited parts of West Minas

The mineral ore types as assigned to the block model were as follows with the corresponding percentage of occurrence (by total tonnage) within the ore bearing pre-mineral tuff and within the reserve pit design:

- 49% Fine Black Sulfide (FBS)
- 24% Pyrite Marcasite (PM)
- 12% Plumbogummite (PG)
- 7% Quartz / Sulfide / Barite (QSB)
- 4 % Iron Oxide (4) (Poor flotation response) (FeO)
- 3 % Iron Oxide (3) Transitional flotation response (FeO)
- 1% Coarse Sulfide and Celadonite (CSC)
- Less than 1 % Manganese Oxide (MnO)

Figure 7-6 illustrates the location of the mineral ore types on the final Pit and topography as a result of reserve production.

Within this feasibility study, the geo-metallurgical types were combined into two broad categories. The FBS, PM, QSB, and CSC were combined into a common group referred to as the “Mixed Sulfides”. The PG and FeO(3) were combined and referred to as “Transition” material. FeO(4) and MnO were not considered for processing within the mine plan or mineral reserve. FeO(4) was considered as potential heap leach material within the mineral resource.

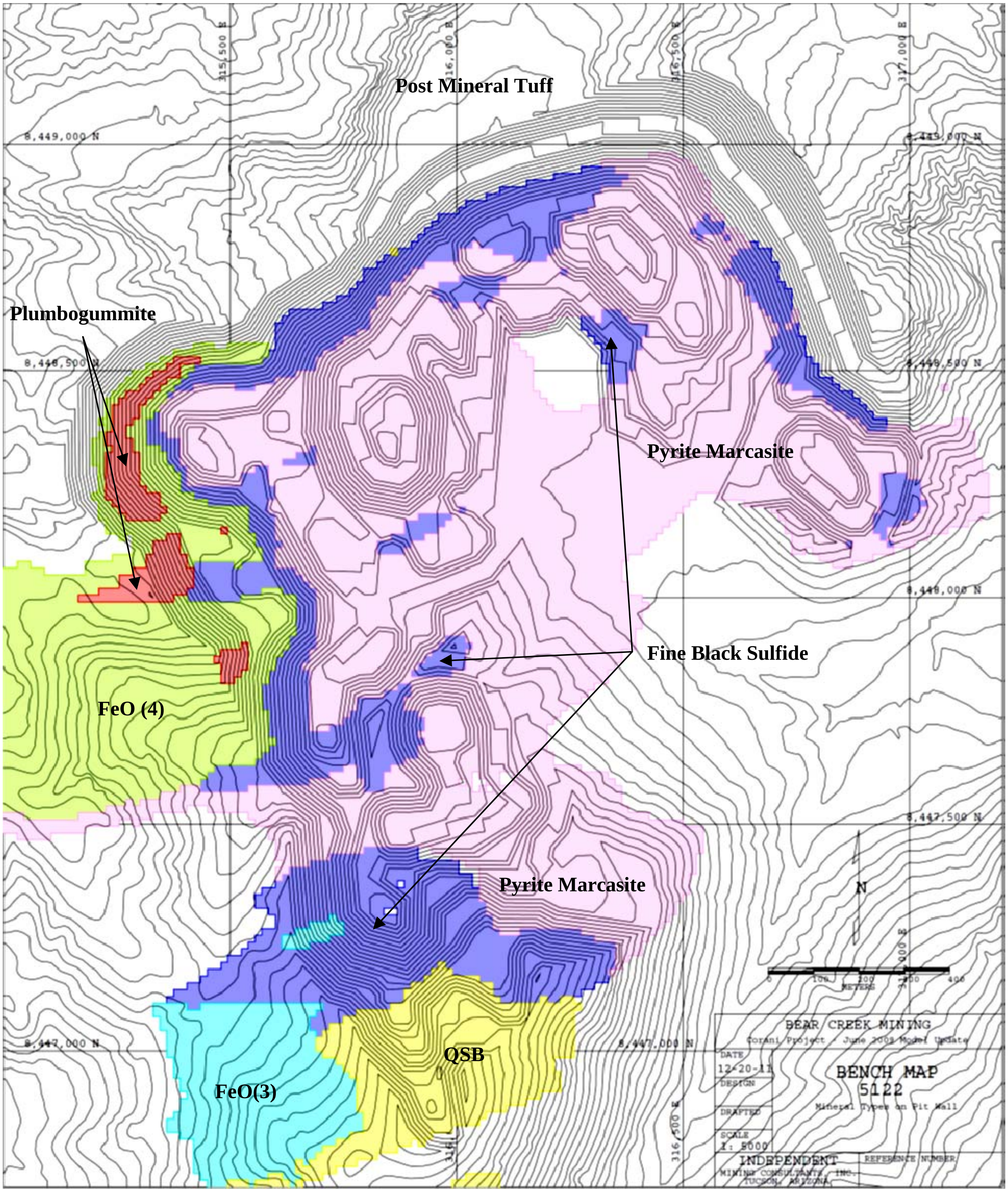


Figure 7-6: Mineral Types in Final Pit

8 DEPOSIT TYPES

The Project resource is a low sulfidation epithermal silver, lead, and zinc deposit 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 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.

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 four additional trenches in the Gold Zone to the south.

Diamond core drilling was in progress during the IMC site visit in July 2006 and is reported to have continued through March 2008. The total Corani district diamond drilling completed by BCM prior to May 2009 has been 544 holes representing 93,576.5 meters of drilling Samples were sent to an assay lab in Lima for preparation and assay. The database has been maintained by the BCM staff in the Lima office.

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 exploration plan calls for recovering samples for detailed metallurgical testing which will concentrate on developing specific composites of the material that will be seen 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 drill targets.

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 of 6.36 cm diameter (2.5 inches). The block model for this study was based on the drill hole data that was complete by May of 2009.

The drill pattern is generally a series of drill fans on sections spaced 50 m apart. The fans are located perpendicular to the strike of the individual deposits. Angle holes are used to try and cross 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 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 IMC resource estimate, 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 as they have not been closed off by drilling in all directions.

Figure 10-1 presents a drill hole location map of the available Project data as of May 2009. 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 May 2009 was as follows:

- Number of drill holes 519
- Number of sample intervals 37,073
- Total meters of drilling 90,652.7 meters
- Number of silver assays 35,958
- Number of lead assays 35,958
- Number of zinc assays 35,958
- Number of copper assays 35,958

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 544 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 458
- Number of sample intervals 33,405
- Total meters of drilling 83,122.8 meters
- Number of silver assays 32,404
- Number of lead assays 32,404
- Number of zinc assays 32,404
- Number of copper assays 32,404

Drill hole collars were originally surveyed by hand held GPS with a reported accuracy of +-3.0 meters. Since 2008, BCM has had all of the drill hole collars resurveyed by conventional survey with substantially higher precision. Comparisons of the new collar elevations with the surface topography map indicate that there are a number of collars (about 15%) where the difference between the topography and the collar survey exceeds 5 m in elevation. Most of these (2/3) occur on steep topography where the hole collar is below topography, indicating the cut required to establish the drill pad.

Prior to late 2007, down hole surveys were not applied to 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 and down hole surveyed to confirm the presence and location of the grade intercepts. The average depth of the 12 surveyed holes was 220 meters. If one compares the down hole survey location of the last interval in the drill hole with the location had one used only 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 drill hole location without survey would have been within one model block.

The indication from the 12 surveyed holes is that the lack of down hole 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 down hole surveys should be implemented for all future drilling.

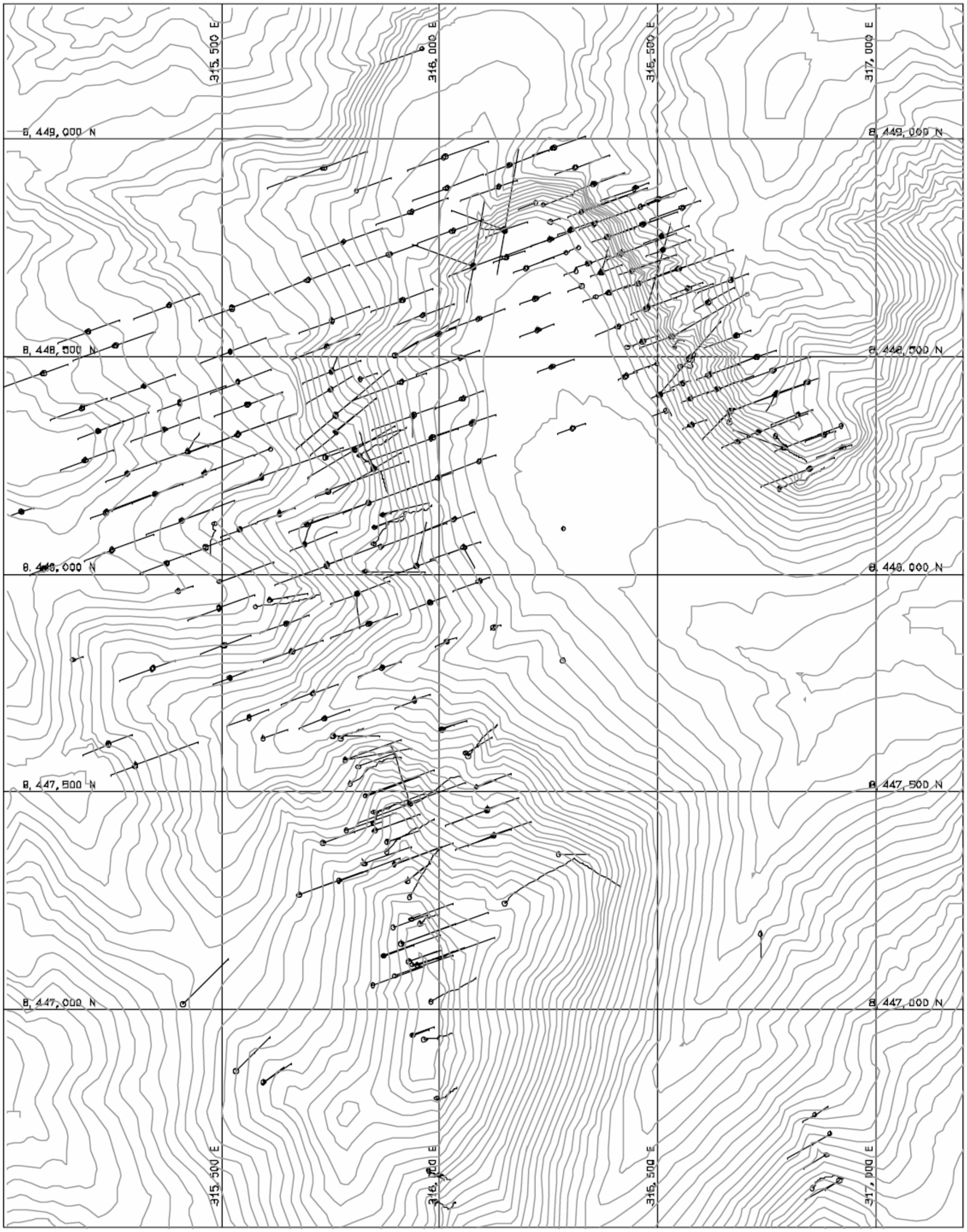


Figure 10-1: Drill Hole Location Map

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 core. The whole core is boxed into plastic and weatherproof cardboard boxes at the drill rig by the drill crew. IMC personnel were able to observe the procedures outlined below during 2006. The following list summarizes the drill core handling process:

- Whole HQ 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. 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 reported to IMC personnel by BCM geologists in Peru and some of the trenches were observed in the field:

- The trench is hand dug by local workers to remove surface alluvium and establish a clean bed rock 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;
- Sample intervals are 2 m horizontal intervals, based on GPS survey of the start and end point;
- 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.

The channel sample database was provided to IMC after processing by BCM and a previous consultant. IMC compared the sample lengths against the recorded plunge or dip of the trench to confirm proper application of the 2 m horizontal distance to the true length of the sample on the hill side.

11.1.3 Density Data

The initial Technical Report prepared by IMC concluded that the density determinations previously used for the earlier resource calculations were unreliable. BCM has since initiated an extensive program of regular testing the rock to determine the bulk density. To do this, 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 meters. This data was reviewed by IMC and the in situ densities used in the resource estimate are shown in Table 11-1.

Table 11-1: Assigned Block Densities

ROCK TYPE	ZONE	AG GRADE	DENSITY t/m^3
Post Mineral Tuff	All	None	2.2714
Sediments	All	None	2.5214
Pre-Mineral Tuff	Main	Less than 30 gm	2.3275
Pre-Mineral Tuff	Main	Greater than 30 gm	2.4768
Pre-Mineral Tuff	Este	Less than 130 gm	2.3205
Pre-Mineral Tuff	Este	Greater than 130 gm	2.3926
Pre-Mineral Tuff	Minas	Less than 90 gm	2.3468
Pre-Mineral Tuff	Minas	Greater than 90 gm	2.5178

The density assignment methods as defined above are based on silver grade and the zone of the deposit as established in the previous resource models assembled by IMC. IMC recommends that future models evaluate the correlation of the mineral ore types and base metal grades versus the measured density. A more refined and locally precise method may be the result of such an effort.

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 reported to IMC 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);
- 250 g is split by 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 3 acids: hydrofluoric, nitric, and perchloric (Aqua Regia). This results in a cake;

- The remaining cake is leached with hydrochloric acid; and
- The hydrochloric acid solution is subjected to 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, 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 ICP analysis is applied to selected intervals at the Project site. This information is used for both exploration guidance and for process trace element information. The multi-element ICP 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

IMC and John Marek acting as qualified person have 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 IMC. 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.

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, IMC and John Marek (the qualified person) find 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. The standards are prepared by outside commercial labs. 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 now 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. IMC uses the standards to check on assay bias, and sample handling procedures.

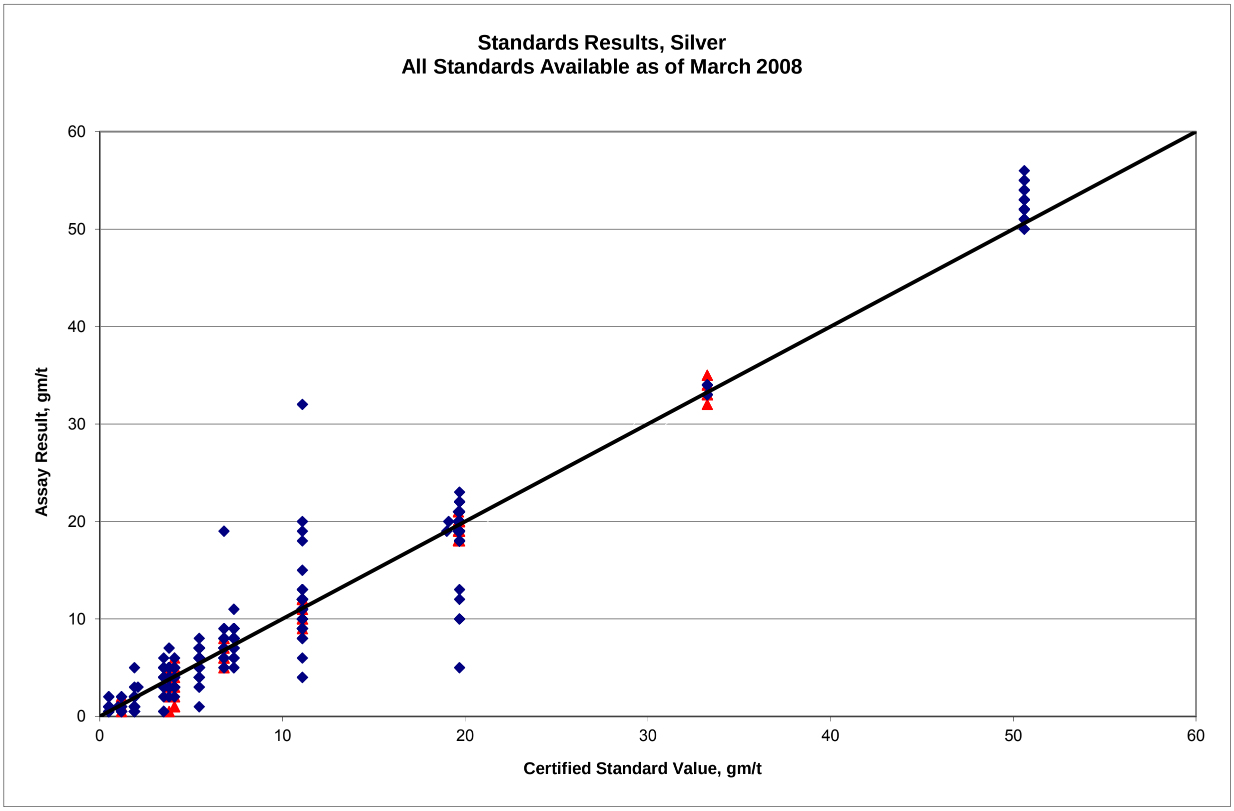
Figure 12-1 is an X-Y plot of the certified value of each standard versus the multiple reports of each standard available from ALS-Chemex. The line on the graph represents a 45 degree line or the ideal result from all values. The statistical results of the ALS-Chemex silver assays are summarized on the 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 60 gm/tonne with cutoffs in the range of 15 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. These plots represent all of the checks available since 2005. 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.

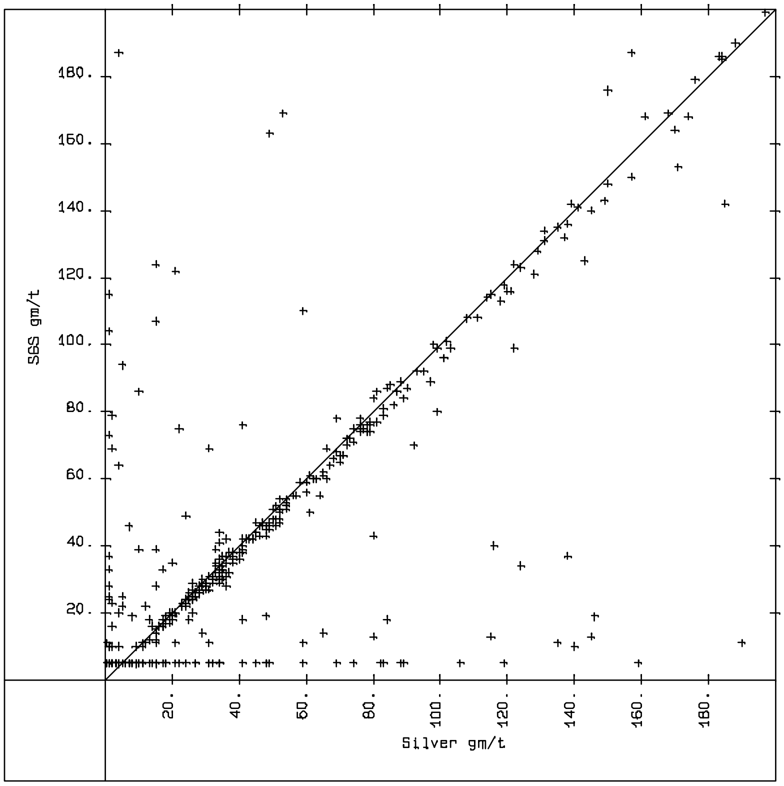
Additional plots were scanned for the period of late 2007 through March 2008. There is effectively no scatter in the plots for the most recent checks. It is not certain if the issue has been corrected or that it is not apparent with only 115 samples in the most recent check set.

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

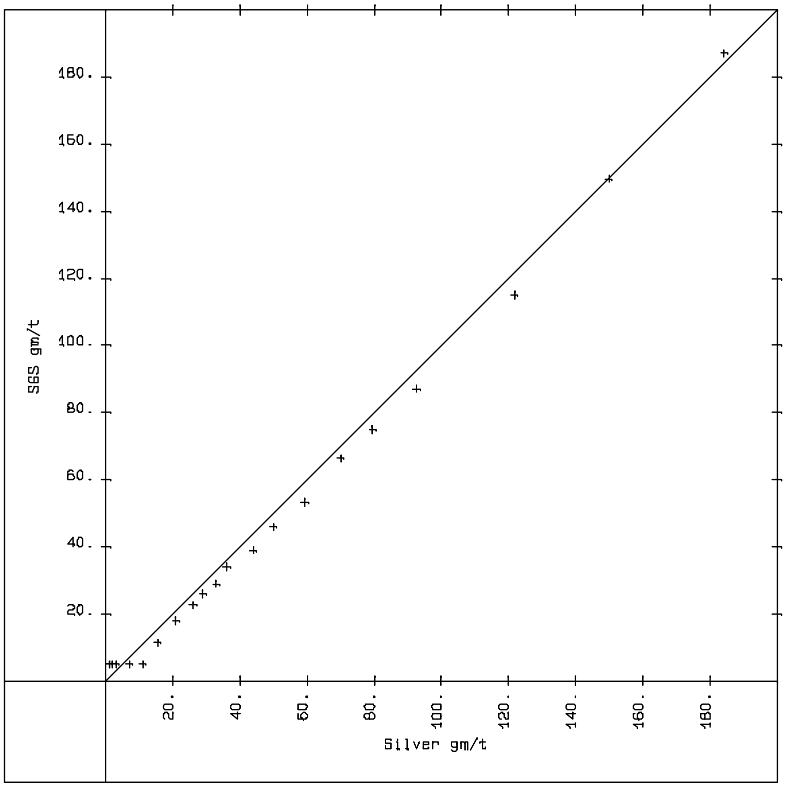
Hypothesis tests for each set of check assays do indicate that they can be accepted with 95% confidence and there is effectively no bias in the check assay result. However, the variability issue should be understood. It could simply be a function of miss-assignment of batch results to the working spreadsheet,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



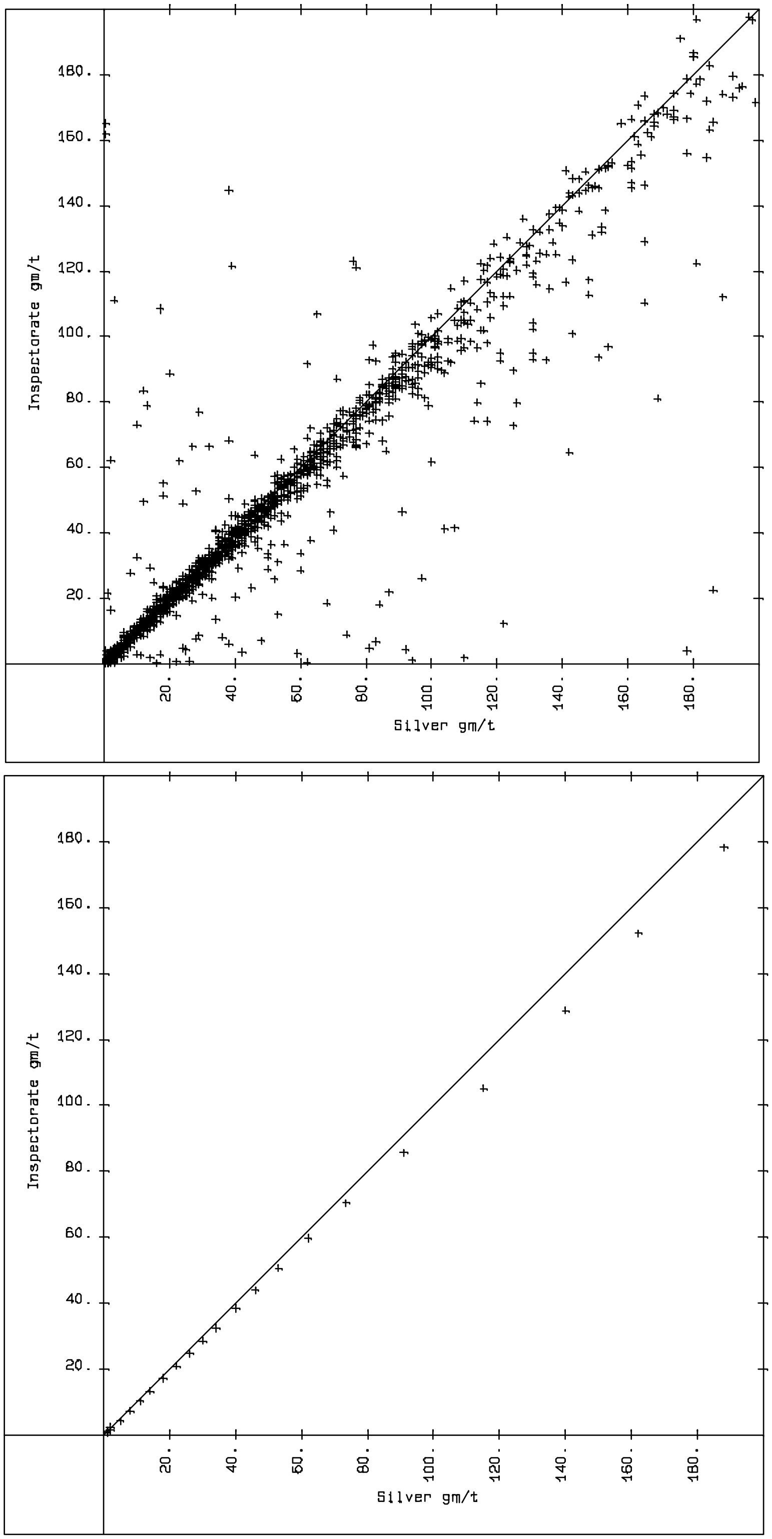
Number = 442
Original = 65.2 gm/t
SGS Check = 63.7 gm/t
T-Test = Pass at 95%
Paired T = Pass at 95%

XY Plot



QQ_Plot
5% Cell

Figure 12-2: SGS Check Assays (Silver)

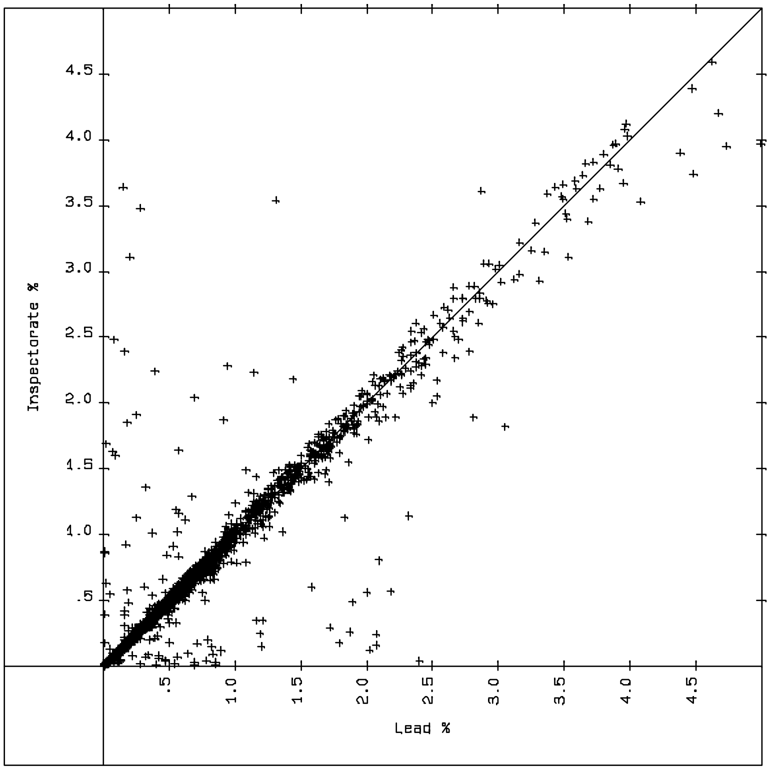


Number = 2093
Original = 48.8 gm/t
Ins Check = 47.1 gm/t
T-Test = Pass at 95%
Paired T = Fail at 95%

XY Plot

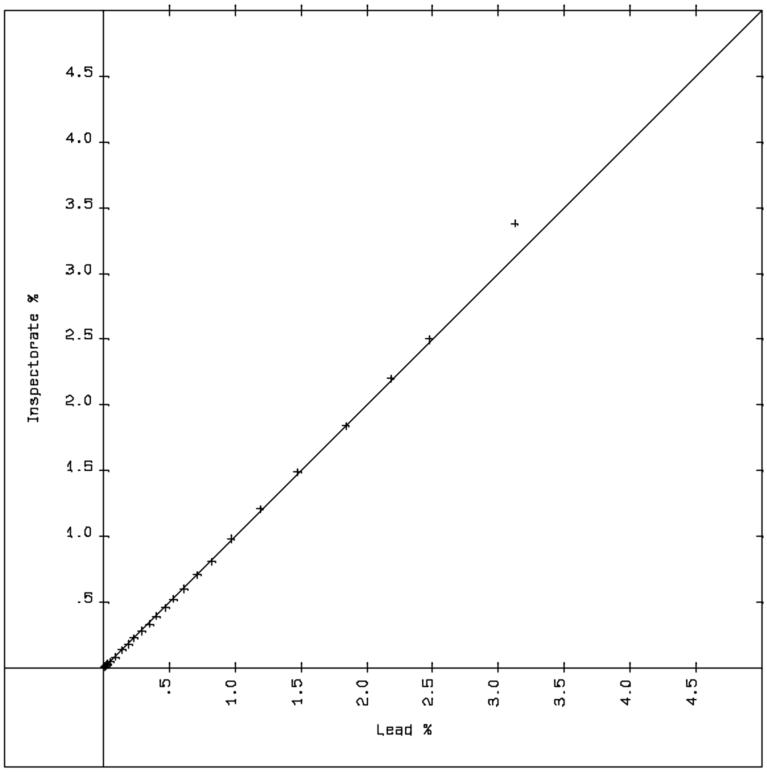
QQ Plot
5% Cells

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



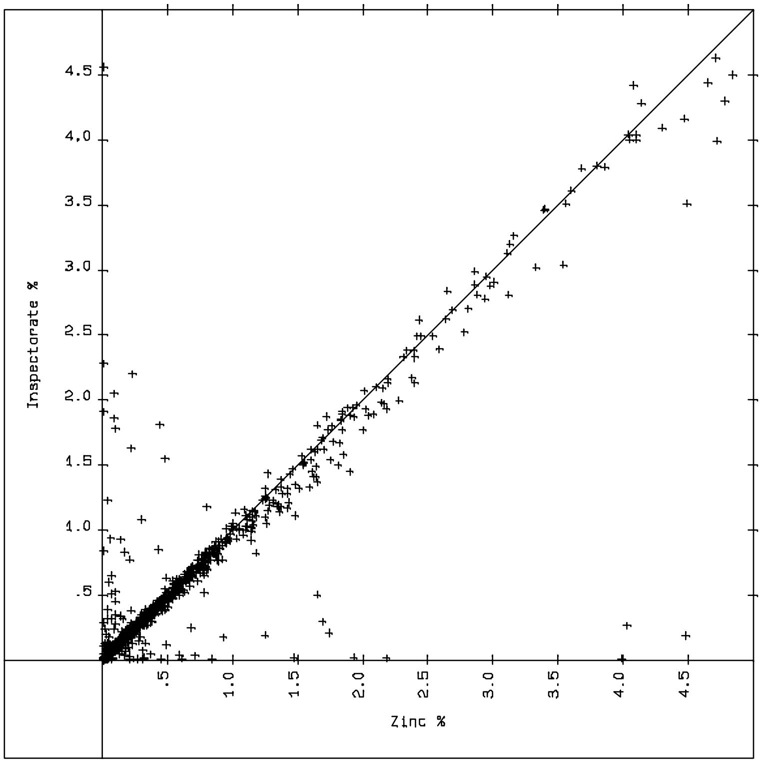
Number = 2098
Original = 0.777 gm/t
Ins Check = 0.793 gm/t
T-Test = Pass at 95%
Paired T = Pass at 95%

XY Plot



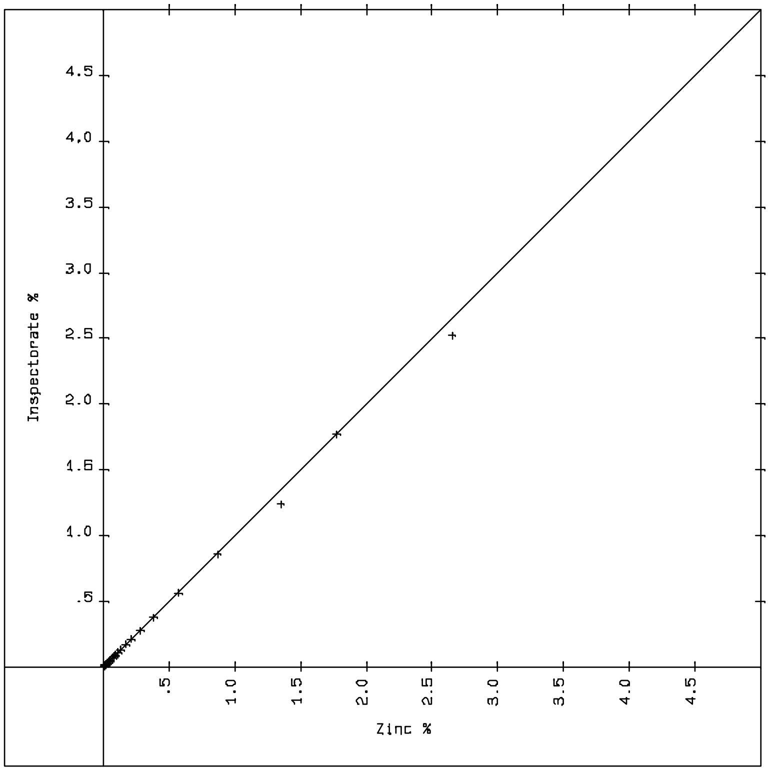
QQ Plot
5% Cells

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 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 during the project site visit. 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 that 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 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

tab12-1.xls

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 METALLURGICAL TEST PROGRAMS

Metallurgical testing of Corani ores has been on-going since approximately 2005-2006 in a variety of testing facilities across North America. In total, some 130 different composites have been tested in approximately 600 flotation tests.

Preliminary test work prior to 2007 showed that the Corani deposit exhibited significant metallurgical variability ranging from ores exhibiting good results in flotation testing to others yielding little or no flotation response. Consequently the various test work programs since 2007 have in general terms taken two forms; (1) variability programs whereby a number of composite samples are subjected to a standard, although non-optimized flow sheet to assess this variability, and (2) where a single composite sample is taken through flow sheet optimization typically over the course of about 50 tests.

The causes of this variability have been investigated extensively through use of QEMSCAN-based automated mineralogy, conducted in tandem with flotation testing. Correlations between mineral abundance or grain size data with flotation response have become a cornerstone of metallurgical understanding of Corani ores so now the mineralogical drivers governing the metallurgy of Corani ores are well understood. A broad overview of the chronology of testing programs relevant to the feasibility study is shown in Table 13-1:

Table 13-1: Chronology of Metallurgical Testing of Corani Ores

Corani Feasibility Level Metallurgical testing Chronology						
	Variability Test Programmes			Flowsheet Development Programmes		
Year	Composite Names	Testing Lab	Description	Composite Names	Testing Lab	Description
<2007						
				Preliminary Samples	Dawson	Early stage testing
				Preliminary Samples	G & T	Early stage testing
2007	G&T Variability Samples	G&T	180 batch test program on 76 variability samples			
	G&T Variability Samples QEMSCAN Analysis	SGS Lakefield	Mineralogical characterization of each G&T variability sample	Alphabet D & B Composites	SGS Vancouver	32 batch test program on D Composite and 68 batch test program on B2 Composite. B2 Composite testing considered bulk and sequential flotation options
	Alphabet Composites A-U	SGS Vancouver	2-4 Batch Tests plus selected LCT's on 20 Alphabet coded samples			
2008				Alphabet G & H Composites	SGS Vancouver	46 batch test program on G composite and 14 batch test program on H Composite
	3 Zone Individual Composites	SGS Vancouver	2 Tests conducted on 10 3-Zone Composites			
2009				3 Zone Master Composites (Pb-Zn-Ag & Pb-Ag)	SGS Vancouver	47 Batch Test + 3*LCT program on mixed sulphide master composite and 38 Batch test + LCT program on Transitional Ore master composite.
2010	1-5 Yr Individual Composites	SGS Vancouver	3-4 Batch tests on 6 new early year composites			
2011				1-5 Yr Master Composites (Pb-Zn-Ag & Pb-Ag)	SGS Vancouver	46 Test + LCT program on early mine life mixed sulphide and transitional composites

13.2 GRINDABILITY AND MILL SELECTION

Two phases of grindability testing were conducted, at SGS Lakefield in 2008 and at SGS Chile in 2010, on samples selected from around the deposit. The results are summarized below:

Table 13-2: Summary of SGS Lakefield 2008 Grindability Testing Results

	Number of samples				LEIT	Dwi	A	b	SPI	Crusher	Bond Rod	Bond Ball	Abrasion	SG	Bulk
	Total	Minas	Este	Main	KWh/mt	KWh/m3			min	index	KWh/mt	KWh/mt	index		Density
SGS Lakefield 2008	10	6	2	2					43.1	37.3		15.8			
SGS Lakefield 2008	2	1	0.5	0.5									0.398		
SGS Chile 2010	6	2	2	2	4.75	1.89	68.7	1.8	21.8	27.1	9.8	13.2	0.112	2.82	1.64

Earlier modeling studies were conducted by SGS Lakefield, the exercise being repeated by DJB Consultants Inc., in August 2011 as part of the feasibility study. This latter design exercise targeted a throughput of 22,500 tpd, and a grind size of 90 microns. This led to selection of a 28' diameter by 13' EGL SAG and 22' diameter by 36.5' EGL Ball mill. DJB concluded that with the soft ore (based on samples tested) at Corani no pebble crusher would be required. For more details on the grindability test work the reader is referred to the Blue Coast Metallurgy report (Blue Coast, 2011).

13.3 CORANI ORE VARIABILITY AND MINERALOGY

Two types of flotation testwork have been conducted through the program, variability testing using a standard flowsheet, and flowsheet fine-tuning work, culminating in locked cycle confirmation testing.

Sampling for the initial variability program was designed to, as much as possible, spatially cover the extent of the Corani deposit. While this exercise was done long before the feasibility pit was designed, by checking where each sample occurs versus the feasibility pit design, it is now possible to identify when each of the samples would be mined through the life of the operation. (If a sample contained material to be mined in two years, 0.5 "samples" were assigned to each of the years – hence fractions of samples occur in some cases.)

For example, more than 16 of the samples tested consist of material to be mined in Year 1, while roughly 10 samples are mined from material in year two. While biased slightly in favor of the first years of operation, the variability samples span the life of the mine quite well, so it can be concluded from a chronological (and therefore spatial) basis these variability samples span the deposit quite well.

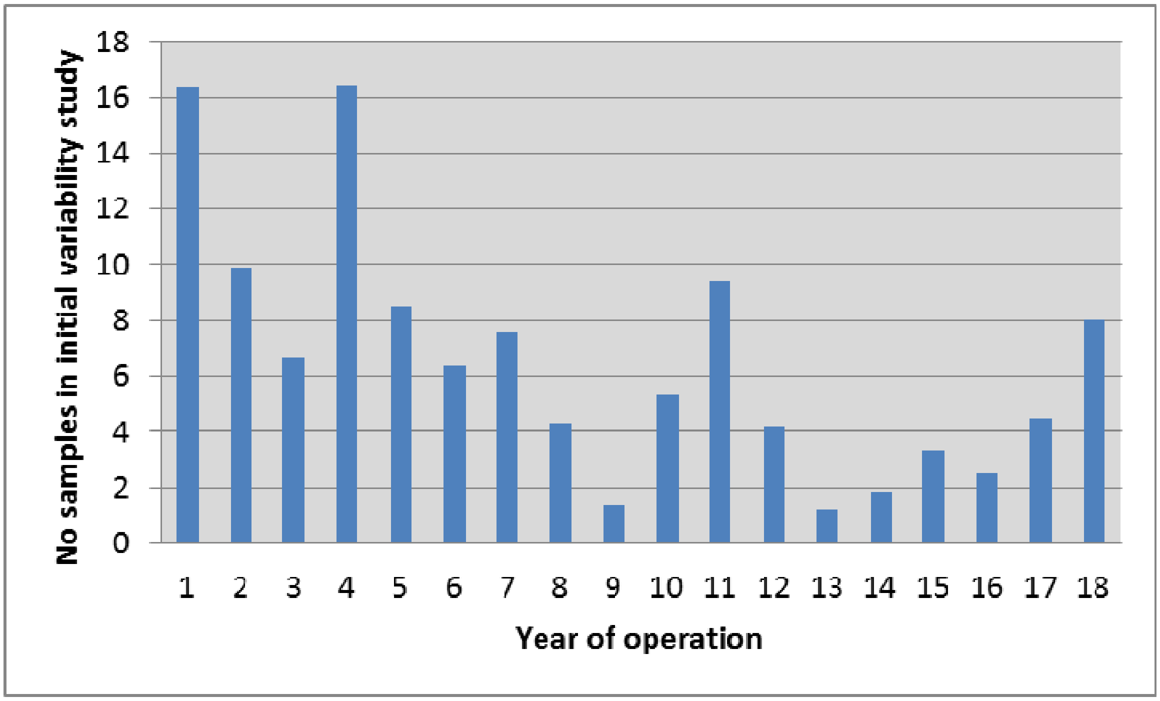


Figure 13-1: Distribution of Samples in Initial Variability Study by Expected Year Mined

Mineralogical analyses were conducted on the variability samples using QEMSCAN employing a carefully created custom "Corani" specimen identification protocol (SIP). These studies provided insight into the causes of flotation behavior of Corani ores, and enabled a classification system to be introduced, which has since been used to guide composite formation for flowsheet development.

Lead occurs as both galena and the altered lead aluminum phosphate mineral plumbogummite ($PbAl_3(PO_4)_2(OH)_5{\bullet}(H_2O)$), which is often found in fine textures with the barium aluminum phosphate end member, gorceixite. These phosphate minerals do not respond to flotation, hence only galena can be recovered, and the ratio of Pb in Galena to Plumbogummite is a driver of Pb recovery. Silver is present as tetrahedrite and various other silver sulfosalts (including myargyrite, pyrargyrite-proustite, boulangerite, acanthite) along with minor native silver. Zinc mineralization is mostly sphalerite.

Grouping the samples by the flotation response revealed several indicators for the degree of alteration of each sample, which corresponded to the flotation behavior. The best results were achieved on samples with the least alteration and hence more of the Pb in the form of galena. Samples with little or no flotation response were almost entirely altered with few primary sulfides remaining. Based on this work an ore classification system was developed describing the metallurgical behavior as follows:

- Type I: Good Pb/Zn selectivity by conventional flotation, high Pb, Zn, Ag recoveries.
- Type II: More challenging Pb/Zn selectivity, high Zn and Ag recoveries, and moderately high Pb recoveries.

- Type III: Pb/Ag ores with moderately high Ag recoveries, low Pb recoveries to low-grade Pb concentrates.

- Type IV: Poor or no flotation of all metals.

These classifications corresponded to the following averaged modal balances shown below, with distinct mineralogical trends in the transition from Type I to Type IV being evident.

Table 13-3: Averaged QEMSCAN Modal Balance for Ore Type Groupings

	I	II	III	IV
Tetrahedrite/Ag Minerals	0.02	0.01	0.03	0.00
Sphalerite	8.22	3.58	1.73	0.26
Galena	3.95	1.51	1.10	0.43
Other Pb-Minerals	0.04	0.03	0.08	0.01
Plumbogummite	0.11	0.03	0.27	0.28
Gorceixite	0.11	0.09	0.26	0.36
Plumbo/Gorce Mix	0.47	0.19	1.52	1.53
Chalcopyrite	0.08	0.16	0.11	0.01
Pyrite	11.19	5.93	5.94	1.15
Other Sulphides	0.16	0.14	0.22	0.05
Mn-Pb-Ba Oxide	0.00	0.00	0.00	0.95
Fe-Mn Oxide	0.00	0.00	0.08	1.59
Mn-Oxides	0.00	0.00	0.27	1.68
Ti Oxides	0.36	0.46	1.04	3.23
Fe-Oxides	0.06	1.09	12.15	7.86
Quartz	30.84	37.38	44.90	42.49
K-Feldspar	19.33	19.83	9.60	8.18
Plagioclase	0.71	0.45	0.56	1.43
Kaolinite	10.06	10.71	8.55	13.96
Chlorite	0.05	0.10	0.38	0.38
Amphibole	0.02	0.04	0.54	0.20
Biotite	0.36	0.48	0.31	1.14
Muscovites/Clays	13.10	16.24	7.84	8.54
Muscovite (Fe)	0.49	1.15	1.08	0.89
Barite	0.05	0.24	0.89	3.31
Carbonates	0.02	0.03	0.19	0.02
Apatite	0.02	0.01	0.08	0.02
Other	0.16	0.10	0.28	0.06

Key mineral abundance trends correlating with metallurgy (from Type I to IV):

- Reduced sulfides
- Higher ratio of plumbogummite to galena
- More Ba, Mn oxides
- More alteration from feldspar to kaolinite

Subsequent samples tested were all mineralogically benchmarked against this standard and, usually, an accurate prediction of the challenges and metallurgical performance could be made. Over time, as the difficulties in sequential Pb-Zn flotation in Type II ores were largely overcome, Types I and II became classified together as mixed sulfides processed by the same flow sheet, albeit recoveries being higher from ore with more Type I mineralogical characteristics.

Type III ores, being highly altered and for the most part containing insufficient zinc to warrant sequential flotation, were classified separately as transition material.

Type IV material, being fully oxidized and not responsive to flotation, was removed from the resource, would not be milled and is considered as waste in this feasibility study.

Galena grain size was also found to decrease with the degree of sample alteration, and hence the transition from Type I to Type III, which along with the increasing presence of plumbogummite are the biggest drivers of lead recovery. Similarly zinc recovery is driven mainly by grain size. These relationships are shown below:

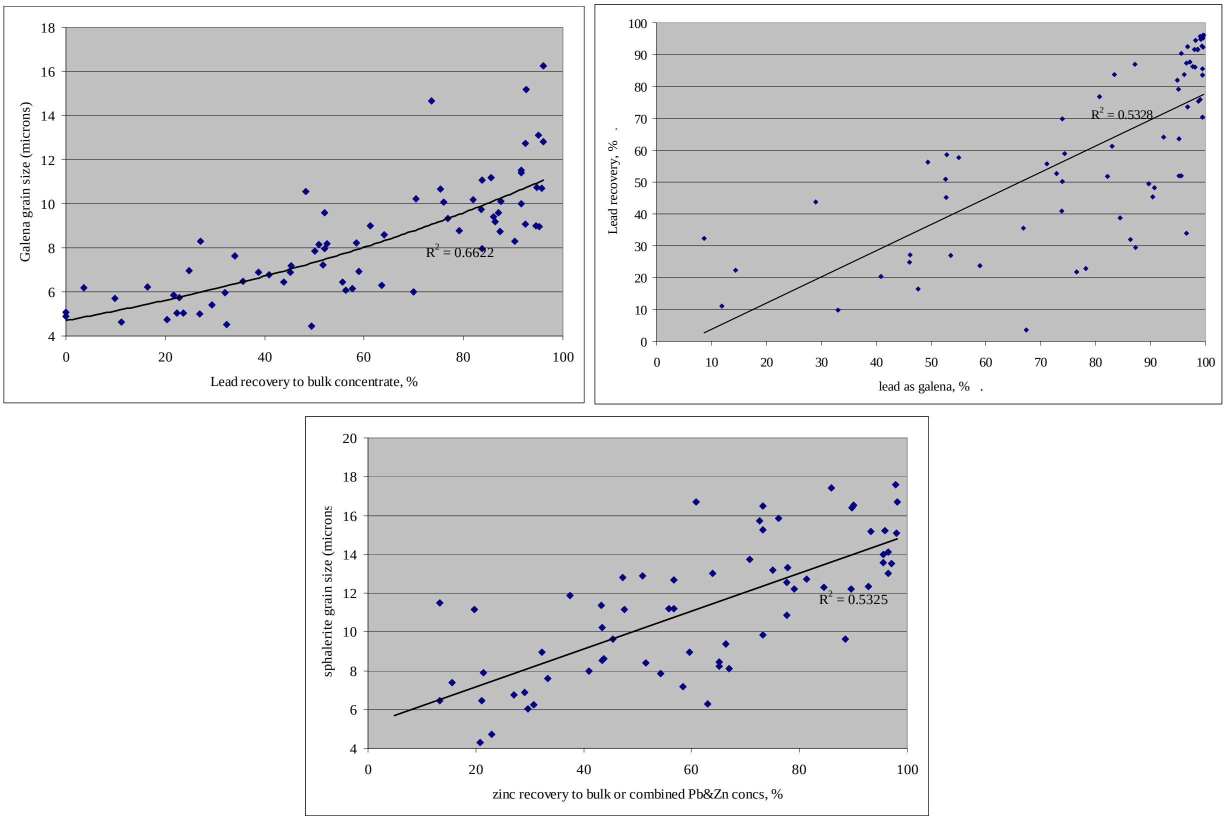


Figure 13-2: Mineralogical Drivers of Pb and Zn Flotation

Key data from the flowsheet development program is described in the next pages, first on depressant selection and primary grind ahead of lead/silver flotation, then lead rougher and cleaner flotation itself, both on Type I and II mixed sulfide ores and Type III transition ores. Development of the zinc circuit is also briefly described. Locked cycle test confirmation test data are described, from which projections of metallurgical performance are made.

13.4 DEPRESSANTS AND EDO

A key objective in flotation flowsheet development is been to repeatedly and reliably produce saleable lead and zinc concentrates through sequential flotation. Initially, lower doses of conventional zinc depressants such as zinc sulfate and sodium cyanide, were tested both alone and with sodium sulfide and SO_2. The chemical combinations tested on the B2 composite is shown below.

Table 13-4: Pb Circuit Reagent Test Conditions for B2 Composite (Type II Mixed Sulfide)

Test	Grind k80, µm	Na_2S g/t	$ZnSO_4$/NaCN g/t	ZnO/NaCN g/t	$ZnSO_4$ g/t	CMC g/t	AF242 g/t	A404 g/t	3418A g/t	Na_2SiO_3 g/t	MIBC g/t	pH
F9	102		200					50	15	50		nat
F1	102		100					50	15	50	15	nat
F4	102		200					50	15	50	10	8
F6	102	500	200					50	15	50	?	nat
F7	102			200				50	15	50	?	nat
F8	102			200				50	15	50	?	8.5
F10	102	1000	200					50	15	50	?	nat
F11	102	500			2000		10				?	nat
F13	85	500			2000		10				?	nat
F14	102	500			2000			50	15		?	nat
F15b	85	500			2000			50	15		?	nat
F15b	85	500			2000			10	5		15	nat
F16	85				2350		20				?	5 (SO2)
F17	85	500			2000		22.5				10	nat
F18	85	1000			2000		10				10	nat
F19	85	500			1000		10				10	nat
F20	85	500			2000		10				10	nat
F25	85	1000			2000	400	22.5			1000	25	nat
F27	85	500			2000		10				?	nat

For the most part, these tests yielded poor selectivity as shown below. A fine grind was used in these studies, and sliming of lead and silver sulfides could have contributed to the problem, although sphalerite activation was evident.

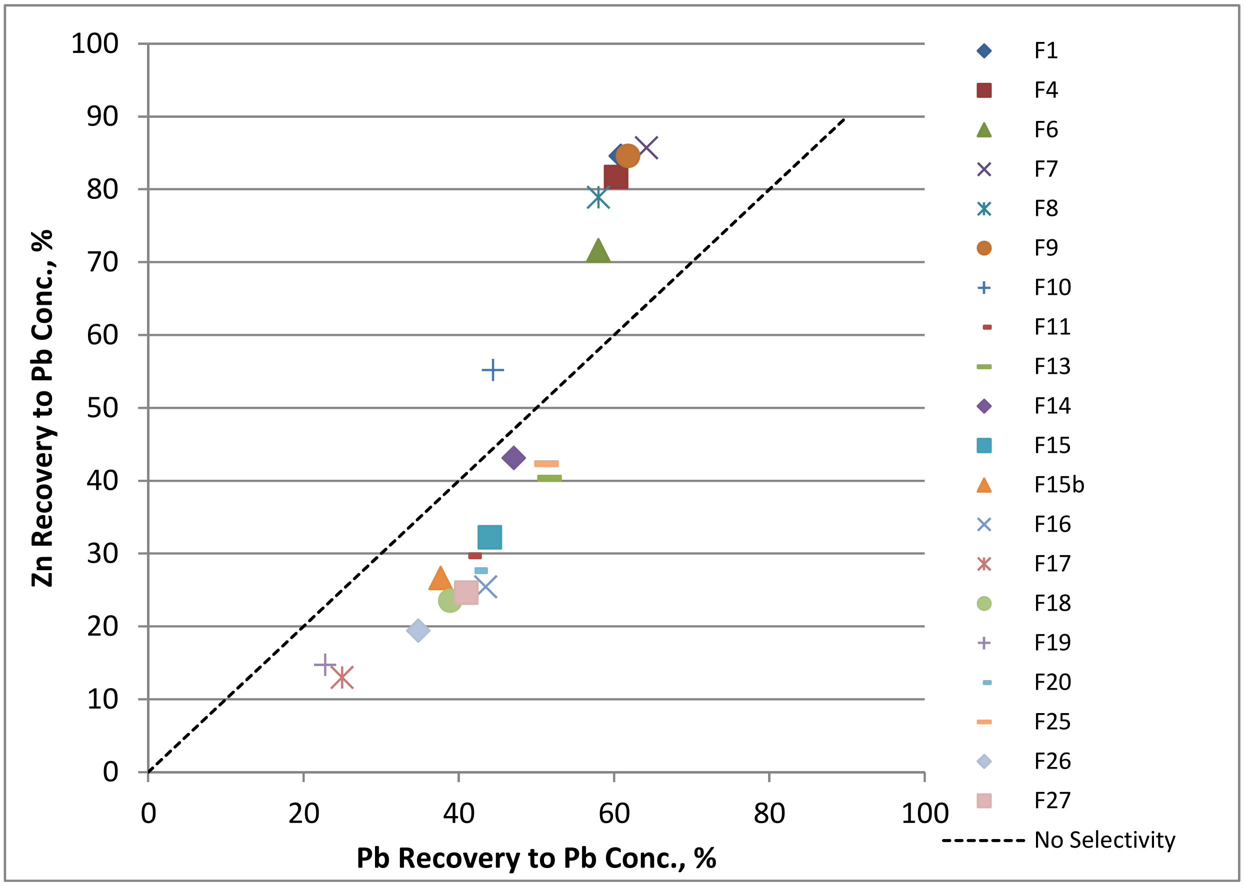


Figure 13-3: Pb-Zn Selectivity Testing – B2 Composite (Type II Mixed Sulfide)

Parallel studies employing solubility tests and surface analysis of floating sphalerite using Time-of-flight Secondary Ionization Mass Spectrometry (TOF-SIMS) concluded that lead ions were responsible for sphalerite activation during lead flotation of these more challenging Type II ores.

For such challenging materials, emulsified diesel oil (EDO) in conjunction with $ZnSO_4$/CN depressants offers a solution. EDO is believed to agglomerate fine weakly floatable lead and silver sulfides allowing for their selective flotation from sphalerite. The effect of EDO was investigated in a series of tests evaluating EDO in combination with depressants on B2 material:

Table 13-5: Test Conditions for Depressant and EDO Series – B2 Composite (Type II Mixed Sulfide)

	47	48	49	50	51	52	53	54
$ZnSO_4$ g/t					500		1000	
$ZnSO_4$/NaCN, g/t	500	500	500	500		500	0	500
Na_2SO_3, g/t	500	500	500	500	500	500	0	500
EDO, g/t	300	250	0	300	300	300	300	300
Regrind, minutes	0	0	0	5	0	10	0	20

The results of these tests are shown in the following plots, showing lead recovery vs. zinc misplacement in the Pb circuit, and vice versa in the zinc circuit. The effect of EDO, not present in test 49 (in yellow) is quite apparent while comparison of these data with those shown in the graph above illustrate the value of EDO in processing challenging Type II materials.

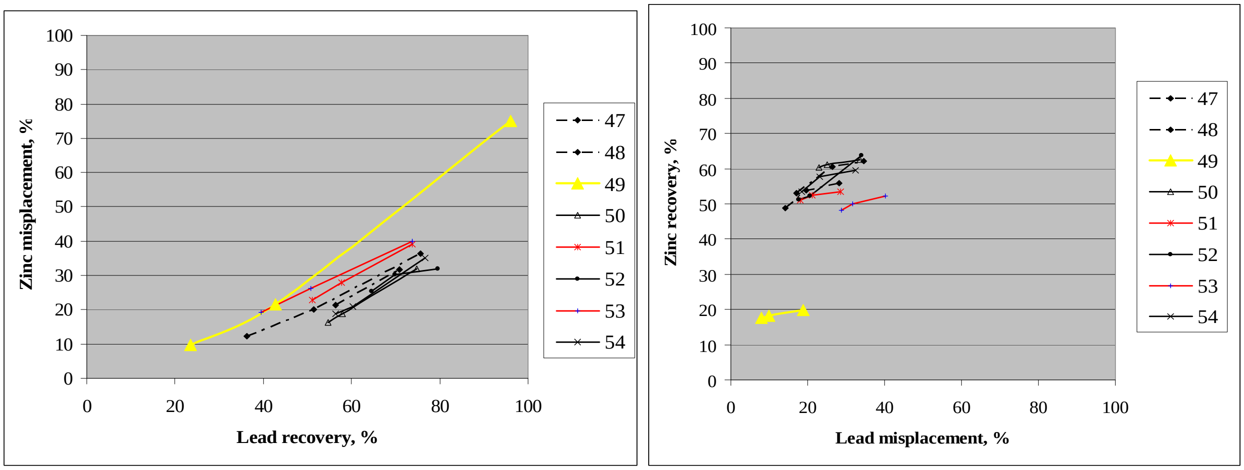


Figure 13-4: Effect of EDO on Selective Pb-Zn Flotation – B2 Composite (Mixed Sulfide)

EDO is not always necessary, indeed it seems likely that at the coarser grind selected for Corani, the role of EDO may be much reduced, as more recent studies at grinds in the 80-120 micron size range have shown far less of a need for the reagent. It is, however, a useful tool in the Corani flotation toolbox and its dosage has been allowed for in the feasibility plant design.

Aside from the above-described work on Composite B2, zinc and pyrite depressant selection was tested extensively on a variety of composites. Zinc sulfate, cyanide and sodium sulfite were the core depressants tested. Often zinc sulfate and cyanide were used together, premixed in a 3:1 ratio (a commonly used industry standard) typically at doses of roughly 500 g/t. Results from testing on D Composite, typical of a better behaving Type I mixed sulfide, are shown below. Out of the 5 tests, the best results were found with $ZnSO_4$ and CN added together in a pre-mixed 3:1 dosage ratio.

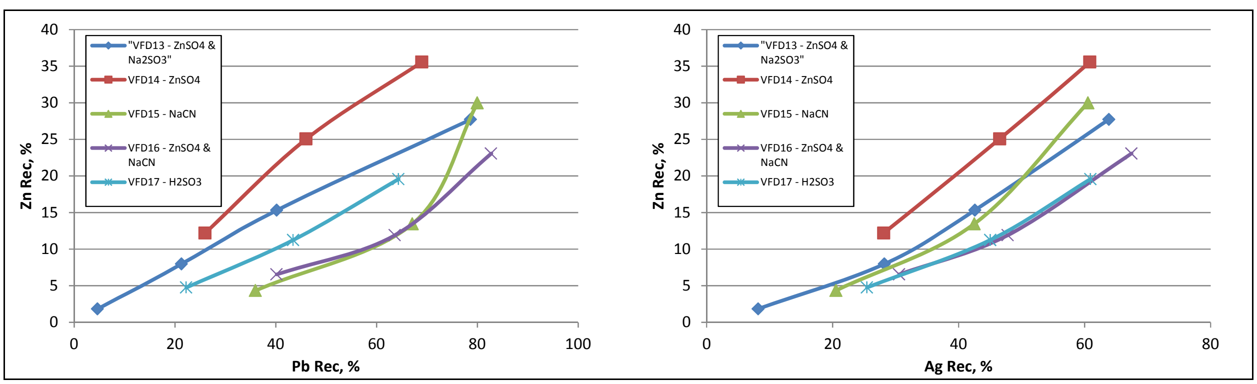


Figure 13-5: Depressant Selection Testing on D Composite

Testing on Composite G, a material characterized by mineralogy somewhere between Composites B2 and D also showed the zinc sulfate/cyanide reagent system to be the most effective, with sodium sulfite consistently being a valuable player in pyrite depression.

Accordingly, the results from Composites B2, G and D, spanning the whole mineralogical range of Corani Ag/Pb/Zn ores, all showed zinc sulfate/sodium cyanide and sodium sulfite to be the best depressant system. Accordingly more recent work focused on optimizing this dosage of $ZnSO_4$/CN and Na_2SO_3. Work for the pre-feasibility study established the optimum dosage at 500 g/t $ZnSO_4$/NaCN and 500 g/t Na_2SO_3.

13.5 GRIND SIZE

Early mineralogical (galena and sphalerite textural) studies pointed to the need for a relatively fine grind. The grain size distribution of the galena and sphalerite in many composites pointed to the need for grinds in the order of 50 microns, though some composites contained coarser lead and zinc sulfides, pointing to grind k80's in the range of 100 microns. Metallurgical results, however, contradicted the mineralogical information. Several test programs employed a variety of grinds on different composites, most of these being more challenging, finer grained Type II composites. The results are detailed in the Blue Coast report in the appendix and summarized below:

- 3 zone composite: 150um=100um>75um>45um, optimum at least 150 microns
- G composite: 76um>64um=46um, optimum at least 76 microns
- H composite: 73um=60um=50um, optimum at least 73 microns
- 3 zone Transition composite: 100um=75um>180um=50um, optimum 75-100 microns

In a meeting at BCM offices in Denver in March 2011, a 106 micron grind was decided as the design basis for the primary grind based on all the evidence assembled at that time. The objective of the most recent composite testing was pyrite depression rather than grind size determination. Although ~100 microns was targeted, the actual size came back at 90 microns, which was then adopted for all the tests for comparative testing purposes. This grind size is deemed to be slightly on the conservative (fine) side of optimal.

Ultimately there is no evidence of 106 micron grind being inferior to 90 microns. M3 used 106 microns to design the grinding circuit as, agreed upon earlier.

13.6 LEAD ROUGHING COLLECTORS

Collector testing focused on the use of Sodium Isopropyl Xanthate (SIPX) vs. Potassium Amyl Xanthate (PAX) as the primary collector along with various secondary collectors. SIPX with the Aero-404 a mercaptonbenzothiozole yields some of the best metallurgy (red below) and became the standard suite:

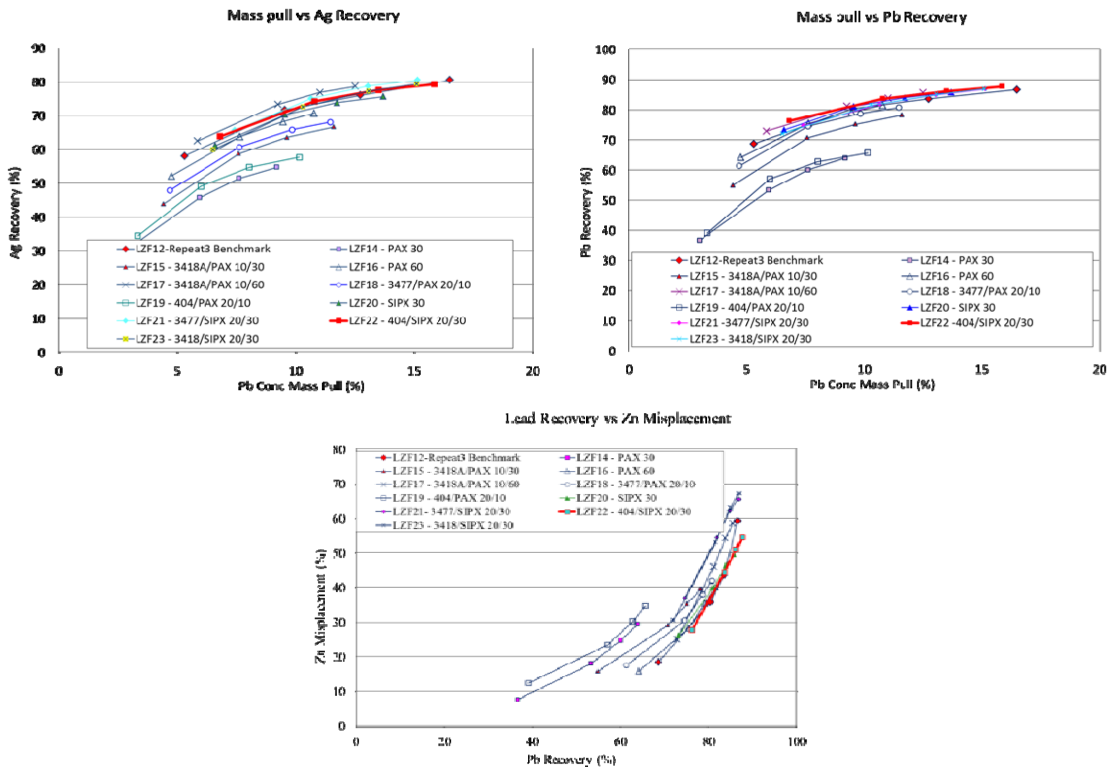


Figure 13-6: Collector Optimization Results in Pb Roughers – 3 Zone Pb-Zn-Ag Composite

13.7 GRINDING MEDIA AND SODA ASH

Only discovered in work conducted during 2011 on the challenging “1-5 Yr Master Composite”, grinding media selection appears to be a key factor at Corani. Results from an exhaustive test program targeting the rejection of pyrite are shown below. The details behind these tests are in Blue Coast (2011), but the key test is Test 19 (red) – the only one using inert ceramic media in place of the usual mild steel balls.

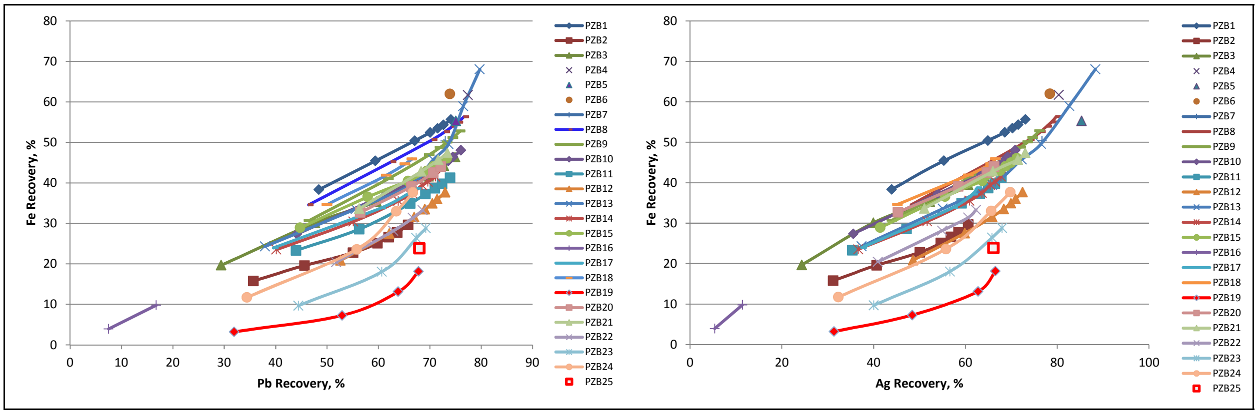


Figure 13-7: Pyrite Depression Testing – 1-5 Year Mixed Sulfide Composite

Fe recoveries to the Pb rougher concentrate dropped from 40-50% to ~20% without a significant drop in Pb and Ag recovery. While this mechanism is not completed understood, it is thought that galvanic interactions occur between mild steel and pyrite during grinding resulting in enhanced floatability of the pyrite. This process is mitigated by replacing the mild steel with inert grinding media and the pyrite becomes controllable. This effect is presently exploited in the treatment of complex Pb/Zn/Ag ores in Mexico. In reality, chrome grinding media are used as an approximation to fully inert media. These were tested on Corani ores, the benefit continuing to be observed.

The other departure from previous practice was the inclusion of soda ash as a pH modifier in the lead circuit. Soda ash is commonly used in Pb/Zn circuits in Canada, Russia and Australia and has been shown to aid zinc and pyrite rejection where lead activation of the sphalerite and pyrite is a problem. It is, however, an expensive reagent, relatively hard to obtain in Peru and accordingly would be used sparingly – we believe that the simpler Type 1 ores will not need soda ash, and an objective of future work would be to identify the economic optimum dosage of soda ash.

13.8 PB CONCENTRATE REGRINDING AND CLEANING

Mineralogical evidence points to the need for a moderately fine regrind for optimum liberation and hence concentrate grade. Accordingly, the requirement for rougher concentrate regrinding ahead of cleaning has been investigated, in two programs. Both programs led to similar findings - that a regrind size of about 25 microns appears to be optimal. The example of the study using the Type II mineralogy 3 zone composite is shown below. The mid-size regrind product k_{80} was roughly 25 microns.

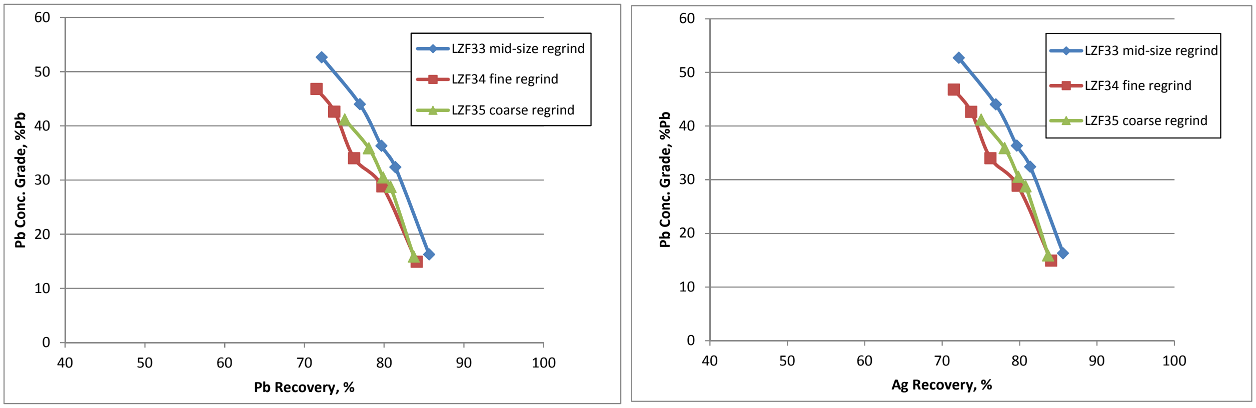


Figure 13-8: Effect of Pb Rougher Concentrate Regrind on Pb Cleaning – 3 Zone Pb-Zn-Ag Master Composite

The reagent dosing philosophy applied to the cleaners followed the pattern established in the rougher, with a small depressant (pre-mixed $ZnSO_4$/cyanide) addition to regrind and small collector dose in each stage of cleaning.

Achieving Pb concentrate grade proved straightforward on the "Type I" composites, where in excess of 60% Pb was routinely achieved and more difficult in the "Type II" B2 and G composites where the range was 35-40% Pb and ~50% Pb respectively. Ultimately, for all Pb/Zn ores tested in locked cycle mode Pb concentrate grades in the 55-60% Pb range were achieved and generally good recoveries projected for locked cycle testing (see later in this section).

13.9 LEAD FLOTATION IN TRANSITION ORE TESTING

In transition (type III) ores, the zinc content is usually too low to warrant a sequential float and hence just a silver-rich Pb concentrate is produced. As has been described previously, transition ores have a higher degree of alteration and lead recoveries are usually quite low, although silver recoveries typically remain reasonably good. A common challenge with these ores is the Pb concentrate grade, which in much of the batch testing was lower than 30% Pb (albeit with a high silver grade).

Two significant flow sheet development programs were conducted on transition ores. Using the relatively challenging H Composite, sulfidization was tested, without success, as well as collectors designed to float tarnished sulfides as well as oxides (hydroxamate and fatty acid collectors).

Testing on the 3 Zone Pb-Ag master composite was a bigger program, but again struggled to yield a saleable product. After extensive rougher and cleaner flotation testing, an approach of using starvation doses of a highly selective collector suite (A242 and SIPX) was adopted, followed by slow cleaning with deep froth depths in the batch cell – conditions similar to what might be found in a column cell.

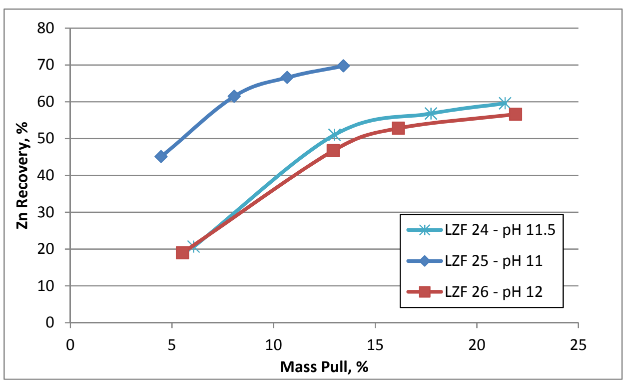
Although Ag recoveries were slightly lower, Pb concentrate grades in excess of 40-45% were achieved (along with ~3000 g/t Ag), which were considered a more marketable product. This flow sheet was carried successfully to locked cycle testing on several transition composites.

Transition ore samples have not to date been tested using inert grinding media, hence further gains in concentrate grade (or Ag recovery) may be possible based on all current knowledge. The mine plan calls for blending of mixed sulfide ore with transitional ore during operations and the transitional ore makes up 16% of the feed ore tonnes, so for the short periods of time the mill will treat higher proportions of transitional material (as high as 50% transitional material), a Pb concentrate assaying 45-50% will be produced albeit with high silver content. The effect of blending of Type I/II mixed sulfide and Type III transitional ores, likely the more common scenario, is discussed later.

13.10 ZINC FLOTATION

As is often the case with polymetallic ore testing programs, less focus has been placed on development of the zinc circuit at Corani. Corani sphalerite tends to float well and losses to the zinc tails tend to be low. The amount of zinc lost to Pb concentrate is only determined by locked cycle testing where sphalerite in middling streams in the Pb cleaner circuit finally report to either Pb concentrate or Pb cleaner tailing.

The majority of zinc circuit optimization work was conducted during the pre-feasibility study using the 3 Zone Pb-Zn-Ag master composite, with zinc rougher pH, collector type and dosage being tested. Results at pH 11, 11.5 and 12 using fixed $CuSO_4$ and PAX dosage is shown as follows.



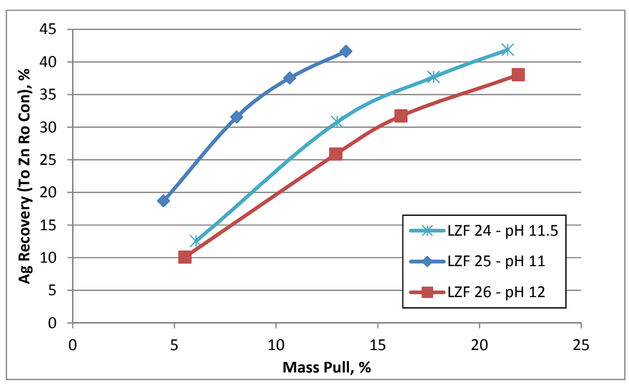


Figure 13-9: Sensitivity of Pb & Ag Flotation to pH in Zn Roughers - 3 Zone Pb-Zn-Ag Master Composite

The results suggest that pH 11 is optimal for both zinc and silver flotation, with a drop off in performance as pH is increased further. It is unlikely further benefit from a lower pH will be found as pyrite flotation would likely become more troublesome.

The collectors assessed were again SIPX and PAX. SIPX proved to be at least as effective as PAX (see below), so the fact that SIPX had also demonstrated clear benefit in the Pb circuit made it the natural choice as the zinc collector.

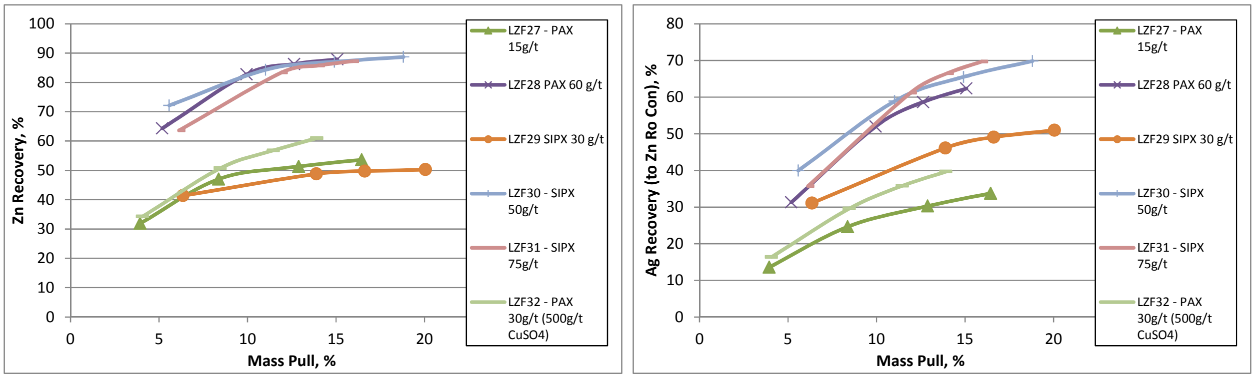


Figure 13-10: Zn Rougher Concentrate Sensitivity to Collector Type and Dosage – 3 Zone Pb-Zn-Ag Composite

Recent work on the 1-5 year master composite added further insight into optimal conditions for zinc flotation (see below). It should be noted that the first 2 tests, which essentially failed to float any zinc at all, both featured an experimental ZnO/CN depressant regime in the Pb circuit. It seems plausible that this could have played a role in preventing zinc flotation from taking place.

Good zinc flotation only occurred at $CuSO_4$ dosages higher than the typical industry standards of ~80-100 g/t per percent Zn. This is different from earlier results and suggests that the combination of the existing depressant suites and inert grinding media may now be over-depressing the sphalerite. This warrants further testing later. Test 44 split the zinc feed into two parts and tested with and without EDO, so the effect of EDO could be established. Both were successful tests with similar results, suggesting EDO is not a significant factor in zinc flotation.

	Grinding Cct		Pb Circuit	Zn Circuit				
Test	Grinding Media	Grind Size	Zn Depressant in Pb Cct	pH	CuSO4 g/t	SIPX g/t	EDO g/t (Pb Cct)	EDO g/t (Zn Cct)
38	Ceramic	76	ZnO/CN	11	160	20	0	0
39	Chromium Steel	70	ZnO/CN	11.5	300	30	100	0
40	Chromium Steel	81	ZnSO4/CN	11.5	300	30	100	0
41	Chromium Steel	80	ZnSO4/CN	11	160	20	0	50
42	Chromium Steel	80	ZnSO4/CN			Experimental		
43	Chromium Steel	80	ZnSO4/CN			Experimental		
44a	Chromium Steel	80	ZnSO4/CN	11	350	40	0	200
44b	Chromium Steel	80	ZnSO4/CN	11	350	40	0	0

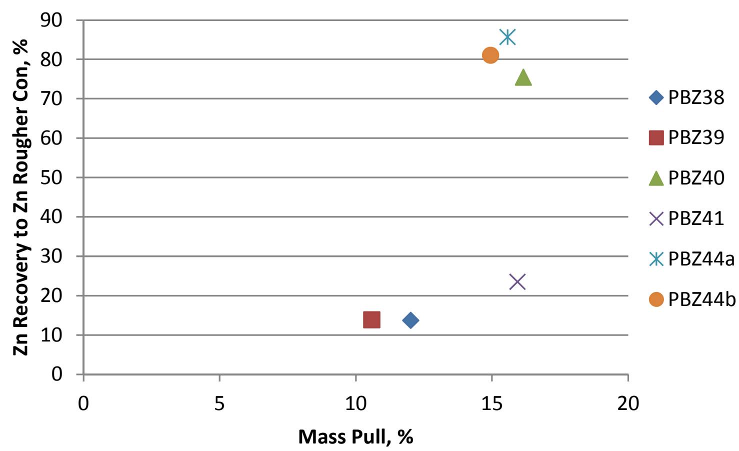


Figure 13-11: Results of Zn Optimization Test Work on 1-5 Yr. Pb-Zn-Ag Composite

13.11 MIXED SULFIDE AND TRANSITION ORE BLENDING

For the most part the current mine plan features a mill feed of either mixed sulfides or a blend of mixed sulfides and transitional ores. Some testing has been conducted on such blends to ensure transitional material does not “poison” the mixed sulfides. Composites L (Transitional) and M (Mixed Sulfide) were combined in a sequence of proportions (25:75%, 50:50%, 75:25%) for testing using the mixed sulfide flow sheet. These Results, showing both the Pb circuit performance (Ag recovery vs. Pb conc. grade and Pb recovery vs. Pb conc. grade) and the zinc circuit performance (Zn grade vs. recovery) are shown below. Combining transitional and mixed sulfides equals a “sum of the parts” result for recoveries, although little change in either Pb or Zn concentrate grade. This demonstrated flexibility for blending strategies without concern for unpredictable results. The blended L+M composite was also successfully tested in locked cycle mode.

A similar finding was made from blend testing in the 3 Zone master composite programs, where the mixed sulfide master composite was combined with the transitional master composite in a 5:1 ratio, which is approximately the same ratio of the two ore types in the Corani ore reserve. Once again, no adverse effect of blending was observed.

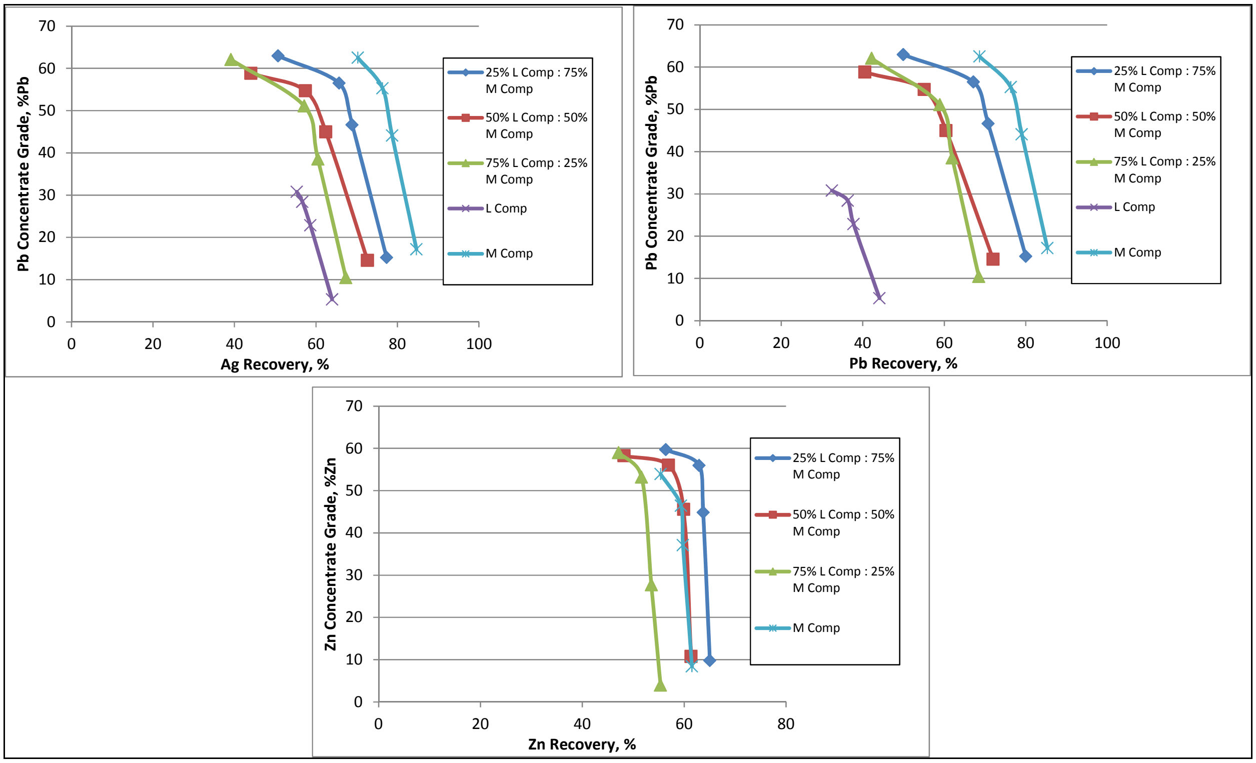


Figure 13-12: Effect of Blending Mixed Sulfide and Transitional Ore on Pb and Zn Circuit Performance – Composites L and M

13.12 LOCKED CYCLE CONFIRMATION TESTING AND METALLURGICAL PERFORMANCE PROJECTIONS

All but three years of the projected Corani mine life were represented by samples tested for the locked cycle test dataset, of which 42% of the dataset was from the first 5 years of operation. Some 13.8% of the locked cycle dataset was on samples located presently outside of the projected pit.

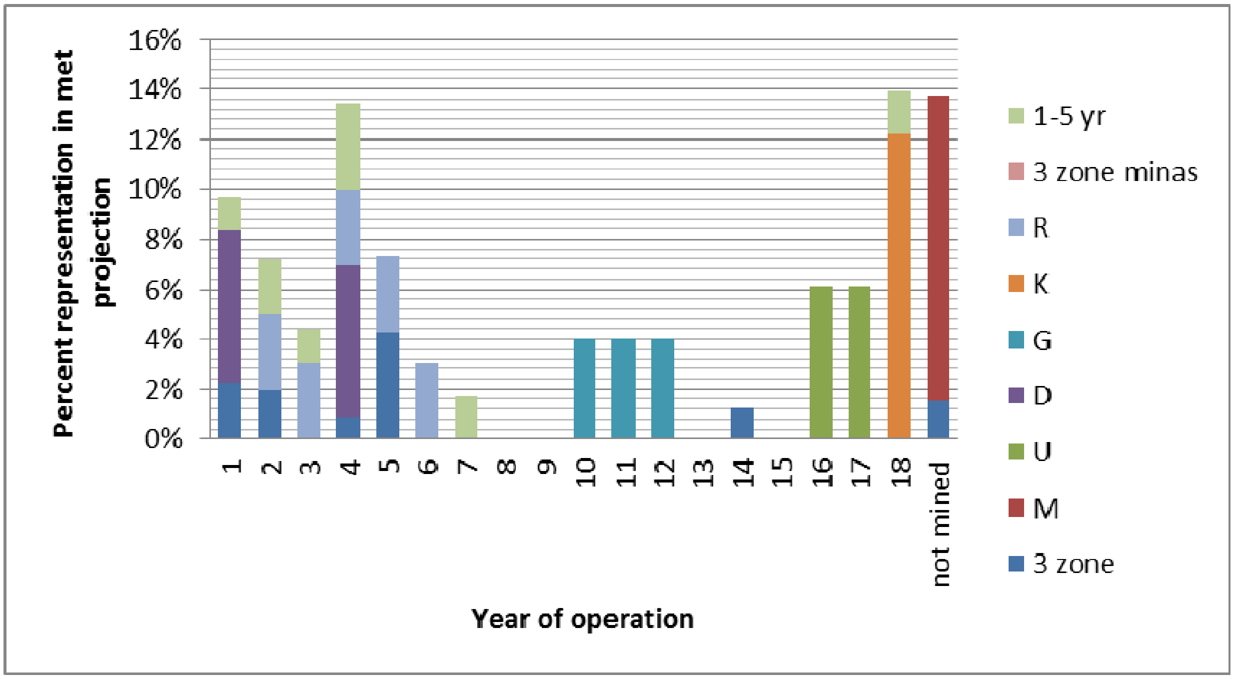


Figure 13-13: Representation through life of mine of Mixed Sulfide Composites Tested in Locked Cycle Mode

The various composites subjected to locked cycle testing along with the associated projected recoveries are presented below.

Table 13-6: Summary of Locked Cycle Test Results on Corani Composites and Overall Averages

Mixed Sulphide Composites	Head Assay				LCT Recovery				LCT Concentrate Grades			
	Pb	Zn	Fe	Ag	Pb	Zn	Ag(Pb Conc)	Ag(Zn Conc)	Pb Conc		Zn Con	
	%	%	%	g/t	%	%	%	%	Pb, %	Ag, g/t	Zn, %	Ag, g/t
3 Zone Mixed Sulphide Master Comp LCT 2	2.3	2.5	5.7	87	76.0	75.4	62.01	15.5	59.6	1686	53.1	390
Alphabet M Composite LCT1	2.2	1.4	4.2	69	80.3	69.4	79.7	7.2	53.5	1671	54.0	295
LCT 1 U Composite	0.8	1.4	3.5	36	87.7	78.4	73.3	13.1	54.9	2392	55.2	283
Alphabet D Composite LCT1	1.6	1.9	1.4	58	72.3	82.1	51.9	35.3	65.0	1680	49.3	651
Alphabet G Composite LCT2	1.2	1.1		62	55.8	68.5	55.8	15.2	49.7	1646	51.8	374
Alphabet K Composite LCT1	1.1	1.6		27	80.5	65.3	54.3	12.8	50.2	904	58.1	192
Alphabet R Composite LCT1	2.6	0.7		25	81.6	60.2	56.3	9.1	57.4	441	46.6	297
3 Zone Minas 3 Composite LCT1	5.1	1.9		154	75.3	71.7	69	12.2	79.5	2199	52.0	729
1-5 Yr Mixed Sulphide Composite LCT 2	0.9	1.3		50	53.6	64.4	40.2	19.5	51.2	2155	51.8	595
AVERAGE	2.0	1.5	3.7	63	73.7	70.6	60.3	15.5	57.9	1641	52.4	423
UNIT WEIGHTED					75.2	72.0	62.5	15.4	63.0	1830	52.7	524

Transitional Ore Composites	Head Assay				LCT Recovery				LCT Concentrate Grades			
	Pb	Zn	Fe	Ag	Pb	Zn	Ag		Pb Conc		Zn Con	
	%	%	%	g/t	%	%	%		Pb, %	Ag, g/t	Zn, %	Ag, g/t
Alphabet H Composite	0.9	0.09	1.6	51	42.9		53.7		35.8	3034		
Alphabet Q Composite	1.3	0.04	2.2	42	54.8		44.8		55.7	1625		
Alphabet T Composite	2.1	0.35	8.0	155	73.7		73.2		51.1	3708		
3 Zone Minas 1 Composite	3.5	0.08		99	79.3		58.7		72.8	1409		
3 Zone Transitional Master Composite	2.6	0.08	3.0	112	52.3		55.6		41.8	2404		
AVERAGE	2.1	0.13	3.7	92	60.6		57.2		51.4	2436		
UNIT WEIGHTED					65.3		61.0		57.9	2735		

Not including the blend composites, an effective weighted average Pb recovery of 75% is projected on mixed sulfide samples with an associated silver recovery of 62%. While the average silver head grade is quite close to the resource average, the lead head grade is significantly higher.

However Pb/Ag head grade/recovery relationships tend not to exist. For example the most Pb-poor sample yielded the higher Pb recovery. It is believed that the effect of head grade on recovery is overwhelmed by the effect of ore alteration, where correlations are most evident (see below).

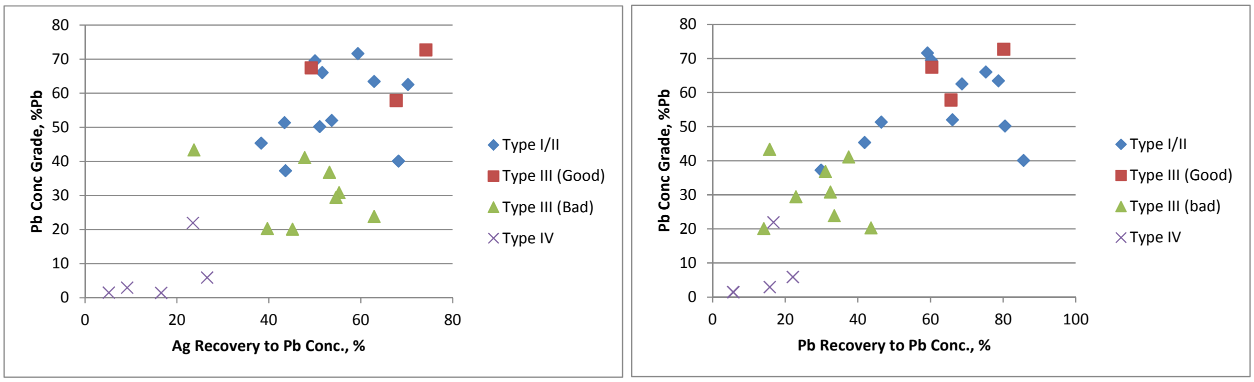


Figure 13-14: Batch Cleaner Test Performance for Alphabet and 3 Zone Variability Samples Grouped by Ore Classification

Accordingly, whether the locked cycle samples are representative of the deposit as a function of alteration is the more pertinent question – a question only a full geometallurgical study can answer, but care was taken, as much as possible, to ensure the samples represent the spectrum of alteration in the deposit. Alteration is not such a factor in zinc flotation, so head grade effects become evident here, and the lower grade zinc material must be taken into account. Based on data thus far it is judged that 0.3% Zn should be considered the cut-off for workable zinc flotation. Such low zinc grades would likely result in lower Zn (and Ag) recoveries than the LCT average, with recoveries then projected to increase at 0.5% Zn and 0.7% zinc. Further work is required on low grade zinc samples to firm up these projections.

Table 13-7: Projected Recoveries for Mixed Sulfide and Transitional Ore Types

Ore	Recovery (%) to Lead Con			Recovery (%) to Zinc Con		
Type	Pb	Zn	Ag	Pb	Zn	Ag
Mixed Sulphide						
Zn > 0.7%	75	9	62	5.0	67.5	14.0
0.5% <= Zinc <0.7%	75	9	62	5.0	50.0	10.4
0.3% <= Zinc <0.5%	75	9	62	5.0	30.0	6.3
Zinc < 0.3%	75	9	62	0	0	0
Transitional	38%+10.9*Lead Grade (%) - Max 65% recovery	5	38.5%+0.2*Ag Grade (g/t) - Max 70% recovery	0	0	0

13.13 FINAL FLOWSHEETS

A simplified schematic diagram of the final recommended lead and zinc flotation flow sheets is presented below.

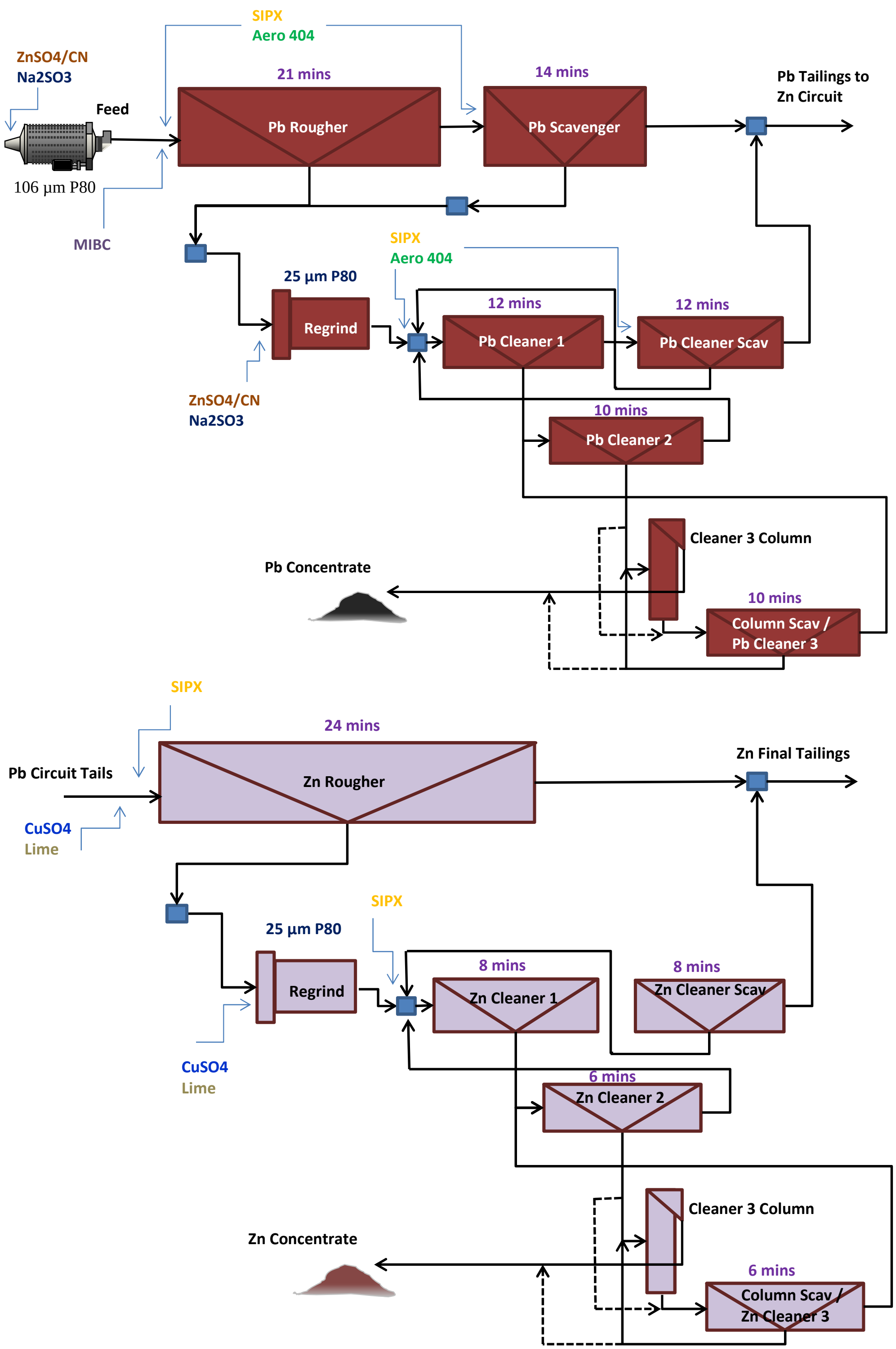


Figure 13-15: Simplified Schematic Flow sheets of Pb and Zn Circuits

It should be noted that the addition of columns in the final cleaning stage would likely benefit concentrate grades although the addition of a column scavenger, that can also perform the 3rd cleaner duty alone, would provide a good degree of operating flexibility. The stated residence times use a standard scale up multiplier of 3.

The concentrates produced in tests on the 3 Zone Master Composite and the 1-5 Year Composite were subjected to a full analytical suite to determine the presence of penalty elements, shown below.

Table 13-8: Pb and Zn Concentrate Multi-Element Scans

Corani Pb Concentrate Multi-Element Analysis

	3 Zone Mixed Sulphide Master Composite LCT1	3 Zone Mixed Sulphide Master Composite LCT2	1-5 Yr Mixed Sulphide Master Composite LCT2	AVERAGE		3 Zone Mixed Sulphide Master Composite LCT1	3 Zone Mixed Sulphide Master Composite LCT2	1-5 Yr Mixed Sulphide Master Composite LCT2	AVERAGE
Ag g/t	1465.8	1686.2	2100	1750.7	Mn g/t	600.0	480.0	278	452.7
Pb %	50.0	59.6	51	53.5	Cr g/t			287	287.0
Zn %	9.2	7.1	8.1	8.1	V g/t			< 4	<4
Cu %	1.2	1.3	1.6	1.4	Na g/t	58.0	59.0	72	63.0
Fe %	9.9	9.0	8.1	9.0	P g/t	< 200	< 200	266	266.0
Au g/t	0.06	0.52	0.37	0.32	As g/t	660.0	390.0	2750	1266.7
S %	22.6	18.0	21.3	20.6	Ba g/t	220.0	230.0	396	282.0
C(t) %	0.25	0.29	1.77	0.77	Co g/t			23	23.0
Cl g/t	15.0	< 10	< 10	15.0	Be g/t			1.74	1.7
F g/t	0.020	< 0.005	0.014	0.017	Li g/t			13	13.0
Hg g/t	24.4	19.7	16.8	20.3	Mo g/t			14	14.0
Bi g/t	< 50	62.0	74	68.0	Ni g/t			30	30.0
Cd g/t	770.0	670.0	1390	943.3	Sb g/t	5900.0	5700.0	12100	7900.0
SiO2 %	2.8	3.1	4.15	3.3	Se g/t	< 30	< 30	< 30	<30
Al2O3 %	0.37	0.42	0.61	0.47	Sn g/t	< 30	< 30	< 20	<30
Mg g/t	120.0	110.0	117	115.7	Sr g/t			64.3	64.3
Ca g/t	260.0	220.0	232	237.3	Tl g/t			98	98.0
K g/t	1100.0	1300.0	2330	1576.7	U g/t			< 20	<20
Ti g/t			228	228.0	Y g/t			2.2	2.2

Corani Zn Concentrate Multi-Element Analysis

	3 Zone Mixed Sulphide Master Composite LCT1	3 Zone Mixed Sulphide Master Composite LCT2	1-5 Yr Mixed Sulphide Master Composite LCT2	AVERAGE		3 Zone Mixed Sulphide Master Composite LCT1	3 Zone Mixed Sulphide Master Composite LCT2	1-5 Yr Mixed Sulphide Master Composite LCT2	AVERAGE
Ag g/t	380.3	389.8	463	411.0	Mn g/t	540.0	570.0	287	465.7
Pb %	4.4	3.7	4.8	4.3	Cr g/t			139	139.0
Zn %	50.8	53.1	52.3	52.1	V g/t			< 4	<4
Cu %	0.31	0.26	0.63	0.40	Na g/t	260.0	120.0	49	143.0
Fe %	4.2	5.0	5.2	4.8	P g/t	430.0	310.0	< 200	370.0
Au g/t	0.02	0.18	0.39	0.20	As g/t	320.0	250.0	722	430.7
S %	28.3	28.8	31.7	29.6	Ba g/t	550.0	380.0	179	369.7
C(t) %	0.2	0.2	0.16	0.2	Co g/t			16	16.0
Cl g/t	76.0	49.0	< 10	62.5	Be g/t			1.36	1.4
F g/t	0.027	0.014	0.008	0.016	Li g/t			6	6.0
Hg g/t	93.5	84.7	69.4	82.5	Mo g/t			< 10	<10
Bi g/t	< 50	< 20	< 20	<50	Ni g/t			< 20	<20
Cd g/t	2700.0	2600.0	4940	3413.3	Sb g/t	1400.0	1500.0	2140	1680.0
SiO2 %	7.20	8.10	2.66	5.99	Se g/t	< 30	< 30	< 30	<30
Al2O3 %	1.80	0.95	0.25	1.0	Sn g/t	< 30	< 30	57	<50
Mg g/t	390.0	220.0	71	227.0	Sr g/t			29.3	29.3
Ca g/t	2700.0	970.0	1090	1586.7	Tl g/t			109	109.0
K g/t	5300.0	3300.0	1240	3280.0	U g/t			< 20	<20
Ti g/t			108	108.0	Y g/t			1.4	1.4

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 will summarize the development of the model as well as the development of the mineral resource.

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 Table 14-1 below:

Table 14-1: Block Model Information

Corani Block Model, SIze and Location			
Block Size			15 x 15 meters
Bench Height			8 meters
Number of Blocks	East		184
	North		214
	Benches		64
Model Limits			
	East	314,745	317,505
	North	8,446,250	8,449,460
	Elevation	4,618	5,130

The model was assembled in the UTM coordinate system and is parallel to the UTM grid. Topographic information was assigned to the model based on topographic maps provided to IMC from Bear Creek. The topography maps were consistent with field observations and the elevations of the drill holes at their coordinates.

14.1.1 Rock Type Boundaries

IMC developed cross sections of the drill hole data in each of the deposits and confirmed the observations of previous authors and Bear Creek personnel regarding the distribution of mineralization within the Corani rock units.

Post-mineral volcanic tuffs are barren and appear to have been emplaced after the mineral events. Pre-mineral volcanic tuffs contain all of the ore of current economic interest at Corani. The mineral reserves and resources are completely contained within the pre-mineral tuffs.

The basement sediments are generally barren at Corani, 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. In addition, the block grade estimation was limited so that there was no extrapolation below the bottom of the drill holes.

IMC staff utilized the drill hole geologic logs provided by Bear Creek personnel to develop wire frame surfaces for: 1) top of sediments (bottom of pre-mineral), and 2) top of pre-mineral volcanics. These surfaces were extrapolated beyond the outside drill holes a sufficient distance so as not to limit the ability to assign rock type or grade when grade estimation was completed.

The wire frame interpretation was checked by plotting detailed cross sections through each of the three deposits. The rock type interpretation was then checked against the logged drill hole data by IMC staff and Bear Creek geology and engineering staff. Minor corrections were made and the check repeated.

The wire frame surfaces were used to assign a code to the block model on a nearest whole block basis. 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 there were mined as waste within the floating cones or mine plan.

Figure 7-3 illustrates the geologic assignment to the model blocks on illustrative cross sections.

14.1.2 Density Assignment

An in place density for each block has been assigned according to the rock type, deposit area, and the block silver grade. The specific densities used in the block model are shown in on Table 14-2. The default density assigned to fringe blocks that have no rock type assignment was 2.3643 t/m^3.

Table 14-2: Assigned Block Densities

ROCK TYPE	ZONE	AG GRADE	DENSITY t/m^3
Post Mineral Tuff	All	None	2.2714
Sediments	All	None	2.5214
Pre-Mineral Tuff	Main	Less than 30 g	2.3275
Pre-Mineral Tuff	Main	Greater than 30 g	2.4768
Pre-Mineral Tuff	Este	Less than 130 g	2.3205
Pre-Mineral Tuff	Este	Greater than 130 g	2.3926
Pre-Mineral Tuff	Minas	Less than 90 g	2.3468
Pre-Mineral Tuff	Minas	Greater than 90 g	2.5178

14.1.3 Block Grade Estimation

Block grades were estimated for silver, lead and zinc using ordinary linear kriging. Kriging was bounded by the rock type assignment and a computer developed grade contour for each metal in each deposit. The grade contour was used to segregate a high grade population from a lower grade population within each deposit. Grades were estimated above and below that population break value.

Prior to compositing, individual assay values were cut in order to limit the influence of high grade outliners on the block grade distribution.

Assay Cuts Prior to Compositing

Silver	Main,	+ 600 gm/t set to 600 gm/t	5 assays
	Minas	+ 600 gm/t set to 600 gm/t	16 assays
	Este	+1,000 gm/t set to 1,000 gm/t	13 assays
Lead	Main,	+ 7% set to 7%	6 assays
	Minas	+ 10% set to 10%	13 assays
	Este	+ 8% set to 8%	12 assays
Zinc	Main,	+ 6% set to 6%	7 assays
	Minas	+ 6.5% set to 6.5%	15 assays
	Este	+ 10% set to 10%	14 assays

The Corani drill holes were composited to nominal 8 m down hole or length composites respecting rock type. IMC utilized a technique that changed the composite length slightly within each rock type in order to have composites of equal length that respected the lithologic boundary.

The pre-mineral tuff is the unit of importance for potentially economic mineralization and can be used as the example of the composting process. Within each drill hole, the length of the intercept of the 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 if there was an improvement in ore selectivity with smaller composites versus the cost of production compares 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. The 8 m down hole composites were used as the basis for grade estimation within the pre-mineral tuff. The block grade estimation started with the determination of grade boundaries as described below.

Within each deposit, a cumulative frequency plot was generated for each of the metals. These were evaluated to determine breaks or boundaries within the mineral population. Detailed review of those graphs resulted in the use of the following population break points: Silver = 30 gm/t, Lead = 1%, Zinc = 1%.

These break points or discriminators were assigned to the model using a simple indicator process. For example, composites with silver value greater than 30 gm/t were assigned an indicator or 1.0. Composites with silver value less than 30 gm/t were assigned an indicator of 0.0. The 1's and 0's were used as input to a kriging run that results in fractions assigned to each block between 0.0 and 1.0. Blocks were contoured at the 50% level to establish whole blocks that had better than a 50:50 chance of being above and below a 30 gm/t discriminator value.

The above process was repeated for each of silver, lead, and zinc at the discriminator values listed above. Table 14-3 summarizes the kriging parameters used to establish the indicator codes within the model.

Table 14-3: Kriging Parameters For Indicator Grade Breaks

Area, Metal	Major Axis		Sub-Major	Range			Search			Spherical Variogram	
Discriminator	Bearing	Plunge	Dip	Major	Inter.	Minor	Major	Inter.	Minor	Nugget	Total Sill
Silver, 30 gm/t Discriminator											
Main	330	30 Dn	60 SW	260	120	40	175	80	25	0.10	1.00
Minas	330	15 Dn	60 SW	250	130	40	165	85	25	0.10	1.00
Este	330	15 Dn	15 SW	270	270	40	180	180	25	0.10	1.00
Lead, 1% Discriminator											
Main	330	30 Dn	60 SW	260	120	40	175	80	25	0.10	1.00
Minas	330	15 Dn	60 SW	180	120	40	120	80	25	0.10	1.00
Este	330	15 Dn	15 SW	180	180	40	120	120	25	0.10	1.00
ZInc, 1% Discriminator											
Main	330	30 Dn	40 SW	260	120	40	175	80	25	0.10	1.00
Minas	330	15 Dn	45 SW	150	112	45	100	75	30	0.10	1.00
Este	330	15 Dn	15 SW	180	180	40	120	120	25	0.10	1.00

Composite Count	Max	Mininum	Max/Hole
	10	3	3

tab14-2.xls

Once the blocks were coded as being above or below the discriminator value, grades were assigned using ordinary linear kriging respecting the indicator assigned grade contour as a hard boundary. The pre-mineral rock type was also respected as a hard boundary, meaning that composites from outside the pre-mineral rock type were not used to estimate the pre-mineral volcanics.

The composites were assigned a code to indicate if the composite were contained within a high grade or low grade block. This composite code was assigned based on the indicator code of the block that contained the composite. The composite codes were matched to the block indicator codes as hard boundaries during the grade estimation process.

Table 14-3 summarizes the kriging parameters used to estimate the high grade and low grade component of each metal for within the pre-mineral volcanics.

Figure 14-1 through Figure 14-3 present example silver grade variograms from the high grade indicator zone of each deposit. These are presented as background support for the selection of grade estimation parameters presented on Table 14-4.

Table 14-4: Kriging Parameters for Block Grade Estimation

Area, Metal	Major Axis		Sub-Major	Range			Search			Spherical Variogram	
Discriminator	Bearing	Plunge	Dip	Major	Inter.	Minor	Major	Inter.	Minor	Nugget	Total Sill
Silver Grade for + 30 gm/t Indicator Blocks in Pre-Mineral Volcanics											
Main	330	30 Dn	60 SW	130	100	50	130	100	50	0.10	1.00
Minas	330	30 Dn	60 SW	115	75	50	115	75	50	0.10	1.00
Este	330	15 Dn	30 SW	130	145	50	130	145	50	0.10	1.00
Silver Grade for less than 30 gm/t Indicator Blocks in Pre-Mineral Volcanics											
Main	330	30 Dn	60 SW	130	100	25	130	100	25	0.10	1.00
Minas	330	30 Dn	90 SW	100	100	25	100	100	25	0.10	1.00
Este	330	15 Dn	60 SW	130	130	25	130	130	25	0.10	1.00
Lead Grade for + 1% Indicator Blocks in Pre-Mineral Volcanics											
Main	330	30 Dn	60 SW	100	75	50	100	75	50	0.10	1.00
Minas	330	30 Dn	60 SW	100	75	50	100	75	50	0.10	1.00
Este	330	15 Dn	60 SW	75	75	25	75	75	25	0.10	1.00
Lead Grade for less than 1% Indicator Blocks in Pre-Mineral Volcanics											
Main	330	30 Dn	60 SW	130	100	50	130	100	50	0.10	1.00
Minas	330	30 Dn	90 SW	100	75	75	100	75	75	0.10	1.00
Este	330	15 Dn	15 SW	130	130	50	130	130	50	0.10	1.00
Zinc Grade for + 1% Indicator Blocks in Pre-Mineral Volcanics											
Main	330	30 Dn	40 SW	100	75	50	100	75	50	0.10	1.00
Minas	330	15 Dn	45 SW	100	75	30	100	75	30	0.10	1.00
Este	330	15 Dn	15 SW	75	75	25	75	75	25	0.10	1.00
Zinc Grade for less than 1% Indicator Blocks in Pre-Mineral Volcanics											
Main	330	30 Dn	40 SW	100	100	50	100	100	50	0.10	1.00
Minas	330	30 Dn	90 SW	100	75	75	100	75	75	0.10	1.00
Este	330	15 Dn	30 SW	130	130	25	130	130	25	0.10	1.00

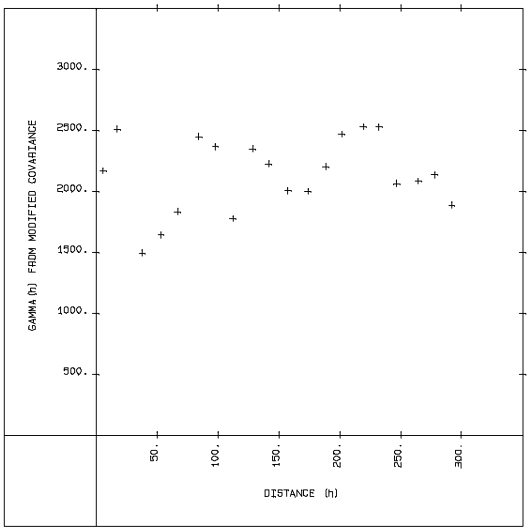
Composite Count	Max	Mininum	Max/Hole
	10	1	3

tab14-3.xls

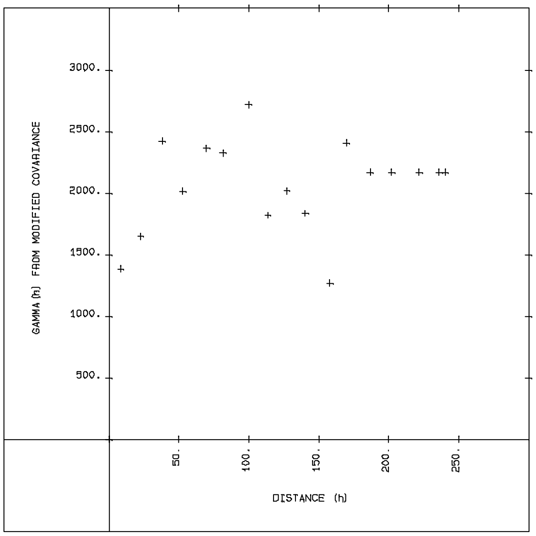
Silver Variograms in High Grade

Indicator Zone, + 30 gm/t

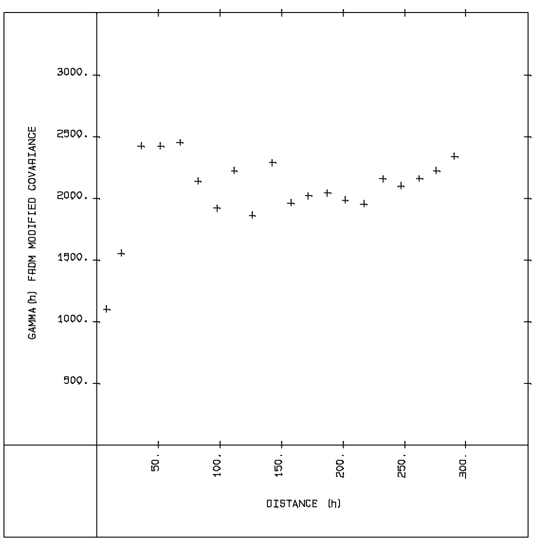
Gamma (h) from Modified Covariance



330 Bearing, 30 Deg Plunge



240 Bearing, 60 Deg Plunge



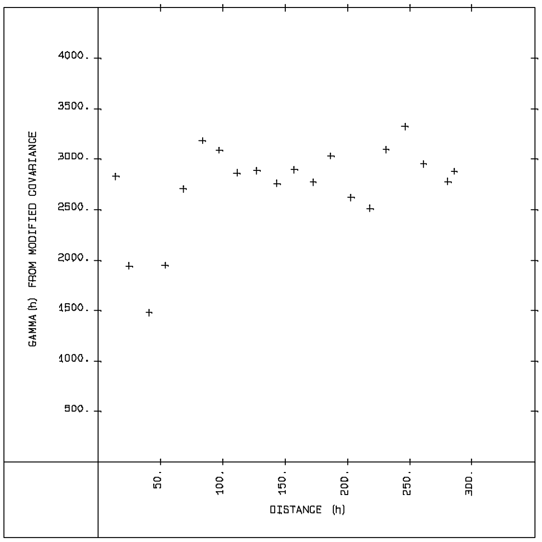
60 Bearing, 30 Deg Plunge

Figure 14-1: Corani Main

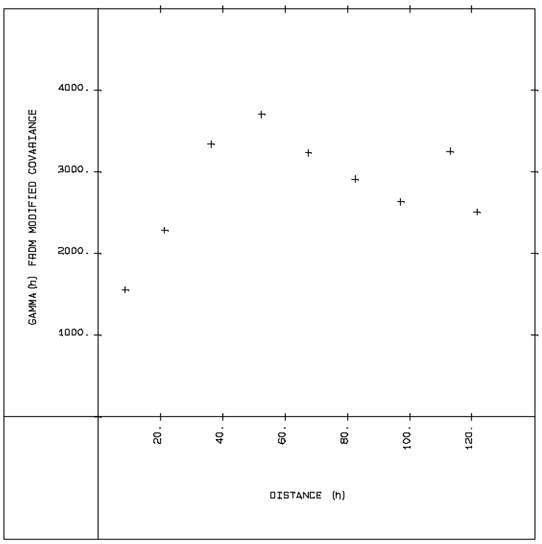
Silver Variograms in High Grade

Indicator Zone, +30gm/t

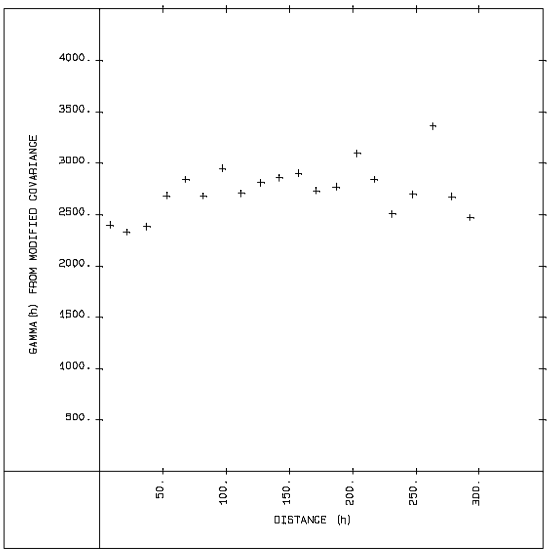
Gamma(h) from Modified Covariance



330 Bearing, 30 Deg Plunge



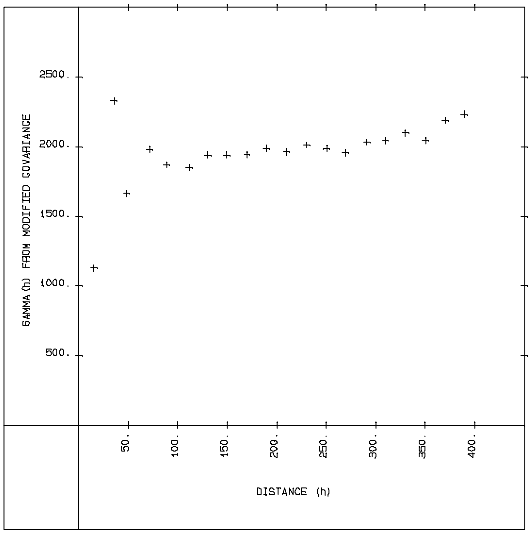
240 Bearing, 60 Deg Plunge



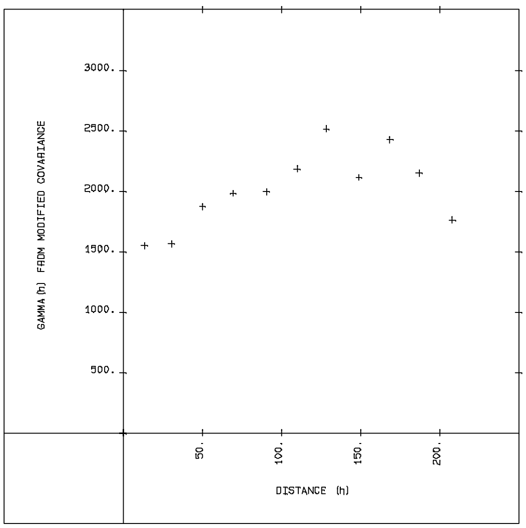
60 Bearing, 30 Deg Plunge

Figure 14-2: Minas Corani

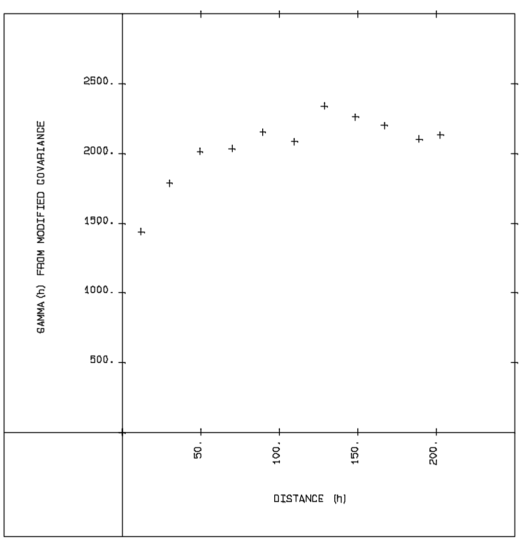
Silver Variograms in High Grade
Indicator Zone, +30 gm/t
Gamma (h) from Modified Covariance



330 Bearing, 15 Deg Plunge



240 Bearing, 30 Deg Plunge



60 Bearing, 60 Deg Plunge

Figure 14-3: Corani Este

14.1.4 Classification

Blocks were coded as measured, indicated, or inferred based on the silver grade estimate. The kriged standard deviation (square root of kriged variance) and the number of composites used to estimate a block silver grade were stored in the model.

The classification procedures were as follows based on silver grade kriging:

- **Measured:** Silver kriged standard deviation < = 0.40 and 10 composites were used
- **Indicated:** Silver kriged standard deviation < = 0.975
- **Inferred:** Otherwise if the silver grade was estimated

14.1.5 Mineral Codes

Mineralization codes were assigned to the block model based on interpretations developed by Bear Creek staff. The mineral zones reflect types of ore mineralogy that correlate with and categorize the metallurgical responses of the deposit. The mineral codes and an illustration of their geometry on the final pit wall were presented in Section 7.

Process testing was completed on a wide range of ore types within all three deposit areas. These process tests were categorized by response and underlying mineral control. The Bear Creek geology and engineering staff scanned the geologic logging data to identify those mineral occurrences within the core drilling and used that information to assemble the interpreted geometries.

The mineral zone interpretations were provided to IMC as wire frame solids. IMC edited those solids to conform to the lithologic interpretation prior to assignment to the block model on a nearest whole block basis. The list of mineral codes is presented below.

- Fine Black Sulfide
- Pyrite Marcasite
- Plumbogummite
- Quartz / Sulfide / Barite
- Iron Oxide (4) (Poor flotation response)
- Iron Oxide (3) Transitional flotation response
- Coarse Sulfide and Celadonite
- Manganese Oxide.

14.2 MINERAL RESOURCES

The mineral resources were developed with the floating cone algorithm to determine the component of the deposit with reasonable prospects of economic extraction. For the resource cone, economic benefit was applied to inferred mineralization.

However, no economic benefit was applied to inferred in the later determination of mineral reserves or in the economic analysis of the project.

The floating cone computer algorithm is a tool for guidance to mine design. The algorithm applies approximate costs and recoveries along with approximate pit slope angles to establish theoretical economic breakeven pit wall locations.

The floating cone 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 Floating Cone

1) The resource floating cone did receive economic credit for inferred class material. Inferred was treated as waste for the mineral reserve.
2) Metal prices for the mineral resource were: $30/oz Silver, $1.00/lb Lead, and $1.00/lb Zinc.
3) Oxide material (mineral zone FeO_4) was assumed to be processed by heap leaching within the mineral resource. It was treated as waste within the mineral reserve. The heap leach process recovery for FeO_4 was 85% and the process cost was $9.25/tonne of heap leach material.

The Corani mineral resource was contained in the floating cone and is summarized on Table 14-5. The resource statement on Table 14-5 includes the mineral reserves that are presented in Section 15.

The qualified person for the estimation of the mineral resource was John Marek of Independent Mining Consultants, Inc. 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 socio-economic conditions that would put the Corani mineral resource at a higher level of risk than any other Peruvian developing resource.

Table 14-5: Total Mineral Resources (Includes Both Resources and Reserves)

Total Mineral Resource, Includes the Mineral Reserve							
Category	Ktonnes	Silver Gm/t	Lead %	Zinc %	Silver Million Ozs	Lead Million Lbs	Zinc Million Lbs
Measured	40,961	53.57	0.866	0.531	70.5	782.0	479.5
Indicated	249,630	35.89	0.629	0.377	288.1	3,461.6	2,074.8
Measured + Indicated	290,591	38.38	0.66	0.40	358.6	4,243.6	2,554.3
Inferred	49,793	30.00	0.464	0.278	48.0	509.4	305.2

Notes:

Cutoff Grade: $1.20 / tonne, Net of Process = NSR - Process Cost which is equivalent to an NSR cutoff grade of $9.20/tonne of ore.

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 grade greater than the breakeven value. The final pit design and the internal phase designs were guided by the results of the floating cone algorithm.

15.1 FLOATING CONES

The floating cone computer algorithm is a tool for guidance to mine design. The algorithm applies approximate costs and recoveries along with approximate pit slope angles to establish theoretical economic breakeven pit wall locations.

Economic input applied to the cone algorithm is necessarily preliminary as it is one of the first steps in the development of the mine plan. However, the cone geometries should be considered as approximate as they do not assure access or working room. The important result of the cones is the relative changes in geometry between cones of increasing metal prices. Lower metal prices result in smaller pits which provide guidance to the design of the initial phase designs. The change in pit geometry as metal prices are increased indicates the best directions for the succeeding phase expansions to the ultimate pit.

Table 15-1 and Table 15-2 summarize the economic input applied to the floating cones that were used for pushback design guidance at Corani. The base case design economics for the final pit utilized metal prices of $18.00/oz silver, $0.85/lb Lead, and $0.85/lb Zinc. The resulting floating cone used as guidance to final pit design is shown on Figure 15-1.

Figure 15-2 presents the surface outlines of 7 nested floating cones with silver prices between $6.00/oz and $18.00/oz. The concentric geometries in each mine area are indicative of the best starting point in each pit and the expansion direction that will result in the best extraction sequence. The final $18.00/oz cone was the guide for design of the final pit.

Table 15-1: Recoveries, Concentrate Grades, and Process Cost by Mineral Type

Mineral Code	Recovery (%) to Lead Con			Recovery (%) to Zinc Con			Pb Con	Zn Con	Leach	Process
	Pb	Zn	Ag	Pb	Zn	Ag	Pb Grade (%)	Zn Grade (%)	Ag Rec(%)	Cost ($)
CSC	75	9	62	5	variable	variable	60.00	53.00		8.00
FBS	75	9	62	5	variable	variable	60.00	53.00		8.00
QSB	75	9	62	5	variable	variable	60.00	53.00		8.00
PM	75	9	62	5	variable	variable	60.00	53.00		8.00
Pg	variable	5	variable	0	0	0	40.00	N/A		8.00
FeO(3)	variable	5	variable	0	0	0	40.00	N/A		8.00
FeO(4)									85.00	9.25
MnO										

Variable Recovervies

	Recovery (%) to Lead Con	
	Pg, FeO(3)	
Pb	38%+10.9*Lead Grade (%)	Max 65%
Ag	38.5%+.2*Ag Grade (g/t)	Max 70%

	Recovery (%) to Lead Con
	CSC/FBS/QSB/PM
Pb	75%
Ag	62%

	Recovery (%) to Zinc Con	Recovery (%) to Zinc Con	Recovery (%) to Zinc Con
	CSC/FBS/QSB/PM	CSC/FBS/QSB/PM	CSC/FBS/QSB/PM
	Zinc >.7	.5<=Zinc <.7	.3<=Zinc < .5
Zn	67.5%	50.0%	30.0%
Ag	14.0%	10.4%	6.3%

Mineral Code Description

- CSC = Coarse Sulfide and Celadonite
- FBS = Fine Black Sulfide
- QSB = Quartz, Sulfide, Barite
- PM = Pyrite - Marcasite
- Pg = Plumbogummite
- FeO(3) = Iron Oxide (3) Transitional Flotation Reponse
- FeO(4) = Iron Oxide (4), Poor Flotation Repsonse
- MnO = Manganese Oxide, Minor Constituent

FeO(4) was considered as potential leach ore

Notes for 2011

FeO(4) was not considered in the Floating cones for the 2011 Feasibility Reserves.

Table 15-2: Floating Cone Input Information Including Smelter Terms for Floating Cone Input

Mining Cost / Tonne Material	$1.34	/tonne total material
General and Administrative Costs	$1.20	/tonne ore
Process Costs	Table 15-1	
Slope Angles:	42	degrees, pre-mineral tuff
	46	degrees, post mineral tuff
Bench Discounting:	1%	per bench

Base Case Metal Price	
Silver= Variable in $/oz, Base Case for Final Pit = $18.00/oz Lead = 4.72% of the price of silver in $/lb Zinc = 4.72% of the price of silver in $/lb Multiple Cones were run at Silver Prices Ranging from $8.00/oz to $20.00/oz	
Lead Concentrate	**Zinc Concentrate**
Payable Metal	
Lead:	Zinc:
Pay the lesser of: 95% contained metal or 3% deduct from concentrate grade (equivalent at 60% lead) Silver:	Pay the Lesser of: 85% contained metal or 8% deduct from concentrate grade (equivalent at 53.33% zinc) Silver:
Pay the lesser of: 95% contained metal or 50 g/tonne deducted from concentrate grade (equivalent at 1000 g/tonne)	Pay 65% of (contained metal - 108.9g/tonne)
Treatment Charge	
$155.00/tonne + 15% of (lead price($/tonne) - $850/tonne) if price greater than $850/tonne	For zinc price less than $1000/tonne: $190/tonne - 14% of ($1000/tonne-price) For zinc price greater than $1000/tonne and less than $1100/tonne $190/tonne plus 16% of (price-$1000/tonne) For zinc price greater than $1100/tonne $190/tonne+$16/tonne+17% of (price-$1100/tonne)
Penalties	
Subtract $1.50/tonne from smelter payment for each one percentage of zinc above 8% in the concentrate.	None
Transportation	
Land: $50.00/tonne Ocean: $75.00/tonne	Land: $50.00/tonne Ocean: $75.00/tonne

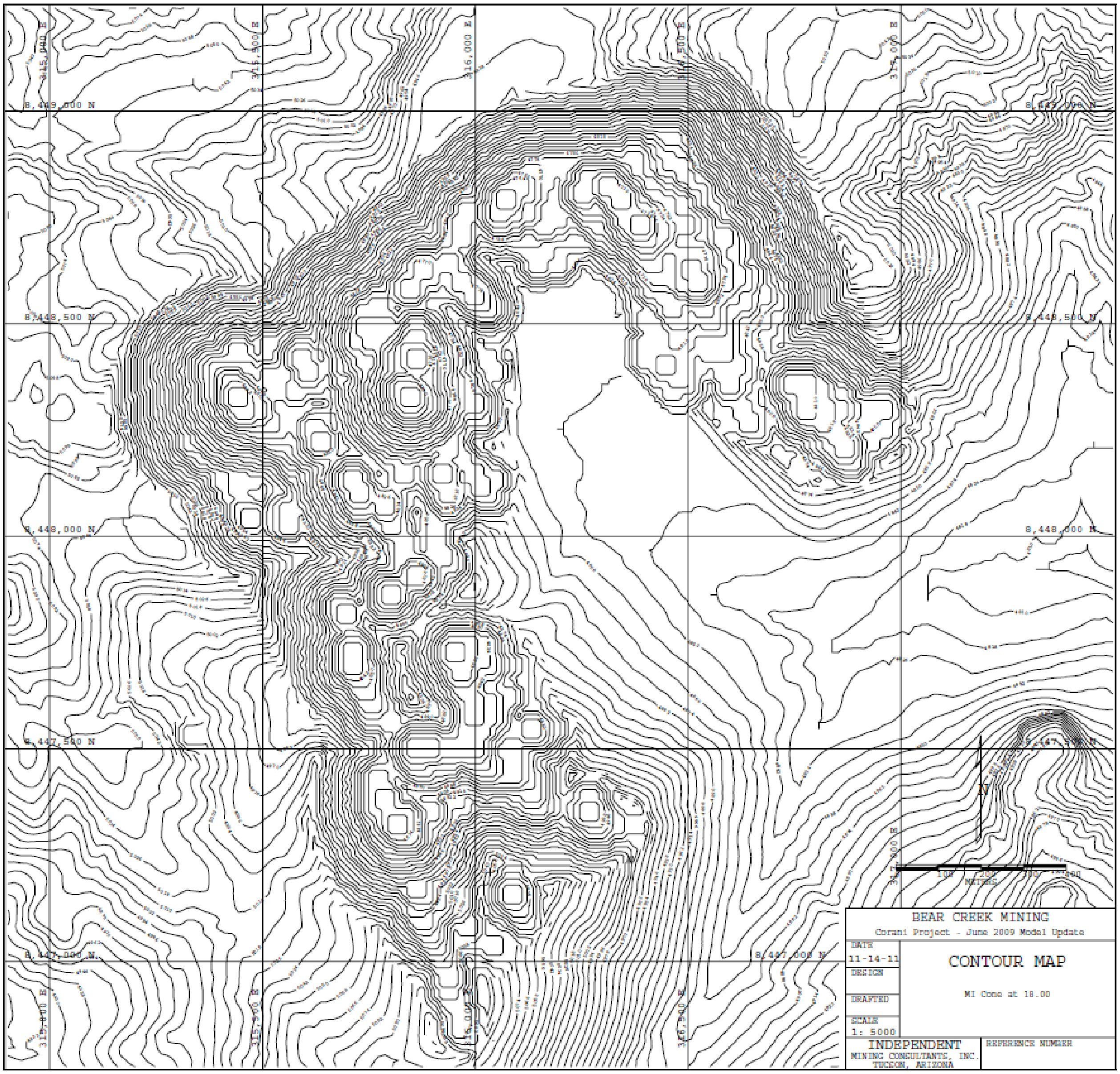


Figure 15-1: Floating Cone Used as Guidance for Final Pit, Metal Prices of: $18.00/oz Silver, $0.85/lb Lead, $0.85/lb Zinc

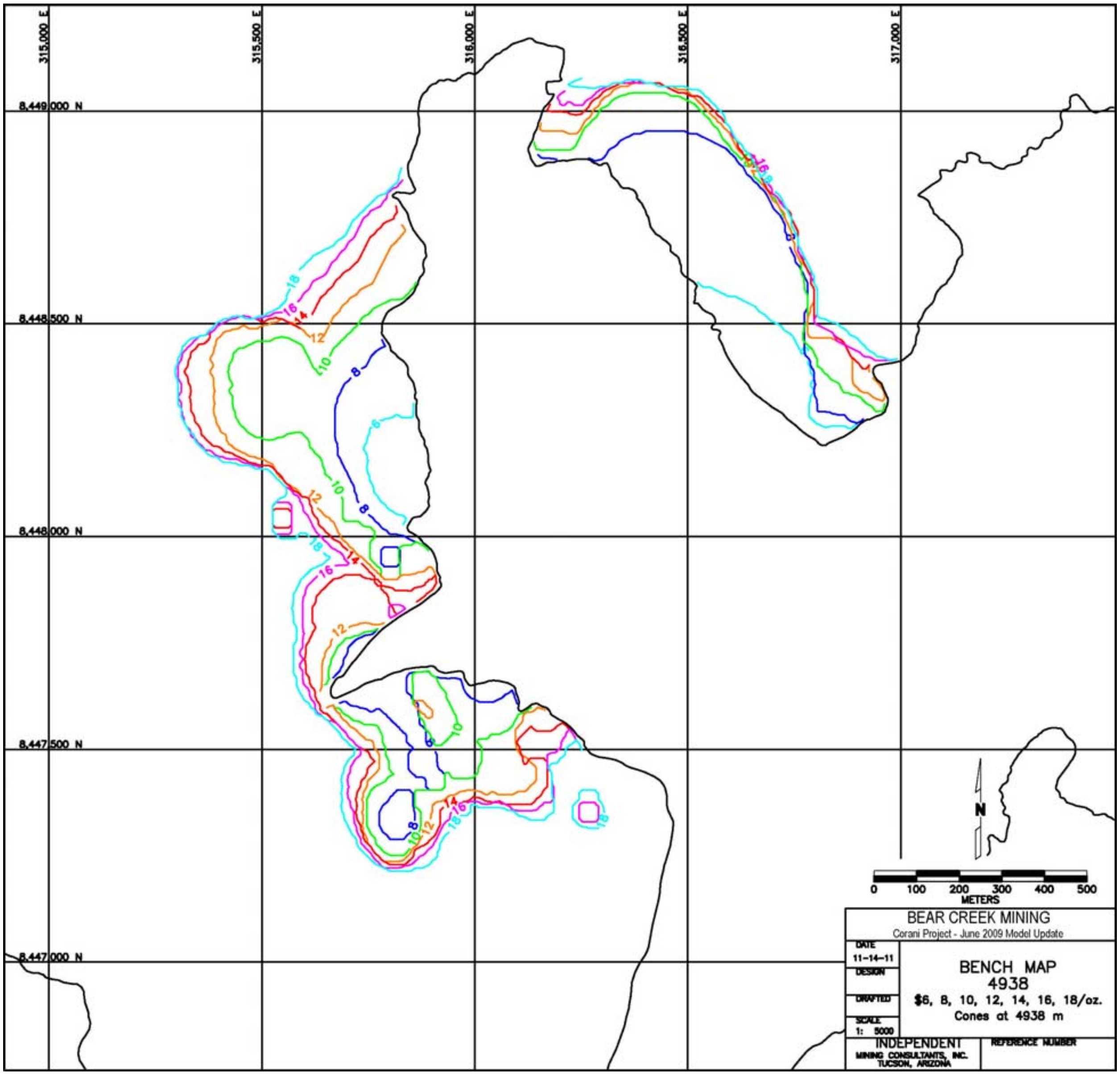


Figure 15-2: $6/oz- $18/oz Cones Sliced at 4938 m Elevation

15.2 PHASE DESIGN

A drawing of the final pit is presented below at the same scale for comparison against the $18.00 floating cone on Figure 15-1. This pit is the end result of mining 11 intermediate phases that are described in more detail in Section 16.

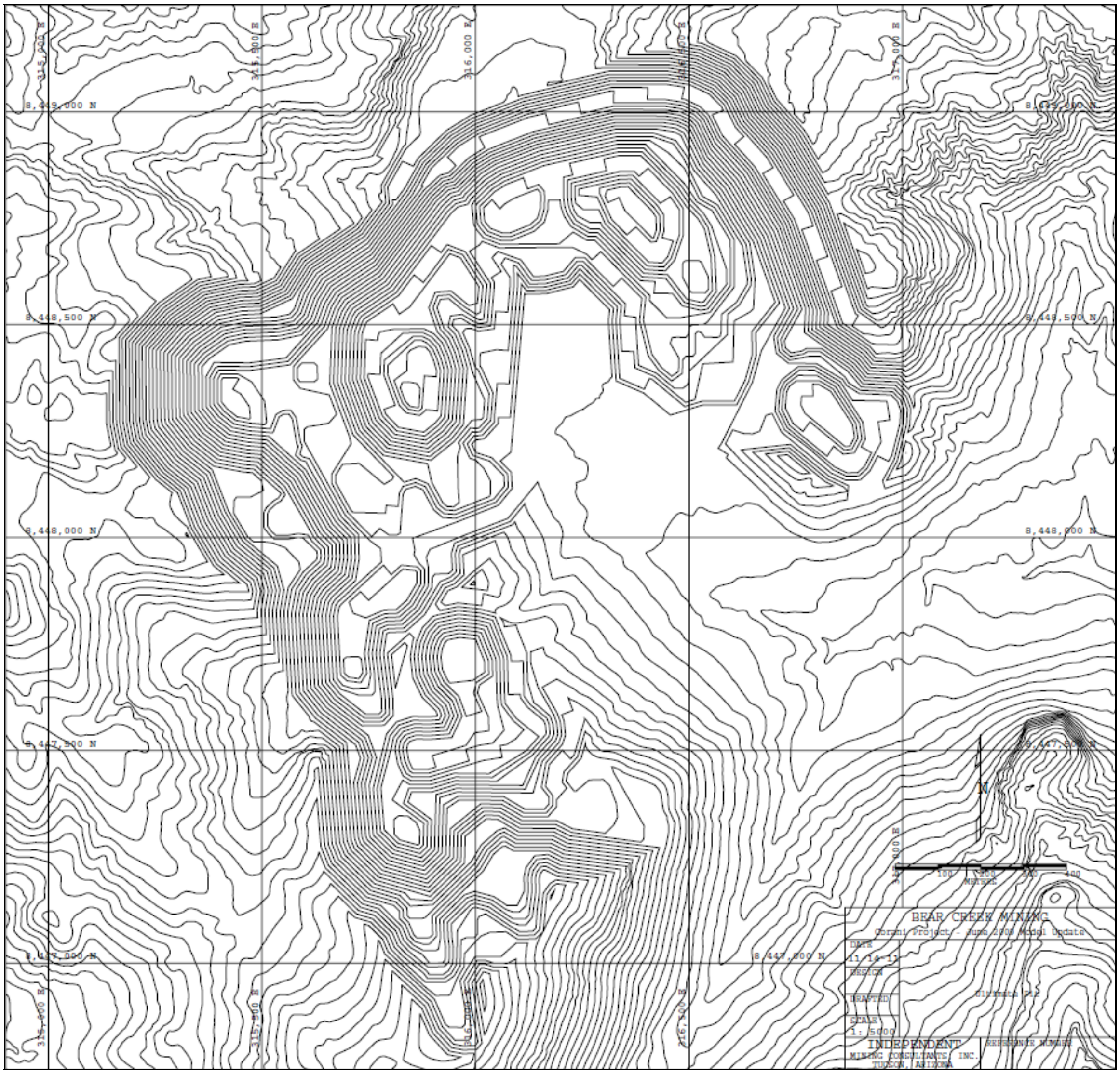


Figure 15-3: Final Pit Design

15.3 MINERAL RESERVE ESTIMATE

The Mineral Reserve is the sum of the Proven and Probable material above a Net of Process Cutoff of $2.54/tonne that is contained within the final pit geometry. This corresponds with the total material that is planned for processing which equates to the sum of mill ore and low-grade on Table 16-1.

The cutoff grade is defined by a term called Net of Processing.

Net of Processing = Net of Smelter Return – Processing Costs.

A processing cost of $8.00/tonne was assumed for all ore material; therefore the $2.54/tonne Net of Processing cutoff is equivalent to a $10.54/tonne Net Smelter Return cutoff. The Mineral Reserves and the Mineral Resources in addition to the mineral reserve are summarized in Table 15-3.

Table 15-3: Mineral Reserves and Mineral Resources

Mineral Reserves, $10.54 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	30,083	66.6	1.04	0.60	64.4	690.4	399.9	115.7	119.6
Probable	126,047	50.7	0.87	0.47	205.6	2,422.6	1,297.7	381.5	94.1
Proven + Probable	156,130	53.8	0.90	0.49	270.0	3,113.0	1,697.6	497.2	99.1

Mineral Resources in Addition to Reserves, $9.20 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	10,878	17.5	0.38	0.33	6.1	91.1	79.1	13.9	39.6
Indicated	123,583	20.8	0.38	0.29	82.6	1,035.3	790.1	166.7	42.0
Measured + Indicated	134,461	20.5	0.38	0.29	88.7	1,126.4	869.2	180.6	41.8
Inferred	49,793	30.0	0.46	0.28	48.0	509.4	305.2	86.2	53.9

The qualified person for the estimation of the mineral reserve was John Marek of Independent Mining Consultants, Inc. 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 at a higher level of risk than any other Peruvian developing resource.

15.4 SENSITIVITY CHECK

As the feasibility study project neared completion, improved cost estimates for mining and processing became available as the result of first principals calculations from the process and mine engineers.

Process costs generally increased from the by about 6 to 7% compared with the floating cone input table. General and Administration (G&A) increased about 17%, and mining costs increased about 10% compared with the cone inputs. There were other changes throughout the project cost structure including minor reductions to smelting costs.

IMC completed an additional floating cone run at the end of the project to see if the cost changes had a negative impact on the statement of mineral reserves. A tabulation of the ore contained within the Final Pit design was also completed at alternative cutoff grades and costs. The results of the final pit tabulation and the floating cone were quite similar.

The table below summarizes the potential impact of changing process, mining, and G&A costs on the ore within the mine plan.

Table 15-4: Impact of Cost Changes on Ore in the Final Put, Measured and Indicated Category Ore

Cutoff $/t	Ore Ktonnes	Net Proc. $/tonne	Silver gm/t	Lead %	Zinc %	Total Ktonnes	Process $/t	Mining $/t	G&A $/t	Silver Pr. $/oz
Reserve as in Report										
2.54	156,130	18.69	53.79	0.904	0.493	420,199	8.00	1.34	1.20	18.00
Change costs Keeping Silver Price the Same										
2.88	146,570	19.18	56.00	0.934	0.504	420,199	8.53	1.48	1.40	18.00
Change costs and Increase the Silver Price to get to Previous Tonnage										
2.88	156,126	20.72	53.88	0.903	0.490	420,199	8.53	1.48	1.40	20.50

In summary, the increased costs and corresponding increased cutoff grade would reduce the mill ore tonnes by about 6%. If one were to increase the silver price to offset that ore loss, a metal price of $20.50/oz would establish the same estimate of ore production. This analysis places all of the metal price change on silver, without modifications to the lead or zinc prices.

The results above suggest that a silver price of $20.50/oz is a value of significant interest when completing the price sensitivity during financial analysis.

16 MINING METHODS

16.1 SUMMARY

The mineral reserve is the total of all measured and indicated (proven and probable) material that is planned for processing within the mine plan. The mine plan was developed to deliver 7,875 ktonnes of ore per year (22,500 tpd) to the crusher for processing by flotation to produce two concentrates: 1) lead-silver, and 2) zinc-silver concentrate.

The steps applied by IMC in the development of the mine plan were as follows:

1) Floating cone guidance to phase design
2) Phase designs
3) Mine production schedule (strategy to maximize project return on investment)
4) Waste storage design and material allocation
5) External haul road design
6) Time sequence mine plan drawings
7) Equipment and Manpower requirements

The steps taken to develop the mine plan are described in the following sub-sections.

16.2 PHASE DESIGN

The suite of floating cones described in the previous section was used as a design guide for phase geometry and direction of phase progression. Phases or pushbacks are practical expansions of an open pit that culminate in the final pit design. Phase designs include all internal access roads and assure proper operating room for mine equipment. A total of 11 phases were designed to arrive at the ultimate pit; the phases in extraction order are:

Este Phase 1
Este Phase 2
Minas Phase 1
Main Phase 1
Este Phase 3
Minas Phase 2
Este Phase 4
Este Final
Minas Phase 3
Main Final
Minas Final

The phases are extracted in economic and practical sequence with preference given to phases that have the lowest cost of metal production. The mine will never look like any one phase design until the final pit. Except for pre-production and the final year, multiple phases are active at any given time. For example, waste is being removed from Phases 2 and 3 while ore is being mined from Phase 1.

The mine design parameters for the phase designs were as follows:

Haul Road Width Including Ditches and Berms:	28 meters
Maximum Haul Road Grade:	10%
Interramp Slope Angles:	42 degrees in Pre-Mineral Tuff
	46 Degrees in Post-Mineral Tuff

Figure 16-1 illustrates the 11 pushbacks that were designed as input to the mine schedule at the 4938 m bench elevation. The number in each area indicates the extraction sequence of the phases starting with Este Phase 1 and finishing with the Minas Final Pit. Figure 16-1 is on the same bench as Figure 15-2 so the theoretical cone results and the practical phase extraction sequence can be compared.

Figure 15-3 in the previous chapter represents the final pit design for the Corani Feasibility study. Figure 15-3 is at the same scale as Figure 16-1 so the $18.00 cone and the final pit design with haul roads can be compared directly.

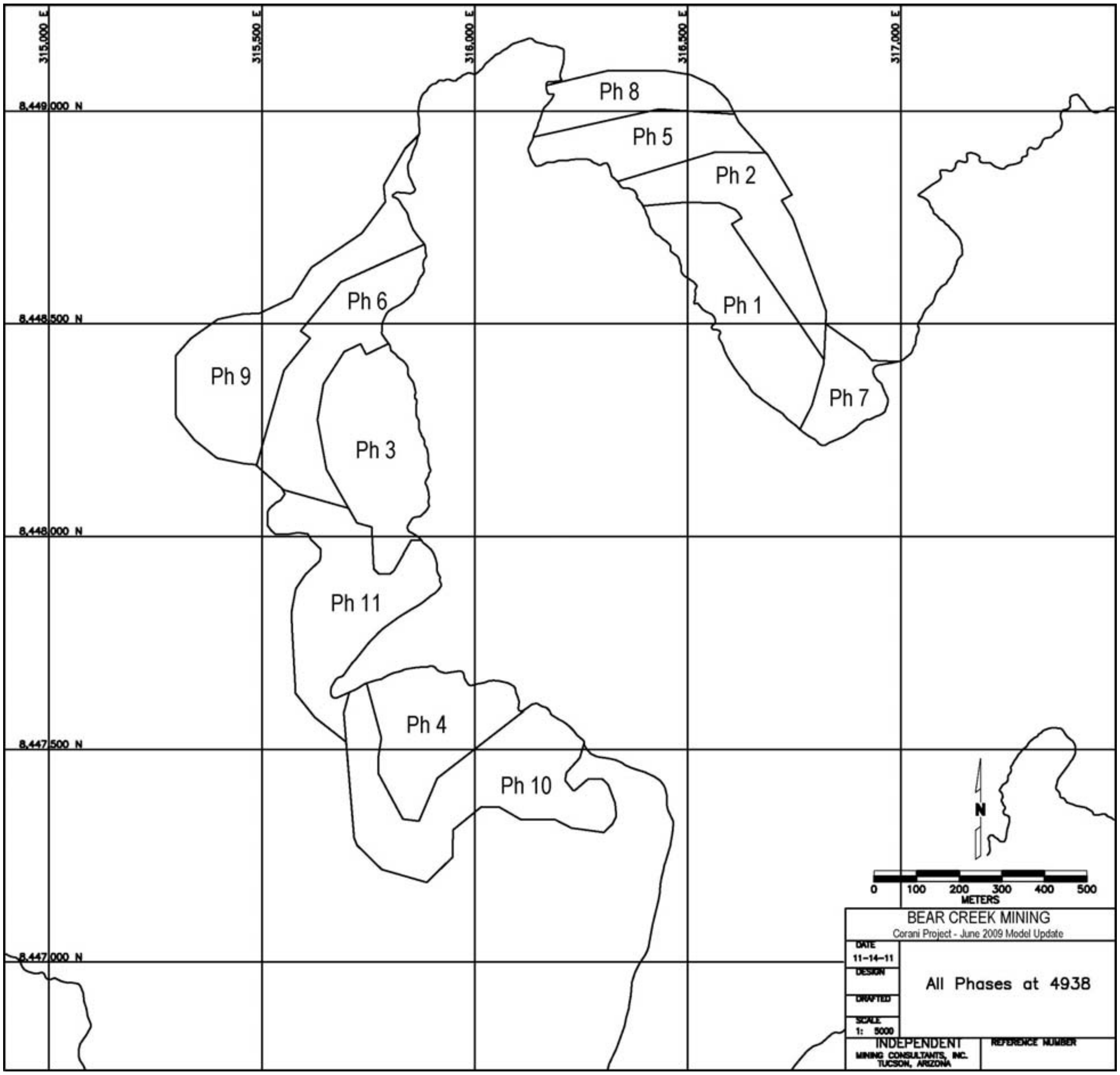


Figure 16-1: Phases Sliced at 4938 m Elevation

16.3 MINE PRODUCTION SCHEDULE

The mine schedule was developed based on the phase designs and the block model. The material contained within each pushback design was tabulated at multiple cutoff grades for input to the mine schedule process. As with the floating cones, only measured and indicated category mineralization was tabulated from the pushback designs. All other material was treated as waste in the mine schedule.

A tradeoff study was completed between the pre-feasibility and feasibility studies so that the optimum ore production rate would be used for the feasibility study. 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 of moving profits forward and increasing capital costs. The mine schedule was developed to produce 22,500 tpd or 7,875 ktonnes of ore per year to the crusher and flotation concentrator. Sufficient waste stripping is planned to assure continued release of the ore at the design level.

Many iterations of the production schedule were developed in order to establish a sound overall approach to the mine operating strategy. Maximizing project NPV at the design prices of \$18.00/oz silver, \$0.85/lb lead, and \$0.85/lb zinc was part of the scheduling criteria incorporating a discount rate of 7% to reflect the changing value of costs and benefits over time.

Cutoff grades were based on Income Net of Process at the design prices in the previous paragraph. Initially mining costs were estimated to be \$1.34/tonne material with G&A costs of \$1.20/tonne ore. Consequently, the breakeven cutoff grade on this basis would be \$2.54 Net of process/tonne. Net of Process = NSR – Process Cost.

Table 16-1 summarizes the mine production schedule that was developed for the feasibility study. The mill cutoff is higher than breakeven for preproduction through year 8 as part of the effort to maximize project return on investment.

Figure 16-2 is a graphic summary of the material movements and metal produced from the mine plan. It should be noted that the minor ore produced during preproduction stripping will be stockpiled and rehandled during year 1 production.

During the period of preproduction through year 8, the material with grade between the breakeven cutoff and the mill cutoff is stockpiled for later processing. That material is planned for rehandle and processing in years 18 through 20.

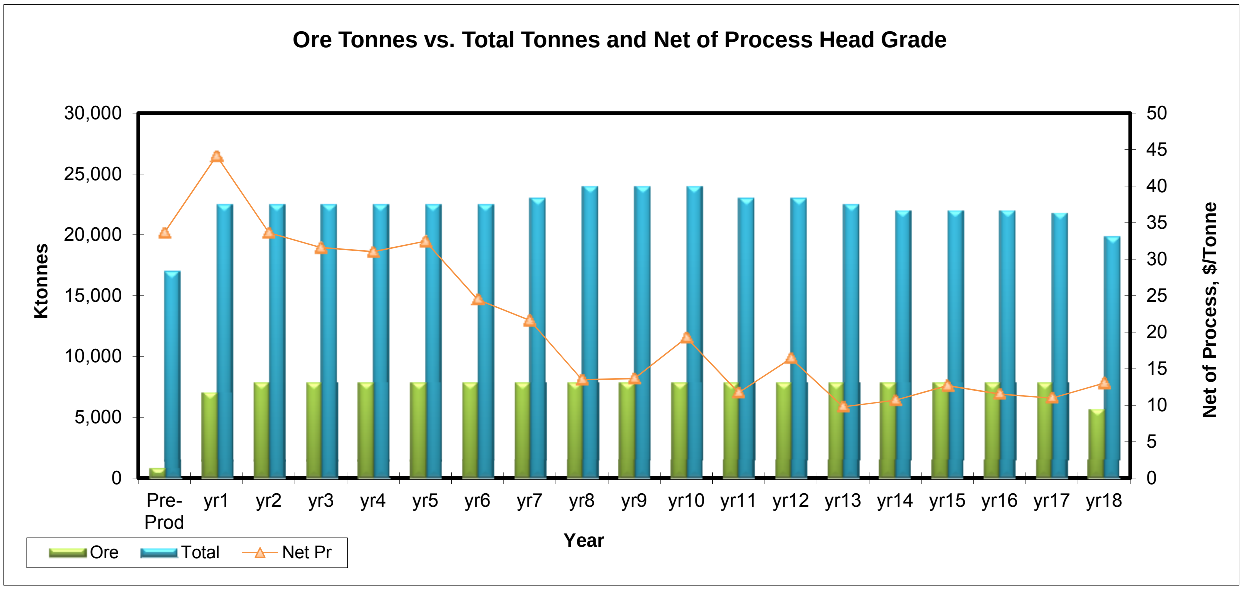
Figure 16-3 is an overall illustration of the end of the mine life prior to remining of the low grade stockpiles. This figure illustrates the mine, waste storage, and low grade stockpile locations along with the crusher location.

Table 16-1 represents the mineral reserve because the combined proven and probable mineral reserve corresponds to the total ore processed in the mine. The proven material and probable category material were split out and reported in combination with the mineral resources in the previous chapter.

Table 16-1: Corani Mine Production Schedule

$18/oz Silver, $0.85/lb Lead, $0.85/lb Zinc

	Direct Feed Mill Ore												Material to Low Grade Stockpile $2.54 Net of Process Cutoff														Rehandle				
				Ore Head Grade			Rec. Metal to Lead Concentrate			Rec. Metal to Zinc Concentrate					Ore Head Grade			Rec. Metal to Lead Concentrate			Rec. Metal to Zinc Concentrate				Total			Pit	PMT	Total
Year	Cutoff $/tonne	Ore kTonnes	Net Proc. $/tonne	Silver g/t	Lead %	Zinc %	Silver g/t	Lead %	Zinc %	Silver g/t	Lead %	Zinc %	Low Grd kTonnes	NSR $/tonne	Silver g/t	Lead %	Zinc %	Silver g/t	Lead %	Zinc %	Silver g/t	Lead %	Zinc %	Waste kTonnes	Mined kTonnes	Ore stkpl kTonnes	LG stkpl kTonnes	Backfill kTonnes	Cap kTonnes	Material kTonnes
ppQ-5																								576	576					576
ppQ-4																								1,478	1,478					1,478
ppQ-3																								3,696	3,696					3,696
ppQ-2	13.00	1	19.94	85.11	0.666	0.032	47.26	0.301	0.002	0.00	0.000	0.000												5,624	5,625					5,625
ppQ-1	13.00	803	33.66	104.81	1.047	0.061	64.27	0.675	0.005	0.22	0.002	0.003	45	11.71	60.59	0.605	0.025	31.97	0.300	0.001	0.00	0.000	0.000	4,777	5,625					5,625
yr1Q1	13.00	1,251	39.78	110.33	1.226	0.216	67.79	0.892	0.019	2.20	0.011	0.079	93	8.53	48.78	0.336	0.032	26.90	0.197	0.002	0.00	0.000	0.000	4,281	5,625	718				6,343
yr1Q2	13.00	1,978	43.70	106.98	1.205	0.917	66.17	0.900	0.082	6.04	0.028	0.561	276	8.83	49.53	0.296	0.038	27.71	0.174	0.003	0.01	0.000	0.000	3,371	5,625					5,625
yr1Q3	13.00	1,908	45.71	105.27	1.127	1.449	65.19	0.844	0.130	8.11	0.039	0.939	185	8.51	45.18	0.475	0.113	24.91	0.286	0.010	0.26	0.004	0.024	3,532	5,625	60				5,685
yr1Q4	13.00	1,934	45.92	103.59	1.211	1.452	64.23	0.908	0.130	8.22	0.044	0.936	84	7.81	33.38	0.663	0.434	19.74	0.471	0.039	1.22	0.018	0.201	3,607	5,625	35				5,660
yr2Q1	12.00	1,969	39.92	96.55	1.320	0.853	59.50	0.969	0.077	5.07	0.031	0.518	253	7.15	35.71	0.770	0.259	18.75	0.459	0.022	0.57	0.014	0.098	3,403	5,625					5,625
yr2Q2	12.00	1,969	36.61	88.44	1.402	0.660	54.51	1.034	0.059	3.77	0.028	0.384	776	5.93	31.63	0.676	0.209	17.17	0.427	0.017	0.31	0.007	0.064	2,880	5,625					5,625
yr2Q3	12.00	1,969	32.46	82.12	1.409	0.408	50.57	1.036	0.037	1.76	0.020	0.205	1,021	5.81	27.32	0.632	0.257	15.77	0.438	0.022	0.32	0.006	0.098	2,635	5,625					5,625
yr2Q4	12.00	1,968	25.45	77.42	0.942	0.178	47.36	0.670	0.015	1.06	0.006	0.047	701	6.21	36.57	0.490	0.104	20.55	0.310	0.008	0.00	0.000	0.000	2,956	5,625					5,625
3	12.00	7,875	31.58	84.31	1.047	0.621	51.30	0.743	0.054	4.66	0.024	0.345	2,471	7.39	41.75	0.510	0.283	22.79	0.308	0.022	0.61	0.004	0.109	12,154	22,500					22,500
4	12.00	7,875	31.02	80.53	1.263	0.647	48.80	0.880	0.055	4.02	0.032	0.332	4,049	6.96	37.35	0.590	0.289	20.03	0.358	0.022	0.42	0.006	0.087	10,576	22,500					22,500
5	11.00	7,875	32.46	77.95	1.496	0.531	47.66	1.091	0.046	3.83	0.044	0.258	2,602	5.96	30.57	0.631	0.323	16.96	0.405	0.026	0.52	0.008	0.116	12,023	22,500					22,500
6	10.00	7,875	24.44	65.53	1.175	0.449	39.74	0.839	0.039	2.22	0.025	0.214	2,610	6.05	27.95	0.607	0.284	16.21	0.422	0.025	0.55	0.012	0.114	12,015	22,500					22,500
7	5.00	7,875	21.60	61.84	0.998	0.529	37.06	0.682	0.046	2.17	0.021	0.288	1,066	3.68	23.08	0.496	0.304	13.39	0.341	0.027	0.59	0.008	0.118	14,059	23,000					23,000
8	3.00	7,875	13.47	44.75	0.805	0.322	26.71	0.563	0.028	0.88	0.013	0.132	325	2.76	17.65	0.489	0.338	10.78	0.363	0.030	0.58	0.010	0.128	15,800	24,000					24,000
9	2.54	7,875	13.63	35.30	0.969	0.439	21.89	0.727	0.040	1.30	0.026	0.228												16,125	24,000					24,000
10	2.54	7,875	19.22	53.93	0.811	0.470	33.40	0.607	0.042	2.23	0.019	0.270												16,125	24,000					24,000
11	2.54	7,875	11.71	47.67	0.828	0.259	26.43	0.515	0.020	0.43	0.006	0.073												15,125	23,000					23,000
12	2.54	7,875	16.39	55.99	0.845	0.201	33.18	0.577	0.017	0.47	0.006	0.053												15,125	23,000					23,000
13	2.54	7,875	9.76	34.81	0.615	0.346	21.34	0.456	0.031	0.80	0.013	0.151												14,625	22,500					22,500
14	2.54	7,875	10.66	34.79	0.782	0.376	20.92	0.565	0.033	1.02	0.018	0.161												14,125	22,000					22,000
15	2.54	7,875	12.67	37.98	0.861	0.564	22.36	0.604	0.050	1.48	0.025	0.293												14,125	22,000					22,000
16	2.54	7,875	11.53	34.91	0.632	0.571	21.50	0.470	0.051	2.16	0.023	0.308												14,125	22,000					22,000
17	2.54	7,875	10.95	29.11	0.670	0.735	18.05	0.502	0.066	2.39	0.024	0.446												13,869	21,744				1,200	22,944
18	2.54	5,698	13.01	33.37	0.660	0.890	20.69	0.495	0.080	2.46	0.021	0.546												11,257	16,955		2,177		2,288	21,420
19																											7,875	4,344	4,154	16,373
20																											6,505		5,576	12,081
Totals:		139,573	20.15	56.18	0.943	0.520	34.067	0.675	0.05	2.292	0.022	0.273	16,557	6.36	33.61	0.577	0.273	18.606	0.370	0.02	0.465	0.007	0.10	264,069	420,199	813	16,557	4,344	13,218	455,131



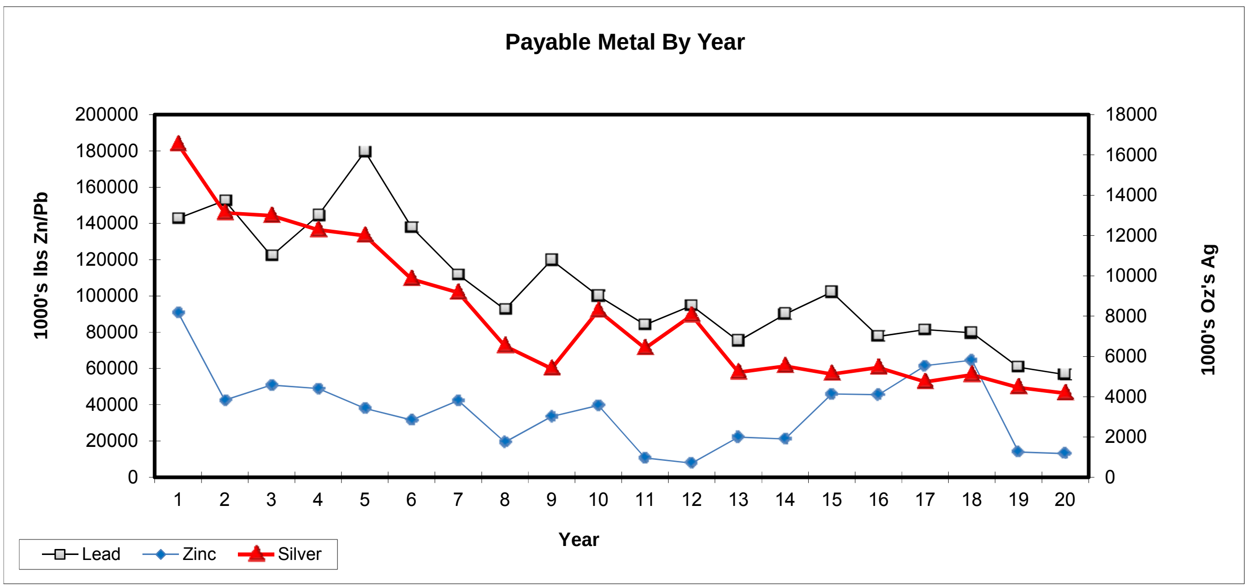


Figure 16-2: Mine Material Movement and Payable Metal

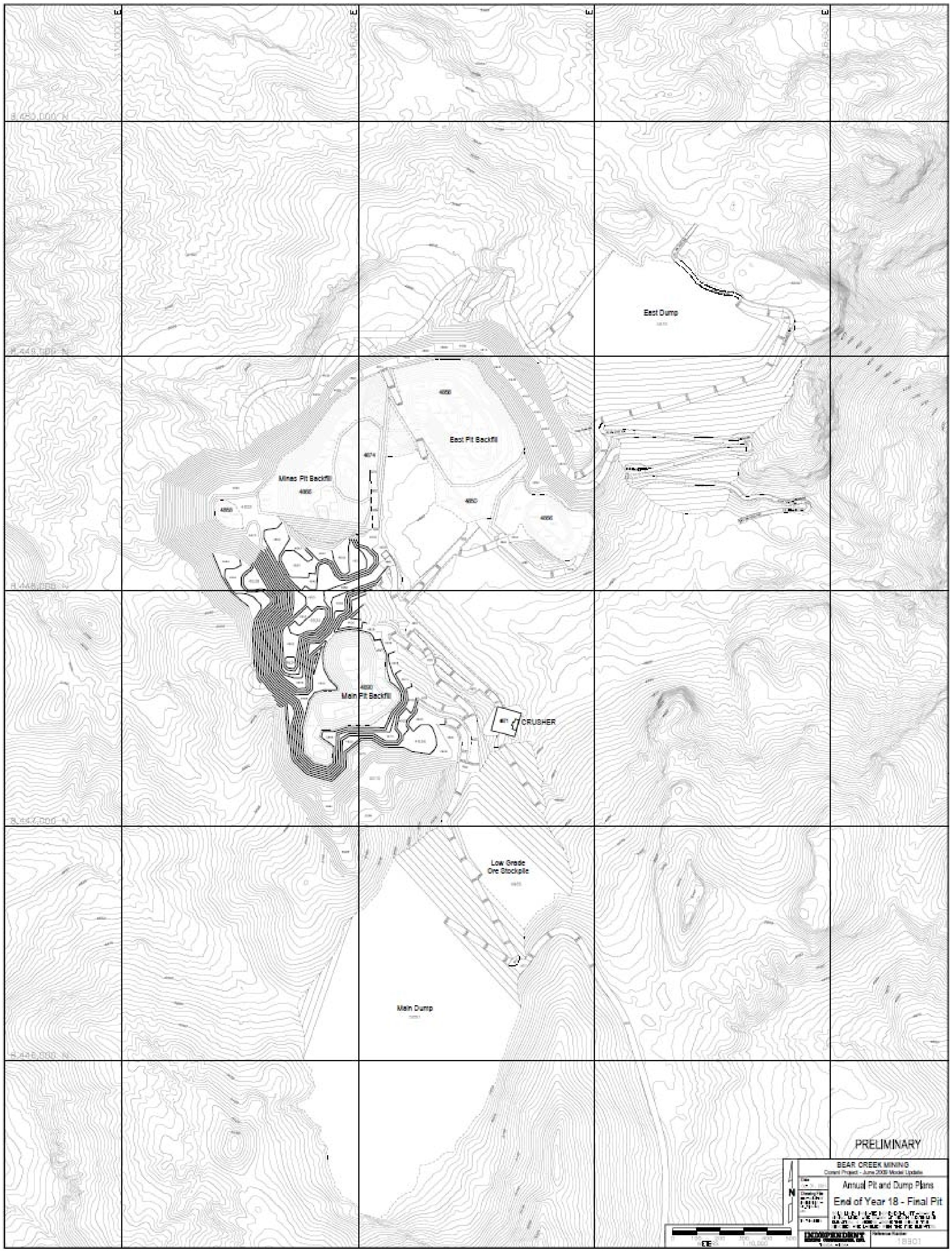


Figure 16-3: End of Year 18

Refer to Figure 18-7 for the final configuration of the site layout.

16.4 WASTE STORAGE

All waste is sent to two external waste dumps for the first 11 years of mine life except for a small amount used in the construction of the TSF dam. The Main dump is south of the Main pit and the East dump is east of the Este pit. For this Feasibility study, waste was separated into two different categories of Pre-Mineral waste and Post Mineral Tuft (PMT). Selective handling has been incorporated to manage Pre-Mineral waste, which is potentially acid-generating. Only PMT is sent to the East dump and Pre-Mineral waste is only sent to the Main dump. PMT is also scheduled to be placed in the main dump throughout the dump's operating period. As well, 17.2 million tonnes of PMT are required for the construction of the tailing dam. Waste movement is presented in Table 16-2. Both the Este and Minas pit have an overlying layer of Post-Mineral Tuft that is considered environmentally benign.

Roughly 33% of the waste material sent to the Main dump is PMT material and it is planned to be dumped at the perimeter of the dump to provide a buffer zone around the pre-mineral waste.

Starting in Year 12, finished pits begin getting backfilled with waste to prevent the formation of pit lakes and to take advantage of shorter haul distances. After the completion of mining, the final Minas pit is backfilled from the remining of an adjacent backfill dump so that no pit lakes form. Also, all Pre-Mineral dumps are capped with PMT material and this material movement is shown towards the end of mine life in the mine schedule in Table 16-1.

A more detailed discussion of mine waste management is described in Section 18.2.2.3.

Table 16-2: Waste Movement Allocation

Post Mineral Waste (PMT) Movement Plan

Phase → Year	East1a Ktonnes	dest	East1 Ktonnes	dest	Min1 Ktonnes	dest	Main1 Ktonnes	dest	East2 Ktonnes	dest	Min2 Ktonnes	dest	East3 Ktonnes	dest	EastFin Ktonnes	dest	Min3 Ktonnes	dest	MinF Ktonnes	dest	MainFin Ktonnes	dest	Destination Codes 1: East Ktonnes	Destination Codes 2: Main Ktonnes	PMT to TSF
0	14,694	1	1,324	1																			14,118		1,900
1	2,347	1	11,786	1																			12,833		1,300
2			7,623	1	288	2																	4,870	1,141	1,900
3			312	2	5,886	2			2,524	1													2,524	3,448	2,750
4					2,995	2			3,534	1													3,534	1,345	1,650
5									4,114	1	3,343	1			397	1							5,354		2,500
6									2,538	1	1,591	2			3,979	1							6,517	291	1,300
7											1,688	2	2,874	2	5,534	1							5,534	3,262	1,300
8											5,310	2			2,548	1							2,548	4,010	1,300
9											28	2			10,587	2								9,315	1,300
10															3,667	2	7,648	1					7,648	3,667	
11															1	2	7,948	1					7,948	1	
12																	6,877	2						6,877	
13																	1,319	2	14	2				1,333	
14																	6	2						6	
15																									
16																									
17																									
18																									
TOTAL	17,041		21,045		9,169		0		12,710		11,960		2,874		26,713		23,798		14		0		73,428	34,696	17,200

Pre-Mineral Waste Movement Plan

Phase → Year	East1a Ktonnes	dest	East1 Ktonnes	dest	Min1 Ktonnes	dest	Main1 Ktonnes	dest	East2 Ktonnes	dest	Min2 Ktonnes	dest	East3 Ktonnes	dest	EastFin Ktonnes	dest	Min3 Ktonnes	dest	MinF Ktonnes	dest	MainFin Ktonnes	dest	Destination Codes 2: Main Dump Ktonnes	Destination Codes 3: EstBkfil Ktonnes	Destination Codes 4: Min Bkfil Ktonnes
0	133	2																					133		
1	532	2	126	2																			658		
2	1,275	2	497	2	20	2	2,172	2															3,964		
3	407	2	1,379	2	1,149	2	497	2															3,432		
4	1,056	2	793	2	1,676	2	495	2	28	2													4,048		
5			704	2	662	2	2,005	2	144	2	655	2											4,170		
6					897	2	1,421	2	1,089	2	500	2											3,907		
7							1,504	2	484	2	933	2	1,036	2	6	2							3,963		
8											3,556	2	4,355	2	31	2							7,942		
9											4,156	2	643	2	711	2							5,510		
10											20	2	87	2	4,499	2	203	2					4,809		
11															1,755	2	2,215	2			3,206	2	7,176		
12															1,274	2	6,447	2			527	2	4,281	3,967	
13																	5,198	3	1,761	2	6,332	2	8,093	5,198	
14																	4,797	3	1,389	3	7,933	2	7,933	6,186	
15																	3,962	3	2,957	3	7,206	3		14,125	
16																	84	3	5,616	4	8,424	4		84	14,040
17																			6,517	4	7,352	3		4,352	9,517
18																			11,257	4					5,761
19																									-4,344
																							Negative = rehandled for backfill		
TOTAL	3,403		3,499		4,404		8,094		1,745		9,820		6,121		8,276		22,906		29,497		40,980		70,019	33,912	24,974

Low Grade Material Schedule to the Low Grade Stockpile

Phase → Year	East1a Ktonnes	dest	East1 Ktonnes	dest	Min1 Ktonnes	dest	Main1 Ktonnes	dest	East2 Ktonnes	dest	Min2 Ktonnes	dest	East3 Ktonnes	dest	EastFin Ktonnes	dest	Min3 Ktonnes	dest	MinF Ktonnes	dest	MainFin Ktonnes	dest	4: Low Grade Ktonnes
0	45	4																					45
1	638	4																					638
2	647	4	294	4	28	4	1,782	4															2,751
3	158	4	972	4	1,053	4	288	4															2,471
4	711	4	445	4	2,592	4	301	4															4,049
5			220	4	759	4	1,390	4			233	4											2,602
6					835	4	970	4	522	4	283	4											2,610
7							556	4	77	4	166	4	267	4									1,066
8											105	4	220	4									325
9																							
10																							
11																							
12																							
13																							
14																							
15																							
16																							
17																							
18																							
TOTAL	2,199		1,931		5,267		5,287		599		787		487		0		0		0		0		16,557

de st = Destination Code on Right Side of Table

16.5 EXTERNAL HAUL ROADS AND MINE TIME SEQUENCE DRAWINGS

The ore in the Corani project is located for the most part in the walls of a glacial valley and as a result, the phases are accessed for much of the mine life from external haul roads rather than in pit haul roads. The design of these surface roads can be seen in the time sequence drawings presented in Figure 16-4 to Figure 16-12.

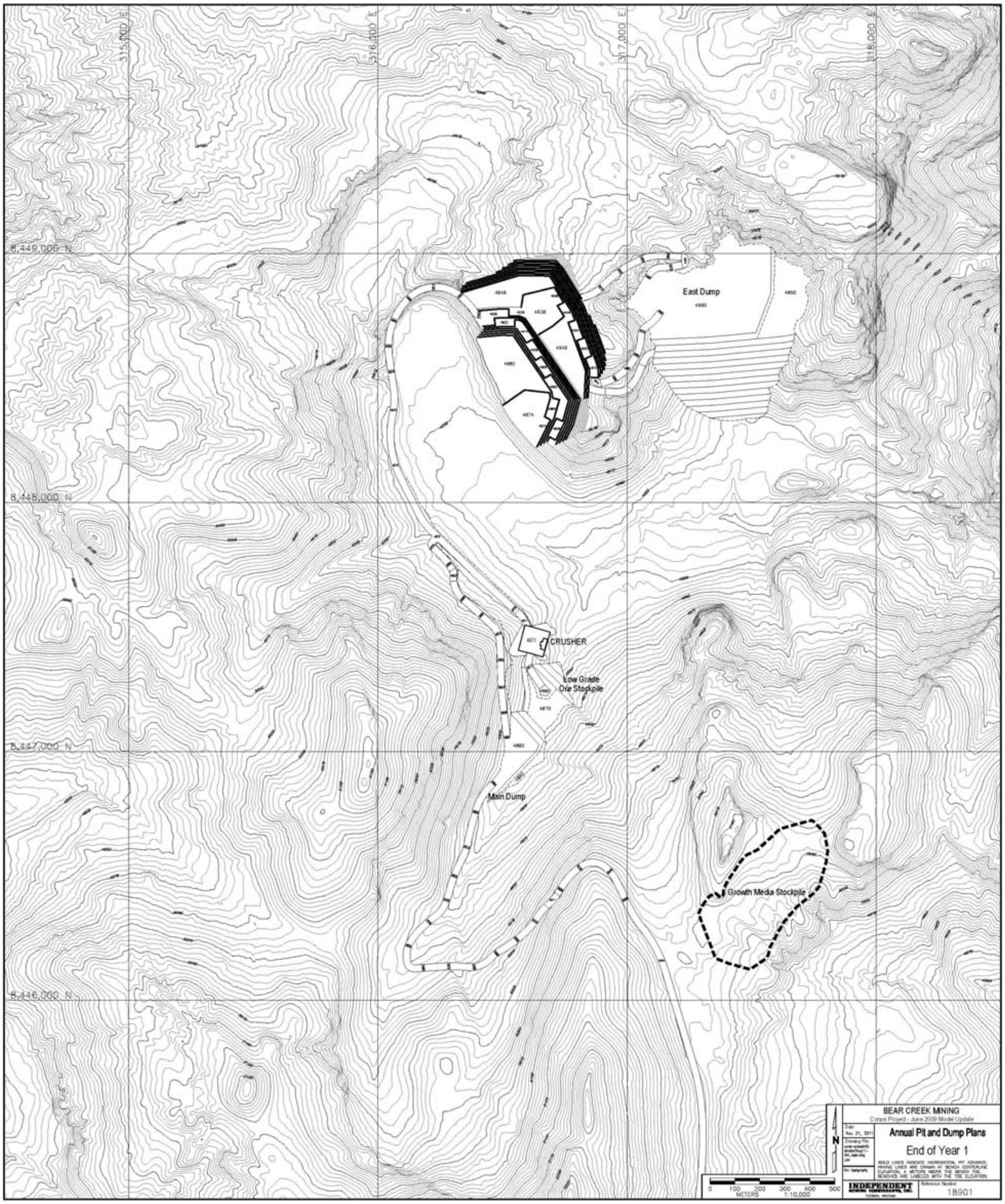


Figure 16-4: Annual Pit and Dump Plans – End of Year 1

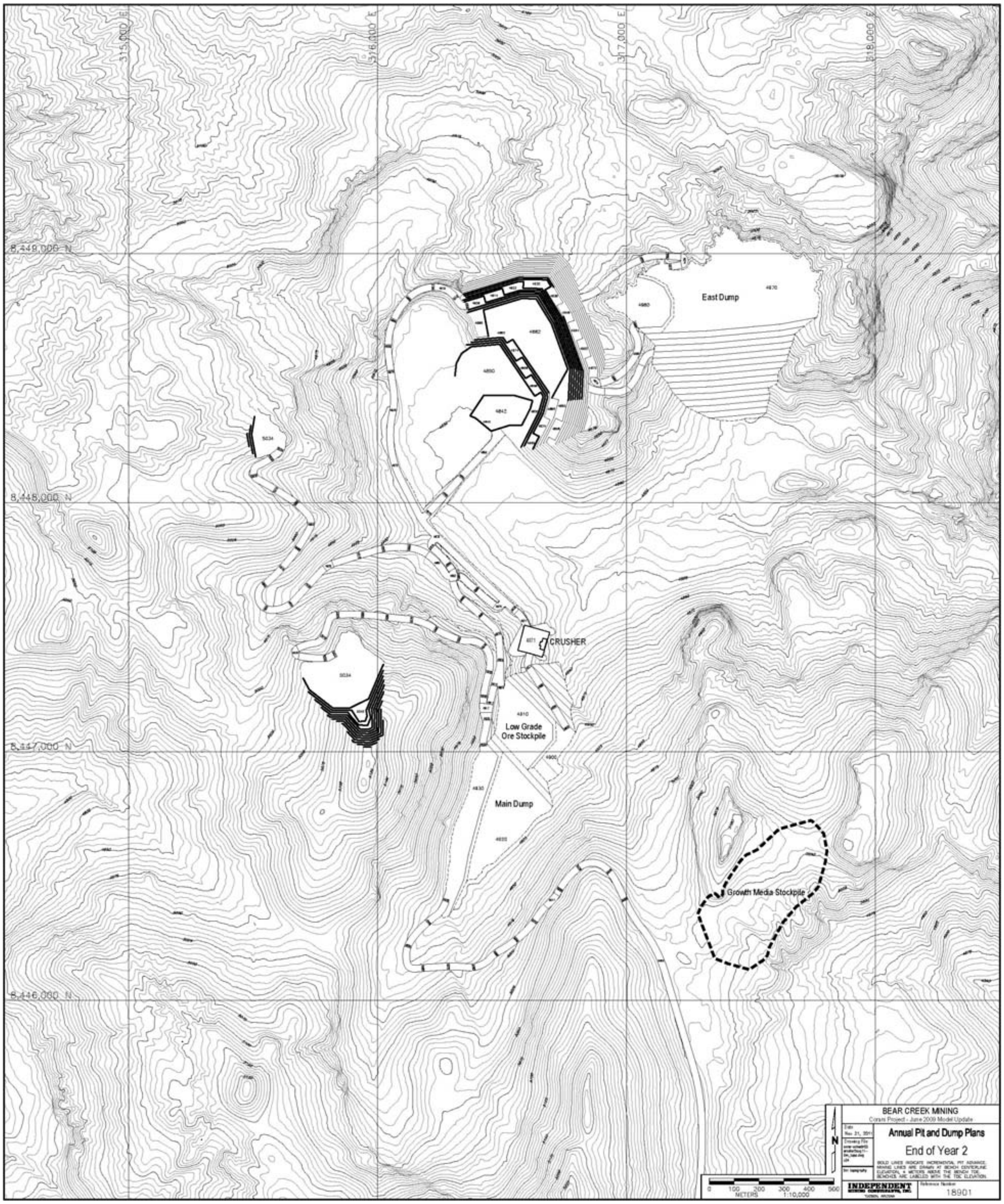


Figure 16-5: Annual Pit and Dump Plans – End of Year 2

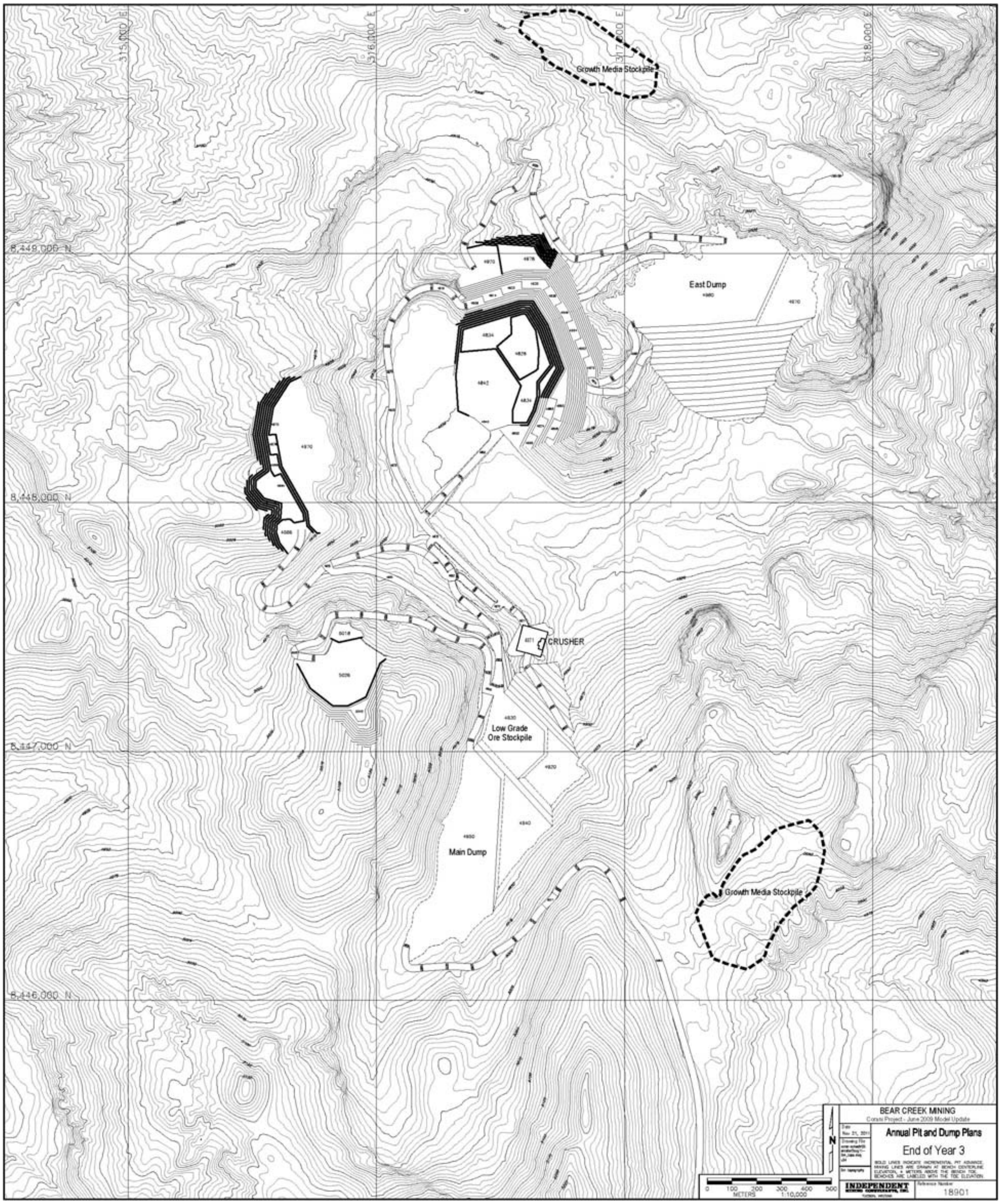


Figure 16-6: Annual Pit and Dump Plans – End of Year 3

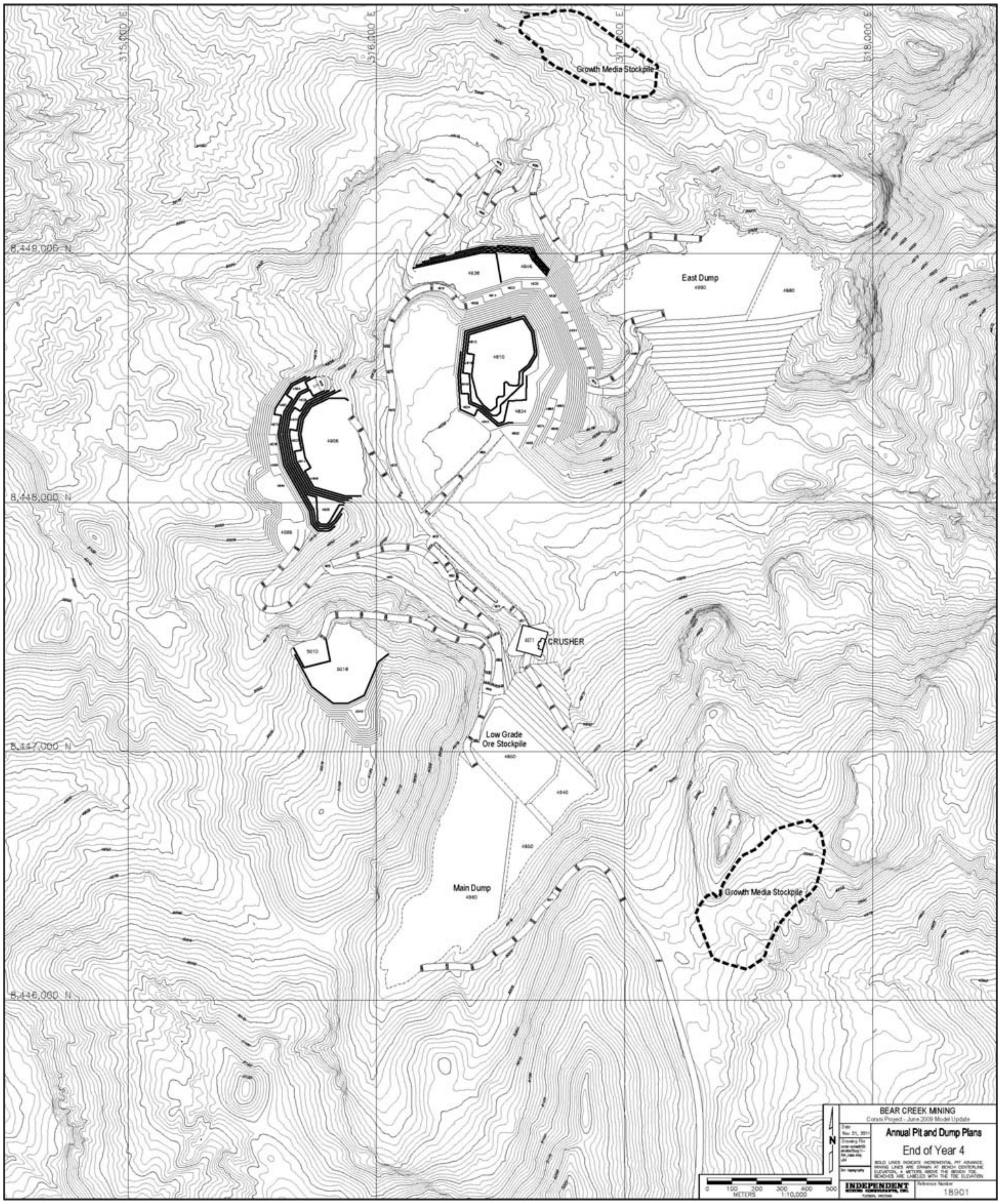


Figure 16-7: Annual Pit and Dump Plans – End of Year 4

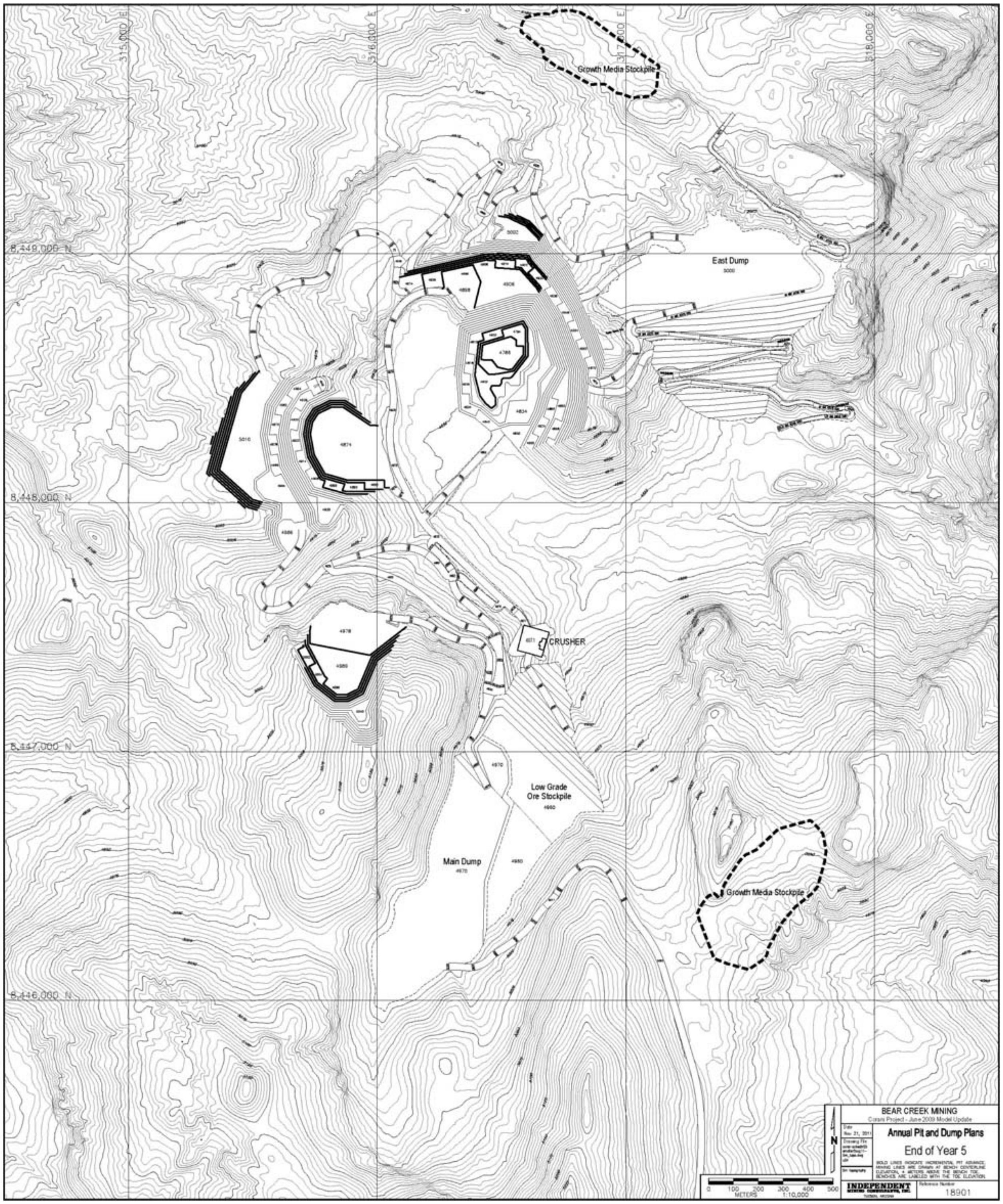


Figure 16-8: Annual Pit and Dump Plans – End of Year 5

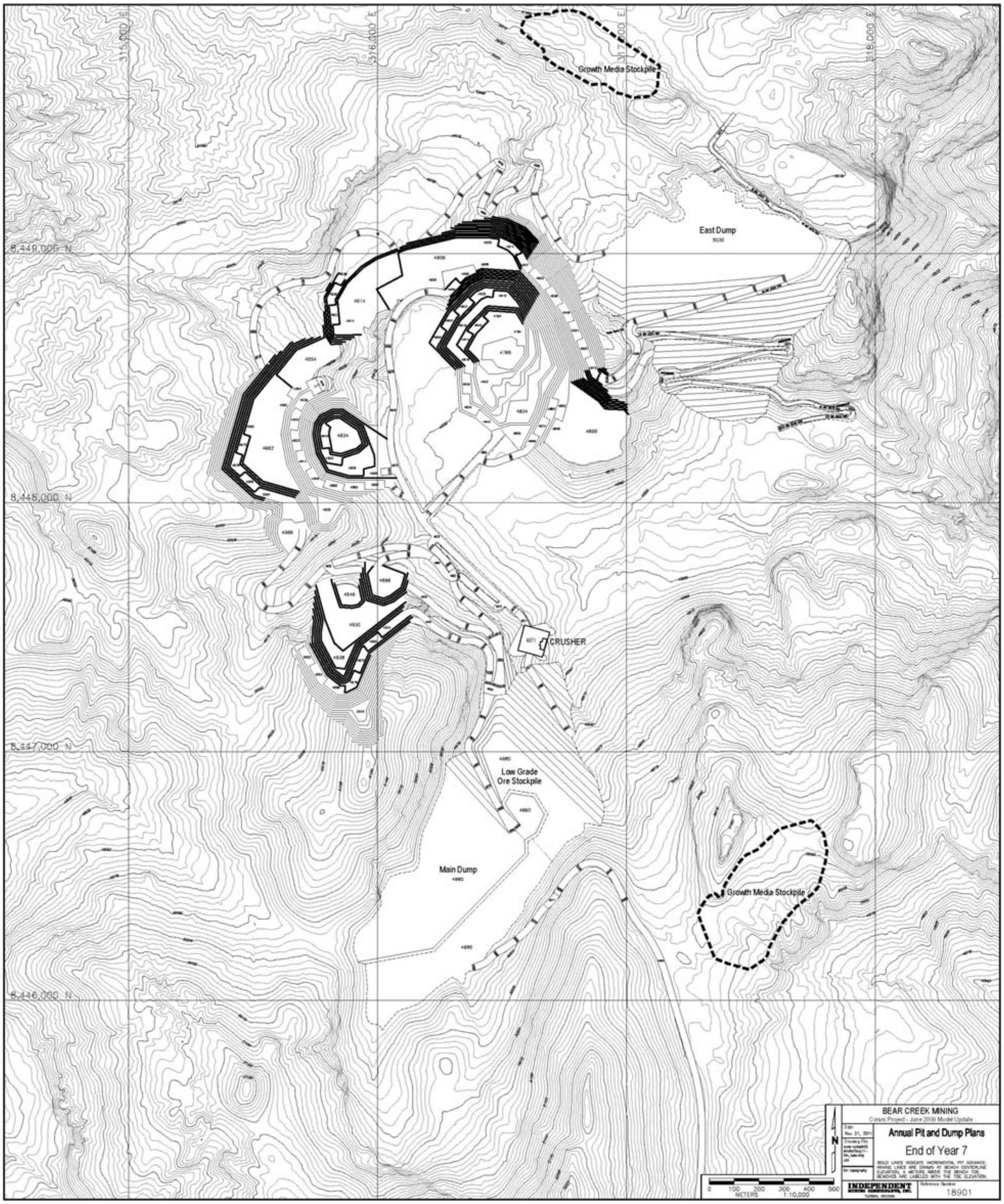


Figure 16-9: Annual Pit and Dump Plans – End of Year 7

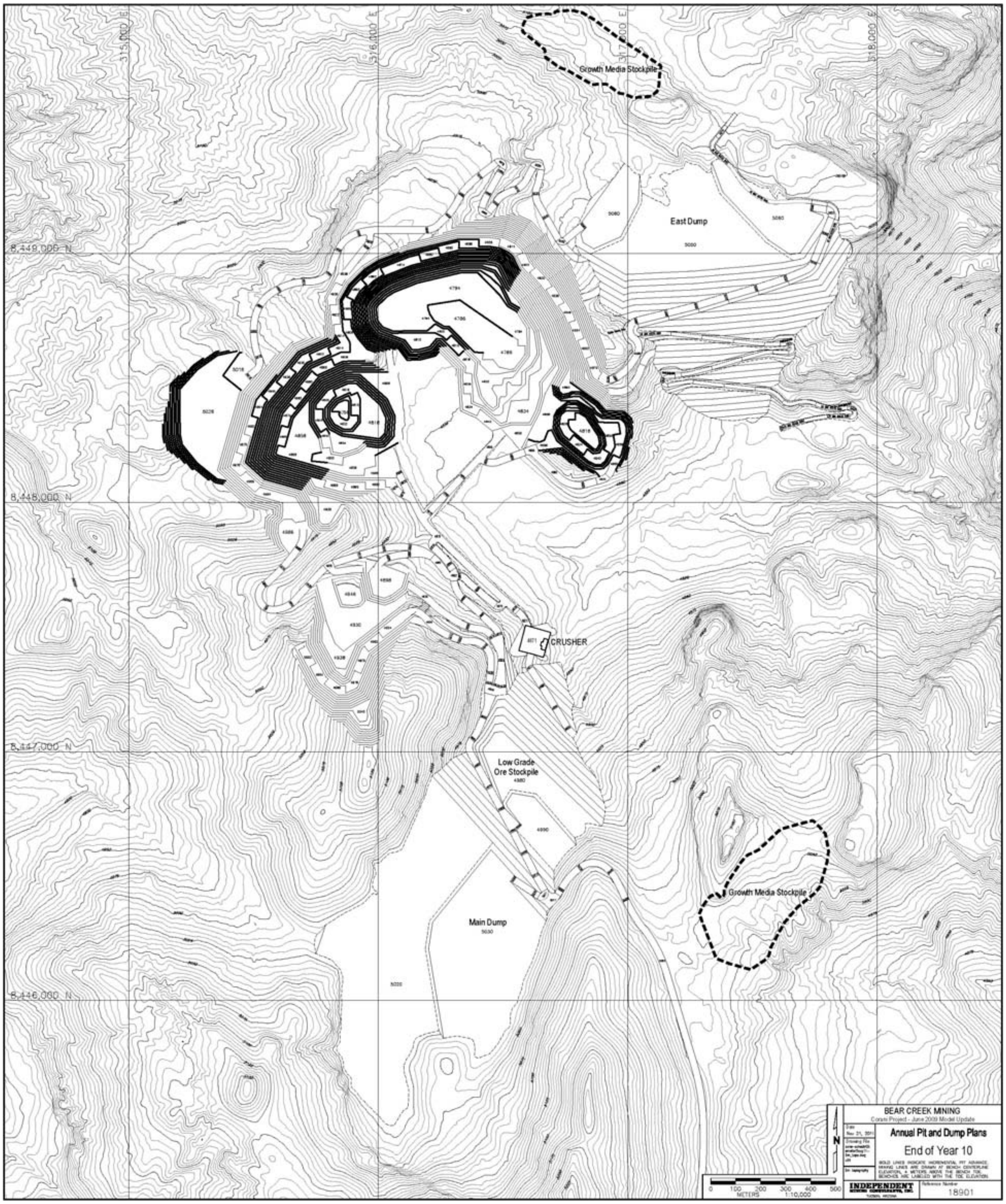


Figure 16-10: Annual Pit and Dump Plans – End of Year 10

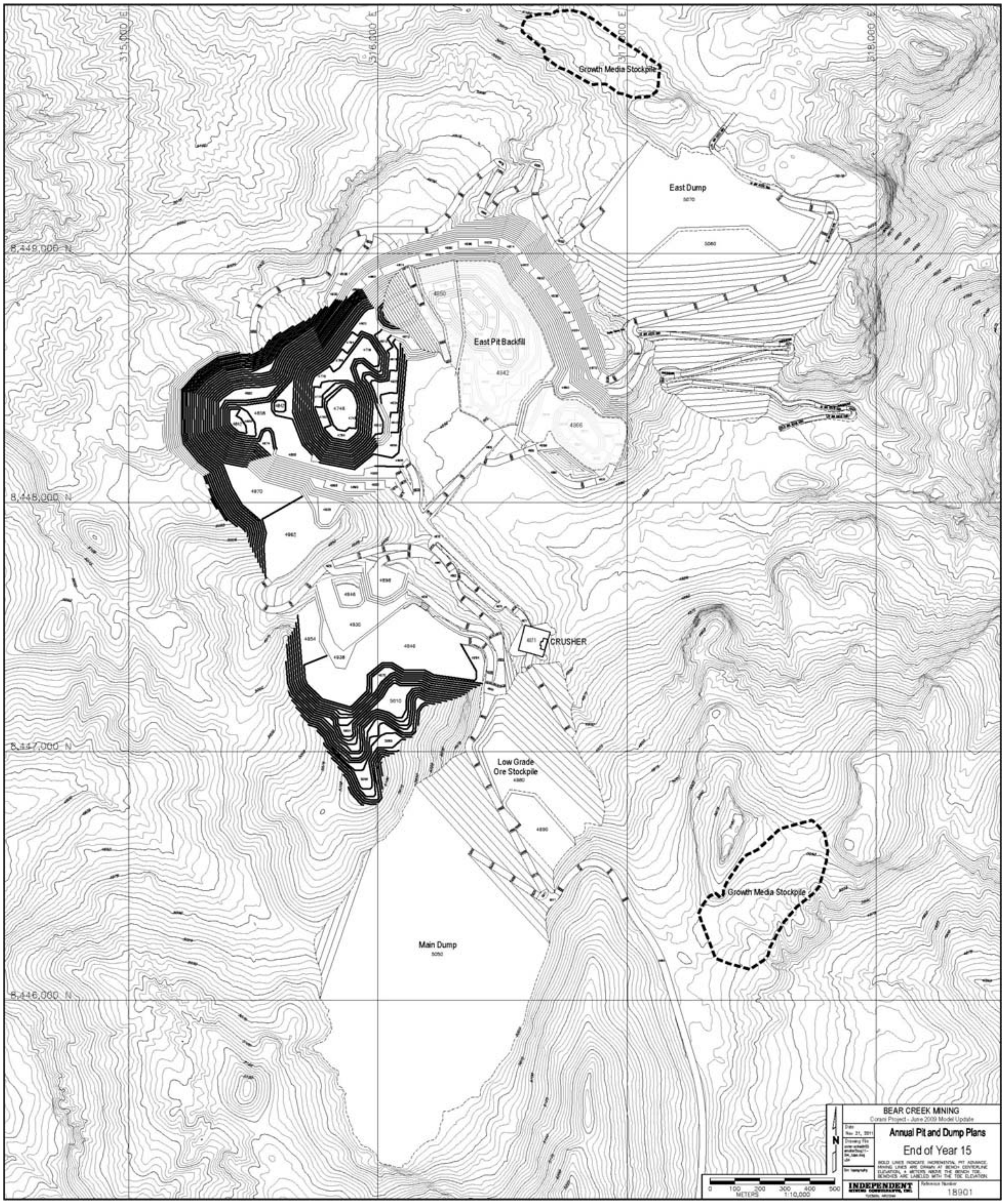


Figure 16-11: Annual Pit and Dump Plans – End of Year 15

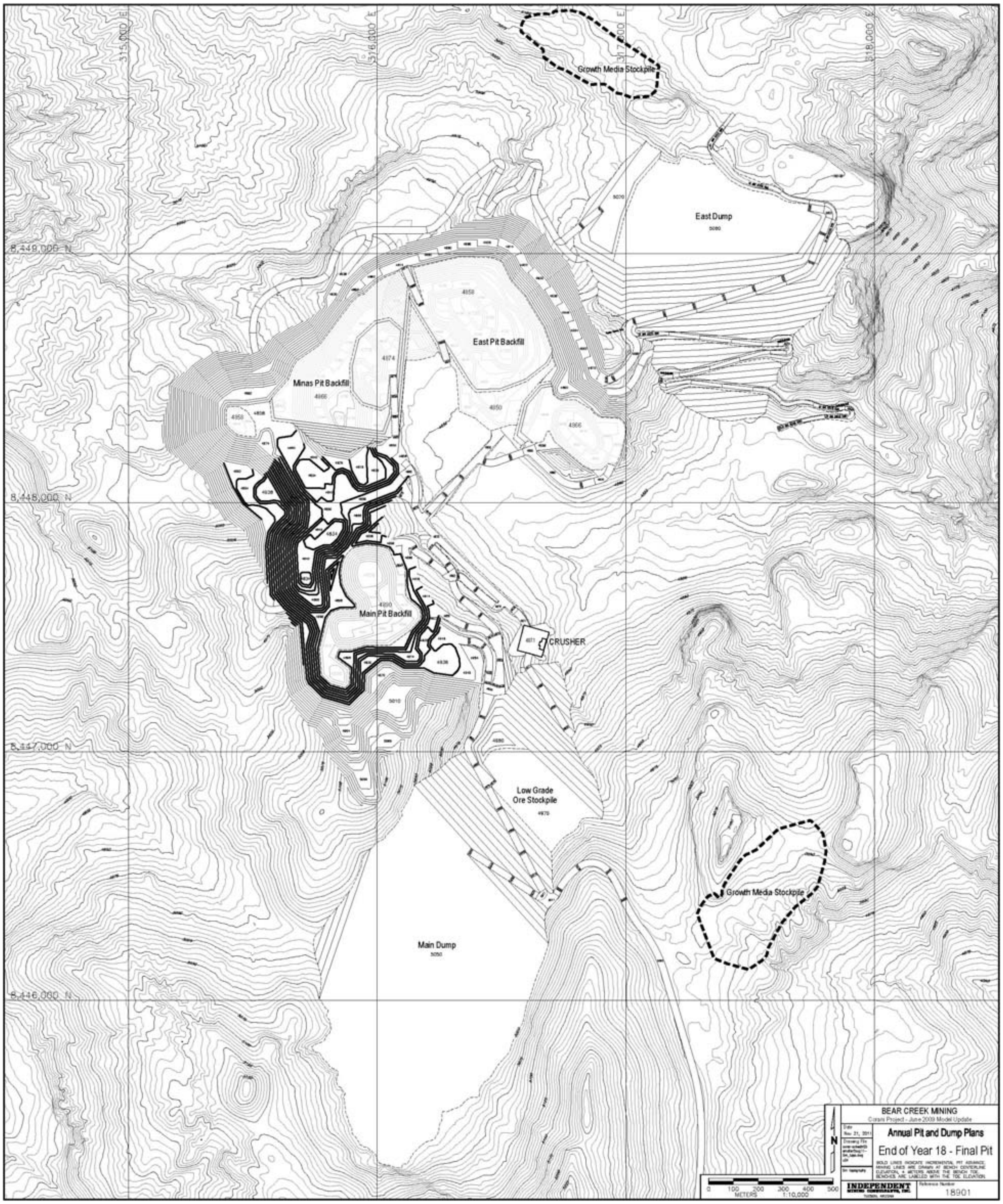


Figure 16-12: Annual Pit and Dump Plans – End of Year 18 – Final Pit

16.6 MINE OPERATIONS AND EQUIPMENT

Mine mobile equipment was selected to meet the production requirements as outlined in Table 16-1. All mine equipment within this study are standard off-the-shelf units. When actual orders are placed, particular attention will need to be given to high altitude options on all mobile equipment.

Mining is scheduled for 365 days/year and 2 shifts/day of 12 hours duration. Fifteen shifts per year are assumed to be lost due to weather delays and holidays. A 3 crew system has been used when manning the mine equipment.

Drilling will be accomplished with conventional track mounted rotary blast hole drills. Pull down force of 45,000 lbs (20,000 kg) was judged to be appropriate for the material at Corani with a bit diameter of 8 inches (20.3 cm). Drill holes will be sampled and assayed for ore control. Holes will be loaded with ANFO and blasted prior to loading.

Loading is planned to utilize a mixed fleet of front end loaders with 13.8 cu meter buckets and 15 cu meter front shovels. A wheel loader was included in the loading fleet for the advantage of having more maneuverability between mining areas. Maneuverability in the loading fleet will be important for the Corani project because the mill recovery is sensitive to the ore type and head grade and also the mine will have 3 pit areas operating simultaneously

Hauling is planned to utilize 135 tonne haul trucks for all blasted and rehandled material.

Track dozers are 410 hp units (D9 Class) and a wheel dozer is provided in the 485 hp class for clean up around shovels. Graders will have 16 ft moldboards and the water truck will hold 76,000 liters on a 90 tonne unit.

A small, track mounted hydraulic drill is provided for secondary blasting if required and for road pioneering duties. Two, 2 cubic meter backhoes are provided for general support and maintenance of drainage structures and for necessary movement of organics in the valley. Two, 40 tonne articulated haul trucks are planned to be used for moving organic material to stockpiles above the waste dumps.

Equipment productivity was calculated on a shift basis based on Corani rock and operating conditions. The productivity per shift and the tonnage requirements set the number of operating shifts needed per year to move the material. Availability and utilization were applied to determine the required number of operating units.

Haul truck productivity was based on detailed haul time simulation over measured haul profiles. Haul profiles were measured for each material type, from each pushback to each destination on a year by year basis.

Table 16-3 summarizes the mine mobile equipment fleet for the mine life.

The requirements for mine supervision, operations, and maintenance personnel were calculated based on the equipment list and mine schedule. For most of the mine life there will be 34 salaried staff for supervision, engineering, geology, training, and ore control.

Mine operations and maintenance labor increases to 175 persons in the end of year two and stays between 175 and 178 persons until labor requirements begins to decline in year 13. Maintenance personnel requirements are set to be around 70% of operations labor required.

Table 16-3: Summary of Major Mine Equipment

Mine Major Equipment Fleet On Hand

	Time Period															
Equipment Type	PPQ-6	PPQ-5	PPQ-4	PPQ-3	PPQ-2	PPQ-1	Yr1Q1	Yr1Q2	Yr1Q3	Yr1Q4	Yr2Q1	Yr2Q2	Yr2Q3	Yr2Q4	3	4
45,000 Lb Pulldown Drill		1	1	2	3	3	3	3	3	3	3	3	3	3	3	3
15 cu m Hydraulic Shovel		1	1	1	1	1	1	1	1	1	1	1	1	1	1	2
13.8 cu m Loader				1	2	2	2	2	2	2	2	2	2	2	2	2
135 mT Haul Truck		2	4	7	8	8	8	8	9	9	9	9	9	10	10	10
410 HP Track Dozer	3	3	3	3	3	3	4	4	4	4	4	4	4	4	4	4
498 HP Wheel Dozer				1	1	1	1	1	1	1	1	1	1	1	1	1
16 ft Motor Grader	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3
76,000 liter Water Truck		2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
8.6 cu m Wheel Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
40 tonne Articulated Truck	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2
Pioneer Drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
2 cu m Excavator	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2
TOTAL	9	15	17	23	26	26	29	29	30	30	31	31	31	32	32	33

	Time Period															
Equipment Type	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
45,000 Lb Pulldown Drill	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
15 cu m Hydraulic Shovel	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1
13.8 cu m Loader	2	2	2	2	2	2	2	3	3	3	3	3	3	2	1	1
135 mT Haul Truck	10	10	11	11	12	13	13	13	11	11	11	11	11	11	10	10
410 HP Track Dozer	4	4	4	4	4	4	4	4	4	4	4	3	3	3	3	3
498 HP Wheel Dozer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
16 ft Motor Grader	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2
76,000 liter Water Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
8.6 cu m Wheel Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
40 tonne Articulated Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Pioneer Drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
2 cu m Excavator	2	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1
TOTAL	33	33	34	33	34	35	35	36	34	34	34	33	33	31	29	28

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. The most significant consideration in determining the optimum location for the processing facilities is selecting a location that minimizes the need to pump tailing to the TSF and to return reclaims water to the processing plant.

17.2 PROCESS DESCRIPTION

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. These studies, including the report by Blue Coast (2011) are listed in Section 27 of this report.

Figure 17-1 is a simplified schematic of the overall process for the Project. It provides the basis for the process description that follows.

After crushing and grinding, the ore will be processed in a flotation circuit where lead will be floated first, and then zinc will be floated from the lead flotation tails. Details of the process are discussed in later sections.

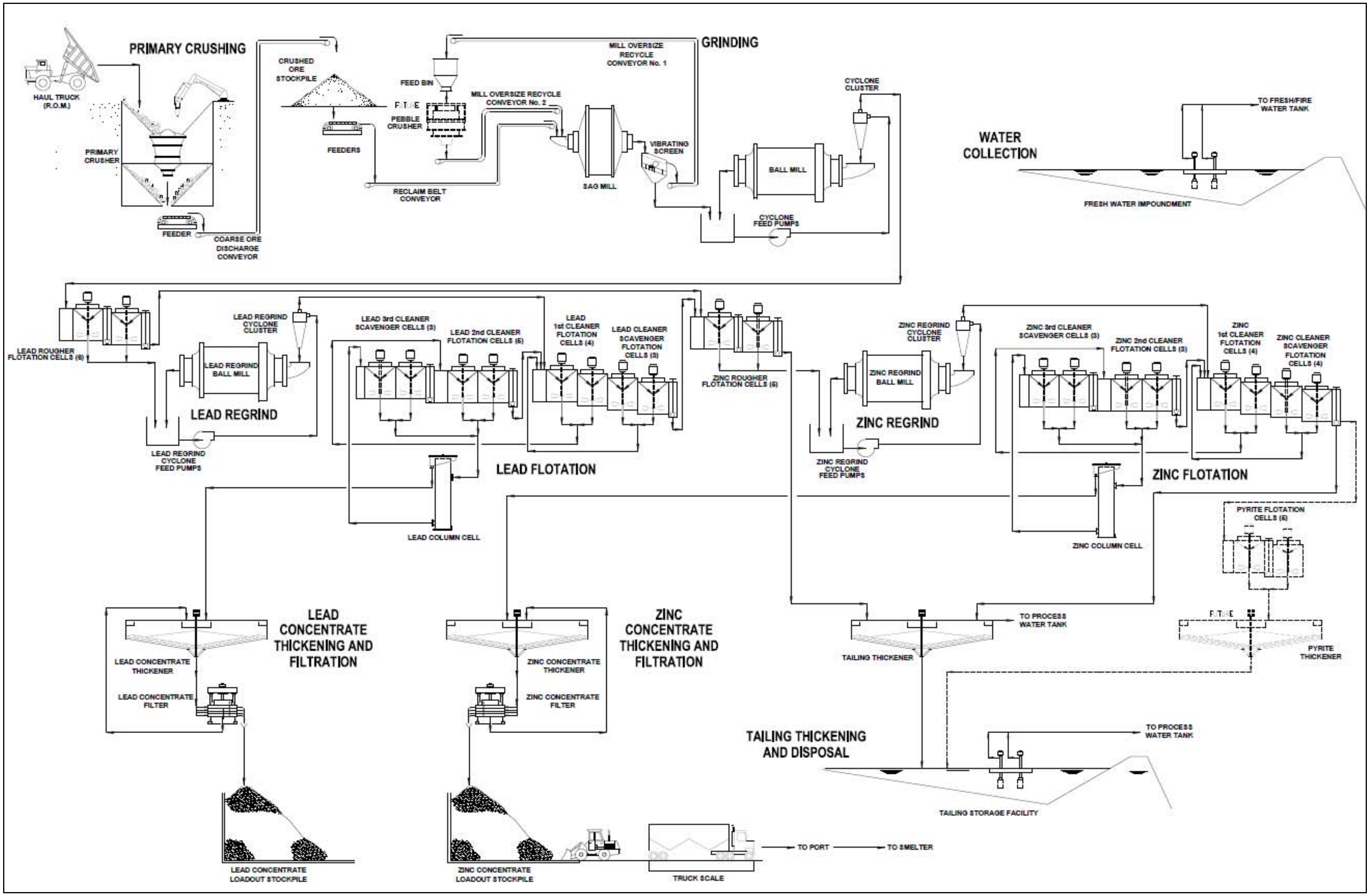


Figure 17-1: Simplified Process Flow Diagram for the Corani Project

17.2.1 Process Design Criteria

Bear Creek Mining Corporation tasked M3 Engineering 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 of 92 %, except for the crushing area, which was designed at an availability of 75%. These design availabilities are fairly common for current and recent projects at M3.

The current mine plan developed for the project was based on a 350-day calendar year. The yearly ore tonnage of 7,875,000 metric tons can be processed through the mill in 322 days, which is less than the designed available mill days of 336 days. This will leave an additional margin for the process plant to absorb increases in ore hardness through the mine life.

Figure 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 Crusher	
Feed F80, mm	1,200
Product P80, mm	150
Crushing work index (LEIT), kWh/t	4.75
SAG Mill Grinding	
Feed F80, mm	150
Product P80, microns	1,600
SAG Mill Work Index, kWh/t	5.37
Ball Mill Grinding	
Feed F80, microns	1,600
Product P80, microns	106
Ball Mill Work Index, kWh/t	13.2
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
Laboratory Third Cleaner Flotation Time, min	5
Plant Third Cleaner Scavenger Flotation time, min	10
Zinc Flotation	
Rougher Conditioning Time, min	10
Plant Rougher Flotation Time	24
Plant First Cleaner Flotation Time	8
Plant Cleaner Scavenger Flotation Time	8
Plant Second Cleaner Flotation Time, min	6
Laboratory Third Cleaner Flotation Time	3
Plant Third Cleaner Scavenger 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	8.7	62
Zinc Concentrate	5	67.5	14

These recoveries were based on the recovery equations supplied by Blue Coast and provided to M3 on August 19, 2011, and the yearly mine grades provided by IMC. The 90th percentile grade in the mine plan was used as the basis for the simulation, namely 1.17% Pb and 0.7% Zn. M3 assumed that at these grades, the feed to the mill will be predominantly sulfidic and will follow the recovery curves for the mixed sulfide ores.

17.3 CRUSHING AND CRUSHED ORE STOCKPILE

Run-of-mine (ROM) ore will be transported by haul trucks from the mine to the primary crusher, and fed to the crusher via a dump pocket with a two-truckload capacity. The primary crusher will be a 42” x 65” gyratory crusher, with an open side setting of 140 mm (5½ inches), a feed opening of 1,065 mm (42 inches). It will powered by a 375-kW motor. The crusher will discharge through a discharge bin onto the primary crusher discharge apron feeder.

The feeder will meter ore onto the stacking conveyor, which will deliver the coarse ore to the stockpile. The coarse ore stockpile will have a total capacity of 100,000 tons and a live capacity of 24,500 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 be in a closed circuit with a pebble screen, and the ball mill in a closed circuit with hydrocyclones. A future pebble crusher is planned to crush the pebble-screen oversize before returning it to the SAG mill, making the circuit a standard SABC arrangement.

The SAG feed conveyor will feed ore to the semi-autogenous grinding (SAG) mill (9.8 m diameter by 4.2 m long). The SAG mill product will discharge to a pebble wash screen into the cyclone feed pump box. This will constitute fresh feed to the ball mills mix with the discharge from the ball mill and get pumped to the primary cyclone cluster 30x26 cyclone feed pump with a 1,120 kW variable frequency drive (VFD). The cyclone underflow will be fed to the ball mill (7.32 m diameter by 13.4 m long) while the overflow will be fed to the flotation circuit. The pebbles separated by the pebble wash screen will be collected on the pebble crusher feed conveyor, which will return them to the SAG mill via the SAG feed conveyor or divert them to a pebble stockpile for further handling, as deemed appropriate. The product from the grinding circuit is targeted to have a size distribution of 80 percent finer than 106 microns.

17.5 FLOTATION

The Corani process plant will produce separate lead and zinc concentrates using a conventional froth flotation process. Lead will be floated first from the primary cyclone overflow slurry, followed by zinc from the lead flotation tails. Lead flotation will be conducted at a pH of 8 which will be achieved by the addition of soda ash 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, 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, where silver is payable at a higher rate.

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, and emulsified fuel oil 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 will have a residence of 5 minutes.

The conditioned slurry will then flow to the rougher flotation bank by gravity. Tailing from the lead rougher flotation circuit will report to the zinc flotation circuit. In the standard flow scheme, the concentrate from all the cells in the rougher bank will be sent to the regrind mill. An alternate scheme is to forward the concentrate from the first two cell to the 3rd cleaner column through the second cleaner concentrate pump box. This scheme will be implemented when the ore type and grade coming to the mill result in high-grade concentrates from the first two rougher cells. Concentrate from the rest of the cells will follow the standard flow.

Regrinding of the lead rougher concentrate will be implemented in 3-m x 6-m ball mill in a closed circuit with hydrocyclones. The rougher concentrate will be sent to the regrind pump box where it will combine with the discharge from the regrind mill. From the pump box, the slurry will be pumped to the hydrocyclones for classification. The hydrocyclone underflow will return to the regrind mill, while the overflow will flow to the lead first cleaner flotation circuit. The target particle size distribution for the reground material is 80 percent finer than 25 microns.

Three stages of cleaning will upgrade the reground lead concentrate to meet smelter specifications. In addition, a first cleaner scavenger stage will be installed to produce tailing that can be forwarded to the zinc flotation circuit with no significant loss of lead. The first lead cleaner concentrate will be transferred to the second lead cleaner flotation circuit. Tailing from the first lead cleaner circuit will be processed in the lead cleaner scavenger flotation circuit. Concentrate from the cleaner scavenger flotation circuit will be returned to the feed of the first lead cleaner flotation circuit. Tailing from the cleaner scavenger circuit will be pumped to the zinc flotation circuit.

The concentrate from the second lead cleaner flotation circuit will be pumped to the third lead cleaner flotation column. Concentrate from this column will be pumped to the lead concentrate thickener as final lead concentrate. The tailing from the second lead cleaner flotation circuit will be recycled to the first lead cleaner flotation circuit.

A third cleaner scavenger bank will process tailing from the third cleaner flotation column. Concentrate from this stage will be returned to the column while the tails will flow to the second cleaner stage. The purpose of the third cleaner scavenger stage is to reduce the circulating load around the column. In addition, the third cleaner scavenger stage was designed to have enough volume to take over the function of the column in case of column shutdowns or as called for due operator preference.

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	6	270
First Cleaner	4	80
Cleaner-Scavenger	3	53
2nd Cleaner	5	28.3
3rd Cleaner Column	1	3-m dia.
3rd Cleaner Scavenger	3	28.3

17.5.2 Zinc Flotation

The zinc flotation flow design will be a copy of the lead flotation circuit, except it will not have the flexibility to advance part of the rougher concentrate to the third cleaner column. Laboratory test results show that this flexibility will not be 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. Sodium isopropyl xanthate (SIPX), methyl isobutyl carbinol (MIBC) and milk of lime will be added to the second conditioning tank. Each conditioning tank will have a residence time of 5 minutes.

The conditioned slurry will flow by gravity into the zinc rougher flotation bank. The zinc rougher concentrate will be pumped to the zinc regrind pump box where it will combine with the discharge from the regrind ball mill. From the pump box, the slurry will be pumped to the zinc regrind cyclones.

The zinc regrind cyclone overflow will report to first zinc cleaner flotation circuit. The target particle size distribution is 80% finer than 25 microns. The cyclone underflow will be returned to the ball mill for further size reduction until the particles are reduced to the appropriate size.

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 second zinc cleaner flotation circuit. Tailing from the first zinc 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 first zinc cleaner flotation circuit. Tailing from the cleaner scavenger circuit will be pumped to the final tailing thickener.

The concentrate from the second zinc cleaner flotation circuit will be pumped to the third zinc cleaner flotation column. Concentrate from the third zinc cleaner flotation column will be pumped to the zinc concentrate thickener as final zinc concentrate. The tailing from the second zinc cleaner flotation circuit will recycled to the first zinc cleaner flotation circuit.

As in the lead circuit, a third cleaner scavenger bank will be included in the zinc circuit to process tailing from the third cleaner flotation column. Concentrate from this stage will be returned to the column while the tails will flow to the second cleaner stage. The purpose of the third cleaner scavenger stage is to reduce the circulating load around the column. In addition, the third cleaner scavenger stage was designed to have enough volume to take over the function of the column in case of column shutdowns or as called for due operator preference.

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 that will be installed in 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	219
First Cleaner	4	32
Cleaner-Scavenger	4	22
2nd Cleaner	3	13
3rd Cleaner Column	1	3-m dia.
3rd Cleaner Scavenger	3	13

17.6 FUTURE PYRITE FLOTATION

At around Year 10 of operation, BCM plans to build a facility to recover pyrite from the mill tailing in order to reduce its acid-generating potential. This will entail the addition of sulfuric acid to the zinc flotation tails to neutralize the slurry pH, sodium isopropyl xanthate and frother, as required, to float pyrite and other sulfides. Since the objective of the facility is recovery rather than concentrate grade, a one-pass rougher flotation bank or banks will suffice. The plan is to deposit the pyrite concentrate at the deepest section on the tailing storage facility where the dissolved oxygen levels will be low and where the pyrite will later be covered with low-sulfur tails.

The flotation conditions for pyrite recovery have not been optimized. BCM has plans to conduct laboratory tests in the future to establish these conditions, including target recoveries and flotation times.

The pyrite flotation facility may be built as an addition to the process plant, or near the tailing storage facility. The advantage of the second option would be the relatively simpler design to pump and pipe the pyrite concentrate. However, additional power will have to be supplied to the tailing site to run the flotation cells. Storage tanks for sulfuric acid and flotation reagents will also have to be built.

17.7 CONCENTRATE THICKENING, FILTRATION, STORAGE

Lead concentrate from the third lead cleaner flotation circuit will be dewatered in the 5 m diameter lead concentrate thickener. The thickened lead concentrate will be pumped to two recessed-plate dewatering filters that will operate in parallel. The filtered concentrate will be conveyed to the lead concentrate surge hopper. From the hopper, the lead concentrate will be loaded and transported from the Project via trucks.

Zinc concentrate from the third zinc cleaner flotation circuit will be dewatered in the 5 m diameter zinc concentrate thickener. The thickened zinc concentrate will be pumped to a recessed-plate dewatering filter. The filtered concentrate will be conveyed to the zinc concentrate surge hopper. From the hopper, the zinc concentrate will be loaded and transported from the Project via trucks.

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 reagents used in the process plant.

Table 17-5: Process Reagents and Consumption Rates

Reagent	Consumption, g/t
Na Isopropyl Xanthate (SIPX)	40
Cytec AP 404	15
Lime	3,000
Sodium Carbonate	370
Zinc Sulfate	620
Sodium Cyanide	210
Copper Sulfate	290
Methyl Isobutyl Carbinol (MIBC)	30
Sodium Sulfite	505
Na Hydroxide	10
Fuel Oil (added as an emulsion)	1
Flocculant	20
Antiscalant	5
Grinding Balls	98
Crusher and Mill Liners	1,020

17.9 WATER REQUIREMENT

A water balance for the process plant was developed for the Corani project using MetSim modeling. The Corani process plant is projected to require 363 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 510 m^3/h. This is equivalent to 0.5 m^3 of water per tonne of ore processed, which is within typical operating ranges.

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 or US$2.10 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)	Total Cost, US$
Concentrator			
Primary Crushing & Conveying	838	3,973,301	218,532
Grinding	23,437	152,587,405	8,392,307
Flotation	11,528	66,212,612	3,641,694
Concentrate Thickening/Filtration	984	5,753,610	316,449
Reagents Storage	222	1,468,600	80,773
Tailing Management & Reclaim Systems	3,348	23,958,123	1,317,697
Water Supply System	835	3,746,138	206,038
Ancillary	245	839,592	46,178
Total Connected (kW)	41,438		
Total Consumption (kW-hr)		258,539,382	14,219,666
Cost Per kW-hr			0.055

17.11 TAILING THICKENING

The tailing from the zinc rougher flotation circuit and the zinc cleaner scavenger flotation circuit will flow by gravity to the tailing thickener. The slurry will be dewatered from approximately 35 percent solids by weight in the feed to approximately 45 to 50 percent solids by weight in the thickener underflow. The thickened tailing will be pumped to the tailing storage facility (TSF). The water recovered from the thickener overflow and the TSF will be pumped to the process water tank for use in the process.

17.12 CONTROL PHILOSOPHY

17.12.1 Process Control Philosophy

The plant will include standard process control systems that can be operated from a central control room. Two x-ray analyzers and two particle size monitors will provide important information about 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.

17.12.2 Control Systems

A crusher control room, located in the primary crusher area at the mine will be the operating and control centre 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.

17.13 PLANT SERVICES

17.13.1 Mobile Equipment

Table 17-7 below lists the mobile equipment that is provided for in the project capital cost estimate. The mine capital estimate includes a 60 t crane that will be shared between the process and mine department for maintenance activities.

Table 17-7: Mobile Equipment List

DESCRIPTION	QTY	DUTY
Wheeled Loader - used CAT 988F	1	Crushed ore stockpile
12 m^3 End Dump Truck	1	Utility
2 t Flatbed Truck	1	Haul Warehouse Supplies
4 t Fork Lift	1	Distribute Warehouse Supplies
1,5 t Fork Lift	1	Distribute Warehouse Supplies
25 t Boom Truck	1	Maintenance
Welders Truck	1	Maintenance
Bob Cat	2	Spills Clean-up
Water Truck	1	Road Maintenance

17.13.2 Assay / Metallurgical Laboratories

A 15.3-meter wide by 42.7-meter long analytical laboratory building has been provided for in the capital cost estimate. Provision has been made for facilities that include sample receiving, sample drying, sample preparation, metallurgical lab, wet lab, and fire assay for mine and process plant samples on a daily basis.

18 PROJECT INFRASTRUCTURE

18.1 INFRASTRUCTURE

18.1.1 Site Access

Access will be via a new 63 km road to be built generally over flat and gently sloping topography resulting in low construction costs. Design details remain unchanged from Vector (2009).

The new Mine Access Road will connect the town of Macusani to the process plant, and for much of its length follows an existing access track. The Interoceanic Highway (a two-lane, paved highway) currently connects Macusani to the Peruvian port cities of Matarani and Ilo.

Traffic studies will need to be undertaken in future studies, together with establishing an appropriate connection to the Interoceanic Highway either through or around Macusani.

18.1.2 Site Layout

The layout of the site has been developed in two areas:

1. The layout of the area that includes the mine ancillary facilities; and
2. The area where the camp is located.

These areas are at different elevations. The mine facilities are at an elevation of 4850 m. The camp facilities are at an elevation of 4400 m. These two layouts will be described in separate sections in the following pages.

Figure 18-1 shows the general arrangement of the site.

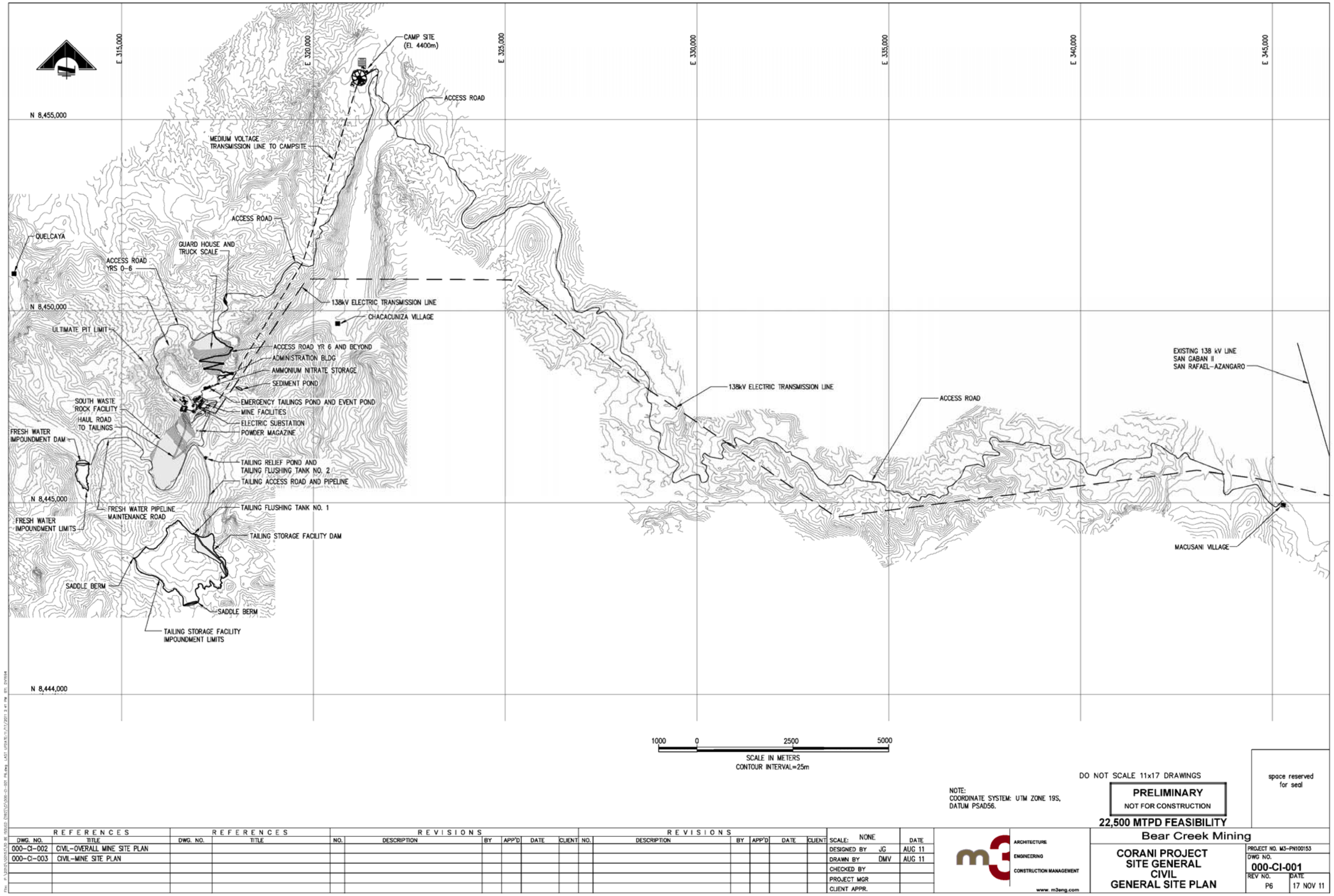


Figure 18-1: General Site Plan 000-CI-001

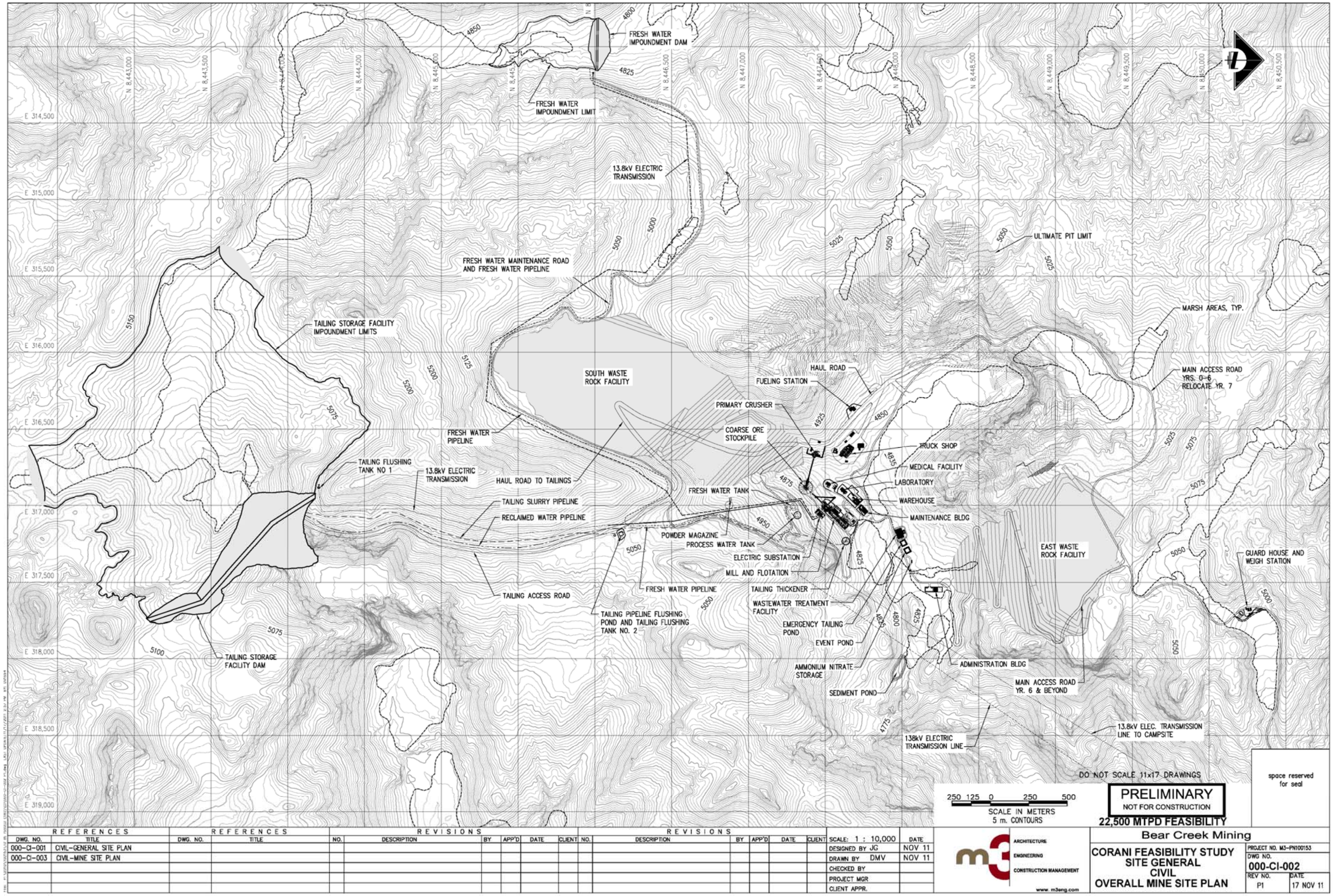


Figure 18-2: Overall Mine Site Plan 000-CI-002

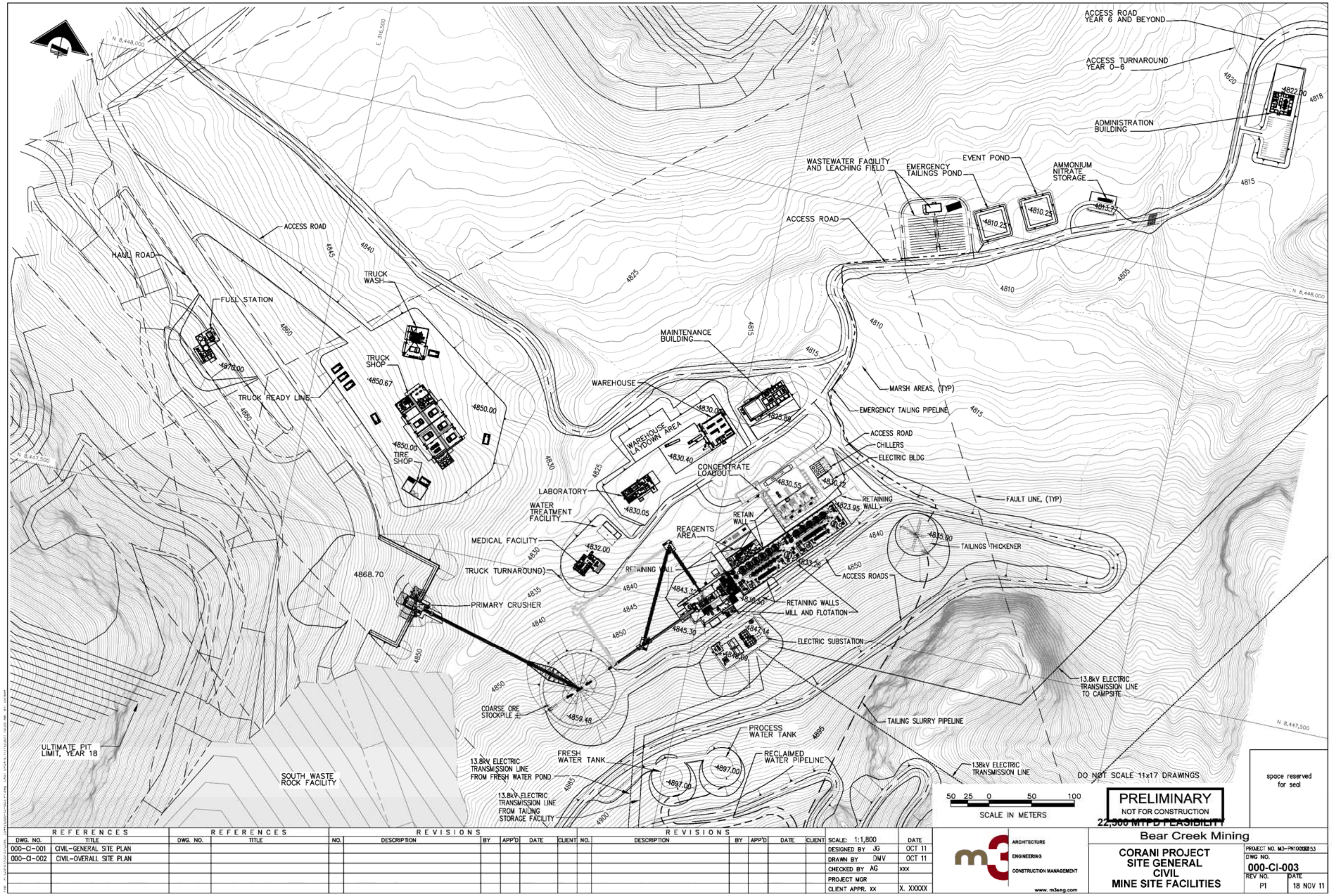


Figure 18-3: Mine Site Facilities Drawing 000-CI-003

18.1.3 Site Roads

M3 has developed site roads from the Pit to the primary crusher, access from the plant to the tailing Storage Facility and Fresh Water Impoundment and access to the various components of the processing plant.

M3 reviewed the access road alignment from Macusani to the camp site. From the camp site two roads were developed to access the plant in two phases. In Phase 1, a road alignment will be built from year 0 to year 6 to the east around the pit which then enters the pit in the NE-SW direction. For Phase 2, after year 6, at this junction, a road will be built across the East waste rock facility from north to south.

These preliminary alignments are shown on Figure 18-1, Figure 18-2, and Figure 18-3.

18.1.4 Site Buildings

The process facility and ancillary buildings include the following (see Figure 18-3):

- Primary Crushing;
- Grinding and Flotation;
- Concentrate Handling;
- Administration Building;
- Medical unit and ambulance station;
- Analytical Laboratory;
- Plant Maintenance; and
- Tailing Storage Facility (TSF).

The mine infrastructure and ancillary buildings include the following:

- Explosive Storages (one building for the powder magazine and one for ammonium nitrate storage);
- Warehouse;
- Yard Storage;
- Truck Wash;
- Truck Fuel Storage and Fueling Station; and
- Tire Change Station.

18.1.4.1 Mine Ancillary Facilities

Mine service facilities will be located on a graded area adjacent to the primary crusher and will house mine administration personnel and provide service and support to the mining operation. 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.

The building will house offices, men's and women's dry, lunch room, warehouse, tool cribs, electrical and mechanical rooms, lube room, welding/repair area, two large equipment repair bays and two small equipment repair bays. An outside secure storage area, a tire changing apron, and a truck wash bay will also be provided.

Other mine facilities include ready-line and fuel storage and dispensing facilities, a mine control tower and explosive storage facility.

18.1.4.2 Process Plant and Administration 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.

In closer proximity to the crusher there will be a truck shop, a truck wash a tire shop and a fuel station.

Security

The main mine gate house and security office will be located nearby the administration building, at the entrance to the camp site at the property entrance. This small building will house security offices, toilet and 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.

Primary Crusher

The primary crusher structure will be reinforced concrete and structural steel construction and the concrete dump pocket is designed to receive two dump-trucks at the same time. A hydraulic rock breaker for crushing oversize boulders will be provided.

Building area lighting and dust control will be provided. Internal structural steel platforms that provide maintenance access will be covered with metal grating, and kick-plates and handrails are provided for safety.

Crushed Ore Storage

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. Tunnel lighting, ventilation, and an escape tunnel will be provided.

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.

Grinding Building

The Grinding Building will be a pre-engineered structure with insulated roofing and siding. Internal structural steel platforms that provide maintenance access are supported independent of the main building structure. Maintenance platforms will be covered with metal grating, and kick-plates and handrails are provided.

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 and monorails are provided for maintenance of the grinding equipment. The main building areas will be heated and ventilated and controls rooms and offices within the structure will be air conditioned and heated with high-bay lighting provided through-out.

Flotation Area Building

The flotation circuit has a building with roofing and siding. 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 provided. Maintenance on the flotation circuit will be facilitated by two cranes to serve the lead and zinc sections separately. Pole mounted high-bay area lighting will be provided to illuminate the area.

Concentrate Dewatering and Load-out Building

Product dewatering and load-out facilities will be located within a pre-engineered structure with insulated roofing and siding. Internal structural steel platforms that provide maintenance access are supported independent of the main building structure. Maintenance platforms will be covered with metal grating, and kick-plates and handrails are provided. The building will be ventilated and lighting provided through-out.

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.

Tailing Storage Facility (TSF)

The TSF is located to the south of the plant at a distance of approximately 4.2 km. The TSF includes a process water tank, two flushing tanks for cleaning and emergency events, a tailing relief pond, an emergency tailing pond, a pipeline and a system of pumps to transport the tailing to the TSF. The reclaimed water from the barge pumps will flow by gravity to the process water tank. A 13.8 kV power line will be servicing the booster station.

In the final half of the mine life a pyrite flotation circuit will be added to separate the pyrite from the zinc flotation tailing.

Plant Maintenance Building

The maintenance workshop building will be a pre-engineered structure with insulated roofing and siding and a reinforced concrete mat foundation. Offices, restrooms, work areas and a warehouse area are provided within.

Warehouse Building

An enclosed storage area is provided, a dock for truck unloading and an external fenced warehouse yard is also provided adjacent to the building. Offices in the building will be heated/air conditioned and workshop areas will be ventilated.

Reagent Building

A reagent mixing and storage facility will be located adjacent to the concentrator area to receive and store reagent supplies. A pre-engineered reagent mixing building will be provided to prepare and supply reagents to the process. Material and reagents storage areas are provided, and vehicle parking areas are also provided.

Administration Buildings

There are two administration buildings, one that is located close to the concentrator building that will house facilities such as cafeteria, kitchen, men's dry, toilets and showers, women's dry, toilet, offices, and training room. 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.

A second administration building, located close to camp site and to the main property entrance and access road, will house facilities such as senior personnel offices, records and archives, safety and training rooms, accounting, engineering offices, conference rooms, toilets and 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.

Assay / Metallurgical Laboratory

The cost of providing an analytical laboratory is included in the capital cost estimate. The 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 and women's toilet and locker facilities, lunch room, loading dock and various offices. The building will be heated/air conditioned, and fume extraction and dust collection equipment is provided.

Medical Facility

The main medical facility is centrally located in relation to the main ancillary facilities in the mine site to attend accidents and injuries closer to the workplace.

A medical unit, ambulance station and mine rescue facilities are provided to handle major medical requirements. The infirmary will be fully equipped in order to attend to such injuries and illness. More serious injuries will be transferred via ambulance to Macusani where appropriate facilities and services are provided.

18.1.4.3 Camp

The camp site is located at an elevation of approximately 4400 m and at a distance of about 10 km to the northeast of the mine facilities.

The camp coordinates are:

Northing 8456222.15 and Easting 321428.83 under UTM, Zone 19S, Datum PSAD56.

The camp, see drawing 950-GA-001, includes the following buildings:

- Cafeteria and kitchen
- Cafeteria Dining
- Laundry
- Recreation
- Store
- Camp Administration
- Camp Sewage Treatment
- Camp Water Treatment
- Camp Maintenance
- 49 man Dormitories
- 24 man Dormitories
- 12 man Dormitories
- Camp Guard House

In this layout, areas have been set aside where contractors can build their housing for their respective workers.

In addition, areas have been set aside for open recreation such as Fulbito (a popular sport in Peru) fields/courts.

18.1.5 Water Supply and Distribution

As a consequence of the various optimization steps taken during 2011 in defining the current project components and GA, the need for a reliable source of make-up water for project operations was identified. The analysis of the project-wide water balance is ongoing at this time.

However, a sufficient level of analysis has been completed to date to support the need for the fresh water dam. Under average climatic conditions, the potential exists that sufficient make-up water can be obtained from the collection of natural runoff from the areas of disturbance related to the project. However, seasonally, this water supply could be limited and, under dryer than average climatic conditions, insufficient water could be available to maintain operations at design levels. Although the make-up water dam was not identified as required in the 2009 Technical Report, such a facility has been incorporated in the current project design. Figure 18-4 and Figure 18-5 show the proposed layout and sections of the facility.

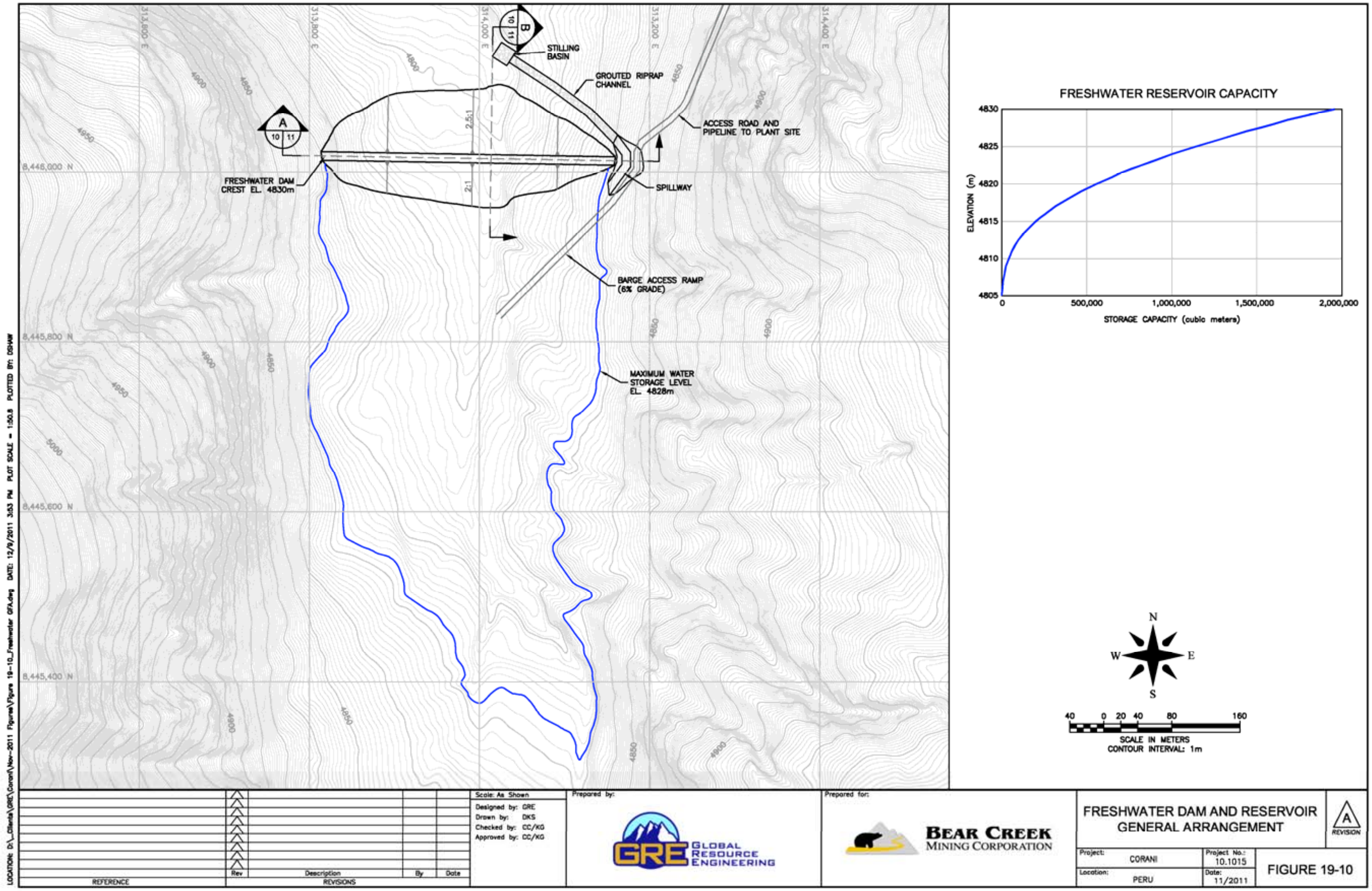


Figure 18-4: Freshwater Dam and Reservoir General Arrangement

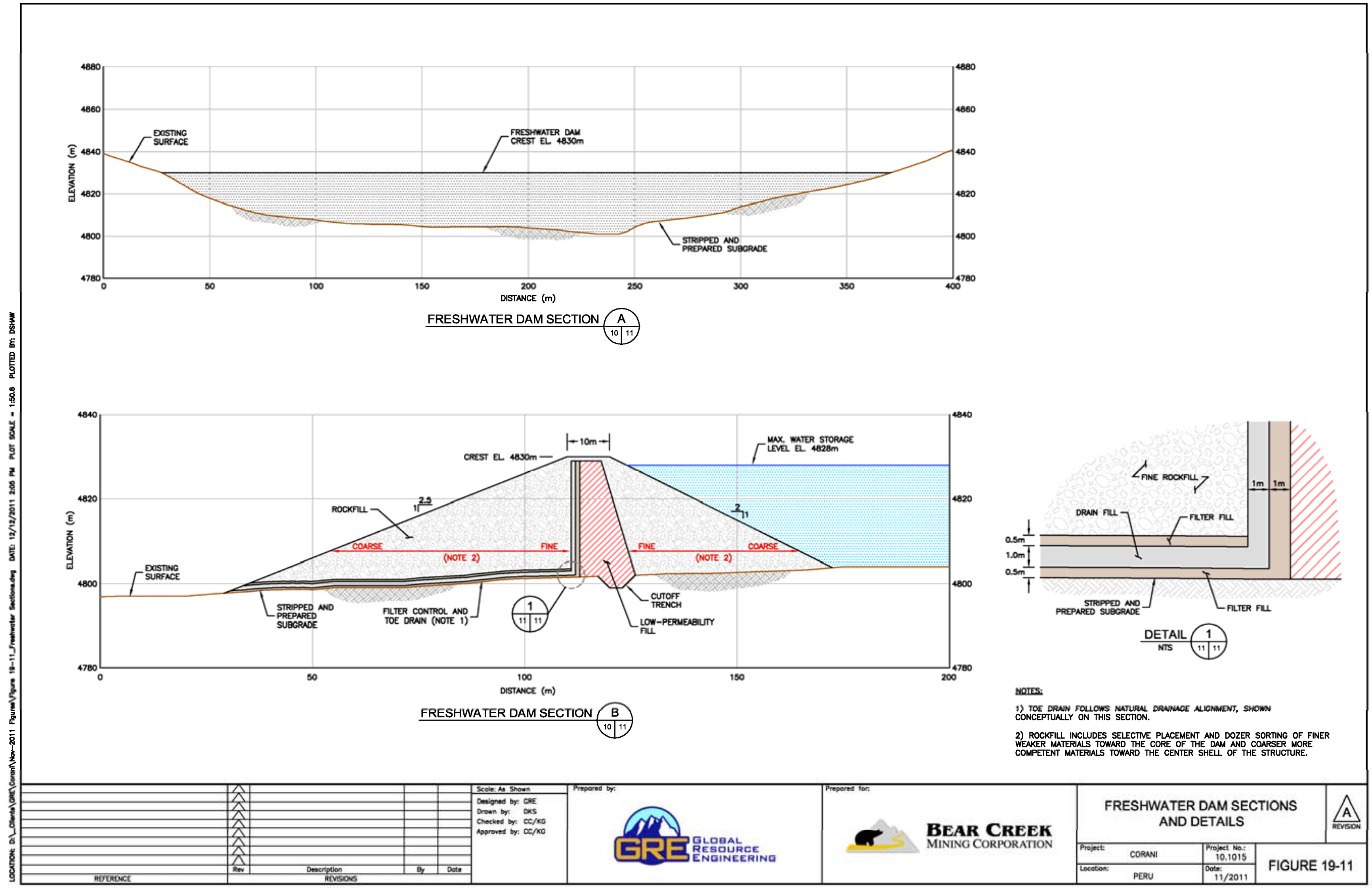


Figure 18-5: Freshwater Dam Sections and Details

The fresh water dam will be located in the upper catchment of the Huaynahuayco drainage. In this portion of the drainage, the volume of water stored will increase in the wet season and decrease in the subsequent dry season as water is withdrawn to meet process needs. A similar zoned dam to the TSF main embankment consisting of a low permeability core and compacted rockfill shells would be constructed across the valley with an impoundment height of approximately 30 meters. The dam will store up to 2 million m^3 of water, with allowances for freeboard and discharge of excess or storm flows through an engineered spillway.

Site investigation of the main embankment area of the dam and of the reservoir area is in progress. However, no fatal flaws to the selected site, construction and operation of the facility have been identified at this time. At closure, the overall capacity of the dam would be reduced with the construction of a post-closure spillway located in competent bedrock abutment materials. The resulting structure represents a potential post-closure resource to downstream communities in the form of fresh water supply and as a source of irrigation water. The transfer of custodianship for the dam following closure to the local communities is currently considered in the preliminary Closure Plan.

In addition to the fresh water dam, water supply is available from the sediment pond that will be located downstream of the mine pit, plant, and waste rock facilities. The sediment pond is primarily intended to reduce suspended solids in the runoff from mine facilities that are not captured by first-line best management practice (BMP) measures. Additionally, the sediment pond will serve as a water supply for plant operations and for dust suppression needs. Water can be supplied from the sediment pond at a reduced cost relative to the fresh water dam due to its close proximity to the mine and plant facilities. The pond facilitates environmental management by allowing contact water to be recycled to the process circuit. The sediment pond will have limited water storage capacity for dry season operations and is therefore intended to supplement rather than replace the fresh water dam.

A fresh-water pipeline will deliver the water to the project Raw Water/Fire Water Tank, located at the Project site at an elevation of approximately 4,900 meters. Any water in excess of the plant raw water requirements would be directed to the Process Water Tank located adjacent to the process facilities.

Raw water for plant use would be drawn from the upper of two nozzles located at an elevation high enough on the tank to ensure an adequate reserve for a fire water supply, and distributed by gravity to end users. From the lower nozzle, a fire water loop would supply the requirements to the processing facilities. A volume of the fresh water will be delivered to a water treatment plant where it will be treated and distributed to supply potable water to eye-wash stations and drinking fountains throughout the facilities.

The water supply for the camp site will come from the Chacaconiza river. A storage tank will be placed to deliver water to a second water treatment plant located in the camp site.

Water for camp use will be drawn from the upper of two nozzles located at an elevation high enough on the tank to be treated and distributed by gravity to the various camp facilities: kitchen, laundry, recreation center, and dorms. From the lower nozzle, a fire water loop would supply the requirements for fire suppression.

18.1.6 Power Supply and Power Distribution

The 2006 high voltage power supply alternatives study for the Project determined that the project requires the construction of a new power substation (Macusani substation), which will connect with Power Transmission Line L-1013 (San Gabán II – San Rafael – Azángaro). A new 138 kV power transmission line will need to be built to connect the Macusani substation to a receiving substation located near the Project's grinding building.

The proposed alignment for the 138 kV line is indicated on Figure 18-1 and details on the power supply infrastructure are included in PEPSA's report (2009). The transmission line route was selected based on using the route already provided by the Project's Mine Access Road. The existing facilities and projected facilities are at an altitude of approximately 5000 masl.

A 13.2 kV Transmission Line with a length of approximately 10 km will connect the mine site to the camp site.

The proposed configuration consists of:

- The new Macusani Substation,
- The Transmission Line, and
- The new Corani Substation.

18.1.6.1 New Macusani 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.

18.1.6.2 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.

18.1.6.3 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.

In addition, three distribution lines from the New Corani Substation will be connected to the tailing storage facility at a distance of 4.2 km to the south, the fresh water impoundment dam at a distance of 4.0 km to the southwest, and to the camp site at a distance of 10 km to the northeast.

18.1.7 Fire Protection

A fresh-water pipeline will deliver 40 m^3/h (545 m^3/hr) of fresh water to the project Raw Water/Fire Water Tank, located at the project site at an elevation of approximately 5,000 meters (4897 meters), and to the Crushing station and Truck-shop Raw Water/Fire Water Tank, also located at a higher elevation to ensure sufficient head to pressurize the fire water loops.

Raw 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 administrative services building area, and the laboratory.

Individual hand-held fire extinguishers will also be located throughout the offices and work areas.

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.

18.1.8 Sanitary Sewage

A sewage treatment plant will treat sewage from the concentrator facilities, the administrative services building, the laboratory, warehouse, plant maintenance building, truck shop, truck wash facilities.

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 packaged sewage treatment plant, and the treated water will be transferred to an adjacent leach field.

18.1.9 Communications

The project off-site telecommunications will be served by a satellite telecommunications system that will link to a data center in Lima. From this center, telephone and data communications including voice, data and internet communications will be provided for the mine site and Macusani.

18.1.10 Communications System

Optical fiber is considered as a primary way and the power line carrier as backing.

The communications system, after passing through a satellite transceiver, 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 the outside world.

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.

18.2 RE-OPTIMIZED INFRASTRUCTURE COMPONENTS

18.2.1 Introduction

Since the previous 43-101 Technical Report for the Corani Project, issued on October 14th 2009, Bear Creek Mining Company (BCM) has performed an optimization study of the project. The optimization study has included:

- Additional exploration of the mineralized zones within the Corani Project area;

- Extensive metallurgical studies focused on confirming and improving target mineral recoveries;
- An updated mineralogical database and mine model;
- Improved capital and operating cost estimation;
- Optimization of the mine plan to define the mineable reserves based upon the model;
- Geotechnical site investigations and waste characterization studies;
- Detailed process design;
- Project-wide water balance studies; and
- Engineering and design targeted to advance the project to feasibility study levels.

Although a number of the above items are still in progress, the project currently has a well-defined approach to project development and operations. Several modifications to the project have been performed based upon the increased level of knowledge and better definition of project parameters. In this section of the Report, a description of the current components and arrangement of facilities is presented. Several of these project components are described in more detail in other sections of this report and a general description of the relevant aspects of the project components is given herein. The following is aimed at providing an understanding of why the facilities are in their current location and why the changes were made. The following components of the project are described:

- Rescheduled Mine Plan incorporating the ability to partially backfill the open pits;
- Reconfigured waste rock management facilities (WRF);
- Reconfigured low grade ore stockpile;
- The project-wide surface water management plan;
- The relocation and resizing of the tailing storage facility (TSF); and
- The inclusion of a fresh water make-up dam to provide the required make-up for the project.

The fieldwork portion of the feasibility study is scheduled for completion in February 2012, with ongoing laboratory work and engineering studies targeted for completion in the second quarter of 2012. It is important to recognize that detailed engineering is in progress at the time of writing this report. However, the work completed to date has not revealed any fatal flaws and supports the technical feasibility of the project.

18.2.2 Optimized Project Components

18.2.2.1 General

The general arrangement (GA) for the project in the October 2009 Technical Report was shown on Drawing 100-02 of that report. In large part the selection of locations for and configuration of the various project components (with the exception of the pit area) was made with limited site knowledge of both the geotechnical conditions influencing component development and operations. In addition, limited depth of understanding existed at the time of that report related to mine waste management requirements for the pit areas, waste rock, process tailing and low grade ore stockpiles. Since the October 2009 report, extensive work has been performed to characterize the site conditions and to better define the geochemical and physical nature of the site materials. Studies have also been performed to develop baseline conditions for the project area for climate, water quality and the potential earthquake loading required to be considered in project design.

Following completion of the October 2009 report several technical issues related to the previously proposed GA and the ability to construct and effectively operate the facilities were identified. The preliminary studies on the potential for oxidation of the sulfide-bearing materials in the waste rock, exposed pit walls and in the process tailing indicted that they would have a significant influence upon project design and operational procedures. In addition, the data from the baseline study on surface water quality suggested that an increased level of environmental management would be necessary to meet International and Peruvian requirements. Concurrent with these studies, advancement of the mine plan with the incorporation of the supplemental exploration results increased the projected tonnages of the ore contained within the open pits, necessitating a re-evaluation of the suitability of the conceptual facilities to accommodate the increased tonnages of tailing and waste rock. An increased sensitivity to the influence of local attitudes and politics of communities surrounding the project area has also been recognized and resulted in a re-evaluation of portions of the GA and, particularly, the location of the TSF.

In 2011, BCM elected to re-evaluate the GA based upon the supplemental data obtained during 2010. This included the identification of alternative sites for the TSF, revised approaches to defining the mine plan and associated waste rock management, and the locations of process-related facilities and infrastructure. The following describes the optimization of the conceptual design based on the results of work completed to date during 2011. The optimized GA is illustrated in Figure 18-6.

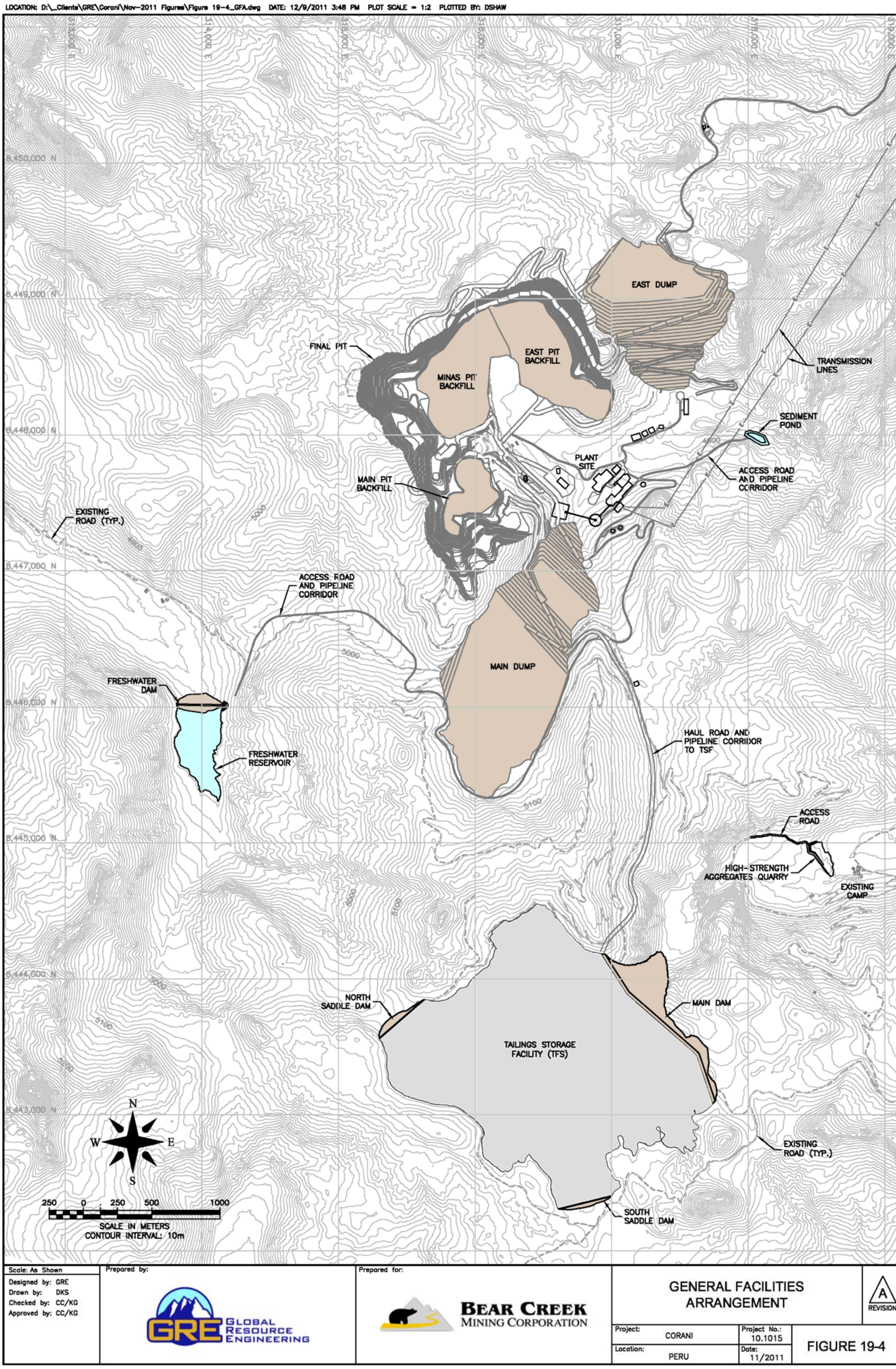


Figure 18-6: General Arrangement

18.2.2.2 Mine Plan

The Mine Plan, described in detail in other sections of this report, was updated to incorporate supplemental exploration data, a more detailed assessment of capital and operating costs and the results of the metallurgical studies performed during 2010. The Mine Plan also incorporated an additional objective of allowing mining of 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 the Main Corani Pit is also incorporated in the Mine Plan, leaving only minor portions of the Corani Minas Pit not backfilled at the end of active mining. Final backfilling of the Corani Minas Pit 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 would consist of the rock formations identified as post-mineral tuff (PMT) while the remaining PAG material would consist of all other waste rock types. This production schedule was incorporated into the optimized design of the WRF.

18.2.2.3 Waste Rock Management Facilities

With the modification of the Mine Plan to facilitate pit backfilling, the design of the WRF was optimized to reduce the size and number of external waste rock dump facilities previously proposed. The WRF design was optimized to avoid the construction of a North waste rock dump, to reduce the size of the East waste rock dump and to increase the capacity of the Main waste rock dump. 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 Este Dump and a combination of PAG and NAG waste would be scheduled for managed placement within the Main Dump.

Optimization of the waste rock facilities so that the North Dump is no long required alleviates the potential of management issues in developing the dump related to the footprint of this dump being primarily over bofedal materials. 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 in construction of the TSF. 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 and PAG 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.

18.2.2.4 Low-Grade Ore Stockpile

Although not specifically described in the 2009 Technical Report, mining and stockpiling of low-grade ore will take place early in the mine life. This material would be processed in the last 2 to 3 years of plant operation after active mining has been completed. Several options for stockpiling the low-grade material were identified in iterations of the Mine Plan since 2009. The low-grade ore is considered a PAG material and, with the exposure of these materials for a period of up to 18 years, consolidation of the stockpiles was incorporated into the current project design. In addition, the location for the stockpiles was selected to allow collection of any contact water for potential management during operations. As described further in the description of the surface water management plan, seepage from this and other sources of water from the project's areas of disturbance will be utilized as make-up water for the process circuit. The location for the low-grade ore stockpile is in the area of the northern toe of the Main waste rock dump, directly above the location of the processing (mill) facilities.

18.2.2.5 Surface Water Management Plan

In order to accommodate the site conditions that will be created during project development, operation and reclamation and closure, a project-wide surface water management plan (SWMP) was developed. The SWMP also considers the operating water balance for the TSF which indicates the need for a make-up water supply during at least portions of the year and over the life of mine. This make-up water would be derived from dewatering of the pit areas, the collection of runoff from the Este and Main waste rock dump areas and from fresh water runoff collected in the Fresh Water Dam, described further below.

The SWMP includes the development of a collection ditch and sediment pond system, as shown on Figure 18-7. The sediment pond that will be located downstream of the mine pit, plant, and waste rock facilities. The sediment pond is primarily intended to reduce suspended solids in the runoff from mine facilities that are not captured by first-line best management practice (BMP) measures. Additionally, the sediment pond will serve as a water supply for plant operations and for dust suppression needs. Water can be supplied from the sediment pond at a reduced cost relative to the fresh water dam due to its close proximity to the mine and plant facilities. The pond facilitates environmental management by allowing contact water to be recycled to the process circuit. The sediment pond will have limited water storage capacity for dry season operations and is therefore intended to supplement rather than replace the fresh water dam.

During the reclamation and closure phase of the project the system will be converted for water treatment. At closure, portions of the processing circuit will be converted to a water treatment plant which will be operated for a period of time following closure when natural runoff will be required to be released from the project area. During operations, all natural runoff not collected within the SWMP areas, and any post-closure treated water will be released into the valley downstream of the project area.

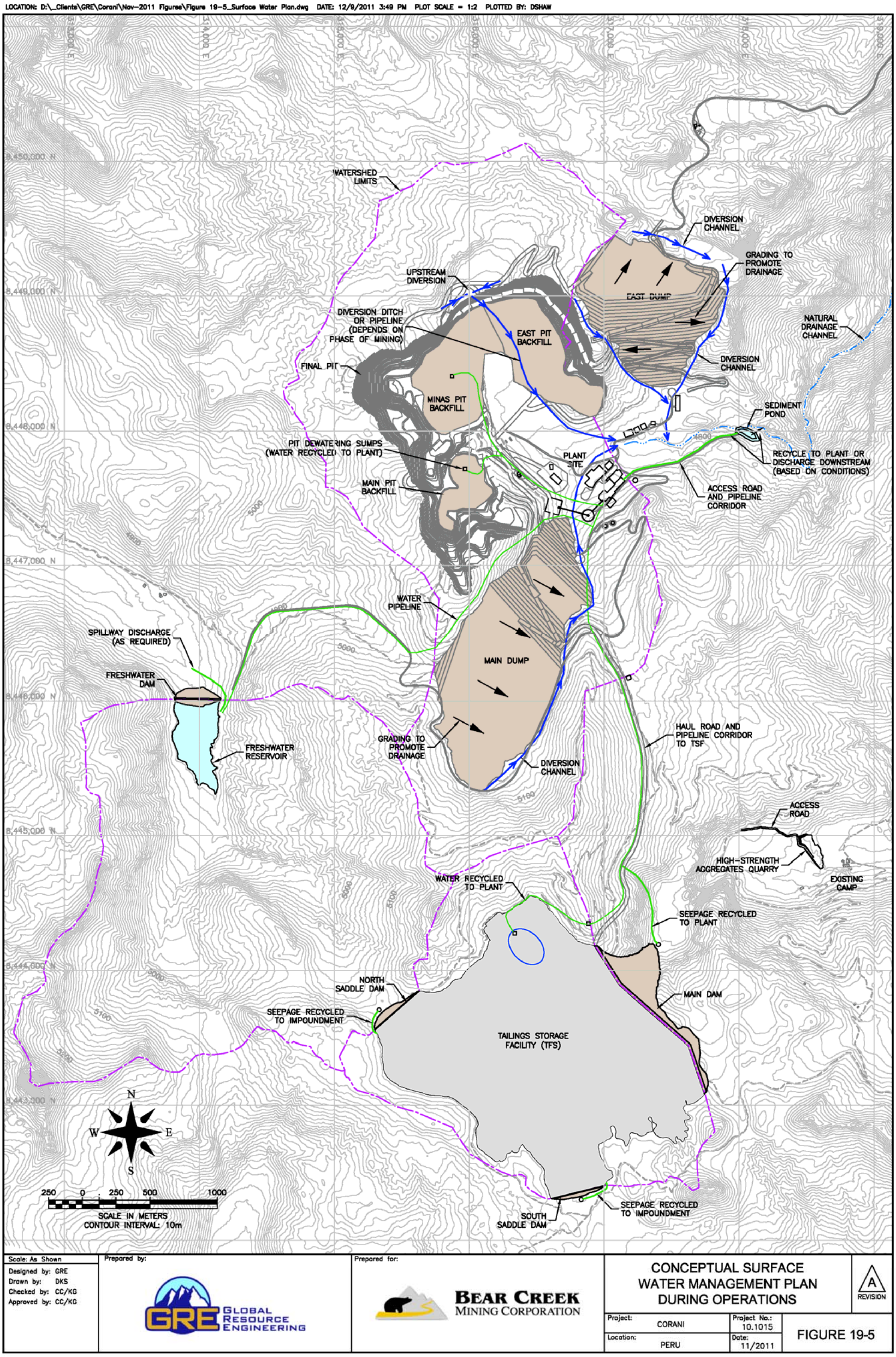


Figure 18-7: Conceptual Surface Water Management Plan During Operations

18.2.2.6 Tailing Storage Facility

In the 2009 Technical Report and in work performed during 2010, the TSF was proposed to be located in the Quelcaya valley. This drainage basin comprises part of a separate catchment area from the remainder of the project. Baseline studies in this drainage indicate surface water quality significantly different from that directly within the project area. Natural stream flow in the Quelcaya valley is only minimally impacted by the naturally occurring ARD found in the other drainage basins in the project area.

Within the footprint of the previously proposed dam and tailing impoundment area, the stream runs through deposits of alluvial and glacial materials in a classic glaciated "U" shaped valley. The catchment area for the TSF is of significant size and the previous design incorporated an extensive diversion system to route natural runoff around the TSF. During the development of the design of this facility, technical issues were identified in regards to the construction of the diversion system and operation of the TSF in this location.

Coupled with the revision of the ore reserve as a result of the updating of the Mine Plan and BCM's desire to consolidate the project facilities within a single catchment area, an alternative site was identified to the south of the main project area. This site lies on the strike of the mineralized trend associated with the Corani ore bodies. Water quality in this drainage has been degraded by the presence of exposed mineralization on the valley walls and the presence of historic mine workings in the vicinity of the proposed main tailing embankment area. The design of the South TSF has been advanced to allow the storage of up to 125 percent of the current ore reserve. With the reduced catchment area, the South TSF does not require diversion of runoff from upstream catchments and will operate with a negative water balance during the operating life of the facility. The South TSF has capacity to store the planned 157 million tonnes of ore production, including allowances for tailing beach slope, process water storage, and freeboard.

Geotechnical investigations of the main embankment dam area and of the impoundment area indicate that bedrock conditions exhibit suitable foundation properties for embankment construction and a low permeability resulting in the low potential of seepage from the basin. In addition, the hydrogeologic characterization of the basin indicates that a groundwater condition around the perimeter of the impoundment exists such that groundwater flow gradients are into the impoundment, further alleviating the potential of seepage.

Construction of the main TSF embankment will incorporate the use of a low permeability core material and compacted rock fill shells. The low permeability fill will be derived from weathered portions of the main project facilities and from the footprints of the WRFs. Inert NAG rockfill will be developed from local quarry sources and from PMT waste rock from the mine pit. This dam design is recognized as the most suitable for tolerating significant earthquake loading conditions required for facility design in the project area. Figure 18-8 to Figure 18-11 show the proposed layout and sections of the TSF.

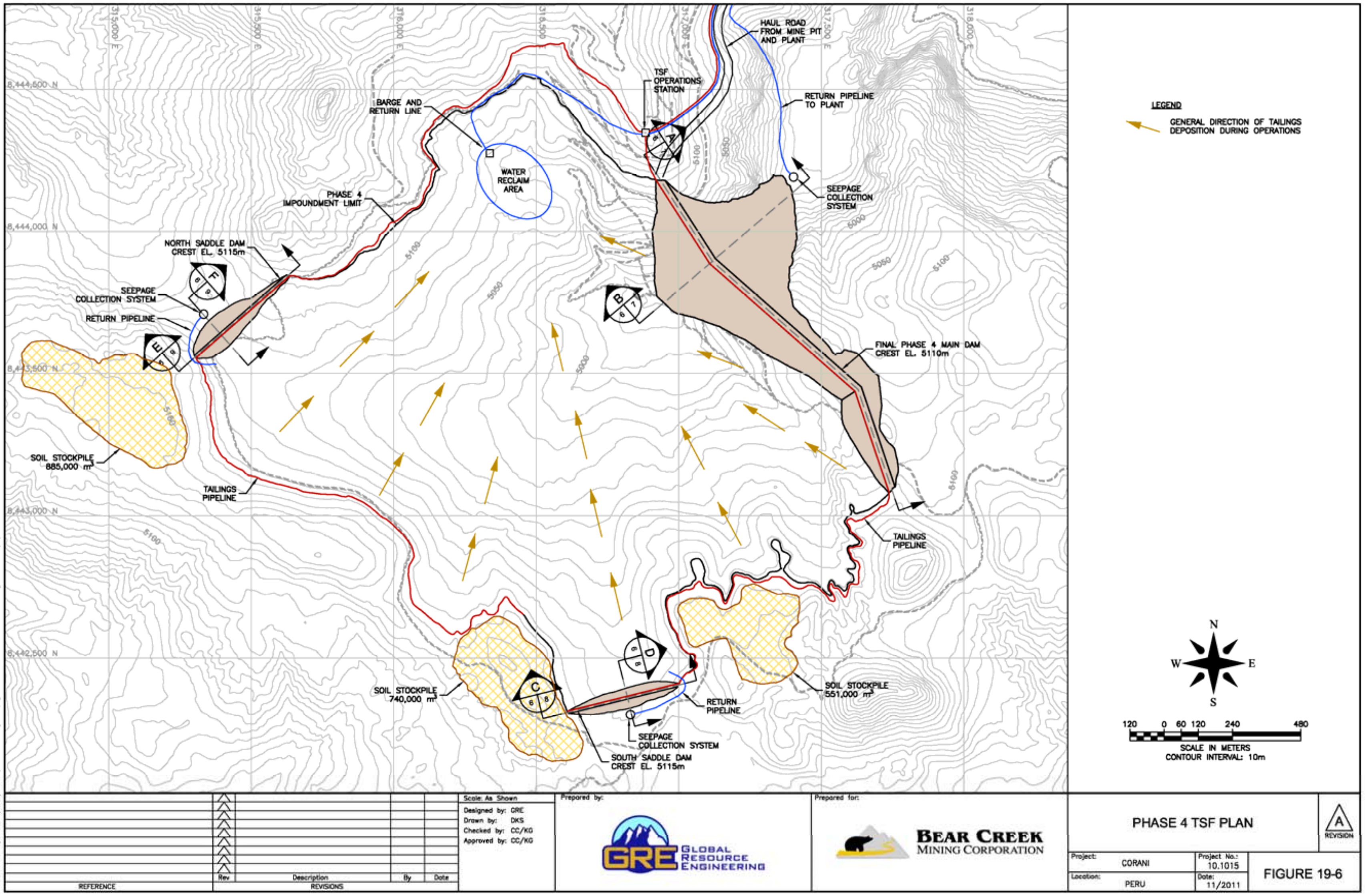


Figure 18-8: Phase 4 TSF Plan

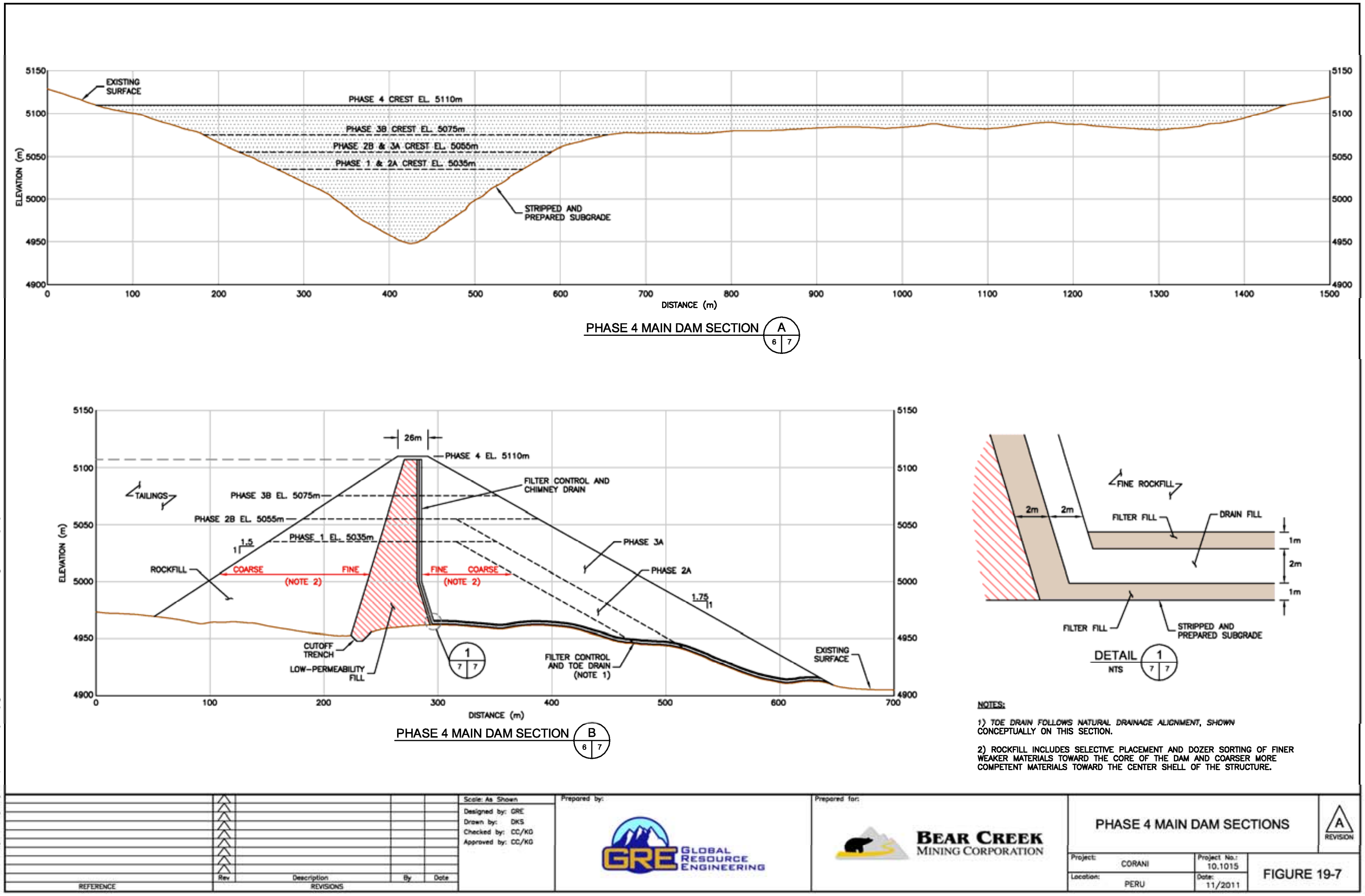


Figure 18-9: Phase 4 Main Dam Sections

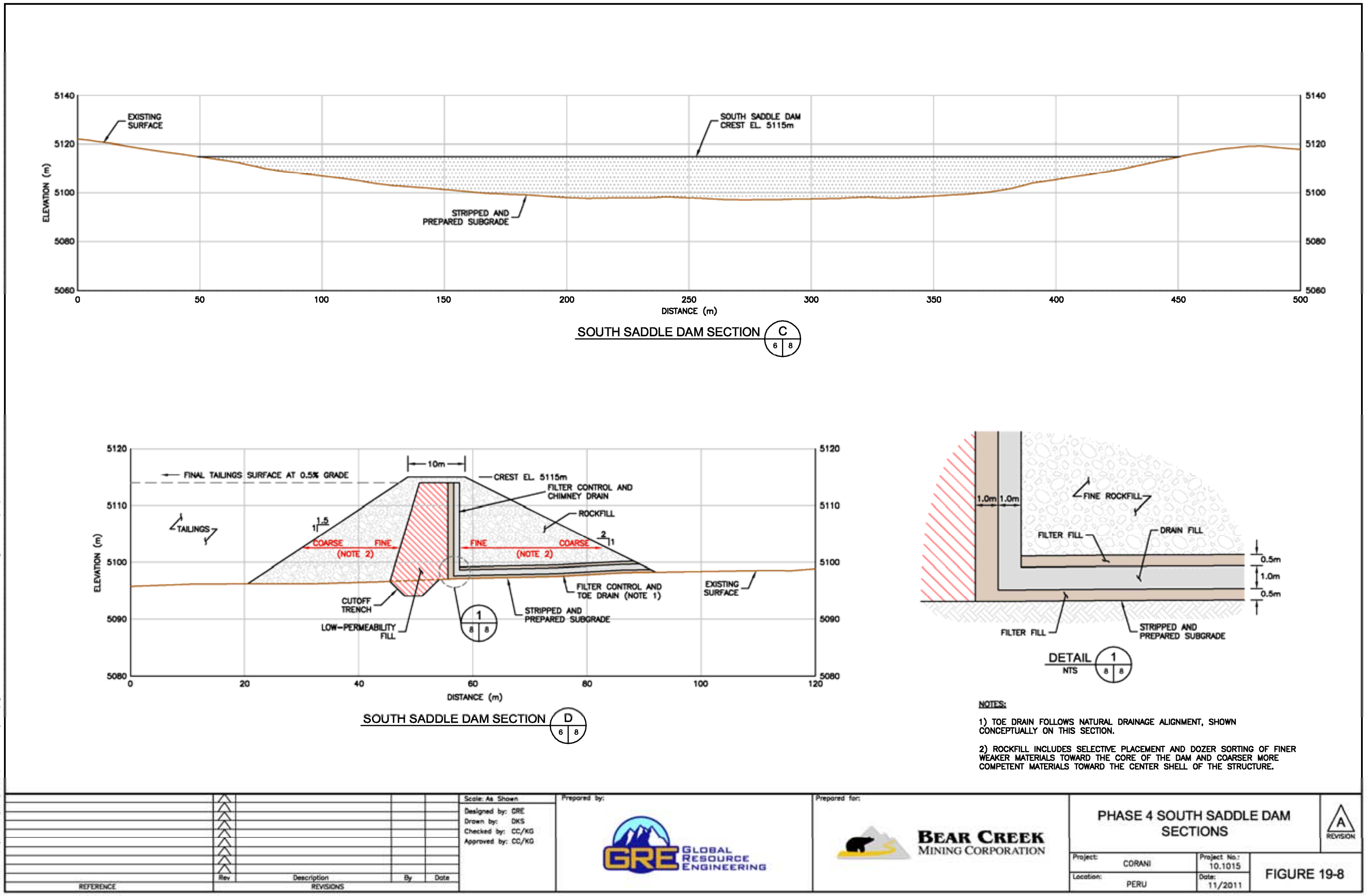


Figure 18-10: Phase 4 South Saddle Dam Sections

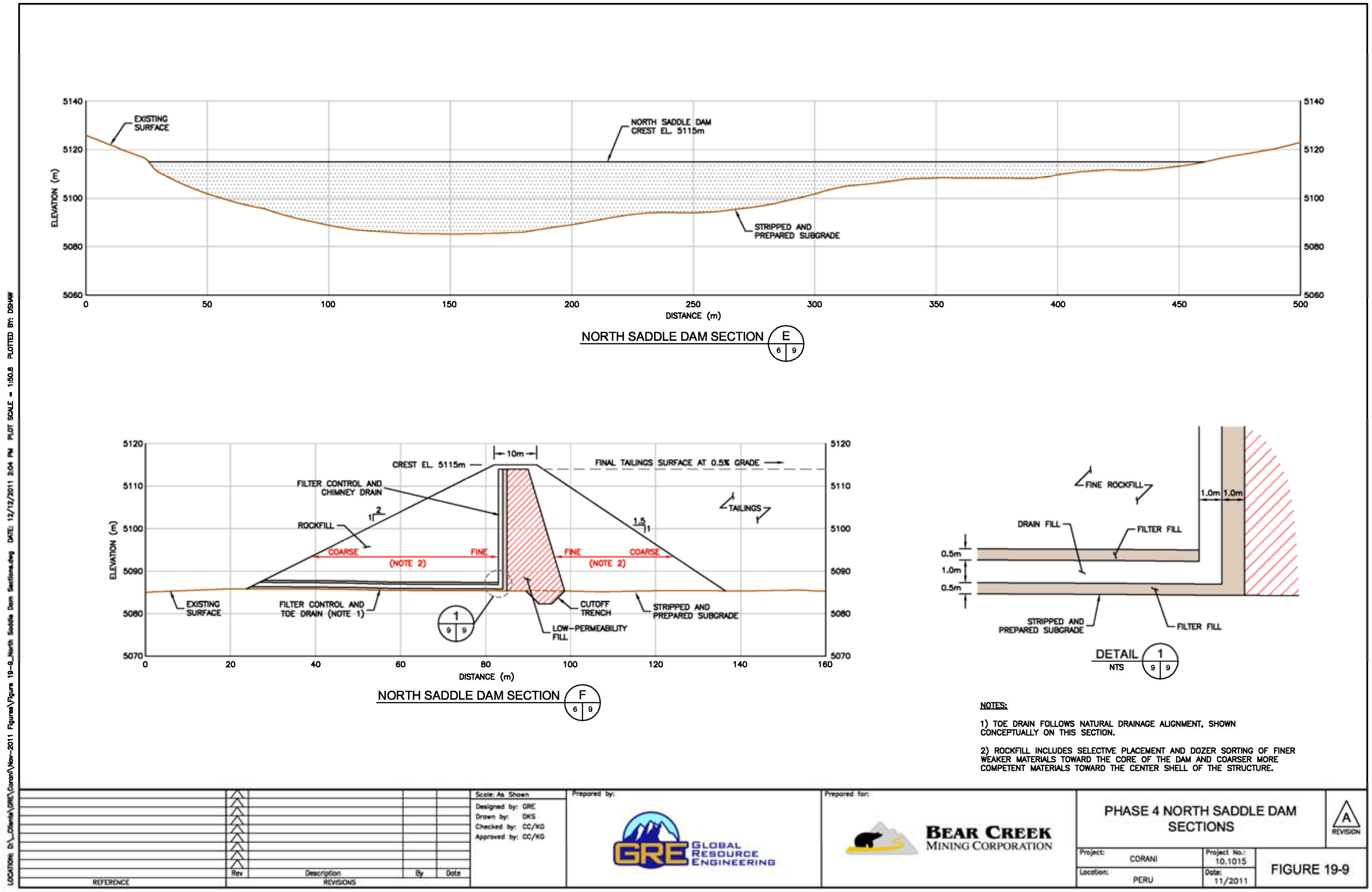


Figure 18-11: Phase 4 North Saddle Dam Sections

With the negative operating water balance, recycle of process-related solutions to the process circuit will be maximized during the TSF operating life. At the end of the operating life of the facility, water volumes can be reduced within the process circuit and, following reclamation and closure activities, the TSF can be closed with the construction of a post-closure spillway utilizing a natural topographic saddle. This spillway would function in perpetuity and would allow for periodic post-closure releases of runoff from the impoundment basin.

Considering the advantages of the South TSF in relation to the previous TSF design, the design of the South TSF is in the process of being advanced and is anticipated to be advanced through detailed engineering and construction level design.

18.2.2.7 Fresh Water Dam

As a consequence of the various optimization steps taken during 2011 in defining the current project components and GA, the need for a reliable source of make-up water for project operations was identified. The analysis of the project-wide water balance is ongoing at this time. However, a sufficient level of analysis has been completed to date to support the need for the fresh water dam. Under average climatic conditions, the potential exists that sufficient make-up water can be obtained from the collection of natural runoff from the areas of disturbance related to the project. However, seasonally, this water supply could be limited and, under dryer than average climatic conditions, insufficient water could be available to maintain operations at design levels. Although the make-up water dam was not identified as required in the 2009 Technical Report, such a facility has been incorporated in the current project design. Figure 18-4 and Figure 18-5 show the proposed layout and sections of the facility.

The fresh water dam will be located in the upper catchment of the Huaynahuayco drainage. In this portion of the drainage, the volume of water stored will increase in the wet season and decrease in the subsequent dry season. A similar zoned dam to the TSF main embankment consisting of a low permeability core and compacted rockfill shells would be constructed across the valley with an impoundment height of approximately 30 meters. The dam will store up to 2 million m^3 of water, with allowances for freeboard and discharge of excess or storm flows through an engineered spillway.

Site investigation of the main embankment area of the dam and of the reservoir area is in progress. However, no fatal flaws to the selected site, construction and operation of the facility have been identified at this time. At closure, the overall capacity of the dam would be reduced with the construction of a post-closure spillway located in competent bedrock abutment materials. The resulting structure represents a potential post-closure resource to downstream communities in the form of fresh water supply and as a source of irrigation water. The transfer of custodianship for the dam following closure to the local communities is currently considered in the preliminary Closure Plan.

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 truck transported from the mine site to the port of Matarani at a charge of US$55.00/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

Two options for transporting the concentrate from the plant to the port were studied: truck and rail.

The decision was made to transport concentrate by container trucks to the Peruvian Port of Matarini (See Figure 19-1). The decision was due to the high initial capital cost given by Peru Rail. Several Peruvian shipping (trucking) companies were contacted and bids obtained. The concentrate transport cost is estimated at US$55.00/wmt from the Corani plant to the port.

Meetings were held with various port facility operators for concentrate storage and loading. It was found that sufficient handling facilities could be made available, given sufficient notice. A handling fee of US$17.00 per ton is used in the study. This handling fee also includes the costs for insurance and owner's representation services.

The container trucks will meet all required environmental regulations and are fully enclosed.

The first seven years of Lead and Zinc concentrate production will be approximately 170,000 mtpy, dropping down to an average of 103,000 mtpy for the remainder 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$55.00 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$127.00/ 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.

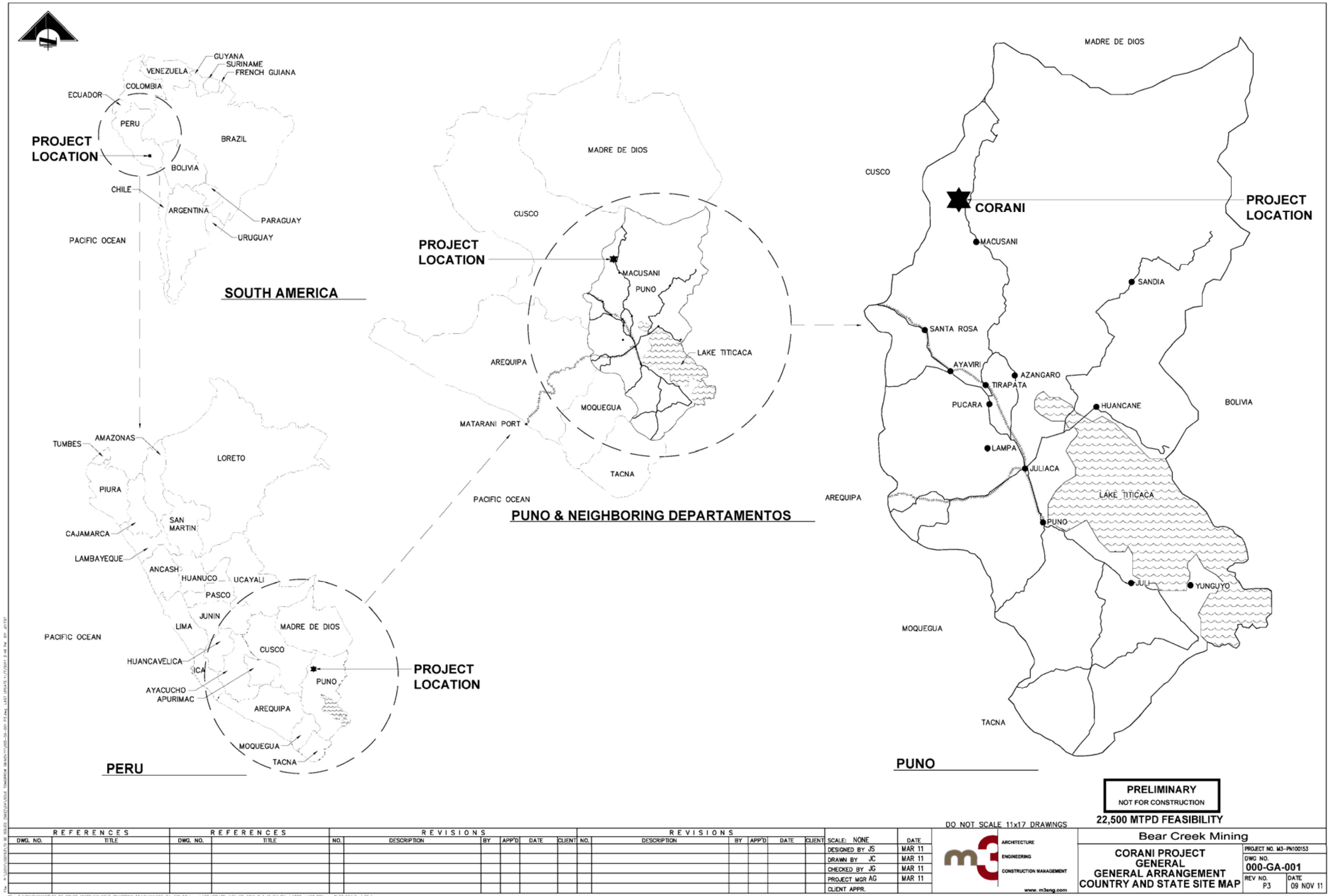


Figure 19-1: Corani Project Country and State Site Map

19.1.4 Smelter Terms

In the absence of letters of interest or letters of intent from potential smelters or buyers of concentrate, in-house database numbers were used to benchmark the terms supplied by BCM.

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 months 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	411.0	1750.7
Pb	%	4.3	53.5
Zn	%	52.1	8.1
Cu	%	0.40	1.37
Au	g/t	0.20	0.32
S	%	29.6	20.6
C(t)	%	0.19	0.77
Cl	g/t	45.0	11.7
Hg	g/t	82.5	20.3
As	g/t	430.7	1266.7
Ba	g/t	369.7	282.0
Ca	g/t	1586.7	237.3
Cd	g/t	3413.3	943.3
Fe	%	4.8	9.0
Sb	g/t	1680.0	7900.0
SiO_2	%	6.0	3.3

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 2011 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 (minimum deduction of 50 g/t).

- **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 2011 Pacific Rim Benchmark Terms for lead concentrates are as follows:

- **Payable Metals**
  - Lead: 95 percent
  - Silver: 95 percent (minimum deduction 50 grams per DMT)
  - Gold: 95 percent (minimum deduction 1.0 gram 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 – 0.20% 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

Zinc	US$0.85/lb
Lead	US$0.85/lb
Silver	US$18/troy oz

Penalty element assay values were based on chemical analyzes of concentrates produced during the composite locked-cycle test performed by SGS.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 ENVIRONMENTAL CONSIDERATIONS

Physical and biological baseline studies suitable for use in the preparation of the ESIA have been proceeding since July of 2010 and include field sampling during the dry and the wet seasons and are on-going.

Principal environmental risks associated with this type of Project fall into these categories:

- Potential impacts 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 ESIA effort will quantify the magnitude, extent, and mitigation of these impacts. It must be noted that in a number of cases the development of the project is anticipated to improve existing environmental conditions.

20.2 ENVIRONMENTAL BASELINE STUDIES

Environmental sampling has been ongoing since July 2009. As part of the Feasibility Study Investigation, extensive additional characterization was conducted in 2011, but the results of these studies have not yet been published. The following sections summarize the existing environmental data.

20.2.1 Summary of Prior Air, Noise, Groundwater and Surface Water Studies

Below is a summary of the results of 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 pollutant generating activities in the zone.
- Noise measurements were taken near populated areas. The results were below the maximums 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 sit on the low-conductivity basement rocks. The shallow aquifers

appear to 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, their protection from BCM mining activities will be part of the focus of the ESIA and future environmental planning.

- Surface water samples exhibited highly to slightly acidic characteristics during sampling events at all times of year. These acidic conditions are not related to BCM activities and relate 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 the Canadian Environmental Quality Guidelines. As a result, the major drainages leaving the Project site are sufficiently degraded to be unusable for many kilometers downstream from the project.

20.2.2 Summary of Prior Biological Studies

The biological baseline study describes the ecosystem of the site and has measured 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 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.

20.2.3 Summary of Prior Geochemical Studies

The results of waste rock geochemical characterization studies have shown that some of the Corani mine waste is potentially acid generating (PAG). These results were derived from 23 acid-base accounting analyses. Ongoing geochemical investigations are underway to better define the characteristics, quantity and production of the various waste rock types to be mined at the Project.

It is important to note that this ARD-potential has already been realized by the mine waste left on site from prior operations. Leachate from this waste and discharge from old open mine portals has exacerbated naturally occurring ARD conditions in all major site drainages.

20.2.4 Ongoing Studies

As part of the ongoing work to complete the Feasibility Study and the ESIA, BCM is executing a multiple-phase environmental investigation. This investigation includes additional borings, aquifer tests, and additional environmental samples of waste rock, tailing, surface water, and groundwater. This field work will be followed by data analysis and a revision of the Baseline Environmental Conditions Report. Predictive models (particularly of surface water, groundwater and geochemistry) will be created to determine the environmental impact of the project, and the best migration measures to alleviate environmental impacts. The following section briefly outlines the completed, ongoing, and scheduled work.

20.2.4.1 Geochemical Characterization

The previous geochemical characterization has been expanded to include the following:

- Additional geochemical samples collected from mine waste;
- Geochemical testing of mine tailing; and
- On-site long-duration kinetic cell testing.

The results of the characterization will be used to determine the extent to which various waste rock types have a potential to affect water and provide data for incorporation in waste rock management planning.

20.2.4.2 Groundwater Characterization

The project has conducted the following groundwater work since the 2009, NI 43-101 update:

- The installation of additional groundwater sampling points and an expansion of the permanent groundwater well network;
- The execution of additional aquifer tests including long-duration multiple well aquifer tests;
- The execution of additional packer tests in hard-rock formations;
- The installation of additional piezometers in the pit and TSF area; and
- The collection of additional groundwater samples.

The work summarized above augments the characterization and understanding of groundwater over the Project area.

20.2.4.3 Surface Water Characterization

The Project is currently executing an expanded work plan designed to characterize the surface water dynamics of the site. The purpose of this work is to determine the following:

- Runoff volumes and the response site drainages have to different types of precipitation events; and
- Surface water chemistry and the variations in the chemistry over time and space.

This work will be incorporated into the Feasibility Study and will assist with the design of a surface water management plan, a site water balance, and the engineering of mine water management structures.

20.3 PERMITTING CONSIDERATIONS

Refer to Section 4.4 for an explanation of the permits required to execute the Project.

20.4 RECLAMATION AND CLOSURE

Peruvian Law 28090 regulates the obligations and procedures mine owners must follow in relation to mine closure and requires that a mine closure plan is approved and financial guarantee for the cost of implementation is required. 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 on progress to the Ministry of Energy and Mines with implementation of the approved plan.

The ESIA being developed for the Project will include a conceptual closure plan. The final closure plan must be submitted within a year of the ESIA approval. The development of the closure concepts has considered IFC guidelines and industry standards in addition to the Peruvian regulatory requirements. The general approach to mine reclamation and closure developed at this time are described in the following Report sections.

20.4.1 Objective of Preliminary Closure Plan

The objective of the Preliminary Closure Plan is to identify potential and viable measures which can be implemented during the operation, reclamation and closure period and the post-closure period at Corani. The measures should alleviate the potential long-term impacts from the mining operations and minimize long-term liability. Current conditions at the project site exhibit 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 the pH and an increase in dissolved metals and salts of the surface water leaving the project area. In addition, historic mining activity within the main Corani basin has resulted in the presence of underground mine workings and of historic mine tailing which are a significant source of acid rock drainage (ARD).

It is the intent of the measures to be applied for reclamation and closure of the project to return the receiving environment to a condition, at worst, equal to the measured baseline conditions and, to the extent practical, to improve these conditions. It is not the intent of the closure measures to change the naturally occurring conditions at the site. Neither is it the intent to mitigate the effects of historic mining within the project area, apart from where 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.

Within the following subsections of this document, the overall site conditions that affect the selection and implementation of the reclamation and closure measures are identified. The specific aspects of the various project components, which require consideration in defining closure measures, and the projected overall execution of the reclamation and closure are described. In addition, the general items to be considered in the estimation of costs for the reclamation and closure are identified. A large proportion of the reclamation and closure measures will be completed as part of the operations of the mine. In the cost estimation for reclamation and closure only those activities required following the end of mining and mineral processing are included. For example, pit backfilling only considers that volume of material which will not be derived directly from mining activities during operations. Additional information in regard to the cost estimation is provided in the relevant sections of this document.

20.4.2 General Site Conditions Affecting Reclamation & Closure

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 mineralization has the potential of oxidizing at various rates when exposed to oxygen and moisture. As a result of the widespread glaciation that has occurred in the project region, surficial materials have been eroded to expose the mineralized areas as reflected by the current site conditions. In addition, the glaciation has periodically removed zones of natural weathering on the ground surface and naturally occurring growth media is limited to non-existent within the project area.

The natural environment of the project area, however, has limited the effects of oxidation and metal leaching from the mineralized zones in the form of the passive treatment capability of the bofedal (swamp) areas. The climate and altitude of the project area is also not compatible with the rapid establishment of reclamation measures in the form of vegetation which has also been limited by the acidic soil conditions that exist in portions of the project area. However, the presence of an extensive formation of post mineral tuff (PMT) exists overlying portions of the mineralized zone. This material is classified as being inert from a geochemical viewpoint due to the absence of sulfide mineralization within this formation. A large volume of this material will be mined during the development of the open pits.

The reclamation and closure measures identified in the following subsections have been developed to minimize the post-closure management of the areas of disturbance related to the mining activities. However, as described further below, some long-term post-closure management and monitoring is anticipated to be required for an extended period following mine reclamation and closure.

20.4.3 Project Components

The main project components considered in the preliminary closure plan and for cost estimation purposes consist of the following.

- The Este, Main and Corani open pit areas;
- Waste rock facilities consisting of the Este dump, Main dump and in-pit backfill areas;
- The surface water management and water collection pond systems;
- The plant facilities and related infrastructure;
- The fresh water dam located in the Quelcaya drainage; and
- The tailing storage facility (TSF).

It has not been the intent to identify all aspects of the reclamation and closure measures at this time. Rather, only the major aspects of the reclamation and closure, and their significant contribution to closure costs have been considered. Nevertheless, the estimate of quantities and the unit costs used in developing the cost estimate have tended to be conservative in order to incorporate the cost of lesser reclamation and closure activities within the overall cost estimate.

20.4.3.1 Open Pits

Mining of the open pit will occur over a period of approximately 18 years. At completion of the mining, the various pit areas identified above, will form a contiguous horseshoe shaped excavation around the east, south and west sides of the Corani Valley. Mining will be performed by developing a sequence of near vertical pit walls offset by benches to form an overall pit slope of 1h:1v. This pit wall configuration is anticipated to be stable based upon the geotechnical work completed related to the design of the pit walls. The southern pit wall is designed to intersect the bofedal to the south; and a portion of this material will be removed prior to and during mining activities. In addition, the bofedal to the north of the pit area will be intersected by the northern pit wall and portions of this material will also be removed. In these areas, the pit slope will be configured to be stable within the bofedal zones.

The pits will extend beneath the current lower bofedal elevation and several low points within the pit will extend to a depth of up to 130 m below the pit rim. In addition to the upper bofedal area, several natural drainages will be tributary to the fully developed pit area and surface runoff will exist from these drainages.

As indicated above, excavation of the bofedal material will be performed as part of the mining activity. Localized stockpiles of this material will be established for use in reclamation and closure activities. Collection and storage of these materials will be maximized to the extent practical. During the mining activity, the historic underground workings located in the southwest portion of the pit will be mined out. In addition, the historic mine tailing located adjacent to the workings will be removed from the bofedal and placed in the TSF. Although this activity will be performed during the operational period, the cost estimation has assumed that the cost of excavating and hauling the historic tailing to the plant site is to be considered as a reclamation and closure cost. Once at the processing plant, the historic tailing will be combined with the normal process tailing and pumped to the TSF. Removal of the workings and the tailing material will eliminate a major source of ARD and dissolved metals currently existing in the pit area.

As described in the mine plan, portions of the pit are completed prior to the end of mining activities. From approximately year 12 through year 18, continued mining within the pit will produce waste rock which can be used to backfill the deeper portions of the pit and place material against the lower pit walls. As shown on Figure 18-6, mine pit backfill is placed to a level at or slightly above the southern pit rim. Backfilling will also occur during this period to fill the pit bottom on the south and west sides to a minimum elevation of the northern pit rim and in areas of the pit floor in order to result in partial re-grading of the pit wall. However, the last area of mining will occur (year 18) in such a manner as to limit the ability to backfill the pit through active mining. This area will be backfilled after mining activity has ended by the importation of material from the external waste rock dumps and/or the excavation and placement of previously backfilled areas with elevations above the pit rim. Based upon the current mine plan, an estimate of 5 million m^3 of material will be required to backfill the final mining area as a reclamation and closure measure.

During the period that backfilling takes place, the waste rock will be obtained from mineralized zones within the pit wall. As described in more detail in other sections of this document, this material has a potential of oxidizing and generating acidic conditions and mobilizing elevated dissolved metals and salts. Similarly, the floor of the pit terminates in mineralized, yet sub-economic, materials that have the potential of oxidation and acid generation. During mining activities the pits will be maintained in a dewatered condition with only limited flooding during and following significant precipitation events. This water will be pumped to the mineral processing area and used as make-up water and will ultimately be placed in the TSF as part of the overall tailing slurry deposition process. In effect, with the required pH adjustment and chemical additions related to the mineral processing, the processing plant acts as a large water treatment plant for this water. However, in the absence of active dewatering of the pit, the lower portions of the pit floor have a potential of forming pit lakes that would, over time, potentially contain acidic water with elevated dissolved metal and salt constituents.

By backfilling the pit to a minimum of the elevation of the northern pit rim, the areas of backfill will accumulate runoff and direct precipitation and the backfill material will become flooded. Sub-aqueous conditions will be established beneath the northern pit rim elevation. Under these conditions, the absence of oxygen will inhibit long-term oxidation of the pit walls and the backfill material to the extent that long-term issues related to acid rock drainage from these areas will be alleviated. There will be no pit lakes left within the pit area following backfilling and reclamation and closure of the pits. Hydrogeological investigations of the pit area have also indicated that the rock formations have extremely low permeability and the potential of migration of water from within the lower backfill to the receiving environment is considered to be minimal. In addition, the post-mining water level in the lower bofedal will re-establish itself to the elevation of the pit rim and will result in low hydraulic gradients away from the backfill zone similar to those existing under pre-mining conditions.

As part of the reclamation and closure activities, a surface water management system will be established to collect runoff and direct precipitation so that it is rapidly routed through the pit area. This will minimize the contact of water with the material exposed in the pit walls and portions of the backfilled pit areas. The main components of the surface water management system will be collection areas for the drainages tributary to the pit and the construction of lined, erosion resistant channels. As discussed further below, this water will also be routed through the surface water management system external to the pit.

For the exposed backfill areas and pit floor, a layer of inert PMT material will be placed to form a cover over the PAG backfill. This material will be selected to have a grain size distribution engineered to create a store and release, evapotranspiration cover. Selection of portions of the PMT which will naturally break down through traffic and compaction equipment will be made. The function of an evapotranspiration cover is to accumulate infiltration (precipitation minus runoff) during rainfall periods and store it so that it can be evaporated following the rainfall events. In addition, a layer of bofedal material will be placed over the pit floor. The bofedal material contains organic matter in addition to fine-grained aeolian tuff. This material creates anoxic conditions and is anticipated to form an oxygen barrier in the areas where it is placed within the pit. Again, in the absence of oxygen, oxidation of the PAG material will be alleviated. Over time, it is anticipated that a natural bofedal will be created within the backfilled pit which will also function as a passive treatment system for contact waters originating from exposed PAG material within the pit walls. It is anticipated, however, that the backfilled pit area will produce ARD-impacted water that will require active treatment for an indefinite period of time.

During reclamation and closure it is assumed that active treatment of water originating from within the pit will be required for a period following closure. However, the mine plan includes the stockpiling of low grade ore that will be processed following the completion of mining. This will occur over a period of two to three years during which time the reclamation and closure of the pit area will have been performed. Water draining from the pit area will be collected and used as makeup water in the process circuit and pumped with the tailing slurry to the TSF. During the period of low grade ore processing, post closure conditions in the pit area will become established and the effectiveness of the reclamation and closure activities will be monitored. Also during this period the need any supplemental reclamation and closure activities will assessed and implemented.

20.4.3.2 Waste Rock Facilities

In addition to the waste rock backfilled into the pit, two waste rock facilities (WRF) areas will be developed. These consist of the East dump and the Main dump (Figure 18-6). Segregation of the waste rock based on the mineralized (PAG) and non-mineralized non-acid generating (NAG) nature of the material will be performed throughout mining activity. Non-acid generating PMT material will be placed in the East dump. PAG waste rock will be placed in the Main dump and forms approximately 60 to 70 percent of the volume of material to be placed in the Main dump. The remaining material will consist of PMT and managed placement of the two materials will be performed within the dump.

Both dumps will be developed from the bottom up. Initially, placement of a zone of PMT material at the face of the dumps and within the immediate placement area will be formed. Lifts of waste rock material will be placed with an external slope at approximately the natural angle of repose of the material. Each lift will be offset with a bench to form an overall slope for the dump face of 2.5h:1v. All lift surfaces will be placed to drain away from the outside dump slope. As the dumps are raised, engineered surface water management on the face of the dump will be established as part of regular mine operations. Each dump has an access/haul road(s) passing across the face of the dump to the active surface of the dump. Routing of runoff from the dump slope will be made to both the groins where the waste rock material contacts natural ground and to the inside edge of the access/haul road. Runoff collected from the road will also be routed to the groin ditches and to the toe of the dump.

Completion of placement of waste rock in both the East and Main waste dumps will occur prior to the completion of mining in the pit area. Once backfill areas are available in the pit (approximately year 12) the reclamation and closure of the dumps will be performed.

At the end of operations for the East dump, the upper surface will be graded to shed water to the sides of the dump and a permanent collection ditch connecting to the groin ditches will be constructed against the contact with natural ground. The geochemical nature of the waste rock within the East dump does not suggest that water quality issues will relate to this dump. Runoff water and any seepage water emanating from the toe of the dump will be collected and routed through the surface water management system in the event that beneficial effects can be achieved by mixing waters and/or in reducing the amount of treatment that may be required. With the concurrent reclamation and closure of the dump face, the reclamation and closure measures required for the East dump at the end of operations will be restricted to establishment of the surface water management system for the dump surface. In addition, during preparation of the footprint of the dump, the removal of the soft and organic materials within the dump footprint will be performed. This material will be stockpiled for use at the closure of the dump and will be placed over the dump surface. As well, bofedal material from above the pit area will have been stockpiled adjacent to the ultimate top surface of the East dump. A layer of these materials will be placed across the dump surface and, over the long-term, are anticipated to recreate a natural bofedal condition.

Placement of material in the Main dump will involve the sequenced placement of both PMT and PAG waste rock. Based upon the relative volumes of these materials to be placed in the dump, sufficient PMT material exists to create a minimum of a 50m wide PMT zone on the face of the dump. PMT is also available for the placement of a rind of inert material beneath the footprint of the dump and to form an inert zone of material at the base of the dump. The dump design includes the placement of PMT material so that it remains above the level of the PAG material to be placed upstream of the PMT zone. During mining of the PMT waste rock, a stockpile of the PMT material will be developed adjacent to the ultimate surface of the dump so that at closure this material can be moved over the dump to form a cap of inert waste rock material. In a similar manner to the PMT layer to be placed within the pits, material will be placed on the ultimate surface of the Main dump to form a store and release evapotranspiration cover zone.

The availability of bofedal material from within the Main dump area is limited. Nevertheless, the bofedals within the footprint of the dump and any soft or organic zones encountered during dump development will be excavated and stockpiled for future use in surface reclamation and closure of the dump. The final surface grading of the dump will be similar to the East dump and will be designed to shed water to the sides of the dump where the material contacts natural ground. A similar surface water management system will be established connecting to the groin ditches associated with the dump face and will route any runoff from the area to the toe of the dump for collection and management. Collection and diversion of the runoff from the areas tributary to the dump will be performed. At reclamation and closure the diversion will be constructed to perform as a low maintenance, post closure ditch system and will be located at the contact between the reclaimed dump and the natural ground on the east side of the dump.

It should be understood that during operations the amount of waste rock being placed at the natural moisture content that exists within the mine area will result in unsaturated conditions existing within the waste rock materials. Insufficient natural precipitation occurs on an annual basis to significantly change the moisture content of the waste rock during operations. That is, the in-place moisture content in the dumps will not be sufficient to create percolation through the material in any measurable amount. The potential exists, however, that a minor percentage of precipitation may, over the long-term, percolate through the cover materials, pass through the dump and migrate down drainage to the toe of the dump. Percolation water would be treated as necessary to comply with regulations.

20.4.3.3 Surface Water Management & Water Collection Pond Systems

During project development and operations, a surface water management system will be developed to route runoff and potentially acidic runoff and/or waters from areas of disturbance to a collection pond system. The primary pond is located below the toe of the east waste rock dump and is adjacent to the stream leaving the project area. The capacity of this pond is approximately 200,000 m^3 and the pond will include an emergency spillway and a method of releasing water from the pond should this become necessary or viable during operations. Under normal operating conditions, the collected water will be routed to the mineral processing plant for use as make-up water in a similar manner to the dewatering of the pit. This water would ultimately be treated within the plant and would be routed to the TSF as part of the tailing slurry. At the end of the ore processing, the consumptive use of the water by the plant will cease and water will ultimately require release from the system.

As part of the reclamation and closure activities, a portion of the plant previously utilized in the mineral processing will be converted to a water treatment plant. Water from the management system would be routed to the plant and treated as necessary prior to release to the receiving environment. The water treatment will primarily consist of pH adjustment with lime, settling of sludge and a sludge disposal system. The plant required for this would include reagent mixing, tankage, thickeners, pumping systems and control systems. The majority of this equipment will be available within the mineral processing plant. Sludge from the water treatment plant would be disposed of in an onsite landfill area.

During the reclamation and closure period, the collection ditches routing water from various locations around the property would be upgraded to create low-maintenance erosion resistant channels and ditches for long-term operation during the post closure period.

20.4.3.4 Plant Facilities & Related Infrastructure

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 would be decommissioned and demolished. Where viable, equipment would be salvaged and sold and/or transported offsite as scrap material. A small amount of infrastructure would be left in place for post-closure use. This would include a water treatment and equipment storage area, office building and equipment storage required for the maintenance and monitoring of the post-closure site conditions. All other structures would be demolished and the demolition debris buried in an on-site landfill. This would include the majority of the plant buildings, truck shop, crushing system, warehouses, reagent storage areas, etc.

The main power line utilized during operations would be removed and the cable salvaged. The main transformers and electrical control systems would be removed. Towers would be removed and the power line replaced by a lesser power supply for use in the water treatment plant and to provide power to the limited infrastructure remaining at the project site. All major pipelines would be either salvaged or, if buried, left in place.

Once the off-site man camp was no longer required, the camp would 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 the necessary locations for maintenance and monitoring of the project areas will be reduced in size to that of similar local roads in the project region. All other haul roads would be removed by ripping and reclaimed to conditions similar to the surrounding area.

20.4.3.5 Fresh Water Dam

The fresh water dam has a capacity of approximately 2 million m^3 of water. The operating level of the dam will fluctuate on a seasonal basis with a maximum depth of stored water of up to approximately 30 m. In its location above the Quelcaya Village, the dam, during operations, constitutes a structure that will require monitoring. At the end of ore processing, the requirement of supplemental make-up water for the operation of the processing plant will cease to exist. If left in its operational condition, the dam will remain relatively full throughout the year and will discharge during the rainy season into the downstream valley. As a structure with a long-term potential of posing a threat to life ("high-risk") and the downstream environment in the event of failure of the dam, the dam will require monitoring and maintenance in perpetuity. (It should be noted that the term 'high-risk' as used here is a classification based on risk to life and property downstream, and does not imply that the structure would be at a high risk of failure.) In the absence of a long-term custodian for the dam, the dam will be decommissioned during reclamation and closure to reduce the risk associated with the structure.

Approximately 75 percent of the dam's storage capacity exists within the uppermost 10m of the reservoir. At reclamation and closure, a permanent spillway with a depth of approximately 10m will be excavated through the north abutment of the dam. This area consists of a rock foundation and the spillway will be routed to the downstream drainage at which a stilling basin will be constructed. With the reduced storage capacity of the dam and the provision of the large capacity spillway, the risk profile of the structure will be significantly reduced. The resulting dam consists of a potential resource for the downstream communities as a fresh water supply dam and/or irrigation water source. Transfer of ownership and operation of the dam to a group of the local communities or government organizations would be performed following the implementation of reclamation and closure measures. To convert the dam so that a controlled release of water could be made, the future custodian would be required to install an outlet facility at relatively minimal cost. All operational pumping systems and infrastructure would be removed, which primarily consists of power lines and transformers and the above ground pumping systems and pipelines at the dam.

20.4.3.6 Tailing Storage Facility (TSF)

The TSF will remain a “high-risk” dam structure following reclamation and closure and will require monitoring and maintenance for the foreseeable future. (It should be noted that the term “high-risk” as used here is a classification based on risk to life and property downstream, and does not imply that the structure would be at a high risk of failure.) The dam has been designed to withstand the 1 in 10,000 year recurrence interval earthquake event during operations and the post-closure period. At reclamation and closure, a spillway will be constructed capable of passing the probable maximum flood (PMF) event. Prior to this, the dam maintains sufficient freeboard to store the PMF. The spillway will be located in the saddle at the southeast end of the main tailing dam embankment. At this location, the discharge through the spillway will enter the valley downstream of the main dam at the downstream bofedal.

In addition to the main dam embankment, a minimum of 1, and potentially 2, saddle dams will be constructed towards the end of the operating mine life. These structures are designed to withstand the same earthquake event as the main dam. At reclamation and closure the TSF will meet internationally recognized design criteria. Due to the potential that liquefaction of the tailing material contained within the impoundment could occur in portions of the saturated tailing material, the post-closure land use would be restricted to grazing. As the mine’s owner will retain long-term responsibility for the TSF, establishment of a custodian for the long-term monitoring and management of the facility will be required.

In order to configure the impoundment surface such that water entering the impoundment during the post-closure period runs to the spillway without the impoundment of a significant volume of water upstream of the dam, tailing deposition will be modified over the last 5 to 6 years of operations. This will involve the deposition of tailing primarily from the north and west limits of the impoundment. Towards the end of operations, additional deposition will be performed from the south side of the impoundment as required to create an overall slope of the impoundment surface towards the spillway. With this deposition sequence, the operational freewater pond, from which water is recycled to the processing plant, will migrate to the south towards the spillway location.

Once the freewater pond has moved towards the south, additional deposition would be placed over the deepest portion of the impoundment in order to form a mound of tailing in this area. This is to be performed so that the potential long-term settlement of the tailing material will, to a large extent, be alleviated so that supplemental re-grading of the impoundment will be minimized during reclamation and closure. In order to maintain recycle capacity, modification of the pumping system will be required by extending the pipeline, pumping system and power supply along the crest of the dam concurrent with the migration of the freewater pond.

During this deposition period, all tailing entering the impoundment will be de-pyritized. The mineral processing circuit has been designed with a supplemental flotation circuit to reduce the residual sulfide levels in the tailing material. This part of the processing circuit will be constructed remote from the main processing circuit at a position on the north side of the TSF. The flotation circuit will produce a pyrite concentrate that will be deposited sub-aqueously within the impoundment. The purpose of the secondary flotation is to produce an inert tailing material as the final upper layers of the impoundment area.

To dispose of the pyrite concentrate material, a floating pipeline would be installed from the flotation plant to the deepest point of the operating freewater pond on the impoundment. As the pond migrates southwards, the floating tailing line would be moved to maintain its position above the deepest point in the pond. As additional inert tailing are deposited, the pyrite concentrate would be buried beneath the subsequent tailing deposition. At the end of the operating period, a zone of the pyrite concentrate would remain beneath the residual pond. In order to cover these materials and displace the pond water, processing of inert waste rock material would be performed for a period of several months and this material would be deposited within the center of the pond. No reagent or additional processing would be performed during the placement of the inert material within the pond.

During the last 12 to 18 months of processing, the volume of make-up water added to the circuit would be reduced. The objective of reducing the make-up water is to reduce the volume of water in the freewater pond on the impoundment to the minimum required to maintain water recycle. In this manner, the volume of water within the tailing system would be minimized prior to reclamation and closure. With the deposition of the inert, non-processed material in the freewater pond following the end of ore processing, the chemistry of the water within the freewater pond will change and the constituent levels present in the operating supernatant solution will decrease. A small shallow pond will remain on the impoundment surface adjacent to the spillway inlet.

As the impoundment surface remote from the spillway area dries and consolidation access will be possible for light equipment to the impoundment. Over a period of 2 to 3 years after the completion of the above steps, the impoundment surface will be progressively covered by a layer of bofedal material that is to be salvaged from the impoundment footprint and stockpiled prior to commissioning of the facilities. Sufficient material to place a minimum of 150mm of this material over the majority of the impoundment area is available within the impoundment area. In the long-term it is anticipated that a natural bofedal will establish itself on the impoundment surface. Figure 20-1 and Figure 20-2 show the Tailing Storage Facility closure plan and sections.

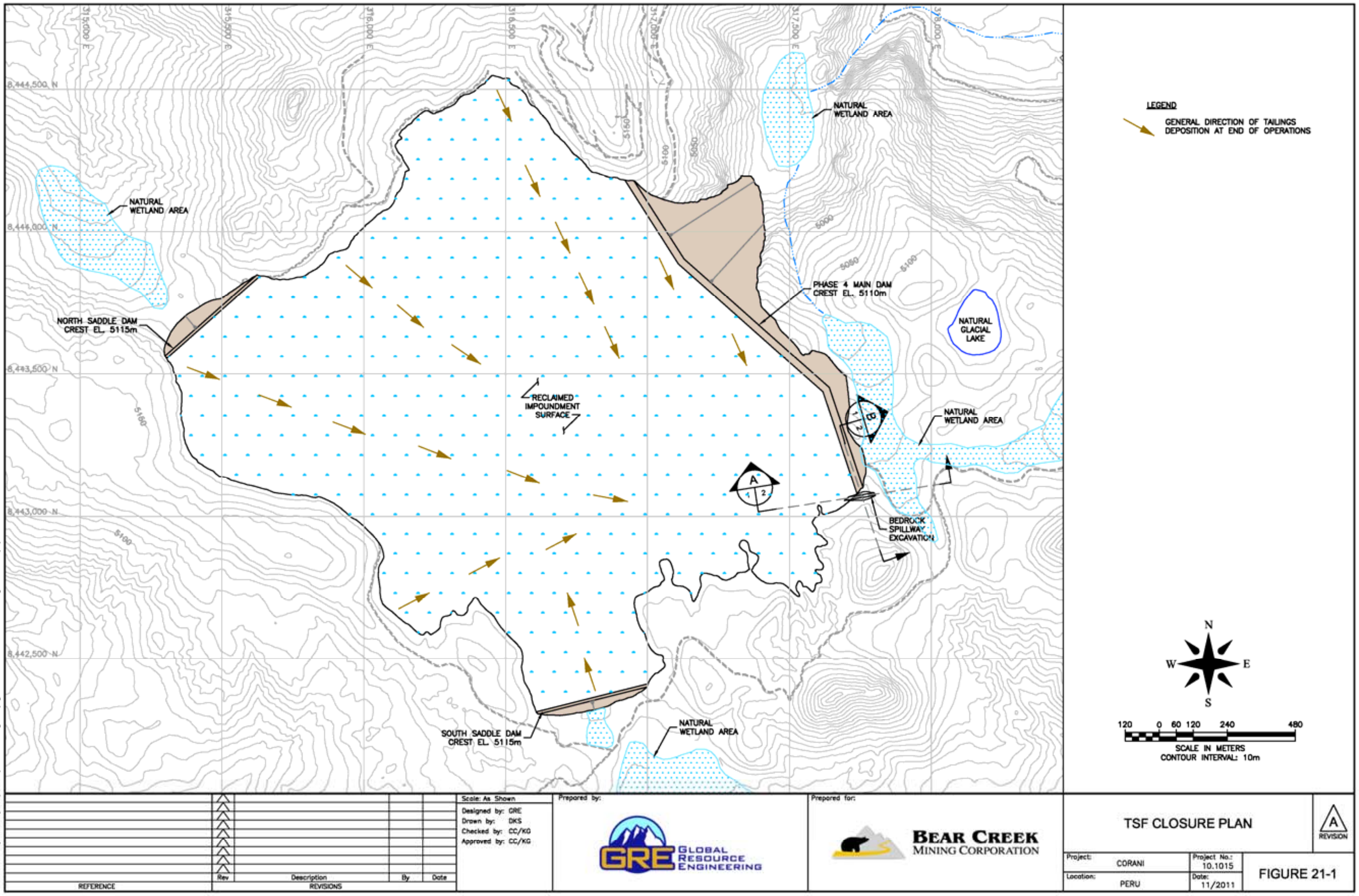


Figure 20-1: TSF Closure Plan

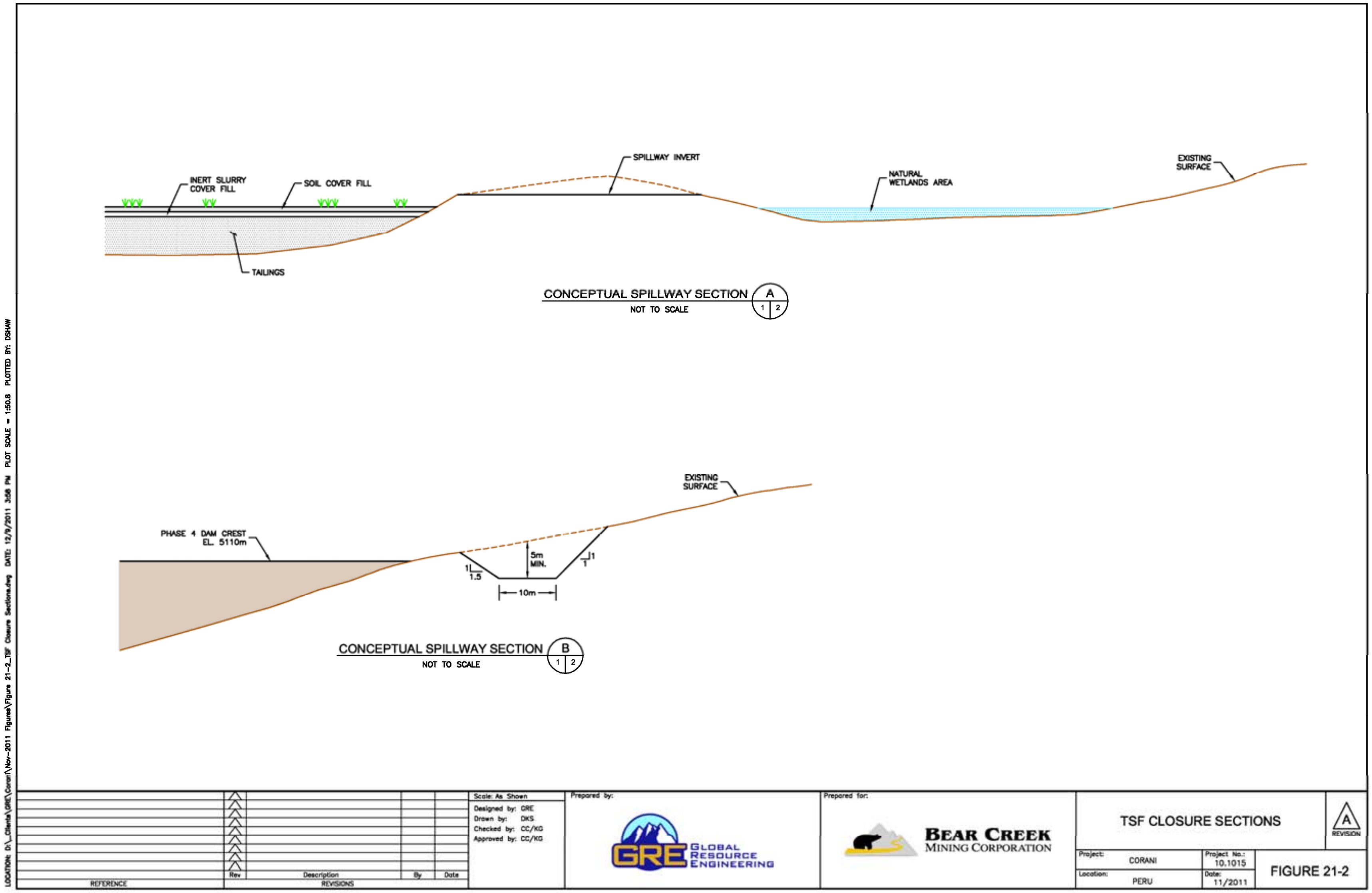


Figure 20-2: TSF Closure Sections

The primary seepage collection system, located at the toe of the main tailing impoundment will be maintained during the reclamation and closure and post-closure period in order to collect and monitor the seepage quantity and quality at that point. Similarly, seepage collection systems at the saddle berms will also be maintained during this period. Seepage analyses for the main and saddle dams indicate that seepage through the dams will be minimal. For the main dam, the majority of water reporting to the seepage collection system is from direct precipitation falling on the downstream section of the dam and on areas tributary to the dam downstream of the centerline. As measured in the project's baseline studies, the quality of this water is currently degraded due to contact with mineralized material exposed on the valley walls. The bulk of these materials will be covered by the tailing impoundment and upstream section of the main embankment; and the impact to precipitation and infiltration reporting to the seepage collection system will be significantly reduced from baseline conditions. Initially, if required, water from the seepage collection system will be recycled into the impoundment. It is anticipated, however, that the long-term water quality at the seepage collection system will develop a chemistry at or better than the baseline conditions.

20.4.3.7 Monitoring and Maintenance

Monitoring and maintenance of the mine reclamation and closure activities and in the post closure period will be performed. This will consist primarily of a team of onsite personnel. Equipment to monitor the reclaimed areas of the project and to perform regular maintenance will be available for this purpose. The monitoring will include water quality sampling and analyses in addition to regular inspection of the TSF, WRFs and the pit areas. The required duration of the monitoring and maintenance is undefined at this time however, a minimum period of 25 years has been considered in the cost estimate.

20.4.3.8 Closure Schedule

The schedule for performing reclamation and closure activities and anticipated duration can be summarized as follows:

- East waste rock dump – Year 12 over 12 months
- Main waste rock dump – Year 14 over 12 months
- Pit area – Year 18 over 12 months
- Operational surface water management system – Year 20 over 6 months
- Water treatment plant – Year 20 over 9 months
- Fresh water makeup dam – Year 20 over 9 months
- TSF – Year 20 over 3 years

20.5 SOCIOECONOMICS AND COMMUNITY

The Project site is located within the jurisdiction of two rural communities (comunidades campesinas), specifically Chacaconiza y Quelcaya. These communities, situated at over 4,000 masl are small isolated rural settlements consisting of around 120 individuals. The majority of the population speaks Quechua.

The geographical isolation and lack of infrastructure in the zone result in reduced economic and social development opportunities and high levels of poverty result. However both communities have access to water and 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 will be undertaken as part of the ESIA. The ESIA process will include 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 will be developed to avoid, remedy or mitigate the identified community impacts including the development of social programs aimed at providing enhanced economic and social development opportunities.

Further community engagement will occur concurrently with the impact assessment process. This will include two initial workshops with each of the two communities (Chacaconiza and Quelcaya) affected by the project. A location and time that is convenient for the community members will be agreed for each of the workshops. The first workshops are undertaken before the ESIA studies begin and the ESIA process will be explained to the community. The second workshops will be undertaken during the elaboration of the ESIA. Finally, a public meeting is held. At this meeting the ESIA and Community Participation Plan are presented to the approving authorities. Currently, the first workshops have been carried out at the site and the ESIA process has begun.

21 CAPITAL AND OPERATING COSTS

21.1 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 21-1 shows a detail schedule of the G&A area showing the annual cost by category.

Table 21-1: G&A Costs by Area

Sulfide Ore Tonnes Processed	**7,875,000**	
	Total	
Cost Item	**Annual Cost - $**	**$/ton ore**
Labor & Fringes	$ 3,088,968	$ 0.392
Power (@ 0.5% of processing power)	90,166	0.011
Vehicle Operating & Maintenance	547,500	0.070
Communications	151,500	0.019
Safety Supplies / Incentives	180,000	0.023
Offsite Training & Conferences	36,000	0.005
Insurance	1,501,363	0.191
Corporate Services and Travel	894,000	0.114
Environmental	108,000	0.014
Security & Medical	247,500	0.031
Professional Membership Costs	6,000	0.001
Community Development	300,000	0.038
Bussing (150 weekly and 40 per day)	224,000	0.028
Staff Living Expences (250 people at the camp)	1,642,500	0.209
Consultants	67,500	0.009
Computer Equipment/Software	37,500	0.005
Misc. Office Supplies	18,000	0.002
Misc. Freight & Couriers	18,000	0.002
Recruiting and Relocation	198,000	0.025
Mine Access Road Maintenance	391,230	0.050
Legal, Permits, Fees	275,000	0.035
Contingency (10%)	1,002,273	0.127
Total General & Administrative Cost	**$ 11,025,000**	**$ 1.400**

21.2 MINE OPERATING COST

Mine operating costs were developed based on first principals for the mine plan and equipment list presented earlier in Section 16. The unit costs for labor were provided by BCM and M3 engineering. Fuel costs were set at $1.05 USD per liter.

Table 21-2 summarizes the mine operating costs by the unit operations. The operating costs have been broken into quarterly time periods for preproduction and years 1 and 2 to parallel the mine plan. Preproduction is established to be 18 months or 6 quarters. During the first quarter of preproduction (Qtr -6), the costs shown are for development of the initial access roads to the mine working areas. No in-pit mining tonnage is moved during that period so there is no calculation of “cost per tonne”. The cost per tonne in all remaining periods is based on the total tonnage moved within the mine plan.

Table 21-3 summarizes the total mine operating cost per time period along with the total mine capital cost. This table should provide a clear indication of the mine operating costs by year of operation.

The mine operating costs include:

1. Drilling, blasting, loading, and hauling of material from the mine to the crusher, low grade stockpile or waste storage facility. Maintenance of the waste storage areas and stockpiles is included in the mining costs. Maintenance of mine mobile equipment is included in the operating costs.
2. Mine supervision, mine engineering, geology and ore control are included in the G&A category.
3. Operating labor and maintenance labor for the mine mobile equipment are included.
4. Mine access road construction and maintenance is included. If mine haul trucks drive on the road, its cost and maintenance is included in the mine operating costs.
5. Removal of the marsh alluvial material (bofedal) as required for mine operations is included.
6. Delivery of post mineral tuff mine waste to the tailing dam construction is included. However, placement and compaction of that material at the tailing facility is not included in the mine operating costs but is included in the construction capital cost of the Tailing Storage Facility.
7. The small stockpile (804 Ktons) that is generated during preproduction stripping is rehanded to the plant in Year 1.
8. The cost of remining the low grade stockpile in years 18 through 20 is included.
9. The cost of spreading post mineral tuff (PMT) over the exposed waste storage areas in the South dump and the in-pit dumps is included.
10. A general mine allowance is included that is intended to cover mine pumping costs and general operating supplies that cannot be assigned to one of the unit operations.
11. A general maintenance allowance is included that is intended to cover the general operating supplies of the maintenance group.

The mine operating costs DO NOT include:

1. Crushing, conveying or processing
2. The only reclamation costs are the costs to deliver and spread PMT over all of the waste dumps at the end of the mine life. No recontouring or other reclamation costs are included.

The mine is planned to work 2 shifts per day for 365 days per year. Fifteen days (30 shifts) of loss time are assumed due to weather delays and holidays.

Table 21-2: Corani Project Mine Operating Costs

Summary of Mine Operating Costs - Per Total Tonne ($US x 1000)												
Mining Year	Total Material (kt)	Drilled/ Blasted (kt)	Drilling	Blasting	Loading	Hauling	Auxiliary	General Mine	General Maint.	G&A	TOTAL	Total Cost
PPQ-6	0	0	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	0.000	819
PPQ-5	576	576	0.136	0.145	0.182	0.517	1.228	0.110	0.108	0.653	3.079	1,774
PPQ-4	1,478	1,478	0.133	0.147	0.182	0.722	0.692	0.073	0.076	0.303	2.326	3,438
PPQ-3	3,696	3,696	0.133	0.138	0.201	0.518	0.295	0.046	0.049	0.127	1.506	5,567
PPQ-2	5,625	5,625	0.132	0.136	0.212	0.420	0.208	0.040	0.042	0.085	1.276	7,179
PPQ-1	5,625	5,625	0.132	0.136	0.212	0.422	0.226	0.040	0.042	0.085	1.295	7,283
Yr1Q1	6,343	5,625	0.117	0.121	0.206	0.370	0.263	0.038	0.041	0.076	1.233	7,819
Yr1Q2	5,625	5,625	0.131	0.136	0.211	0.423	0.298	0.040	0.043	0.086	1.369	7,698
Yr1Q3	5,685	5,625	0.130	0.135	0.211	0.453	0.295	0.040	0.043	0.086	1.392	7,912
Yr1Q4	5,660	5,625	0.130	0.135	0.211	0.424	0.296	0.040	0.043	0.086	1.364	7,723
Yr2Q1	5,625	5,625	0.131	0.136	0.207	0.461	0.301	0.040	0.043	0.087	1.406	7,907
Yr2Q2	5,625	5,625	0.131	0.136	0.207	0.477	0.301	0.040	0.043	0.087	1.421	7,994
Yr2Q3	5,625	5,625	0.130	0.136	0.206	0.475	0.301	0.040	0.043	0.087	1.419	7,981
Yr2Q4	5,625	5,625	0.131	0.136	0.207	0.533	0.310	0.040	0.044	0.087	1.487	8,363
3	22,500	22,500	0.131	0.136	0.207	0.510	0.332	0.040	0.044	0.087	1.486	33,425
4	22,500	22,500	0.131	0.136	0.204	0.529	0.332	0.040	0.044	0.087	1.503	33,811
5	22,500	22,500	0.131	0.136	0.204	0.532	0.334	0.040	0.044	0.087	1.507	33,909
6	22,500	22,500	0.131	0.136	0.204	0.486	0.325	0.040	0.044	0.087	1.453	32,686
7	23,000	23,000	0.131	0.136	0.205	0.541	0.319	0.039	0.043	0.085	1.499	34,469
8	24,000	24,000	0.131	0.136	0.211	0.536	0.293	0.039	0.043	0.082	1.470	35,281
9	24,000	24,000	0.131	0.136	0.211	0.557	0.254	0.039	0.042	0.081	1.452	34,852
10	24,000	24,000	0.131	0.136	0.211	0.617	0.254	0.039	0.043	0.082	1.512	36,294
11	23,000	23,000	0.131	0.136	0.211	0.656	0.265	0.039	0.043	0.085	1.567	36,031
12	23,000	23,000	0.131	0.136	0.211	0.633	0.293	0.039	0.043	0.085	1.572	36,159
13	22,500	22,500	0.130	0.136	0.211	0.371	0.308	0.040	0.043	0.086	1.326	29,827
14	22,000	22,000	0.131	0.136	0.212	0.501	0.276	0.040	0.044	0.088	1.428	31,411
15	22,000	22,000	0.131	0.136	0.212	0.433	0.273	0.040	0.043	0.088	1.355	29,806
16	22,000	22,000	0.131	0.136	0.207	0.338	0.244	0.040	0.043	0.087	1.226	26,974
17	22,944	21,744	0.124	0.129	0.207	0.312	0.234	0.039	0.042	0.083	1.170	26,834
18	21,420	16,955	0.104	0.109	0.206	0.274	0.203	0.040	0.042	0.088	1.066	22,839
19	16,373	0	0.000	0.000	0.191	0.193	0.134	0.036	0.042	0.064	0.660	10,809
20	12,081	0	0.000	0.000	0.189	0.158	0.174	0.038	0.047	0.084	0.690	8,338
TOTAL	455,131	420,199	0.121	0.126	0.207	0.467	0.279	0.040	0.043	0.087	1.369	623,212
PERCENT			8.8%	9.2%	15.1%	34.1%	20.3%	2.9%	3.2%	6.3%	100.0%	
Per Tonne Drilled/Blasted			0.131	0.136								

Table 21-3: Summary of Mine Capital and Operating Costs ($US x 1000)

Year	Mine Equipment Initial Capital Cost	Mine Equipment Sustaining Capital Cost	Mine Preprod. Development	(1) Total Mine Capital	Mine Operating Cost	MINE TOTAL COST
PPQ-6	8,622		819	9,441		9,441
PPQ-5	18,194		1,774	19,967		19,967
PPQ-4	5,375		3,438	8,813		8,813
PPQ-3	16,108		5,567	21,676		21,676
PPQ-2	9,425		7,179	16,604		16,604
PPQ-1	23		7,283	7,306		7,306
Yr1Q1		2,074		2,074	7,819	9,893
Yr1Q2		0		0	7,698	7,698
Yr1Q3		2,477		2,477	7,912	10,389
Yr1Q4		0		0	7,723	7,723
Yr2Q1		847		847	7,907	8,754
Yr2Q2		46		46	7,994	8,040
Yr2Q3		0		0	7,981	7,981
Yr2Q4		2,478		2,478	8,363	10,841
3		0		0	33,425	33,425
4		3,013		3,013	33,811	36,824
5		1,722		1,722	33,909	35,631
6		6,062		6,062	32,686	38,748
7		6,121		6,121	34,469	40,590
8		1,287		1,287	35,281	36,568
9		6,318		6,318	34,852	41,170
10		20,652		20,652	36,294	56,946
11		7,313		7,313	36,031	43,344
12		10,716		10,716	36,159	46,875
13		2,033		2,033	29,827	31,860
14		4,802		4,802	31,411	36,213
15		46		46	29,806	29,852
16		4,433		4,433	26,974	31,407
17		0		0	26,834	26,834
18		0		0	22,839	22,839
19		0		0	10,809	10,809
20		0		0	8,338	8,338
TOTAL	57,748	82,440	26,060	166,248	597,152	763,400

(1) Preproduction Development is shown as capital on this table.

21.3 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. A summary of the process plant operating costs for a typical year of operations is shown in Table 21-4.

Table 21-4: Process Plant Operating Cost

Processing Units Base Rate (tonnes/year mill ore)	7,875,000	
Cost Item	Total Annual - $	$/ton ore
Primary Crushing		
Operating Labor and Fringes	$249,000	$ 0.032
Power	218,532	0.028
Liners	296,100	0.038
Maintenance	527,600	0.067
Supplies & Services	140,000	0.018
Subtotal Primary Crushing	**$ 1,431,231**	**0.182**
Grinding		
Operating Labor and Fringes	$238,200	$ 0.030
Power	8,392,307	1.066
Grinding Media	9,646,875	1.225
Liners	1,638,000	0.208
Maintenance	1,832,741	0.233
Supplies and Services	390,000	0.050
Subtotal Grinding	**$ 22,138,123**	**$ 2.811**
Flotation		
Operating Labor and Fringes	$487,200	$ 0.062
Power	3,641,694	0.462
Reagents	23,854,163	3.029
Grinding Media	212,625	0.027
Liners	204,750	0.026
Maintenance	1,463,164	0.186
Supplies and Services	165,000	0.021
Subtotal Flotation	**$ 30,028,595**	**$ 3.813**
Concentrate Thickening,Filtration and Tailings		
Operating Labor and Fringes	$525,000	$ 0.067
Reagents	614,250	0.078
Power	1,634,145	0.208
Maintenance	551,887	0.070
Supplies and Services	381,581	0.048
Subtotal Concentrate Thickening,Filtration and Tailings	**$ 3,706,863**	**$ 0.471**
Ancillary		
Operating Labor and Fringes	$552,600	0.070
Power	153,069	0.019
Maintenance	625,510	0.079
Water Charges	-	-
Supplies and Services	763,604	0.097
Subtotal Ancillary	**$ 2,094,783**	**$ 0.266**
Total Process Plant	**$ 59,399,595**	**$ 7.543**

21.3.1 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 119 employees will be employed at the process plant with 74 in operations and 45 in maintenance. A summary of the staffing plan and gross annual labor costs are shown in Table 21-5.

Table 21-5: Process Plant Staffing Summary

Department	Number Of Personnel	Total Labor ($)/year
Mill Operations		
Mill Operations	74	$2,052,000
Mill Maintenance	45	$1,055,200
Total Labor Cost	**119**	$3,107,200

21.3.2 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 $0.055 per kWh with a consumption of 32.8 kWh per ore tonne. A summary of the power consumption and cost are shown in Table 21-6.

Table 21-6: Summary of Electric Power

Cost Item	Area	Connected kW	Total (kW hr/yr)	Total Cost
Concentrator				
Primary Crushing & Conveying		838	3,973,301	$ 218,532
Grinding		23,437	152,587,405	$ 8,392,307
Flotation		11,528	66,212,612	$ 3,641,694
Concentrate Thickening/Filtration		984	5,753,610	$ 316,449
Reagents Storage		222	1,468,600	$ 80,773
Tailings Management & Reclaim Systems		3,348	23,958,123	$ 1,317,697
Water Supply System		835	3,746,138	$ 206,038
Ancillary		245	839,592	$ 46,178
Total Connected (kW)		41,438		
Total Consumption (kW-hr)			258,539,382	$14,219,666
Cost Per kW-hr				$ 0.055

21.3.3 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 below.

Table 21-7: Summary of Reagents

Process Reagent (Consumption Basis)	Consumption		Unit Rate	
	kg/tonne ore	kg/year	$/kg	Annual Cost
Sodium Isopropyl Xanthate (SIPX)	0.040	315,000	$ 1.92	$ 604,800
Lime	3.000	23,625,000	$ 0.17	$ 4,016,250
Methyl Isobutyl Carbinol (MIBC)	0.030	236,250	$ 2.98	$ 704,025
AP 404 Promoter	0.015	118,125	$ 2.50	$ 295,313
Emulsified Diesel Oil	0.001	7,875	$ 1.45	$ 11,419
Sodium Cyanide	0.210	1,653,750	$ 2.75	$ 4,547,813
Copper Sulfate	0.290	2,283,750	$ 2.70	$ 6,166,125
Sodium Carbonate	0.370	2,913,750	$ 0.60	$ 1,748,250
Sodium Hydroxide	0.010	78,750	$ 0.69	$ 54,338
Sodium Sulfite	0.505	3,976,875	$ 0.61	$ 2,425,894
Zinc Sulfate	0.620	4,882,500	$ 0.65	$ 3,173,625
Antiscalant	0.005	39,375	$ 2.70	$ 106,313
Flocculent (thickening)	0.020	157,500	$ 3.90	$ 614,250

21.3.4 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 21-8.

Table 21-8: Grinding Media and Wear Parts

Process Plant Ore Tonnes (annual production)		7,875,000		
	Consumption		Unit Rate	
Grinding Media & Wear Parts	kg/tonne ore	kg/year	$/kg	Annual Cost
Primary Crusher Liners	0.008	63,000	$ 4.70	$ 296,100
SAG Mill Liners	0.050	393,750	$ 2.60	$ 1,023,750
Ball Mill Liners	0.030	236,250	$ 2.60	$ 614,250
Lead Regrind Vertimill Liners	0.005	39,375	$ 2.60	$ 102,375
Zinc Regrind Vertimill Liners	0.005	39,375	$ 2.60	$ 102,375
SAG Mill - Balls	0.500	3,937,500	$ 1.25	$ 4,921,875
Ball Mill - Balls	0.500	3,937,500	$ 1.20	$ 4,725,000
Lead Regrind Vertimill - Balls	0.010	78,750	$ 1.35	$ 106,313
Zinc Regrind Vertimill - Balls	0.010	78,750	$ 1.35	$ 106,313

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.

21.3.5 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 21-9. The laboratory labor cost is based on a staff of 24 for an annual cost of $1.1 million.

All other laboratory costs were developed as allowances based on M3's information from other projects and guidance from Bear Creek Mining.

Table 21-9: Laboratory Cost

Sulfide Ore Tonnes Processed	**7,875,000**	
	Total	
Cost Item	**Annual Cost - $**	**$/ton ore**
Labor & Fringes	760,603	$ 0.097
Power Allocation Ancillary Facilities	1,573	0.000
Reagents & Fuel	16,000	0.002
Assay Consumables	230,000	0.029
Wear & Maintenance Parts	49,500	0.006
Maintenance Labor, Fringes, and Allocations (2.5%)	21,104	0.003
Supplies and Services	5,000	0.001
Total Laboratory Cost	**$1,083,780**	**$ 0.138**

21.4 CONCENTRATE HANDLING, TRANSPORTATION AND STORAGE

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. The truck transportation and port costs are estimated to be $72.00 per wet metric tonne. The shipping cost is estimated to $55.00 per wet metric tonne. A total of $346.2 million is estimated to be paid for concentrate transportation for the life of the mine.

21.5 CAPITAL COST ESTIMATE

21.5.1 Introduction

The capital cost estimate prepared for this Report addresses a silver-lead-zinc concentrator capable of processing 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 $574 million. Table 21-10 summarizes the capital costs by major area.

Table 21-10: Capital Cost Summary

AREA	TOTAL	
General Site Including Access Road	$	47,046,501
Mine Capital + Preproduction	$	84,315,000
Primary Crushing	$	15,038,626
Reclaim Stockpile	$	6,381,502
Grinding	$	44,958,665
Flotation and Regrind	$	39,552,771
Concentrate Thickening	$	10,161,130
Tailing Disposal	$	62,288,241
Fresh Water/ Plant Water	$	17,900,934
Power Supply Infrastructure	$	13,873,238
Reagents	$	3,277,208
Ancillaries	$	21,894,455
Off Sites (Camp)	$	13,822,799
Direct Cost	**$**	**380,511,068**
Contractor Indirects	$	11,505,867
EPCM Services	$	47,161,819
Commissioning and Vendor Reps	$	2,489,369
Capital Spare Parts & Initial Fills	$	8,859,561
Owner's Cost	$	30,771,413
Freight, Duties	$	27,192,086
Indirect Cost	**$**	**127,980,115**
Contingency (Process Plant)	$	59,010,715
Contingency (Mine)	$	6,872,450
Total	**$**	**574,374,347**

Note: Total shown in table is inconsistent because of rounding of the inputs

21.5.2 Currency

The estimate is expressed in third-quarter 2011 United States dollars. No provision has been included to offset future escalation. No funds have been allocated in the estimate to offset potential currency fluctuations.

21.5.3 Scope

The capital cost estimate for the Project addresses the engineering, procurement, construction and start-up of a 22,500 tpd silver-lead-zinc concentrator located near Macusani, Peru.

The boundary conditions for M3 scope-of-work include the process plant from primary crusher to tailing discharge, associated process and mine support infrastructure such as administration buildings, truck shop, truck wash, plant maintenance building, warehouse, laboratory, maintenance shops, medical facility, including emergency power generation and distribution to support the site, fresh water supply pipeline and pumping costs. M3 scope also included the camp site with dormitories, cafeteria, dining, laundry, recreational center. Owner's costs were provided by BCM.

Also included in the capital cost estimate are the tailing impoundment costs, site access road costs, waste stockpiles costs, mining costs provided by IMC, and incoming power costs provided by M3/PROMOTORA.

M3 is responsible for the assembly of the overall capital cost estimate with supporting data provided by others for the key areas noted below:

Table 21-11: Capital Cost Estimate Areas and Responsible Parties

Area	Responsible Party
Mine	IMC
Tailing Storage Facility	GRE
Transmission Line	PROMOTORA/ M3
Mine Access Road	M3
Port Costs (concentrate tote bins)	M3
Indirect Cost	All
Owner's Cost	BCM
Contingency	All

21.5.4 Tailing Storage Facility Basis of Cost Estimate

The cost estimate for the South Tailing Storage Facility (TSF) and fresh water dam was completed by applying estimated unit costs to the engineering quantity estimate. These unit costs were developed through different resources including price quotes, comparable projects, cost indices, detailed first principle estimates of equipment operating costs and productivities

Allowances were also made for the remote location of the project and its high elevation of around 5,000 meters above sea level. Eighty-seven percent, 87%, of the construction cost for the TSF and fresh water dam can be attributed to the following construction components:

- Rock Fill
- Low Permeability Fill
- Bofedal Stripping
- Drain Fill
- Filter Fill

Rock fill is the main construction component of both the TSF and fresh water dam embankments followed by low permeability fill, drain fill, and filter fill. As the embankments are the main construction components of these facilities, these items represent the main quantity component of the facilities themselves. In addition to the aforementioned materials, bofedal stripping is a major component of the TSF construction due to the large area of the impoundment and the intent to remove and stockpile this material for final reclamation and closure of the impoundment.

The unit costs for rock fill and low permeability fill were developed using detailed first principle estimates of equipment costs and productivities. Rock fill is sourced from two principle locations during construction, a borrow area located within the TSF impoundment and the open pit. Unit operations associated with the borrow area rock fill are drilling, blasting, loading, hauling, and placement. This particular scenario only occurs during the first and final phases of construction requiring a total of 3 separate haul profiles to develop construction scenario specific productivities based on the final location of the fill: the main dam (1^{st} phase), north saddle dam (final phase), or south saddle dam (final phase). Unit operations associated with the open pit rock fill only include placement since the remaining unit operations are already covered within the mining costs.

Caterpillar's Fleet Production and Cost (FPC) was used to estimate the different number of required trucks by haul. An additional two trucks were added to that estimate to account for periodic short dumping of unsuitable material which cannot be used for construction fill. Due to the time constraints for preproduction construction a total of 3 excavator truck fleets are required to attain the required production. Each fleet consists of a Caterpillar (CAT) 345 excavator and CAT 725 articulated trucks.

The equipment productivities for each unit operation drilling, blasting, loading, hauling and placement, were combined to determine the total operating hours by equipment type for each construction scenario. An allowance for lower productivity during the wet season, approximately 6 months of the year, was included in the operating hour estimate. These operating hours were multiplied by published small equipment rental rates along with operating costs and labor costs adjusted for local conditions. The division of this total cost and productivity resulted in the estimated unit cost by construction scenario. Likewise, the unit costs for low permeability fill were estimated accounting for phase specific haul profiles. In both cases, the average unit cost for all rock fill and low permeability fill was used for the cost estimate.

The unit cost for bofedal stripping was developed using the actual cost from a recent Peruvian project. This cost was escalated to account for lower machine productivity at high elevations along with an adjustment to account for the recent rise in diesel prices.

Both drain and filter unit costs were derived from a past project where the materials were produced onsite using a portable crushing and screening unit. These costs were scaled using the US Department of Labor Producer Price Index for crushed stone. Finally, an allowance was made to account for the lower machine productivity realized due to the project's high elevation.

The remaining unit costs, which represent thirteen percent, 13%, of the overall construction cost were primarily estimated from comparable projects with escalation factors and professional judgment along with some current price quotes for specialized equipment.

21.5.5 Mine Capital Cost

Mine capital cost for mobile equipment was developed from the mine equipment list presented in Section 16. Quotes for the mine equipment were obtained by the client staff in Lima. IMC reviewed the quotes against other recent data on file for reasonability.

Mine capital costs do include:

- All mine mobile equipment required to drill, blast, load, and haul the material from the pit to the appropriate destinations.
- Auxiliary equipment to maintain the mine and material storage areas in good working order as well as construct the mine haul roads and maintain them.
- Equipment to maintain the mine fleet such as tire handlers and forklifts.
- Light vehicles for mine operations and staff personnel.
- An allowance is included for initial shop tools.
- An allowance is included for initial spare parts inventory.
- Mine engineering equipment (computers, survey equipment etc.) is included.
- Equipment replacements are included as required based on the useful life of the equipment.

Mine capital costs DO NOT include:

1. Mine office buildings, or shop facilities. They are included elsewhere in the project capital list.
2. Mobile equipment that is not required by the mine. (i.e. no mobile units for the plant)
3. Infrastructure or process plant related costs

The equipment is shown as purchase in the year it is required for operation.

Table 21-12 presents the detailed purchase schedule for the mine equipment.

Table 21-13 summarizes the mine capital costs by year along with the mine operating costs.

Table 21-14 shows preproduction stripping as part of the mine capital cost. That cost is broken out separately to illustrate that it is not a cost to purchase mine equipment.

Table 21-12: Mine Mobile Equipment Capital, Preproduction Through Year 2

Mine Equipment Capital Costs , USD x $1,000, Preproduction Through Year 2

	Unit Cost	Life	PPQ-6		PPQ-5		PPQ-4		PPQ-3		PPQ-2		PPQ-1		Yr1Q1		Yr1Q2		Yr1Q3		Yr1Q4		Yr2Q1		Yr2Q2		Yr2Q3		Yr2Q4	
	($1000)	Years	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)
MINE MAJOR EQUIPMENT:																														
45,000 Lb Pulldown Drill	1,261	60,000	0	0	1	1,261	0	0	1	1,261	1	1,261	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
15 cu m Hydraulic Shovel	4,965	80,000	0	0	0	0	0	0	1	4,965	1	4,965	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
13.8 cu m Loader	2,624	30,000	0	0	1	2,624	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
135 mT Haul Truck	2,477	60,000	0	0	2	4,954	2	4,954	3	7,431	1	2,477	0	0	0	0	0	0	1	2,477	0	0	0	0	0	0	0	0	1	2,477
410 HP Track Dozer	906	30,000	3	2,718	0	0	0	0	0	0	0	0	0	0	1	906	0	0	0	0	0	0	0	0	0	0	0	0	0	0
498 HP Wheel Dozer	1,210	30,000	0	0	0	0	0	0	1	1,210	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
16 ft Motor Grader	823	30,000	2	1,646	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1	823	0	0	0	0	0	0
76,000 liter Water Truck	2,117	50,000	0	0	2	4,234	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
8.6 cu m Wheel Loader	1,722	30,000	1	1,722	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
40 tonne Articulated Truck	634	30,000	1	634	0	0	0	0	0	0	0	0	0	0	1	634	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pioneer Drill	731	40,000	1	731	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
2 cu m Excavator	335	20,000	1	335	0	0	0	0	0	0	0	0	0	0	1	335	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Subtotal Major Equipment				7,786		13,073		4,954		14,867		8,703		0		1,875		0		2,477		0		823		0		0		2,477
MINE SUPPORT EQUIPMENT:		Years																												
Blasthole Stemmer (skid steer)	43	4		0	1	43		0		0		0		0		0		0		0		0		0		0		0		0
Blasters Flatbed Truck (2 T)	54	6		0	1	54		0		0		0		0		0		0		0		0		0		0		0		0
ANFO/Slurry Truck (40,000 Lb)	630	6		0	1	630		0		0		0		0		0		0		0		0		0		0		0		0
Fuel/Lube Truck 5000 gal (19 k ltr)	404	6		0	2	808		0		0		0		0		0		0		0		0		0		0		0		0
Crane Truck (8 - 10 ton)	208	6		0	1	208		0		0		0		0		0		0		0		0		0		0		0		0
T62H Tire Handler	475	6		0	1	475		0		0		0		0		0		0		0		0		0		0		0		0
Mechanics Truck	290	6		0	3	870		0		0		0		0		0		0		0		0		0		0		0		0
Welding Truck	158	6		0	1	158		0		0		0		0		0		0		0		0		0		0		0		0
Shop Forklift (Hyster H100XM)	61	18		0	1	61		0		0		0		0		0		0		0		0		0		0		0		0
RT Forklift (Sellick SD-100)	81	9		0	1	81		0		0		0		0		0		0		0		0		0		0		0		0
Man Van	68	4		0	2	136		0		0		0		0		0		0		0		0		0		0		0		0
Pickup Truck (4x4)	26	4	2	52	6	156		0		0		0		0		0		0		0		0		0		0		0		0
light Plants	23	18		0		0	1	23	2	46	1	23	1	23	2	46		0		0		0	1	23		0		0		0
Mine Radios	1	6	11	11	27	27	2	2	6	6	3	3		0	3	3		0		0		0	1	1		0		0	1	1
Mine Communications Network	368	9		0	1	368		0		0		0		0		0		0		0		0		0		0		0		0
Water Pipe - Dewatering	23	5		0		0		0		0		0		0		0		0		0		0		0	1	23		0		0
Mine Pumps	23	5		0		0		0		0		0		0		0		0		0		0		0	1	23		0		0
Subtotal Mine Support Equipment				63		4,075		25		52		26		23		49		0		0		0		24		46		0		1
Engineering/Geology Equipment	150	6	1	150		0		0		0		0		0		0		0		0		0		0		0		0		0
Shop Tools (3% of Major Equipment)		3.0%		234		392		149		446		261		0		56		0		0		0				0		0		
Initial Spare Parts (5% of Major Equipment)		5.0%		389		654		248		743		435		0		94		0		0		0				0		0		
TOTAL EQUIPMENT/FACILITIES CAPITAL				8,622		18,194		5,375		16,108		9,425		23		2,074		0		2,477		0		847		46		0		2,478

Table 21-13: Mine Mobile Equipment Capital, Years 3 through 16

Mine Equipment Capital Costs , USD x $1,000, Years 3 through 16	Unit Cost	Life	3		4		5		6		7		8		9		10		11		12		13		14		15		16		Project
	($1000)	Years	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	No.	($1000)	Total
MINE MAJOR EQUIPMENT:																															
45,000 Lb Pulldown Drill	1,261	60,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	2	2,522	0	0	1	1,261	0	0	0	0	7,566
15 cu m Hydraulic Shovel	4,965	80,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	1	4,965	0	0	0	0	0	0	0	0	14,895
13.8 cu m Loader	2,624	30,000	0	0	1	2,624	0	0	0	0	0	0	0	0	0	0	0	0	1	2,624	0	0	0	0	0	0	0	0	0	0	7,872
135 mT Haul Truck	2,477	60,000	0	0	0	0	0	0	0	0	1	2,477	0	0	1	2,477	8	19,816	1	2,477	0	0	0	0	0	0	0	0	0	0	52,017
410 HP Track Dozer	906	30,000	0	0	0	0	0	0	3	2,718	0	0	1	906	0	0	0	0	0	0	0	0	0	0	3	2,718	0	0	0	0	9,966
498 HP Wheel Dozer	1,210	30,000	0	0	0	0	0	0	0	0	1	1,210	0	0	0	0	0	0	0	0	0	0	1	1,210	0	0	0	0	0	0	3,630
16 ft Motor Grader	823	30,000	0	0	0	0	0	0	1	823	1	823	0	0	0	0	1	823	0	0	0	0	1	823	1	823	0	0	0	0	6,584
76,000 liter Water Truck	2,117	50,000	0	0	0	0	0	0	0	0	0	0	0	0	1	2,117	0	0	1	2,117	0	0	0	0	0	0	0	0	0	0	8,468
8.6 cu m Wheel Loader	1,722	30,000	0	0	0	0	1	1,722	0	0	0	0	0	0	1	1,722	0	0	0	0	0	0	0	0	0	0	0	0	1	1,722	6,888
40 tonne Articulated Truck	634	30,000	0	0	0	0	0	0	0	0	2	1,268	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	2,536
Pioneer Drill	731	40,000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	731
2 cu m Excavator	335	20,000	0	0	0	0	0	0	0	0	1	335	0	0	0	0	0	0	0	0	1	335	0	0	0	0	0	0	0	0	1,340
Subtotal Major Equipment				0		2,624		1,722		3,541		6,113		906		6,316		20,639		7,218		7,822		2,033		4,802		0		1,722	122,493
MINE SUPPORT EQUIPMENT:		Years																													
Blasthole Stemmer (skid steer)	43	4		0	1	43		0		0		0	1	43		0		0		0	1	43		0		0		0	1	43	215
Blasters Flatbed Truck (2 T)	54	6		0		0		0	1	54		0		0		0		0		0	1	54		0		0		0	1	54	216
ANFO/Slurry Truck (40,000 Lb)	630	6		0		0		0	1	630		0		0		0		0		0	1	630		0		0		0	1	630	2,520
Fuel/Lube Truck 5000 gal (19 k ltr)	404	6		0		0		0	1	404		0		0		0		0		0	1	404		0		0		0	1	404	2,020
Crane Truck (8 - 10 ton)	208	6		0		0		0	1	208		0		0		0		0		0	1	208		0		0		0	1	208	832
T62H Tire Handler	475	6		0		0		0		0		0		0		0		0		0		0		0		0		0		0	475
Mechanics Truck	290	6		0		0		0	3	870		0		0		0		0		0	3	870		0		0		0	3	870	3,480
Welding Truck	158	6		0		0		0	1	158		0		0		0		0		0	1	158		0		0		0	1	158	632
Shop Forklift (Hyster H100XM)	61	18		0		0		0		0		0		0		0		0		0		0		0		0		0		0	61
RT Forklift (Sellick SD-100)	81	9		0		0		0		0		0	1	81		0		0		0		0		0		0		0		0	162
Man Van	68	4		0	2	136		0		0		0		0		0		0		0	2	136		0		0		0	2	136	544
Pickup Truck (4x4)	26	4		0	8	208		0		0		0	8	208		0		0		0	8	208		0		0		0	8	208	1,040
light Plants	23	18		0		0		0		0		0		0		0		0		0	1	23		0		0		0		0	207
Mine Radios	1	6		0	2	2		0	1	1	8	8	3	3	2	2	13	13	3	3	10	10		0		0		0		0	96
Mine Communications Network	368	9		0		0		0		0		0		0		0		0		0		0		0		0		0		0	368
Water Pipe - Dewatering	23	5		0		0		0	1	23		0	1	23		0		0	2	46		0		0		0	1	23		0	138
Mine Pumps	23	5		0		0		0	1	23		0	1	23		0		0	2	46		0		0		0	1	23		0	138
Subtotal Mine Support Equipment				0		389		0		2,371		8		381		2		13		95		2,744		0		0		46		2,711	13,144
Engineering/Geology Equipment	150	6		0		0		0	1	150		0		0		0		0		0	1	150		0		0		0		0	450
Shop Tools (3% of Major Equipment)		3.0%		0																								0			1,538
Initial Spare Parts (5% of Major Equipment)		5.0%		0																								0			2,563
TOTAL EQUIPMENT/FACILITIES CAPITAL				0		3,013		1,722		6,062		6,121		1,287		6,318		20,652		7,313		10,716		2,033		4,802		46		4,433	140,188

Table 21-14: Corani Project Summary of Mine Capital and Operating Costs

Year	Mine Equipment Initial Capital Cost	Mine Equipment Sustaining Capital Cost	Mine Preprod. Development	(1) Total Mine Capital	Mine Operating Cost	MINE TOTAL COST
PPQ-6	8,622		819	9,441		9,441
PPQ-5	18,194		1,774	19,967		19,967
PPQ-4	5,375		3,438	8,813		8,813
PPQ-3	16,108		5,567	21,676		21,676
PPQ-2	9,425		7,179	16,604		16,604
PPQ-1	23		7,283	7,306		7,306
Yr1Q1		2,074		2,074	7,819	9,893
Yr1Q2		0		0	7,698	7,698
Yr1Q3		2,477		2,477	7,912	10,389
Yr1Q4		0		0	7,723	7,723
Yr2Q1		847		847	7,907	8,754
Yr2Q2		46		46	7,994	8,040
Yr2Q3		0		0	7,981	7,981
Yr2Q4		2,478		2,478	8,363	10,841
3		0		0	33,425	33,425
4		3,013		3,013	33,811	36,824
5		1,722		1,722	33,909	35,631
6		6,062		6,062	32,686	38,748
7		6,121		6,121	34,469	40,590
8		1,287		1,287	35,281	36,568
9		6,318		6,318	34,852	41,170
10		20,652		20,652	36,294	56,946
11		7,313		7,313	36,031	43,344
12		10,716		10,716	36,159	46,875
13		2,033		2,033	29,827	31,860
14		4,802		4,802	31,411	36,213
15		46		46	29,806	29,852
16		4,433		4,433	26,974	31,407
17		0		0	26,834	26,834
18		0		0	22,839	22,839
19		0		0	10,809	10,809
20		0		0	8,338	8,338
TOTAL	57,748	82,440	26,060	166,248	597,152	763,400

(1) Preproduction Development is shown as capital on this table.

21.5.6 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 fourth-Quarter 2011;
- 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.

21.5.7 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%.

21.6 QUANTITIES AND PRICING METHODOLOGY

21.6.1 Quantities

Discipline engineers developed material take-off quantities (MTO's) for earthworks, concrete, steel, overland pipelines, and overland power-lines based on general arrangement drawings, site plans and plot plans developed for the Project.

21.6.2 Pricing Methodology

The 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.

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.

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.

Buildings

The process facility and ancillary buildings include the following:

- Primary Crushing Building;
- Concentrator Building;
- Administration Building;
- Medical Facility;
- Analytical Laboratory;
- Plant Maintenance Building; and
- Warehouse with Yard Storage.

The mine infrastructure and ancillary buildings include the following:

- Explosive Storages;
- Truck Shop;
- Truck Wash;
- Truck Fuel Storage and Fueling Station; and
- Tire Change Station.

The camp site includes the following:

- Dormitories;
- Kitchen and Dining;
- Laundry;
- Recreation Center;
- Store; and
- Administration Building.

The structural components (concrete & steel) for the buildings have been included in the discipline MTOs. The remaining components are factored from data on similar facilities.

Power Costs

The referential capital budget is presented in Table 21-15 below.

Table 21-15: Power Transmission Capital Cost Budget

ITEM	COST
Macusani connection substation	2,054,562
Power transmission line	7,684,173
Corani substation	3,459,641
Power transmission line to camp site	395,092
Power transmission line to Tailing Storage Facility	143,803
Power transmission line to fresh water impoundment	135,968
Total	**13,873,239**

These budgets include direct taxes that the project shall bare, excluding the general sales tax (IGV).

Contracted Indirect Costs

Indirect costs are based on the either the mechanical equipment cost or the total direct cost depending upon which is most appropriate. Percentages based on historical data of similar projects are applied to develop pricing.

Other Costs

Transportation costs have been included for the delivery of equipment and materials to the site. In general, it has been assumed that most equipment and bulk materials will be purchased outside of Peru and shipped to Matarani, where it will then be trucked to site.

A contingency of approximately 15 percent, or $59 M, has been included in the capital cost in recognition of the degree of detail on which the estimate is based.

The contingency allowance has been assessed by considering possible ranges of cost uncertainties for various elements of the estimate. Each element is assigned a percentage rate based on the best judgment of the project team.

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 concentrate, both of which contain 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.

The life of mine ore quantities and ore grades are presented in the table below.

Table 22-1: Life of Mine Ore Quantities and Ore Grade

	Tonnes (kt)	Zinc (%)	Lead (%)	Silver Grade (g/t)
Ore	156,130	0.49%	0.90%	53.79
Waste	264,069			
Total Material Mined	420,199			

22.3 PLANT PRODUCTION STATISTICS

The design basis for the process plant is 22,500 tonnes per day at 96% mill availability. The metal recoveries are projected to average 51.6% for zinc, 71.1% for lead and 64.2% for silver.

The estimated life of mine metal production is presented in Table 22-2.

Table 22-2: Metal Production

	Life of Mine
Zinc (million lbs)	876.0
Lead (million lbs)	2,213.4
Silver (million ozs)	173.3

Note: production values are prior to smelter deductions

22.4 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 22-3 and Table 22-4.

Table 22-3: Smelter Treatment Factors (Lead Concentrate)

Lead Concentrate	
Payable lead	95.0%
Minimum Deduction (%)	0%
Payable silver	95.0%
Minimum Deduction (oz/dmt)	1.61
Treatment charge ($/dmt)	$175.00
Refining charge – Ag ($/payable oz.)	$0.50
Lead Concentrate Transportation	
Concentrate Trucking and port ($/wmt)	$72.00
Concentrate Shipping ($/wmt)	$55.00
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 22-4: Smelter Treatment Factors (Zinc Concentrate)

Zinc Concentrate	
Payable zinc	85.0 %
Minimum Deduction (%)	8.0 %
Price Participation Basis ($/tonne metal)	$2,500.00
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)	$72.00
Concentrate Shipping ($/wmt)	$55.00
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

22.5 CAPITAL EXPENDITURE

22.5.1 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 $574.4 million. Approximately 88% of these expenditures will be incurred over a two-year period.

Presented in Table 22-5 is the initial capital.

Table 22-5: Initial Capital

	$ in millions
Mining	91.2
Process Plant	452.4
Owner's Cost	30.8
Total	**574.4**

22.5.2 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 $143.9 million, and is expected to be spent over the 18-year mine life as presented in Table 22-6.

Table 22-6: 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	4.6	3.4	-	3.0	1.7	6.1	6.1	1.3	6.3	20.7
Process Plant	1.9	5.6	1.4	1.3	1.4	7.5	7.3	6.9	7.3	18.8
Total	6.5	9.0	1.4	4.3	3.1	13.6	13.4	8.2	13.6	39.5

	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Total
Mining Equipment	7.3	10.7	2.0	4.8		4.4	-	-	-	82.4
Process Plant	0.3	0.2	0.2	0.6	0.2	0.4	0.2	-		61.5
Total	7.6	10.9	2.2	5.4	0.2	4.8	0.2	-		143.9

22.5.3 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 19% 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.

22.5.4 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	$0.85
Lead	$0.85
Silver	$18.00

22.5.5 Total Operating Cost

The average Total Operating Cost over the life of the mine is estimated to be $18.99 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 22-7 shows the estimated operating cost by area per tonne of ore processed.

Table 22-7: Life of Mine Operating Cost

Operating Cost	$/ore tonne
Mine	$3.82
Process Plant	$7.91
General Administration	$1.40
Smelting/Refining Treatment & Concentrate Transport	$5.85
Total Operating Cost	$18.99

Note: Total shown in table is inconsistent because of rounding of the inputs

22.6 TOTAL CASH COST

The average Total Cash Cost over the life of the mine is estimated to be $3.66 per ounce of payable silver, including reclamation and net of lead and zinc credits. Total Cash Cost for the project is summarized in Table 22-8 below.

Table 22-8: Life of Mine Total Cash Cost

Direct Mining Costs	$2,050,474
Transportation and Refining Charges	$913,937
Subtotal	**$2,964,411**
Lead Payable Revenue	($1,787,288)
Zinc Payable Revenue	($632,216)
Total Cash Cost, Net of Lead and Zinc Revenues	**$544,907**
Reclamation	$41,208
Total Cash Cost, Including Reclamation	**$586,115**
Payable Silver Ounces	160,171
Total Cash Cost per Ounce of Payable Silver, Net of Lead and Zinc Revenues	**$3.40**
Total Cash Cost per Ounce of Payable Silver, Including Reclamation and Net of Lead and Zinc Revenues, and Reclamation	**$3.66**

22.6.1 Salvage Value

A $6.9 million allowance for salvage value has been included in the cash flow analysis.

22.6.2 Reclamation & Closure

An allowance of $41.2 million for the cost of reclamation and closure of the property has been included in the cash flow projection.

22.7 DEPRECIATION

Depreciation was calculated using the following assumptions for both initial and sustaining capital.

- Buildings 20 year straight line method
- Mobile equipment 8 year straight line method
- All other assets units of production on a silver equivalent basis
- Last year of production is the catch up year if assets are not fully depreciated

22.8 TAXATION

22.8.1 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 $62.6 million of royalty tax will be paid during the life of the mine.

22.8.2 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 $44.3 million of special tax will be paid during the life of the mine.

22.8.3 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 $119.3 million of labor profit sharing tax will be paid during the life of the mine.

22.8.4 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 30%. It is estimated that $411.5 million of income taxes will be paid during the life of the mine.

22.9 NET INCOME AFTER TAX

Net income after taxes for the project amounts to $947.9 million.

22.10 PROJECT FINANCING

The financial model has been prepared on the assumption that the project will be financed 100% with equity.

22.11 NET PRESENT VALUE, INTERNAL RATE OF RETURN, PAYBACK

The economic analyses for the project are summarized below in Table 22-9.

Table 22-9: Financial Analysis Results

	Pre-Tax	After Tax
NPV @ 0% ($000)	$1,585.6	$947.9
NPV @ 5% ($000)	$906.9	$462.5
IRR	29.7%	17.6%
Payback (Years)	2.4	3.8

22.12 SENSITIVITY ANALYSIS

The results of the sensitivity analysis for the project both before taxes and after taxes are shown on Table 22-10 to Table 22-13 and Figure 22-1 to Figure 22-4**.**

Table 22-10: NPV Sensitivity Analysis @ 5% - Before Taxes

	NPV @ 5% - Before Taxes			
	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	$1,559,076	$673,083	$800,458	$1,379,639
10%	$1,232,985	$789,989	$853,676	$1,143,267
0%	$906,894	$906,894	$906,894	$906,894
-10%	$580,803	$1,023,800	$960,112	$670,522
-20%	$254,713	$1,140,705	$1,013,330	$434,149

Table 22-11: IRR% Sensitivity Analysis – Before Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	41.6%	25.8%	23.7%	38.9%
10%	35.9%	27.8%	26.5%	34.4%
0%	29.7%	29.7%	29.7%	29.7%
-10%	22.7%	31.5%	33.4%	24.6%
-20%	14.5%	33.2%	38.0%	19.0%

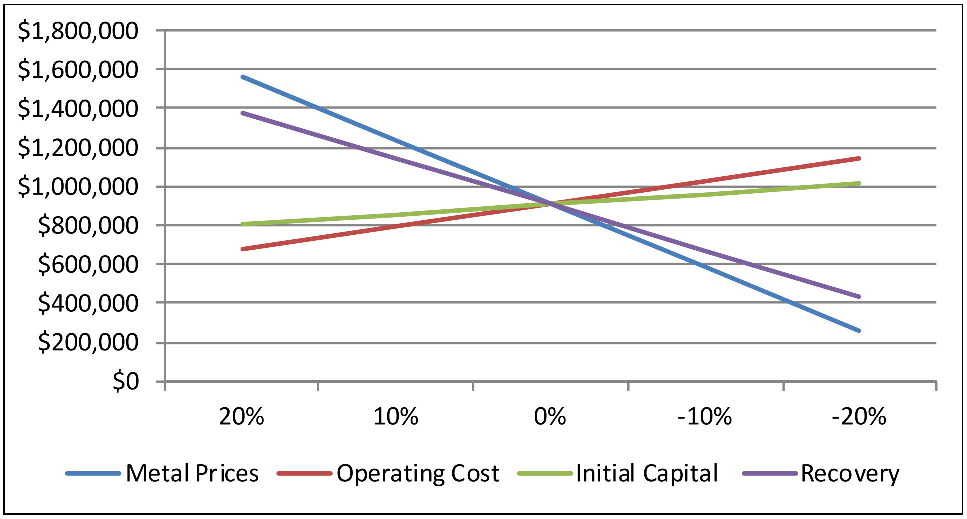


Figure 22-1: Sensitivity Analysis on NPV (before tax) at 5%

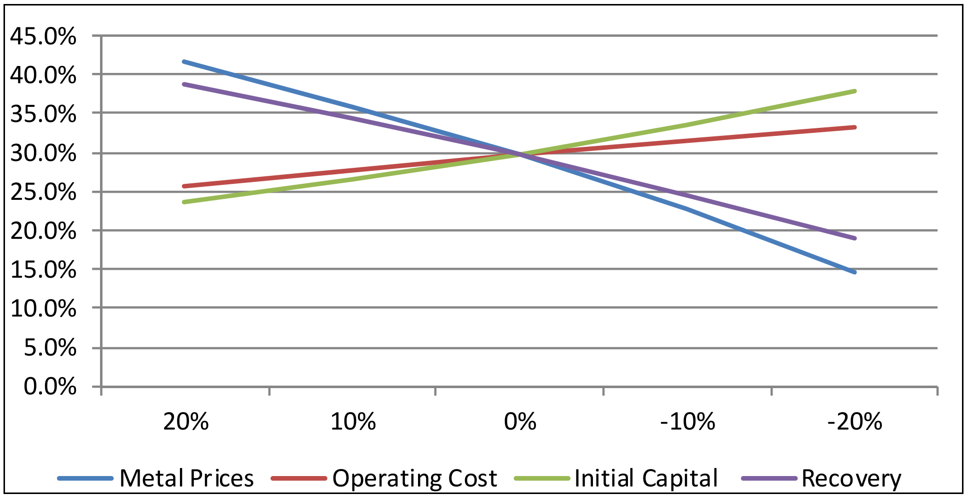


Figure 22-2: Sensitivity Analysis on IRR% (before tax)

Table 22-12: NPV Sensitivity Analysis @ 5% – After Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	$851,342	$312,182	$383,852	$745,557
10%	$657,756	$388,688	$423,224	$604,386
0%	$462,521	$462,521	$462,521	$462,521
-10%	$261,464	$533,855	$501,606	$318,001
-20%	$40,378	$604,528	$540,456	$167,874

Table 22-13: IRR% Sensitivity Analysis – After Taxes

	Metal Prices	Operating Cost	Initial Capital	Recovery
20%	25.5%	14.6%	13.9%	23.6%
10%	21.7%	16.1%	15.6%	20.7%
0%	17.6%	17.6%	17.6%	17.6%
-10%	12.9%	18.9%	19.9%	14.2%
-20%	6.5%	20.2%	22.8%	10.3%

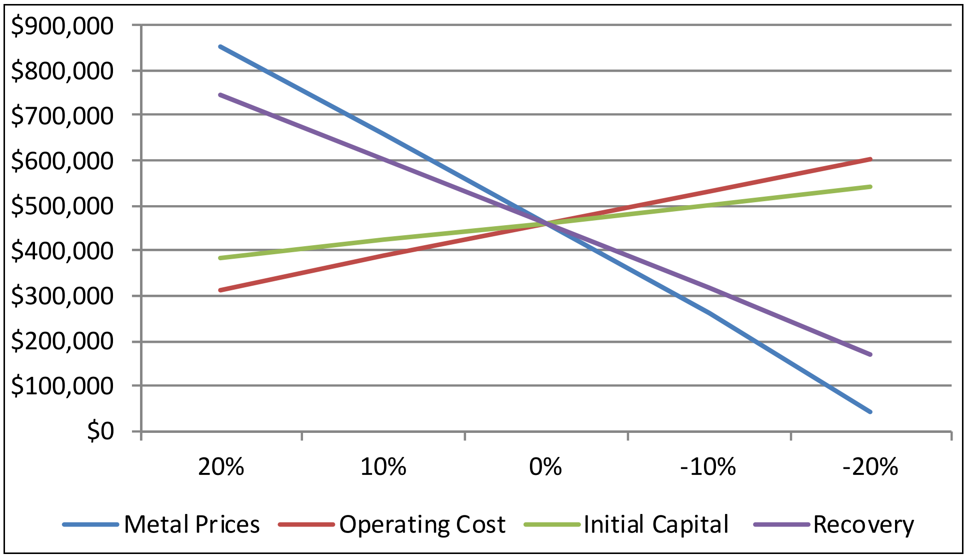


Figure 22-3: Sensitivity Analysis on NPV (after tax) at 5%

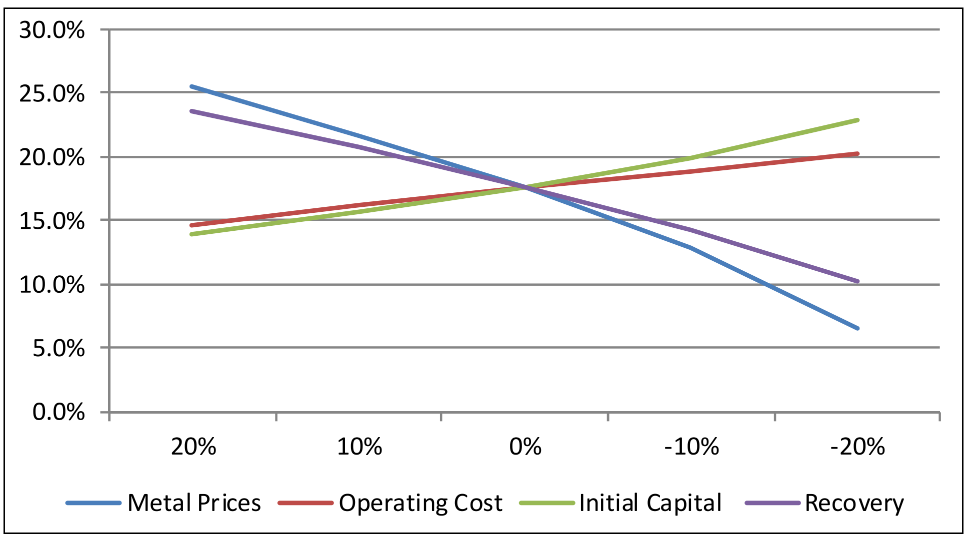


Figure 22-4: Sensitivity on IRR% (after tax)

Table 22-14: 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	19	20
Mining Operations																								
Ore to Mill																								
Beginning Inventory (kt)	139,573	139,573	139,573	139,573	138,769	131,698	123,823	115,948	108,073	100,198	92,323	84,448	76,573	68,698	60,823	52,948	45,073	37,198	29,323	21,448	13,573	5,698	-	-
Mined (kt)	139,573	-	-	804	7,071.000	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	5,698	-	-
Ending Inventory (kt)	-	139,573	139,573	138,769	131,698	123,823	115,948	108,073	100,198	92,323	84,448	76,573	68,698	60,823	52,948	45,073	37,198	29,323	21,448	13,573	5,698	-	-	-
Silver Grade (g/t)	56.18	-	-	104.78	106	86.13	84.31	80.53	77.95	65.53	61.84	44.75	35.30	53.93	47.67	55.99	34.81	34.79	37.98	34.91	29.11	33.37	-	-
Zinc Grade (%)	0.520%	0.000%	0.000%	0.060%	1.083%	0.525%	0.621%	0.647%	0.531%	0.449%	0.529%	0.322%	0.439%	0.470%	0.259%	0.201%	0.346%	0.376%	0.564%	0.571%	0.735%	0.890%	0.000%	0.000%
Lead Grade (%)	0.943%	0.000%	0.000%	1.046%	1.189%	1.268%	1.047%	1.263%	1.496%	1.175%	0.998%	0.805%	0.969%	0.811%	0.828%	0.845%	0.615%	0.782%	0.861%	0.632%	0.670%	0.660%	0.000%	0.000%
Contained Silver (kozs)	252,103	-	-	2,709	24,140	21,807	21,346	20,389	19,736	16,590	15,656	11,331	8,939	13,655	12,070	14,175	8,814	8,808	9,616	8,839	7,370	6,113	-	-
Contained Zinc (klbs)	1,598,629	-	-	1,072	168,772	91,102	107,767	112,408	92,229	77,942	91,809	55,966	76,260	81,564	45,016	34,848	60,069	65,261	97,899	99,173	127,651	111,821	-	-
Contained Lead (klbs)	2,902,002	-	-	18,544	185,413	220,216	181,786	219,213	259,777	203,968	173,209	139,686	168,249	140,815	143,674	146,723	106,719	135,697	149,532	109,660	116,271	82,850	-	-
Ore to Stockpile																								
Beginning Inventory (kt)	16,557	16,557	16,557	16,557	16,512	15,874	13,123	10,652	6,603	4,001	1,391	325	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	16,557	-	-	45	638	2,751	2,471	4,049	2,602	2,610	1,066	325	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	16,557	16,557	16,512	15,874	13,123	10,652	6,603	4,001	1,391	325	-	-	-	-	-	-	-	-	-	-	-	-	-
Silver Grade (g/t)	33.61	-	-	60.59	46.03520	31.66	41.75	37.35	30.57	27.95	23.08	17.65	-	-	-	-	-	-	-	-	-	-	-	-
Zinc Grade (%)	0.27%	-	-	0.03%	0.1110%	0.20%	0.28%	0.29%	0.32%	0.28%	0.30%	0.34%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Lead Grade (%)	0.58%	-	-	0.61%	0.4021%	0.62%	0.51%	0.59%	0.63%	0.61%	0.50%	0.49%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Contained Silver (kozs)	17,890	-	-	88	944	2,801	3,317	4,862	2,557	2,346	791	184	-	-	-	-	-	-	-	-	-	-	-	-
Contained Zinc (klbs)	99,652	-	-	25	1,562	12,411	15,400	25,802	18,541	16,342	7,149	2,421	-	-	-	-	-	-	-	-	-	-	-	-
Contained Lead (klbs)	210,558	-	-	600	5,656	37,642	27,767	52,635	36,185	34,913	11,658	3,501	-	-	-	-	-	-	-	-	-	-	-	-
Total Ore																								
Beginning Inventory (kt)	156,130	156,130	156,130	156,130	155,281	147,572	136,946	126,600	114,676	104,199	93,714	84,773	76,573	68,698	60,823	52,948	45,073	37,198	29,323	21,448	13,573	5,698	-	-
Mined (kt)	156,130	-	-	849	7,709	10,626	10,346	11,924	10,477	10,485	8,941	8,200	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	7,875	5,698	-	-
Ending Inventory (kt)	-	156,130	156,130	155,281	147,572	136,946	126,600	114,676	104,199	93,714	84,773	76,573	68,698	60,823	52,948	45,073	37,198	29,323	21,448	13,573	5,698	-	-	-
Silver Grade (g/t)	53.79	-	-	102.44	101.21	72.03	74.15	65.87	66.18	56.17	57.22	43.68	35.30	53.93	47.67	55.99	34.81	34.79	37.98	34.91	29.11	33.37	-	-
Zinc Grade (%)	0.49%	0.00%	0.00%	0.06%	1.00%	0.44%	0.54%	0.53%	0.48%	0.41%	0.50%	0.32%	0.44%	0.47%	0.26%	0.20%	0.35%	0.38%	0.56%	0.57%	0.74%	0.89%	0.00%	0.00%
Lead Grade (%)	0.90%	0.00%	0.00%	1.02%	1.12%	1.10%	0.92%	1.03%	1.28%	1.03%	0.94%	0.79%	0.97%	0.81%	0.83%	0.85%	0.61%	0.78%	0.86%	0.63%	0.67%	0.66%	0.00%	0.00%
Contained Silver (kozs)	269,993	-	-	2,796	25,084	24,608	24,664	25,251	22,293	18,936	16,447	11,515	8,939	13,655	12,070	14,175	8,814	8,808	9,616	8,839	7,370	6,113	-	-
Contained Zinc (klbs)	1,698,281	-	-	1,097	170,333	103,513	123,167	138,210	110,770	94,284	98,958	58,387	76,260	81,564	45,016	34,848	60,069	65,261	97,899	99,173	127,651	111,821	-	-
Contained Lead (klbs)	3,112,560	-	-	19,145	191,070	257,859	209,553	271,849	295,961	238,881	184,868	143,187	168,249	140,815	143,674	146,723	106,719	135,697	149,532	109,660	116,271	82,850	-	-
Waste																								
Beginning Inventory(kt)	264,069	264,069	264,069	264,069	247,918	233,127	221,253	209,099	198,523	186,500	174,485	160,426	144,626	128,501	112,376	97,251	82,126	67,501	53,376	39,251	25,126	11,257	-	-
Mined (kt)	264,069	-	-	16,151	14,791	11,874	12,154	10,576	12,023	12,015	14,059	15,800	16,125	16,125	15,125	15,125	14,625	14,125	14,125	14,125	13,869	11,257	-	-
Ending Inventory (kt)	-	264,069	264,069	247,918	233,127	221,253	209,099	198,523	186,500	174,485	160,426	144,626	128,501	112,376	97,251	82,126	67,501	53,376	39,251	25,126	11,257	-	-	-
Total Material Mined (kt)	420,199	-	-	17,000	22,500	22,500	22,500	22,500	22,500	22,500	23,000	24,000	24,000	24,000	23,000	23,000	22,500	22,000	22,000	22,000	21,744	16,955	-	-
Waste to Ore Ratio	1.69	-	-	19.02	1.92	1.12	1.17	0.89	1.15	1.15	1.57	1.93	2.05	2.05	1.92	1.92	1.86	1.79	1.79	1.79	1.76	1.98	-	-
Process Plant Operations																								
Mixed Sulfide Ore Milled																								
Ore Processed (kt)	131,428	-	-	-	7,277	7,148	6,434	5,721	6,946	6,605	5,717	6,580	7,871	7,822	4,050	5,980	7,589	7,088	6,370	7,702	7,874	7,139	5,211	4,304
Silver Grade (g/t)	51.46	-	-	-	106.95	86.29	85.30	77.90	78.23	63.07	56.10	41.34	35.30	53.86	37.81	53.87	34.23	31.93	31.08	34.35	29.11	31.77	25.45	25.45
Zinc Grade (%)	0.55%	0.00%	0.00%	0.00%	1.05%	0.57%	0.72%	0.78%	0.56%	0.49%	0.68%	0.36%	0.44%	0.47%	0.33%	0.23%	0.35%	0.40%	0.67%	0.58%	0.74%	0.78%	0.33%	0.33%
Lead Grade (%)	0.90%	0.00%	0.00%	0.00%	1.20%	1.27%	1.04%	1.28%	1.52%	1.15%	0.97%	0.77%	0.97%	0.81%	0.77%	0.82%	0.62%	0.76%	0.83%	0.63%	0.67%	0.63%	0.53%	0.53%
Contained Silver (kozs)	217,454	-	-	-	25,022	19,830	17,646	14,329	17,469	13,394	10,311	8,746	8,933	13,545	4,924	10,358	8,353	7,277	6,365	8,505	7,368	7,292	4,264	3,522
Contained Zinc (klbs)	1,605,079	-	-	-	169,220	89,513	101,509	98,842	85,858	71,840	85,912	52,514	76,255	81,254	29,173	30,035	59,110	62,486	93,428	98,930	127,640	122,299	37,931	31,332
Contained Lead (klbs)	2,594,041	-	-	-	191,846	200,148	147,937	161,587	232,496	168,182	122,516	111,552	168,189	139,350	68,634	107,540	103,301	119,045	116,184	107,203	116,257	99,843	61,461	50,769
Zinc Concentrate																								
Ag Recovery (%)	4.84%	0.00%	0.00%	0.00%	5.94%	3.72%	6.69%	7.11%	5.54%	4.20%	5.33%	2.55%	3.69%	4.17%	2.21%	1.16%	2.41%	3.53%	5.89%	6.43%	8.20%	6.62%	2.76%	2.76%
Zinc Recovery (%)	54.58%	0.00%	0.00%	0.00%	62.70%	55.91%	58.99%	58.39%	52.15%	51.67%	58.15%	43.80%	51.89%	57.67%	43.41%	30.47%	44.31%	44.61%	54.46%	54.00%	60.60%	59.83%	43.75%	43.75%
Zinc Concentrate (kt)	750	-	-	-	91	43	51	49	38	32	43	20	34	40	11	8	22	24	44	46	66	63	14	12
Zinc Concentrate Grade (%)	53.00%	0.00%	0.00%	0.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%
Recovered Silver (kozs)	10,531	-	-	-	1,486	738	1,180	1,018	969	562	549	223	329	565	109	120	201	257	375	547	604	483	118	97
Recovered Zinc (klbs)	876,012	-	-	-	106,099	50,043	59,876	57,710	44,771	37,120	49,955	23,001	39,567	46,859	12,664	9,151	26,192	27,876	50,882	53,419	77,349	73,174	16,595	13,708
Lead Concentrate																								
Ag Recovery (%)	62.00%	0.00%	0.00%	0.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%	62.00%
Lead Recovery (%)	75.00%	0.00%	0.00%	0.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%	75.00%
Lead Concentrate (kt)	1,471	-	-	-	109	113	84	92	132	95	69	63	95	79	39	61	59	67	66	61	66	57	35	29
Lead Concentrate Grade (%)	60.00%	0.00%	0.00%	0.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%	60.00%
Recovered Silver (kozs)	134,821	-	-	-	15,514	12,295	10,940	8,884	10,831	8,304	6,393	5,423	5,538	8,398	3,053	6,422	5,179	4,512	3,946	5,273	4,568	4,521	2,644	2,184
Recovered Lead (klbs)	1,945,532	-	-	-	143,885	150,112	110,954	121,191	174,373	126,136	91,887	83,664	126,141	104,512	51,476	80,655	77,476	89,284	87,138	80,402	87,192	74,883	46,096	38,077
Transition Ore Milled																								
Ore Processed (kt)	24,702	-	-	-	598	727	1,441	2,154	929	1,270	2,158	1,295	4	53	3,825	1,895	286	787	1,505	173	1	736	2,664	2,201
Silver Grade (g/t)	66.16	-	-	-	95.00	84.59	79.88	87.51	75.87	78.27	77.04	62.08	46.20	64.30	58.11	62.66	50.14	60.50	67.20	60.07	59.36	49.56	49.56	49.56
Zinc Grade (%)	0.17%	0.00%	0.00%	0.00%	0.05%	0.10%	0.20%	0.29%	0.31%	0.22%	0.12%	0.12%	0.11%	0.27%	0.19%	0.12%	0.15%	0.16%	0.13%	0.06%	0.08%	0.16%	0.16%	0.16%
Lead Grade (%)	0.95%	0.00%	0.00%	0.00%	0.92%	1.25%	1.07%	1.21%	1.33%	1.28%	1.07%	0.99%	0.72%	1.26%	0.89%	0.94%	0.54%	0.96%	1.01%	0.64%	0.72%	0.66%	0.66%	0.66%
Contained Silver (kozs)	52,541	-	-	-	1,827	1,977	3,701	6,060	2,266	3,196	5,345	2,585	6	110	7,147	3,817	461	1,531	3,251	334	2	1,174	4,245	3,507
Contained Zinc (klbs)	93,129	-	-	-	605	1,577	6,260	13,568	6,366	6,107	5,900	3,450	10	311	15,842	4,812	953	2,778	4,466	240	2	2,614	9,457	7,812
Contained Lead (klbs)	518,513	-	-	-	12,107	20,063	33,849	57,626	27,275	35,787	50,692	28,132	63	1,467	75,041	39,183	3,416	16,653	33,348	2,456	16	10,695	38,687	31,957
Zinc Concentrate																								
Ag Recovery (%)	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Zinc Recovery (%)	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Zinc Concentrate (kt)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Zinc Concentrate Grade (%)	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
Recovered Silver (kozs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Recovered Zinc (klbs)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Lead Concentrate																								
Ag Recovery (%)	53.20%	0.00%	0.00%	0.00%	58.76%	56.73%	55.32%	57.27%	54.57%	54.95%	55.94%	51.82%	47.74%	52.69%	50.90%	51.83%	48.62%	51.32%	52.73%	51.02%	50.37%	48.69%	48.69%	48.69%
Lead Recovery (%)	51.65%	0.00%	0.00%	0.00%	50.47%	54.10%	53.12%	54.87%	55.33%	54.66%	52.39%	50.33%	45.82%	57.14%	50.60%	49.66%	47.71%	52.56%	53.28%	48.49%	45.84%	47.02%	47.02%	47.02%
Lead Concentrate (kt)	304	-	-	-	7	12	20	36	17	22	30	16	0	1	43	22	2	10	20	1	0	6	21	17
Lead Concentrate Grade (%)	40.00%	0.00%	0.00%	0.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%	40.00%
Recovered Silver (kozs)	27,954	-	-	-	1,073	1,122	2,047	3,470	1,237	1,756	2,990	1,339	3	58	3,638	1,979	224	786	1,715	170	1	571	2,067	1,707
Recovered Lead (klbs)	267,827	-	-	-	6,111	10,855	17,980	31,622	15,092	19,561	26,556	14,160	29	838	37,971	19,459	1,630	8,753	17,767	1,191	7	5,029	18,191	15,026
Total Zinc Concentrate																								
Zinc Concentrate (kt)	750	-	-	-	91	43	51	49	38	32	43	20	34	40	11	8	22	24	44	46	66	63	14	12
Zinc Concentrate Grade (%)	53.00%	0.00%	0.00%	0.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%	53.00%
Recovered Silver (kozs)	10,531	-	-	-	1,486	738	1,180	1,018	969	562	549	223	329	565	109	120	201	257	375	547	604	483	118	97
Recovered Zinc (klbs)	876,012	-	-	-	106,099	50,043	59,876	57,710	44,771	37,120	49,955	23,001	39,567	46,859	12,664	9,151	26,192	27,876	50,882	53,419	77,349	73,174	16,595	13,708
Total Lead Concentrate																								
Lead Concentrate (kt)	1,775	-	-	-	116	126	104	127	149	118	100	79	95	80	82	83	60	77	86	62	66	62	55	46
Lead Concentrate Grade (%)	56.58%	0.00%	0.00%	0.00%	58.80%	58.04%	56.09%	54.37%	57.70%	56.23%	53.95%	55.95%	59.99%	59.76%	49.49%	54.69%	59.39%	57.44%	55.32%	59.57%	60.00%	58.17%	52.56%	52.56%
Recovered Silver (kozs)	162,775	-	-	-	16,587	13,416	12,988	12,354	12,068	10,061	9,383	6,762	5,541	8,456	6,691	8,401	5,403	5,298	5,661	5,443	4,569	5,092	4,711	3,891
Recovered Lead (klbs)	2,213,359	-	-	-	149,996	160,966	128,934	152,813	189,464	145,697	118,443	97,824	126,170	105,350	89,447	100,114	79,106	98,037	104,904	81,593	87,199	79,912	64,287	53,103
Payable Metals																								
Zinc Concentrate																								
Payable Silver (kozs)	5,535	-	-	-	818	412	701	592	584	316	280	108	147	297	50	65	86	122	156	271	261	184	48	39
Payable Zinc (klbs)	743,784	-	-	-	90,084	42,490	50,838	48,999	38,013	31,517	42,415	19,529	33,594	39,786	10,752	7,770	22,239	23,668	43,202	45,356	65,674	62,129	14,090	11,639
Lead Concentrates																								
Payable Silver (kozs)	154,636	-	-	-	15,757	12,746	12,338	11,737	11,464	9,558	8,913	6,424	5,264	8,033	6,356	7,981	5,133	5,033	5,378	5,171	4,341	4,838	4,475	3,697
Payable Lead (klbs)	2,102,691	-	-	-	142,496	152,918	122,487	145,172	179,991	138,412	112,521	92,933	119,862	100,083	84,974	95,108	75,151	93,135	99,659	77,513	82,839	75,916	61,073	50,448
Income Statement ($000)																								
Zinc ($/lb.)	$ 0.85	$ -	$ -	$ -	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85
Lead ($/lb)	$ 0.85	$ -	$ -	$ -	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85	$ 0.85
Silver ($/oz)	$ 18.00	$ -	$ -	$ -	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00	$ 18.00
Revenues																								
Zinc Concentrates - Zn	632,216	$ -	$ -	$ -	$ 76,571	$ 36,116	$ 43,212	$ 41,649	$ 32,311	$ 26,790	$ 36,052	$ 16,600	$ 28,555	$ 33,818	$ 9,140	$ 6,604	$ 18,903	$ 20,118	$ 36,721	$ 38,553	$ 55,823	$ 52,810	$ 11,977	$ 9,893
Zinc Concentrates - Ag	99,633	$ -	$ -	$ -	$ 14,724	$ 7,413	$ 12,612	$ 10,655	$ 10,514	$ 5,685	$ 5,033	$ 1,942	$ 2,654	$ 5,347	$ 893	$ 1,166	$ 1,549	$ 2,187	$ 2,807	$ 4,879	$ 4,690	$ 3,318	$ 856	$ 707
Lead Concentrates - Pb	1,787,288	$ -	$ -	$ -	$ 121,122	$ 129,980	$ 104,114	$ 123,396	$ 152,992	$ 117,650	$ 95,643	$ 78,993	$ 101,883	$ 85,070	$ 72,228	$ 80,842	$ 63,878	$ 79,165	$ 84,710	$ 65,886	$ 70,413	$ 64,529	$ 51,912	$ 42,881
Lead Concentrates - Ag	2,783,451	$ -	$ -	$ -	$ 283,633	$ 229,420	$ 222,091	$ 211,260	$ 206,358	$ 172,037	$ 160,442	$ 115,629	$ 94,751	$ 144,596	$ 114,411	$ 143,650	$ 92,392	$ 90,591	$ 96,801	$ 93,083	$ 78,134	$ 87,077	$ 80,556	$ 66,541
Total Revenues	5,302,588	$ -	$ -	$ -	$ 496,050	$ 402,930	$ 382,029	$ 386,960	$ 402,176	$ 322,162	$ 297,170	$ 213,164	$ 227,843	$ 268,831	$ 196,672	$ 232,263	$ 176,722	$ 192,060	$ 221,040	$ 202,401	$ 209,060	$ 207,733	$ 145,300	$ 120,023
Operating Cost																								
Mining	597,152	$ -	$ -	$ -	$ 31,152	$ 32,245	$ 33,425	$ 33,811	$ 33,909	$ 32,686	$ 34,469	$ 35,281	$ 34,852	$ 36,294	$ 36,031	$ 36,159	$ 29,827	$ 31,411	$ 29,806	$ 26,974	$ 26,834	$ 22,839	$ 10,809	$ 8,338
Process Plant	1,234,740	$ -	$ -	$ -	$ 60,483	$ 60,483	$ 60,483	$ 60,483	$ 60,483	$ 60,483	$ 60,483	$ 60,483	$ 60,483	$ 60,483	$ 64,106	$ 64,106	$ 64,106	$ 64,106	$ 64,106	$ 64,106	$ 64,106	$ 64,106	$ 64,106	$ 52,953
General Administration	218,582	$ -	$ -	$ -	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 11,025	$ 9,107
Treatment & Refining Charges																								
Zinc Concentrates																								
Treatment Charges	171,686	$ -	$ -	$ -	$ 20,794	$ 9,808	$ 11,735	$ 11,310	$ 8,775	$ 7,275	$ 9,791	$ 4,508	$ 7,755	$ 9,184	$ 2,482	$ 1,794	$ 5,133	$ 5,463	$ 9,972	$ 10,469	$ 15,159	$ 14,341	$ 3,252	$ 2,687
Price Participation	(5,607)	$ -	$ -	$ -	$ (679)	$ (320)	$ (383)	$ (369)	$ (287)	$ (238)	$ (320)	$ (147)	$ (253)	$ (300)	$ (81)	$ (59)	$ (168)	$ (178)	$ (326)	$ (342)	$ (495)	$ (468)	$ (106)	$ (88)
Penalty	3,243	$ -	$ -	$ -	$ 393	$ 185	$ 222	$ 214	$ 166	$ 137	$ 185	$ 85	$ 146	$ 173	$ 47	$ 34	$ 97	$ 103	$ 188	$ 198	$ 286	$ 271	$ 61	$ 51
Transportation	102,832	$ -	$ -	$ -	$ 12,455	$ 5,874	$ 7,029	$ 6,774	$ 5,256	$ 4,357	$ 5,864	$ 2,700	$ 4,645	$ 5,501	$ 1,487	$ 1,074	$ 3,075	$ 3,272	$ 5,973	$ 6,271	$ 9,080	$ 8,590	$ 1,948	$ 1,609
Lead Concentrate																								
Treatment Charges	310,539	$ -	$ -	$ -	$ 20,248	$ 22,014	$ 18,247	$ 22,309	$ 26,064	$ 20,569	$ 17,426	$ 13,879	$ 16,694	$ 13,993	$ 14,345	$ 14,532	$ 10,573	$ 13,549	$ 15,054	$ 10,873	$ 11,537	$ 10,905	$ 9,708	$ 8,019
Price Participation	-	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Penalty	10,534	$ -	$ -	$ -	$ 687	$ 747	$ 619	$ 757	$ 884	$ 698	$ 591	$ 471	$ 566	$ 475	$ 487	$ 493	$ 359	$ 460	$ 511	$ 369	$ 391	$ 370	$ 329	$ 272
Silver Refining Charges	77,318				$ 7,879	$ 6,373	$ 6,169	$ 5,868	$ 5,732	$ 4,779	$ 4,457	$ 3,212	$ 2,632	$ 4,017	$ 3,178	$ 3,990	$ 2,566	$ 2,516	$ 2,689	$ 2,586	$ 2,170	$ 2,419	$ 2,238	$ 1,848
Transportation	243,392	$ -	$ -	$ -	$ 15,870	$ 17,254	$ 14,301	$ 17,485	$ 20,428	$ 16,122	$ 13,658	$ 10,878	$ 13,084	$ 10,967	$ 11,243	$ 11,390	$ 8,287	$ 10,619	$ 11,799	$ 8,522	$ 9,042	$ 8,547	$ 7,609	$ 6,285
Stockpile inventory adjustment	(0)	$ -	$ -	$ -	$ (1,086)	$ (4,376)	$ (3,931)	$ (6,441)	$ (4,139)	$ (4,152)	$ (1,696)	$ (517)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,463	$ 12,527	$ 10,347
Total Operating Cost	2,964,412	-	-	-	179,220	161,311	158,942	163,226	168,296	153,742	155,934	141,857	151,629	151,812	144,350	144,538	134,881	142,347	150,797	141,051	149,136	146,407	123,507	101,430
Salvage Value	(6,903)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Reclamation & Closure	41,208	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Total Production Cost	2,998,717	$ -	$ -	$ -	$ 179,220	$ 161,311	$ 158,942	$ 163,226	$ 168,296	$ 153,742	$ 155,934	$ 141,857	$ 151,629	$ 151,812	$ 144,350	$ 144,538	$ 134,881	$ 142,347	$ 150,797	$ 141,051	$ 149,136	$ 146,407	$ 123,507	$ 101,430
Operating Income	$ 2,303,871	$ -	$ -	$ -	$ 316,830	$ 241,618	$ 223,088	$ 223,734	$ 233,880	$ 168,419	$ 141,236	$ 71,307	$ 76,214	$ 117,019	$ 52,322	$ 87,726	$ 41,841	$ 49,714	$ 70,243	$ 61,350	$ 59,924	$ 61,326	$ 21,794	$ 18,592
Depreciation on BC Sucursal Cost	$ 23,241				$ 2,174	$ 1,766	$ 1,674	$ 1,696	$ 1,763	$ 1,412	$ 1,302	$ 934	$ 999	$ 1,178	$ 862	$ 1,018	$ 775	$ 842	$ 969	$ 887	$ 916	$ 910	$ 637	$ 526
Depreciation	$ 718,317				$ 60,200	$ 51,170	$ 49,053	$ 49,938	$ 51,733	$ 44,348	$ 42,638	$ 34,248	$ 28,519	$ 34,828	$ 28,403	$ 33,019	$ 27,385	$ 28,841	$ 30,990	$ 29,514	$ 29,361	$ 26,632	$ 19,308	$ 18,189
Total Depreciation	$ 741,558	$ -	$ -	$ -	$ 62,374	$ 52,936	$ 50,728	$ 51,634	$ 53,495	$ 45,760	$ 43,940	$ 35,183	$ 29,518	$ 36,006	$ 29,265	$ 34,037	$ 28,160	$ 29,683	$ 31,959	$ 30,401	$ 30,277	$ 27,543	$ 19,945	$ 18,715
Net Income After Depreciation	$ 1,562,313	$ -	$ -	$ -	$ 254,456	$ 188,682	$ 172,360	$ 172,100	$ 180,384	$ 122,660	$ 97,296	$ 36,124	$ 46,696	$ 81,013	$ 23,057	$ 53,689	$ 13,681	$ 20,031	$ 38,284	$ 30,949	$ 29,647	$ 33,783	$ 1,849	$ (123)
Interest Expense	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Net Income After Interest	$ 1,562,313	$ -	$ -	$ -	$ 254,456	$ 188,682	$ 172,360	$ 172,100	$ 180,384	$ 122,660	$ 97,296	$ 36,124	$ 46,696	$ 81,013	$ 23,057	$ 53,689	$ 13,681	$ 20,031	$ 38,284	$ 30,949	$ 29,647	$ 33,783	$ 1,849	$ (123)
Royalty Tax	$ 62,609	-	-	-	8,696	5,885	5,191	5,111	5,402	3,222	2,972	2,132	2,278	2,688	1,967	2,323	1,767	1,921	2,210	2,024	2,091	2,077	1,453	1,200
IEM (special tax)	$ 44,275	-	-	-	8,175	5,760	5,164	5,117	5,387	3,371	2,497	746	1,012	2,010	447	1,210	261	381	796	624	592	690	37	-
Worker's Participation Tax	$ 119,285	-	-	-	19,007	14,163	12,960	12,950	13,568	9,285	7,346	2,660	3,473	6,105	1,651	4,012	932	1,418	2,822	2,264	2,157	2,481	29	-
Income Taxes at 30%	$ 411,532	$ -	$ -	$ -	$ 65,574	$ 48,862	$ 44,713	$ 44,677	$ 46,808	$ 32,034	$ 25,344	$ 9,176	$ 11,980	$ 21,063	$ 5,697	$ 13,843	$ 3,216	$ 4,893	$ 9,737	$ 7,811	$ 7,442	$ 8,561	$ 99	$ -
Total Taxes	$ 637,700	$ -	$ -	$ -	$ 101,451	$ 74,670	$ 68,029	$ 67,854	$ 71,165	$ 47,913	$ 38,159	$ 14,713	$ 18,743	$ 31,866	$ 9,763	$ 21,388	$ 6,176	$ 8,613	$ 15,565	$ 12,723	$ 12,282	$ 13,809	$ 1,618	$ 1,200
Net Income After Taxes	$ 924,613	-	-	-	153,005	114,012	104,331	104,246	109,219	74,747	59,136	21,410	27,954	49,147	13,294	32,301	7,505	11,418	22,719	18,226	17,366	19,975	231	(1,323)
Cash Flow																								
Operating Income after Depreciation & Interest	$ 1,562,313	$ -	$ -	$ -	$ 254,456	$ 188,682	$ 172,360	$ 172,100	$ 180,384	$ 122,660	$ 97,296	$ 36,124	$ 46,696	$ 81,013	$ 23,057	$ 53,689	$ 13,681	$ 20,031	$ 38,284	$ 30,949	$ 29,647	$ 33,783	$ 1,849	$ (123)
Add back Depreciation	$ 741,558	$ -	$ -	$ -	$ 62,374	$ 52,936	$ 50,728	$ 51,634	$ 53,495	$ 45,760	$ 43,940	$ 35,183	$ 29,518	$ 36,006	$ 29,265	$ 34,037	$ 28,160	$ 29,683	$ 31,959	$ 30,401	$ 30,277	$ 27,543	$ 19,945	$ 18,715
Working Capital																								
Account Recievable (60 days)	$ -	$ -	$ -	$ -	$ (81,542)	$ 15,308	$ 3,436	$ (811)	$ (2,501)	$ 13,153	$ 4,108	$ 13,809	$ (2,413)	$ (6,738)	$ 11,862	$ (5,851)	$ 9,130	$ (2,521)	$ (4,764)	$ 3,064	$ (1,095)	$ 218	$ 10,263	$ 4,155
Accounts Payable (30 days)	$ -	$ -	$ -	$ -	$ 14,820	$ (1,202)	$ (231)	$ 558	$ 228	$ (1,195)	$ (22)	$ (1,254)	$ 761	$ 15	$ (613)	$ 15	$ (794)	$ 614	$ 695	$ (801)	$ 664	$ (509)	$ (2,627)	$ (1,635)
IGV Payment	$ (113,145)	$ (1,053)	$ (10,231)	$ (13,347)	$ (4,428)	$ (4,474)	$ (4,525)	$ (4,541)	$ (4,545)	$ (4,493)	$ (4,569)	$ (4,604)	$ (4,586)	$ (4,647)	$ (4,800)	$ (4,805)	$ (4,534)	$ (4,602)	$ (4,534)	$ (4,413)	$ (4,407)	$ (4,236)	$ (3,721)	$ (3,049)
IGV Refund	$ 113,145		$ 1,053	$ 10,231	$ 13,347	$ 4,428	$ 4,474	$ 4,525	$ 4,541	$ 4,545	$ 4,493	$ 4,569	$ 4,604	$ 4,586	$ 4,647	$ 4,800	$ 4,805	$ 4,534	$ 4,602	$ 4,534	$ 4,413	$ 4,407	$ 4,236	$ 3,721
Stockpile inventory adjustment	$ (0)	$ -	$ -	$ -	$ (1,086)	$ (4,376)	$ (3,931)	$ (6,441)	$ (4,139)	$ (4,152)	$ (1,696)	$ (517)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,463	$ 12,527	$ 10,347
Inventory - Parts, Supplies	$ -	$ -	$ -	$ (3,489)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 872	$ 2,616
Total Working Capital	(0)	$ (1,053)	$ (9,178)	$ (6,605)	$ (58,890)	$ 9,683	$ (777)	$ (6,709)	$ (6,417)	$ 7,858	$ 2,315	$ 12,004	$ (1,634)	$ (6,784)	$ 11,096	$ (5,841)	$ 8,607	$ (1,975)	$ (4,001)	$ 2,384	$ (424)	$ 3,343	$ 21,549	$ 16,156
Debt Financing	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Capital Expenditures																								
Initial Capital																								
Mine	$ 91,187	$ -	$ 31,820	$ 59,367	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Process Plant	$ 452,415	$ 22,621	$ 180,966	$ 203,587	$ 45,242	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Owners Cost	$ 30,771	$ -	$ 7,000	$ 23,771	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
Sustaining Capital																								
Mining	$ 82,440	$ -	$ -	$ -	$ 4,551	$ 3,371	$ -	$ 3,013	$ 1,722	$ 6,062	$ 6,121	$ 1,287	$ 6,318	$ 20,652	$ 7,313	$ 10,716	$ 2,033	$ 4,802	$ 46	$ 4,433	$ -	$ -	$ -	$ -
Process Plant	$ 61,503	$ -	$ -	$ -	$ 1,927	$ 5,580	$ 1,359	$ 1,269	$ 1,442	$ 7,491	$ 7,305	$ 6,876	$ 7,262	$ 18,839	$ 320	$ 213	$ 170	$ 556	$ 213	$ 426	$ 213	$ 40	$ -	$ -
Total Capital Expenditures	$ 718,317	$ 22,621	$ 219,787	$ 286,725	$ 51,719	$ 8,951	$ 1,359	$ 4,282	$ 3,164	$ 13,553	$ 13,426	$ 8,163	$ 13,580	$ 39,491	$ 7,633	$ 10,929	$ 2,203	$ 5,358	$ 259	$ 4,859	$ 213	$ 40	$ -	$ -
Cash Flow before Taxes	$ 1,585,554	$ (23,674)	$ (228,965)	$ (293,330)	$ 206,221	$ 242,350	$ 220,951	$ 212,742	$ 224,298	$ 162,725	$ 130,124	$ 75,147	$ 61,000	$ 70,744	$ 55,785	$ 70,956	$ 48,245	$ 42,380	$ 65,983	$ 58,875	$ 59,287	$ 64,629	$ 43,343	$ 34,749
Cummulative Cash Flow before Taxes		$ (23,674)	$ (252,638)	$ (545,968)	$ (339,748)	$ (97,398)	$ 123,554	$ 336,296	$ 560,594	$ 723,319	$ 853,444	$ 928,591	$ 989,591	$ 1,060,334	$ 1,116,120	$ 1,187,076	$ 1,235,321	$ 1,277,701	$ 1,343,684	$ 1,402,559	$ 1,461,847	$ 1,526,476	$ 1,569,818	$ 1,604,567
					1.0	1.0	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Taxes																								
Income Taxes	$ 637,700	$ -	$ -	$ -	$ 101,451	$ 74,670	$ 68,029	$ 67,854	$ 71,165	$ 47,913	$ 38,159	$ 14,713	$ 18,743	$ 31,866	$ 9,763	$ 21,388	$ 6,176	$ 8,613	$ 15,565	$ 12,723	$ 12,282	$ 13,809	$ 1,618	$ 1,200
Cash Flow after Taxes	$ 947,853	$ (23,674)	$ (228,965)	$ (293,330)	$ 104,770	$ 167,680	$ 152,923	$ 144,888	$ 153,133	$ 114,812	$ 91,965	$ 60,434	$ 42,257	$ 38,877	$ 46,022	$ 49,568	$ 42,069	$ 33,767	$ 50,419	$ 46,152	$ 47,005	$ 50,820	$ 41,725	$ 33,548
Cummulative Cash Flow after Taxes		$ (23,674)	$ (252,638)	$ (545,968)	$ (441,199)	$ (273,519)	$ (120,596)	$ 24,292	$ 177,425	$ 292,238	$ 384,202	$ 444,636	$ 486,893	$ 525,771	$ 571,793	$ 621,361	$ 663,430	$ 697,197	$ 747,615	$ 793,767	$ 840,773	$ 891,593	$ 933,318	$ 966,867
					1.0	1.0	1.0	0.8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Economic Indicators before Taxes																								
NPV @ 0%	0%	$ 1,585,554																						
NPV @ 5%	5%	$ 906,894																						
IRR %		29.7%																						
Payback	Years	2.4																						
Economic Indicators after Taxes																								
NPV @ 0%	0%	$ 947,853																						
NPV @ 5%	5%	$ 462,521																						
IRR %		17.6%																						
Payback	Years	3.8																						

23 ADJACENT PROPERTIES

This report focuses on the areas of the Project concessions that contain 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

Additional relevant information includes the project execution plan originally developed as part of the Pre-Feasibility study.

24.1 PROJECT EXECUTION

24.1.1 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.

24.1.2 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 consultants input, client input and historical project data. Construction durations were based on quantities and man-hours developed in the capital cost estimate:

- Bankable Feasibility Study – 4 months
- ESIA Preparation/Review and Permitting – 17 months
- Detailed Engineering – 18 months
- Construction – 23 months
- Commissioning/Start-Up – 6 months

ESIA baseline work includes further ARD, surface water and groundwater assessment. Detail engineering will begin after the Bankable Feasibility Study has been filed. Funding must be available during the 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 mills is also planned during detail engineering as their quoted lead times are approximately 58 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.

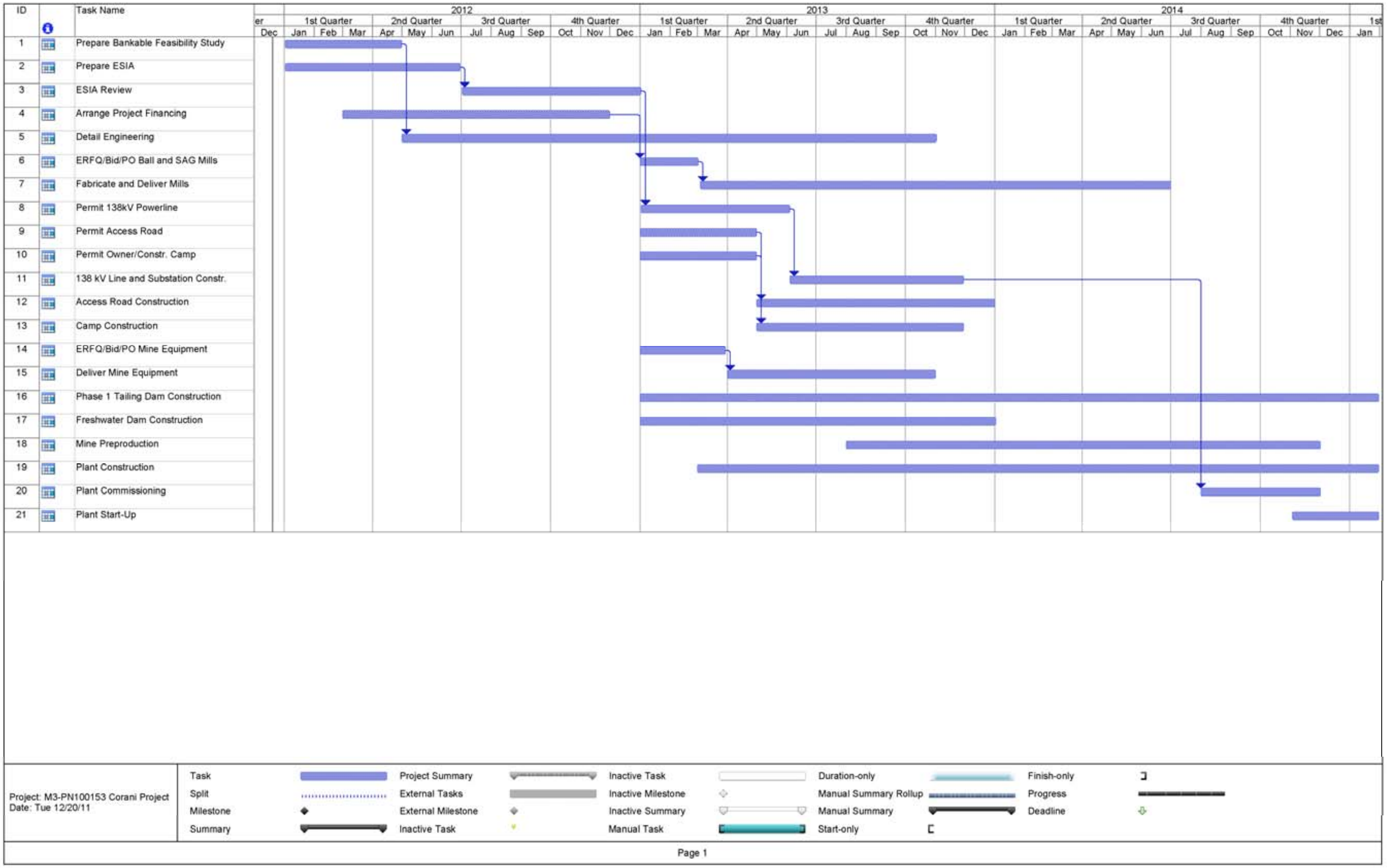


Figure 24-1: Project Schedule

The critical path runs through test work and permitting and then through the completion of construction activities. The 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 starter Tailing Impoundment dam and the Fresh Water Impoundment 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 24-1 for a breakdown of scheduled activities and milestones.

24.1.3 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

24.1.4 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 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 EPC schedule that will plan all the major activities in accordance with contractual requirements.

24.1.5 Engineering

Execution of engineering work would likely take place in North America. 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.

24.1.6 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.

24.1.7 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.

24.1.8 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.

24.1.9 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.

24.1.10 Procurement

Purchasing

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.

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; and
- Execute contract close-out activities to ensure compliance with contract scope of work and contract financial terms.

24.1.11 Construction

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

Construction of the process plant and infrastructure facilities would be performed by contractors specialized in the scope of work as described in the Engineering Requests for Proposal (ERFP) documents. Contracts would be awarded to major construction contractors following competitive bidding based on bid documents prepared by the engineer.

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

24.1.12 Commissioning and Start up

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, robust silver and base metals 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 270 million ounces of silver, 3.1 billion pounds of lead and 1.7 billion pounds of zinc.

- The base case after-tax net present value ("NPV") is $463M at a 5% discount rate with an internal rate of return ("IRR") of 17.6% ($18/oz silver, $0.85/lb lead and $0.85/lb zinc). On a pre-tax basis, the base case NPV at a 5% discount rate is $907M with an IRR of 29.7%.

- At spot metals prices ($34.64/oz silver, $0.89/lb zinc, $0.90/lb lead on November 8, 2011, the date of the initial press release of the FS results), Corani has an after-tax NPV of approximately $1.5 billion at a 5% discount rate and a 38% IRR ($2.7 billion NPV and 60% IRR on a pre-tax basis).

- Average annual payable silver production is 13.4M ounces per year for the first five years and 8 million ounces per year over the life-of-mine ("LOM"). On a silver equivalent ounce basis, average annual payable production is 23.0 million ounces per year for the first five years and 14.7 million ounces per year over the LOM.

- Cash cost is a negative $(0.45) per ounce of silver for the first five years, with a LOM cash cost of $3.66 per ounce of silver (net of base metal credits at $0.85/lb lead and $0.85/lb zinc).

- Project produces Lead and Zinc concentrates. Extensive metallurgical testing has established conventional flotation recoveries.

- Initial capital cost is $574 million with capital payback of 3.8 years at base case metal prices, and 2.0 years at metal prices on November 11, 2011.

- Mine life is 20 years.

- Mill capacity is 22,500 tonnes per day.

- Stripping ratio is 1.69:1 (waste:ore).

- Opportunities include 89M of measured and indicated silver resource ounces, which represent potential future reserve conversion. Additional new mineralization was intersected in recent drilling near perimeter of proposed tailing dam.

25.2 CONCLUSIONS

The Project has an after-tax internal rate of return (IRR) of 17.6%, net present value of $463 million at a 5% discount rate based upon metals prices of $18 per ounce silver, and $0.85 per pound for lead.

Payable silver production averages 13.4 million ounces per year. The project will produce an average of 8 million payable ounces of silver, 105 million pounds of lead and 37 million pounds of zinc annually over a 20 year mine life.

Total cash cost for the first five years is a negative $(0.45)/oz silver, with a mine-of-life cash cost of $3.66/oz silver, net of base metal credits. The initial capital investment on the project is estimated to be $574 million with sustaining capital expenditures during mine operations averaging $7.2 million per year over the 20 year mine life. The project achieves payback of capital in 3.8 years using base case metal prices.

The FS is based upon assumptions derived from mine planning sequences completed by IMC and metallurgical test work performed by SGS Laboratories in Vancouver, BC and reviewed by Blue Coast Metallurgy. The mining sequence primarily derives ore from the higher-grade starter pits in the early years and moves to lower-grade areas in the later years of production. Operations are for 20 years based on current reserves.

In the mine sequence, only 270 million ounces contained within 156 million tonnes have been used as reserves in this plan. An additional 134 million tonnes of measured and indicated resource (containing 88.7 million ounces of silver at 20.5 g/t) and 49.8 million tonnes of inferred resource (containing 48.0 million ounces of silver at 30g/t) remain that could be included in later plans of operations. About 89% of these resources are mixed sulfide and transition material peripheral to the reserve pit. About 11% are contained within oxide mineralization, which outcrops at surface.

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	64.2%
Overall process recovery – lead – into lead cons	71.1%
Overall process recovery – zinc – into zinc cons	51.6%
Total processed tonnes	156,130,000
Average silver grade (g/t)	53.8 g/t
Average lead grade (%)	0.90%
Average zinc grade (%)	0.49%
Payable ounces of silver net of smelter payment terms (total)	160.2 million
Payable pounds of lead net of smelter payment terms (total)	2.1 billion
Payable pounds of zinc net of smelter payment terms (total)	744 million
Overall stripping ratio	1.69 to 1
Life-of-mine (mining only) years	18
Life-of-mine (processing) years	20

Reserves are based on metal prices of $18/oz silver and $0.85 per pound for both lead and zinc. For the resources, metal prices of $30/oz for silver and $1.00/lb for both lead and zinc were used, representing the three-year backward and two-year forward metal prices weighted 60:40 from August 2011 which is consistent with the Company's policy and industry standards.

The Feasibility Study recommends proceeding with project development based on:

- Robust economics at the base case assumptions with excellent exposure to up-side silver and base metals prices,
- Well-defined resources open to expansion and conversion to reserves,
- A solid metallurgical process producing highly marketable, separate lead and zinc concentrates,
- Favorable infrastructure for tailing storage, 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 15-3 represents the Mineral Resources and mineral reserves.

25.2.2 Mineral Reserves

As a result of the work described throughout this report, the authors conclude that the mineral reserve is as stated on Table 15-3. For ease of reference, the table is repeated below.

Mineral Reserves, $10.54 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	30,083	66.6	1.04	0.60	64.4	690.4	399.9	115.7	119.6
Probable	126,047	50.7	0.87	0.47	205.6	2,422.6	1,297.7	381.5	94.1
Proven + Probable	156,130	53.8	0.90	0.49	270.0	3,113.0	1,697.6	497.2	99.1

Mineral Resources in Addition to Reserves, $9.20 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	10,878	17.5	0.38	0.33	6.1	91.1	79.1	13.9	39.6
Indicated	123,583	20.8	0.38	0.29	82.6	1,035.3	790.1	166.7	42.0
Measured + Indicated	134,461	20.5	0.38	0.29	88.7	1,126.4	869.2	180.6	41.8
Inferred	49,793	30.0	0.46	0.28	48.0	509.4	305.2	86.2	53.9

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 at altitudes similar to those of the Project that are successful operations.

Mining will be performed using conventional open pit methods using 136t 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 16.2M 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 Independent Mining Consultants of Tucson, Arizona, the process and portions of the infrastructure capital cost have been prepared by M3 Engineering of Tucson, Arizona and the Tailing Storage Facility ("TSF") and remaining infrastructure costs have been prepared by Global Resource Engineering ("GRE"). The initial startup capital is estimated to be $574M and the sustaining capital cost is estimated to be $7.2M annually over the life of mine. The capital costs include detailed long-term plans for tailing dam expansions as well as ongoing capital (i.e. mine fleet replacement) and mine closure.

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; 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 tailing storage resulting in very low capital and operating costs, as the plant will be located immediately adjacent to the mine and the tailing will be pumped to the TSF. 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 small water storage dam and water capture in the TSF, mine and 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 and is currently negotiating the access rights for the ancillary facilities. 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 TSF has been designed by GRE to the highest standards of containment and stability. Importantly, the latest design technology will facilitate the permitting process. In the TSF, the Feasibility Study calls for the operation of a sulfide flotation plant that will capture and segregate the sulfide material in the central part of the TSF. This will result in the sulfide material never being exposed to the atmosphere during operations and following mine closure. This technology provides assurance that the TSF will not produce acid rock drainage, thus facilitating final closure of the TSF.

Furthermore, the waste rock storage facilities are designed to capture and manage any flows that may originate from the waste rock. A buffer layer of inert rock will be placed on the outside of the waste rock piles to mitigate the acid producing potential of the facilities. Finally, the closure plan provides for the covering of the tailing storage and waste rock facilities assuring safe and environmentally compliant closure of the mine.

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

Over 80% of land is owned by BCM. The remaining 20%, corresponding to the location of the tailing dam and the pipe corridor to the plant, is currently under negotiation for purchase.

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). The statutory timeframe for approval of the ESIA is established as a maximum of 120 days from the time of submission. However, in practice the time for approval is highly variable and could take between 6 and 12 months.

The ESIA procedure contains a community consultation process including community workshops and public hearings and the development of a community relations plan. Reaching agreements take time and this can cause delays.

25.3.6 Finance Risks

Significant raises in the long-term financing charges could affect the cost of capital needed for the start of operation.

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.

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 resources (134 million tonnes of measured and indicated containing 88.7 million ounces averaging 20.5 g/t Ag and 49.8 million tonnes of inferred resources containing 48 million ounces of silver averaging 30.0 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. 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 (see news release dated October 11, 2011).

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.

25.4.4 Incorporation of an Oxide Circuit

The Corani deposit contains a significant amount of oxide material in the resource (16 million tonnes at 33 g/t silver of measured plus indicated resource and 4.4 million tonnes at 28 g/t silver of indicated resource). This is potentially an additional source of silver if it can be leached. The volume of this material will have to be estimated and a testing program implemented to see if silver can be extracted at a profit.

26 RECOMMENDATIONS

The recommendations presented in this section relate to parts of a feasibility study and associated evaluations, studies and permitting. The anticipated scope of work for the evaluations, studies and permitting is expected to cost between $4 and $6 million. The scope of work will be developed with prospective feasibility study firms to establish the associated costs.

26.1 PLANT RELOCATION

The most recent geotechnical studies should be used to optimize the process plant location which would likely involve only minor changes to the plant position and elevation. Once the more optimal site location is established there should be follow-up geotechnical drilling at the new proposed mill locations to confirm the rock foundation conditions.

26.2 GOLD ZONE

For the Gold Zone, a program of metallurgical testing should be undertaken to establish an appropriate recovery method and costs and then a mineral resource should then be calculated for this area.

26.3 MINING AND MODELING

During the course of the work on this Project, IMC has developed recommendations for future consideration and execution by BCM. The list of IMC recommendations is summarized below:

- The scatter or variability in check assays and standards should be explained. If there has been a problem with data storage of check assays, it could imply a problem with database assembly that has not been identified; and

- The existing rock density information should be sorted by mineral ore type to see if there is a better correlation method for future model estimates.

26.4 METALLURGY

The primary mesh of grind for the Corani ore needs to be studied further. While the mill design was based on a grind of 106 microns, recent metallurgical tests show that some ore types may require finer or coarser grinds. In addition, grinding mills in closed circuit with cyclones tend to overgrind heavy mineral. This has been shown for copper sulfides and is very pronounced for lead sulfide. This may represent an opportunity for the operating mill to use coarser grinds and reduce unit power costs.

Metallurgical tests should be performed on composites of ore material that originate from the parts of the mine that make up the ore feed from the early years. The material should be handled in a manner that reduces the material aging and oxidation that may cause reductions in the metallurgical performance. Following these tests, an evaluation should be prepared of whether or not a pilot plant test program is warranted to demonstrate the metallurgical behavior of the

Project ore. If pilot testing is needed the test should be performed on a composite of representative material.

Bench-top flotation tests, whether in batch or in lock cycles, may predict recoveries well but not plant concentrate grades. It is recommended that tests results be benchmarked against existing operations with similar grades. The effect of variations in floatability in the ore body to the process plant may also be predicted by performing simulations that use flotation kinetics parameters measured for the different ore types.

Additional studies are recommended to improve zinc recovery in general and, in particular, improve silver recovery in low-grade zinc ores.

Additional grindability tests are recommended to characterize the variability of hardness of the ore that will be delivered to the mill over the life of mine. Samples for these tests should come from locations in the mining cone that are somewhat evenly distributed in space and representing all the rock types. A geostatistic analysis of these results will allow prediction of the work index of each mining block and aid in estimating mill capacity over the life of mine.

Selectivity of the flotation process for Corani was shown to be optimal with the use of inert grinding media. Flotation tests are recommended to determine the best alloy to be used for grinding balls that will minimize unit cost per tonne of ore while maintaining the required chemistry for flotation.

26.5 PROCESS PLANT DESIGN

The locked cycle flotation tests results were not benchmarked against existing operations. The design used flotation residence time recommendations from the laboratory tests. However, the recoveries and concentrate grades used for the metallurgical balance (MetSim) were estimates that resulted in a convergent simulation and within reasonable limits. Variations from these estimates will affect sizing of pumps, pump boxes and pipes, as well as flotation cell volumes and agitator drives. Additional MetSim simulations are recommended when benchmarked locked-cycle tests results become available.

The regrind mills were sized by vendors and would tend to be conservative. No data on the material work index was available and none was given to the vendors. It is recommended that data from similar existing lead-zinc operations be collected and used to estimate feed size distribution (F80) and the Bond work index (from the operating work index). Better estimates of the feed F80 and work indices could potentially result in smaller mills, and therefore lower capital and operating costs.

Sizing of the thickeners for tails and concentrates were based on typical settling rates. No experimental data were available. It is recommended that tails and concentrate samples be tested to determine settling rates and rheology of the thickened slurry to properly design the thickeners.

Filter presses for concentrate dewatering were sized by the filter vendors based on the fineness of the concentrates. No test data were available for the filtration rate or filter cake moisture. Actual samples at the projected size distribution must be submitted for filtration rate analysis. The actual concentrate fineness may differ from the design targets and may require more filter area or the addition of a filtration aid.

26.6 ENVIRONMENTAL AND SOCIAL

Given the mineralogy of the area, water management is a key environmental consideration. Potential impacts to surface water and ground water in relation to the tailing storage facility and waste rock dumps should be thoroughly assessed and mitigation and management programs developed.

An environmental liability closure plan will be developed and the environmental liabilities will be resolved as the mine is developed.

26.7 RE-OPTIMIZED INFRASTRUCTURE COMPONENTS

The work in progress on the re-optimized infrastructure components indicates favorable conditions for tailing and waste rock management. The current concepts such as pit backfilling to eliminate pit lakes, de-pyritization of tailing, and encapsulation of PAG waste within the Main Dump represent a significant advancement in environmental management practices over previous studies. On this basis it is recommended that the optimization of infrastructure components continue to be advanced in support of basic engineering studies. It is recommended that relevant supporting studies be carried out including hydrology, hydrogeology, geochemistry, seismic design, and construction material characterization.

26.8 OTHER RECOMMENDATIONS

During the course of the work on this Project, the contributors have developed several recommendations for future consideration and execution by BCM, including:

- Further development of the project should proceed with basic engineering to better define and optimize the arrangement of facilities and to confirm equipment sizing and selection.
- More geotechnical tests are required to finalize the location of the mill and surrounding facilities. Geotechnical studies so far completed suggest that a minor relocation or repositioning of the plant may be required to minimize foundation work costs.
- M3 recommends that BCM start talks with prospective lead and zinc smelters to improve estimates for smelter treatment and refining charges, payable rates, transport costs, specifications, and others. The smelters may request BCM to submit samples of typical concentrates that the Corani plant will produce.
- The capital cost estimate is based on budgetary quotes for equipment. Capital cost reduction is possible with the negotiation of firm purchase orders.
- With recent worldwide financial events, the used equipment market is emerging again. Consideration should be given to the purchase of used process equipment.

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APPENDIX A: FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

The groups responsible for this report are listed in the following table.

SECTION	SECTION NAME	COMPANY	RESPONSIBLE PARTY
1	Summary	M3, GRE & Walsh	Daniel H. Neff, P.E., Chris Chapman, P.E. & Edmundo Laporte-Ramírez, P.E.
2	Introduction	M3	Daniel H. Neff, P.E.
3	Reliance on Other Experts	M3	Daniel H. Neff, P.E.
4	Property Description and Location	M3 & Walsh	Daniel H. Neff, P.E. & Edmundo Laporte-Ramírez, P.E.
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	M3, GRE & Walsh	Daniel H. Neff, P.E., Chris Chapman, P.E. & Edmundo Laporte-Ramírez, P.E.
6	History	IMC	John Marek P.E.
7	Geological Setting and Mineralization	IMC	John Marek P.E.
8	Deposit Types	IMC	John Marek P.E.
9	Exploration	IMC	John Marek P.E.
10	Drilling	IMC	John Marek P.E.
11	Sample Preparation, Analyses and Security	IMC	John Marek, P.E.
12	Data Verification	IMC	John Marek, P.E.
13	Mineral Processing and Metallurgical Testing	Blue Coast	Chris Martin, C. Eng.
14	Mineral Resource Estimates	IMC	John Marek, P.E.
15	Mineral Reserve Estimates	IMC	John Marek, P.E.
16	Mining Methods	IMC	John Marek, P.E.
17	Recovery Methods	M3	Art S. Ibrado, Ph.D.
18	Project Infrastructure	M3 & GRE	Daniel H. Neff, P.E & Chris Chapman, P.E.
19	Market Studies and Contracts	M3	Daniel H. Neff, P.E.
20	Environmental Studies, Permitting and Social or Community Impact	Walsh & GRE	Edmundo Laporte-Ramírez, P.E. & Chris Chapman, P.E.
21	Capital and Operating Costs	M3, IMC & GRE	Daniel H. Neff, P.E., John Marek, P.E. & Chris Chapman, P.E.
22	Economic Analysis	M3	Daniel H. Neff, P.E.
23	Adjacent Properties	IMC	John Marek, P.E.
24	Other Relevant Data and Information	M3	Daniel H. Neff, P.E.
25	Interpretation and Conclusions	M3	Daniel H. Neff, P.E.
26	Recommendations	All	Daniel H. Neff, P.E.
27	References	M3	Daniel H. Neff, P.E.

Abbreviations: ALL – All QP Contributors; IMC – Independent Mining Consultants, M3 – M3 Engineering & Technology Corporation, Blue Coast – Blue Coast Metallurgy, Ltd.

Note: Where multiple authors are cited, refer to author certificate (Appendix A) for specific responsibilities.

The certificates of the Qualified Persons are included in this appendix.

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 & 13848)

4. I have practiced civil and structural engineering and project management for 37 years. I have worked for engineering consulting companies for 12 years and for M3 Engineering and Technology, Corporation for 25 years.

5. I have read the definition of "qualified person" set out in National instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of the following sections of the technical report titled Corani Project, Form 43-101F1 Technical Report, Feasibility Study, Dated 22 December 2011 (the "Technical Report"), relating to the Corani Silver, Lead, Zinc project in Peru.

SECTION	SECTION NAME
1	Summary
2	Introduction
3	Reliance on Other Experts
4	Property Description and Location
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography
18	Project Infrastructure
19	Market Studies and Contracts
21	Capital and Operating Costs
22	Economic Analysis
24	Other Relevant Data and Information
25	Interpretation and Conclusions
26	Recommendations
27	References

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. 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.

9. 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.

10. 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.

11. I visited the Corani property on December 8 - 9, 2010.

Dated this 22nd December 2011.

____

Signature of Qualified Person

Daniel H. Neff

Print name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, John M. Marek P.E. do hereby certify that:

a) I am currently employed as the President and a Senior Mining Engineer by:

Independent Mining Consultants, Inc.
3560 E. Gas Road
Tucson, Arizona, USA 85714

b) This certificate is part of the report titled "Corani Project, Form 43-101F1, Technical Report, Feasibility Study", dated 22 December 2011".

c) I graduated with the following degrees from the Colorado School of Mines
Bachelors of Science, Mineral Engineering – Physics 1974
Masters of Science, Mining Engineering 1976

I am a Registered Professional Mining Engineer in the State of Arizona USA
Registration # 12772
I am a Registered Professional Engineer in the State of Colorado USA
Registration # 16191

I am a Registered Member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers

I have worked as a mining engineer, geoscientist, and reserve estimation specialist for more than 35 years. I have managed drill programs, overseen sampling programs, and interpreted geologic occurrences in both precious metals and base metals for numerous projects over that time frame. My advanced training at the university included geostatistics and I have built upon that initial training as a resource modeler and reserve estimation specialist in base and precious metals for my entire career. I have acted as the Qualified Person on these topics for numerous Technical Reports.

My work experience includes mine planning, equipment selection, mine cost estimation and mine feasibility studies for base and precious metals projects world wide for over 35 years.

d) I visited the Corani property during the week of 10 July 2006.

e) I am responsible for the following sections of the report titled "Corani Project, Form 43-101F1, Technical Report, Feasibility Study, 22 December 2011" : 6, 7, 8, 9, 10, 11, 12, 14, 15, 16, 21.2, 21.5.5 and 23.

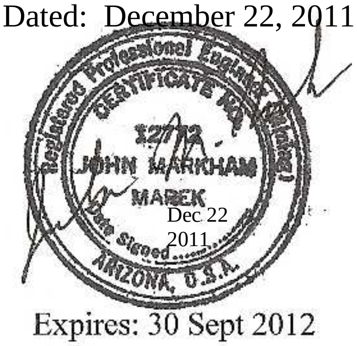
f) I am independent of Bear Creek Mining Corporation applying the tests in Section 1.5 of National Instrument 43-101.

g) Independent Mining Consultants, Inc., and this author have worked on the Corani project previously. Previous Technical Reports were completed on 4 October 2006 and 14 October 2009.

h) I have read National Instrument 43-101 and Form 43-101F1, and to my knowledge, the Technical Report has been prepared in compliance with that instrument and form.

i) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated: December 22, 2011.



John M. Marek P.E.

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CERTIFICATE OF QUALIFIED PERSON

I, Christopher John Martin of 1961 Delanice Way, Nanoose Bay, British Columbia hereby certify:

1. I am a graduate of Camborne School of Mines and hold a B.Sc (Honours) Degree in Mineral Processing Technology (1984).
2. I am a graduate of McGill University and hold a M.Eng Degree in Metallurgical Engineering (1988).
3. I am presently President and Principal Metallurgist at Blue Coast Metallurgy Ltd, 1020 Herring Gull Way, Parksville, British Columbia.
4. I have ten years' plant operations experience as Metallurgist, Senior Metallurgist and Plant Superintendent with Rustenburg Platinum Mines in South Africa, as Operations Engineer at Nerco Con Mine in the Northwest Territories and Chief Corporate Metallurgist at Sunshine Mining in Idaho.
5. I have fifteen years' flowsheet development and plant optimisation consulting experience as manager SGS Lakefield Mineral Processing group, General Manager of SGS Vancouver Metallurgy, Manager Metallurgy at AMTEL and most recently as President of Blue Coast Metallurgy.
6. I have been a licenced Chartered Engineer in good standing with the IMMM since 1990. I am also a member of the Canadian Institute of Mining, Metallurgy and Petroleum.
7. I have read the definitions of "Qualified Person" set out in NI43-101 and certify that, by reason of my education, affiliation to a professional institution and past work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI43-101.
8. I have not visited the Corani site.
9. I have no direct involvement with Bear Creek Mining Corporation, and am independent of Bear Creek Mining Corporation applying all the tests of Section 1.5 of NI43-101.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I have read NI43-101 and NI43-101F1 and the technical report is been compiled in compliance with the instrument and form.
12. I consent to the use of this report for the purpose of complying with the requirements set out in NI43-101 to support the technical report "CORANI PROJECT, FORM 43-101F1 TECHNICAL REPORT, FEASIBILITY STUDY", dated December 22nd, 2011 to be submitted to SEDAR for electronic filing.

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13. I am responsible for section 13 of the technical report titled Corani Project, Form 43-101F1 Technical Report, Feasibility Study, Dated 22 December 2011 (the "Technical Report"), relating to the Corani Silver, Lead, Zinc project in Peru.

Christopher J. Martin, BSc (Hons) ACSM, M.Eng, MIMMM, C.Eng
President and Principal Metallurgist, Blue Coast Metallurgy Ltd.
Dated in Parksville, British Columbia on this 22nd day in December 2011.

CERTIFICATE OF QUALIFIED PERSON

I, Art S. Ibrado, Ph.D., do hereby certify that:

1. I am employed as a project manager and metallurgist at M3 Engineering & Technology Corp., 2051 W Sunset Rd, Suite 101, Tucson, AZ 85704, USA

2. I graduated with the following degrees:
   Bachelor of Science in Metallurgical Engineering, University of the Philippines, 1980
   Master of Science (Metallurgy), University of California at Berkeley, 1986
   Doctor of Philosophy (Metallurgy), University of California at Berkeley, 1993

3. I am a Qualified Professional (QP) member of the Mining and Metallurgical Society of America (MMSA) and a Professional Member of the Society of Mining, Metallurgy, and Exploration, Inc. (SME).

4. I have worked as a metallurgist in the academic and research setting for five years, excluding graduate school research, and in the mining industry for 13 years before joining M3 Engineering.

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 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 entitled "Corani Project, Form 43-101F1 Technical Report, Feasibility Study" dated December 22, 2011 relating to the Corani property. I visited the Corani property on December 8 - 9, 2010.

7. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

8. I am independent of Bear Creek Mining Corporation as defined in section 1.5 of National Instrument 43-101 and I do not own any of their stocks.

9. 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.

10. 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 website accessible by the public, of the Technical Report.

11. I have had no prior involvement with the property that is the subject of the Technical Report.

Dated this 22 day of December 2011.

Art S. Ibrado, Ph. D.

Christopher K. Chapman

Principal Mining & Civil Engineer

Global Resource Engineering, Ltd

7000 S. Yosemite St.

Suite 201

Centennial, Colorado 80112

Telephone: 303-383-5033

Email: cchapman@global-resource-eng.com

CERTIFICATE of AUTHOR

I, Christopher K. Chapman do hereby certify that:

1. I am currently employed as Principal Mining & Civil Engineer by Global Resource Engineering, Ltd at:

   7000 S. Yosemite St.

   Centennial, Colorado 80112

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 & Civil Engineer for a total of 11 years since my graduation from university, as a consulting engineer.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for the preparation of the technical report titled "NI 43-101 Technical Report, Corani Project" dated December 22nd 2011 (the "Technical Report"). Specifically, I am responsible for the following sections of this Technical Report: 1.8, 5.3, 18.2, 20.4, and 21.5.4.
7. I have personally visited the project six times over the past year during the period November 2010 to November 2011 spending over 70 days onsite. My most recent trip was in November 2011 consisting of 6 days onsite.

8. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, this Technical Report contains all the scientific and technical information that is required to be disclosed to make this Technical Report not misleading.
9. I am independent of Bear Creek Mining Corporation as independence is described in section 1.5 of NI 43-101. I verify that in the opinion of a reasonable person aware of all relevant facts, there is no circumstance that could interfere with my judgment regarding the preparation of the Technical Report.
10. I have had no prior involvement with the subject property of the Technical Report prior to this study.
11. I have read National Instrument 43-101, Form 43-101F1, and Companion Policy 43-101CP; the Technical Report has been prepared in compliance with that instrument, form, and policy.
12. I consent to the filing of the Technical Report with any stock exchanges or other regulatory authority and any publication by them, including electronic publication in the public company files on the websites accessible by the public, of the Technical Report.

Dated this 22nd day of December 2011

Christopher K. Chapman (Electronic Signature)

Signature of Qualified Person

"Christopher K. Chapman"

Print name of Qualified Person

Edmundo J. Laporte

ECSI, LLC

Civil – Environmental – Mining

340 South Broadway, Suite 200

Lexington, KY 40508. USA

Telephone: 859-233-2103

Fax: 859-259-3394

CERTIFICATE OF AUTHOR

I, Edmundo J. Laporte, PE, PEng, CPEng, RPEQ, hereby certify that:

a) I am an independent Engineer with:

ECSI, LLC
340 South Broadway, Suite 200
Lexington, KY 40508. USA

b) I graduated with a Bachelor of Science degree from the University of Rafael Urdaneta, in Maracaibo, Venezuela in 1987.

c) I am a Registered Member of the Society for Mining, Metallurgy & Exploration Inc. (SME); a Professional Engineer in the Province of Alberta, Canada; a Professional Engineer in 14 States in the United States; a Chartered Professional Engineer in Australia; a Registered Professional Engineer in Queensland, Australia; a Member of the Association of Professional Engineers, Geologists, and Geophysicists of Alberta (APEGGA), and a Member of the Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

d) I have worked as an engineer for a total of 24 years since my graduation from University.

e) I have read the definition of “Qualified Person” set out in the 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.

f) I am responsible for the contents of the following sections: 1.7 Environmental and Permitting, 4.3 Environmental Liabilities, 4.4 Permitting, 5.4 Physiography and Vegetation, 20 Environmental Studies, Permitting and Social or Community Impact, 26.3 Environmental and Social (Recommendations) of the Corani Project, Form 43-101F1 Technical Report, Feasibility Study (the “Technical Report”). For the preparation of those sections, I am relying on the opinion and statements prepared by Mr. Gonzalo Morante, a Peruvian Environmental Expert and General Manager of Walsh Peru, a reputable Environmental Consulting Firm, who analyzed information obtained from both the issuer and publicly available sources on the subject matters specified in those sections.

g) I am independent of Bear Creek Mining Corporation as defined in section 1.5 of the Instrument.

h) I have not had prior involvement with the property that is the subject of the Technical Report.

i) I have read the National Instrument 43-101 and form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

j) As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Dated this 22nd day of December, 2011

____

Signature of Qualified Person

<u>Edmundo J. Laporte-Ramirez</u>

Print name of Qualified Person

CERTIFICATION

Qualification Certificate for Marc Paul Francois Leduc, P.Eng.
I, Marc P.F. Leduc, hereby certify that:

- I am a US resident living at 32 Buckthorn Drive, Littleton, Colorado 80127, USA;
- I am a professional engineer registered in the Province of Ontario (# 100046093) and in British Columbia (# 29552);
- I am a member of the Canadian Institute of Mines and the American Society of Mining, Metallurgy and Exploration;
- I am graduated from Queen's University of Kingston (B.Sc. Honours Mining Engineering) in 1992;
- I am graduated from University of Ottawa (B.Sc. Geology) in 1989;
- I have experienced in my profession since 1988 in the field of exploring and mining gold, base metals and industrial minerals;
- I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.
- I am responsible for the transfer of technical information developed by Bear Creek Mining to the independent qualified people used in the preparation of the report titled *Corani Project, Form 43-101F1 Technical Report, Feasibility Study, Dated 22 December 2011*, relating to the Corani Silver, Lead, Zinc project in Peru. I have visited the Corani Project in Peru on several occasions as part of my duties as President and COO for the Company;
- I am a Qualified Person in accord with the National Instrument 43-101. I have studied this norm and I understand its terms and implications; and
- I am not an Independent Qualified Person as described by NI 43-101 as I am an Employee, Officer and Insider of the issuer, Bear Creek Mining Corporation.

- I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
- I have read the National Instrument 43-101 ("NI43-101") and the Technical Report has been prepared in compliance with the instrument.

Prepared in Littleton Colorado on December 22, 2011.

PROFESSIONAL
PROVINCE OF
M. P. F. LEDUC
29562
BRITISH COLUMBIA
ENGINEER

Marc P.F. Leduc, P.Eng.
for Bear Creek Mining Corp.