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King-king Copper-Gold Project

NI 43-101 Technical Report
Preliminary Feasibility Study
Mindanao, Philippines

REVISION 0

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KING-KING COPPER–GOLD PROJECT
FORM 43-101F1 TECHNICAL REPORT
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LIST OF APPENDICES

APPENDIX	DESCRIPTION
A	Feasibility Study Contributors and Professional Qualifications • Certificate of Qualified Person (“QP”) and Consent of Author

1 SUMMARY

M3 Engineering & Technology of Tucson, AZ was contracted by St. Augustine Gold & Copper ("SAGC") of Spokane, Washington, to prepare a Preliminary Feasibility Study (the "PFS") and an Independent Technical Report (the "Report"), compliant with National Instrument 43-101 ("NI 43-101") on the King-king Copper-Gold Project (the "Property"). This section briefly summarizes the findings of the Preliminary Feasibility Study.

The project is located in the Municipality of Pantukan, Compostela Valley, near Davao City, Philippines. The mining rate will be approximately 178,000 tons per day (tpd) utilizing contract mining. Over the life of the project, 3.16 billion pounds of copper, 5.43 million ounces of gold, and 11.65 million ounces of silver are projected to be produced.

The proposed project is an open pit copper-gold mine that delivers ore to a 60,000 tpd mill facility and a 40,000 tpd heap leach facility. The average life-of-mine throughput is 73,000 tpd (combined through mill and heap leach). The mill facility treats the mill ore with primary crushing, grinding, flotation, tailing agitated leach (with solvent extraction – electrowinning (SX-EW)), tailing neutralization followed by drystack tailing placement. The heap leach facility treats the heap leach ore with three stages of crushing, agglomeration, leaching on an on-off pad. Pregnant leach solution (PLS) from the leach pad is processed in the same SX-EW facility that treats the agitated leach PLS. Other key components of the project include a power plant and port facility.

SAGC selected M3 Engineering & Technology Corporation (M3) and other respected third-party consultants to prepare mine plans, resource/reserve estimates, process plant designs, and to complete environmental studies and cost estimates used for this report. All consultants have the capability to support the project, as required and within the confines of expertise, from feasibility study to full operation. The costs are based on third quarter 2012 US dollars.

1.1 KEY DATA

Key project parameters are presented in Table 1-1 including a summary of the project size, production, operating costs, metal prices, and financial indicators.

Table 1-1: Key Project Data

Mine Life (years)	23 years		
Mine Type:	Open Pit		
Process Description:	Crushing, grinding, flotation, flotation tail leach, drystack tailing deposition, SX-EW, On/off Leach pad		
Total Material Mined (Tons per day)	178,000		
Design Mill Throughput (Tons per day)	60,000		
Design On/Off Leach Pad Throughput (Tons per day)	40,000		
LOM Copper Ore Grade	0.30%		
Average Life of Mine throughput (mill and leach pad) (Tons per day)	73,000		
LOM Gold Ore Grade	0.44g/t		
Initial Capital Costs ($US Millions)	$2,041.9		
Sustaining Capital Costs ($US Millions)	$248.6		
Adjustment for Escalation	None – Assumed 2012 dollars		
Payable Metals			
Copper (Billion Pounds)	3.1		
Gold (Million troy ounces)	5.2		
Silver (Million troy ounces)	7.4		
Unit Operating Cost: (per payable pound of copper)	**Years 1-5**	**Years 1-10**	**LOM**
Mining Cost	$0.47	$0.60	$0.80
Processing Cost	$0.81	$0.92	$1.06
G&A Costs	$0.13	$0.16	$0.27
Shipping, Smelting and Refining Costs	$0.11	$0.15	$0.18
Government Fees	$0.17	$0.22	$0.26
Total cost	$1.69	$2.04	$2.57
By-Product Credits (Gold & Silver)	($1.66)	($1.85)	($2.17)
Total Consolidated Net Cash Cost	$0.03	$0.19	$0.40
Financial Indicators*	**Base Case**	**High Metal Price (+20%)**	**Low Metal Price (-20%)**
Gold Price (per troy ounce)	$1,250	$1,500	$1,000
Copper Price (per pound)	$3.00	$3.60	$2.40
Silver Price (per troy ounce)	$25.0	$30.0	$20
After Tax Project Internal Rate of Return (IRR)	24.0%	33.0%	13.6%
After Tax NPV at 8% Discount Rate ($ Billions)	$1.8	$3.0	$0.6
After Tax Payback (years)	2.4	1.8	3.8
Major Permit Status			
Draft Environmental Impact Statement Submitted for Comments	February 2012		
Declaration of Mine Project Feasibility Submitted	May 2012		

*Assumes a 6-year income tax holiday.

1.2 PROPERTY DESCRIPTION AND LOCATION

1.2.1 Description

The central project property is the tenement area defined by the Amended Mineral Production Sharing Agreement (MPSA) No. 009-92-XI, between the Philippine government and NADECOR. It covers a total area of approximately one thousand six hundred fifty six (1,656) hectares situated in Sitio Lumanggang, Pantukan. The tenement straddles three (3) barangays: King-king, Magnaga and Tagdangua, with approximately half of its total area being situated within the King-king Barangay.

Large multi-volume reports related to permitting of the MPSA have been submitted to the Department of Environment and Natural Resources (DENR). The draft Environmental Impact Statement (EIS) was submitted to the Environmental Management Bureau (EMB) in February 2012. The Declaration of Mining Project Feasibility (DMPF), including the relocation plan for the project affected people, was submitted in May 2012 to the Mines and Geosciences Bureau (MGB). Additionally, the endorsements required by the DMPF have been obtained from the Local Government Units (LGU).

Being within forest land, the tenement area is also covered by a Certificate of Ancestral Domain Title (CADT) issued by the National Commission on Indigenous Peoples (NCIP) to the Mansaka tribe in the Municipality of Pantukan as provided for by Republic Act 8371 or the Indigenous Peoples Rights Act (IPRA). CADT No. R11-PAN-0908-076 was signed in September 2, 2008 and covers a total area of approximately 141,773 hectares. With the exception of alienable and disposable (A&D) lands covered either by an Original Certificate of Title (OCT) or Transfer Certificate of Title (TCT), all project areas are covered by the CADT.

Project facility areas cover approximately 300 parcels of land covered by OCT or TCT. The OCT and TCT give holders surface rights over the land area covered by the document, with the transfer thereof covered by and subject to Philippine laws and regulations.

Some land acquisition will be an initial project expense necessary to secure the areas intended to be used for project facilities. To some degree the acquisition costs may be spread throughout the life of the project. The project intends to use an option agreement as its preferred instrument in securing its hold on the project facility areas.

Initial estimates of potential project-affected people (PAP) and households indicate the number of PAP within facilities footprint areas at 7,861 individuals and 1,642 households. Any additional buffer zone from facility boundaries would expand somewhat this PAP estimate.

Project facility areas not in the MPSA will need to be reclassified as heavy industrial, as the Municipality of Pantukan has not allocated any land for this particular land use. Reclassification is a prerequisite for land conversion.

Republic Act 6657 or the Comprehensive Agrarian Reform Law (CARL) and other related directives, provide the guidelines for land conversion in the Philippines. Conversion is defined as the act of putting a piece or parcel of land into a type of use other than that for which it is currently being utilized. Based on review of secondary data, no project facilities will be located in areas that are non-negotiable for conversion. DAR (Department of Agrarian Reform) is the primary agency mandated to oversee the conversion of lands for other uses.

Topography in the deposit area has steep gradients and carved valleys draining toward the Kingking River. Natural slopes throughout the area range from zero percent up to 50 percent or greater, and most of the project area lies between sea level and 1,000 meters in elevation (amsl).

The physical, chemical, biological, and social environment of the Project area is summarized in Section 1.11. See Figure 1-1 for an aerial photograph of the site.

Figure 1-1: Aerial Photograph of Site Looking Southwest (Kingking River shown)

1.2.2 Location

The King-king Gold-Copper Project is located in the Philippines on the island of Mindanao, in the Municipality of Pantukan, Province of Compostela Valley. It is on the eastern side of Davao Gulf, approximately 92 km by paved road from Davao City. The proposed mine is approximately 10 km from the coast adjacent to the Kingking River. The centroid of the proposed pit cone is located at the approximate geographical coordinates 7°11'31"N Latitude and 125°58'24"E Longitude. See Figure 1-2 for a location map of the Project.

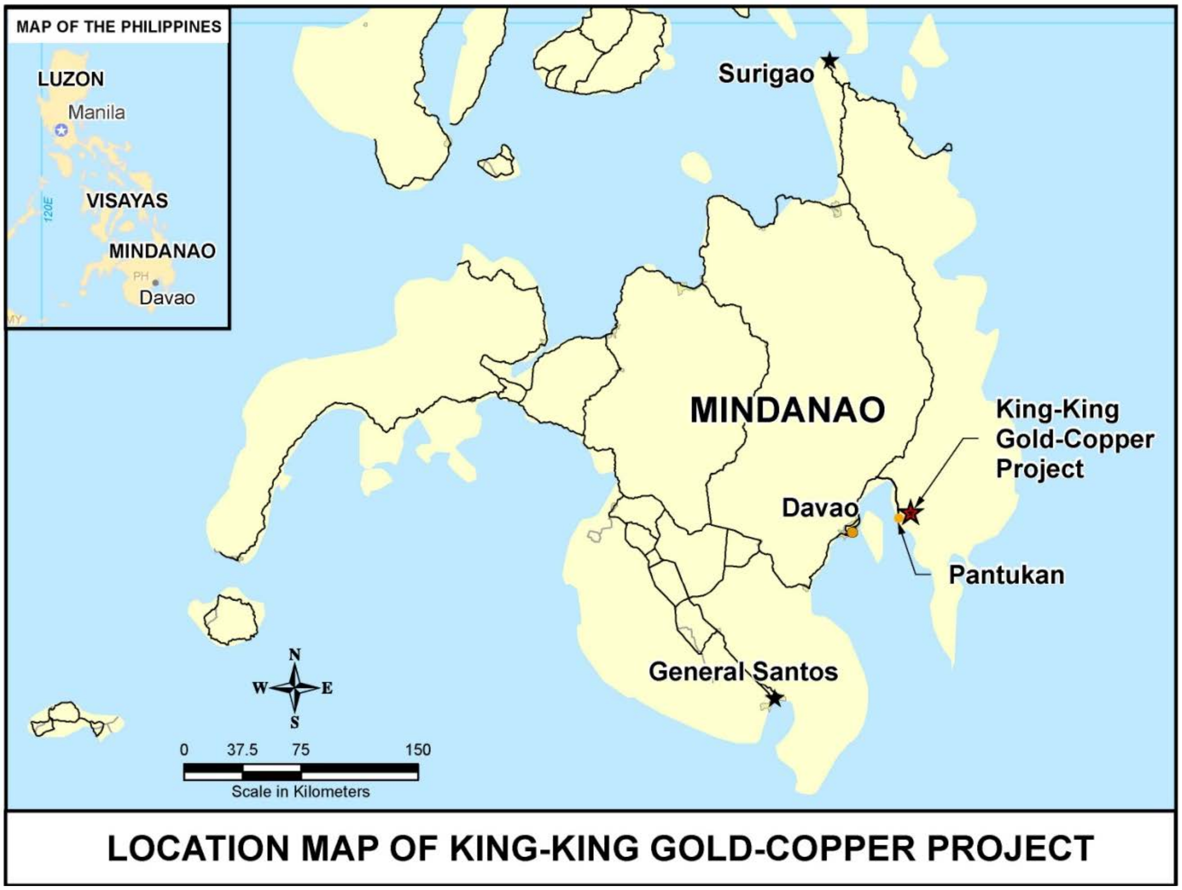

Figure 1-2: Project Location

Topography in the deposit area is very steep, with deeply carved valleys with steep gradients draining toward the Kingking River. Natural slopes throughout the area are on the order of 50% or greater, and most of the project area lies between sea level and 1,000 meters in elevation (amsl).

A large percentage of the natural vegetative cover in the area has been removed via logging and replaced with cultivated hillsides or grassland. However, steep slopes are heavily vegetated with trees and shrubs. Banana tree plantations are present in the coastal plain and extend into the foothills and valley bottoms for a limited distance.

Many site locations were evaluated for the Tailing Storage Facility (TSF), Valueless Rock Management Area (VRMA) and mill facilities. The locations of the facilities selected for the PFS are shown in Figure 1-3. All facilities have alternate locations.

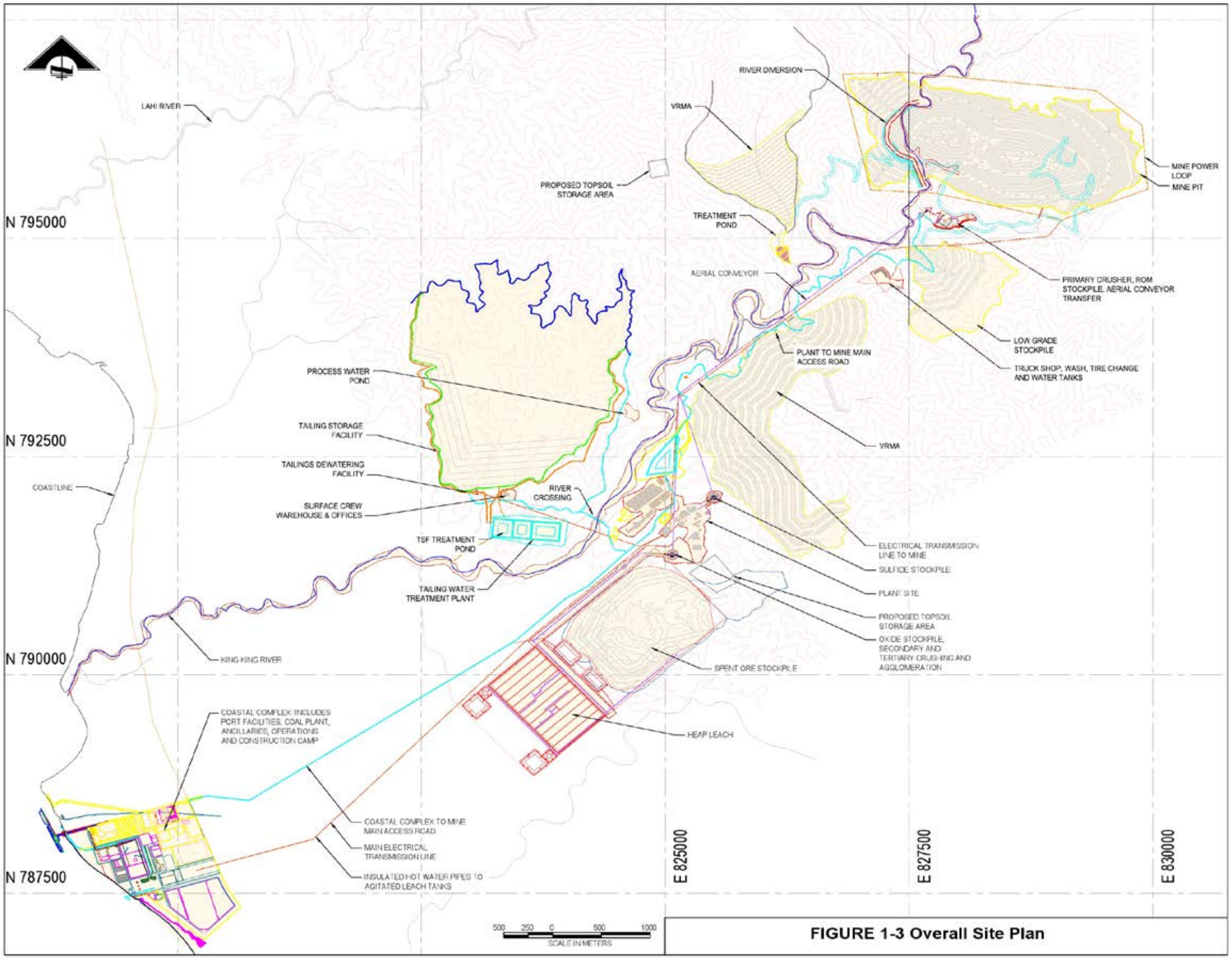

Figure 1-3: Overall Site Facility Map

1.3 HISTORY

The following are highlights in the history of the King-king Copper-Gold Project:

1966-1968	NADECOR discovered the King-king mineralization anomaly;
1969-1972	Mitsubishi Mining Corporation drilled 54 surface diamond drillholes and conducted metallurgical studies;
1981	NADECOR entered into an Operating Agreement with Benguet Corporation (Benguet);
1991-1994	Benguet drilled 69 diamond core holes and 25 reverse circulation (RC) holes; an in-house feasibility study was completed. A draft EIS was completed;
1992	The MPSA was signed among NADECOR (as Leaseholder and Contactor), Benguet (as Operator) and the Philippine Government;
1995-1997	Echo Bay Mines, Inc. obtained an option on the King-king project and drilled 128 holes (52,718 meters) and completed a Feasibility Report. All Echo Bay data were acquired by Kinross Gold, which waived its option to proceed with the King-king project;
2009	NADECOR and Russell Mining and Minerals, Inc. (RMMI) signed a letter of intent (LOI) to work together to develop the King-king project;
2010	The Department of Environmental and Natural Resources (DENR) ordered NADECOR to develop and start a work program, and Benguet to hand over possession in order to allow for immediate resumption of the project. NADECOR and RMMI signed a memorandum of understanding (MOU) to advance the project together through an earn-in process for RMMI to acquire in phases an interest in aggregate (direct and indirect through a Philippine law compliant structure) up to 60% of the project. Ratel Gold Limited (a CGA Mining Limited spinoff) and RMMI published an NI 43-101 compliant resource estimate for King-king. RMMI, NADECOR and Benguet reached a settlement agreement, wherein Benguet relinquished their right, title and interest in the project and in the Operating Agreement.
2011	RMMI assigned its interests in the Project to Ratel Gold Limited and took over management of Ratel and changed its name to St. Augustine Gold and Copper Limited, a publicly traded company on the TSX, as a part of the reverse takeover. NADECOR and SAGC signed a Technical Services Agreement, Onshore and Offshore Services Agreements, and an Interim Funding Agreement. The project was advanced in areas of

	social/environmental studies, drilling programs, engineering studies, and community relations. At the same time, NADECOR formed several companies that are currently intended to be the joint venture companies for the King-king project. SAGC updated the 2010 NI 43-101 compliant resource estimate with new information from the feasibility studies in progress and with new metal prices. An earlier settlement agreement with Benguet was amended for accelerated performance and discharge to the benefit of all parties. A MOU with the project area's indigenous people was signed.
2012	The Preliminary EIS was submitted for comments to the DENR (EMB). The DMPF was completed and submitted to the DENR (MGB). Substantial engineering optimization and trade-off studies were completed. Preliminary feasibility studies and a draft NI 43-101 compliant Technical Report were also completed.

1.4 GEOLOGICAL SETTING AND DEPOSIT TYPE

The King-king deposit is a porphyry copper-gold deposit hosted primarily by porphyritic hornblende diorites, submarine volcanic rocks, and volcanoclastic sediments. The intrusive rocks are believed to be Miocene in age, while the volcanic wall rocks are Cretaceous to early Tertiary. Copper and gold mineralization occurs at or near the apex of the composite diorite intrusive complex within the intrusive rocks and extending well into the surrounding wall rocks.

The majority of the sulfide copper mineralization in the King-king deposit consists of chalcopyrite and bornite, with lesser amounts of chalcocite, digenite, and covellite. Rapid regional uplift and erosion likely caused the nearly complete removal of a classical leached cap and eroded or prevented the development of typically thick oxide and supergene enriched zones such as those found in other major porphyry deposits. Copper mineralization in the oxide zone is observed in silicates and phosphates. Copper silicates are the most abundant oxide mineral group present, with copper silicates minerals containing MgO and FeO being the most prevalent of this group in the oxide zone. Gold is relatively abundant in the oxide zone, in free form formerly in association with the original copper and iron sulfides before they oxidize. Gold also occurs in the sulfide zone of the deposit in free form in close association with bornite and as exsolution intergrowths in other sulfides, particularly pyrite and chalcopyrite. Native gold is occasionally observed on fractures and in quartz veinlets.

In general terms, the King-king gold-copper deposit is consistent in type and form with other bulk-tonnage copper-gold porphyry deposits of the Philippines and elsewhere in the world. The deposit is low in pyrite, averaging less than one percent by volume FeS_2. This is reflected by the relative absence of a pyrite halo that is commonly developed around many porphyry copper deposits. For process development purposes, two types of mineralization are considered: sulfide and oxide (which includes mixed oxide-sulfide material).

1.5 EXPLORATION STATUS, DRILLING, SAMPLE PREPARATION AND SECURITY

Exploration of the King-king deposit has spanned several decades and represents the efforts of numerous companies and individuals. A significant portion of past work focused on drilling to explore, define and confirm the economic potential of the property. The interpretation of the exploration work performed to date indicates that the King-king deposit is a significant copper-gold porphyry system with the potential to become an economically profitable project. The drilling performed through 1998 (Echo Bay period) has also been used to develop an NI 43-101 compliant mineral resource for the deposit, as presented in Section 14 of the Technical Report.

The exploration data provided by previous owners was validated by SAGC and its contractors. The data was used to assist with other analyses.

Three companies completed exploration-level drilling campaigns on the King-king property - Mitsubishi Metal Mining Corp. (Mitsubishi), Benguet Corporation (Benguet), and Echo Bay Mines Ltd. (Echo Bay). The database provided to Independent Mining Consultants (IMC) represents 276 drillholes totaling 89,922 meters of diamond core and reverse circulation (RC) holes.

In addition to this historic drilling, SAGC commissioned 14 holes in 2011: three holes (SAG-01 through SAG-03) designed to further evaluate local areas of the deposit for enhancements to mineral resource estimation (and for metallurgical testing), six holes (SAGT-01 through SAGT-06) to gather geotechnical data for pit slope design, one hole to provide samples for further metallurgical testing (SAM-01), and four holes to provide hydrogeological data for open pit dewatering well design. The total depth of the 14 holes is 5,980 meters.

Estimates of mineralized tonnage and grade for the King-king deposit have historically been based upon assays derived from drill intercepts. Approximately 33,660 samples were collected over the course of the project and processed by four separate analytical laboratories that include Benguet's in-house laboratories at Dizon and Balatoc, McPhar Laboratory in Manila and Inchcape Laboratories in Manila. The sample preparation was completed by the companies previously working on the project.

Sample preparation and analysis procedures for the Benguet and Echo Bay drilling campaigns were reviewed and deemed acceptable. Similar procedures for the Mitsubishi drilling program of 1969-1972 were not available for review, nor are the sample security procedures (chain of custody) known for this program. The chain of custody procedures employed by Echo Bay is believed to have been adequate.

1.6 METALLURGICAL TESTING

Prior metallurgical work by various laboratories under contract with Benguet and Echo Bay was reviewed to determine the scope and direction of the metallurgical work completed for this study. The current process design is mainly based on test work performed by AMEC-Australia and Leach, Inc. in Tucson, Arizona, USA under contract with SAGC. The metallurgical test programs consisted of a series of comminution, flotation, settling, gold deportment and leaching tests on mill feed, flotation tailing, and heap leaching of drill core samples.

Core samples for metallurgical testing were selected to represent the orebody based on the resource and mining schedule developed in November 2010. Samples were classified as sulfide ore or oxide ore, with oxide ore defined as containing copper in oxide form in excess of 35% of the total copper content.

1.6.1 Comminution Tests

The King-king rock mineralization exhibits variable rock competency and ball mill grindability. Sulfide samples had the lowest Axb, averaging 38, and highest Bond ball mill work indices (BWi), averaging 14.2 kWh/t. Oxide samples, scheduled for early processing, were the least competent, exhibiting higher Axb averaging 67, and lower BWi averaging 10.9 kWh/t.

1.6.2 Primary Grind and Regrind Sizes

The grind size optimization flotation tests were conducted at P_{80} values of 150 µm, 106 µm, 75 µm and 53 µm to determine the optimum primary grind size for sulfide composite samples. The recovery of total copper increased from 81% to 85% as grind size decreased from 150 µm to 106 µm but further grinding to a P_{80} of 53 µm did not have a significant effect on the recovery of total copper. Regrind tests were performed at P_{80} of 20 µm and P_{100} of 20 µm. The results indicate that a finer regrind improves the cleaner concentrate copper grade but at a lower copper recovery.

1.6.3 Flotation

The collectors PAX, SIBX, A404 and A3302 were tested at the optimum grind, all at a dosage of 40 g/t. The results demonstrated that the PAX and SIBX collectors yielded the highest total copper, gold, and mass recoveries. The optimum dosage for SIBX was found to be 25 g/t in the rougher stage, with 13 g/t more added in the cleaning stages.

Rougher flotation recoveries and concentrate grades significantly increased by raising the pH to 9. Raising the pH further to 10 or 11 contributed minimally to the recovery of gold and copper. No improvement in the grade of the final copper concentrate was observed by changing the pH in the cleaning stages.

Several cleaner flotation tests, including locked cycles tests were conducted on composites and variability samples. The flowsheet developed includes a rougher stage, 3 cleaning stages and a cleaner scavenger stage to treat the 1st cleaner tailing. Regrinding is performed on the rougher concentrate to meet final concentrate grade requirements. Final flotation tailing come from the rougher and cleaner scavenger stages.

Final concentrates from the locked-cycle tests yielded the results shown in Table 1-2. Detailed analyses of the concentrates show that arsenic may be a penalty concern, reaching a high of 3,700 ppm in the LOM composite. Other penalty elements of concern are fluorine in the Year 2/3 composite and antimony in the Year 4/5 and Year 6/10 composites. Levels are not high enough to cause rejection at the smelter.

Table 1-2: Steady State Results of Locked Cycle Testwork

Composite	3rd Stage Conc. Grade		3rd Stage Conc. Recovery	
	Copper (%)	Gold (ppm)	Copper (%)	Gold (%)
Oxide	23	61	25	53
Sulfide	25	39	71	51

1.6.4 Leaching of flotation tailing

Twelve variability flotation-tailing samples were leached at 35% solids, 50°C, and 50 kg of acid per ton of ore, for 12 hours. The results show that the dissolution of copper (total or WAS) begins to decline after 4 hours and reaches completion after 6 hours for most of the samples. After 12 hours, the total copper recovery ranged between 20% and 94.4%. The recovery of non-WAS copper ranged from 19.7% to 78.1%. The acid consumption levels were estimated to be about 25 kg/t, based on the types and relative abundance of acid consuming gangue minerals in the rock. This consumption rate is consistent with the results of laboratory column leach tests.

Several tests to float oxide copper minerals, including sulfidization and use of hydroxamate collectors, PAX and SIBX were conducted with poor results. The best approach was to use PAX with oxide ore to collect any floatable sulfides, with the recovery of oxides left for later stage leaching of flotation tails.

1.6.5 Gold Deportment

The presence of gold in an oxide composite was examined. When ground to 80% finer than 106 microns, 84% of the gold was in the -38 micron fraction and was not analyzed for deportment or amenability to gravity separation. The +38 micron fraction was subjected to amalgamation test, heavy media separation and magnetic separation. The results showed that 4.75% of the total gold is liberated and may be amenable to gravity separation.

1.6.6 Column Leach Tests

Column leach tests were run on oxide ore samples representing heap leach ore to be placed on the heap leach pad each year for the first six years of heap leach operations. Typical results are shown in Figure 1-4, which plots the recovery of acid soluble copper (Cu(AS)hot) against leaching time for 5 columns. Recoveries from 80 to 90% were typical. The average recovery from 23 column tests was 77.2 % after 38 days, 78.4 % after 45 days, and 79.4 % after 52 days. It was concluded that a 60-day operational leach cycle is adequate for economic recovery of the copper.

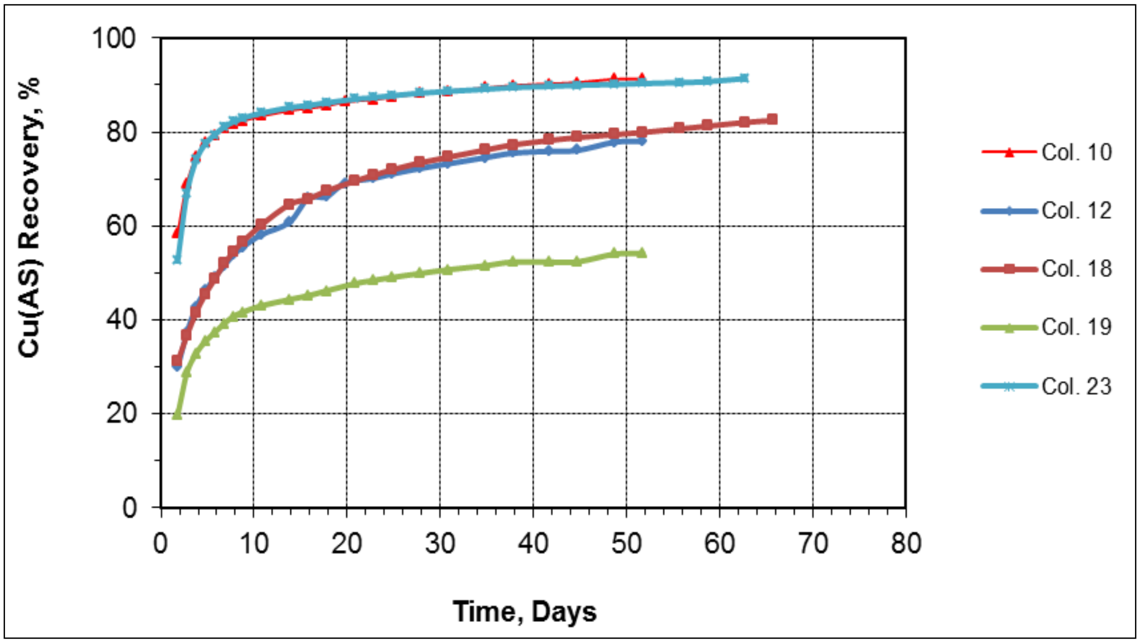

Figure 1-4: Recovery/Time Plot (Year -1)

1.7 MINERAL PROCESSING AND RECOVERY METHODS

The King-king processing facility will recover copper by conventional flotation, by agitated leach of the flotation tails, and by heap leaching of oxide predominant copper ores. Dissolved copper will be recovered through SX-EW into copper cathodes. Most of the recovered gold will report to copper concentrate, but a fraction will be in bullion form produced by gravity concentration, intensive cyanidation, electrowinning and smelting.

The process design was based on metallurgical testwork performed by AMEC-Australia and column heap leach testwork performed by Leach, Inc. (Tucson, AZ). Figure 1-5 is a simplified schematic of the process for the sulfide plant and heap leach operations. The sulfide flotation plant will have a capacity of 60,000 tpd at an availability of 92%, with projected head grades and recoveries that are summarized in Table 1-3. The heap leach operation will have a capacity of 40,000 tpd and will be in operation for thirteen years.

Table 1-3: Metal Recoveries Used for Mass Balance Simulation

Metal	Head Grade (AMEC)	Product	Recovery, %
Cu, Total	0.39 – 0.62 %		
Cu, in sulfides	0.14 – 0.24	Cu Concentrate	86
Au	0.39 – 0.61 g/t	Cu Concentrate	60
		Au Bullion	17

1.7.1 Grinding, Flotation and Agitated Leaching

Run-of-mine (ROM) ore will be crushed to 5½ inches by a gyratory crusher then transported by aerial and overland conveyors to a stockpile. The crushed ore is reclaimed from the stockpile via a conveyor to the grinding circuit. The grinding circuit will be a conventional semi-autogenous grinding (SAG) mill-ball mill-pebble crusher (SABC) system. The SAG mill will be in a closed circuit with a pebble screen and a pebble crusher. The ball mills will be in a closed circuit with hydrocyclone clusters. The target size distribution is 80% finer than 106 microns. A bleed from the cyclone underflow will be processed for recovery of free gold by gravity concentration and followed by intensive cyanidation.

Flotation of copper in the King-king process plant will be accomplished using two banks of rougher flotation cells to achieve recovery, and three stages of cleaning to meet smelter grade requirements. Rougher concentrates will be reground, if required, to 80% finer than 20 microns. Tailing from first cleaner cells will be reprocessed in a cleaner scavenger stage, to allow disposal of this tailing stream with the rougher bank tailing to final flotation tailing.

The third cleaner stage, a flotation column, will produce the final copper concentrate. A third cleaner scavenger bank will process tailing from the third cleaner flotation column to reduce the circulating load around the column. 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 upon due to operator preference. The column flotation cells may be removed from the design altogether if the feed size proves to be too fine for the column to process.

Reagents to be used in the flotation plant include sodium isobutyl xanthate (SIBX) or potassium amyl xanthate (PAX), or possibly an alkyl dithiophosphate-based reagent as collectors, methyl isobutyl carbinol (MIBC) or equivalent as frother, and milk of lime for pH control.

Slurry bleed streams will be taken from each of the two primary cyclone underflow launders and fed to gravity concentrators to recover free gold. Concentrate from the gravity concentrator will be an intensive cyanidation unit. Gold and silver will be finally recovered from the pregnant solution by electrowinning to produce doré bullion. A bleed of the barren cyanide solution will be taken for disposal through a SO_2-air system that will reduce the weak-acid dissociable (WAD) cyanide down to <50 ppm.

During the treatment of oxide dominant ores, the flotation tailing will be leached to recover acid soluble copper, using 25 kg of sulfuric acid per ton of ore, at 50°C. After going through a counter-current decantation (CCD) wash, the dissolved copper will be recovered by SX-EW.

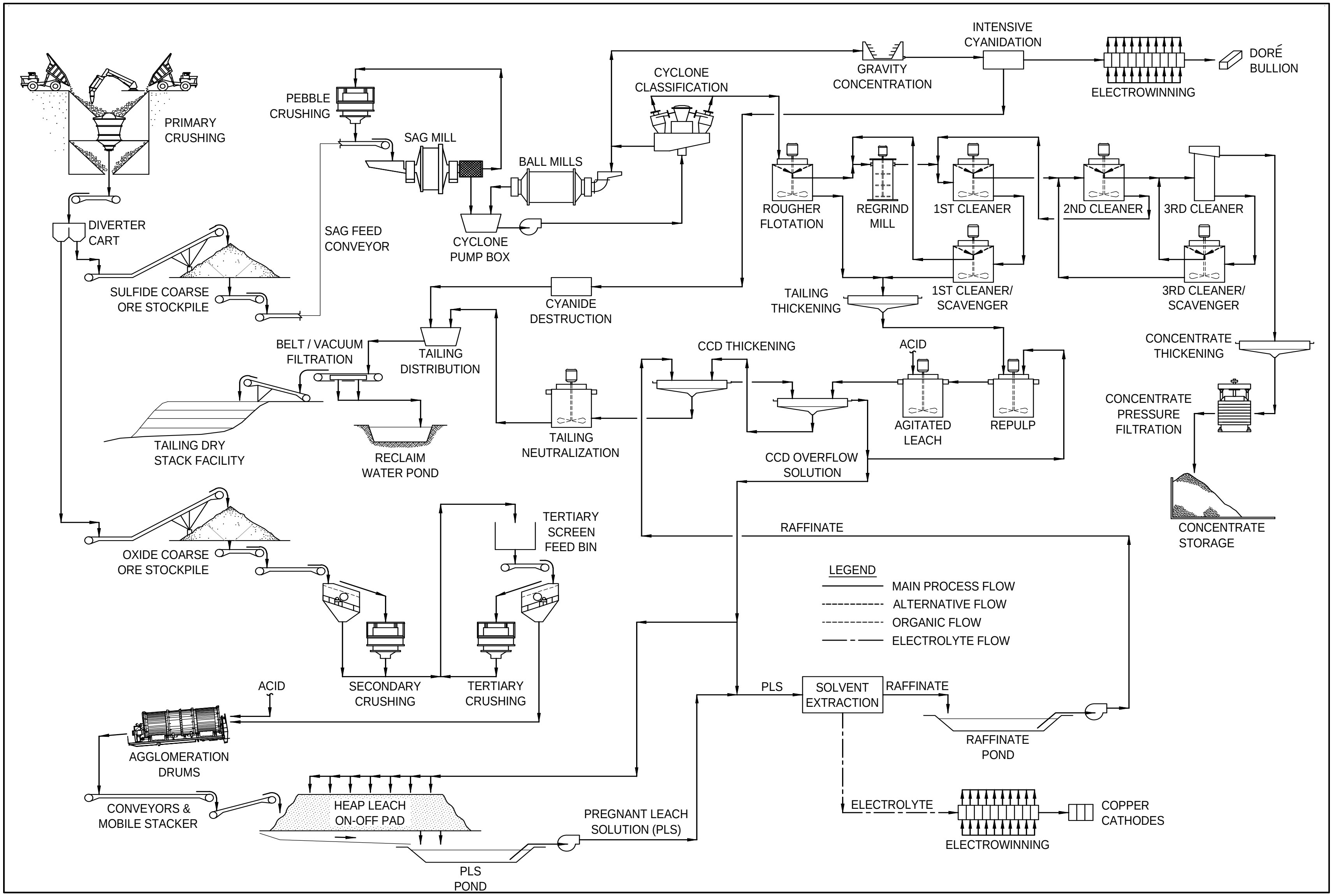

Figure 1-5: Simplified Process Flow Diagram for the King-king Process Facility

1.7.2 Copper Oxide Ore Heap Leaching

Copper oxide ore (with low gold content) will be mined for thirteen years and leached for copper in an on-off leach pad (HLP). Coarse ore from the same primary crusher will be transferred by aerial and overland conveyors to a separate leach operation coarse-ore stockpile that will feed a secondary/tertiary crushing plant. The crushed ore will then be agglomerated with 12.8 kg/t of sulfuric acid and water (CCD overflow solution). The agglomerated ore will be delivered to a stacking conveyor to the heap leach pad and leached with acidified CCD PLS for 60 days. At the end of the leach cycle, the ore will be rinsed and drained then subsequently moved to a spent ore storage facility. The on-off HLP ore stacks will be placed within cell limits to heights of 6 m on grades of less than 5% so as to achieve stability. The edges of the ore stack will be at the natural angle of repose.

The on-off HLP and the Spent Ore Storage Facility (SOSF) will be designed with a minimum static Factor of Safety (FOS) of 1.3. The King-king site is located in an area of active seismicity; therefore, facilities will be designed to resist seismic (earthquake) loads. The King-king site is in an area of high precipitation and moderately high evaporation resulting in a net precipitation environment. The design incorporates conservative measures for comprehensive solution management including measures to control excess influx of meteoric waters into the heap. Geosynthetic liner systems are used for environmental containment to prevent contamination of surface or groundwater by acid solutions used in the copper leaching process. The pond system for the HLP will be designed to store runoff from a 100-yr 24-hr storm event (310 mm) plus the expected drain down volume from a 12-hr power outage. Similarly, the ponds for the SOSF are designed to store the runoff from a 100-yr 24-hr storm event. In addition, temporary removable liners on the surface of the ore will be used to exclude meteoric water from the HLP.

1.7.3 Tailing and Water Systems

Final mill tailing will be thickened and filtered using horizontal vacuum belt filters (HBF). The filtered tailing will then be placed onto a dry-stack tailing storage facility. The filtrate will be recycled to the mill together with overflow from the tailing and concentrate thickeners. A balance of fresh water at a rate of 674m^3/hr will be required to balance moisture in from the heap and mill feed, and moisture out in the mill concentrate, dry stacked tails, and spent ore from the oxide heap.

1.8 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

1.8.1 Mineral Resource

Table 1-4 shows the mineral resource for the project.

Table 1-4: King-king Mineral Resource (August 9, 2011)

Ore Type/Resource Class	Ore Ktonnes	Eq Cu (%)	Tot Cu (%)	Sol Cu (%)	Gold (g/t)	Eq Au (g/t)
Measured Mineral Resource						
Oxide Ore	39,513	1.180	0.431	0.266	0.535	0.843
Sulfide Ore	80,829	0.551	0.258	0.037	0.427	0.803
Total Measured Resource	120,342	0.758	0.315	0.112	0.462	0.816
Indicated Mineral Resource						
Oxide Ore	122,350	0.868	0.334	0.203	0.382	0.620
Sulfide Ore	719,560	0.439	0.230	0.029	0.305	0.640
Total Indicated Resource	841,910	0.501	0.245	0.054	0.316	0.637
Measured/Indicated Mineral Resource						
Oxide Ore	161,863	0.944	0.358	0.218	0.419	0.675
Sulfide Ore	800,389	0.450	0.233	0.030	0.317	0.657
Total Meas/Ind Resource	962,252	0.533	0.254	0.062	0.334	0.660
Inferred Mineral Resource						
Oxide Ore	33,303	0.747	0.276	0.160	0.337	0.534
Sulfide Ore	155,513	0.373	0.202	0.024	0.249	0.544
Total Inferred Resource	188,816	0.439	0.215	0.048	0.265	0.542

Notes:

Eq Cu (oxide) = Total Copper + 1.400 x Gold, Cutoff = 0.30% Eq Cu

Eq Cu (sulfide) = Total Copper + 0.686 x Gold, Cutoff = 0.15% Eq Cu

Alternatively, as Equivalent Gold:

Eq Au (Oxide) = Gold + 0.714 x Total Copper, Cutoff = 0.22 g/t Eq Au

Eq Au (Sulfide) = Gold + 1.458 x Total Copper, Cutoff = 0.22 g/t Eq Au

Total Material in Cone Shell 1,736,371 Ktonnes

Waste:Ore Ratio 0.80 (Inferred as Waste)

Waste:Ore Ratio 0.51 (Inferred as Ore)

1.8.2 Mineral Reserve

The mine and plant production schedules define the mineral reserve for a mining project. Table 1-5 presents the mineral reserve for the King-king Project based on the production schedules.

The mineral reserve amounts to 617.9 million tons at 0.300% total copper and 0.395 g/t gold. For this reserve estimate, measured mineral resource was converted to proven mineral reserve and indicated mineral resource was converted to probable mineral reserve.

Table 1-5: Mineral Reserve

Reserve Classification	Ktonnes	Tot Cu (%)	Sol Cu (%)	Gold (g/t)	NSR (US$)
Proven Mineral Reserve					
Heap Leach Ore	17,791	0.340	0.197	0.132	16.53
Oxide Mill Ore	21,674	0.514	0.328	0.849	45.36
Sulfide Mill Ore	52,942	0.305	0.044	0.543	24.92
Low Grade Mill Ore	6,734	0.184	0.027	0.218	10.80
Total Proven Reserve	99,141	0.349	0.132	0.514	26.92
Probable Mineral Reserve					
Heap Leach Ore	77,373	0.305	0.172	0.145	14.81
Oxide Mill Ore	45,440	0.393	0.259	0.745	35.30
Sulfide Mill Ore	345,715	0.288	0.037	0.398	20.48
Low Grade Mill Ore	50,247	0.191	0.023	0.211	10.93
Total Probable Reserve	518,775	0.290	0.075	0.373	20.01
Proven/Probable Mineral Reserve					
Heap Leach Ore	95,164	0.311	0.177	0.143	15.13
Oxide Mill Ore	67,114	0.432	0.281	0.779	38.55
Sulfide Mill Ore	398,657	0.290	0.038	0.417	21.07
Low Grade Mill Ore	56,981	0.190	0.023	0.212	10.91
Total Prov/Prob Reserve	617,916	0.300	0.084	0.395	21.12

1.9 MINING METHODS

The King-king mine will be a conventional open pit mine. Mine operations will consist of drilling large diameter (32 to 46 cm) blast holes, blasting with either explosive slurries or ammonium nitrate/fuel oil (ANFO) depending on water conditions, and loading the ore onto large off-road trucks with large cable shovels and wheel loaders. Ore will be delivered to the primary crusher and valueless rock to the VRMA facilities. A low-grade ore stockpile will store marginal ore for processing at the end of commercial pit operations. A fleet of track dozers, rubber tired dozers, motor graders, and water trucks will maintain the working areas in the pit, VRMA area, and the roads.

The mine plan was developed to deliver ore at approximately 100,000 tons per day, split between 40,000 tpd to the heap leach facility and 60,000 tpd to the mill. This split is based upon economic parameters developed for mill ore and heap leach ore. LOM average throughput of heap leach plus mill ore is 73,000 tons per day. The mining rate will be approximately 178,000 tpd. The heap leach process is expected to start from 9 to12 months before the mill. Heap leach finishes approximately 13 years into the project, while the mill continues to treat sulfide dominant ore until the end of mine life. Actual production varies year by year due to changes in ore hardness.

AMEC Environment & Infrastructure (AMEC) has developed a range of credible overall slope angles for pit development at the King-king Project commensurate with a scoping-level study. The study utilized drillhole data collected from five oriented core drillholes and from three geohydrology drillholes placed in the predicted final pit walls. This study also used results from unconfined compressive strength (UCS) tests conducted on thirty selected intervals of oriented core from these five holes. Bench design and kinematic analyses are not included as part of the present study. A detailed open pit design and recommendations report, including bench design parameters, will be provided by AMEC in a later phase to support the King-king Project Feasibility Study. However, it should be noted that the interaction of the pit walls with major geologic structures such as faults and shear zones is not included in the present study. The structural model for the King-king Project has been completed. The incorporation of these structures in the geotechnical pit design will be included in the Feasibility Study report. Therefore, the overall slope angles provided herein may be steeper or flatter in some sectors of the pit if favorable or unfavorable geologic structures may exist.

The final pit design is based on a floating cone run at $2.50 per pound copper and $833 per troy ounce gold. Six mining phases are incorporated into the design to mine the pit from the initial starter pit to the final pit limits. The phase designs include haul roads and adequate working room for large mining equipment. The in-pit roads are 33m wide at a maximum grade of 10%. This width will accommodate trucks up to the 230-ton class such as Caterpillar 793 trucks.

1.10 INFRASTRUCTURE

The major support infrastructure includes ancillary buildings, roads, power distribution, communications, water management, shipping, and living facilities for construction and operations personnel. Primary areas of the project requiring this infrastructure are:

- Coastal complex – Includes power plant, ship loading/unloading, bulk material storage (coal and concentrate for example), administration, warehousing and storage, laboratory, medical / fire / rescue, and living quarters.
- Mill – Includes crushing, grinding, sulfide flotation, agitated leach, CCD thickeners, concentrate filtration and SX-EW.
- Heap leach – Includes secondary and tertiary crushing, agglomeration, on/off leach pad, and spent ore stockpile.
- Drystack tailing storage facility – Includes tailing dewatering plant and stacking system for placing dewatered tailing.
- Mine support facility – Includes primary crusher, truck shop, tire shop and fueling bays, truck wash, mine administration, bulk ANFO plant, and powder magazine.

1.10.1 Ancillary Buildings

The coastal complex will contain the main administration building as well as the primary warehouse and laboratory for the mine. Additionally, a medical complex of suitable size to serve the operation of the power plant, process and mine areas will be provided. Smaller support buildings will be located close to each area as required for efficient operations. These will include security posts, warehouses, and various maintenance structures.

1.10.2 Roads

A new two-lane eight-meter wide gravel access road will link the mine with the process plant and the coastal complex and will include widened passing areas. Additional secondary access roads will service other infrastructure areas such as pump stations, powder magazine, and power substations. A wet river crossing will be constructed across the Kingking River opposite from the process plant to access the TSF and dewatering plant. A security gate will be located on the road at the entrance to the mine property. Mine haul roads will be separate from other traffic for safety. The 33-m wide haul-truck roads will serve the primary crusher, truck shop and VRMA.

1.10.3 Power Distribution

A new 160 MW coal fired power plant and 29 MW HFO-fueled generator will be constructed at the coastal complex. A 138 kV transmission line will deliver power from the power plant to the plant substation adjacent to the mill building. The 34.5-kV distribution lines will further distribute power to all plant and mine facility locations.

1.10.4 Communications

High quality voice and data transmission will be provided throughout the site via local area network (LAN) and microwave transmission from a hub in Davao City.

1.10.5 Water Management

Process make up water will be supplied to the project primarily from a well field located in the coastal plain southeast of the plant site adjacent to the Kingking River. Water from these wells will be pumped to fresh and firewater tanks located near the plant.

Potable water at the mill site will be obtained by treating fresh water in a treatment plant to meet drinking water standards. Fresh water wells and treatment packages will provide potable water piped to the various site locations. At the beginning of construction, wastewater will be collected and transported to an off-site treatment facility. Packaged wastewater treatment plants will come on line during the construction period and be utilized for on-site treatment. During routine operations, waste water from the camp and other facilities will be treated in these plants prior to discharge.

Water from horizontal pit depressurization drains is proposed to be pumped and released into the Kingking River south of the open pit. Water from pit de-watering would be pumped to a sediment pond and then released into the Kingking River at the same location as the de-pressurization flow.

Water treatment is planned for the VRMA runoff water commencing in Year 5. Humidity cell test data indicates that potentially acid generating material will not begin to generate acid for at least five years. Excess water from the leach pad will be treated using a reverse osmosis (RO) unit to meet applicable water quality standards prior to discharge. Storm water will be managed to reduce solids to acceptable levels prior to discharge.

1.10.6 Shipping

The port will handle incoming coal, sulfuric acid, limestone, grinding media, fuel, reagents and replacement equipment or parts, and outgoing concentrate and cathodes. The main warehouse for the process plant and mine will be located at the port due to the limited availability of suitable area at the respective sites. Smaller warehouse or storage areas will be located as needed for daily or weekly support.

1.10.7 Construction-Operations Camps

Although local labor will be used to the maximum extent possible, there will be a need to provide living quarters for workers living outside a commutable distance. This is particularly true during the construction period. To accommodate this need, a temporary construction camp and a permanent operations camp will be located at the port complex. The construction camp will be disassembled and removed at the completion of major construction. The permanent operations camp will house 125 workers. Dining, recreation, and laundry buildings are anticipated to serve both camps. A bus terminal will be integral with the camp for transport of workers between the worksites and the coastal complex.

1.11 ENVIRONMENT AND PERMITTING

1.11.1 Physical Environment

The topography in the Project area is steep and rugged with elevations ranging from 260 to 950 meters amsl. The climate is tropical with daytime temperatures ranging from 18 to 35 ºC, and annual precipitation ranging from 1,800 to 3,200 mm. Two weather stations were installed in the Project area in 2011 to collect on-site meteorological data, one on the coastal plain and one in the uplands.

Five soil types were identified in the region, including Banhigan, Camansa, Umingan, San Manuel, and Catanauan, each of which is a mix of silt, clay, and loam.

The project area itself is located largely within the Kingking watershed, which is nearly 20 km long with an average slope of 58 m per km. Small-scale mining activity within the Project area has significantly changed the erosion and sedimentation rates of the lower Kingking watershed, and has significantly impacted water quality. There are two groundwater regimes within the project area: an alluvial aquifer along the coastal plain, and a bedrock fracture-controlled aquifer in the mountains.

Noise levels in most residential areas measured above applicable limits (55 decibels) especially during daytime. Noise levels in non-residential levels were below the limits.

1.11.2 Chemical Environment

The soils are slightly acidic (pH 5.4 - 6.12) and most samples showed high aluminum and iron concentrations (comparable to bedrock composition).

Water samples from the Kingking River showed relatively high total suspended solids (TSS), copper, mercury, cyanide and total coliform concentrations, above applicable water quality criteria. Groundwater quality is generally good within the mountains; while unsanitary sewage disposal has directly impacted portions of the alluvial aquifer in the lowland areas.

Air quality in the Project area is deemed to be good with particulate matter, NO_2, SO_2, CO and Pb all below Philippine standards.

1.11.3 Biological Environment

Six general types of vegetation were recognized: open-canopy mid-mountain forest, brush land, wooded grassland, agricultural plantations (coconut and banana), riparian-riverine vegetation, and coastal vegetation. A total of 301 species were recorded during the biological environment survey, with over half of the species being trees. Twelve species are considered vulnerable or critically endangered.

A total of 74 bird species, 17 mammal species and 10 reptilian species were identified in the region. Several of the species found in the region are listed as near-threatened, vulnerable or protected, including 11 bird species, 2 mammal species, and 5 reptile and amphibian species.

Marine studies showed that several species of sea turtles, dolphins, whales, and seabirds live in the area. Sea cows and whale sharks also live in the region. The sea cow species and all species of sea turtle found in the region are listed as endangered. Phyto-, nano-, zoo-, and ichthyoplankton, as well as coral and benthic species were found in abundance.

Mitigation measures are being developed to protect environmentally sensitive species as a part of the Environmental Impact Statement and will be implemented prior to construction and operation.

1.11.4 Social Environment

Based on the 2000 Census, Pantukan has a population of 61,801 people in 13,311 households (69,656 people in 2007). Pantukan is divided into 13 barangays. Barangays Bongbong, King-king, Magnaga, Napnapan, and Tagdangua may be directly impacted by the proposed Project.

About 75% of the population in the Project region is of Visayan origin. Indigenous people account for 7-32% of each barangay's population, and most belong to the Mansaka, Mandaya, Manobo and Bagobo Tribes. Nearly all people in the region speak Cebuano, a local dialect of Visayan.

The main source of livelihood in Compostela Valley is the production of agricultural products, such as rice, coconut, cacao, coffee, papaya, mango, pineapple, durian, and banana. Some residents have fishponds and culture their own fish. Mining, mostly small-scale, is also a major source of livelihood.

College graduates account for one to six percent of the population, while high school graduates account for 10 to 12% of the population. About 50-90% of each barangay's population earns less than 5,000 Philippine pesos (PhP) per month. The unemployment rate is 12.6% in Pantukan.

Electric lighting is used by more than two-thirds of the households. Wood and charcoal are used as cooking fuel by more than 75% of the households. The majority of households use streams, springs, or wells for their water supply.

1.11.5 Permitting

Based on the baseline information collected to date, there are no environmental issues that would prevent the permitting of the proposed operations. The baseline studies supporting the EIS and SEIA have been completed. Monitoring of selected parameters continues to be performed.

1.12 CAPITAL COST SUMMARY

Initial capital costs have been estimated for the King-king Copper Gold Project in compliance with PFS level design based on the associated material quantities, labor cost estimates and equipment quotations. Unit rates have been based on historic data, published sources and inputs from cost consultant experts local to the Philippines. The estimate includes all evaluated sections of the project such as process, tailing, mining facilities, power station and port facilities. The costs also include pre-production mining, owner's costs and contingency. A more detailed initial capital breakdown is located in Section 21.1 of this report as well as in the economic model, Table 22-10.

Table 1-6: Initial Capital Cost

Area	Description	($ Millions)
Process Plant and General Infrastructure	General Site, Mine support infrastructure, waste disposal, primary crushing, aerial conveyors, heap leach, grinding, flotation, SX-EW, agitated leach, tailing dewatering, drystack tailing, water systems, water treatment, on site power distribution, ancillary facilities, EPCM, freight, import duties	$1,082.3
Mine	Contract mining operating costs before the start of production.	$114.9
Power Plant	Power plant and support facilities. Two 80 MW coal fired power generators and 29MW of HFO, and power line to main project substation.	$320.0
Port Facility	Dock Facility, Coal unloading, Concentrate loading, Coastal Complex	$108.8
Owners Costs	Land Acquisition, Construction/ Operating Camps, Environmental Permits, Initial Fills, Owner's Project Management, Security, Early Staffing, Community relations	$175.8
Contingency	Contingency on all parts of the project	$240.1
Escalation	Not included in this estimate	$0
Total CapEx Before VAT		$2,041.9
Value Added Tax (VAT)		$167.2

1.13 OPERATING COST SUMMARY

The operating cost was estimated in compliance with PFS requirements based on a bottom-up methodology. The operating cost summary is provided in Table 1-7 below, which includes the average cost over the initial 5 and 10 years, as well as the Life of the Mine (LOM). In addition, operating costs are summarized per ton of material processed in Table 1-8. These operating costs illustrate that King-king is expected to be a low cost producer. For instance, the operating cost at King-king for the first 10 years of full production is expected to be $0.19/lb net of by-products (gold and silver). This operating cost places King-king among the 25% lowest cost mines worldwide, based on a pro-rata analysis of cost data presented in the 2012 Copper Cost Curves workbook by World Mine Cost Data Exchange. Details of the operating cost are shown in Section 21.2 below, as well as in the "Detailed Financial Model" shown in Table 22-10.

Table 1-7: Operating Cost Summary

Time Period	Units Years	1-5	1-10	LOM(1)
Payable Pounds of Copper	000'000	1,330	2,046	3,079
Mining	$/lb Cu	$0.47	$0.60	$0.80
Processing	$/lb Cu	$0.81	$0.92	$1.06
Operating Costs	**$/lb Cu**	**$1.28**	**$1.52**	**$1.86**
G&A	$/lb Cu	$0.13	$0.16	$0.25
Reclamation & Closure	$/lb Cu	$0.00	$0.00	$0.02
Cash Costs at Mine	**$/lb Cu**	**$1.41**	**$1.68**	**$2.13**
Government Fees	$/lb Cu	$0.17	$0.22	$0.26
Total Cash Costs at Mine	**$/lb Cu**	**$1.58**	**$1.89**	**$2.38**
Shipping, Smelting and Refining	$/lb Cu	$0.11	$0.15	$0.18
Total Costs	**$/lb Cu**	**$1.69**	**$2.04**	**$2.57**
By-Product Credits	$/lb Cu	-$1.66	-$1.85	-$2.17
Consolidated Net Cash Costs	**$/lb Cu**	**$0.03**	**$0.19**	**$0.40**
(1) Includes year -1 heap leach production				

Table 1-8: Summary of Unit Operating Costs per ton of Material Processed

	Total Cost (Life of Mine $million)	Total Tons Processed (Millions)	$/ton processed	Years of Operation
Contract Mining Costs	$2,454.10	**522.8**	$4.69	1 through 23
Heap Leach & SX-EW	$342.80	**95.0**	$3.61	-1 through 12
Concentrator & Tailing	$2,309.60	**522.8**	$4.42	1 through 23
Agitated leach	$612.10	**194.5**	$3.15	1 through 8
G&A, Lab, Port & Custom Duties	$762.46	**522.8**	$1.46	-1 through 23

1.14 ECONOMIC ANALYSIS

The King-king Project economics were prepared using a discounted cash flow model. The financial indicators examined for the project included the Net Present Value (NPV), Internal Rate of Return (IRR) and payback period (time in years to recapture the initial capital investment). Annual cash flow projections were estimated over the life of the mine based on initial and sustaining capital expenditures, production costs, transportation and treatment charges, government fees and taxes, and sales revenue. The life of the mine is 23 years. Metal price assumptions are $3.00/pound copper, $1,250/ounce gold and $25/ounce silver. The after-tax financial indicators based on a 100 % equity case are summarized as follows:

Table 1-9: Economic Indicators After-Tax

NPV @ 0% ($000)	$4,967,678
NPV @ 5% ($000)	$2,588,925
NPV @ 8% ($000)	$1,757,074
NPV @ 10% ($000)	$1,347,097
IRR %	24.0%
Payback - years	2.4

The economic analysis was performed with the assumption that the project will have a six-year income tax holiday. This incentive has been granted to qualified mining and mineral processing projects operating under a MPSA since the Mining Act of 1995. SAGC will apply for this income tax holiday at the appropriate time.

Figure 1-6 below illustrates NPV sensitivity to metals prices, initial capital, and operating cost. This graph indicates that NPV is mostly sensitive to the metal prices and much less sensitive to initial capital and operating cost. As stated above, the base case of the project was estimated at conservative metal prices.

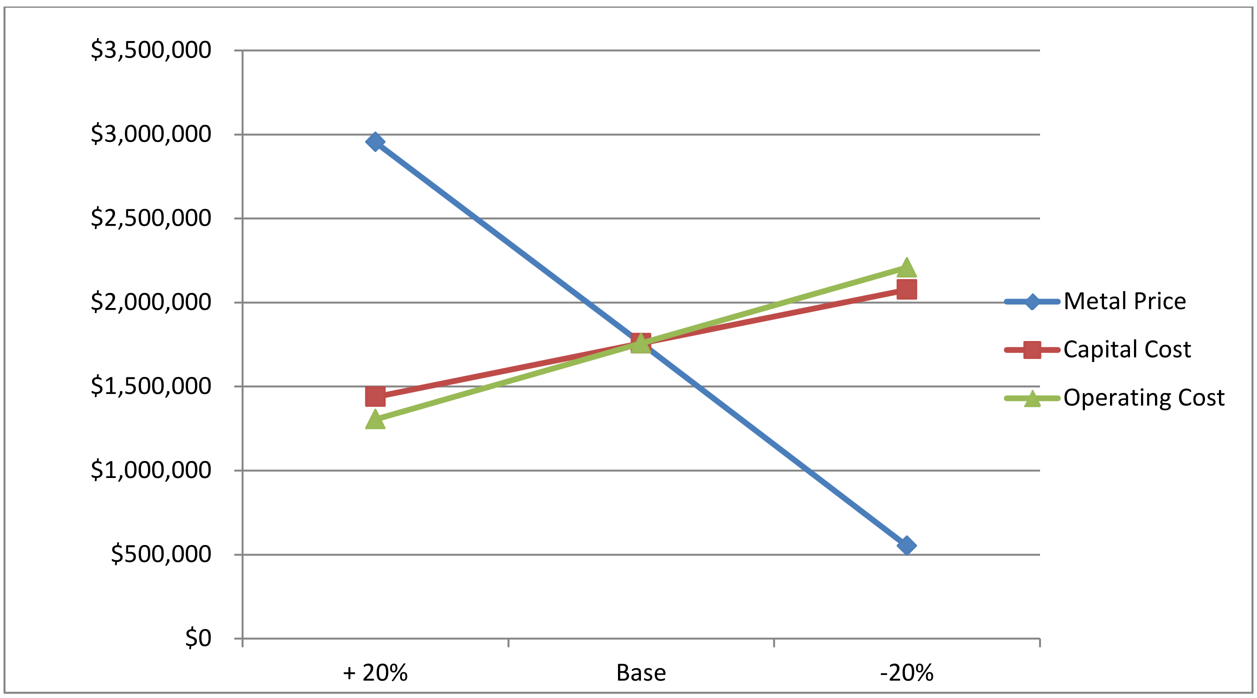

Figure 1-6: After Tax NPV (8%) Sensitivities (000's)

1.15 CONCLUSIONS AND RECOMMENDATIONS

1.15.1 General

The results of this study indicate the King-king project is both technically and economically feasible and demonstrate robust returns even at conservative metal prices. Also, starting up the heap leach operations one year before mill operations provides a financial benefit to the project. Open pit mining methods used in this study are understood and have been applied extensively in the industry.

The project has the advantage of being located relatively close to a large population center, only 10 kilometers from the sea and having a two-lane concrete road within 10 kilometers of the deposit. However, it is located on steep terrain, receives high rainfall, and lacks a reliable grid-supplied power.

The main challenge to processing King-king ore is the presence of a significant amount of copper oxide intermixed with copper sulfide in the deposit which is mined in the first 8 years. Moreover, some of the oxide-dominant materials have gold grades that merit routing to the flotation plant. Once the ore classification and routing schedule was developed, the resulting process plant is more complicated and larger than at typical porphyry copper mines. In addition to a flotation plant, the ore requires leaching of flotation tailing and a heap leach operation to maximize copper recovery. However, these technologies have been proven and operated before at this scale.

M3 recommends that the project be further advanced to feasibility level study. In addition, M3 recommends that SAGC continues to execute their land acquisition plan in a timely fashion.

1.15.2 Economics

This PFS indicates favorable economics for the project at conservative metal prices. As a result, it is recommended to rapidly advance this project to the next phase of development, which is completion of the feasibility study report. Some critical portions of the study are already at feasibility level.

The project economics are summarized below:

- Gold Price – US $1,250 per troy ounce (29% below the EOM Nov 2012 price of $1,762)
- Copper Price – US $3.00 per pound (17% below the EOM Nov 2012 price of $3.62)
- Silver Price – US $25 per troy ounce (27% below the EOM Nov 2012 price of $34.28)
- Average Annual Revenue – US $1.2 billion (during first five years of full production)
- Net Present Value (NPV) – US $1.8 billion at 8% discount rate
- Internal Rate of Return (IRR) After Taxes – 24%
- Payback – 2.4 years

- Initial Capital Cost – US $2.0 billion (includes contingency of US $0.24 billion)
- LOM operating cost (net of metal credits) - US $0.40 per pound of copper (US $0.03 per pound of copper in initial 5 years of full production)

1.15.3 Exploration and Geology

An NI 43-101 compliant mineral resource for the deposit, as presented in Section 14 and 15, has been developed based on drillhole data gathered from drilling programs through 1998.

Fourteen drillholes were recently completed with a total depth of 5,980 meters. New data from these drillholes were obtained to update the geological, geotechnical, metallurgical, and hydrogeological information. At the next stage of the study (feasibility level), the resource geology model update should include the information from these new drillholes to increase the confidence level of the resource and reserve estimates.

1.15.4 Mining

The mining method proposed for King-king is conventional open pit method for bulk mining, which is used extensively in the industry. There are no significant technical challenges to mining at King-king.

This study has developed an updated measured, indicated and inferred resource and a proven and probable reserve. The measured and indicated resource is 962 million tons grading 0.254% total copper and 0.334 g/t gold. The inferred resource is 189 million tons at 0.215% total copper and 0.265 g/t gold. The proven and probable mineral reserve for this project amounts to 617.9 million ore tons at 0.300% total copper and 0.395 g/t gold. These figures equate to 4.1 billion pounds of contained copper and 7.8 million ounces of contained gold within the mineral reserve.

There is potential to add resource and reserve tonnage to the King-king deposit as there are significant quantities of inferred resource where drilling has not found the limits of the mineralization.

1.15.5 Tailing / Geotechnical

The results of this study indicate that dry stacking of tailing material is the most economical and lowest risk option for tailing storage, given the high seismicity and steep topography in the area. Stability analyses were performed on the proposed drystack tailing storage facility. The design meets the Philippine DENR Memorandum Order No. 99-32 requirements.

Pressure and vacuum filtration test work conducted at Pocock Industrial and Amdel Adelaide laboratories indicates 15-17% final moisture content in dewatered tailing should be attainable.

1.15.6 Process Facilities

Processing Technologies used in this study have been proven at large scales in the industry (heap leach and mill ores).

- Gravity concentration to produce gold concentrates applicable to doré metal production by intensive cyanide leaching, electrowinning, and smelting on site;
- Sulfide flotation to produce salable copper chalcopyrite/bornite concentrate containing gold; and
- Tailing leaching of copper oxide minerals with sulfuric acid followed by SX-EW to produce salable copper cathodes.
- Heap leaching of ore containing copper oxide minerals, with very low gold values, using sulfuric acid followed by SX-EW to produce salable copper cathodes.

The milling and process facilities can be expanded within the current process area footprint to accommodate processing additional ore as needed. In the next stage of analysis, some process trade-off studies should be evaluated with regards to optimizing process capital and operating costs.

1.15.7 Infrastructure

The technology assumed in the power plant infrastructure is a circulating fluidized-bed coal-fired boiler, providing a wider range and more flexibility to accept coal from sources that may have different specifications. This is a well-established conventional power generation technology, with a proven track record for efficient energy generation and state-of-the-art environmental controls.

M3 recommends that SAGC continues to evaluate electric power supply options for the project. These include discussions with independent power providers about long term power supply costs. For example, Mindanao suffers from a shortfall in power for its current residential and commercial users. A larger power generating plant supplying power to the King-king project and the Mindanao grid may provide lower cost power than the dedicated plant conceived for the King-king Project.

The proposed port facility as evaluated in this study meets the needs of the project for import of fuel and other consumables as well as export of products.

2 INTRODUCTION

2.1 PURPOSE

This report was compiled by M3 Engineering & Technology Corporation ("M3") for SAGC with respect to the King-king property in Mindanao, Philippines. The purpose of this technical report is to prepare a Preliminary Feasibility Study ("PFS"), including a reasonably executable plan of development for the King-king deposit, and to apply accepted estimation tools to create operating and capital cost estimates for the plan. The financial model incorporates the cost estimates, along with reasonable projections for metal prices, taxes, and other financial elements to predict the economic performance of the project and to analyze the performance using standard economic metrics.

2.2 SOURCES OF INFORMATION

New information, updates to, and review of existing information were provided and performed by the Qualified Persons ("QP's") as shown in Table 2-1.

Table 2-1: Dates of Site Visits and Areas of Responsibility

QP Name	Company	Qualification	Site Visit Date	Area of Responsibility
Joshua W. Snider	M3 Engineering & Technology Corporation – Tucson, AZ	P.E.	N/A	1.1, 1.2, 1.3, 1.10, 1.12, 1.13, 1.14, Parts of 1.15, 2, 3, 4, 5, 6, 18, 19, 21, 22, 23, 24, 25.1, 25.6, 25.8, 25.9, 26.1, 26.7, 26.8, 27
Art S. Ibrado	M3 Engineering & Technology Corporation – Tucson, AZ	Ph.D., QP Member, MMSA	January 25, 2011	1.7, 17, 25.5, 26.5, 26.6
Michael Hester	Independent Mining Consultants – Tucson, AZ	FAusIMM	N/A	1.8, 1.9, 12, 14, 15, 16, 25.3, 26.3
Don Earnest	Resource Evaluation Inc.	P.Geo, SME Registered Member	June 5, 2010 March 23, 2011	1.4, 1.5, 7, 8, 9, 10, 11, 25.2, 26.2
John G. Aronson	AATA International Inc.	Certified Senior Ecologist	February 6 - 9, 2012	1.11, 20
Ronald J. Roman	Leach Inc.	P.E., D.Sc.	N/A	1.6.6, 13.6
Charles C. Rehn	AMEC	P.E., SME Registered Member	N/A	18.4, 18.5, 21.1.4, 21.1.6, 21.1.7, 25.4, 25.7, 26.4
Greg J. Harbort	AMEC Australia, Australia	RPEQ, FAusIMM BE (Met), Ph.D.	N/A	1.6.1-1.6.5, 13.1-13.5

2.3 UNITS AND ABBREVIATIONS

The report considers US Dollars ($) only. Unless otherwise noted, all units are metric. Salable base metals are described in terms of tons or pounds. Salable precious metals are described in grams or troy ounces. Table 2-2 is a list of abbreviations and terms that may be used in this report.

Table 2-2: Units, Terms and Abbreviations

AATA International, Inc.	AATA	G-force (seismic)	g
Above mean sea Level	amsl	Giga (billion)	G
Acidity	pH	Gigajoule	GJ
AMEC Australia	AMEC Au	Gold	Au
AMEC USA	AMEC	Gram	g
Ammonium nitrate/fuel oil	ANFO	Grams per litre	g/L
Ampere	A	Grams per ton	g/t
Annum (year)	a	Greater than	>
Benguet Corporation	Benguet	Hectare (10,000 m^2)	ha
Billion pounds	Glb	Hertz	Hz
Billion years ago	Ga	Horsepower	hp
Billion	G	Hour	h
Biotite diorite porphyry	BDP	Hours per day	h/d
Brinell Hardness Number	BHN	Hours per week	h/wk
Canadian Dam Association	CDA	Hours per year	h/a
Canadian Institute of Mining	CIM	Independent Mining Consultants	IMC
Centimeter	cm	Indigenous Peoples Rights Act	IPRA
Centimeters per second	cm/s	Internal Rate of Return	IRR
Certificate of Ancestral Domain Title	CADT	Intra-mineral dacite porphyry	IDAP
Copper	Cu	Intra-mineral hornblende diorite porphyry	IHDP
Crushed Ore Stockpile	COS	Joule	J
Cubic centimeter	cm^3	Joules per kilowatt-hour	J/kWh
Cubic meter	m^3	Kelvin	K
Cubic meters per day	m^3/d	Kilborn International, Inc.	Kilborn
Cubic meters per hour	m^3/hr	Kilo (thousand)	k
Dacite porphyry	DAP	Kilobyte	kB
Day	d	Kilogram	kg
Days per week	d/wk	Kilograms per cubic meter	kg/m^3
Days per year (annum)	d/a	Kilograms per hour	kg/h
Dead weight tons	DWT	Kilograms per year	kg/a
Decibel	dB	Kilojoule	kJ
Degree	°	Kilometer	km
Degrees Celsius	°C	Kilometer	km
Development Rock Stockpile	DRS	Kilometers per hour	km/h
Dry metric ton	dmt	Kilonewton	kN
Echo Bay Mines Ltd.	Echo Bay	Kilopascal gauge	kPa(g)
Electromotive Force	emf	Kilopascal	kPa
Equivalent (metal grades)	Eq	Kilotons	ktons
Fisher & Strickler Rock Engineering, LLC	FSRE	Kilovolt	kV
Foot/feet	ft	Kilovolt ampere	kVA
Gallon	gal	Kilowatt	kW
Gallons per minute	gpm	Kilowatt hour	kWh
General & Administration	G&A	Kilowatt hours per ton	kWh/t

Kilowatt hours per year	kWh/a
King-King Mines Inc.	KMI
Lead	Pb
Less than	<
Life of Mine	LOM
Litre	L
Litres per hour	L/hr
Litres per minute	L/min
Litres per second	L/s
Load-Haul-Dump	LHD
M3 Engineering & Technology Corporation	M3
Maximum Credible Earthquake	MCE
Maximum Design Earthquake	MDE
Mega (million)	M
Megabyte	MB
Megabytes per second	MB/s
Megapascal	MPa
Megavolt ampere	MVA
Megawatt	MW
Megawatt hours	MWh
Meter	m
Meters above sea level	masl
Meters per minute	m/min
Meters per second	m/s
Micrometer (micron)	?m
Microsiemen (electrical)	?s
Milliamperes	mA
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimeter	mm
Millimeters per hour	mm/h
Million cubic meters	Mm^3
Million litres	ML
Million tons	Mt
Million Years Ago	Ma
Million	M
Mineral Production Sharing Agreement	MPSA
Minute (plane angle)	'
Minute (time)	min
Mitsubishi Metal Mining Corp.	Mitsubishi
Month	mo
Movement Magnitude (of an earthquake)	Mw
National Instrument 43-101	NI 43-101
Nationwide Development Corporation	NADECOR
Net Present Value	NPV
Net Smelter Prices	NSP
Net Smelter Return	NSR
Neutralization Potential	NP
Newton	N
Newtons per meter	N/m
Ounce	oz

Oxidation-Reduction Potential	ORP
Parts per billion	ppb
Parts per million	ppm
Pascal (newtons per square meter)	Pa
Pascals per second	Pa/s
Peak Ground Acceleration (earthquake)	PGA
Percent	%
Pound(s)	lb
Preliminary Feasibility Study	PFS
Preliminary Economic Assessment	PEA
Probable Maximum Flood	PMF
Probable Maximum Precipitation	PMP
Qualified Persons	QPs
Quartz-Sericite-Chlorite	QSC
Ratel Gold Limited	Ratel
Resource Evaluation Inc.	REI
Reverse Circulation	RC
Rock Quality Designation	RQD
Second (plane angle)	"
Second (time)	s
Sierra Madre Occidental	SMO
Silver	Ag
Solvent Extraction -Electrowinning	SX-EW
Specific gravity	SG
Square centimeter	cm2
Square kilometer	km2
Square meter	m2
St. Augustine Gold & Copper, Ltd.	SAGCL
Tailing Storage Facility	TSF
Thousand tons	kt
Ton (metric, 1,000 kg = 2205 lb)	t
Tons per cubic meter	t/m^3
Tons per day	tpd
Tons per hour	t/h
Tons per year	t/a
Toronto Stock Exchange	TSX
Total dissolved solids	TDS
Total suspended solids	TSS
Troy ounce (31.1035 g)	oz
Unspecified scale magnitude for earthquakes	M
Valueless Rock Management Area	VRMA
Volt	V
Week	wk
Weight percent	wt%
Weight/weight	w/w
Wet metric ton	wmt
Yard	yd
Year (annum)	a
Year (U.S.)	yr

3 RELIANCE ON OTHER EXPERTS

In cases where the M3 Preliminary Feasibility Study author has relied on contributions of the Qualified Persons, 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.

M3 relied upon the following consultants for various parts of the project who are not listed as QP's in the project.

- AMEC, Denver, CO, and Salt Lake City, UT, USA- Provided design information and costing data for drystack tailing facility, valueless rock management areas, water diversion structures, and pit slope stability, water balance, and pit depressurization.
- Golder Associates, Inc., Washington, USA– Provided design and costing for water treatment, and wellfield design.
- AV Garcia Power Systems Corporation, Quezon City, Philippines – Provided the design and capital/operating costing for the coal and HFO fired power plants.
- Halcrow, A CH2M Hill Company, Manila, Philippines- Provided the design and capital/ operating costing for the port facilities.
- Simon Hunt Strategic Services, Surrey, United Kingdom – Provided a study on Asian smelting and refining concentrate terms.
- Fertecon Research Centre, Twickenham, United Kingdom – Provided a study on Asian sulfur and sulfuric acid markets.
- Landgon & Seah Philippines Inc., Manila, Philippines – Provided the Asian material unit rates used in initial capital and operating cost estimates
- SAGC – Provided owners cost estimate and camp cost estimate as well as history description, property description, and adjacent property summary.
- SyCip Salazar Hernandez & Gatmaitan, SyCipLaw Center, Makati City, Philippines – Provided legal review on matters pertaining to land control, land acquisition, reclassification and conversion, and standing of the MPSA.

Reports received from other experts have been reviewed for factual errors by SAGC and M3. However, M3 does not attest to or assume responsibility for the accuracy of any information or data from the reports. 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.

Other persons or companies relied on during the preparation of this report include those listed in Section 2.2 and the reports referenced in Section 27, References.

All maps, as well as many of the tables and figures for this report were supplied by SAGC or their consultants.

M3 relied upon SAGC for project ownership data and adjacent property data. M3 did not verify ownership or underlying agreements.

3.1 USE OF THIS REPORT

This Technical Report was prepared for SAGC ("Client") by M3 pursuant to the contract agreement ("Agreement") between the Client and M3.

The report is based in whole or in part on information and data provided to M3 by Client and/or third parties. The results and opinions outlined are dependent on the aforementioned information being current, accurate, and complete as of the date of this report, and it has been assumed that no information has been withheld which would have an impact on the conclusions or recommendations made herein.

M3 represents that it exercised reasonable care in the preparation of this report and that the report complies with published industry standards for such reports.

The recommendations and opinions contained in this report assume that unknown, unforeseeable or unavoidable events, which may adversely affect the cost, progress, scheduling or ultimate success of the Project, will not occur. Except as may be expressly stated in writing in the Agreement, the use of this report or the information contained herein is at the user's sole risk. M3 does not assume any liability other than performing this technical study to normal professional standards.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 PROJECT LOCATION

The King-king Project is centered at approximate geographical coordinates 7°11'31"N Latitude and 125°58'40"E Longitude on the Philippine Island of Mindanao. Figure 4-1 shows the location. The project site is located at Sitio Gumayan, Barangay King-king, Municipality of Pantukan, Province of Compostela Valley, in Mindanao.

Figure 4-1: Project Location

4.2 LOCATION OF THE MINERAL PRODUCTION SHARING AGREEMENT (MPSA)

The central project property is the tenement area defined by the Amended Mineral Production Sharing Agreement (MPSA) No. 009-92-XI, which covers a total area of approximately one thousand six hundred fifty six (1,656) hectares situated in Sitio Lumanggang, Pantukan. The MPSA was approved in May 27, 1992, was amended in December 11, 2002 and is set to expire on May 27, 2017.

The tenement straddles three (3) barangays: King-king, Magnaga and Tagdangua, with approximately half of its total area being situated within the King-king Barangay. The tenement boundaries are defined by 14 points, which are summarized in Table 4-1 below.

Table 4-1: Geographic References for MPSA No. 009-92 XI Boundaries

Point No.	Latitude			Longitude		
1	7°	10’	33.6”	125°	59’	12.4”
2	7°	10’	33.6”	125°	58’	13.7”
3	7°	10’	43.4”	125°	58’	13.7”
4	7°	10’	43.4”	125°	57’	34.5”
5	7°	12’	1.5”	125°	57’	34.5”
6	7°	12’	1.5”	125°	58’	13.6”
7	7°	12’	40.6”	125°	58’	13.6”
8	7°	12’	40.6”	125°	58’	52.7”
9	7°	13’	49.0”	125°	58’	52.7”
10	7°	13’	49.0”	126°	00’	1.1”
11	7°	12’	50.3”	126°	00’	1.1”
12	7°	12’	50.3”	125°	59’	41.5”
13	7°	11’	51.7”	125°	59’	41.5”
14	7°	11’	51.7”	125°	59’	12.4”

The King-king tenement is shown in Figure 4-2.

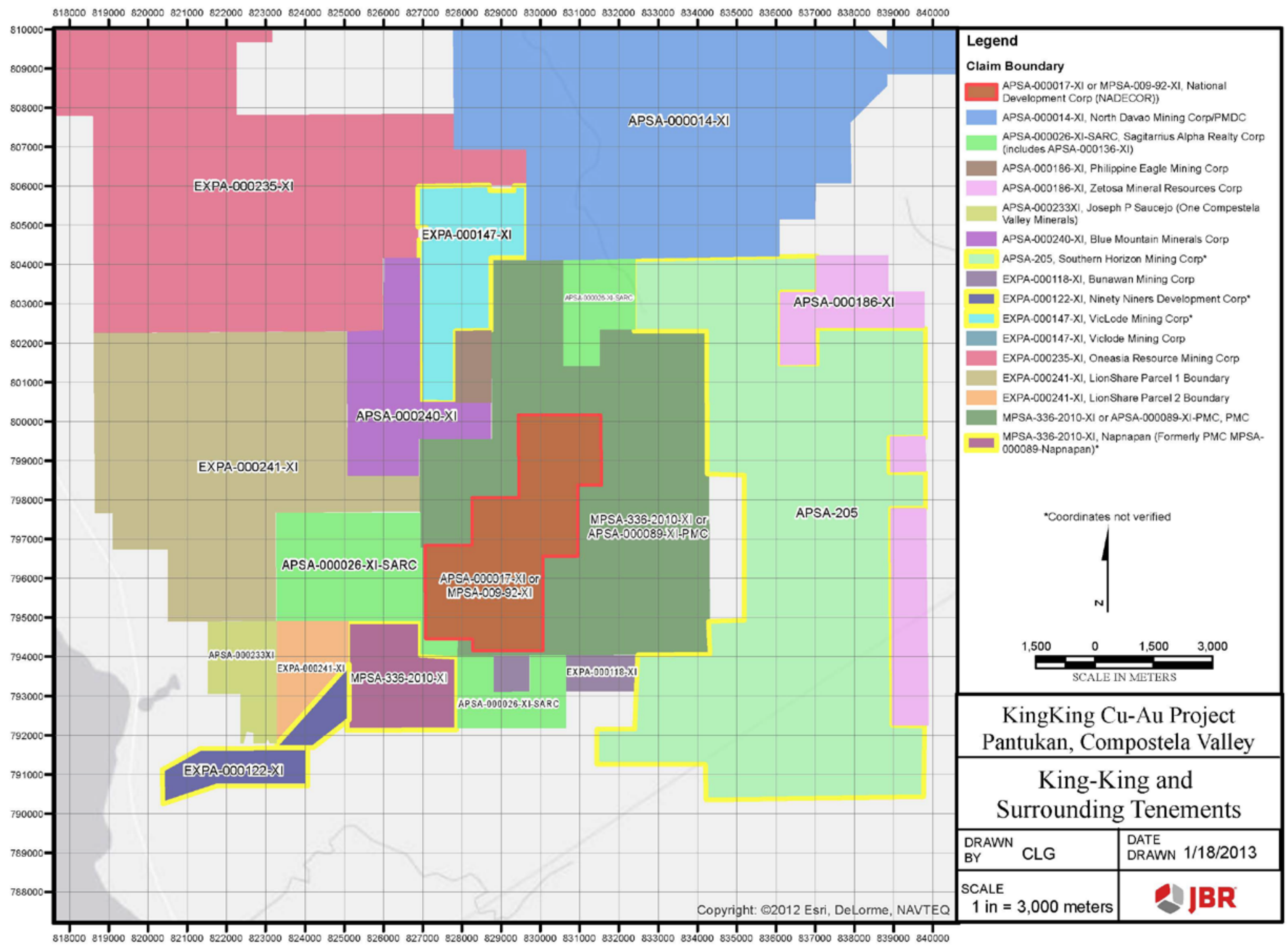

Figure 4-2: Mining Tenement Boundary

Nationwide Development Corporation (NADECOR) is the Leaseholder and Contractor under this MPSA. As the tenement's leaseholder, NADECOR is required to undertake and execute, for and on behalf of the Philippine government, sustainable mining operations in accordance with the provisions of the MPSA and is constituted and appointed the exclusive entity to conduct mining operations in the contract area.

4.3 SAGC EARN-IN BASIS

St. Augustine Gold & Copper Ltd. (SAGC) and its affiliates have entered into agreements, listed below, with NADECOR for development of the King-king Project:

- In April of 2010, a Memorandum of Understanding between NADECOR and St. Augustine Mining Ltd. (SAML), a subsidiary of SAGC, was signed and it set out the basis under which SAML would acquire in phases an interest in aggregate (direct and indirect through a Philippine-law compliant structure) of up to 60% of the project in exchange for certain payments, investments and other deliverables.[1]
- In June 2011, NADECOR and SAGC signed a Technical Services Agreement, as well as Onshore and Offshore Services Agreements. These agreements allowed wholly owned technical service companies of SAGC (MDCA and San Augustin Services Inc. (SASI)) to provide technical services to the project, including services related to the feasibility and permitting studies.
- In August 2011, an Interim Funding Agreement was signed between NADECOR and SAGC, setting out the detailed terms under which SAGC continues to invest in the project and how this investment will be further protected. At the same time, NADECOR formed several companies that are currently intended to be the joint venture companies for the King-king project. This joint venture structure takes into account Philippine legal requirements (including applicable nationality restrictions) and provides for a legally compliant mechanism under which SAGC can participate in the management and ownership of the project. In addition, both parties are currently planning and implementing the process of assigning the MPSA into one of the joint venture companies mentioned above.
- The law firm of Sycip, Salazar, Hernandez & Gatmaitan (Makati City) has reviewed the status of the MPSA, and the Memorandum of Understanding, Technical Services Agreement, Onshore and Offshore Services Agreement, the Interim Funding Agreement and a Preferred Shares Investment Agreement between the parties. The law firm recently issued an opinion verifying that when the MPSA is transferred into one of the joint venture companies, SAGC will have an interest in the MPSA. The opinion also verified the agreements stated above.

[1] On October 15, 2013, St. Augustine and NADECOR announced that they intend to pursue a restructuring plan that will change the ownership described above. This plan is subject to various conditions and approval requirements that are pending.

4.4 CERTIFICATE OF ANCESTRAL DOMAIN TITLE (CADT)

Being within forest land, the tenement area is also covered by a Certificate of Ancestral Domain Title (CADT) issued by the National Commission on Indigenous People (NCIP) to the Mansaka tribe in the Municipality of Pantukan as provided for by Republic Act 8371 or the Indigenous Peoples Rights Act (IPRA). CADT No. R11-PAN-0908-076 was signed in September 2, 2008 and covers a total area of approximately 141,773.3097 hectares. With the exception of alienable and disposable (A&D), which are covered either by an Original Certificate of Title (OCT) or Transfer Certificate of Title (TCT), forestlands covered by a Community-Based Forest Management Agreement (CBFMA) executed by the DENR, and unregistered parcels that are covered by the relevant tax declarations, all Project facility areas are covered by the CADT.

One of the requirements after the issuance of a CADT is the formulation and implementation of an Ancestral Domain Sustainable Development and Protection Plan (ADSDPP) by the indigenous people awarded the CADT.

The ADSDPP for CADT No. R11-PAN-0908-076 allows large-scale mining within the ancestral domain and proffers explicit guidelines for negotiating with the Pantukan Federation of Mansaka Tribal Council (PFMTC), permitting systems, exploration, extraction and utilization of minerals and environmental protection, preservation, and sustainability

4.5 LAND TITLES

Project facility areas cover approximately 300 parcels of land, most of which are covered by OCTs or TCTs. An OCT or TCT evidences the holder's ownership over the land area covered by said document. Project facility areas that are not covered by OCTs or TCTs are covered by the CADT of the Mansaka tribe, the CBFMA executed by the DENR or tax declarations issued in the names of the current occupants of such areas.

The transfer of ownership over the Project facility areas are covered by and subject to Philippine laws and regulations. Property owners are required to pay an annual real estate tax, currently pegged at approximately 1.00% of the total assessed value of the land. The sale of land is also covered by a 6% capital gains tax that the selling landowner needs to remit to the Philippine government through the Bureau of Internal Revenue (BIR) within thirty (30) days following the date of the purchase.

Quantifying any encumbrances in establishing property ownership of these land parcels, and identifying any unpaid tax accounts, will be part of the due diligence work that will be completed prior to the acquisition of the parcels required to construct and operate the Project.

4.6 LAND ACQUISITION

An effective land acquisition strategy for the project has been developed that utilizes option agreements for establishing the price and terms for purchase of the required property. Parcels needed for the project have been identified and the ownership will be investigated as part of the due diligence to be completed prior to entering into an option agreement to purchase the parcels.

Approximately 300 parcels of land outside of the tenement must be purchased for the project. A detailed land acquisition strategy has been developed for the Project that will be implemented to obtain the required parcels to meet schedule and cost objectives.

Initial acquisition estimates included in the project economics depend on Provincial Order 08-2011, which was enacted by the Sangguniang Panlalawigan of Compostela Valley last September 7, 2011 and defines the schedule of base market values for land and base unit construction cost for buildings and other structures in the entire province.

4.7 ENVIRONMENTAL LIABILITIES

Environmental liabilities in relation to the project will be defined by the Environmental Compliance Certificate (ECC) to be issued by the Environmental Management Bureau (EMB) of the DENR.

The guidelines for the issuance of an ECC are provided for by Presidential Decree No. 1586, or a decree establishing the Environmental Impact Statement (EIS) System including other Environmental Management and Related Measures, and further defined by several memoranda, department administrative orders, memorandum circulars and other official documents.

The most significant document to support the issuance of an ECC would be an Environmental Impact Assessment (EIA) study that defines all short and long-term environmental impacts of the project in the natural and built environments. The project team submitted the initial draft of the EIS to the EMB of the DENR in February 2012. The EIS establishes the baseline conditions for the project.

Aside from the mitigating measures enumerated in the ECC, the project has also continued to implement environmental protection and conservation activities over and above the requirements of Philippine government. These activities include active participation in the National Greening Program, performed in partnership with upland communities awarded Community-Based Forest Management (CBFM) agreements by the DENR, mangrove protection initiatives, community support for the implementation of Republic Act 9003 or the Ecological Solid Waste Act, among others.

Additional environmental and permitting details are presented in Section 20.

4.8 TERMS OF THE MPSA

The MPSA and the approved Work Plans (exploration and environmental) allow work to be carried out that is necessary to obtain an approved DMPF (Declaration of Mine Project Feasibility) and an ECC (Environmental Compliance Certificate), which allow the future development of the mine. This would include work proposed for the property, i.e. to drill, sample, transport, survey, conduct baseline studies, etc.

The MPSA was approved in May 27, 1992, (Effective Date), and was amended in December 11, 2002. The MPSA has a term of twenty five (25) years from Effective Date, and may be renewed for another term not exceeding twenty five (25) years.

The MPSA was under the 6th renewal of the Exploration Period (EP). Under conditions of this EP's renewal, the Declaration of Mining Project Feasibility Study (DMPF), and the relocation plan for the affected people within the claim project area had to be submitted within the period specified by the Mines and Geosciences Bureau (MGB). These requirements were fulfilled with the submission of the DMPF and relocation plan on May 4, 2012.

The MPSA for King-king has the following provisions:

- The contractor has the exclusive right to conduct exploration, development and operation in the contract area.
- The contractor is required to carry out activities according to an approved work program and commit expenditure for the environment, the community and the development of geo-sciences. NADECOR/SAGC has complied with the terms of the approved work plan and the MPSA including the submittal of the EIS and DMPF to MGB/DENR.
- The financial requirement includes the payment of occupation fees in the amount prescribed by the DENR. These fee payments are current with the MGB. A payment in the amount of PhP 124,200 (approximately US$ 2,650) was made August 2012, with the next payment scheduled for May 26, 2013 for the period from May 27, 2013 through May 26, 1014.

After the completion of this study, the MGB issued a Certificate of Compliance stating that the MPSA holder (NADECOR) has substantially complied with the terms and conditions of the MPSA including the payment of occupation fees and submission of required reports from May 6, 2010 through the effective date of this report. The law firm of Sycip, Salazar, Hernandez & Gatmaitan (Makati City) has recently reviewed the compliance certificate and issued an opinion concurring that the MPSA is in good standing.

4.9 PROJECT-AFFECTED PEOPLE

Tenured and non-tenured (i.e. informal) people and households will be affected by the project. Initial estimates of the potential project-affected people (PAP) within the facilities footprint are 7,861 individuals and 1,642 households. A buffer zone of one kilometer from facility boundaries yielded an additional 8,579 individuals and 1,747 households. This brings the total estimated number of PAP to 16,440 individuals and 3,389 households. The mandated buffer zone for mining facilities is 50 to 150 meters. The 1-km buffer zone was used to come up with conservative estimates for planning purposes.

These estimates were based on the annual average population growth rate in the Municipality of Pantukan, and on the projection of a population and household census conducted by Barangay Health Workers in 2011 to 2012 (the data gathered from this census was generated for the project).

Table 4-2 summarizes the estimated number of PAP and PAP households for each of the project components, and presents figures for both the facility footprints and the one kilometer buffer zone.

Table 4-2: Summary of Project Affected People (PAP) and Household (HH)

Facility	Footprint		Buffer		TOTAL	
	HH	PAP	HH	PAP	HH	PAP
Coastal Complex	200	907	819	4,069	1,019	4,976
Heap Leach *	165	897	121	629	286	1,526
Southwest Drystack Facility	462	2,178	187	826	649	3,004
West VRMA	94	489	189	904	283	1,393
MegaPit Cone	562	2,570	43	173	605	2,743
Low Grade Ore Stockpile	63	315	0	0	63	315
SW VRMA *	96	505	0	0	96	505
Road/Transmission Line	0	0	388	1,978	388	1,978
TOTAL	**1,642**	**7,861**	**1,747**	**8,579**	**3,389**	**16,440**

*The buffer for the Heap Leach and SW VRMA includes the people living in the area of the process facility.

The PAP will be compensated for any displacement and interruption of their livelihood activities in a way that is consistent with international standards. In particular, the project intends to compensate PAP following, at the very least, International Finance Corporation (IFC) standards and the Equator Principles.

4.10 LAND RECLASSIFICATION

Land acquisition is the first step to ensure project implementation within the desired facilities areas. However, ownership does not imply explicit permission to proceed with project construction activities.

This is especially true for lands that are currently classified as agricultural, as Republic Act 7160 or the Local Government Code (LGC) and Republic Act 8435 or the Agriculture and Fisheries Modernization Act (AFMA) provide stringent processes that need to be followed for the reclassification of agricultural land to other types of land uses.

All project facility areas will need to be reclassified as heavy industrial, as the Municipality of Pantukan has not allocated any land for this particular land use.

Reclassification is done through a legislative act by the Sangguniang Bayan of the Municipality of Pantukan. Said legislation should be pursuant to Section 20 of the Local Government Code, Memorandum Circular No. 54, s. 1993 and other relevant directives, as well as all requirements that the local government unit (LGU) may petition from the project proponents.

4.11 LAND CONVERSION

Republic Act 6657 or the Comprehensive Agrarian Reform Law (CARL), as well as Department of Agrarian Reform (DAR) Administrative Order No. 363 and other related directives, provide the guidelines for land conversion in the Philippines. Conversion is defined as the act of putting a piece or parcel of land into a type of use other than that for which it is currently being utilized. Based on review of secondary data, no project facilities will be located in areas that are non-negotiable for conversion.

DAR is the primary agency mandated to oversee the conversion of lands for other uses, and in the case of the project, it is the DAR Secretary that will issue the conversion order for heavy industrial use in the facilities areas. Compliance with DAR requirements and the issuance of the final conversion order alone is projected to take approximately seven (7) months in the absence of constraints or externalities that may adversely affect approval of the order.

4.12 RISKS AND RISK MANAGEMENT

It is prudent to identify potential risks that come with the options being developed for land acquisition, as this is the phase that is most vulnerable to constraining externalities. Previous experiences by other companies have resulted in their paying large relatively large sums for land, or non-compliance of landowners to previous agreements – leading to surging project costs or worse, non-implementation.

Risks:

- Dishonored agreements - Dishonored agreements in the banana industry, despite the presence of legally binding agreements, began in 2005 and continue up to this day.
- Implications of extended purchasing period - Negotiations with landowners need to be completed quickly, as protracted negotiations increase the risk that a successful agreement will not be reached. These risks include:
    - Demand for higher prices as opposed to what is stipulated in the option agreement
    - Non-cooperation of heirs
    - Seller claims misrepresentation by the buyer
    - Opportunistic informal settlers - A long, drawn-out period to complete all acquisition transactions makes the project facility areas vulnerable to opportunistic informal settlers

Risk Management:

- Ensure that the options agreements are drafted by an expert legal team, with the documents duly notarized to make them a matter of public record.
- Licensed realtors will be retained for the negotiation process, to ensure that all terms and conditions are acceptable to both parties and enforceable under Philippine laws for the entirety of the period defined in the agreement.
- The project team has identified alternative locations for all project facilities with the sole exception of the heap leach. The presence of alternative locations lessens implementation vulnerabilities in the event that the project fails to secure the first choice areas it intends to use for its facilities.
- All legal heirs will be included in the drafting of the option agreements with landowners.
- Provide a full disclosure to landowners where prudent.

- The best land acquisition strategy will not only help cushion the company from the unreasonable demands of opportunists, it will also facilitate and streamline the reclassification and conversion process for the entire project, minimize redundancy of resource use.

4.13 RECOMMENDATIONS

1. Given the numerous modes of tenure (i.e. CADT, OCT/TCT, MPSA, CBFM, and others) that define the project area, it is important to pursue all land acquisition and development efforts in such a way that harmonizes and complies with the various requirements of these different instruments.
2. There is also a need to conduct detailed work programming and planning in all land-related project deliverables. Such programming and planning should:
   a. include a thorough risk assessment of potential show stoppers that are in addition to externalities related to land acquisition;
   b. realistically align with the project construction schedule;
   c. give sufficient time for reclassification and conversion after acquisition and before construction;
3. Ground verification needs to be conducted prior to land acquisition to reflect actual titled properties, identify actual homeowners, and whittle down large parcels that may have been subdivided since the last Lands Management Bureau cadastral mapping activity.
4. This early, it is also important to identify the mechanism for consolidating all properties secured by the company. This is in light of Philippine laws that limit ownership of agricultural land.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

5.1.1 Road Access

The project area is approximately 35 aerial kilometers (km) east-northeast of Davao City, and some 1,000 aerial km southeast of Manila on the island of Mindanao. Locally, it is approximately 10 aerial km northeast of the Municipality of Pantukan, Province of Compostela Valley. Pantukan is approximately 92 km by road from Davao City via the paved Tagum–Mati National Road. From Pantukan town proper, the project can be accessed through the 18 km Buko-buko sa Anay-Lawaan dirt road, which can be negotiated using conventional four-wheel drive vehicles. The proposed access route will be further west and follow above the Kingking River in the upper section. The process plant will be on this route immediately above the coastal plain. The new access road will require an overpass over the existing main highway to connect the port facility with the plant and mine (see section 18.1).

5.1.2 Air Access

The nearest commercial airport to the project is located at Davao City where daily flights to and from Manila and Cebu are available. From the airport, it is a 1½ to 2 hour trip to Pantukan via the Tagum-Mati National Road. Once the access roads are completed, the mine site will be approximately another ½ hour away with the process plant about halfway.

5.1.3 Sea Access

There is currently no port, or wharf facility adequate for the project needs at or near Pantukan. Most commercial facilities are privately owned and by law are only for the owners' use.

5.2 CLIMATE

The climate at the project site is humid tropical and is considered a Coronas Classification Type 2 (Dames & Moore, 1997). The project is south of the normal typhoon path, so it does not have pronounced wet and dry seasons as experienced elsewhere in the Philippines.

Monthly rainfall data from Davao City (approximately 44 km west of the mine site) were reviewed for the periods from 1961 to 1994 and 2002 to 2010, and data from Tagum City (approximately 29 km north of the mine site) were reviewed for the periods from 1979 to 1997 and 2002 to 2010 (AMEC, 2011a). In evaluation of the earlier climate data, Dames & Moore (1997) applied a 38% increase factor to account for the higher elevation of the project area, and to better reflect the conditions found on the east side of the Davao Gulf. Using the same methodology and incorporating the more recent climatic data, AMEC (2011a) estimates that the average annual rainfall at the project site is 2,766 millimeters (mm), but this can vary by 1,000 mm above or below this amount depending on whether a wet or dry year. Rainfall is spread evenly throughout the year with no distinct wet or dry season. Two new weather stations have been installed on the site. These have been recording data at an upper station (near the pit) and a lower station (near the coast) since April 2011 (AATA, 2011a). The climatic design criteria for the project will be updated as additional site data becomes available from these stations.

Daytime temperatures range from 18^{o} C to 35^{o} C with an average ambient temperature of 27^{o} C.

Typhoons are very rare but torrential rains and subsequent flash floods are not uncommon.

There are no climatic conditions that should cause great operational difficulty for the project. The greatest climatic issue will be managing storm waters that will result from excessive rainfall at intermittent times during the life of the project. However, this is a common operating issue at many tropical mine sites and will be manageable with proper storm water management controls and planning.

5.3 LOCAL RESOURCES

The local unemployment is approximately 12.6%. In 2009, the local Pantukan Municipal government sent a letter to the DENR requesting the King-king Project be developed as swiftly as possible. The local community is favorable to the project.

Primary employment in the region is on plantations growing bananas or coconuts. Secondary jobs exist for a limited number of workers in the several small scale mines in the mountains northeast of Pantukan.

According to the National Statistics Office of the Philippines, the 2007 populations of communities near the King-king Project were as follows:

Pantukan Municipality	69,656
Magnaga	7,743
Napnapan	9,983
King-king	21,444
Davao City	1,366,153

5.4 INFRASTRUCTURE

Some of the basic infrastructure is in-place for exploration and development of the King-king deposit. A paved highway from Davao City runs 10 kilometers southwest of the project. The project mine area in the 250 to 950-meter elevation range can be reached via the previously mentioned 18 km Buko-buko sa Anay-Lawaan dirt road, which is now passable by large four-wheel drive vehicles such as drilling rigs and supply, fuel and water trucks. Planned low-land facilities, including the tailing area, mill site, port facility, and power plant location can be accessed via local area roads.

Water for exploration has been taken from low pressure artesian wells, including two wells developed from exploration diamond drillholes located on the southern side of the deposit or from nearby small surface drainages that run through the southern and northern ends of the project area. Potential sources for water for mining and processing include wells planned to be situated in the alluvium deposits located south of the Kingking River.

Power availability is currently too limited in Mindanao to assume that grid-supplied power will be available for operation of King-king. Construction of a 189 MW power plant (160 MW from coal and 29 MW from HFO) is envisioned for the project.

Anticipated concentrate shipment volumes and the requirements for importing coal and other essentials necessitate the construction of a dedicated port facility. The only port facility in the Pantukan area consists of a concrete barge landing ramp, which should be available to handle barges from the existing deep water port facilities at Davao and Tagum for transport of inbound materials for construction and early mine operation.

Currently there is a temporary drill core storage building in Pantukan (approximately 1,000 square meters). Most of the drill core is located at a warehouse in Davao. Several buildings from Echo Bay's tenure in 1997 remain at the project site and have been recently rehabilitated in 2011.

5.5 PHYSIOGRAPHY

The coastal plain extends a length of six (6) kilometers from Davao Gulf to the base of the mountains where the King-king project is located. The majority of the population lives along the coastal plain with significantly lower population densities in the mountains. Figure 5-1 below shows the topography of the local area.

The topography in the immediate project area is steep and rugged with elevations ranging from 260-950 meters above mean sea level (amsl) and averaging 650 meters amsl. The porphyry copper-gold mineralization outcrops between 400 m and 700 m elevations. The terrain gradually transitions through moderately rugged to rolling, moving westward toward the coastline. The dominant drainage pattern in the area is dendritic. The property itself is drained by the Casagumayan and Lumanggang creeks, tributaries of the Kingking River which enters the Davao Gulf at Pantukan.

The project area is covered generally by sparse tropical rainforest mostly left over from past commercial mahogany logging operations. Old growth mahogany trees are mostly gone, and large areas of the previously timbered slopes have been cleared, cultivated and planted with corn and other crops by local mountain tribes and lowland settlers. In the foothills toward Davao Gulf, what used to be forest-covered slopes are now dominated by cogon grass. Vegetables and fruit-bearing trees are grown in some places but these are limited and concentrated in localized flat or rolling terrain.

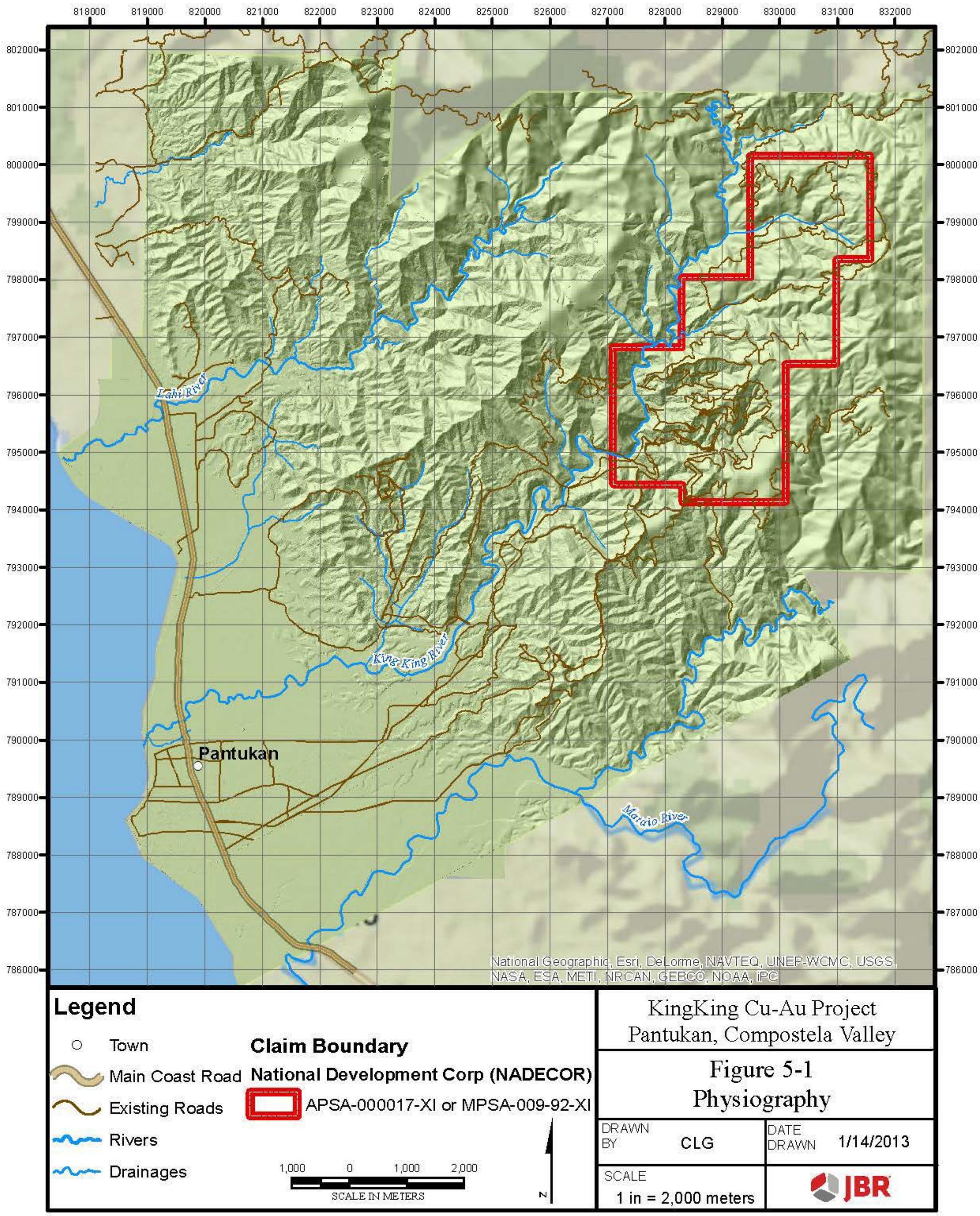

Figure 5-1: Physiography

5.6 MINING SURFACE RIGHTS AND MINING PERSONNEL

Under the Philippine Mining Act of 1995, NADECOR, as a holder of mining rights by virtue of the MPSA No. 009-92-XI, may not be prevented from entry into private lands and concession areas by surface owners, occupants, or concessionaires when conducting mining operations within the area covered by MPSA No. 009-92-XI. However, any damage done to the property of the surface owner, occupant, or concessionaire as a consequence of such operations, must be properly compensated by NADECOR in accordance with applicable regulations, and to guarantee such compensation, NADECOR must, prior thereto, post a bond (in an amount computed based on the type of properties and the prevailing prices in and around the area where the mining operations are to be conducted) with the Regional Director of the MGB.

Operations and maintenance staffing would be sourced from Pantukan and neighboring municipalities, the province of Compostela Valley, from the island of Mindanao, from the Philippines and from outside of the Philippines. The Municipality of Pantukan is home to approximately 69,000 people. There is a large craft trained work force to draw from in the Davao area. The population of Davao is 1.4 million people. Thus, there is a sizable work force to draw from near the mine site.

6 HISTORY

NADECOR discovered the King-king mineralization anomaly in 1966 through 1968. From 1969 to 1972, Mitsubishi Mining Corporation undertook initial exploration of the deposit, completing 54 surface diamond drillholes for a total of 13,031 meters of drilling. These initial holes were all drilled within the present resource outline. The Mitsubishi drill cores were only assayed for total copper and acid soluble copper. None of the cores from this program are known to exist.

Benguet signed an Operating Agreement with NADECOR on August 21, 1981 for the exploration and development of the King-king property. However, litigation regarding ownership did not allow any activity within the project. In 1991, all legal issues were resolved in favor of NADECOR's rights over the mineral claims.

From 1991 until 1994, Benguet completed 69 diamond core holes (19,247m), 25 reverse circulation holes (4,926m), 326 m of confirmatory adits and underground raises, 2,500 hectares of geological mapping, and the collection of 2,172 surface rock samples. The Benguet drilling was concentrated in the Lumanggang and Casagumayan areas in the central and west areas of the currently known deposit. Benguet produced an in-house "pre-definitive" feasibility study in March 1994.

From 1995-1997 King-king Mines, Inc. (KMI), an Echo Bay Mines, Inc. company, entered into an option agreement with Benguet and NADECOR to develop the King-king Project. KMI drilling amounted to 128 core holes and 52,718 m of drilling. Kilborn International, Inc. (Kilborn) was retained by KMI to complete a plus or minus 20 percent capital and operating cost estimate for the King-king Project, the scope of which was based on several specific items and on Kilborn's interpretation of Echo Bay Mines' generic requirements for what was termed by Echo Bay to be a Level l Study. The scope included those activities necessary for evaluation of equipment, processes, environmental and regulatory considerations, and economic factors sufficient to confirm a technically viable and cost effective project.

Several other consulting groups provided services for the project. DCCD Engineering of Manila, under subcontract to Kilborn, provided capital cost estimates for port facilities, local labor rates, and local costs for services and consumables. Knight Piésold Ltd. (Knight Piésold), which is under contract with KMI, provided costs for the various tailing dam, waste rock storage alternatives, and closure costs. Fluor Daniel, under contract with KMI, completed the mine planning and mine cost estimate portions of the report.

In mid-1997, KMI's "Level I" study estimated a total mineral resource of 1,040 million tons containing 0.306% Cu and 0.41grams Au per ton for the King-king deposit. This resource included a "mineable reserve" of 403 million tons @ 0.332% Cu and 0.488g/t Au. The authors of this report emphasize that neither the KMI "Level 1" mineral resource estimate nor the "mineable reserve" estimate is compliant with current NI 43-101 guidelines. These estimates are included in this Technical Report only because they are an important part of the project history. Upon completion of the KMI "Level 1" study, the property reverted to original ownership.

In 1998, Benguet completed a revised mineral resource estimate that was based on all available exploration drilling data and on a 0.20% total Cu cut-off grade. This estimate, which the authors of this report emphasize, is not compliant with current NI 43-101 guidelines, totaled 749 million tons containing 0.387% Cu and 0.433 g/t Au.

All Echo Bay data were acquired by Kinross Gold (Kinross) through its merger with Echo Bay in 2002. Kinross subsequently waived its option to proceed with the project. Kinross provided all the available data in its archives to RMMI.

NADECOR and RMMI signed a Letter of Intent (LOI) in August 2009 to work together to develop the King-king project, with RMMI undertaking extensive analysis to update the project information and mine plan.

In April of 2010, a Memorandum of Understanding between NADECOR and St. Augustine Mining Ltd. (SAML), a subsidiary of SAGC, was signed and it set out the basis under which SAML would acquire in phases an interest in aggregate (direct and indirect through a Philippine law compliant structure) of up to 60% of the project in exchange for certain payments, investments and other deliverables.

Per DENR's requirement, NADECOR started exploration and environmental work programs. NADECOR submitted Work Programs to the DENR. The Work Programs were approved in May 2010. A settlement was reached with Benguet in October 2010 where they agreed to their removal as Operator under the MPSA through a series of payments, one initially in 2010 with other payments over the course of commercial development of the property.

Ratel Gold and RMMI issued a NI 43-101-compliant technical report on the King-king resource in October 2010. The resource contained measured and indicated mineralization of 792 million tons with 0.279% total copper, 0.072% weak acid soluble copper and 0.371g/t gold. It also contained an inferred resource of 125 million tons with 0.237% total copper, 0.061% weak acid soluble copper and 0.308g/t gold.

In January 2011, RMMI assigned its interests in the Project to Ratel Gold Limited and took over management of Ratel and changed its name to St. Augustine Gold and Copper Limited (SAGC), a publicly traded company on the TSX, as a part of the reverse takeover. SAGC was used to raise capital for the feasibility and permitting studies through a series of private placement stock sales with several institutional investors.

In June 2011, NADECOR and SAGC signed a Technical Services Agreement, as well as Onshore and Offshore Services Agreements. These agreements allowed wholly owned technical service companies of SAGC (MDCA and SASI) to provide technical services to the project including in respect of the feasibility and permitting studies. Several studies were started in 2011 or in-progress with some completing in the same year:

- Environmental and social baseline studies – most were completed by end of year and compiled in several reports - Meteorology and Air Quality; Geology, Soils, Sediments &

Natural Hazards; Surface Water, Hydrology and Quality; and Groundwater Hydrology and Quality – to name a few.

- Feasibility studies – a few were completed by end of the year and reports prepared. Most studies were in-progress at year end. Metallurgical studies were almost all completed to feasibility level status by year end except for the heap leach study. Completion of the metallurgical studies allowed the facility design studies to proceed. Studies were in progress regarding mine design (including dumps), processing plants, tailing storage facility, power plant, port and project infrastructure.

In August 2011, an Interim Funding Agreement was signed between NADECOR and SAGC, setting out the detailed terms under which SAGC continues to invest in the project and how this investment will be further protected. At the same time, NADECOR formed several companies that are currently intended to be the joint venture companies for the King-king project. This joint venture structure takes into account Philippine legal requirements (including applicable nationality restrictions) and provides for a legally compliant mechanism for SAGC to participate in the management and ownership of the project. In addition, both parties are currently planning and implementing the process of assigning the MPSA into one of the joint venture companies mentioned above.

SAGC issued a press release in August 2011, updating the 2010 NI 43-101 compliant resource with new data from its feasibility studies and new metal prices. The resource contains measured and indicated mineralization of 962 million tons containing 0.254% total copper, 0.062% weak acid soluble copper and 0.334 g/t gold. It also contains an inferred resource of 189 million tons with 0.215% total copper, 0.048% weak acid soluble copper and 0.265g/t gold.

The earlier settlement agreement with Benguet was amended in August 2011 for accelerated performance and discharge for the benefit of all parties. The payment obligations were discharged in September 2011.

The MOU between NADECOR and the project-area indigenous people (Mansaka tribe) was signed in October, 2011. This agreement will lead to an important future agreement with the indigenous people that will set up issuance of the Certificate Precondition (CP). The CP is an important step for attaining approval of the Declaration of Mine Project Feasibility documentation (DMPF).

Some preliminary feasibility and permitting studies started in 2011 continued into 2012. Three important large multi-volume reports were completed or nearing completion during the year (2012).

- Environmental, social and facility design studies were completed to a level of detail allowing a preliminary EIS to be submitted for comments to the DENR (EMB) in February, 2012.

- Environmental, social and facility design studies were completed to a level of detail allowing a DMPF to be submitted to DENR (MGB) in May, 2012.

- Metallurgical and facility design studies were completed to a level of detail, allowing a preliminary feasibility study (PFS) report to be prepared. Some supporting studies and cost estimates were completed to a feasibility level.

All PFS studies were completed by year end and compilation and review of the report volumes were in progress for this NI 43-101 Preliminary Feasibility Study Technical Report.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 REGIONAL GEOLOGY

The southeastern Mindanao Peninsula (comprising the mountainous provinces of Davao Oriental, Compostela Valley and Davao del Norte) is bounded by two parallel subduction systems – the north-south trending East Mindanao trench, which is a segment of the Philippine Trench situated off the east coast of Mindanao, and the north-south trending Davao Trench situated between Samal Island and the east coast of Davao Gulf. Active tectonism is manifested in the frequent low to moderate-intensity earthquakes that occur in the area.

The King-king porphyry copper-gold deposit is located on the western flank of the eastern Mindanao Cordillera. It is the southernmost of a group of porphyry copper and gold deposits that are situated within a 75-km long, NNW-trending mineralized belt that runs across southeastern Mindanao, which is believed to be related to tension relief faulting induced by the Philippine Fault (Philippine Rift Zone). These deposits include the currently inactive Hijo and Amacan Mines of North Davao Mining Corporation, the old Masara Mines of Apex Mining Company, the Kalamatan Mine of Sabena Mining Company, and the well-known gold-rush areas of Diwalwal in Monkayo farther north (Burton, 1977; Culala, 1987). Numerous other mines and mineral prospects that are likely related to the Philippine Rift Zone lie outside of this belt. These include the Cabadbaran Gold Mine and the Placer Gold Mine of Manila Mining in Agusan del Norte, the Coo Gold Mine of Banahaw Mining in Agusan del Sur, the Siena Gold Mine of Suricon, and, the Asiga porphyry copper prospect, all in Surigao del Norte in northeastern Mindanao.

7.2 LOCAL GEOLOGY

The district in which the King-king deposit is located is bounded by two major splays of the tectonically-active Philippines Fault (see Figure 7-1). About 20 km to the east is the main Agusan Valley fault and its branches, which controlled the courses of the Manat, Agusan and Bitanagan rivers and was likely responsible for the formation of Maragusan Valley (See Figure 7-1). This valley encompasses a broad plain believed to be a sediment-filled graben that is perched high in the Diwata Range at elevations ranging from 650 m to 850 m amsl. Several kilometers to the west is a thrust fault that trends N-S (parallel to Davao Gulf), with King-king situated on the upper (over-riding) plate.

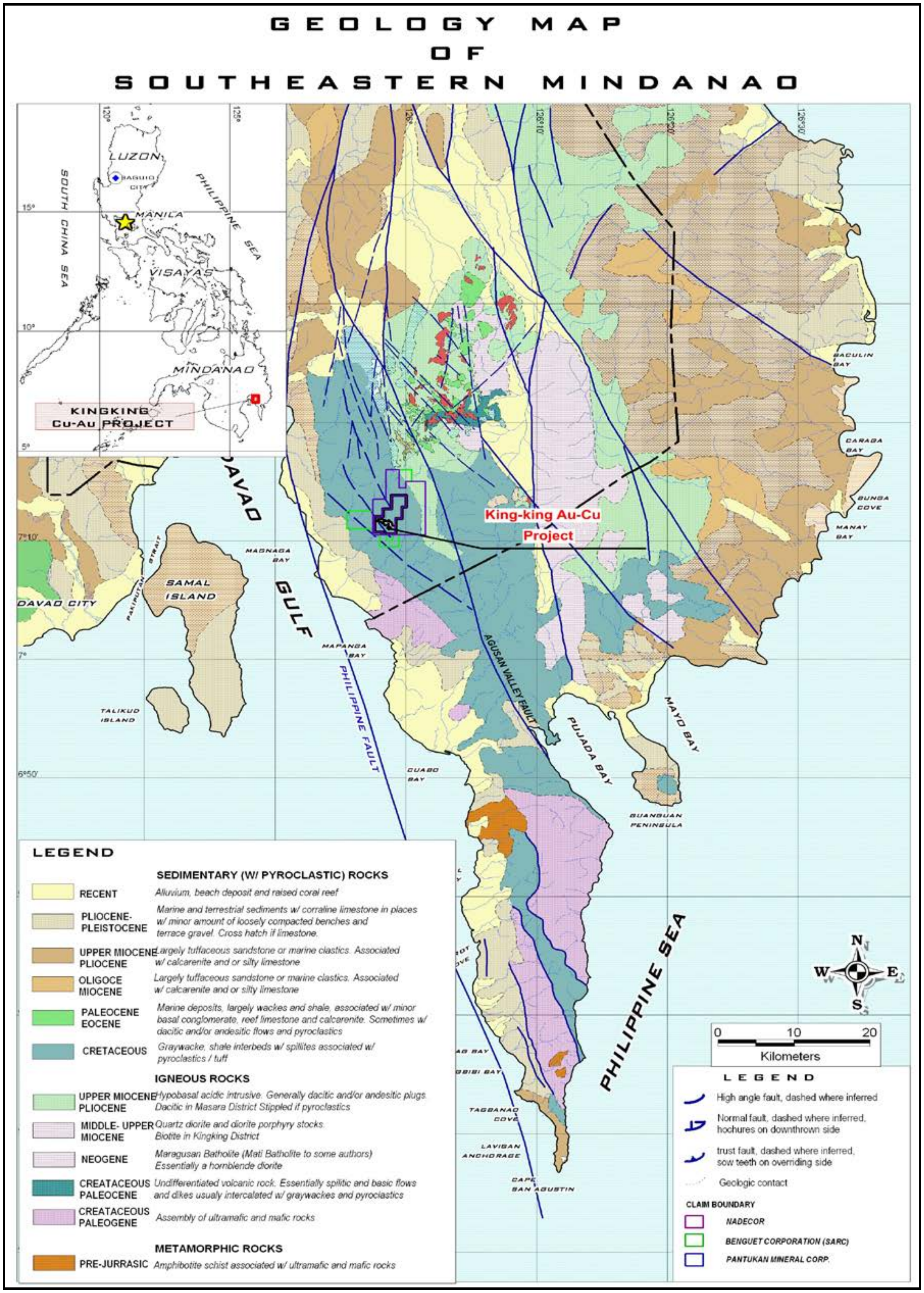

Figure 7-1: Local Geology

The King-king deposit is the largest of several prospects associated with a NE-trending belt of mineralized and post-mineralization intrusive rocks that measures approximately 6 km long by 3 km wide. These intrusives, which consist predominantly of diorites but also include less extensive dacites, were emplaced during the Middle- to Late-Miocene in a folded sequence of Cretaceous-Paleocene volcano-sedimentary rocks, apparently along pre-existing NW-trending anticlinal axes. The axial portions of the anticlines have since been largely eroded, exposing the cupolas of the underlying mineralized intrusives. The intruded volcanics are composed of pyroclastics (tuff, lithic tuff) and flows (andesites) with intercalated sediments (mostly wackes). The sediment/pyroclastic sequence has a general northwest trend with a southwest dip. However, local reversals of dip are common, forming minor anticlines and synclines that are evident along road cuts and gullies south of the main King-king deposit.

7.3 PROPERTY GEOLOGY

7.3.1 Lithology

The King-king deposit is hosted mainly by the diorite intrusive complex (to which it is genetically related) and to a lesser extent by the extrusive volcanics and sediments. The overall shape of the diorite complex is elongate, trending northwesterly and measuring on overage approximately 1,800 m in length and 400 m across. The diorite complex consists of the pre-mineralization biotite diorite porphyry (BDP), the intra-mineral diorite porphyry (IMDP), the intra-mineral hornblende diorite porphyry (IHDP), and two post-mineralization porphyries composed of hornblende diorite (HDP) and diorite (DP). Less-common dacite intrusives associated with the diorite complex include the intra-mineral dacite porphyry (IDAP), which consists of dikes cutting the BDP rocks, and the post-mineralization dacite porphyry (DAP). Local hydrothermal brecciation during the intrusion of the diorites into the older volcanics resulted in the development of intrusion/hydrothermal breccias along contacts.

The BDP (which appears to be the major intrusive underlying the King-king district) is generally brownish, medium- to coarse-grained and is characterized by the presence of primary “book” biotite that accounts for approximately 10% of the rock’s volume. BDP intrusive rocks are the most important intrusive hosts for copper-gold mineralization. Copper mineralization within the BDP consists predominantly of bornite with subordinate chalcopyrite occurring usually as fracture fillings. Bornite appears to increase towards the western half of the deposit from Casagumayan to the Tiogdan area. The copper and gold grades in the BDP average 0.37% Cu and 1.17 g/t Au, respectively. Copper-gold contents vary in the other intra-mineralization intrusives. In the IHDP, metal values average 0.37% Cu and 0.44 g/t Au, respectively. Where bornite is the predominant copper mineral in the dacite dikes (IDAP), copper grades are generally over 0.2% with occasional values exceeding 1% Cu near dike contacts with the intruded BDP. The IMDP is the least well-mineralized intrusive with respect to copper and gold, with grades in the ore zone averaging 0.37% Cu and 0.38 g/t Au respectively.

The main economic portion of King-king deposit (as defined by a 0.20% total copper cut-off) is elongated along a N70°W trend and measures approximately 1,800 m long and from 250 m to 550 m wide, as shown in Figure 7-2.

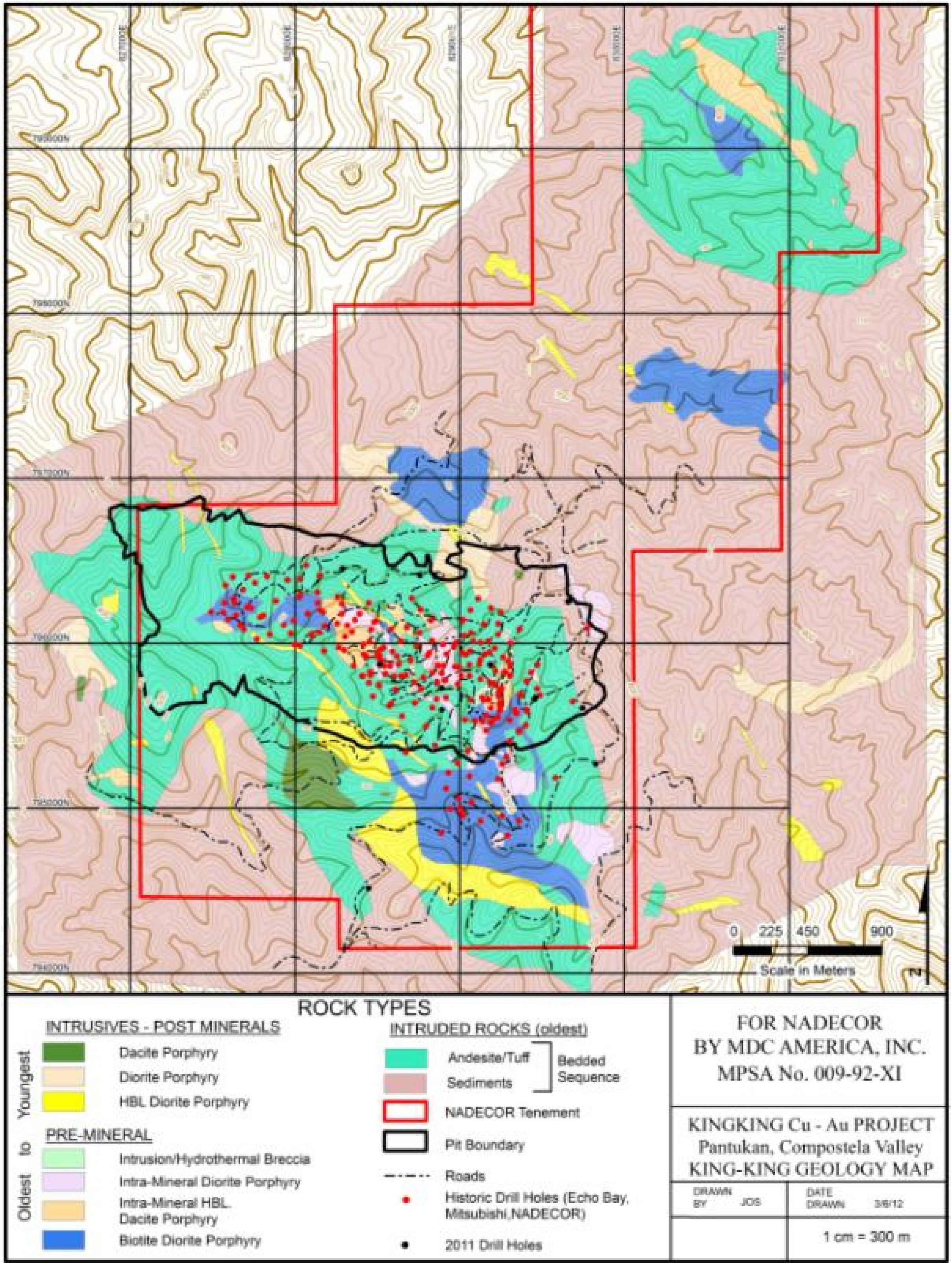

Figure 7-2: Property Geology Map

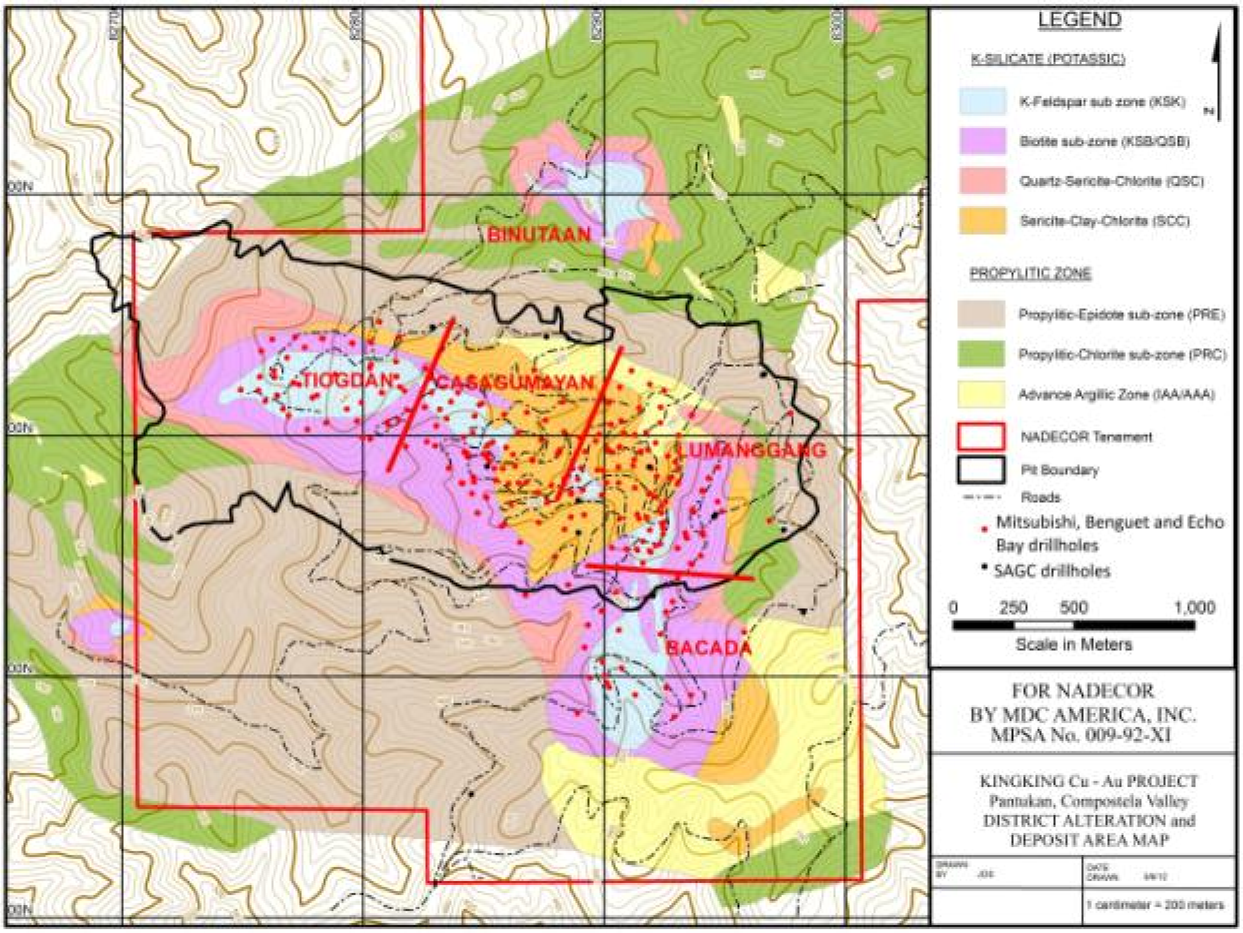

Figure 7-3: Commonly Referenced Deposit Areas

Figure 7-3 shows the relative locations of the commonly referenced areas of the deposit, which include Tiogdan, Casagumayan, Lumanggang, Bacada, and Binutaan, The deposit has an apparent steep NE dip in its central portions. On longitudinal section, it appears as an irregularly-shaped body with an undulating bottom, although for the majority of the deposit the bottom of the mineralization has yet to be fully defined. The deposit is subdivided into two more or less equal segments: 1) the eastern segment underlying Lumanggang, where copper mineralization is extremely erratic in general and where the better gold mineralization occurs in pockets usually associated with localized zones of strong silicification and quartz stockworks, and 2) the western segment within the Casagumayan and Tiogdan areas which generally carries higher copper and gold values and is more uniformly mineralized. The intrusion of dioritic rocks continued after the porphyry copper deposit was emplaced, as evidenced by the presence of post-mineral hornblende diorite porphyry (HDP), diorite porphyry (DP) and dacite porphyry (DAP). These occur as peripheral stocks bounding the Lumanggang and Bacada areas, and as northwest-trending lenticular bodies or dikes flanking the porphyry mineralization. One hornblende diorite porphyry dike measures 5 m to 15 m wide and is traceable for more than 1,000 m along and within the southern flank of the deposit. Elongate hornblende diorite stocks bounding the southern and western portions of Bacada also trend northwest.

7.3.2 Structure

7.3.2.1 Folding

Evidence for folding is found outside of the main King-king deposit in the Lahi, Barricade, Buko-buko sa Anay and Maplag areas, where fold axes are found to generally trend northwest with localized deviations to the east and west. Although the folds observed are generally small in scale, these are believed to be reflective of much larger-scale northwest-trending regional folding, as evidenced by the recumbent folds found along portions of the Maplag - Buko-buko road which appear to have developed as a result of regional stresses.

7.3.2.2 Faulting

A recent structural study commissioned by SAGC that focused on structures logged in diamond drill core and available surface mapping has identified five major structural sets, three of which trend northwest (Structural Geology Model, King-king Project, Compostela Valley, Philippines, Fischer & Strickler Rock Engineering, LLC, March 6, 2012). These structural sets are summarized as follows:

- **Set 1: N17W, 25˚E –** Low-angle structures aligned with the trend of intrusive breccias;
- **Set 2: N88W, 78˚N –** High-angle structures aligned with a post-mineral diorite porphyry intrusive (DP) located on the north side of the planned open pit;
- **Set 3: N31W, 76˚NE –** High-angle structures also aligned with the trend of intrusive breccias (See Set 1);
- **Set 4: N58E, 73˚NW –** High-angle structures similar to Set 5;
- **Set 5: N35W, 65˚NE –** High angle structures aligned with the post-mineral hornblende diorite porphyry (HDP) intrusive situated on the south side of the planned open pit.

The major faults in the main King-king deposit and immediate vicinities are generally northwest-trending and dip steeply to the northeast (Sets 3 & 5 above), sympathetic to the trends of post-mineral intrusives and intrusive breccias. Major structures identified by surface geology mapping include the Soysoy Fault, which apparently influenced the course of Soysoy Creek. The Soysoy Fault is also thought to define the south flank of the main deposit. This fault is traceable for 1.5 km along its strike length, extending northwest beyond Kingking River. Several other faults (particularly those traced across Casagumayan and Tiogdan) have been observed within the deposit, and these display localized silicification and associated quartz veinlets along their contacts.

The dominance of the northwest structural component is reflected by the preferred orientation of the post-mineral HDP dikes, the epithermal quartz stockwork zone in the Casagumayan and Tiogdan “bardown” areas and the general elongation of the entire main deposit. The same trend is also expressed by the HDP intrusives that are situated peripheral to the main King-king deposit. It is apparent that these northwest-trending structures played an active part during the emplacement of the mineralized diorite complex and the post-mineral intrusives, although the north-northwest faults also appear to have influenced the emplacement of the HDP as indicated by the dikes near Tiogdan.

On a district-wide scale, the northwest fabric is also reflected in the orientation of the faults and veins and orientation of the longer axes of post-mineral diorite stocks in Binutaan and Diat and the shape and orientation of the biotite diorite and hornblende diorite porphyries in Diat.

7.3.3 Alteration

Four major porphyry alteration zones and two relatively minor hydrothermal alteration zones have been recognized in and around the main King-king deposit. From the central portion of the deposit outward, the major zones are:

1) The K-silicate (potassic) zone which is further subdivided into K-feldspar and biotite subzones;
2) The quartz-sericite-chlorite (QSC) zone;
3) The sericite-clay-chlorite (SCC) zone; and
4) The propylitic zone which is further subdivided into epidote and chlorite sub-zones.

Important mineralization occurs in the first three major alteration zones.

Locally overprinting these major zones is later-stage epithermal alteration that consists of argillic alteration which includes both an intermediate zone and patches of advanced argillic alteration (AAA), and a quartz-dominated zone that is further subdivided into zones of quartz stockwork and pervasive silicification.

The hydrothermal alteration zoning at King-king is typical of other porphyry copper deposits in the Philippines and in other parts of the world. However, important differences between King-king and other Philippine porphyry deposits include the presence of a very well developed potassic zone characterized by widespread secondary biotite and strong, well-developed K-feldspar; a stockwork-pervasive silicification zone that is much more intense than in other deposits; and a phyllic zone characterized by QSC. The absence of a more typical phyllic (quartz-sericite-pyrite) alteration zone is due to the very low total pyrite (<1%) content of the deposit. Also, epidote is not an exclusive component of the propylitic zone as it is in most other porphyry copper systems, but rather is found in all alteration types at King-king, and advanced argillic alteration (which is extensively developed in other deposits such as the Dizon porphyry copper-gold deposit in Zambales) has been observed at King-king only locally in a few faults and structures that are generally outside of the ore zone.

7.3.4 Mineralization

Gold and copper mineralization in the King-king deposit is hosted primarily by the elongate, dike-like N60°W-striking diorite intrusive complex described earlier in Section 7.3.1. The copper-gold mineralization occurs as fracture fillings and to a lesser extent as disseminations in the diorite porphyries (and to a much lesser extent the dacite porphyry) and adjacent wall rocks. Better gold and copper grades appear to occur where there was interaction between the various rock types, such as along contact zones or where several intra-mineral dikes or intrusives cut the earlier lithologies.

The majority of the mineralization in the King-king deposit is hypogene (sulfide). Rapid regional uplift and erosion likely caused the nearly complete removal of a classical leached cap and the extensive decimation of the underlying oxide and supergene enriched zones (or perhaps prevented the development of significant oxide and supergene enriched zones) typically found in other porphyry deposits. For process development purposes, two types of mineralization are considered: sulfide and oxide (which includes mixed oxide-sulfide material).

7.3.4.1 Oxide Zone

In general, the depth of oxidation is greatest under ridge tops (reaching 150 m in thickness), and thins progressively to the valley bottoms where oxidation may only extend to a depth of a few meters due to active erosion. The Lumanggang area contains the greatest thickness of surface oxidation. The transition between the oxide and sulfide horizons is usually quite abrupt, with mixed zones seldom more than a few tens of meters thick.

In the oxide and oxide-sulfide (mixed) zones, partially oxidized chalcopyrite and bornite are occasionally found along with weak acid soluble copper mineralization mostly occurring in silicates and phosphates that are only observable with combined backscatter electron imaging and x-ray mapping techniques. Copper silicates are the most abundant oxide mineral group present, with copper silicate minerals containing MgO and FeO being the most prevalent in the oxide zone. Because the bright colors of these minerals and their usual association with the more visible, ridge-forming, highly silicified outcrops and quartz stockworks, past impressions of the relative abundance of malachite and chrysocolla in the deposit have been exaggerated, due to these silicified outcrops being generally found only in limited areas.

Gold is relatively abundant in the oxide zone, as evidenced by widespread gold panning and small-scale mining activities on the oxidized slopes of Casagumayan and Tiogdan. Some of the gold particles examined in the possession of the small-scale miners were found to be attached to quartz and/or blebs of magnetite. According those who pioneered gold panning at King-king, coarser gold particles were more abundant in the original soil horizon that existed over the deposit. Gold particles panned along the creeks typically range up to 2 mm in diameter.

7.3.4.2 Mixed Zone

The mixed zone consists of the oxide minerals described in the previous section, partially oxidized chalcopyrite and bornite, and limited supergene chalcocite and covellite mineralization. Chalcopyrite and bornite are partially to completely be replaced by the secondary chalcocite and covellite, with covellite almost always rimming bornite.

7.3.4.3 Sulfide Zone

Hypogene (sulfide) copper mineralization consists predominantly of chalcopyrite with overall lesser amounts of bornite and primary chalcocite, the latter occurring as fracture fillings in the areas of the deposit that are distinctly more bornite-rich. Bornite dominant areas include the biotite diorite porphyry, where bornite partially replaces chalcopyrite and occurs in amounts roughly equal to or greater than chalcopyrite.

Lesser sulfide minerals include molybdenite, which commonly occurs as fracture coatings and in quartz veins, digenite, covellite, tetrahedrite, galena, and sphalerite. The minerals have been observed in trace amounts in petrographic studies. There also appears to be a higher grade molybdenite-bearing shell along the fringes the copper-gold mineralization.

Gold occurs in the sulfide zone of the deposit in free form in close association with bornite and as ex-solution intergrowths in other sulfides, particularly chalcopyrite. Native gold is occasionally observed on fractures and in quartz veinlets.

The King-king deposit is characteristically low pyrite (<1%), as reflected by the relative absence of a pyrite halo that is commonly developed around most porphyry copper deposits. The low pyrite content of the deposit to some extent may have contributed to the deposit's lack of a classic leach cap and supergene enrichment zone, as there may not have been enough pyrite present to generate sufficient acid to form these zones.

8 DEPOSIT TYPES

In general terms, the King-king gold-copper deposit is consistent in type and form with other bulk-tonnage copper-gold porphyry deposits of the Philippines and elsewhere in the world. These consistencies are summarized as follows:

- The King-king gold-copper deposit is associated with and hosted by stock-size intrusive rock bodies ranging in composition from diorites to dacites;
- The four classic alteration assemblages typically found in porphyry deposit are present in the King-king deposit, situated in a typical zonal distribution pattern of shells that extend outwards from a central core potassic zone (subdivided into K-feldspar and biotite subzones) into a phyllic zone consisting of QSC alteration, an argillic zone comprised of SCC alteration, and an outermost propylitic zone, which is subdivided into epidote and chlorite sub-zones;
- The deposit contains a typical suite of porphyry-style copper minerals consisting predominantly of chalcopyrite and lesser bornite in the lower sulfide (hypogene) zone, chalcocite, cuprite, and covellite in a weakly-developed transition zone, and malachite and chrysocolla in the uppermost oxide zone. Gold occurs in all zones as free gold (predominantly in the oxide and transition zones) and to a lesser extent associated with sulfides in hypogene zone.

Three factors that suggest the King-king deposit is somewhat different from other Philippine porphyry copper deposits include:

- The deposit contains a quartz stockwork zone that, with some exceptions, generally has elevated gold values averaging more than 1.0 g/t compared with the surrounding zones;
- The occurrence of widespread biotite alteration and the presence of a strong and well-developed K-feldspar-rich alteration zone, which along with the stockwork/pervasive silicification zone provide assemblages that are much more intense than in other deposits of this type;
- The absence of a typical phyllic (or quartz-sericite-pyrite) alteration zone, which is attributed to the very low total pyrite (<1%) content of the deposit.

9 EXPLORATION

Prior to SAGC's tenure, exploration of the King-king deposit was conducted intermittently by previous project owner/operators beginning in the late 1960s and continuing until 1997. This work included:

1) Surface mapping and sampling;
2) Drilling (primarily diamond core);
3) Underground adit and raise sampling;
4) Geochemistry (soil, stream, and down-hole);
5) Development of cross sections, long sections, and plan maps;
6) Physical and computer-generated three-dimensional modeling.

Between 1997 and the entry of SAGC and its affiliated predecessors into an agreement with NADECOR in 2009, no exploration work of any kind took place on the project (See Section 6 – History). Since assuming control of the King-king Project, SAGC's exploration work has been confined to an extensive and detailed review of all historic geologic information and data from the exploration activities summarized above that were generated by previous project operators. Based on this thorough review and analysis of the historic data (which in the opinion of the Qualified Person constitutes the first step in a logical exploration plan), SAGC determined that three of five exploration drillholes situated in the Diat and Binutaan areas to the northeast of the planned open pit intersected favorable host lithologies and highly anomalous copper and gold values. With further exploration work (geophysics, soil and rock geochemistry, drilling), portions of this mineralization may prove to be economically extractible in separate open pits that would be satellite to the main King-king pit. The most notable intercepts from these data are from Benguet drillhole DD-1 in the Diat area, and Echo Bay drillholes EBD-1 and EBB-1 in the Diat and Binutaan areas, respectively. These holes are located approximately 1 km to 4 km northeast of the current King-king pit limit. Summaries of the significant intervals in these holes are included in Table 10-4 in Section 10 – Drilling. SAGC plans to formulate a phased exploration plan to further define the geometry, size, and grade of these areas of potential after completion of the planned feasibility study and subsequent development of the main King-king deposit.

10 DRILLING

As summarized in Section 6 (History), three companies completed drilling campaigns on the King-king property - Mitsubishi, Benguet, and Echo Bay. The drillhole database provided to IMC consisted of 276 holes drilled by these companies, which represented 89,922 meters of drilling. Table 10-1 shows the drilling by campaign (RC = Reverse Circulation). Figure 10-1 shows the drillholes by drilling campaign.

Table 10-1: Drilling by Campaign

Campaign Description	No. of Holes	Meters	No. of Intervals
Mitsubishi Core Holes	54	13,031	4,352
Benguet Core Holes	69	19,247	6,412
Benguet RC Holes	25	4,926	4,456
Echo Bay Core Holes	128	52,718	18,440
TOTAL DRILLING	276	89,922	33,660

Most of the Echo Bay holes and a significant number of the Benguet core holes are angle holes oriented southwest to intersect structures oriented northwest with a northeast dip.

The core holes were nominally sampled on 3m-down-hole intervals, though a portion of the early Echo Bay holes were sampled on 2m-intervals. The Benguet RC holes were sampled on 1m intervals. Of the 33,600 intervals, 33,466 were assayed for total copper, 33,323 for soluble copper, and 29,192 for gold. Gold analyses were not completed for the Mitsubishi drilling. Soluble copper assays were obtained for almost every interval for which total copper was done.

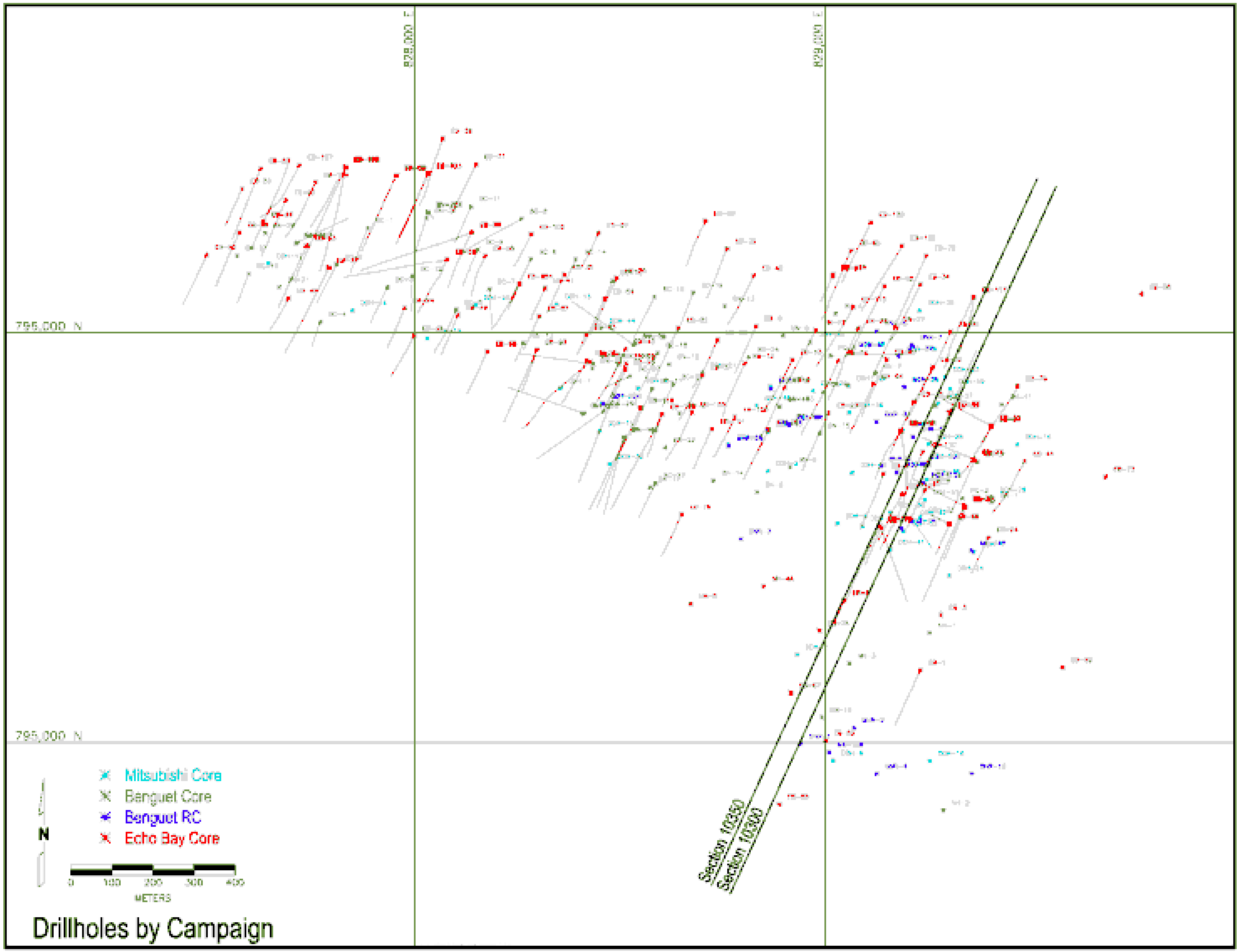

Figure 10-1: Drillhole Locations by Campaign

Table 10-2 shows details of the drilling by hole series and drillhole type – diamond core holes (DDH), and reverse circulation holes (RCH).

Table 10-2: Drilling History by Company

54	DDH Holes	Mitsubishi	1972	DDH 1-54
23	DDH Holes	Benguet	1991-1994	BC 1-23
38	DDH Holes	Benguet	1991-1994	BN 1-31(A&B)
3	DDH Holes	Benguet	1991-1994	NH 1-3
5	DDH Holes	Benguet	1991-1994	PQ 1-5
10	RCH Holes	Benguet	1991-1994	BNR 1-9
13	RCH Holes	Benguet	1991-1994	M-Series Holes
2	RCH Holes	Benguet	1991-1994	PQ-Series Holes
128	DDH Holes	Echo Bay	1996-1997	EB 1-126

It is the opinion of IMC that the drilling done to date is sufficient to develop an NI 43-101 compliant mineral resource for the King-king deposit. Based on the structural zones developed by IMC to control block grade estimation (see Section 14.2.6), the area represented by the drillhole samples is approximately 1,695,000 square meters (m^2), or about 170 hectares. The 400m bench is a central bench in the deposit. It contains 192, 15 m composites assayed for copper and 108 composites with acceptable gold assays. Dividing the sampled area of 1,695,000 m^2 by the number of composites and taking the square root provides a semi-quantitative measure of average sample spacing in plan view. This results in average sample spacing of 94 m for copper and 125 m for gold.

Section 9 (Exploration) discusses three of five historic holes that have been determined by SAGC to be indicative of potential additional mineralization that could prove to be economically extractable after additional exploration work. The significant intercepts from these holes are summarized in Table 10-3, and Figure 10-2 shows the location of the five historic exploration drillholes.

Table 10-3: Significant Intercepts from Outlying Historic Drillholes

Location	Drillhole	From (m)	To (m)	Length	Cu (%)	Au (g/t)
Diat Area	EBD-1	3	683	680	0.151	0.269
	including	3	126	123	0.176	0.190
	including	147	180	33	0.012	0.850
	including	372	683	311	0.234	0.352
	DD-1	3	312	309	0.177	0.254
	including	3	84	81	0.441	0.336
	including	84	237	153	0.051	0.251
	including	237	312	75	0.146	0.172
Binutaan Area	EBB-1	0	409	409	0.098	0.534
	including	78	93	15	0.061	4.160
	including	105	117	12	0.067	7.753
	including	159	366	207	0.143	0.192

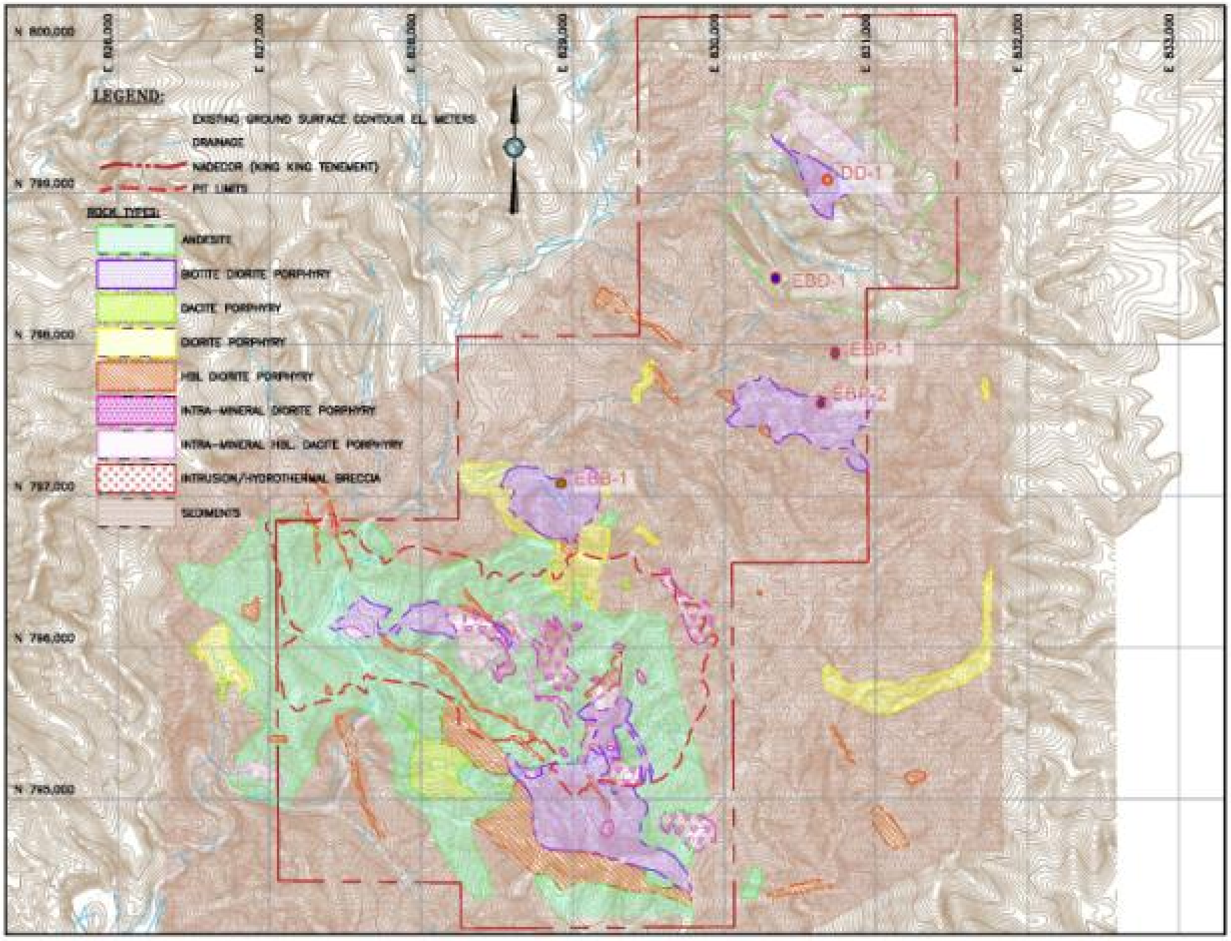

Figure 10-2: Historic Exploration Drilling Location

10.1 HISTORIC DRILLHOLE COLLAR LOCATION CHECK

During the June 5, 2010 site visit, Resource Evaluation, Inc. (REI) attempted to locate 21 randomly-selected drillhole collars in the field. Because of dense vegetation overgrowth and sloughing of cutbanks at drill sites, only six drillhole collars were located. Two of the drillholes located contain steel casing with valves and are currently in use as water wells - NH-1 (a Benguet hole drilled in the early 1990's) and EB-3, an Echo Bay hole drilled in 1995. The collars for two Echo Bay holes (EB-27 and EB-121) were located and both have small (0.3 m) roughly circular concrete pads surrounding open PVC pipe collar casing. Of the remaining two drillholes, an open drillhole collar (no concrete pad) for M31-2R (an RC twin drillhole of the earlier Mitsubishi DDH-31B) was located, as was the collar of the Benguet drillhole NH-4, which contained a cylindrical concrete plug.

In the opinion of REI, the fact that the majority of the drillhole collars selected for field checks were not locatable in the field is not a material issue. In the case of each of the 21 randomly selected holes, it was clearly evident that a drill site had been constructed. The likelihood that any of the drillholes selected were not drilled is remote.

10.1.1 Recent Drilling by SAGC

In addition to the historic drilling described in the previous sections, SAGC commissioned 14 holes in 2011 that consisted of three holes (SAG-01 through SAG-03) designed to further evaluate local areas of the deposit for enhancements to mineral resource estimation, six holes (SAGT-01 through SAGT-06) to obtain geotechnical data for pit slope design, one hole to provide samples for additional metallurgical testing (SAM-01), and four holes (SAH-01 through SAH-04) to provide hydrogeological data for open pit dewatering well design. All of these holes, except SAH-02, were diamond core drillholes drilled by either Drillcorp or Indodrill, both of which are Philippine drilling contractors. Total depth for the 14 holes is 5,980 meters. The drillholes are summarized in Table 10-4. As part of its June 5, 2010 site visit, REI examined the two drill rigs that were in operation (one from each contractor) found the site set-ups and equipment to be acceptable.

Table 10-4 and Figure 10-3 below show the 14 new holes completed by SAGC.

Table 10-4: Recent Holes Completed for SAGC

Hole No.	Total Depth (m)	Purpose	Company	Planned Hole No.
SAG-01	651.1	Mine Engineering	Drillcorp	R-12
SAG-02	454.9	Mine Engineering	Indodrill	R-03
SAG-03	430.0	Mine Engineering	Indodrill	R-02
SAGT-01	498.3	Geotechnical - Open Pit	Drillcorp	GT-03
SAGT-02	556.2	Geotechnical - Open Pit	Drillcorp	GT-02
SAGT-03	462.3	Geotechnical - Open Pit	Indodrill	GT-01
SAGT-04	500.0	Geotechnical - Open Pit	Drillcorp	GT-04
SAGT-05	444.6	Geotechnical - Open Pit	Indodrill	GT-08
SAGT-06	550.0	Geotechnical - Open Pit	Drillcorp	GT-07
SAM-01	250.0	Metalliurgical Testing	Indodrill	Met-1
SAH-01	300.1	Hydrogeology	Indodrill	HG-1
SAH-02	189.0	Hydrogeology	Drillcorp	HG-4
SAH-03	300.0	Hydrogeology	Drillcorp	HG-6
SAH-04	400.1	Hydrogeology	Drillcorp	HG-2

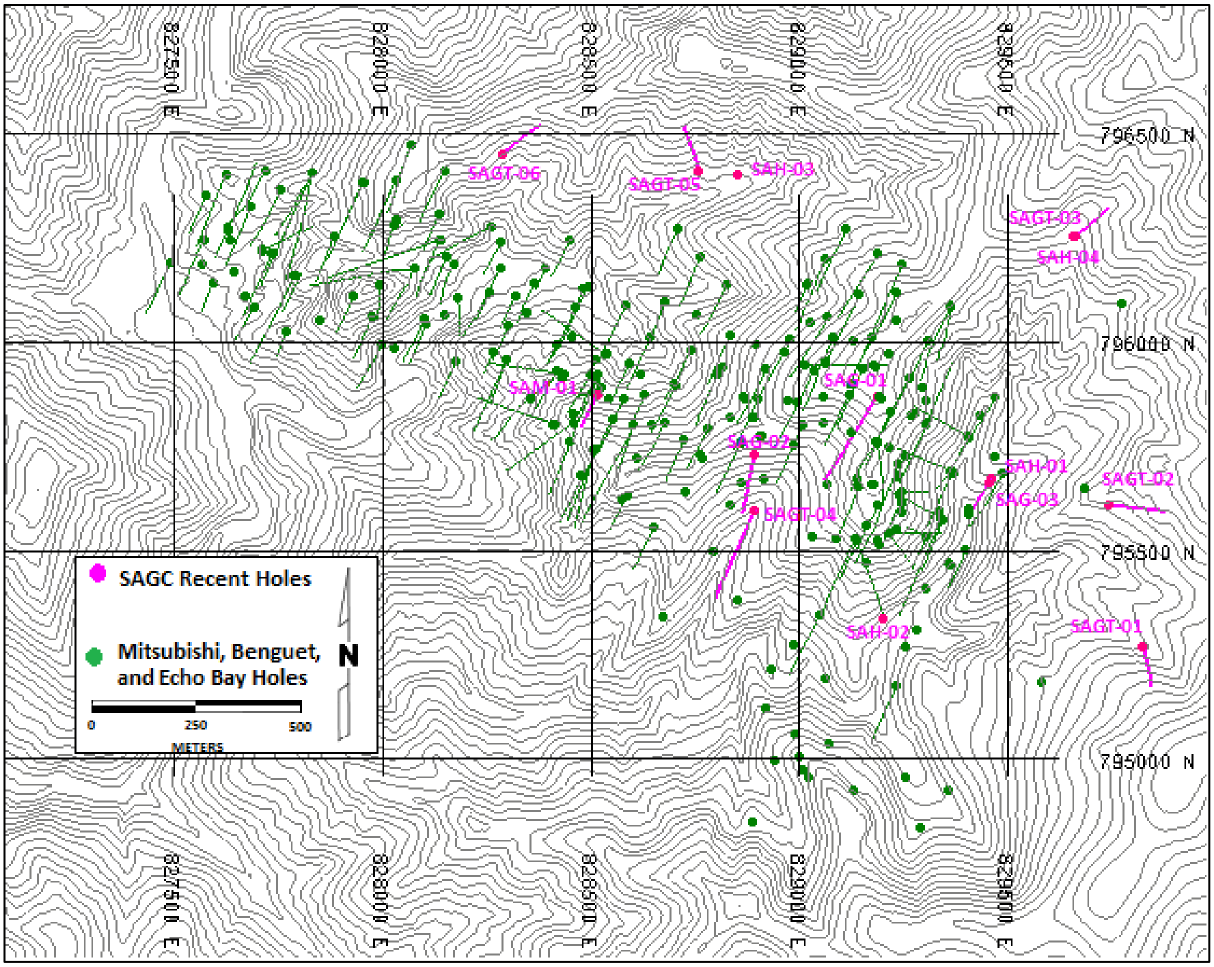

Figure 10-3: Recent Hole Locations by SAGC

IMC conducted initial analysis of the assays received from SAGC holes. The drilling information for holes SAG-01, SAG-02, SAG-03, SAH-01, SAH-02, and SAM-01 was composited into 15m composites and then compared to the resource block model. A total of 128 composites were compared between the resource block model and the new drillhole data. The total copper assay in the resource block model is 15.7% lower than the new drillhole data. The gold assay in resource block model is also lower by 18.6% compared to the new drillhole data.

Table 10-5 below shows the results of the initial analysis.

Table 10-5: New Drilling Assay versus Block Model – 15 m composites

	No. of	Copper (%)			Gold (g/t)		
Hole	Comps	Comp	Block	%Diff	Comp	Block	%Diff
SAG-01	33	0.400	0.273	-31.8%	0.318	0.201	-36.8%
SAG-02	28	0.200	0.282	41.0%	0.131	0.141	7.6%
SAG-03	24	0.218	0.255	17.0%	0.181	0.102	-43.6%
SAH-01	17	0.203	0.317	56.2%	0.122	0.068	-44.3%
SAH-02	11	0.251	0.198	-21.1%	0.095	0.102	7.4%
SAM-01	15	0.861	0.386	-55.2%	0.489	0.551	12.7%
TOTAL	128	0.337	0.284	-15.7%	0.226	0.184	-18.6%

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Estimates of mineralized tonnage and grade for the King-king deposit have historically been based upon assays derived from drilled intercepts. Approximately 33,660 samples were collected over the course of the project and processed by four separate analytical laboratories that include Benguet's in-house labs at Dizon and Balatoc, McPhar Labs in Manila and Inchcape Labs in Manila. The sample preparation was not completed by SAGC or any of SAGC's contractors, but was completed by the companies previously working on the project.

11.1 MITSUBISHI DRILLING PROGRAMS

Sample preparation and analysis procedures for the Mitsubishi drilling program from 1969-1972 were not available for review. The sample chain of custody (COC) and security procedures used by Mitsubishi are unknown.

11.2 BENGUET DRILLING PROGRAMS

Sample preparation and analysis procedures for the Benguet drilling programs are described in the reference titled "Benguet Sample & Assay Procedure." Core samples were collected on 3 m intervals and split at the site, placed in sample bags, and sent to the company's sample preparation laboratory in Davao City. There the samples were dried and crushed to a nominal 1/8 inch size. The crushed sample was split down to approximately 500 grams that was then pulverized to 150 mesh. The pulp was then divided into two 250- to 300-gram samples, one for analysis and one for reserve. The pulps were then shipped to Benguet's in-house analytical labs at either Balatoc or Dizon for analysis. Total copper analysis was completed on a 0.5-gram sample. Three-acid digestion was used (perchloric, nitric, and hydrochloric acids) prior to analysis by atomic absorption (AAS).

Soluble copper analysis was done on a 1.0-gram sample. Digestion was with 5% sulfuric acid at room temperature for two hours, with solution stirring every 15 minutes. As with total copper, final analysis was done by AAS.

Based on the documentation provided to IMC, it appears that the Benguet laboratories also performed gold analysis by solution methods rather than by fire assay. The gold analyses were based on 10.0-gram samples. Nitric acid was first added under low heat to decompose sulfides. Potassium chlorate was then added, followed by hydrochloric acid (HCL), which formed aqua regia and dissolved the gold. Additional HCL was added to dissolve salts that may have formed, and MIBK (methyl isobutyl ketone) was added to collect the gold. Final gold analysis was by AAS. In light of Benguet's gold analytical procedures, Echo Bay's decided to re-assay Benguet samples for gold.

The specific sample COC and security procedures employed by Benguet are not known, although it is likely that the samples were continually under Benguet company control, given that the samples were prepared as well as analyzed in company laboratories.

11.3 ECHO BAY DRILLING PROGRAMS

11.3.1 Core Splitting and Sample Preparation

Core was transported as soon as possible to a centrally located on-site logging area for inventory and geotechnical logging. The geotechnical logging was completed by trained technicians following procedures provided by Knight Piésold (J. Haile, 1995). After completion of the geotechnical logging, core was then transported daily to the Davao office warehouse for detailed geologic logging. The core was photographed prior to splitting and the photographs were transferred to a CD-ROM format for ease of storage and access.

Core splitting was completed by trained technicians using conventional hydraulic knife-blade splitters. One half of the core was placed in permanent storage in a secure, enclosed warehouse. The remainder of each interval was transported daily to a sample preparation facility located in Davao City that was independently operated by Inchcape Testing. The entire sample was crushed to minus one-tenth inch using a jaw crusher. A sample weighing approximately one kilogram was then split from the crushed material using a riffle splitter. This entire split was pulverized using a large capacity disk pulverizer. The pulps were reduced in size to a nominal 90 percent passing through a minus 200 mesh screen. A pulp split, weighing approximately 150 grams, from each sample was then shipped to the Inchcape Testing laboratory in Manila by air freight. The remainder of the pulp and the coarse reject were returned to KMI for secure, permanent storage in an enclosed warehouse.

Gold assaying was completed by fire assay with an AAS finish on fifty-gram charges. Total copper and molybdenum were assayed using a total digestion followed by atomic absorption technique. A weak acid, room temperature digestion followed by AAS analysis was used for acid soluble copper analysis.

11.3.2 Assay Quality Control/Quality Assurance

The Quality Assurance/Quality Control (QA/QC) program used by KMI was designed by Ken Lovstrom, a consulting geochemist, together with KMI staff early in 1996 and was fully implemented in the second quarter of 1996. To provide the highest degree of assurance for assay data, KMI used three reputable independent assay laboratories. The primary lab was Inchcape Testing Services located in Manila. The secondary check laboratory was Cone Geochemical located in Denver, Colorado. Chemex Labs Ltd. of Vancouver, British Columbia was used for limited check assaying and for round robin assays of control samples. Echo Bay’s chain of custody and security procedures were not documented in writing, but it is highly likely that rigid procedures were followed, based on the authors’ first-hand experience with other Echo Bay projects that were overseen by Ken Lovstrom.

11.3.2.1 Assay Reliability

The reliability of numerical data is measured by precision and accuracy. Precision is the degree of reproducibility, regardless of accuracy. Accuracy is the degree of closeness to a true and generally known value. The limit of detection (LOD) is another important term because assay labs define a detection limit as "that point at which precision is plus or minus 100 percent."

Therefore, by definition all assay values for concentrations larger than the LOD will have greater precision. The next step in providing quality assurance for any analytical program is the quantification of results. The limit of quantification (LOQ) is the point that 95 percent of the samples fall within plus or minus 10 percent and assumes that one in every twenty samples falls outside this range. The LOQ varies with the concentration, detection limit and precision factor for the process and assumes a homogenous sample. This is a critical assumption and each lab produces this qualifying statement quickly.

The LOQ for Inchcape Testing was 0.5 parts per million (ppm) gold, for Chemex it was 0.6 ppm gold, and for Cone it was 0.1 ppm gold. The large discrepancy between LOQ for Cone versus both of the other labs was a function of the final separation technique used by Cone. Detection limits and LOQ's for copper are very low relative to copper grades of economic interest and are not critical to the quality control program. For gold, precision decreases from 95 percent within plus or minus 10 percent to 95 percent within plus or minus 100 percent below 0.5 ppm. This does not mean that data below this threshold is unquantifiable. The following table, provided by Inchcape Testing, defines the precision curve for all data. This curve was the accepted tolerance limit to which data generated by the assay labs should be held.

Concentration (ppm gold)	Tolerance
0.005	±100%
0.050	±50%
0.100	±25%
0.500	±10%

Analytical accuracy and precision are dependent on the techniques used:

- Fire Assay
- Atomic Absorption

A variety of other factors including technique, detection limits, sampling, sample preparation, extraction, homogenization, reagent purity, instrumentation and professionalism all contribute to the integrity of analytical data. These factors can have an accumulated effect on assay data. The presence of coarse gold alone can alter an assay value plus or minus 17 percent at the 0.5 ppm concentration level, significantly violating the assumption of “homogeneity” incorporated into the tolerance values listed above. This was nearly double the tolerance for a homogenous sample at the same concentration. A variance in gold assays of plus or minus 11 percent and copper assays of plus or minus 7 percent are accepted as within industry standards due to the nature of the analyses used by KMI. Figure 12-4 in the next section shows an example of tolerance, or precision, versus grade for gold, and that in general, for gold, actual precision is not as good as the table above based on theoretical “homogenous” samples.

Accuracy was established by using control samples. These control samples are used to check for laboratory assay "batch busts", data entry errors, or other analytical problems. Running means for control samples having concentrations above, below, and at the LOQ are compared with accepted true values for those samples as established by a round robin test.

Precision was established by comparing assay pairs and was expressed as percent running standard deviation. Variance was defined by the ratio of the running mean and standard deviation.

11.3.2.2 Quality Control Protocol

The intent of this QA/QC program was to monitor assays on a per batch basis using control samples, duplicates, blanks, replicates and umpire laboratories to insure assay integrity. KMI monitored seven different types of samples to detect the precision and accuracy of assays provided by the various assay laboratories.

- Bulk pulp control samples
- Bulk reject control sample
- Duplicate core sample
- River sand sample
- Second laboratory check sample
- Lab duplicate sample
- Certified standards

11.3.2.2.1 Bulk Pulp Control Samples "A”

During 1996, KMI used six different bulk pulp control samples, KM 1 through 6. Control samples KM 1 through 3 were submitted randomly in oxidized zones, and KM 4 through 6 were used in sulfide zones. All were designated by the letter “A" immediately following the hole and sample number and are easily identifiable on assay sheets. Forty kilograms of split core from the project were composited to obtain a desired grade for copper and gold. Bondar-Clegg in Reno, Nevada pulverized and mixed the bulk samples and generated 75 gram pulp packets. Ten of each of the pulp packets were assayed by Cone Geochemical in Denver, Colorado to establish initial concentration ranges for gold and copper. Subsequently, each control sample has been resubmitted to Cone, Inchcape, and Chemex for round robin assaying to determine the "true" values for each sample.

A bulk pulp control sample was submitted at a frequency of one per every twenty samples. Assays are reviewed against the accepted year to date means and global means as established by the round robin analysis for gold and copper. Assays that fall outside these criteria are re-assayed along with the five preceding and five following drill samples in that batch. After the re-assay returns it will be placed in the database as an original assay.

11.3.2.2.2 Bulk Reject Control Samples "B"

The bulk rock control sample used in this program was identified by the letter "B" immediately following the sample number. The material for this sample was a composite, oxide, coarse reject from drillhole EB-7, with known gold and copper values. Ten samples were initially submitted to Cone for analysis to set assay ranges for gold and copper. A bulk reject was submitted one per day per hole and was specifically designed to check sample preparation. Any assays that fall above or below two standard deviations are re-assayed along with the five preceding and five following drill samples in that batch.

11.3.2.2.3 Duplicate Core Samples "C"

For every fortieth (40th) sample, the second half of the split core was used completely for assay. The purpose of this sample was to check the core splitting and sampling procedures for quality and bias.

11.3.2.2.4 River Sand Samples "D"

This blank control sample was generated by the Inchcape Testing sample prep facility. A sample of ordinary river sand was run through the crushing and pulverizing equipment after each sample. Every tenth sample was submitted for assay. The purpose of this sample was to check the sample preparation procedures for cleanliness and cross contamination.

11.3.2.2.5 Second Lab Check Samples "E"

Each month, duplicate pulps of five percent of the assays received are re-submitted to a second lab. The selection of these samples was random and not biased toward a particular range of concentrations for either gold or copper. Cone was chosen as the second lab. The purpose of this sample was to provide an outside lab check of the primary lab.

11.3.2.2.6 Lab Duplicate Samples "F"

Inchcape re-assays one sample in ten as an internal check. This re-assay was reported on the final sheet of each assay report. This data was tracked by KMI personnel and was given the letter "F" to distinguish it from the various other check samples.

11.3.2.2.7 Certified Standards "G"

Inchcape used internally, several certified standards including Canmet and Gannett standards for copper and gold. KMI's staff monitored and evaluated these assay results. Several standards are included in each batch of samples fired.

11.4 IMC/REI OPINIONS OF SAMPLE PREPARATION, SECURITY AND ANALYTICAL PROCEDURES

It is the opinion of IMC that the Echo Bay sample preparation, security, and analytical procedures are adequate for the nature of mineralization being tested, namely a bulk, relatively low grade base metal deposit that includes precious metals.

It is also the opinion of IMC and REI that the Echo Bay QA/QC program exceeded industry standards at that time and also exceeds current standards in place at most companies. Also the principal authors worked with Ken Lovstrom (now deceased) on other Echo Bay projects and have high regard for his work.

The Benguet sample preparation and analytical procedures, as described in information provided to IMC, also appear appropriate. The total copper and soluble copper analysis methods are also appropriate. The Benguet gold analysis method, however, is complex, and not commonly used. As will be discussed in the next section of this report, there appears to be a bias with regard to the Benguet gold assays. None of the Benguet gold assays were included in the resource model development. Total copper results; however, appear to be in line with Echo Bay results.

12 DATA VERIFICATION

IMC performed the following data verifications on the King-king sampling database:

- A significant portion of the assays in the database were compared with assay certificates and geologic logs,
- For the 1997 Feasibility Study, Echo Bay re-assayed a significant number of Benguet samples for copper and gold. IMC did comparisons of the Echo Bay and Benguet assays for these sample intervals,
- Don Earnest of REI pulled 100 samples from Benguet and Echo Bay existing core to be assayed for copper and gold for comparison with original assays.

The following sections include the details of the various studies.

12.1 COMPARISONS OF ASSAYS WITH ORIGINAL ASSAY CERTIFICATES

12.1.1 Echo Bay Assays

IMC originally selected 14 Echo Bay drillholes to compare assays in the database with original assay certificates. These holes were:

EB-2	EB-7	EB-8	EB-21	EB-26
EB-35	EB-68	EB-86	EB-88	EB-92
EB-95	EB-105	EB-115	EB-121	

These were a relatively random selection of drillholes, though there was a bias toward selecting more of the higher grade drillholes.

Three of the 14 holes, EB-7, EB-8, and EB-68 did not have all the assay certificates available, though the data compared well with the certificates that were available. Other than EB-115 most of the denoted errors are minor in nature except for a gold assay in EB-2 and a total copper assay in EB-92 which were off by an order of magnitude. EB-115 however contained three total copper assays and one gold assay with order of magnitude errors.

Due to the results of EB-115, and also the three holes with incomplete assay certificate coverage, IMC selected seven additional Echo Bay holes to audit:

EB-9	EB-116	EB-119	EB-124	EB-11
EB-89	EB-63			

Certificate data was incomplete for EB-9 and EB-11. Results for EB-116 were relatively poor, similar to EB-115, which indicated the possibility of a significant lapse in the data entry/verification for a portion of the Echo Bay data.

IMC then audited EB-113, EB-114, EB-117 and EB-118 to bracket the problem holes. Drillholes EB-113, EB-114, and EB-117 are confirmed. Certificate data was only available for the first 31 records of EB-118 and none of the assays compared with the database. The first 33 (not 31) assays in EB-118 were the same as EB-117, indicating a portion of EB-117 was copied over the EB-118 data. A review of the cross sections indicated the certificate data compared well with surrounding holes (and what was originally in the database did not). Also, the data in the lower portion of EB-118, the portion not covered by the certificates, looks reasonable compared to surrounding holes.

IMC then checked EB-120, EB-122, EB-123, EB-125, and EB-126, which represent all the Echo Bay drilling after EB-118, plus three additional holes EB-43, EB-53, and EB-77. The latter three were chosen because no other holes from the 40's, 50's, or 70's series had been selected. These holes checked reasonably well.

Overall 33 of the 128 Echo Bay holes were audited which is about 26% of the holes. Certificate entries were available for 84% of the total copper assays, 82% of the soluble copper assays, and 89% of the gold assays. The overall error rate was approximately 1%. The overall error rate is acceptable, though IMC would expect it to be approximately half that in a verified database. The fact that these errors clustered in three holes probably drilled about the same time indicates a lapse in the data entry procedures for a brief period near the end of the Echo Bay drilling program.

IMC corrected the known errors, replacing the database values with certificate values.

12.1.2 Benguet Assays

There were no assay certificates available to IMC for the Benguet holes. There were however, image files from old Benguet drill logs that also included assay values for total copper, soluble copper, and gold. Minimally, this allowed verification that there was not any tampering with, or errors introduced into the database since the Benguet tenure.

IMC selected 14 Benguet holes for review:

BC-5	BC-11	BC-16	BC-21	BN-18
BN-20	BN-25B	BNR-2	BNR-7	BNR-10
M25-3R	NH-1	PQ-3	PQ-5	

BNR-2, BNR-10, M25-3R and the upper portion of BNR-7 were sampled by reverse circulation drilling. Assays were completed on 1m intervals. On the logs, averages over three meter intervals were recorded. IMC averaged the database values to complete the comparison.

Results of the comparison were good. The bottom of the table shows an error rate of 1.2% for total copper, 0.6% for soluble copper, and 1.8% for gold. This is a bit high, but it can be observed that only approximately three of the assays amounted to order of magnitude errors (a gold assay in BN-18 and BNR-2 and a soluble copper assay in BNR-7).

IMC did not change any values in the database based on this comparison. Since the check was against data in logs, not assay certificates, there is no way of knowing which value is the correct one. Also, as noted above, most of the differences are minor.

12.1.3 Mitsubishi Assays

To IMC's knowledge there are no available assay certificates for the Mitsubishi data. Only total copper and soluble copper were assayed for those samples.

12.1.4 Other Data Checks

IMC did a listing of data records with soluble copper greater than or equal to total copper and reviewed these against certificates when available. A cluster of these in EB-59 showed that what was recorded in the database as soluble copper assays were actually gold assays for 18 records. These were replaced with the correct values from the assay certificates.

A listing of records with total copper equal to gold showed a cluster of records in BNR-4 where the gold assays in the database were actually total copper assays. The errant gold assays were replaced with values from the logs.

It was also discovered that several assays were represented in the database as either 0.98 or 0.99 that original certificates indicated were actually 0.098 or 0.099. These were about 10 Echo Bay assays and occurred in total copper, soluble copper and gold. IMC reviewed all 0.98 and 0.99 assays in the database because of this error. It is not certain how, or when, this error was introduced.

Due to the IMC database checks approximately 132 data records were changed compared with the database used for the 2009 due diligence review.

12.2 ECHO BAY RE-ASSAYS OF BENGUET SAMPLES

12.2.1 Re-Assayed Holes

IMC received assay certificates for the following 22 Benguet holes that were re-assayed during the Echo Bay Feasibility Study:

BC-1	BC-2	BC-3	BC-7	BC-10
BC-11	BC-13	BC-14	BC-15	BN-1
BN-4	BN-7	BN-8	BN-18	BN-19
BN-20	BN-26	BN-27	BN-29	BN-30
BN-30B	BN-31			

The assay data was entered into the database and verified by IMC. The data amounted to approximately 1,171 total copper assays, 1,493 gold assays, and 139 soluble copper assays. Most of the assays were on the Benguet pulps, not remaining core samples.

12.2.2 Total Copper

Figure 12-1 shows an xy plot and linear regression for the total copper assays. This represents 1,159 assay pairs because pairs with an assay value less than 0.01% or greater than 3.0% were excluded. The statistics indicate a mean copper grade of 0.239% for the Benguet assays versus 0.237% for Echo Bay. The regression equation (forced through the origin) has a slope of 0.989, very nearly one, which is an excellent result. It can also be observed that the samples generally cluster fairly tightly around the regression line.

Figure 12-2 shows another xy plot, this time a plot of base 10 logarithms to show more details at the lower end of the distribution. All 1,171 re-assays are included on the plot. The line on the plot is at a slope of 1. Again, it can be seen that there is very good correlation between the original Benguet assays and Echo Bay re-assays for total copper. It can be seen that there is quite a bit of scatter at the low end of the distribution, at an x-axis value of about -1.5, which corresponds to a grade of approximately 0.03% total copper. It is expected that the assay precision should be low at these low grades.

Figure 12-3 shows a plot that represents precision and bias calculations for the data. The x axis is the mean value for each assay pair, i.e. (Benguet Assay + Echo Bay Assay)/2. The y axis is the %HRD (Half Relative Deviation), calculated as (Benguet Assay – Average)/Average and expressed as a percentage. The average %HRD value for all the points is a measure of bias between the data sets. Another statistic is the %HARD (Half Absolute Relative Deviation) which is the absolute value of %HRD, which ignores the sense of the error or relative deviation. The %HARD is a measure of assay precision. The bottom of Figure 12-3 shows for all samples the precision estimate is approximately 7.2%, (i.e. any assay should be within +/-7.2% of the true value). As Figure 12-3 also shows, precision is poor for lower grade samples and improves as the grade increases. For samples with a mean copper value greater than (or equal to) 0.05% copper, the precision estimate is 5.7%. The bias estimates shown are -3.5% for all data (Benguet > Echo Bay), but only -1.9% for samples greater than 0.05% total copper. These are considered good results.

According to the assay certificates, the Echo Bay total copper assay was based on four acid digestion (HF, HNO3, HCLO4, and HCL) followed by analysis by atomic absorption.

12.2.3 Gold

Figure 12-5 shows an xy plot and linear regression for the gold assays. This represents 1,485 assays pairs because pairs with an assay value less than 0.01 g/t or greater than 5.0 g/t were excluded. The statistics indicate a mean gold grade of 0.454 g/t gold for Echo Bay versus 0.489 g/t gold for Benguet, an approximate 7.7% difference. The regression equation, forced through the origin, has a slope of 0.914, i.e. Echo Bay gold = 0.914 x Benguet gold, which implies an 8.5% to 9% difference in the assays. The Benguet gold assays are biased high compared with the Echo Bay assays.

Figure 12-6 shows another xy plot, this time a plot of base 10 logarithms to show more details of the distribution. It can be seen that for the assays less than -0.75 along the x-axis, which corresponds to approximately 0.2 g/t gold, there is considerable scatter around the 1:1 line for the assay results. This is fairly typical because assay precision at grades lower than the 0.2 g/t threshold is usually poor for standard fire assays. Above about -0.5 on the x-axis (approximately 0.3 g/t gold) the assays tend to cluster fairly well around the 1:1 line though it is noticeable that a significant majority of the assays plot below the line (Benguet > Echo Bay).

Figure 12-4 shows a plot that represents precision and bias calculations for the data. For all samples the precision estimate is 20.6%, which implies that any assay should be within +/-20.6% of the true value. For samples with a mean greater than 0.2 g/t gold, the precision estimate is 14.4%. Considering that the check assays are duplicate samples (versus say re-assays of the same pulp) this range of precision is acceptable for gold. Bias estimates by the %HRD calculation are -7.3% (Benguet > Echo Bay) for all samples and -4.0% for samples greater than 0.2 g/t gold. The Echo Bay gold assays were based on a 50 g fire assay with an atomic absorption finish.

None of the Benguet gold assays were included in the resource model development as discussed in Section 11.4.

12.2.4 Soluble Copper

Check assays of soluble copper were limited to two holes, BN-1 and BN-18.

Figure 12-7 shows an xy plot of Benguet versus Echo Bay soluble copper assays. The line on the graph is at a slope of 1:1 and it can be seen that the Echo Bay assays are always higher than the Benguet assays. The mean grades are 0.381% soluble copper for Echo Bay versus 0.277% for Benguet.

It appears that the Echo Bay soluble copper assay method was a more aggressive assay than the method used by Benguet, though two holes is not very diagnostic. Comparisons of the results of block grade estimation with and without Benguet assays, as discussed in Section 14.5, did not indicate this magnitude of difference in soluble copper results.

The Echo Bay soluble copper assays are based on sulfuric acid digestion followed by analysis by atomic absorption. The Feasibility Study report describes it as “a weak acid, room temperature digestion.”

```
* REGRESSION ANALYSIS                    LINEAR REGRESSION
  DEPENDENT VARIABLE: ebra_tcu           INDEPENDENT VARIABLE: tcu
  OPTIONS: through origin ,biweight
              total    non-
  variable    cases   missing     mean         std dev       minimum       maximum
  ebra_tcu     1159     1159   .23747E+00   .21917E+00   .10000E-01   .23510E+01
  tcu          1159     1159   .23888E+00   .21476E+00   .18000E-01   .26000E+01
  standard error of estimate        .0002    correlation coefficient    .9879
  ebra_tcu =        .989305 * tcu
```

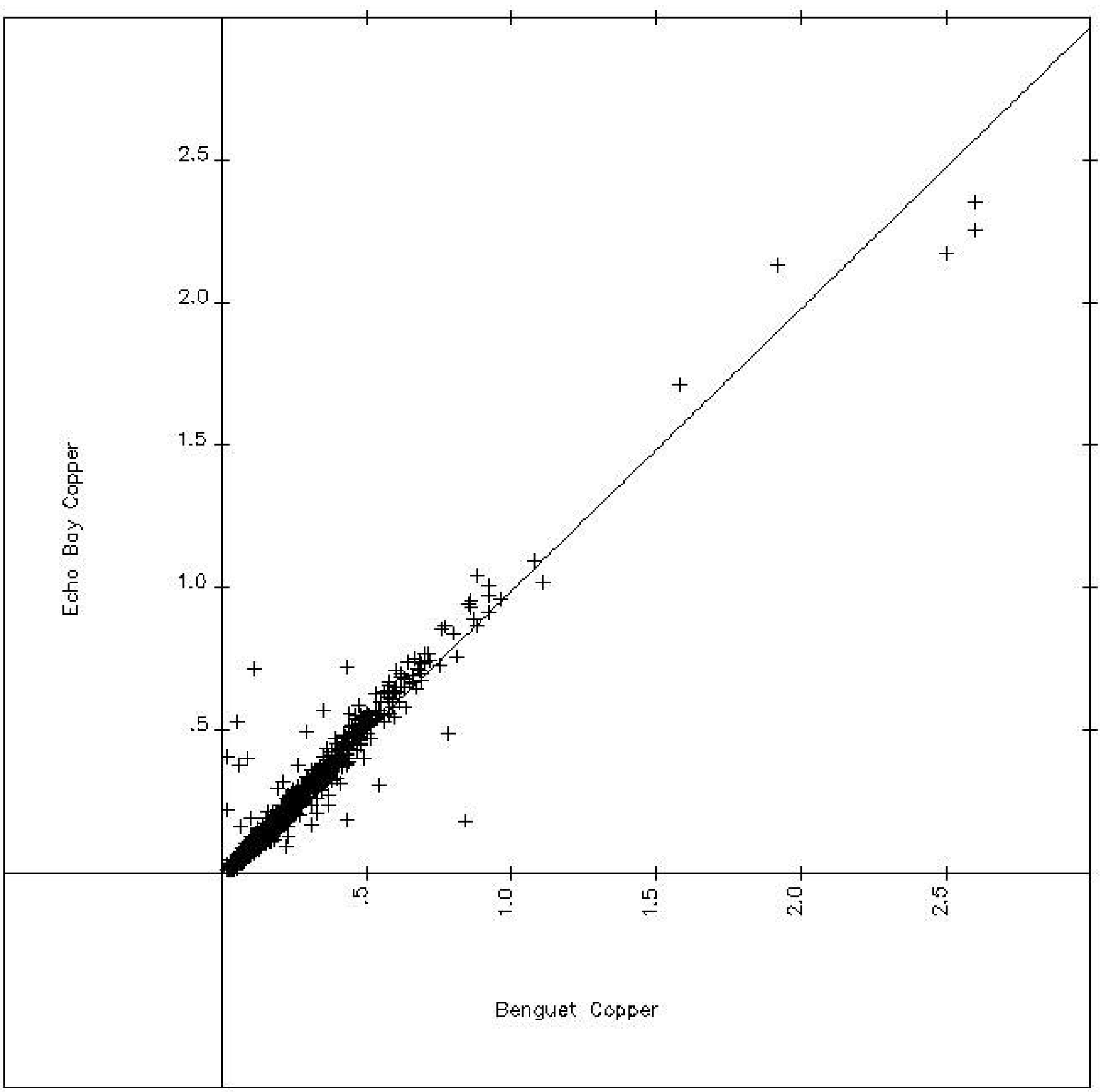

Figure 12-1: Echo Bay Re-Assays of Benguet Samples Total Copper

```
* XY PLOT                                 LINE WITH A SLOPE OF 1
  DEPENDENT VARIABLE: ebra_tcu            INDEPENDENT VARIABLE: tcu
  OPTIONS: through origin
              total
  variable    cases     mean          std dev       minimum        maximum
  ebra_tcu     1171    -.78125E+00    .39804E+00    -.23010E+01    .77085E+00
  tcu          1171    -.75099E+00    .34178E+00    -.17447E+01    .69461E+00
  standard error of estimate      .1254     correlation coefficient      .8997
  ebra_tcu =      1.000000 * tcu
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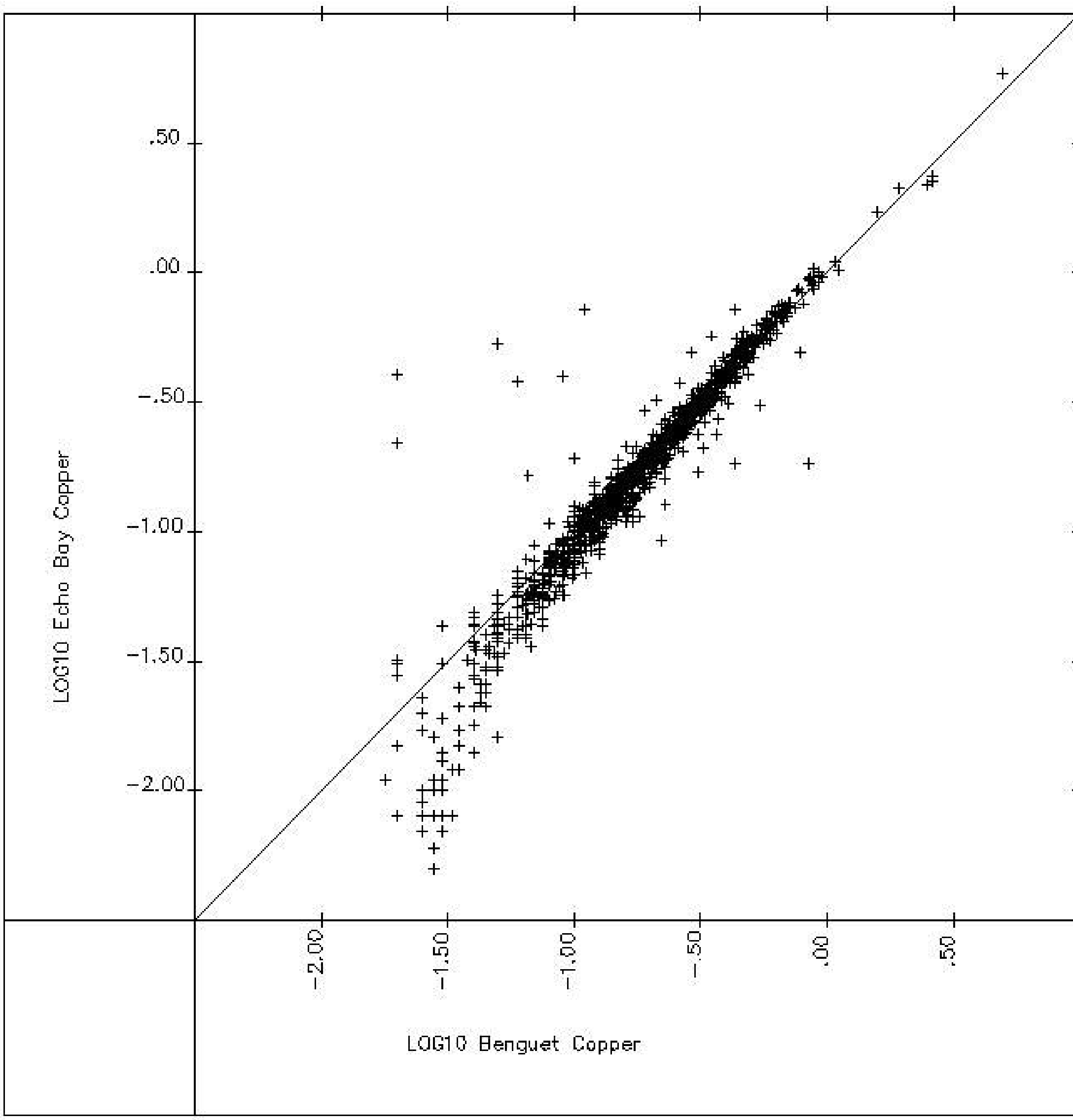
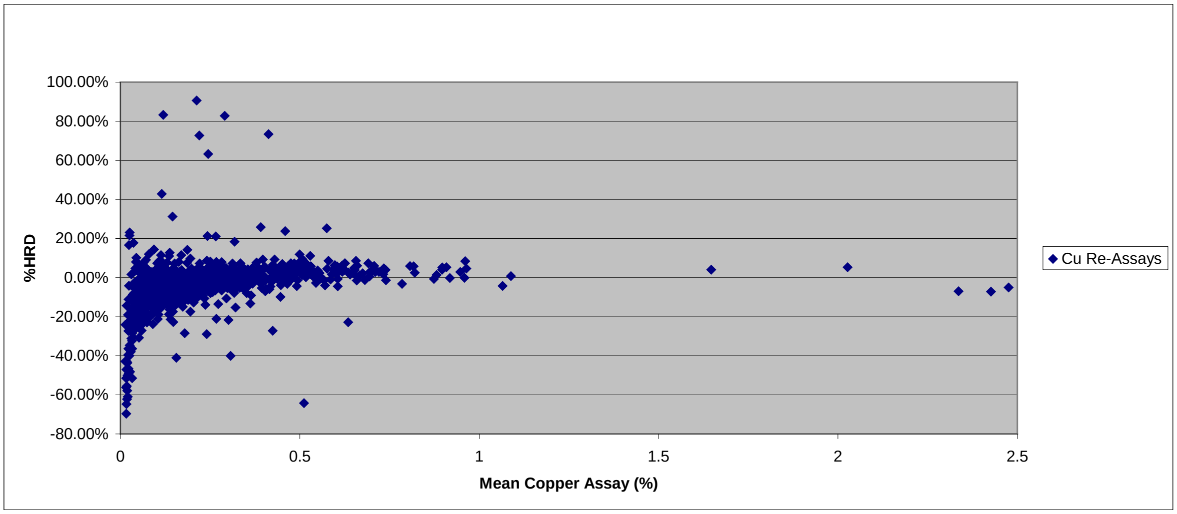
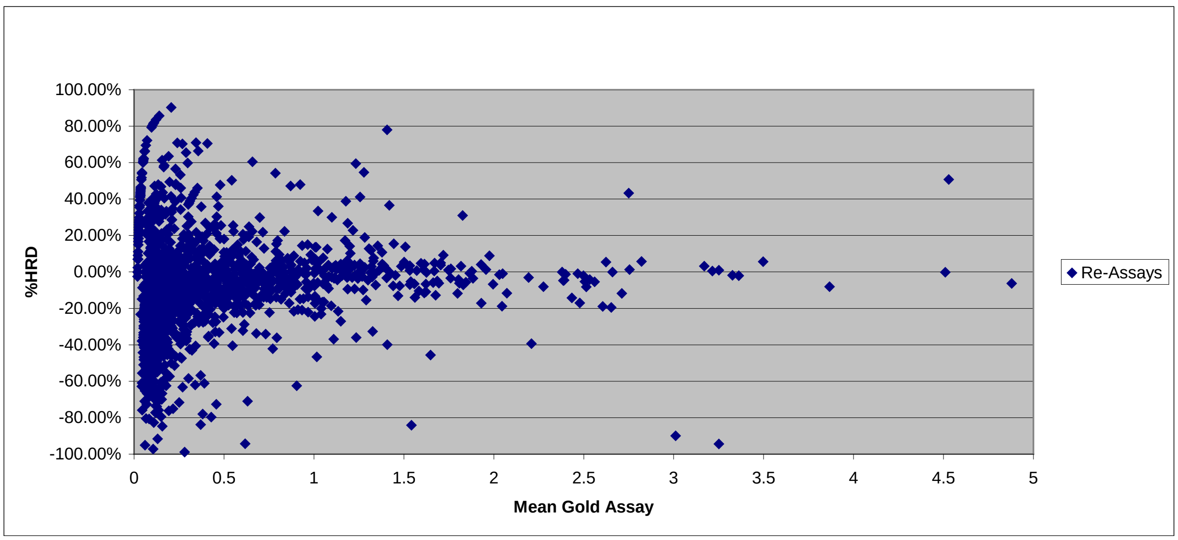


Figure 12-2: Echo Bay Re-Assays of Benguet Samples Total Copper – Logs Base 10



Description	No. of Samples	Benguet Cu (%)	EB Cu (%)	% Diff	Precision (%HARD)	Bias (%HRD)
All Samples	1171	0.241	0.240	-0.32%	7.18%	-3.48%
Avg Cu >=0.05%	1082	0.258	0.258	0.06%	5.70%	-1.89%

Figure 12-3: %Half Rel Deviation vs Mean – Echo Bay Re-Assays of Benguet Copper



Description	No. of Samples	Benguet Au (g/t)	EB Au (g/t)	% Diff	Precision (%HARD)	Bias (%HRD)
All Assays	1493	0.501	0.476	-5.00%	20.59%	-7.34%
Avg Gold >= 0.2 g/t	887	0.762	0.739	-3.08%	14.41%	-4.01%

Figure 12-4: %Half Rel Deviation vs Mean – Echo Bay Re-Assays of Benguet Gold

```
* REGRESSION ANALYSIS              LINEAR REGRESSION
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  OPTIONS: through origin ,biweight
                 total    non-
  variable       cases   missing      mean         std dev       minimum        maximum
  ebra_au         1485      1485   .45447E+00   .54499E+00   .11000E-01   .45000E+01
  au              1485      1485   .48949E+00   .55810E+00   .20000E-01   .45200E+01
  standard error of estimate        .0060       correlation coefficient       .9633
  ebra_au   =       .914260 * au
```

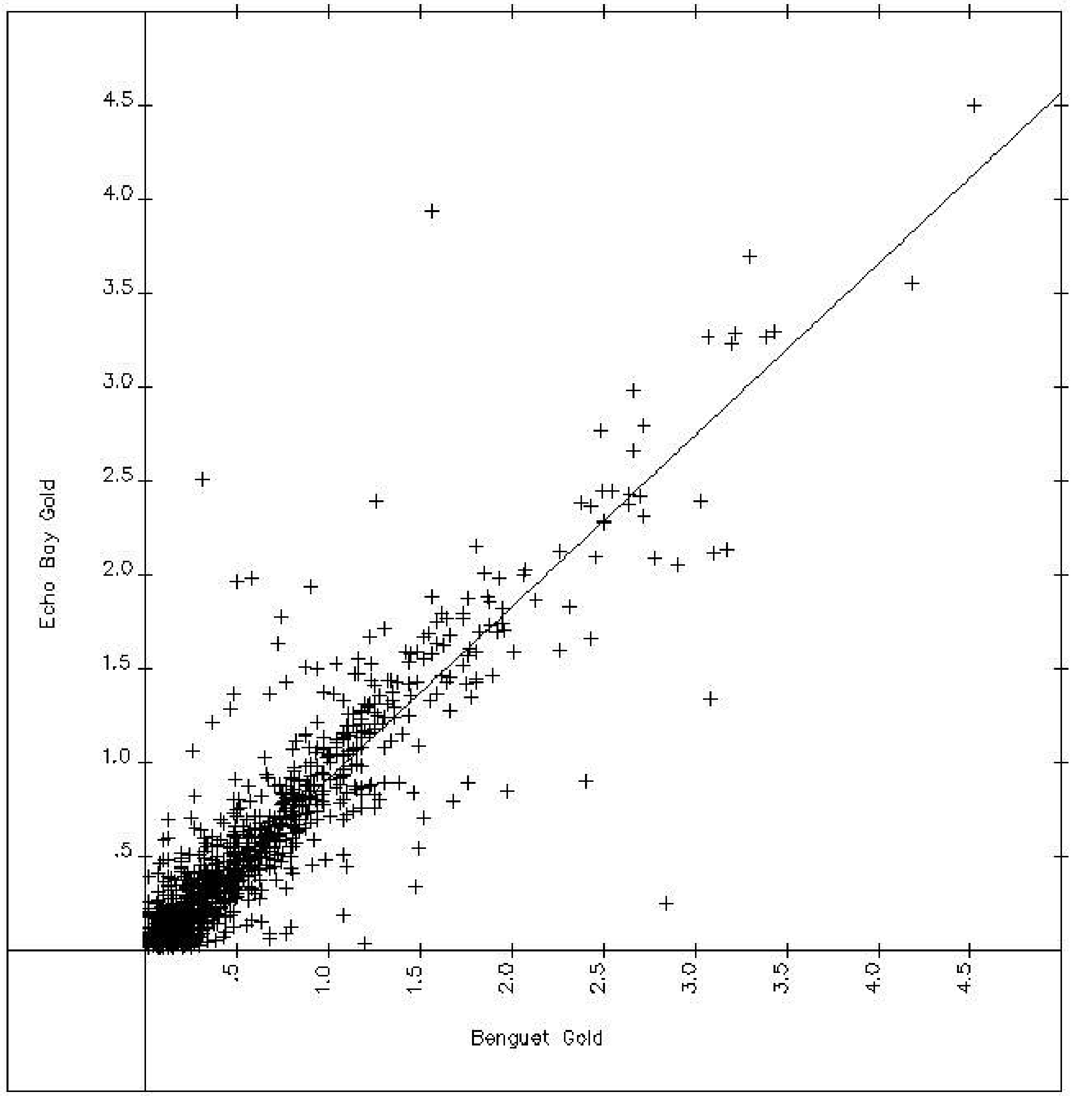


Figure 12-5: Echo Bay Re-Assays of Benguet Samples Gold

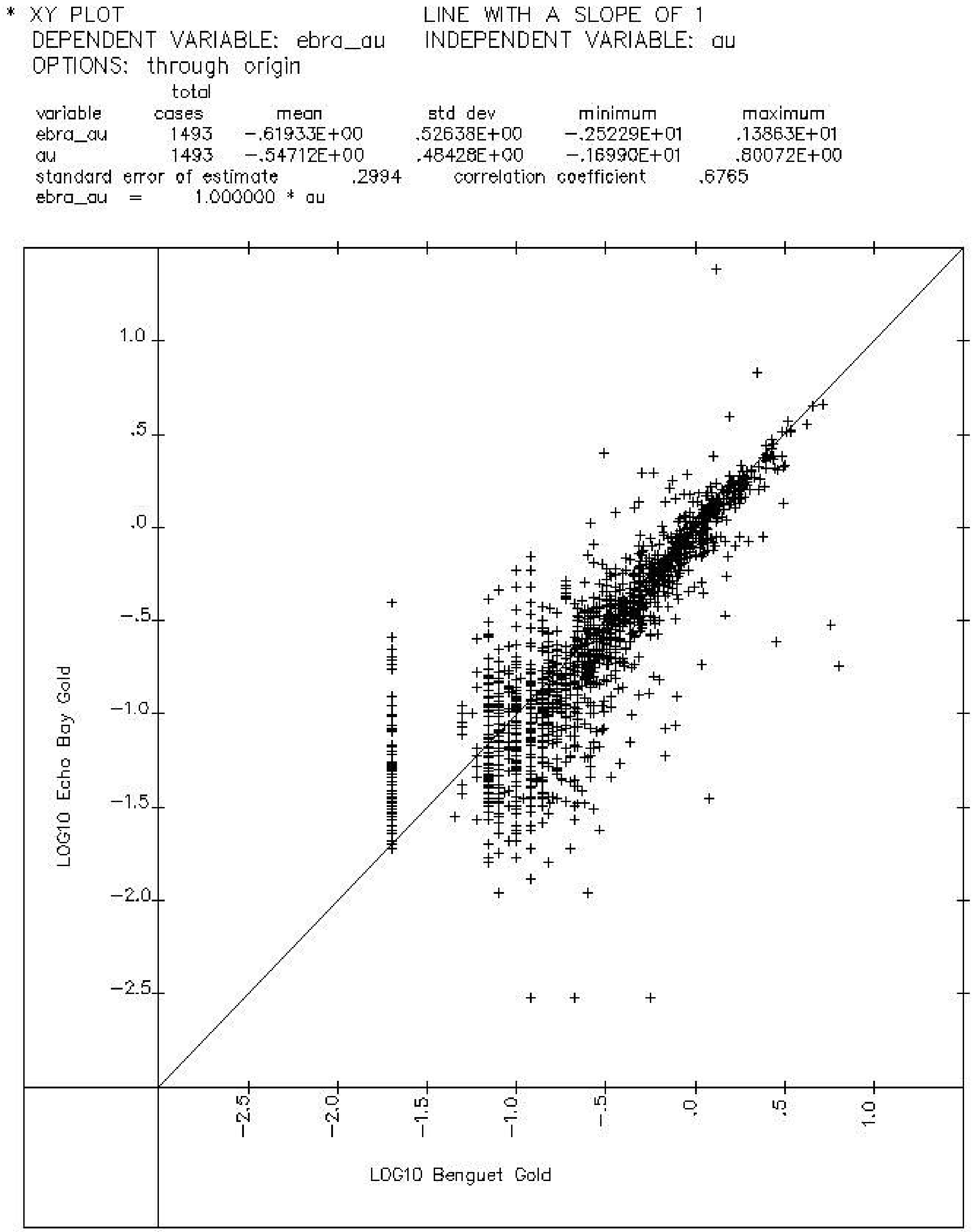


Figure 12-6: Echo Bay Re-Assays of Benguet Samples Gold – Logs Base 10

```
* XY PLOT                                  LINE WITH A SLOPE OF 1
  DEPENDENT VARIABLE: ebra_scu             INDEPENDENT VARIABLE: scu
  OPTIONS: through origin
              total
  variable    cases      mean          std dev        minimum        maximum
  ebra_scu    120      .38082E+00    .45658E+00     .23000E-01     .25290E+01
  scu         120      .27706E+00    .41028E+00     .10000E-01     .22400E+01
  standard error of estimate    .1404     correlation coefficient    .9054
  ebra_scu =      1.000000 * scu
```

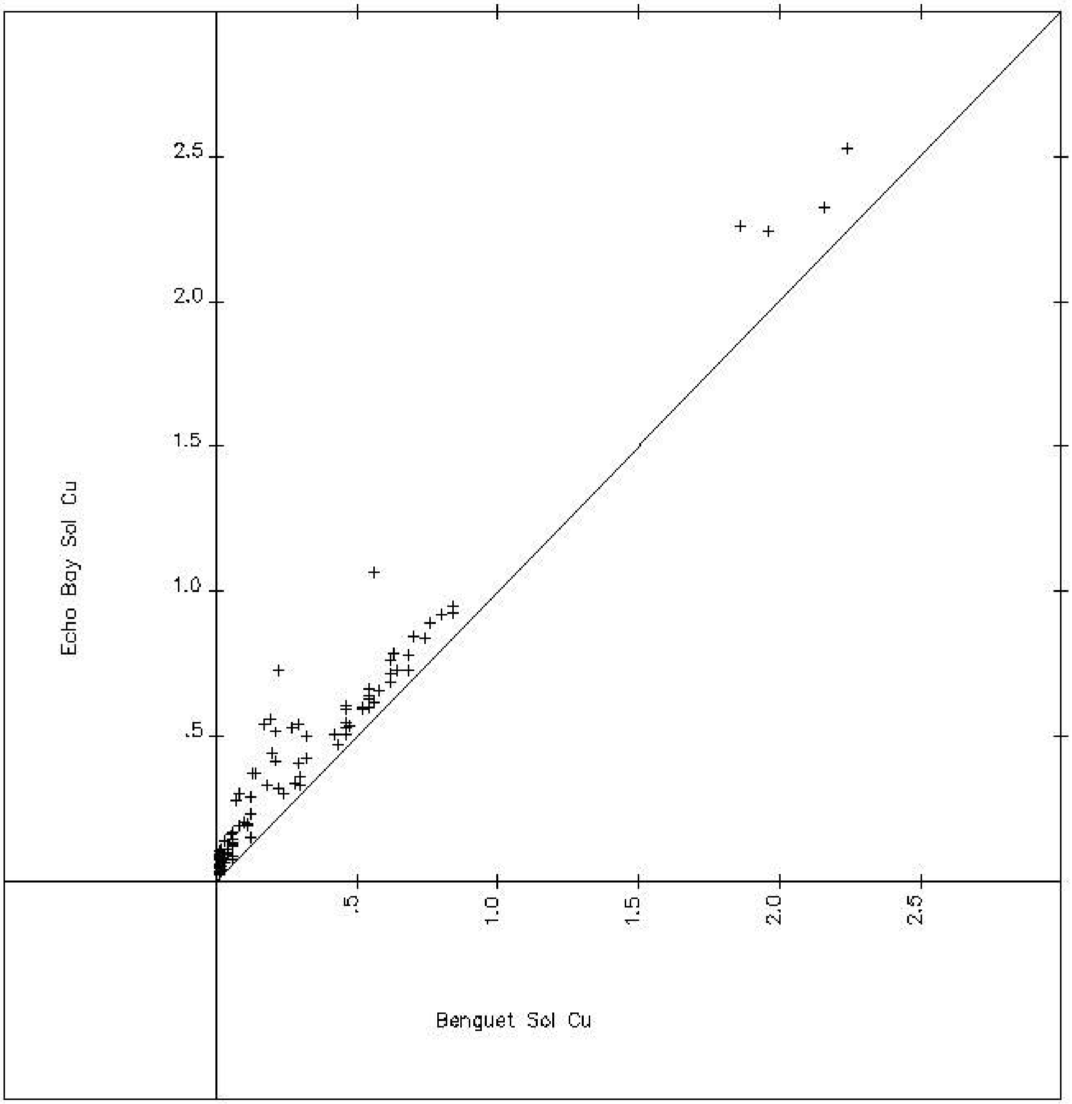


Figure 12-7: Echo Bay Re-Assay of Benguet Samples Soluble Copper

12.3 RMMI CHECK ASSAYS

D. Earnest (REI) collected a suite of 100 diamond drilled, 3-meter intervals on June 5-7, 2010. The samples were obtained from material remaining in the King-king Pantukan core shack, and were considered representative of variations in lithology and grade for samples of the original drilling programs. The intent was to verify the original assay results. The samples included 68 samples from original Echo Bay core and 32 samples from original Benguet core. Analyses were conducted at Independent Australian Labs (IAL).

12.3.1 Total Copper

Figure 12-8 shows an xy plot of original total copper assays on the x axis and the RMMI check assay on the y axis. Original Benguet and Echo Bay samples are distinguished on the graph. Most of the samples cluster relatively closely to the 1:1 line plotted on the graph, though there are five to six significant outliers, and for all of them the check assay was significantly lower than the original assay.

Figure 12-9 shows the plot of the mean total copper grade (mean of the original assay and check assay for each pair) on the x axis and %HRD (Half Relative Deviation) on the y axis. Section 12.2.2 defined the terminology used.

Table 12-1 shows the relative statistics for the comparison. For all samples, the original copper assay averaged 0.468% copper versus 0.424% for the check assay. This is approximately a 9.5% difference in the means. Results are similar for Echo Bay and Benguet samples with the check assays being 9.7% lower than Echo Bay original samples and 9.4% lower than Benguet original samples. Precision estimates by the HARD calculation are approximately 10%, meaning that any one assay is expected to be within +/-10% of the true value. Bias estimates from the %HRD calculation method are -4.9% for all data (original assay > check assay), -4.5% for Echo Bay original samples and -5.9% for Benguet original samples.

These results are not as favorable as those obtained by Echo Bay with their program to re-assay Benguet samples, though assay results were similar for the majority of the 100 samples. The Echo Bay re-assay program showed a lower bias and better precision than the RMMI check assay program. The results may partly be explained by degradation of the samples over time, or possibly that 100 samples do not represent a large enough population.

12.3.2 Gold

Figure 12-10 shows an xy plot of original gold assays on the x axis and the RMMI check assay on the y axis. Original Benguet and Echo Bay samples are distinguished on the graph. Most of the samples cluster reasonably close to the 1:1 line plotted on the graph, though there are a few significant outliers. Note there is one Echo Bay sample with an original assay of 14.3 g/t and an RMMI check assay of 6.8 g/t that is not shown. Note also that this single assay can significantly distort mean value calculations with only 100 samples available.

Figure 12-11 shows the plot of the mean gold grade on the x axis and %HRD (Half Relative Deviation) on the y axis. Table 12-2 shows the relative statistics for the comparison. For all samples the original gold assay averaged 0.962 g/t versus 0.801 g/t gold for the check assay. This is about a 16.7% difference in the means. Also, for all data, the precision estimate is 23.0% (any one assay is expected to be within +/-23% of the true value) and the bias estimated by the %HRD calculation is -13.4% (original assay > RMMI check assay).

The table also shows significantly different results for Echo Bay and Benguet original samples. For all Echo Bay samples the precision estimate is 19.4%, and goes to 15.2% when samples less than 0.12 g/t gold and the outlier at 14.3 g/t are excluded. The bias for Echo Bay samples is 10.7% (Echo Bay > RMMI) for all samples and goes to a very reasonable -5.5% when the low grade and outlier are excluded. For original Benguet samples the precision estimate is 30.7% and the bias -19.0 (Benguet assay > RMMI assay). Truncating a few low grade samples has minimal impact on the results. As with the Echo Bay re-assay program, the RMMI assays indicate the original Benguet gold assays are biased high.

12.4 CONCLUSIONS AND RECOMMENDATIONS

The results of the comparison of assays in the database to assay certificates indicate that the Echo Bay data were not as good as expected. However, based on the IMC checks, and subsequent corrections, IMC is of the opinion that the database now correctly reflects original assay results to an acceptable level of accuracy for the current resource estimation.

The Benguet total copper assays and Echo Bay re-assays compare well and indicate good assay precision for total copper. Based on this, the Benguet total copper assays are acceptable for resource calculation.

The Benguet gold assays are biased high compared with the Echo Bay assays, and also the RMMI check assays, and will not be used for the current resource model. However, they will be replaced with Echo Bay re-assays when available.

The RMMI check assay program was successful in that it broadly validated previous Benguet and Echo Bay copper assays and Echo Bay gold assays. On average, the check assays tended to be lower than the original assays. This can partially be explained by a few outliers since 100 samples is not a particularly large sample population. It is also possible that there has been some degradation of the samples over time.

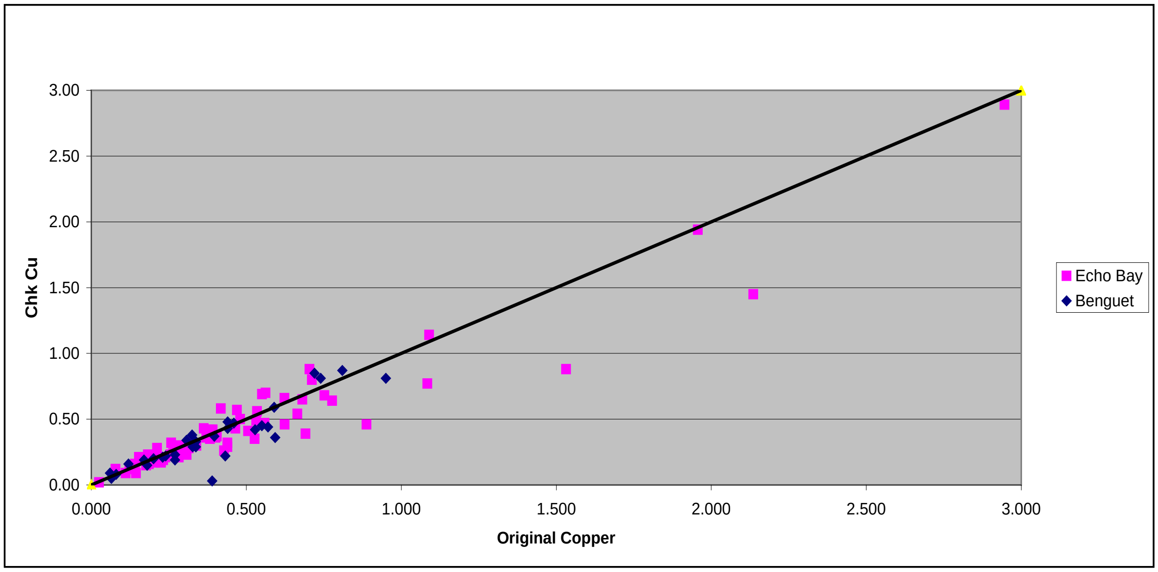


Figure 12-8: Total Copper- RMMI Check Assays vs Original Assays

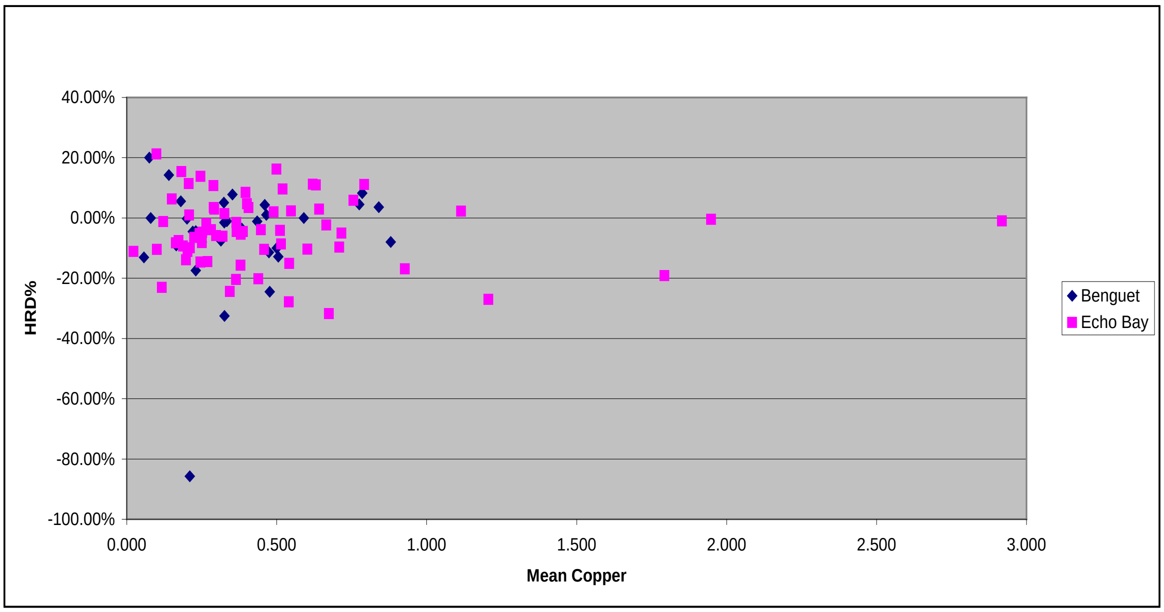


Figure 12-9: HRD% vs Mean Copper Grade for RMMI Check Assays

Table 12-1: RMMI Check Assays vs Original Assays – Total Copper

Description	No. of Samples	Original Cu (%)	Check Cu (%)	% Diff	Precision (%HARD)	Bias (%HRD)
All Data	100	0.468	0.424	-9.54%	9.98%	-4.92%
Echo Bay Data	68	0.505	0.456	-9.68%	9.72%	-4.47%
Echo Bay Data 0.2% < Original Cu	58	0.570	0.512	-10.15%	9.40%	-5.11%
Echo Bay Data Original Cu < 1.5%	64	0.403	0.373	-7.43%	9.59%	-4.01%
Benguet Data	32	0.389	0.354	-9.14%	10.53%	-5.87%
Benguet Data 0.2% < Original Cu	26	0.453	0.408	-10.05%	10.57%	-7.90%

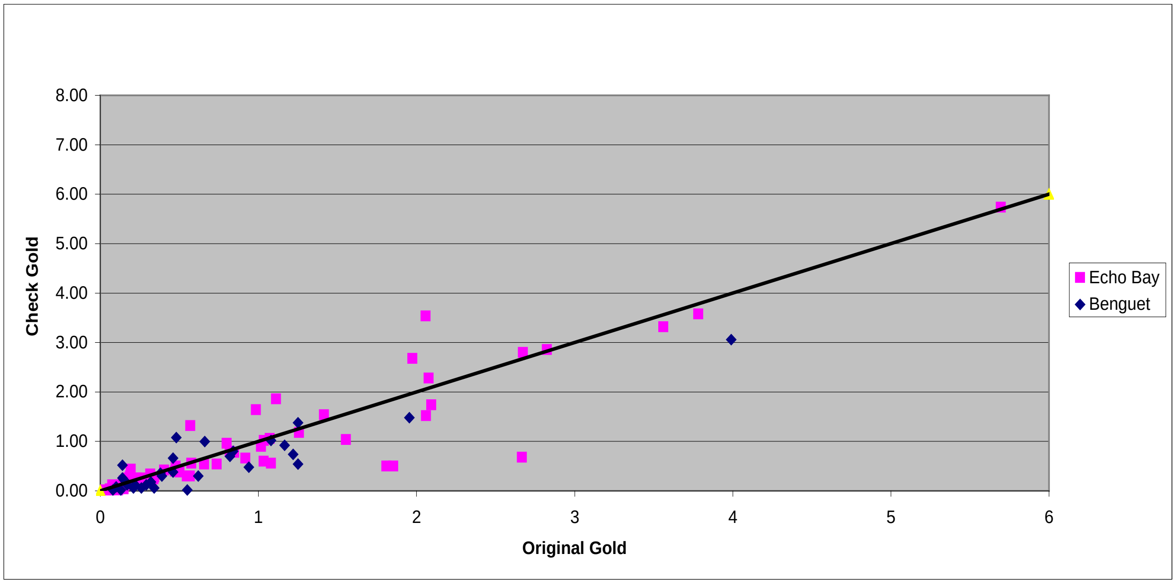


Figure 12-10: Gold- RMMI Check Assays vs Original Assays

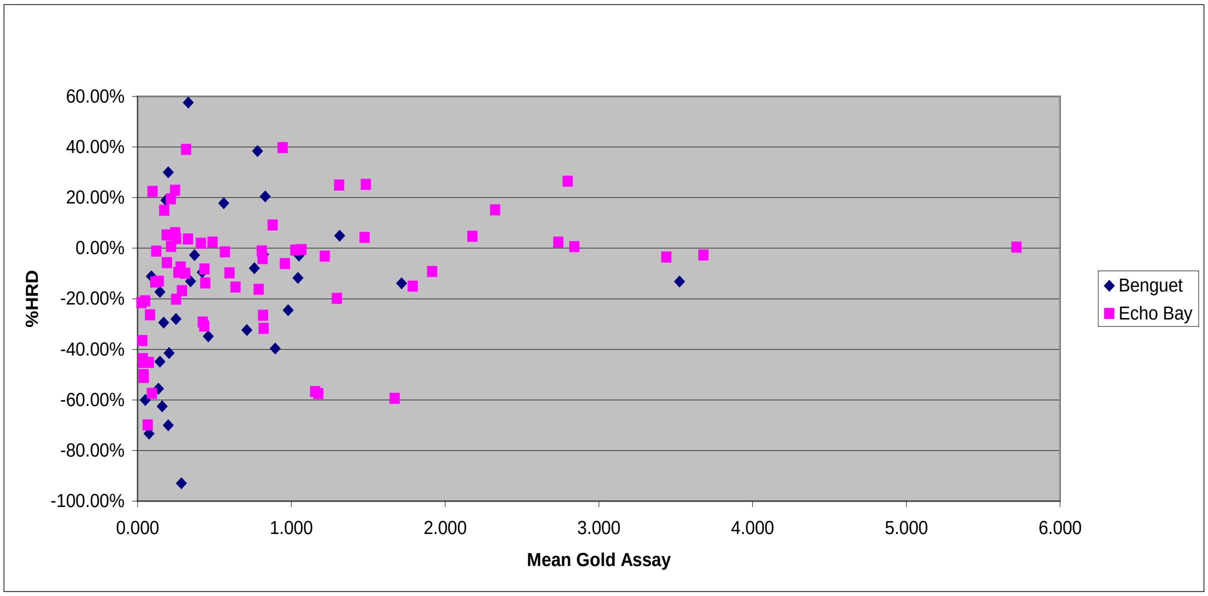


Figure 12-11: HRD% vs Mean Gold Grade for RMMI Check Assays

Table 12-2: RMMI Check Assays vs Original Assays - Gold

Description	No. of Samples	Original Au (g/t)	Check Au (g/t)	% Diff	Precision (%HARD)	Bias (%HRD)
All Data	100	0.962	0.801	-16.68%	23.04%	-13.37%
Echo Bay Data	68	1.101	0.926	-15.89%	19.41%	-10.73%
Echo Bay Data 0.12 < Original Au < 10	56	1.068	0.997	-6.62%	15.20%	-5.46%
Benguet Data	32	0.665	0.535	-19.45%	30.74%	-18.98%
Benguet Data 0.135 < Original Au	29	0.723	0.587	-18.83%	28.93%	-15.96%

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Previous metallurgical testwork on King-king ores was carried out by Metcon Research Inc. (Tucson, AZ) in 1993 for Benguet Corp. and by Lakefield Research Inc. (Lakefield, ON) in 1997 for Echo Bay Mines.

The Metcon work was done to develop parameters at preliminary feasibility level. Two phases of column leaching tests were performed on upper (oxide) ore. About 90 % of the copper from this ore was found to be soluble in acid. Flotation tests were performed on two ore types, mixed and sulfide, to establish basic grinding and flotation parameters and to evaluate final concentrate grade and recovery for each type. Locked-cycle tests yielded 79% recovery of copper and 81% recovery of gold from the mixed ore, and 93% recovery of copper and 91% recovery of gold from the sulfide ore.

The Lakefield work was done to develop a flowsheet and reagent scheme at feasibility level. The process for sulfide ore consisted of grinding, flotation of copper and some of the pyrite, regrinding, copper-pyrite separation and upgrading. The presence of clay in the upper sulfide ore resulted in lower copper recoveries and loss of gold in the cleaner circuit. The process for oxide ore consisted of grinding, magnetic separation, and leaching, followed by a leach-precipitation-float (LPF) step. Recoveries reported were 90% for copper and 70% for gold. The leach solution was determined to be amenable for recovery of copper by solvent extraction.

Following a preliminary assessment of historical King-king metallurgical tests, AMEC Australia conducted a series of comprehensive tests to aid understanding of the individual domain characteristics in order to assess variability of metallurgical performance, and to show the impact of the current mine plan and production composite samples on commercial operation.

The metallurgical test work program consisted of a series of comminution and flotation tests, thickener sizing tests, a gold deportment tests on oxide dominant ore, and leaching tests on flotation tailing. Leach, Inc. of Tucson, AZ performed heap leaching tests on copper oxide dominant samples. The sections that follow summarize the details and results of the metallurgical testing program.

13.1 SAMPLE SELECTION

13.1.1 Comminution and Flotation

Twenty-six core samples were tested to measure SAG Mill Comminution (SMC) parameters, Bond abrasion indices (Ai), Bond ball mill work indices (BWi) and Bond rod mill work indices (RWi).

Core samples for flotation testing were selected to represent the ore body based on the resource and mining plan (dated 20 November 2010). Half core and pre-crushed core materials were supplied for the testing program. From this material, the following samples and composites were prepared:

- 2 ore type composites: Oxide and Sulfide

- 38 'variability' sample taken from individual drill core intervals selected to represent the orebody
- 4 'Life of Mine' (LOM) composites: Yr 2/3, Yr 4/5, Yr 6/10 and "Remaining Life" (RL)

The head samples were assayed for sequential Cu, Au, S and Fe. The sequential copper determination techniques were (1) Deionised Water Dissolution (DWD), (2) Acetic Acid Dissolution (AAD), (3) Weak Acid Dissolution (WAS), (4) Cyanide Dissolution (CNS) and (5) Strong Acid Dissolution (AqRS). Gold was analysed by fire assaying. Samples were classified as sulfide or oxide ore types based on the sequential copper results. Oxide types were those where the acid soluble copper content exceeded 35% of the total copper present.

Initially, sulfidization was tested to investigate the use of NaHS for improving the flotation performance of the oxide composite sample. The results were not encouraging hence it was decided to attempt sequential flotation of sulfide and oxide copper in two stages. The first stage used potassium amyl xanthate (PAX) as collector to float sulfides and the second stage tested several collectors, known as hydroxymates, to float oxides. This scheme is identified in this report as the 'Phase 1 Oxide' scheme in reference to the variability flotation testing, and the 'LCA' scheme in reference to the Locked Cycle testing. It resulted in high consumption of the oxide collector and high mass pulls (ie: low concentrate grades), and was replaced with the 'Phase 2 Oxide' or 'LCB' scheme which used only PAX to collect floatable sulfides in oxide ore, leaving recovery of the oxides for later stage leaching of the tails. A third scheme, the 'Sulfide' scheme, used sodium isobutyl xanthate (SIBX) as the collector when treating sulfide ore.

The two ore type composites (Oxide and Sulfide) and the four Life of Mine composites were subjected to 6 cycle locked cycle testing. The Oxide and Yr 2/3 (LOM) were floated using the oxide float scheme (LCA) employing PAX as the collector, and the Sulfide, Yr 4/5, Yr 6/10 and RL were floated using a sulfide float scheme employing SIBX as the collector. The Oxide and Yr 2/3 (LOM) composites were also floated with an alternative oxide scheme (LCB) that had longer collection times and produced better recoveries but with lower copper and gold grades.

13.1.2 Tails Leach Option

Tails leach testing was conducted on tails from two stages of flotation test work. The Stage I flotation work tested a sequential sulfide-oxide float. This was dropped due to high reagent consumptions and poor concentrate grades. The Stage II flotation work tested a sulfide only float, where the recovery of any soluble oxides present was left to downstream leaching of the flotation tails. Twelve (12) variability samples from the Stage I and Stage II flotation tests were used for the tails leach test work. Five (5) samples were obtained directly as tails from the Stage II sulfide variability flotation tests. Due to a shortage of variability sample quantity, the remaining seven (7) samples were obtained by combining Stage I oxide rougher concentrate and oxide rougher tails from oxide variability flotation tests.

13.1.3 Heap Leaching of Oxide Ore

A group of 25 samples were used for column leach testing. They were selected based on the mine plan and the drill core assays. Because of sample availability, the samples do not necessarily appear in the test program in proportion to the tonnage they represent in the deposit. Therefore, any averages from the column leach test results do not necessarily represent the average that can be expected in the proposed heap leach operation.

13.2 COMMINUTION TESTWORK

The results of the comminution testwork indicated that the King-king rock mineralization exhibits variable rock competency and ball mill grindability. Variation in material competency between the two major ore domains, oxide and sulfide, was observed. There is also significant variation for the samples within each domain. Sulfide samples had the lowest Axb and highest Bond ball mill work indices (BWi). Oxide samples, scheduled for early processing, were the least competent, exhibiting higher Axb and lower Bond ball mill work indices.

The range and average of each parameter obtained from the King-king testwork are summarized in Table 13-1.

Table 13-1: Comminution Testwork Results for Oxide and Sulfide Samples

Property	Unit	No. of Oxide Samples	Oxide Samples Range	Oxide Samples Average	No. of Sulfide Samples	Sulfide Samples Range	Sulfide Samples Average
Axb		10	36.9 – 124.2	67.0	16	25.0 – 56.7	38.0
Bond Abrasion Index (Ai)	g	10	0.03 - 0.16	0.07	16	0.03 - 0.31	0.16
Bond Ball Mill Index (BWi)	kWh/t	10	6.7 - 14.6	10.9	16	9.9 – 20.7	14.2
Bond Rod Mill Index (RWi)	kWh/t	10	7.8 – 14.3	11.5	16	10.3 – 21..9	14.7

13.3 FLOTATION TESTWORK

All flotation test work utilized both sulfide and oxide composite samples. The overall objective of the testing was to determine the preferred processing route and optimal operating conditions.

The flotation test work program involved determining the optimum flotation conditions for both sulfide and oxide composites through batch rougher flotation testing.

The oxide and sulfide composite flotation program included tests to establish:

- the optimal grind size
- the effects of varying pulp potential
- collector dosage for bulk flotation and sequential flotation (oxide composite only)
- the optimal conditions for the cleaner stages

- steady state final concentrate grades and recoveries for copper and gold; and
- the variability effects of individual samples on: primary grind, rougher lime consumption, and the overall grade and recovery achieved.

13.3.1 Oxide Composite Flotation

Details of the oxide composite flotation tests are not to be included here. They can be obtained from the AMEC Australia report "King-King Copper-Gold Project Metallurgical Testwork Report" (AMEC document no. 65007-00000-21-002-005).

13.3.2 Sulfide Composite Flotation

The details and results of the sulfide composite flotation program are summarized in the sections that follow.

13.3.2.1 Grind Size

The grind size optimization flotation tests were conducted at P_{80} values of 150 µm, 106 µm, 75 µm and 53 µm to determine the optimum primary grind size for sulfide composite samples. In these experiments A3302 and MIBC were used as collector and frother respectively. The following observations were made:

- The recovery of total copper increased from 81% to 85% as grind size decreased from 150 µm to 106 µm but further grinding to a P_{80} of 53 µm did not have a significant effect on the recovery of total copper.
- The recovered mass increased almost 8% as grind size decreased from 150 µm to 53 µm.
- The cumulative grade of concentrate decreased considerably as grind size decreased.

13.3.2.2 Collector Type

This series of tests employed the optimal grind size determined above to evaluate the performance of four collectors: PAX, SIBX, A404 and A3302, all dosed at 40 g/t.

The results demonstrated that the PAX and SIBX collectors yielded the highest total copper, gold and mass recoveries.

13.3.2.3 Collector Dosage

As determined in the previous set of experiments, SIBX performed best in terms of copper and gold recovery, thus SIBX was selected as the optimum sulfide collector for subsequent experiments.

The aim of this set of experiments was to determine the optimal SIBX dosage. Four different collector dosages were tested: 40 g/t, 30 g/t, 20 g/t and 10 g/t.

It was observed that as collector dosage increased from 10 g/t to 20 g/t, the recovery of total copper increased from 83% to 89%. Thus the optimum dosage rate of 20 g/t was employed in all subsequent sulfide test work. It was also noted that varying collector dosage had insignificant effects on total mass recovered.

13.3.2.4 pH

Various pH values were tested to investigate the effect of pH on the flotation performance of the King-king sulfide dominant composite. The pH values tested were natural (tap water), 9, 10 and 11.

The results indicated that increasing pH from natural to 9 produced a significant rise in the recovery of total copper from 58.8% to 87.1%, and in the recovery of gold from 60.2% to 82.6%. However, increasing the pH to 10 or 11 contributed minimally to the recovery of gold and copper. Thus the best pH for the sulfide dominant composites is 9.

13.3.2.5 Cleaner Stage Optimization

This set of tests utilized the optimum flotation conditions established so far while varying certain conditions to test their impact on the performance of the cleaning and re-cleaning stages. The variables investigated were cleaner flotation pH, concentrate regrind particle size, collector addition, addition of a cleaner scavenger stage, and addition of a re-cleaner stage.

<u>Cleaner pH</u>: Three experiments with different pH (9, 10, and 11) values in the cleaning stage were performed to evaluate the effects of pH on the final cleaner concentrate grade. No improvement in the grade of the final copper concentrate was observed by changing the pH.

<u>Concentrate Regrind</u>: Two tests were performed to investigate the effect of re-grind size (P_{80} of 20 µm and P_{100} of 20 µm) on overall flotation performance of the King-king sulfide dominant ore. The results indicate that a finer re-grind improves the cleaner concentrate copper grade but at a lower copper recovery.

<u>Collector Dosage to Cleaner</u>: An additional cleaner test was performed to investigate the effect of collector addition in the cleaner stage on flotation performance of the King-king sulfide dominant composite. The test was performed without collector, unlike all the previous tests which were conducted with a collector dosage of 5 g/t. When compared to the re-grind test (re-grind size of P_{100} of 20 µm) performed with the addition of collector to the cleaner stage conditioner, this test indicated that the addition of collector to the conditioner has negligible effect on the grade of the concentrate.

<u>The Addition of Cleaner Scavenger Flotation Stage</u>: A test to investigate the effect of adding a single cleaner scavenger stage on recovery was conducted on the King-king sulfide dominant composite. This single experiment indicated further recovery of gold and copper can be achieved by adding a cleaner scavenger flotation stage. However, the grade of the scavenger concentrate was observed to be low.

The Addition of a Re-Cleaning Stage: It was observed that substantially higher copper and gold grades are attainable with the addition of a re-cleaning stage to the flotation process. The addition of two-stage re-cleaner to the flotation process was also examined to identify further potential improvements in copper and gold grades. Small improvements in the copper and gold grades were observed.

13.3.3 Summary of Oxide and Sulfide Flotation Testwork Results

The oxide and sulfide composite flotation test work collectively produced the optimum flotation conditions. These are summarized in Table 13-2.

Table 13-2: Composite Optimum Conditions

Stage	*Collector Dosage (g/t)	Grind Size (µm)	pH
Grinding		P_{80} of 106	9
Conditioning			
Rougher Stage	25		9
Re-Grinding		P_{100} of 20	
Cleaning Stage	5		9
Cleaning Scavenger	1		
Re-Cleaning 1	5		
Re-Cleaning 2	2		

***PAX for oxide and SIBX for sulfide**

These conditions formed the basis for the locked cycle test work.

13.3.4 Locked Cycle Flotation Tests

Six-cycle locked cycle flotation tests were conducted on the following composites:

- Oxide dominant
- Sulfide dominant
- Year 2/3 Life of Mine
- Year 4/5 Life of Mine
- Year 6/10 Life of Mine

Each locked cycle test was conducted under the optimal conditions determined from the previous flotation testwork. The test scheme included a rougher stage, regrind of the rougher concentrate to 20 µm, three cleaning stages, and a cleaner scavenger stage to treat the first cleaner stage tailing. Tailing from the rougher stage and the cleaner scavenger stage together formed the final flotation tailing. Cleaner scavenger concentrate was recycled to the first cleaner feed. Tailing from the 2nd and 3rd cleaner stages were returned to their corresponding previous stages. The locked cycles tests were run for six cycles.

The results of the locked cycle test are summarized in Table 13-3.

Table 13-3: Steady State Results of Locked Cycle Testwork

Composite	3rd Stage Conc Grade		3rd Stage Conc Recovery	
	Copper (%)	Gold (ppm)	Copper (%)	Gold (%)
Oxide	23	61	25	53
Sulfide	25	39	71	51

Final concentrates from the last two cycles of each locked cycle test were assayed to produce detailed elemental/species analyses. These were used to estimate concentrate quality, penalties (for various impurities) which may be incurred, and saleability of the product.

From these analyses, it was concluded that arsenic may be a penalty concern. The arsenic levels in the life of mine composites reached a high of 3,700 ppm. The concentration of aluminium and fluorine in the Yr 2/3 LOM composites were above the expected penalty limit. Antimony and selenium concentrations also exceeded the expected penalty limits in concentrates produced from the sulfide Yr 4/5 LOM and Yr 6/10 LOM composites.

Gold represented a payable by-product in all concentrates achieving a grade over 2 g/t.

13.3.5 Flotation Variability Tests

Flotation roughing tests were conducted to estimate the impact of the variability of King-king samples on grind size, rougher lime consumption, grade, and recovery. The conditions utilized in these tests were the optimum primary grinding and rougher flotation conditions established in the optimization tests.

Oxide samples were subject to Phase 1 Oxide scheme sequential sulfide/oxide flotation and Phase 2 Oxide scheme sulfide flotation. Sulfide samples were subject to sulfide scheme sulfide flotation (see Section 13.1.1 for an explanation of these terms).

13.3.5.1 Variation in Primary Grind

For these tests individual oxide and sulfide samples were ground for the time determined in previous grind tests to achieve the target value of P_{80} 106 µm grind size. The results are summarized in Table 13-4.

Table 13-4: Flotation Variability Test -Variation in Primary Grind Results

Sample Type	Grind Size Range (µm)	Average (µm)
Oxide	48-130	90
Sulfide	52-176	103

The average grind sizes achieved for individual oxide and sulfide samples were P_{80} of 90 µm and 103 µm respectively. This compares well with target value of P_{80} of 106 µm.

13.3.5.2 Variation in Rougher Lime Consumption

Table 13-5: Flotation Variability Test -Variation in Rougher Lime Consumption

Sample Type	Scheme	Lime Consumption (kg/t)	Average (kg/t)
Oxide	Phase 1 Oxide Sequential	1.16-1.25	1.6
Sulfide	Phase 1 Oxide Sequential	0.5-3.5	1.7
Oxide	Phase 2 Oxide	0.7-1.25	0.95
Sulfide	Sulfide	0.5-3.5	1.0

The average lime consumptions in the sequential flotation were higher than in the bulk flotation. This was expected due to the higher operating pH.

13.3.5.3 Variability of Phase 1 Oxide (Sequential) Flotation Recovery

The results show high variability in rougher recovery of copper for oxide samples (21.7% - 84.5%) relative to sulfide samples (66.1% - 96.6%).

It was also noted that the variability in rougher recovery of gold was moderate for both oxide and sulfide samples. For oxide samples, the gold recovery ranged from 57% to 87% and had an average gold recovery of 74.4%. For sulfide samples, the gold recovery ranged from 72.5% to 97.6% and had an average of 85%.

Table 13-6: Phase 1 Oxide (Sequential) Flotation Individual Grade & Recovery Variability.

Sample Type	Overall Recovery (Cu%)		Overall Recovery (Au%)		Rougher Concentrate Grade (Cu%)		Rougher Concentrate Grade (Au ppm)	
	Range	Average	Range	Average	Range	Average	Range	Average
Oxide	21.7-84.5	21.7	57-87	74.4	0.3-1.8	1.2	0.1-6.2	2
Sulfide	66.1-96.6	81.8	72.5-97.6	85	0.1-6.1	1.8	0.1-12.3	5

13.3.5.4 Variability of Phase 2 Oxide Flotation Recovery

The same observations were made from the Phase 2 Oxide flotation test work – oxide samples (7.8% - 82.2%) displayed very high variability in overall recovery of copper relative to sulfide samples (78.3% - 95.1%).

The overall variability of gold recovery was very high for oxide and sulfide samples, ranging between 14 and 88.4% for oxide and 34 and 87% for sulfide.

Table 13-7: Phase 2 Oxide Flotation Individual Grade and Recovery Variability Results

Sample Type	Overall Recovery (Cu%)		Overall Recovery (Au%)		Rougher Concentrate Grade (Cu%)		Rougher Concentrate Grade (Au ppm)	
	Range	Average	Range	Average	Range	Average	Range	Average
Oxide	7.8-82.2	26.7	14-88.4	51.1	0.4-8.3	2.5	0.5-60	14.4
Sulfide	78.3-95.1	85.2	34-87	70.8	1.1-10	3.3	0.4-10.2	3.1

13.3.5.5 Variability of Sulfide Flotation Recovery

The variability in the recovery of total copper was low for the sulfide samples, ranging from 77%-90% with an average of 83%. The range in total gold recovery was significantly higher, ranging from 60%-92% and with an average recovery of 81%.

Table 13-8 Sulfide Flotation Individual Grade and Recovery Variability Results

Sample Type	Overall Recovery (Cu%)		Overall Recovery (Au%)		Rougher Concentrate Grade (Cu%)		Rougher Concentrate Grade (Au ppm)	
	Range	Average	Range	Average	Range	Average	Range	Average
Sulfide	77-90	83	60-92	81	0.5-7	2	0.3-10.8	3

13.4 GOLD DEPORTMENT

A 2 kg charge of King-king oxide composite was analyzed for gold deportment. The sample was ground to a P_{80} of 106 µm then the -106 µm fraction was washed through 75-µm and 38-µm screens. The -38 µm fraction was classed as undifferentiated and set aside, while the +38 µm size fractions were analyzed for gold deportment as follows:

- mercury soluble (liberated) gold: a portion of each size fraction (+106µm, -106µm+75µm, -75µm+38µm) was analyzed for gold in order to determine total liberated and locked gold.
- mercury insoluble residues were separated in heavy liquid at s.g. = 3.32. The <3.32 s.g. component was analyzed for gold as gold locked in silicates or carbonates.
- the >3.32 s.g. component was passed over a Franz magnetic separator to separate the magnetics and non-magnetics. Each portion was then analyzed for gold. The gold in magnetics was classed as gold locked in iron oxides, and the gold not in magnetics was classed as gold locked in sulfides.

The -38µm fraction was analyzed for gold. This undifferentiated fraction accounted for 83.92% of all gold in the sample. The remaining 16.08% was distributed as summarized in Table 13-9.

Table 13-9: Gold Deportment Summary

Species	Gold Association			
	+106 µm	-106+75 µm	-75+38 µm	Total
Sulfides	4.48%	6.97%	11.33%	22.78%
Iron Oxide	0.56%	1.18%	1.87%	3.61%
Silicates	14.37%	15.99%	13.69%	44.06%
Liberated	1.74%	4.73%	23.09%	29.56%

The following inferences were made from the results:

- The majority (44%) of the differentiated gold in the oxide sample is associated with silicates and would be non-recoverable through either flotation or gravity recovery
- 23% of the differentiated gold is associated with sulfides and is recoverable through flotation
- The percentage of gold association increases with decreasing particle size
- The liberated gold is predominantly within the smaller size fraction

13.4.1 Ancillary Testwork

13.4.1.1 Thickener Sizing Tests

Pocock Industrial (Salt Lake City) was commissioned to conduct solid-liquid separation (SLS) tests on King-king material to generate data for thickener design and sizing criteria. Tests were conducted on samples of pre-leach (that is, flotation) tails, leach residue tails, and neutralized leach tails. The resulting data was used to size the tailing thickeners (treating pre-leach) and Counter Current Decantation (CCD) thickeners (treating leach residue).

This work produced the following high rate thickener design parameter recommendations:

Table 13-10: High Rate Thickener Sizing Test Results

Material	pH	Max Feed Solids %	Max Underflow Solids, %	Max Unit Feed Rate $m^3/m^2\cdot hr$
Pre-leach	7.7	15 - 20	58 - 62	4.25
Leach residue	2.2	15 - 20	58 – 62 CCD 1	3.50 CCD 1
			54 – 58 CCD 2-n	2.75 CCD 2-n

Flocculant screening was conducted on small pulp samples in static settling tests to determine the effectiveness of each flocculant. Pocock selected Hychem AF 304, a widely used high molecular weight anionic polyacrylamide, for best overall performance for thickening the pre-leach tails. For the CCD thickeners, it will be necessary to use a non-ionic flocculant, such as Cytec N100 or one of the Hychem NF series, in order to avoid phase disengagement problems in the downstream solvent extraction process. Flocculant consumption rates of 0.015 kg/ton (tailing) and 0.075 kg/ton (CCD circuit) are assumed based on direct operating experience with a copper beneficiation circuit similar to that planned for King-king.

13.5 TAILS LEACH OPTION STUDY

In Section 13.1.1, it was mentioned that the Phase 2 oxide reagent scheme was designed to collect only the floatable sulfides in oxide dominant ore, with the recovery of the oxides being left for later stage leaching of flotation tails. A Tails Leach Option Study was carried out to investigate this concept.

Twelve variability flotation tails samples from previous sulfide flotation test work were used for the tails leach testing (see discussion in Section 13.1.2). All 12 samples were prepared such that they contained 35 % w/w solids, and were atmospherically leached (batch and agitated) at 50°C.

A leach time of 12 hours and acid to feed ratio of 50 kg/t were selected for this test work. These were kept constant to determine the effects of feed variability (determined by mine plan) on leach kinetics.

Figure 13-1 and Figure 13-2 below illustrate the kinetic variability across all samples for total copper and non-WAS copper.

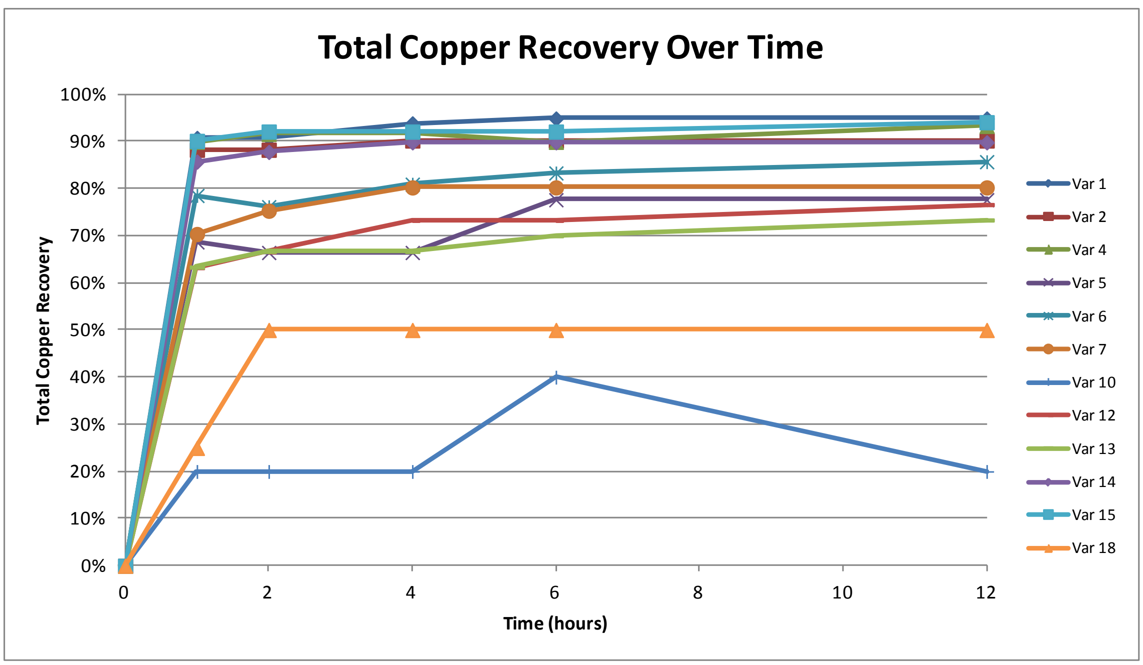


Figure 13-1: Total Copper Dissolution Over Time

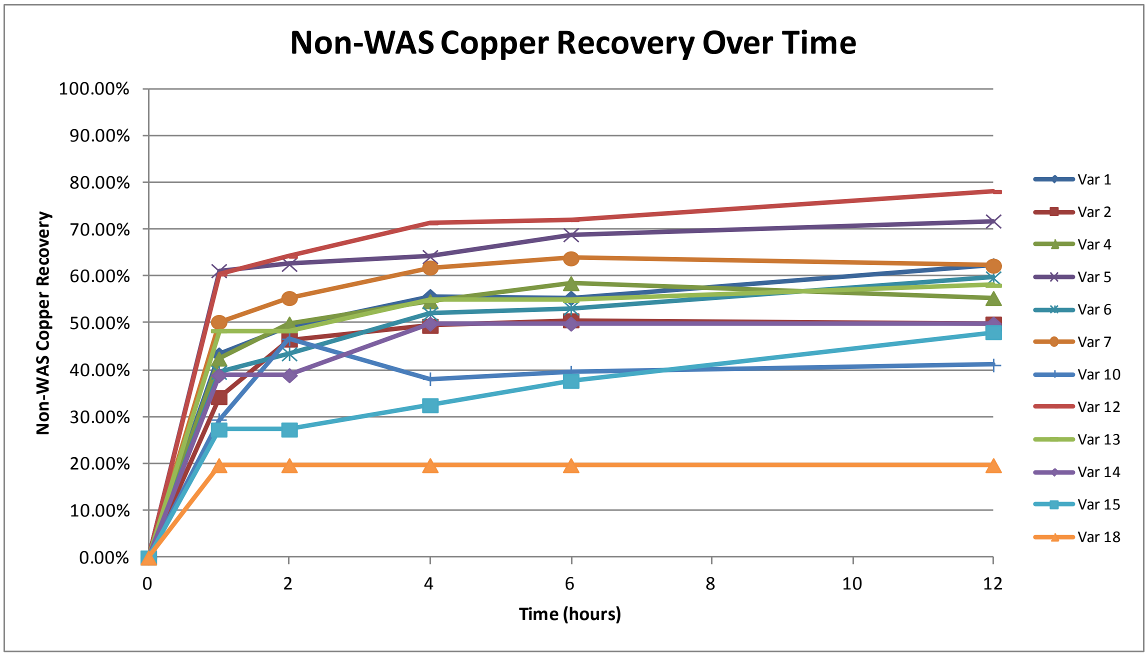


Figure 13-2: Non-WAS Copper Dissolution Over Time

The leach results indicate that the rate of copper (total) dissolution begins to decline between 4 and 6 hours for most variability samples. After 6 hours, most samples had reached their peak dissolution. The same observations were made for the weak acid soluble (WAS) copper and non-weak acid soluble (Non-WAS) copper constituents of the samples, suggesting that 4 to 6 hours of leaching will suffice.

After 12 hours of leaching, a total copper recovery range between 20% and 94.4% was observed across all sample results. A smaller range (19.7%-78.1%) was noted for the non-WAS copper.

Subsequent examination of acid consumption levels for King-king oxide ores considered the types and relative abundance of acid consuming gangue minerals in the rock. Based on this analysis and the results of laboratory column leach testing, a revised estimate for acid consumption in actual practice of 25 kg/t has been reached.

13.6 COLUMN LEACH TESTS

The program for column leach testing of the King-king samples was developed after a review of test results from a previous investigation by Metcon (King-king Project, Upper Ore Type, Column Leach Study, Vol. II, METCON Project M-454-01, May 1994). Twenty-five column leach tests were run on samples of the King-king oxide ore representing oxide ore types to be mined over the 6 years of anticipated life of the heap leach operation. The tests were conducted in six-inch diameter six-foot high columns using acid solution based on mature raffinate solution obtained from the ASARCO Ray (Arizona) heap leach operation. The free acid content of this solution was adjusted before use in the curing step and the column leach test.

13.6.1 Sample Preparation

All sample preparation and column leach testing was conducted at Mountain States R&D (MSRDI) in Vail, AZ, by the staff of Leach Inc. or under the direct supervision of Leach Inc. Assays were done primarily by Metcon in Tucson, AZ with some PLS samples done by the assay laboratory of MSRDI.

Split core samples were crushed to minus one inch. From each crushed sample a 50 kg portion was cut for use as column feed. The remainder, approximately 30 kg, was split with half being retained for future use and the remainder being screened into six size fractions and prepped for assay. The screen fractions were +3/4 in., 3/4x5/8 in., 5/8x1/2 in., 1/2x3/8 in., 3/8x1/4 in., and minus 1/4 in. Each of the screen fractions was crushed to minus 1/4 in., roll crushed to minus 10 mesh, and split. Half the split was discarded; the other half was pulverized to minus 100 mesh and split again. Half of that split was discarded; the remainder, approximately 200 grams, was submitted for sequential copper assay.

It was noted during the early stages of the column leach tests that calculated recoveries of acid soluble copper were in excess of 100 percent suggesting the head assay procedure was underestimating the soluble copper content of the sample. Samples were re-assayed using a hot acid dissolution in place of the default ambient temperature dissolution method. This new procedure gave significantly higher values for the acid soluble copper content of each of the 25 samples.

Each of the six size fractions of each of the 25 column feed head samples was submitted for assay of total copper, Cu(total), and acid soluble copper, Cu(AS). Table 13-11 lists the calculated head assay of the 25 column feeds calculated from the six screen fractions of each sample. The cyanide soluble copper assay, Cu(CNsol), was run on the residue of the ambient acid soluble copper determination.

Table 13-11: Head Assays of Column Feeds

Column Feed	Cu(Total), %	Cu(AS)Amb, %	Cu(AS)Hot, %	Cu(CNsol), %	SI(Amb)	SI(Hot)
KK-1	0.644	0.343	0.638	0.028	0.53	0.99
KK-2	0.326	0.129	0.310	0.013	0.40	0.95
KK-3	0.146	0.063	0.134	0.012	0.44	0.92
KK-4	0.352	0.161	0.345	0.014	0.46	0.98
KK-5	0.355	0.099	0.331	0.019	0.28	0.93
KK-6	0.283	0.100	0.253	0.022	0.35	0.89
KK-7	0.331	0.184	0.322	0.016	0.56	0.97
KK-8	0.563	0.330	0.472	0.149	0.59	0.84
KK-9	0.872	0.763	0.852	0.034	0.88	0.98
KK-10	0.470	0.270	0.366	0.113	0.58	0.78
KK-11	0.716	0.290	0.382	0.365	0.41	0.53
KK-12	0.166	0.054	0.152	0.014	0.33	0.92
KK-13	0.183	0.053	0.149	0.013	0.29	0.81
KK-14	0.463	0.336	0.445	0.028	0.73	0.96
KK-15	0.772	0.470	0.598	0.182	0.61	0.77
KK-16	0.619	0.315	0.598	0.045	0.51	0.96
KK-17	0.450	0.284	0.394	0.083	0.63	0.88
KK-18	0.394	0.262	0.384	0.021	0.67	0.97
KK-19	0.136	0.051	0.122	0.011	0.38	0.90
KK-20	0.182	0.103	0.164	0.016	0.56	0.90
KK-21	0.292	0.134	0.281	0.024	0.46	0.96
KK-22	0.343	0.142	0.345	0.032	0.41	1.01
KK-23	0.797	0.530	0.720	0.125	0.67	0.90
KK-24	0.619	0.352	0.506	0.139	0.57	0.82
KK-25	0.323	0.041	0.067	0.070	0.13	0.21

The samples for column leach testing were selected to include a wide range of solubility indices; however, when the samples were re-assayed with the "hot acid" soluble procedure, it appears that the copper in the oxide ore is predominantly acid soluble.

It should be noted that the assay results shown in Table 13-11 are used only for monitoring the progress of the column leach test while it is underway. Because there will be a slight variance between the head sample used for these assays and the head sample used for the column feeds, the results of the column leach tests are based on column feed assays calculated at the conclusion of the test from the copper assays of the column residue and all of the PLS solutions generated.

13.6.2 Column Leach Test

13.6.2.1 Cure

Column feed samples were split from drill core in fifty-kilogram portions. Each sample was placed on a large sheet of plastic and a stock cure solution was sprayed over the ore as it was rolled back and forth to mix the solution with the ore, until the right consistency of ore/solution was reached. The stock cure solution was made by adjusting the free acid content of the ASARCO raffinate to a desired level. The volumes of solution added were recorded and the ore/solution mixtures were loaded into the leach columns. The 50 kg sample size was intentionally selected to be more than the columns could contain and the excess from each sample was weighed to determine the exact weight of ore loaded into each column.

13.6.2.2 Leaching

Twenty five columns were loaded with the acid-cured composites. The ore was allowed to cure for three days before the 90-day irrigation cycle with leach solution (adjusted ASARCO raffinate) was started. In the first few days, two of the columns, Columns KK-5 and KK-13, were stopped because they were plugged by fine particles. The irrigation rate was initially set to 0.0045 gpm/ft^2 then reduced to 0.003 gpm/ft^2. PLS was collected initially every day, then after two weeks, three times per week, and after four weeks, twice a week. The volume of PLS collected was determined by weighing the PLS and measuring the specific gravity of the solution. The pH and the oxidation–reduction potential (ORP), also known as electromotive force (emf), of the PLS were measured within a few hours after collection. Free acid was determined by titration, and copper and iron contents by Atomic Absorption Spectroscopy (AAS). Ferrous and ferric iron concentrations were calculated from the total iron assay and the ORP using the Nernst equation.

Recovery of both total copper, Cu(total), and hot acid soluble copper, Cu(AS)hot, were calculated for each PLS sample and cumulatively for the test. In addition, acid consumption per ton of ore and per pound of copper recovered were also calculated for each PLS sample and cumulatively for the test.

13.6.3 Leach Test Results

Copper recoveries are calculated as a percentage of total copper, Cu(total), in the sample and percentage of hot acid soluble copper, Cu(AS)hot, in the sample. Recoveries are based on the calculated head of the column test, as previously defined. Because it is not possible to directly calculate the Cu(AS)hot content of the column feed, it was estimated by adjusting the assayed head of the Cu(AS)hot by the ratio of the assayed head of Cu(total) to the assayed head of Cu(AS)hot. This assumes that the variance between the assayed head and the calculated head are the same for Cu(total) and Cu(AS)Hot.

13.6.3.1 Recovery as a Function of Leach Time

Figure 13-3 to Figure 13-8 show the column leach test results from the 23 columns, in terms of recovery of Cu(AS)hot as a function of leach time. The results are grouped according to the year each composite represents in the mine plan. Based on these results, a nominal leach cycle time of 60 days was selected.

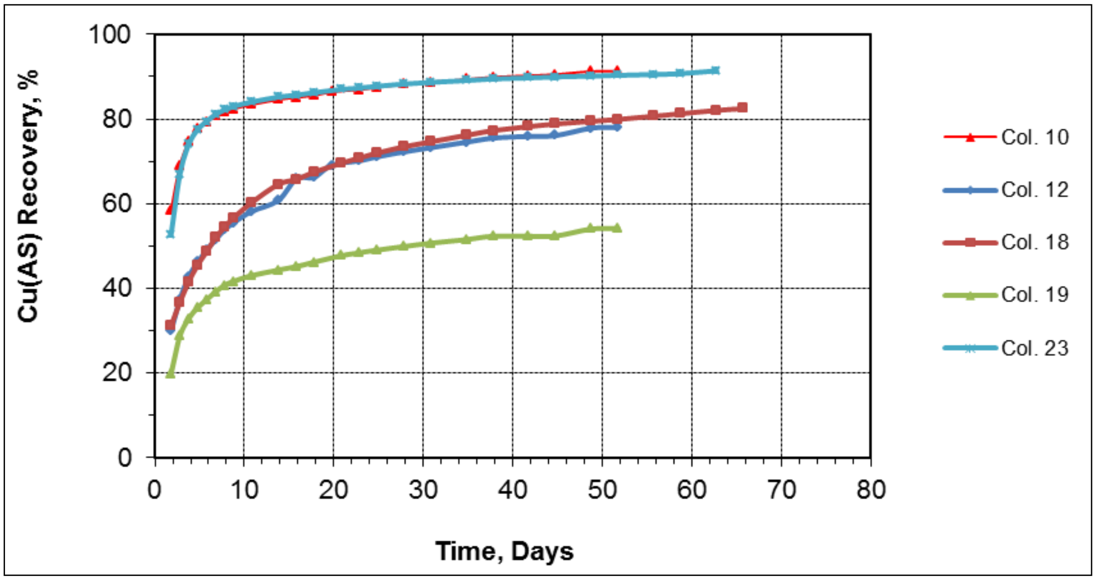


Figure 13-3: Recovery/Time Plot (Year 0)

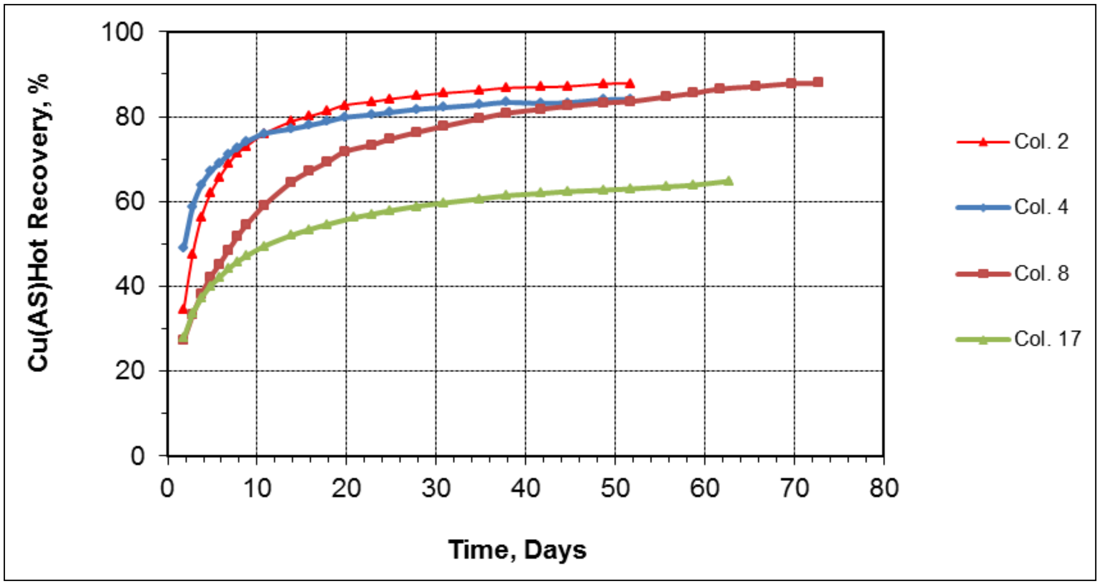


Figure 13-4: Recovery/Time Plot (Year 1)

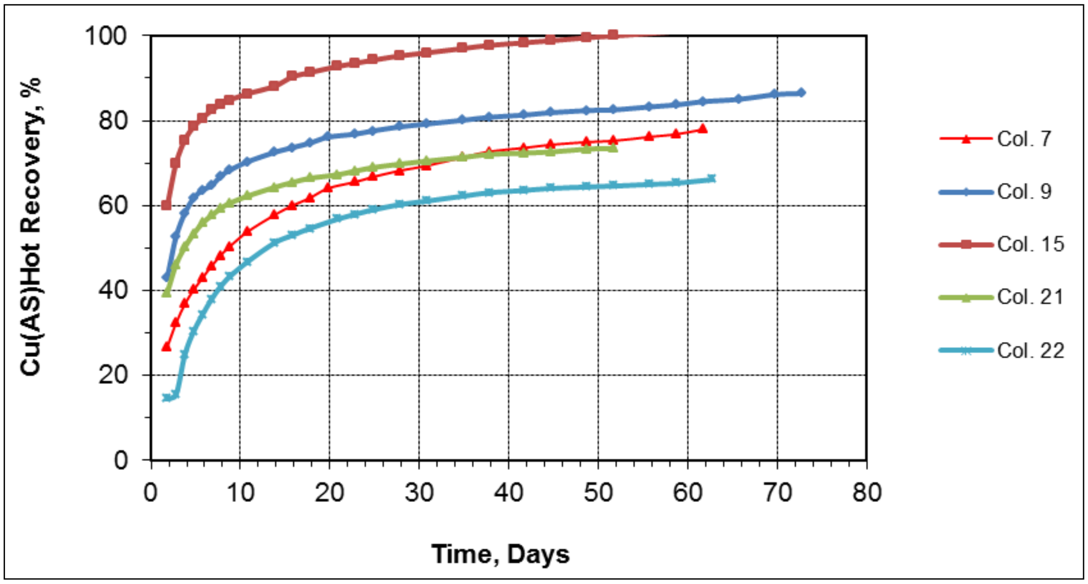


Figure 13-5: Recovery/Time Plot (Year 2)

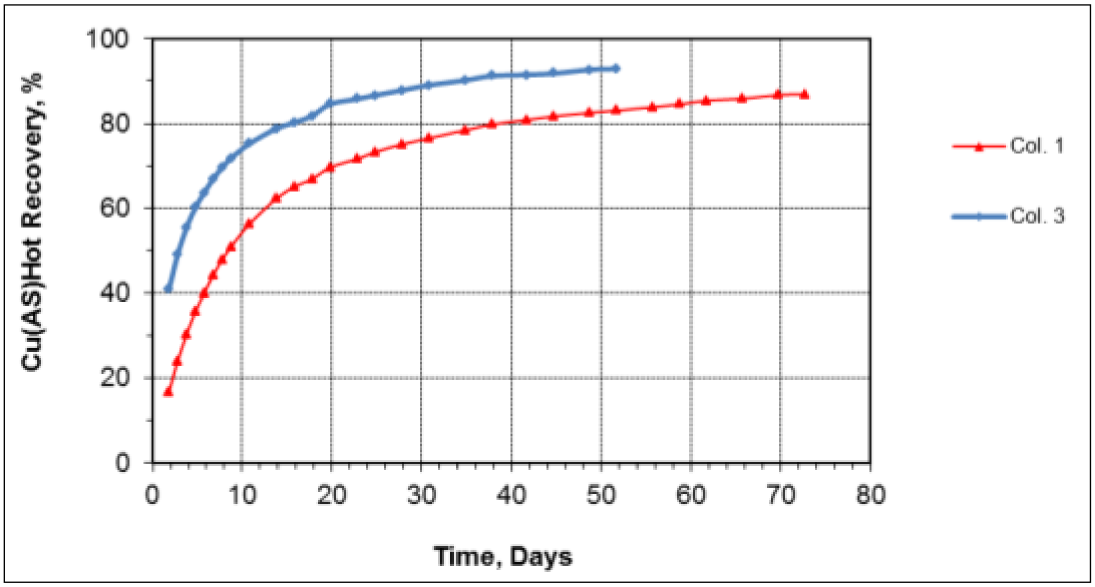


Figure 13-6: Recovery/Time Plot (Year 3)

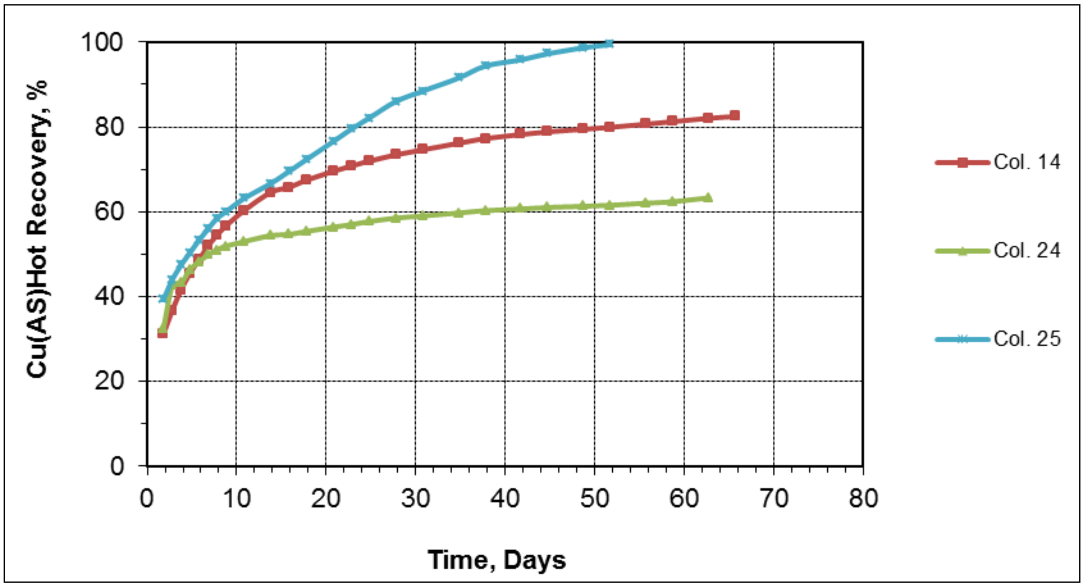


Figure 13-7: Recovery/Time Plot (Year 4)

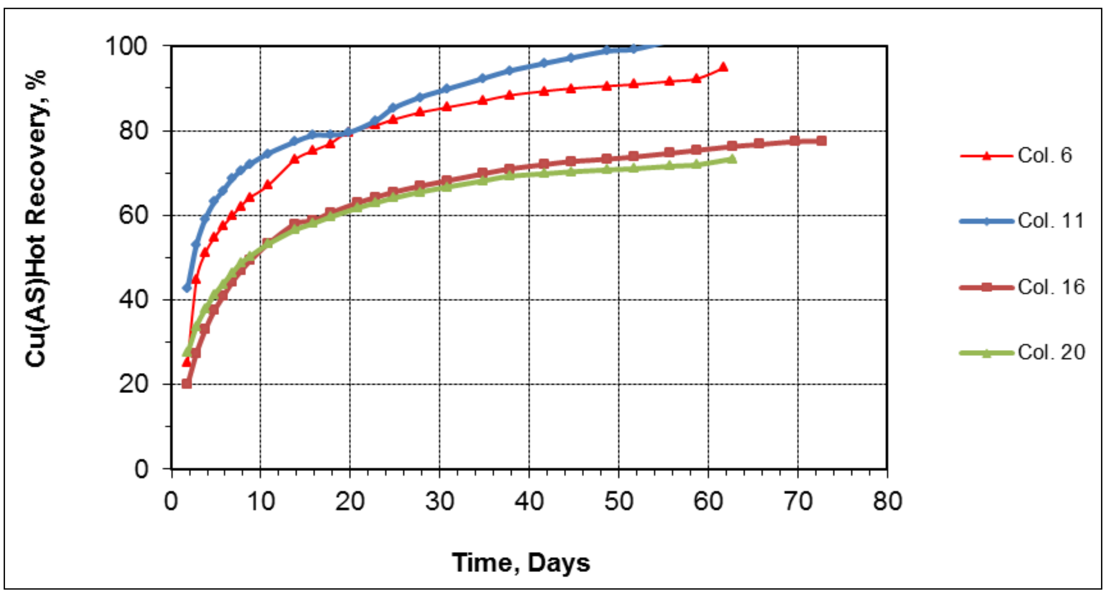


Figure 13-8: Recovery/Time Plot (Year 5 & 6)

13.6.3.2 Effect of Particle Size on Recovery

The residues of each column were screened and each size fraction was assayed for total copper, Cu(total). Because the recovery from the individual particles in the size fraction is of interest, not the recovery of the entire size fraction, no adjustment was made for the change in the weight of the size fraction resulting from the leach test. Table 13-12 lists the recoveries from each size fraction for the 23 column test.

The recovery by particle size suggests that all of the soluble copper has been leached with no significant particle size effect. Based on this, a heap feed crush size of -1 inch is indicated. The exception is Column KK-25, where the lower recoveries suggest that the hot assay procedure may have overestimated the soluble copper content.

Table 13-12: Cu(total) Recovery vs Particle Size

Column	Size Fraction					
	+3/4"	3/4" x 5/8"	5/8" x 1/2"	1/2" x 3/8"	3/8" x 1//4"	-1/4"
KK-1	82.4	85.3	87.2	87.9	89.1	88.9
KK-2	74.0	74.8	80.0	77.8	77.6	76.8
KK-3	75.7	78.9	76.6	81.0	75.2	74.6
KK-4	77.7	81.6	82.4	81.5	83.0	85.1
KK-6	50.3	48.6	68.0	67.0	66.5	77.4
KK-7	59.6	63.9	75.1	77.5	82.9	84.1
KK-8	66.2	70.6	72.0	84.3	83.7	86.3
KK-9	78.6	81.7	84.6	87.3	86.5	93.2
KK-10	87.7	84.0	81.1	80.6	77.3	69.2
KK-11	53.8	57.7	60.4	58.1	65.9	64.4
KK-12	60.2	59.2	59.7	59.7	62.3	66.7
KK-14	79.6	83.5	85.4	84.8	86.1	88.3
KK-15	69.8	75.1	77.0	79.7	80.4	78.3
KK-16	68.9	73.8	77.8	81.1	85.2	87.9
KK-17	66.1	79.9	74.5	73.8	82.3	82.6
KK-18	68.1	72.8	78.7	79.0	84.1	83.3
KK-19	47.3	51.7	62.2	62.8	62.9	48.7
KK-20	67.3	72.4	72.4	77.6	71.9	82.3
KK-21	90.7	73.2	74.4	76.0	78.9	80.1
KK-22	80.9	82.1	82.2	83.1	83.9	85.5
KK-23	80.7	82.9	81.8	79.7	81.8	78.6
KK-24	63.2	72.7	72.5	75.8	77.7	75.3
KK-25	36.9	22.5	27.4	33.6	35.7	25.0

13.6.3.3 Copper Recovery, Acid Consumption, Leach Cycle Time Relationship

Table 13-13 on the next page shows the relationship between copper recoveries, acid consumption, PLS grade, and leach cycle time for each of the 23 samples tested.

Based on the results of the column leach test program, Independent Mining Consultants, Inc. (IMC) calculated a heap leach copper recovery over the 14 year heap leach life. The basis for this calculated recovery was an assumed constant heap leach residue grade of 0.08 percent Cu(total) with a recovery cap of 85 percent. The calculation was done in annual increments based on the average grade of the ore placed on the heap each year as determined by a mine plan developed by IMC. The average grades were 0.311 percent Cu(total) and 0.177 percent Cu(acid soluble). Copper recoveries ranged from 66.7% to 78.4%, with an average of 73.7% of Cu(total). The cap of 85 percent was never reached in this calculation.

Table 13-13: Summary of Cu(AS)hot Recovery, PLS Grade, and Acid Consumption

Column	38 days			45 days			52 days		
	Cu(AS)hot Recovery, %	Average PLS Grade, gpl Cu	Gangue Acid Consumption, lb/ton	Cu(AS)hot Recovery, %	Average PLS Grade, gpl Cu	Gangue Acid Consumption, lb/ton	Cu(AS)hot Recovery, %	Average PLS Grade, gpl Cu	Gangue Acid Consumption, lb/ton
KK-1	81.5	1.59	45.0	83.5	1.42	48.0	84.9	1.28	56.3
KK-2	81.2	0.88	48.8	81.5	0.78	52.1	82.2	0.68	52.1
KK-3	83.8	0.52	41.3	84.4	0.47	43.2	85.4	0.41	43.2
KK-4	83.5	1.01	51.5	83.5	0.88	54.8	84.2	0.77	54.7
KK-6	79.3	0.86	57.9	80.7	0.76	61.1	81.6	0.68	69.3
KK-7	69.3	0.88	36.6	70.8	0.79	38.2	71.7	0.73	42.8
KK-8	81.0	2.14	11.8	82.7	1.88	13.0	83.8	1.72	17.5
KK-9	88.4	1.34	39.5	89.7	1.18	41.7	90.4	1.09	47.5
KK-10	93.6	1.11	14.7	94.3	0.98	14.9	95.0	0.88	14.8
KK-11	98.0	1.35	32.4	101.2	1.19	34.1	103.3	1.10	39.3
KK-12	69.2	0.50	48.8	69.7	0.46	52.1	71.5	0.42	52.1
KK-14	77.2	1.18	44.6	78.8	1.05	47.2	79.9	0.95	55.2
KK-15	97.7	1.96	22.9	98.8	1.71	24.0	99.9	1.53	30.4
KK-16	71.0	1.37	38.0	72.7	1.22	40.3	73.8	1.10	48.0
KK-17	61.4	0.82	37.8	62.4	0.74	39.3	62.4	0.67	46.4
KK-18	59.4	0.82	39.1	60.3	0.73	41.6	61.0	0.67	49.5
KK-19	52.3	0.35	42.7	52.4	0.32	43.6	54.1	0.30	43.6
KK-20	69.2	0.51	33.4	70.3	0.47	35.5	71.0	0.45	40.9
KK-21	72.0	0.76	37.0	72.7	0.67	39.6	73.4	0.59	39.5
KK-22	63.0	0.84	45.7	64.1	0.75	48.2	64.6	0.68	55.9
KK-23	89.5	1.75	37.0	89.9	1.53	39.7	90.3	1.36	48.5
KK-24	60.3	0.95	47.9	61.0	0.84	51.0	61.6	0.76	59.9
KK-25	94.3	0.37	32.7	97.2	0.35	34.4	99.4	0.31	34.4
Average	77.2	1.04	38.6	78.4	0.92	40.8	79.4	0.83	45.3

14 MINERAL RESOURCE ESTIMATES

14.1 MINERAL RESOURCES

Table 14-1 shows the mineral resource for the project.

Table 14-1: King-king Mineral Resource (August 9, 2011)

Ore Type/Resource Class	Ore Ktonnes	Eq Cu (%)	Tot Cu (%)	Sol Cu (%)	Gold (g/t)	Eq Au (g/t)
Measured Mineral Resource						
Oxide Ore	39,513	1.180	0.431	0.266	0.535	0.843
Sulfide Ore	80,829	0.551	0.258	0.037	0.427	0.803
Total Measured Resource	120,342	0.758	0.315	0.112	0.462	0.816
Indicated Mineral Resource						
Oxide Ore	122,350	0.868	0.334	0.203	0.382	0.620
Sulfide Ore	719,560	0.439	0.230	0.029	0.305	0.640
Total Indicated Resource	841,910	0.501	0.245	0.054	0.316	0.637
Measured/Indicated Mineral Resource						
Oxide Ore	161,863	0.944	0.358	0.218	0.419	0.675
Sulfide Ore	800,389	0.450	0.233	0.030	0.317	0.657
Total Meas/Ind Resource	962,252	0.533	0.254	0.062	0.334	0.660
Inferred Mineral Resource						
Oxide Ore	33,303	0.747	0.276	0.160	0.337	0.534
Sulfide Ore	155,513	0.373	0.202	0.024	0.249	0.544
Total Inferred Resource	188,816	0.439	0.215	0.048	0.265	0.542

Notes:

Eq Cu (oxide) = Total Copper + 1.400 x Gold, Cutoff = 0.30% Eq Cu
Eq Cu (sulfide) = Total Copper + 0.686 x Gold, Cutoff = 0.15% Eq Cu
Alternatively, as Equivalent Gold:
Eq Au (Oxide) = Gold + 0.714 x Total Copper, Cutoff = 0.22 g/t Eq Au
Eq Au (Sulfide) = Gold + 1.458 x Total Copper, Cutoff = 0.22 g/t Eq Au

Total Material in Cone Shell	1,736,371	Ktonnes
Waste:Ore Ratio	0.80	(Inferred as Waste)
Waste:Ore Ratio	0.51	(Inferred as Ore)

Measured and indicated mineral resource amounts to 962.3 million tons at 0.533% copper equivalent, 0.254% total copper, 0.062% soluble copper, and 0.334 g/t gold. Inferred mineral resource is an additional 188.8 million tons at 0.439% copper equivalent, 0.215% total copper, 0.048% soluble copper, and 0.265 g/t gold. The last column of the table also shows that with metal grades defined in terms of equivalent gold, instead of equivalent copper, the equivalent gold grade of the measured and indicated mineral resource is 0.660 g/t gold equivalent (0.675 g/t for the oxide resource and 0.657 g/t for the sulfide resource). The measured and indicated mineral resource consists of 5.4 billion pounds of contained copper and 10.3 million troy ounces of contained gold.

The resources are contained within a floating cone pit shell and are compliant with the "reasonable prospects for economic extraction" clauses of Canada's NI 43-101 regulations and also Australia's JORC code. The cone shell is based on a copper price of US$ 2.50 per pound and a gold price of US$ 1,100 per Troy ounce. Table 14-2 shows the cost and recovery parameters used to develop the cone shell. The parameters are based on bulk open pit mining of the ore followed by crushing, grinding, and flotation to produce copper concentrates.

Table 14-2: **Economic Parameters for Mineral Resource Determination**

Parameter	Units	Oxide Mill	Sulfide Mill
Copper Price Per Pound	(US$)	2.500	2.500
Gold Price Per Troy Ounce	(US$)	1100	1100
Base Mining Cost Per Tonne Material	(US$)	1.250	1.250
Mine Replacement Capital Per Tonne	(US$)	0.100	0.100
Process Cost Per Ore Tonne	(US$)	5.000	5.000
G&A Cost Per Ore Tonne	(US$)	0.270	0.270
Process Recovery of Copper (Average)	(%)	37.8%	77.2%
Process Recovery of Gold (Average)	(%)	75.0%	75.0%
Smelting/Refining Payable for Copper	(%)	96.4%	96.4%
Smelting/Refining Payable for Gold	(%)	95.0%	95.0%
SRF (or SXEW) Cost Per Pound Copper	(US$)	0.260	0.260
Gross Royalty	(%)	3.0%	3.0%
NSR Factor for Total Copper	(US$)	17.455	35.649
NSR Factor for Gold	(US$)	24.443	24.443
Gold Factor for Copper Equivalent	(none)	1.400	0.686
Total Copper Equivalent Cutoff Grades			
Breakeven (without lift)	(%Cu)	0.38	0.19
Internal	(%Cu)	0.30	0.15
Copper Factor for Gold Equivalent	(none)	0.714	1.458
Gold Equivalent Cutoff Grades			
Breakeven (without lift)	(g/t)	0.27	0.27
Internal	(g/t)	0.22	0.22

The mining related costs, base mining and mine capital replacement are preliminary IMC estimates. The process cost, G&A cost, SRF cost (smelting, refining, and freight) and process recoveries were provided by MDCA personnel. The bottom of Table 14-2 shows gold factors for copper equivalent calculations and also oxide and sulfide copper equivalent cutoff grades. For $2.50 copper and $1,100 gold, the copper equivalents are defined as:

Eq Cu (Oxide Ores) = Total Copper + 1.400 x Gold

Eq Cu (Sulfide Ores) = Total Copper + 0.686 x Gold

The Net Smelter Returns (NSR) Factors ($US/t) shown on Table 14-2 for copper and gold represent the NSR for one (1) ton of 1% copper and one (1) ton of 1 g/t gold respectively:

NSR Factor (Oxide Copper) = (\$2.50-\$0.26)(0.378)(0.964)(0.97)(22.046) = \$17.455
NSR Factor (Oxide Gold) = (\$1100)(0.75)(0.95)(0.97)/31.103 = \$24.443

NSR Factor (Sulfide Copper) = (\$2.50-\$0.26)(0.772)(0.964)(0.97)(22.046) = \$35.649
NSR Factor (Sulfide Gold) = (\$1100)(0.75)(0.95)(0.97)/31.103 = \$24.443

The 0.97 term in the above equations account for the 3% royalty.

The gold factors shown on Table 14-2 are calculated from the NSR factors as follows:

Gold Factor = NSR Factor for Gold / NSR Factor for Copper, so
Gold Factor (Oxide Ore) = \$24.443 / \$17.455 = 1.400
Gold Factor (Sulfide Ore) = \$24.443 / 35.649 = 0.686

Equivalent copper cutoff grades are then calculated as:

Breakeven Cutoff (Eq Cu) = (Mining+Processing+G&A Costs) / Copper NSR Factor
Breakeven Cutoff (Oxides) = (\$1.35+\$5.00+\$0.27)/\$17.455 = 0.38% Eq Cu
Breakeven Cutoff (Sulfide) = (\$1.35+\$5.00+\$0.27)/\$35.649 = 0.19% Eq Cu

Internal Cutoff (Eq Cu) = (Processing + G&A Costs) / Copper NSR Factor
Internal Cutoff (Oxides) = (\$5.00+\$0.27)/\$17.455 = 0.30% Eq Cu
Internal Cutoff (Sulfides) = (\$5.00+\$0.27)/\$35.649 = 0.15% Eq Cu

Internal cutoff grade treats mining costs as sunk costs, i.e. it applies to blocks that have to be removed from the pit.

Note also, to perform the analysis in terms of equivalent gold, instead of equivalent copper, the relevant factors are:

Copper Factor = NSR Factor for Copper / NSR Factor for Gold
Copper Factor (Oxide Ores) = \$17.455 / \$24.443 = 0.714
Copper Factor (Sulfide Ores) = \$35.649 / \$24.443 = 1.458

Eq Au (Oxide Ores) = Gold + 0.714 x Total Copper
Eq Au (Sulfide Ores) = Gold + 1.458 x Total Copper

Using the gold NSR factors for the cutoff grade calculations, instead of the copper NSR factor, results in breakeven gold equivalent cutoff grades of 0.27 g/t for both oxide and sulfide and internal cutoff grades of 0.22 g/t for oxide and sulfide.

Only measured and indicated resource blocks were allowed to contribute to the development of the floating cone shell used for the resource tabulation; inferred blocks were treated as valueless rock to develop the cone shell.

Total material in the cone shell is 1.7 billion tons. An overall slope angle of 45^o was used to develop the cone shell.

The resource is based on an updated block model developed by IMC and Resource Evaluation Inc. (REI) during June 2011. The mineral resource estimate was developed by Michael G. Hester, FAusIMM of IMC, a qualified person. Mr. Hester is independent of the issuer. The August 2011 update was a relatively minor update of the August 2010 model. The main features of the update were incorporation of better topography into the estimate than was previously available and the conversion of the model and drillhole database from the old Benguet coordinate system to WGS84 coordinates. The collar elevations of a few holes were adjusted to match new topography.

It should be noted that this mineral resource estimate preceded the mine plan and mineral reserve for this study by approximately six months and there was considerable refinement between the economic parameters shown on Table 14-2 and Section 15, the next report section. In particular process costs are considerably higher in Section 15, but result in higher copper recoveries for oxide ores. IMC considers that Table 14-1 is a valid statement of mineral resources.

There is no guaranty that any of the mineral resource will be converted to mineral reserve. There is also no guaranty that inferred mineral resource will be upgraded to measured or indicated mineral resource or mineral reserves.

IMC is not aware of any environmental, permitting, legal, title, taxation, socio-economic, or marketing issues that may materially impact the mineral resource. There is however, some degree of political risk associated with Mindanao. It is reported to IMC that political risk assessments will be conducted as part of on-going studies.

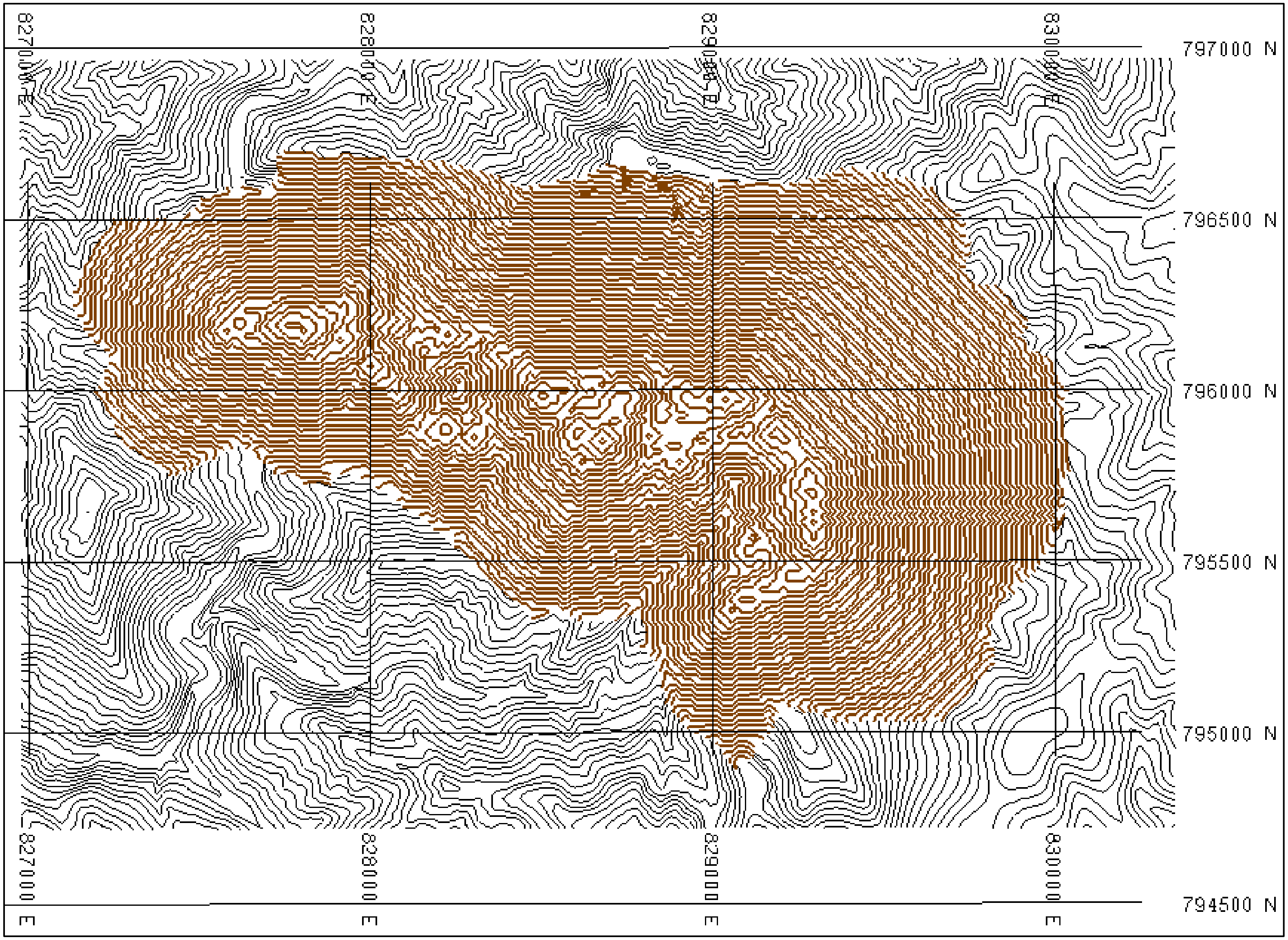


Figure 14-1: Resource Floating Cone

14.2 DESCRIPTION OF THE BLOCK MODEL

14.2.1 General

The deposit was modeled as 15m by 15m by 15m blocks.

14.2.2 Cap Grades, Corrections, and Compositing

The drillhole database provided to IMC consisted of 276 holes which represented 89,922 meters of drilling. As discussed above, Benguet gold assays were not used, however Echo Bay re-assays of Benguet samples amounted to 1,493 assays that were used. Gold assays were capped at 10 g/t, which affected six assays with original values of 44.3, 18.6, 17.3, 14.3, 11.98, and 10.7 g/t.

Copper assays were not capped. The highest assay (3m composite) was 7.2%.

The assay database was composited to 15m bench composites for block grade estimation. Based on the bench composites, the data available for resource estimation consisted of 88,597m of sample with a total copper assay (5,672 composites with an average length 15.6m) and 57,315m of sample with a gold assay (3,607 composites with an average length of 15.9m). Samples with a retained gold assay represent approximately 64.7% of sample with a copper assay.

14.2.3 Topography

New topography, in the world geodetic system 1998 (WGS84) coordinate system was available for this study. The drilling data and block model were translated into the new coordinate system.

14.2.4 Lithology Model

King-king lithology is quite complex. The original host rocks included sedimentary and volcanic flows that were intruded by multiple intrusive events. For this study, the rock types were categorized as shown in Table 14-3. It can be seen that the multiple intrusions were broadly categorized into pre-mineral/syn-mineral intrusions and post mineral intrusions.

Table 14-3: King-king Lithology for Resource Modeling

Rock Code	Description
10	Overburden
20	Host Rocks
30	Pre-mineral / Syn-mineral Intrusions
40	Post Mineral Intrusions
50	Breccias

A cross sectional interpretation of lithology was developed by RMMI personnel. REI personnel developed the interpretation on bench level maps from the sectional data. This was then digitized, checked, and incorporated into the block model.

The Benguet core drilling generally recorded a depth of overburden. This was often only a few meters, up to approximately 15m in some areas. IMC used this data to develop a surface to represent depth of overburden and used it to code overburden in the model.

The original lithology codes in the drillhole data base (actually 15m bench composites) were reconciled against the model geology. If the lithology code in the composites was not reasonable given the new interpretation, it was changed to match the code of the block it was located in. Figure 14-2 and Figure 14-3 show an example level map and cross section of model lithology.

14.2.5 Ore Types

IMC developed ore type or oxide/sulfide domains in the drilling database and block model. Table 14-4 shows how the ore type codes were initially assigned to the 15m drillhole composites based on the ratio of soluble copper to total copper grades.

Table 14-4: General Ore Type Criteria

Ore Type	Name	Description
1	"Leached"	Not used; reserved for low grade in oxide/mixed zone.
2	Oxide	Soluble copper / total copper $\geq$ 0.40
3	Mixed	0.20 < soluble copper / total copper < 0.40
4	Primary	Soluble copper / total copper $\leq$ 0.20

The codes for oxide, mixed, and primary ore types were first assigned to 15m composites based on these criteria. The assignments were then reviewed on a hole by hole basis on data listings and also on cross sections to develop a reasonable interpretation of the top of primary mineralization in each hole.

An interpretation of the top of primary mineralization was then developed from the drilling data and represented as a triangulated surface. Model blocks below the surface were coded as primary and blocks above the surface as oxide. Once the block grade estimates were completed (Section 14.2.9) the oxide zone was further segregated into oxide and mixed blocks based on the soluble copper to total copper ratio of the block. Blocks below the top of primary surface retained the primary coding though there are some areas where the soluble copper to total copper ratio is higher than would normally be expected for primary mineralization, i.e. there are local zones in the primary that might be considered as "mixed" based on the criteria of Table 14-4.

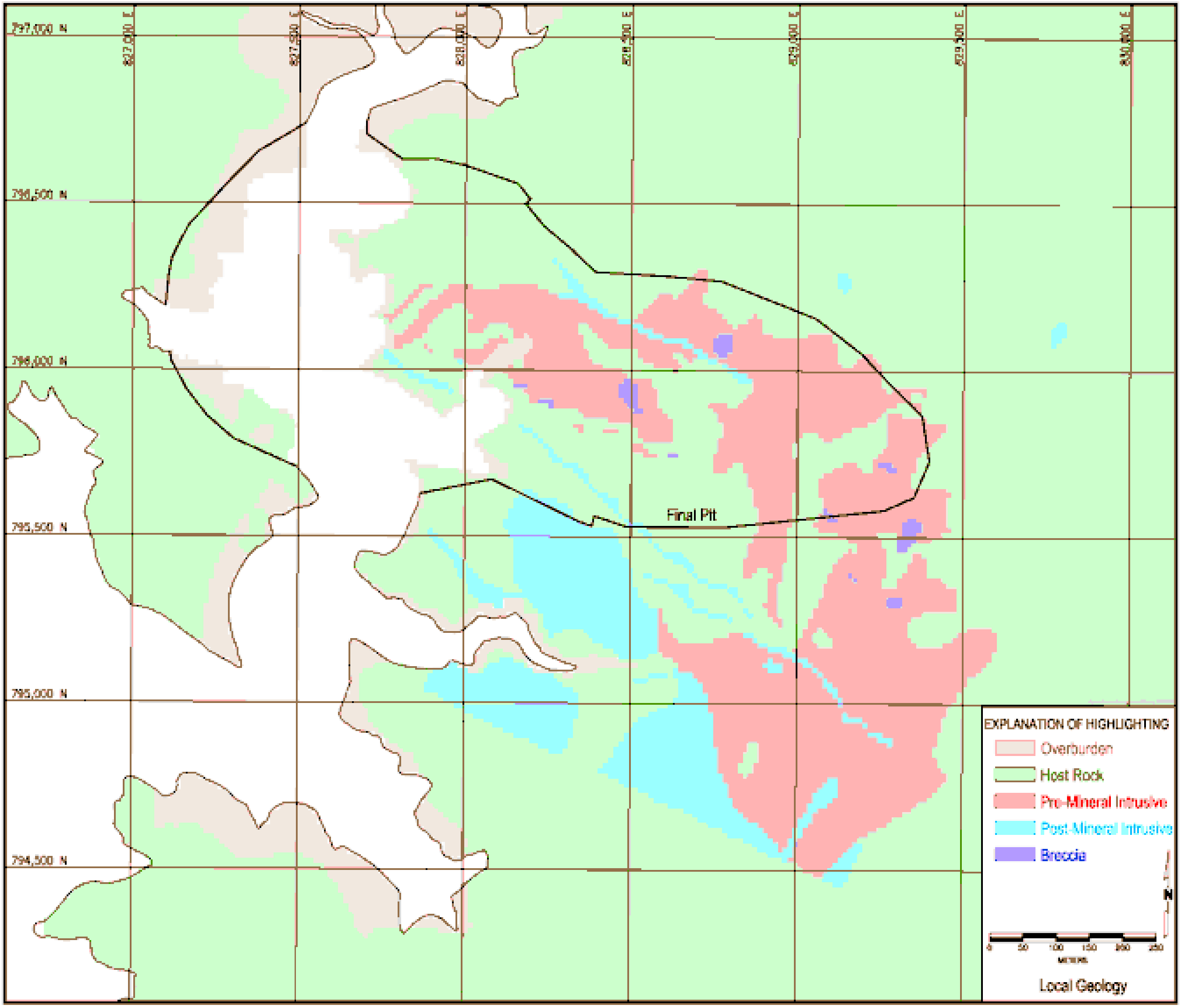


Figure 14-2: Model Lithology – 330 Bench

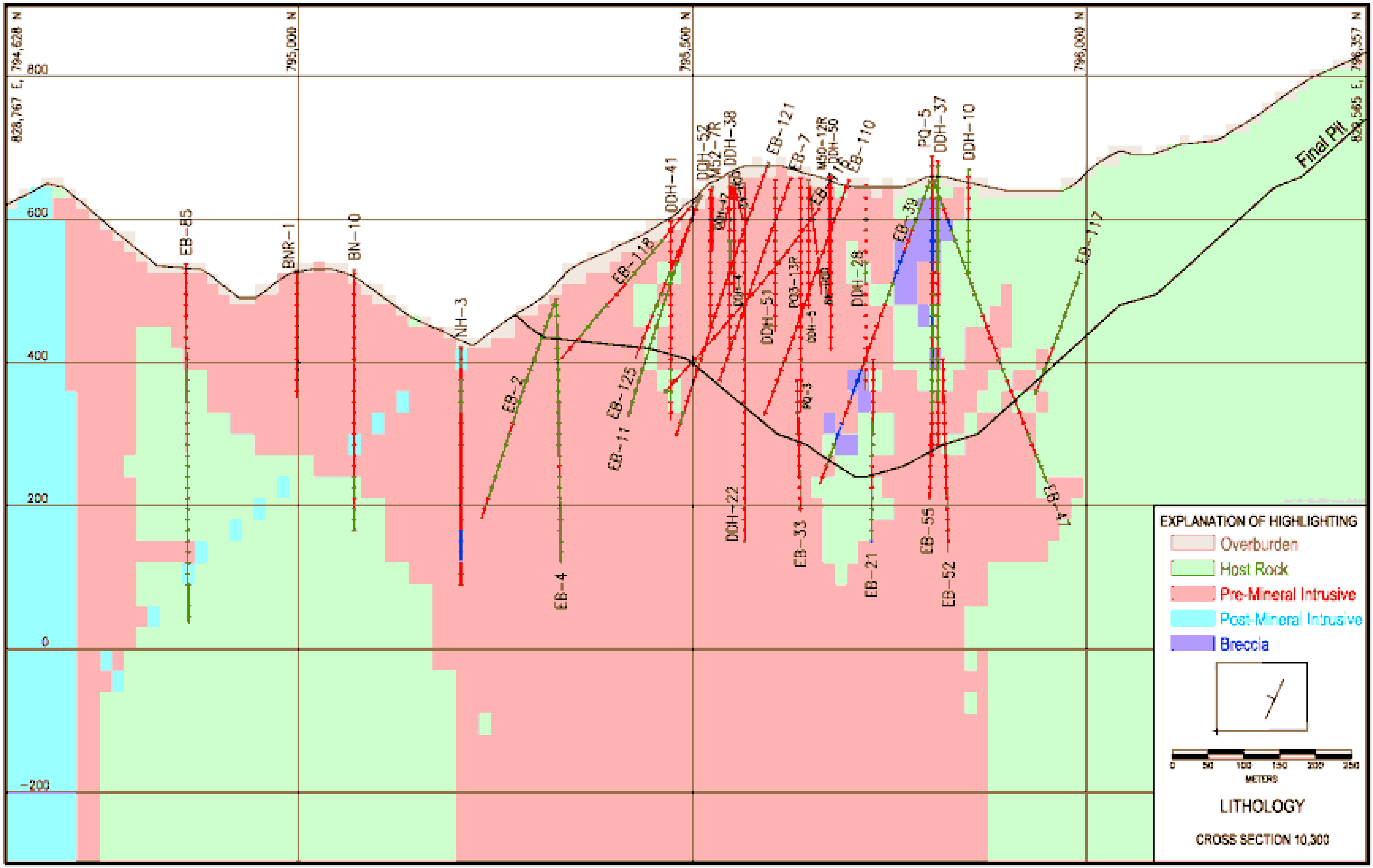


Figure 14-3: Model Lithology on Section 10,300

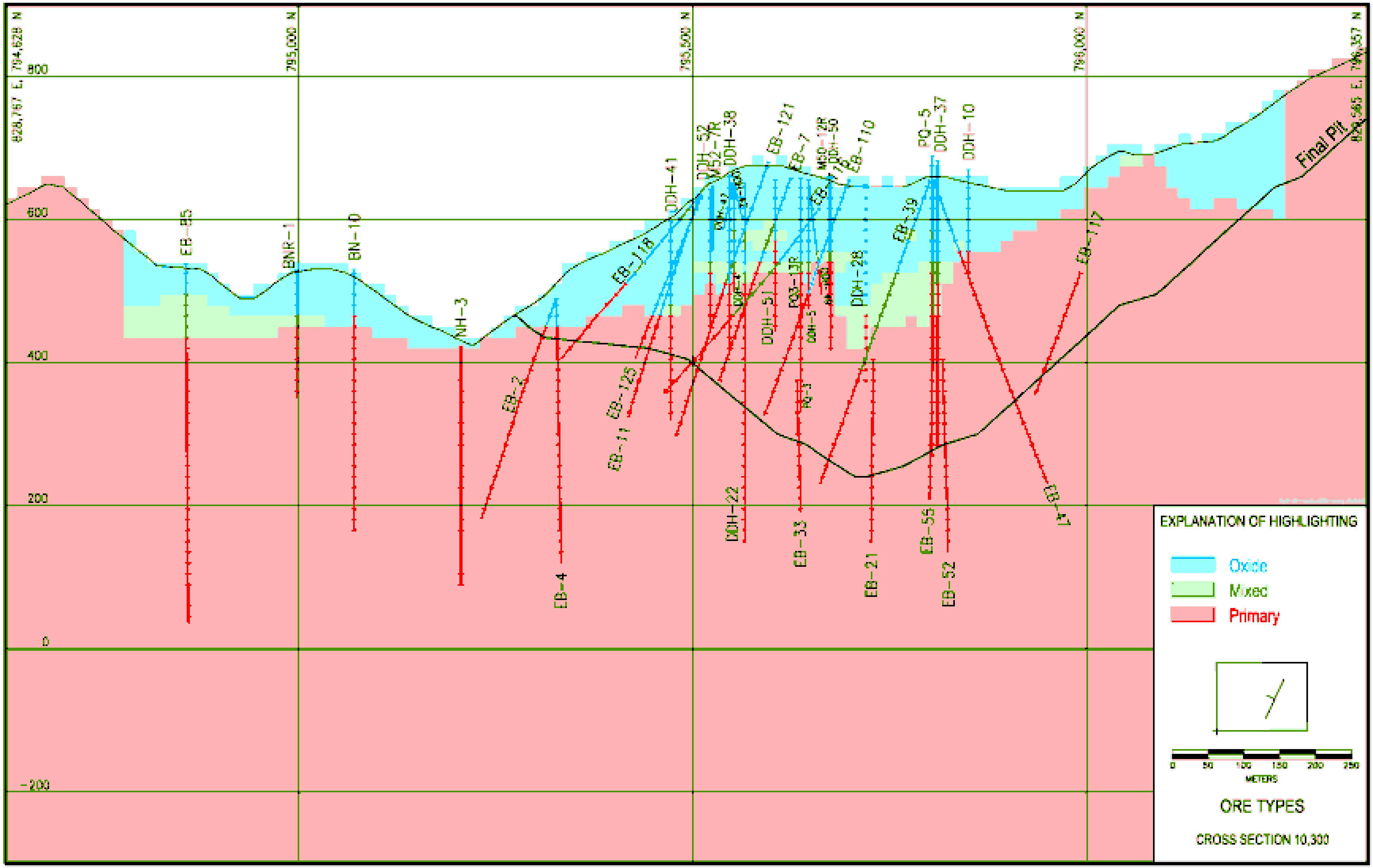


Figure 14-4: Model Ore Types on Section 10,300

14.2.6 "Structural" Zones

IMC also developed five "structural" zones for the model. These were actually based on review of grade thickness maps of copper and gold mineralization rather than any identifiable structures. Figure 14-5 shows the grade thickness map for copper with the zones. Zone 20 is slightly anomalous; it is characterized as relatively low in copper grade, but also relatively high in gold grade compared to the other zones. The outer boundary of the zones represents an approximate 100m boundary outside of the drilling. Block grades were not estimated outside the shown boundaries.

These zones also appear to correspond to historic regional names that were used to describe the deposit as follows:

Zone	Regional Name
10	Tiogdan
20	Casagumayan
30, 40	Lumanggang
50	Bacada

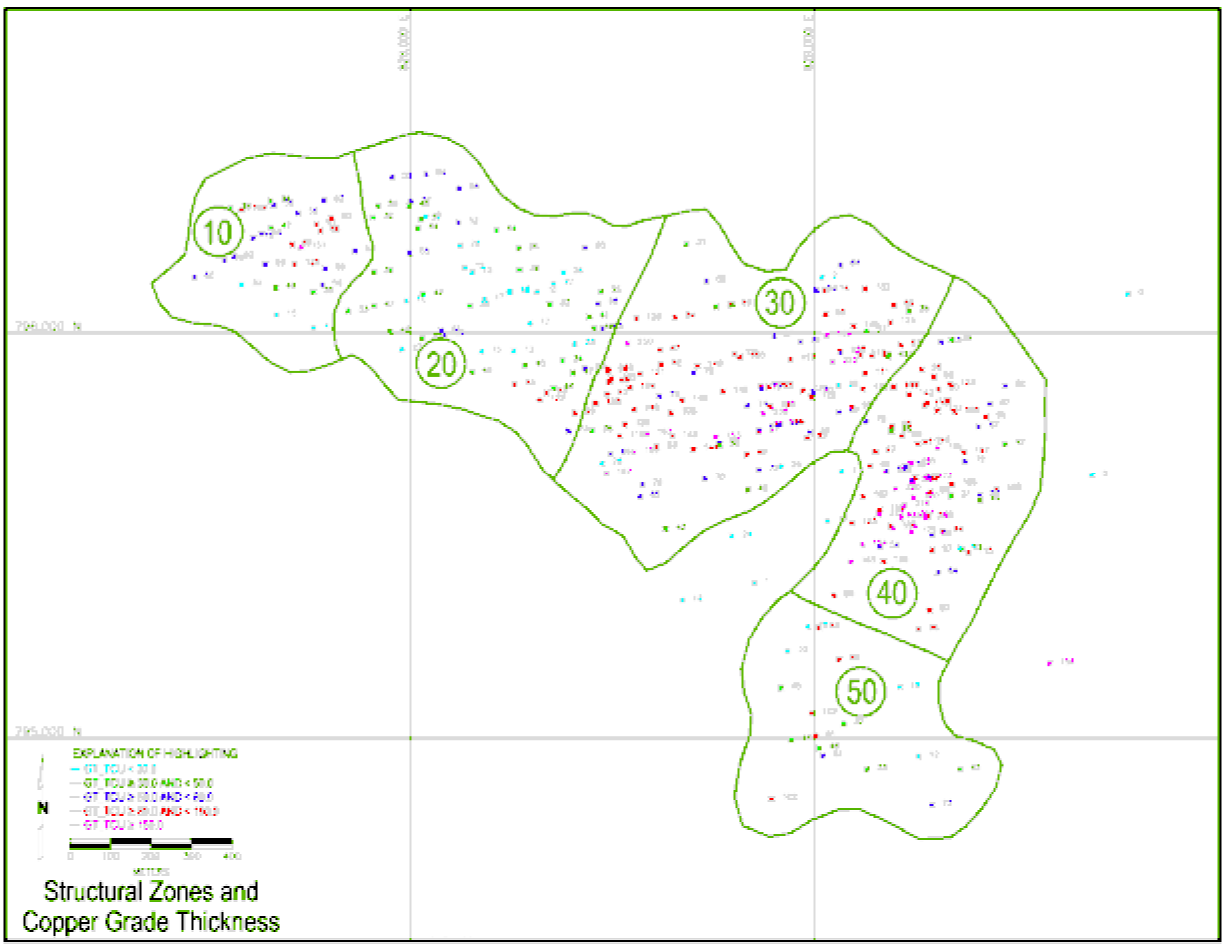


Figure 14-5: Structural Zones with Copper G x T Data

14.2.7 Basic Statistics of Drillhole Composites

Figure 14-6 and Figure 14-7 present box plots of total copper and gold respectively for 15m composites by rock type. Descriptive statistics for each population are also shown along the bottom of the plots. Graphically, the plots show the representation of the population minimum and maximum, the 25 and 75 percentiles (bottom and top of the light gray boxes), median (middle of light gray box), and mean (middle of dark gray box). The dark gray box represents a ±95% confidence interval of the mean, based on classical statistics.

For total copper, Figure 14-6 shows, as expected, that the post mineral intrusives are significantly lower in grade than the other rock types. It can also be seen that the box plots indicate relatively similarity in the other rock units. For gold, Figure 14-7 indicates slightly elevated values in the pre-mineral intrusions.

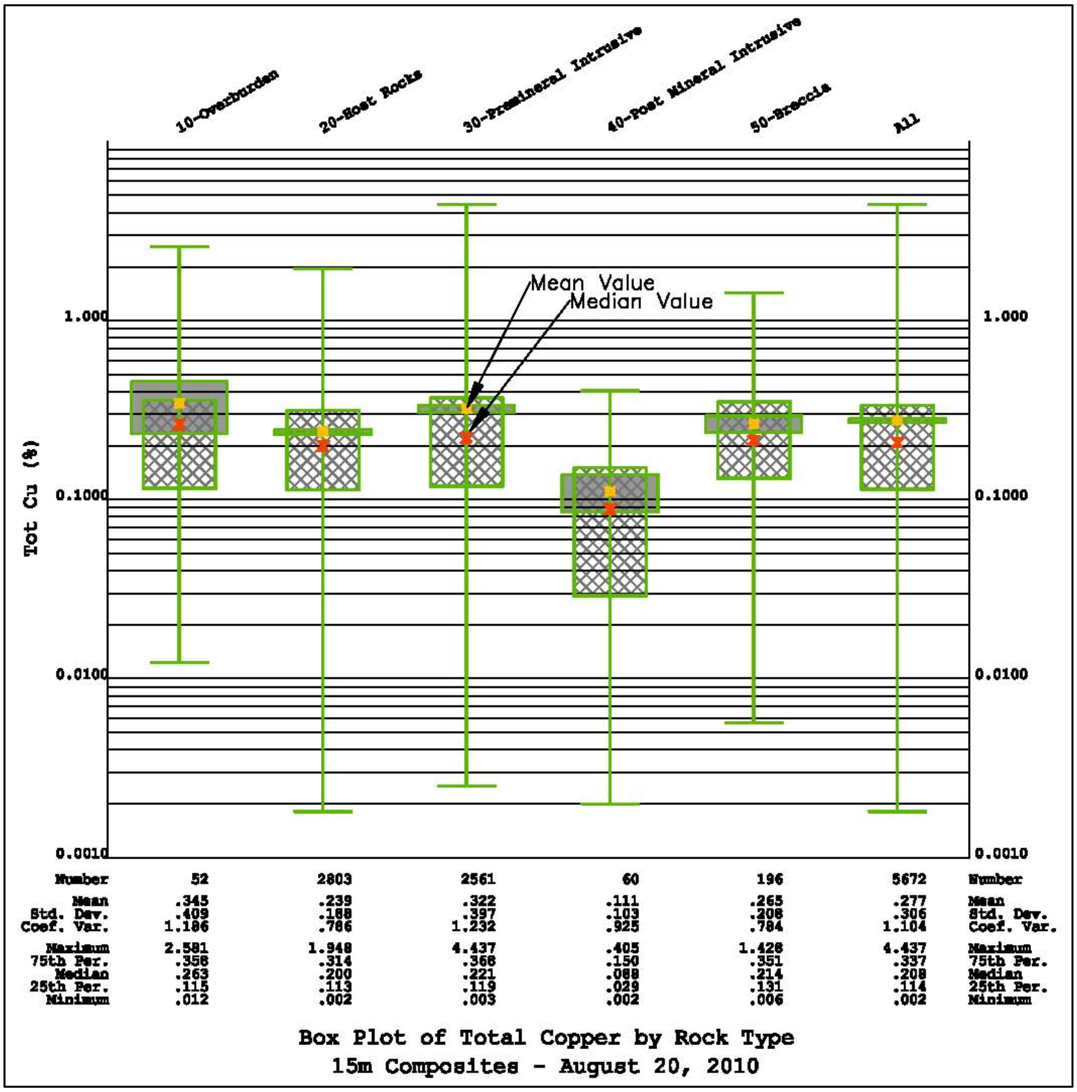


Figure 14-6: Box Plot of Total Copper by Rock Type

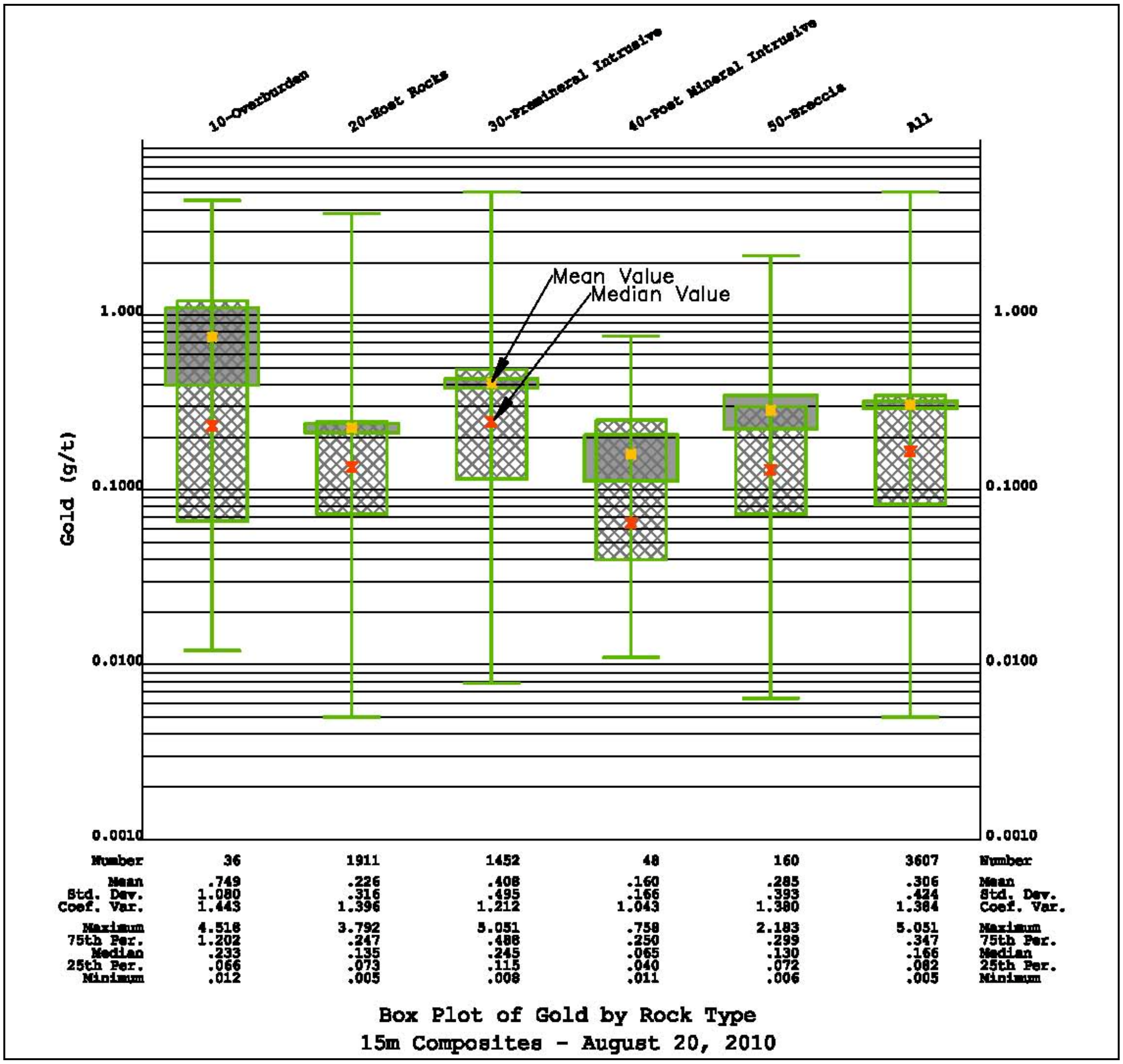


Figure 14-7: Box Plot of Gold by Rock Type

14.2.8 Variograms

A variogram analysis of total copper was completed for host rocks and pre-mineral intrusive rocks to establish search orientations for block grade estimation. First, approximately 60 directional variograms were calculated to search the entire sphere in about 22.5 degree increments. These were examined to find longest range, highest clarity, variograms that might be considered to define the primary direction. Given a candidate, or candidates, for a primary direction, a series of eight variograms were calculated to search the plane perpendicular to the primary direction, to look for the best secondary axis direction.

Figure 14-8 shows variograms for total copper for host rocks. The variograms represent the primary and secondary direction as interpreted by IMC. The primary direction has an azimuth of 300^{o} and an upward plunge of 65^{o}, or alternatively an azimuth of 120^{o} with a downward plunge of 65°. The secondary direction has an azimuth of 300° with a downward plunge of 25°. The ranges of the two variograms are about 659m and 513m, respectively

Figure 14-9 shows variograms for total copper for pre-mineral intrusive rocks. The variograms represent the primary, secondary, and tertiary directions as interpreted by IMC. The primary direction is at an azimuth of 45° with a downward plunge of 45°. The secondary axis has an azimuth of 280° and downward plunge of 30°. The ranges of the three variograms are 410m, 346m, and 207m, respectively.

All variograms were calculated by the pairwise relative method. It is also considered that the directions are reasonable given the geology and perceived orientation of mineralization as observed on sections.

14.2.9 Block Grade Estimation

14.2.9.1 General

Block grades of total copper, soluble copper, and gold were estimated by inverse distance with a power weight of 3 (ID3). This was performed to prevent over-smoothing of block grades. Search radii were typically 200m in the primary and secondary axes directions and 50m in the tertiary direction. For all estimations, a maximum of 12 and a minimum of one composite were used and a maximum of three composites per hole were allowed.

Post mineral intrusive rocks were considered a separate population for grade estimation and only post intrusive composites were used to estimate post intrusive blocks. Host rocks, pre-mineral intrusive rocks, and the breccias were considered a single population for block grade estimation. Though the pre-mineral intrusive rocks are slightly higher grade than host rocks, an analysis of the boundary indicated the boundary was of no-significance for total copper and only of slight significance for gold. Also, overburden blocks were not estimated. This represents a very small amount of material as it generally occurs as only a thin veneer at the surface.

The structural boundary was used as an outer boundary for grade estimation. Blocks not coded as one of the five zones were not estimated and composites outside the zones were not used. The boundaries between respective zones were not used as hard boundaries however. Composites in Zone 20 could be used for Zone 10 blocks, etc. Note that the ID3 estimation will tend to honor the data pretty closely regardless of boundaries.

The oxide/sulfide domain boundary was used as a hard boundary for total and soluble copper, but not for gold.

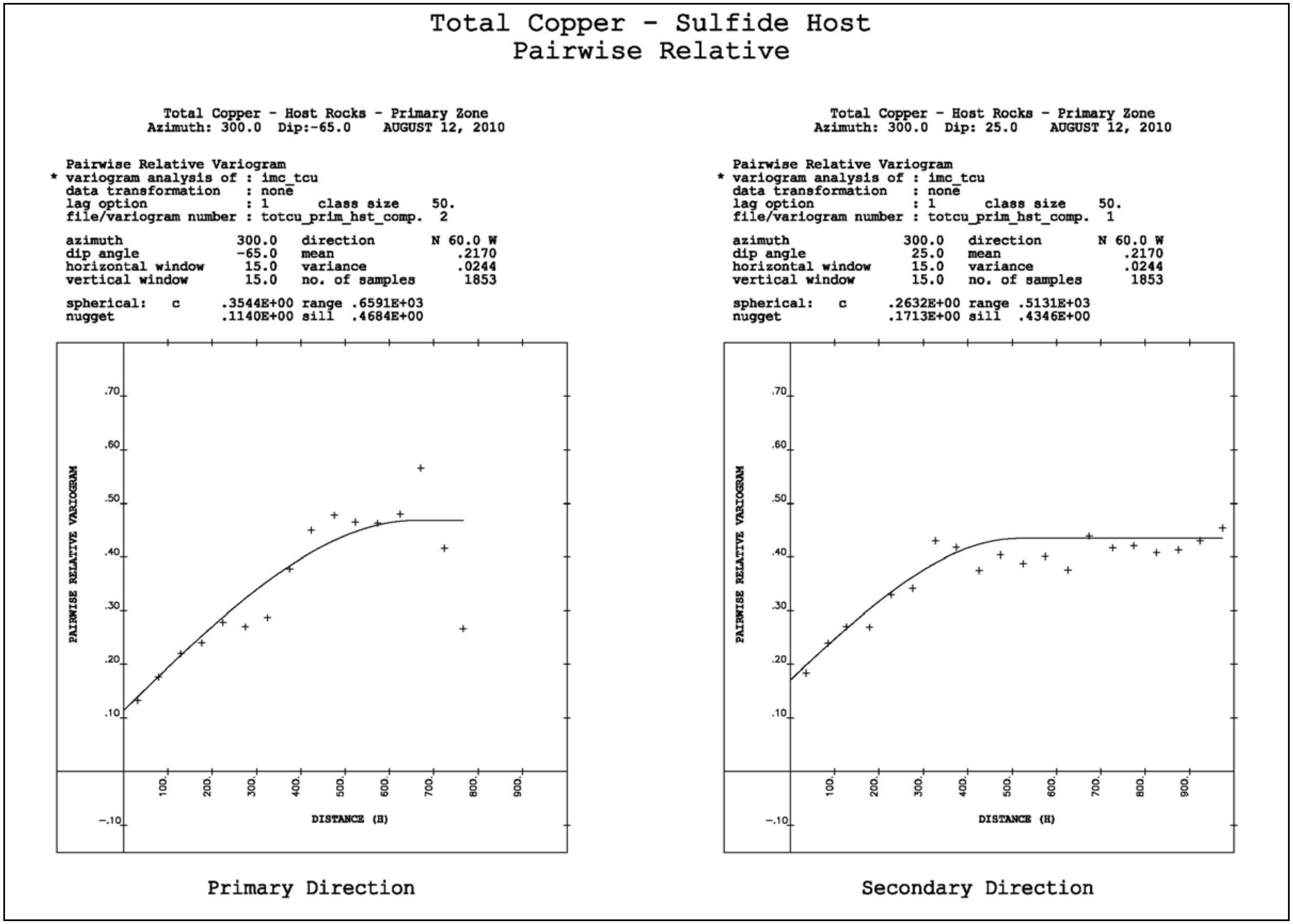


Figure 14-8: Total Copper Variograms Host Rocks in Sulfide Zone

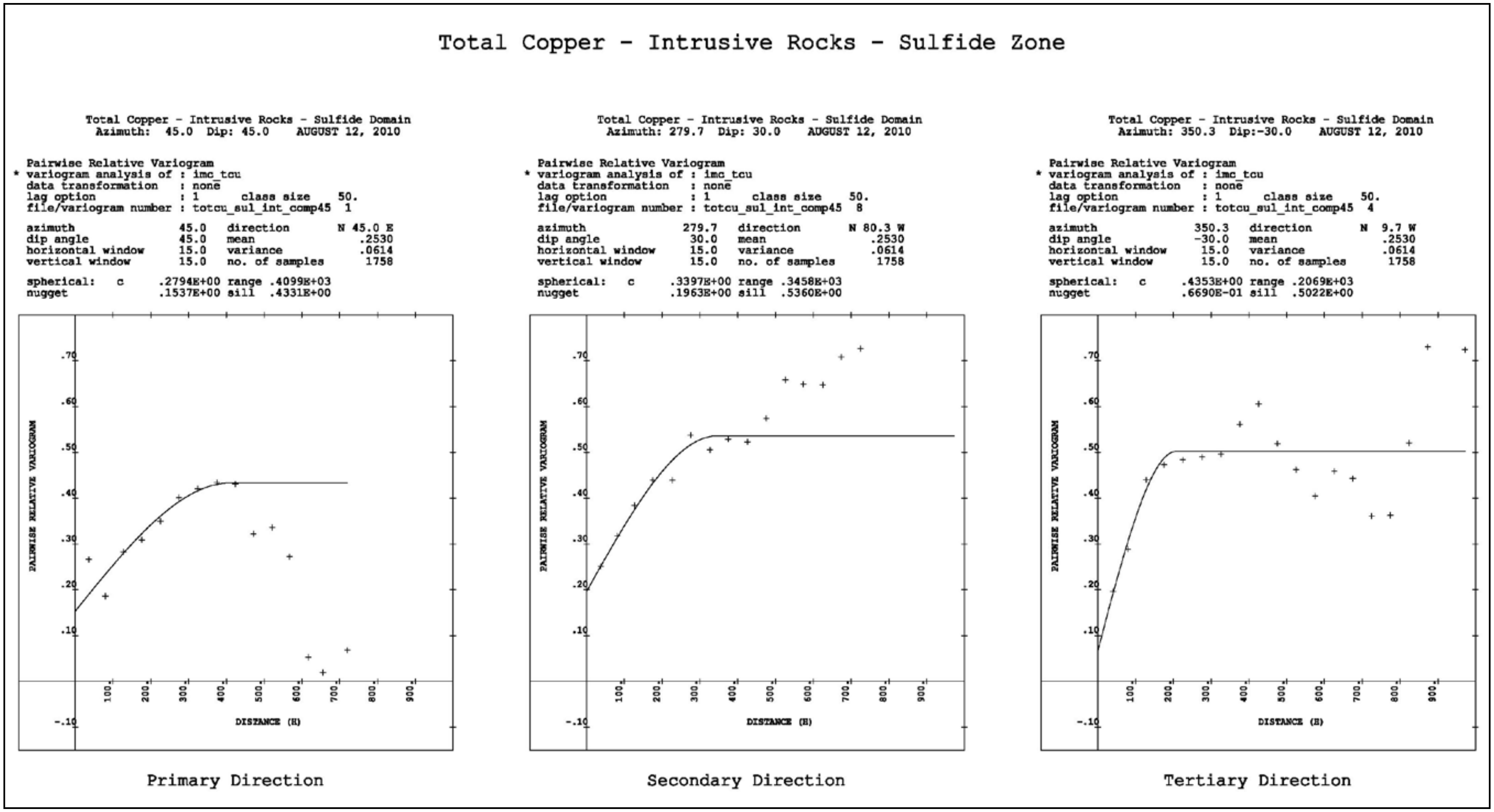


Figure 14-9: Total Copper Variograms Intrusive Rocks in Sulfide Zone

14.2.9.2 Copper

For total and soluble copper, the oxide/sulfide boundary was used as a hard boundary; i.e. sulfide domain blocks were only estimated with sulfide domain composites and oxide/mixed domain blocks were only estimated with oxide/mixed composites. The oxide/mixed boundary was not a hard boundary however. Actually, the oxide/mixed block designations were completed after grade estimation, based on soluble copper to total copper block grades.

A flat, circular search of 200m by 200m by 50m vertical was used for the estimation of total copper and soluble copper grades in the oxide/mixed domain.

Based on the variogram analysis of total copper for host rocks in the sulfide zone, the primary axis appears to be orientated with an azimuth of 120^o (S60^{o}E) and a plunge of 65^o and the secondary axis is oriented with an azimuth of 300^o (N60^{o}W) with a plunge of 25^o. The tertiary axis is oriented with azimuth of 30^o (N30^{o}E) with no plunge. Note that this alignment is consistent with the NW-SE trend in the area. In GSLIB convention, the rotation angles are 120^o, -65^o, and 0^o, representing rotation of major axis, plunge of major axis, and rotation of secondary axis, etc.

For pre-mineral intrusive rocks in the sulfide, the variogram analysis indicates a primary axis orientation of N45^{o}E with a plunge of 45^o. The secondary axis is orientated about N80^{o}W with a plunge of approximately 30^o. The GSLIB convention angles are 45^o, -45^o, and 45^o.

Due to the relatively small size and complex orientations of the post mineral intrusive rocks the search radius was opened up to 200m by 200m by 200m to match post mineral intrusive composites to blocks.

The Mitsubishi, Benguet, and Echo Bay total and soluble copper assays were used for block grade estimation. Soluble copper was estimated with the same search parameters as total copper in all cases. Figure 14-10 shows an example of the block grade estimations on a cross section.

14.2.9.3 Gold

Gold was estimated with the same search orientations as sulfide zone copper for host and pre-mineral intrusive rocks. The oxide/sulfide surface was not considered as a hard boundary for gold. The search orientations for gold in the oxide zone were also orientated according to directions established for primary copper.

Benguet gold assays were not used, except for the sample intervals that were re-assayed by Echo Bay. Figure 14-11 shows an example of block grade estimations on a cross-section.

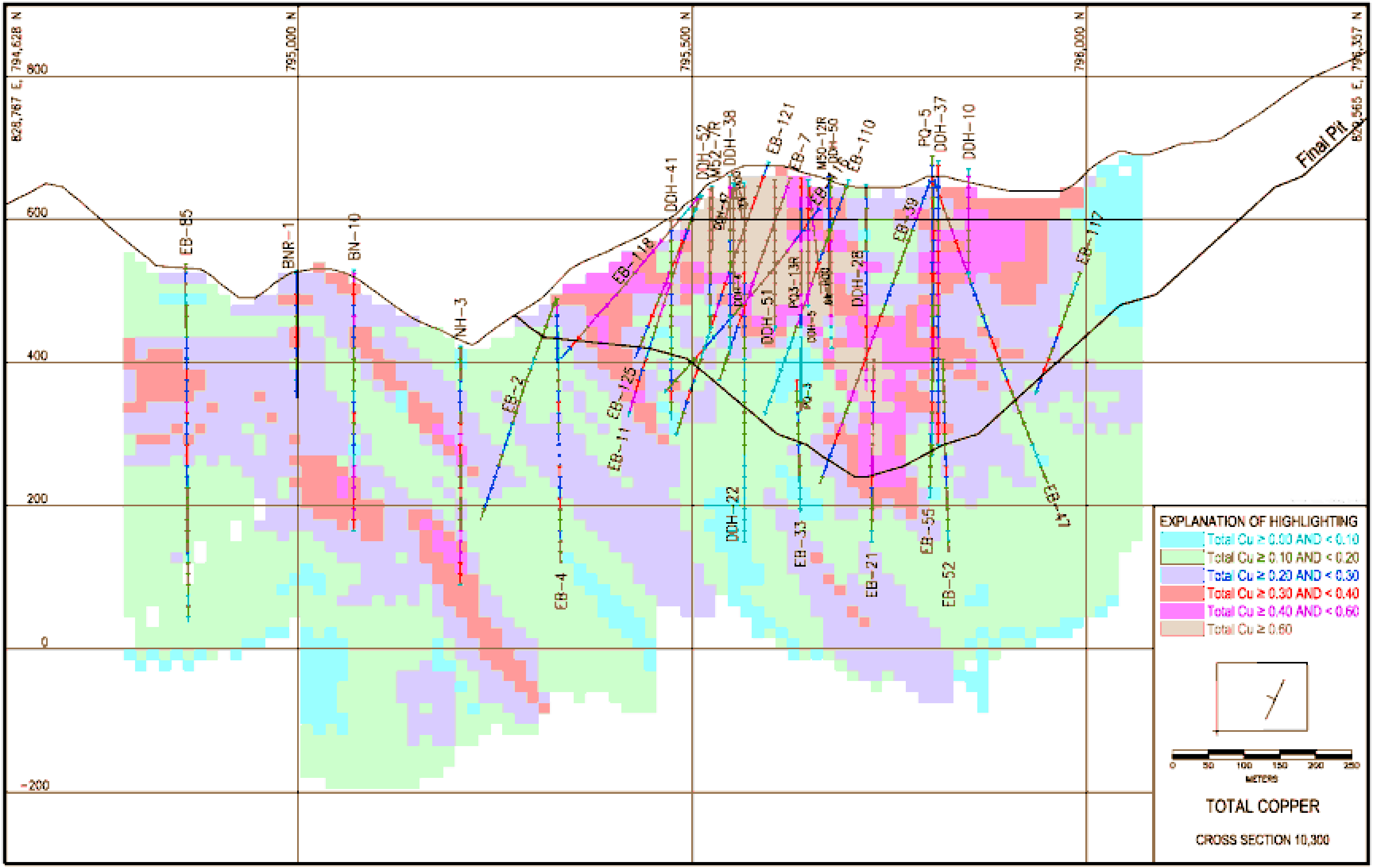


Figure 14-10: Copper Grades on Section 10,300

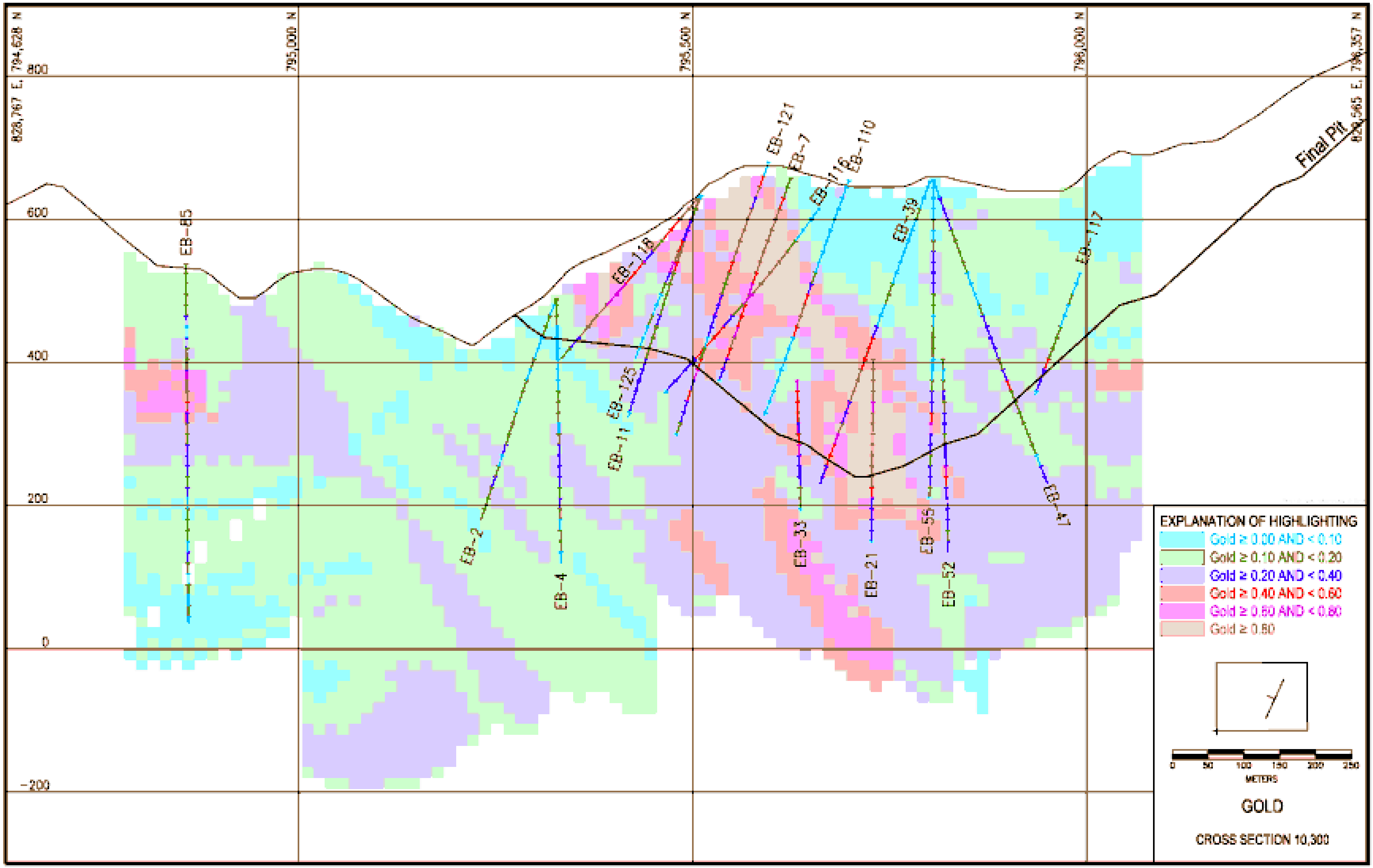


Figure 14-11: Gold Grades on Section 10,300

14.3 RESOURCE CLASSIFICATION

The number of composites and the average distance to the composites were stored in the block model and used for resource classification. This was completed for both the total copper and gold grade estimation. The following procedure was then used to establish the resource classification for each:

All blocks with a grade estimate were set to inferred resource.

The following blocks were then upgraded to indicated resource.

Blocks estimated with 10, 11, or 12 composites and average distance $\leq$ 150m

Blocks estimated with 7, 8, or 9 composites and average distance $\leq$ 125m

Blocks estimated with 4, 5, or 6 composites and average distance $\leq$ 100m

The following blocks were then upgraded to measured resource.

Blocks estimated with 7 or more composites and average distance $\leq$ 75m

Note that the block grade estimation limited the number of composites to three per hole, thus 4+ composites indicates a minimum of two holes, 7+ composites a minimum of three holes, and 10+ composites a minimum of four holes.

This procedure was completed independently for total copper and gold. The final block classification was taken as the lower confidence of the two classifications; i.e. if the classification of a block was measured based on total copper and indicated based on gold the final classification was indicated resource.

Though copper and gold grade estimates were completed by inverse distance, IMC also completed an ordinary kriging estimate for total copper and gold to obtain a relative kriging standard deviation to assist in establishing the resource classification. Figure 14-12 show a cross tabulation of blocks by number of composites and average distance for total copper. The cells of the figure show the number of blocks in the cell and also the average kriging standard deviation for the blocks. Measured blocks generally correspond to a relative kriging standard deviation less than 0.45 and indicated blocks less than 0.73. The standard deviations are relative because they were calculated with a variogram with the sill normalized to 1 and a nugget value of 0.15. Figure 14-13 shows a similar cross tabulation for gold. Figure 14-14 shows the resource classification on a cross section.

Crosstabulation of Number of Composites and Average Distance
for Total Copper Kriging - With Kriging Standard Deviation
Crosstabulation of Variables: ncmp_cu and avgdst_cu

Number of Composites

Average Distance (m)

n_stdev stdev_cu	= 1	= 2	= 3	= 4	= 5	= 6	= 7	= 8	= 9	= 10	= 11	= 12	Row Total
	1 Hole			2 Holes			3 Holes			4 Holes			
>= .0 < 25.0	26 .55	14 .41	319 .38	0 .00	0 .00	2 .41	0 .00	0 .00	0 .00	0 .00	0 .00	48 .24	409 .38
>= 25.0 < 50.0	110 .86	125 .79	1478 .60	58 .50	44 .50	243 .40	15 .35	8 .34	9 .33	2 .32	0 .00	8532 .33	10624 .38
>= 50.0 < 75.0	385 1.11	449 .96	2931 .73	340 .58	273 .56	1161 .50	169 .45	137 .43	257 .43	86 .44	66 .44	38464 .39	44718 .43
>= 75.0 < 100.0	572 1.15	620 1.02	4651 .83	839 .72	801 .66	3554 .60	708 .58	741 .55	2390 .52	708 .50	709 .48	47866 .47	64159 .52
>= 100.0 < 125.0	739 1.17	943 1.04	5901 .92	1340 .82	1594 .77	6141 .72	1663 .70	1829 .67	5582 .63	1902 .61	2020 .60	31524 .58	61178 .66
>= 125.0 < 150.0	1169 1.19	1262 1.07	7334 1.00	1750 .90	2024 .87	6834 .84	2092 .80	2485 .77	5916 .75	2204 .73	2407 .70	15153 .69	50630 .81
>= 150.0 < 175.0	1744 1.20	2019 1.10	7723 1.06	1949 .96	1874 .94	4063 .93	1134 .88	1204 .87	1981 .86	696 .82	675 .81	1962 .80	27024 .97
>= 175.0 < 200.0	3488 1.22	2981 1.14	3624 1.10	584 1.00	460 .99	448 .98	82 .93	80 .92	47 .92	24 .91	17 .92	20 .94	11855 1.13
Column Total	8233 1.19	8413 1.09	33961 .94	6860 .87	7070 .84	22446 .77	5863 .75	6484 .73	16182 .68	5622 .67	5894 .65	143569 .49	270597 .66

MEASURED

INDICATED

INFERRED

INDICATED

INFERRED

Figure 14-12: Resource Classification for Total Copper

Crosstabulation of Number of Composites and Average Distance
for Gold Kriging - With Kriging Standard Deviation
Crosstabulation of Variables: ncmp_au and avgdst_au

Number of Composites

Average Distance (m) n_stdev stdev_au	= 1	= 2	= 3	= 4	= 5	= 6	= 7	= 8	= 9	= 10	= 11	= 12	Row Total
	1 Hole			2 Holes			3 Holes			4 Holes			
>= .0 < 25.0	0 .00	3 .41	586 .37	2 .28	0 .00	0 .00	0 .00	0 .00	3 .43	0 .00	0 .00	0 .00	594 .37
>= 25.0 < 50.0	31 1.14	72 .95	2419 .57	89 .47	45 .45	155 .43	7 .36	2 .69	37 .60	6 .35	7 .33	1090 .30	3960 .50
>= 50.0 < 75.0	284 1.19	384 .98	4008 .72	329 .58	179 .58	1443 .49	144 .47	107 .47	435 .44	58 .42	42 .46	16371 .38	23784 .47
>= 75.0 < 100.0	425 1.18	576 1.00	6060 .82	738 .71	580 .65	4023 .61	710 .57	741 .55	3736 .51	698 .50	630 .48	47437 .45	66354 .52
>= 100.0 < 125.0	660 1.18	899 1.04	7276 .92	1311 .80	1557 .77	8443 .72	1511 .70	1780 .66	7734 .62	1710 .60	2030 .57	33465 .56	68376 .65
>= 125.0 < 150.0	1132 1.19	1233 1.07	8799 1.00	1685 .90	2174 .86	9883 .83	1827 .79	2207 .77	6827 .75	1694 .71	2089 .69	14790 .68	54340 .81
>= 150.0 < 175.0	1614 1.20	1909 1.10	8767 1.06	1797 .96	1926 .93	5586 .92	1021 .87	1155 .85	2293 .84	522 .82	596 .80	1782 .78	28968 .97
>= 175.0 < 200.0	3348 1.22	2909 1.14	4162 1.10	514 1.00	477 .98	530 .97	69 .92	64 .90	57 .89	8 .87	10 .89	14 .84	12162 1.13
Column Total	7494 1.21	7985 1.09	42077 .92	6465 .86	6938 .84	30063 .77	5289 .74	6056 .72	21122 .66	4696 .65	5404 .63	114949 .50	258538 .69

Figure 14-13: Resource Classification for Gold

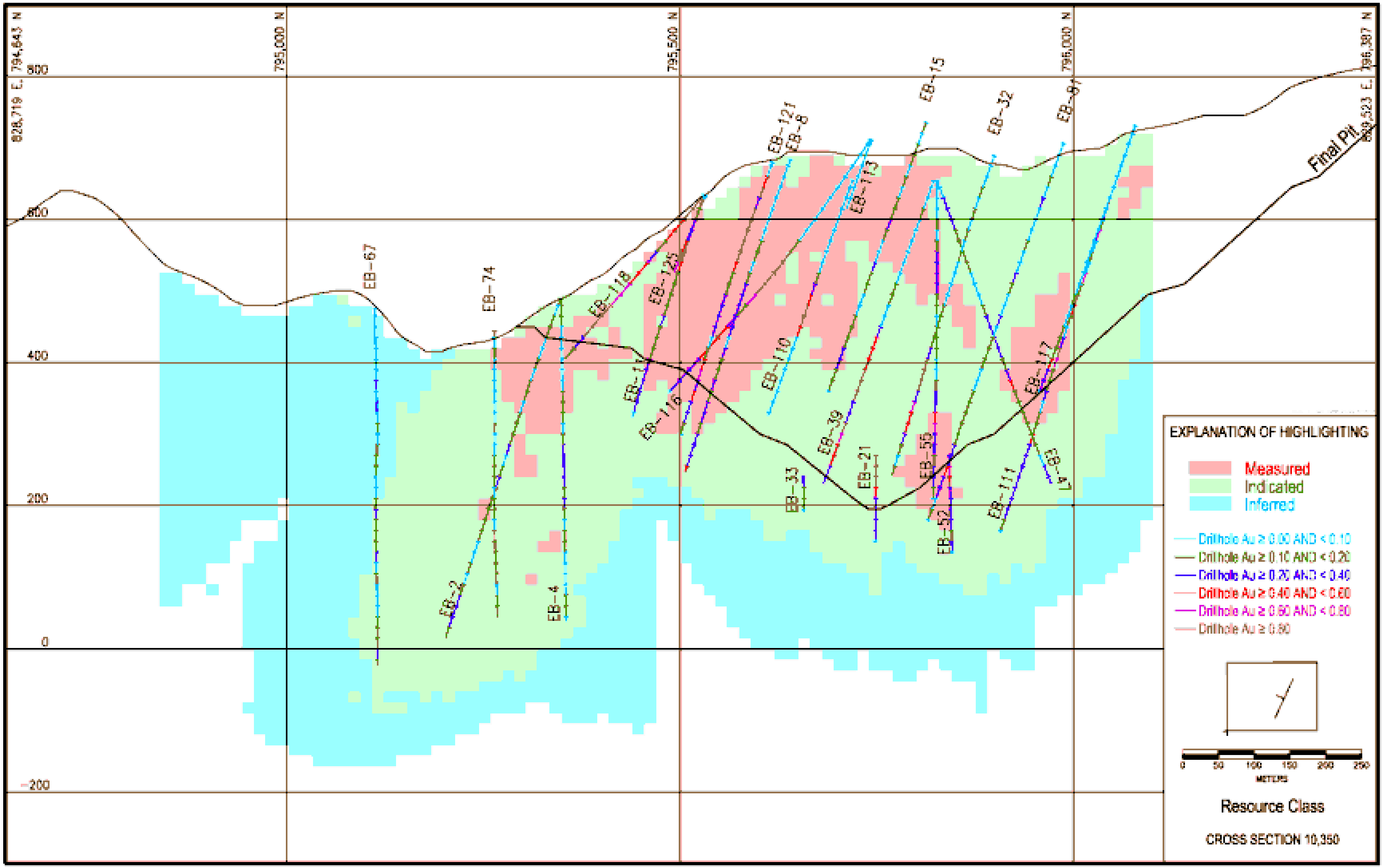


Figure 14-14: Cross Section 10350 Showing Resource Classification

14.4 BULK DENSITY

Specific gravity measurements, by the water immersion method, were performed on 100 core samples selected for the RMMI check assay program. Figure 14-5 shows an xy plot of the specific gravity measurement versus soluble copper to total copper ratio by the various rock types. It can be seen that specific gravities are lower for the samples with a soluble copper to total copper ratio greater than approximately 40%, which would also correspond to oxide/mixed ore types. Table 14-5 shows basic statistics of the data by rock type and also by higher versus lower soluble copper to total copper ratio.

The values shown on Table 14-5 were incorporated into the model as dry bulk densities without additional adjustments. Oxide and mixed blocks were assigned bulk density values of 2.41 t/m^3 and 2.36 t/m^3 for host rocks and intrusive rocks, respectively. Primary (sulfide) blocks were assigned bulk densities of 2.54 t/m^3 and 2.47 t/m^3 for host and intrusive rocks, respectively. Breccia blocks were assigned a bulk density of 2.47 t/m^3. IMC assigned overburden blocks a bulk density of 2.0 t/m^3.

The Echo Bay study was based on bulk densities of 2.69 t/m^3 for oxide and 2.76 t/m^3 for sulfide, which are considerably higher than the new measurements. The report says these were measurements completed by Lakefield using picnometer readings of the metallurgical samples. The report did not indicate how many measurements were completed. This is effectively the specific gravity of a ground pulp, which is of interest for ore processing design, but would generally not be considered an appropriate measurement method for ore reserve calculations because small fractures and voids are removed.

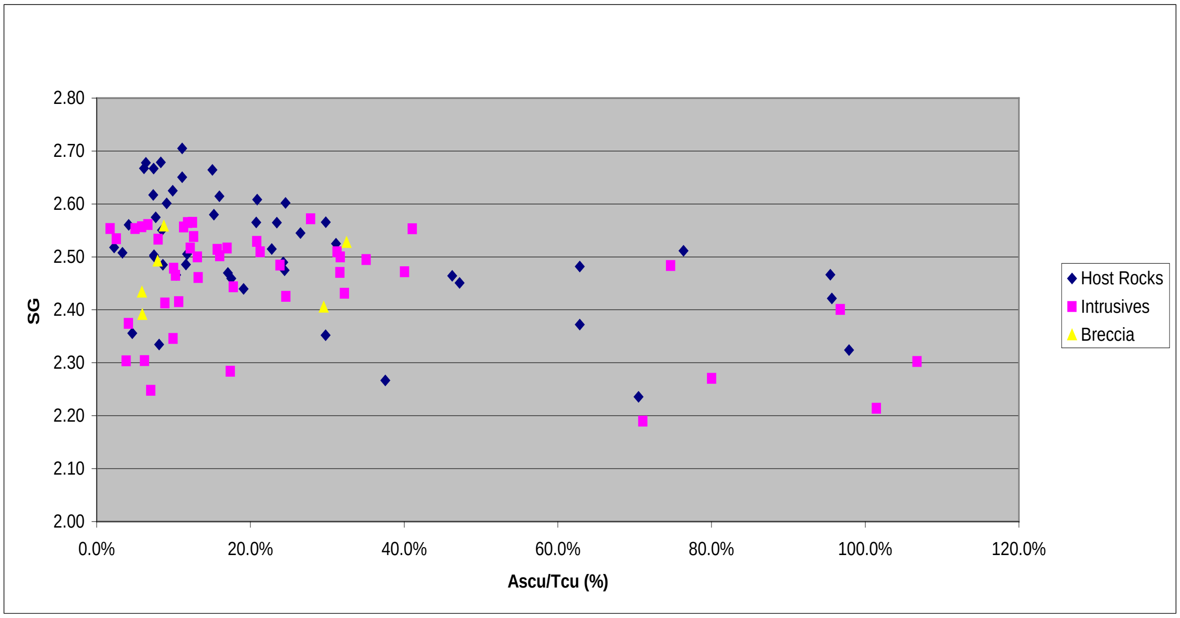


Figure 14-15: Specific Gravity versus Ascu/Tcu Ratio

Table 14-5: Specific Gravity Measurement by Rock Type

Code	Description	Number	Mean	Std Dev	Min	Max
20	Host Rocks	49	2.52	0.109	2.24	2.70
20	Ascu/Tcu < 40%	40	2.54	0.100	2.27	2.70
20	Ascu/Tcu > 40%	9	2.41	0.088	2.24	2.51
30	Intrusives	45	2.45	0.105	2.19	2.57
30	Ascu/Tcu < 40%	37	2.47	0.086	2.25	2.57
30	Ascu/Tcu > 40%	8	2.36	0.135	2.19	2.55
50	Breccia	6	2.47	0.068	2.39	2.56
ALL	All Rock Types	100	2.48	0.109	2.19	2.70

14.5 IMPACT OF VARIOUS DRILLING CAMPAIGNS

IMC reviewed the various drilling campaigns to determine the data that was appropriate for use in resource modeling. In particular, the Echo Bay study indicated that the Benguet gold assays were biased high compared with the Echo Bay results.

First, IMC tested the various drilling campaigns for total and soluble copper. Table 14-6 summarizes the results. Case 1 shows an ore tonnage and copper grades for the new resource model developed using all available copper assays. The tabulation is inside a pit design IMC developed for an August 2009 due diligence review of the project. The tabulations are at 0.2% total copper cutoff grades and include only measured and indicated resource blocks. This model resulted in 435.6 million tons at 0.358% total copper and 0.105% soluble copper.

Case 2 shows the results of removing the Benguet core holes from the estimation. Ore tons and total copper results are very similar to Case 1. The soluble copper grade increased 4.8% to 0.110%. This implies the Benguet core soluble copper grades tended to be lower than Echo Bay results (since removing them increased the grade), but IMC deemed the difference is not significant.

Cases 3 and 4 show the results of removing the Benguet RC data and Mitsubishi data, respectively. The differences with Case 1 are not significant. From this, IMC concluded that all the copper data was acceptable for the grade estimations.

For the 2009 due diligence review, IMC completed a similar analysis for gold, which was not repeated for this current study. The analysis showed that excluding gold assays for Benguet core holes decreased the gold grade 9.7%. The Benguet core data is higher than Echo Bay data for gold since removing it caused a significant reduction in grade. From this 2009 analysis, IMC determined that the Benguet gold assays would not be used for resource modeling. As noted previously, Benguet samples re-assayed for gold by Echo Bay were used.

Recall that the Mitsubishi drilling did not have any gold assays.

Table 14-6: Comparison of Various Drilling Campaigns for Copper

Case	Description	Ore Ktonnes	Tot Cu (%)	Sol Cu (%)
1	All Drilling Campaigns	435,620	0.358	0.105
2	Excluding Benguet Core Holes	430,765	0.360	0.110
	%Difference Versus Case 1	-1.1%	0.6%	4.8%
3	Excluding Benguet RC Holes	435,609	0.360	0.106
	%Difference Versus Case 1	0.0%	0.6%	1.0%
4	Excluding Mitsubishi Holes	435,978	0.356	0.105
	%Difference Versus Case 1	0.1%	-0.6%	0.0%

Note: Tabulation at 0.2% total copper cutoff inside pit designed for August 2009 Due Diligence Review. Only measured and indicated resource blocks.

15 MINERAL RESERVE ESTIMATES

15.1 MINERAL RESERVE

The mine and plant production schedules define the mineral reserve for a mining project. Table 15-1 presents the mineral reserve for the King-king Project based on the production schedules presented in the previous section.

The mineral reserve amounts to 617.9 million tons at 0.300% total copper and 0.395 g/t gold. For this reserve estimate, measured mineral resource was converted to proven mineral reserve and indicated mineral resource was converted to probable mineral reserve.

Table 15-1: Mineral Reserve

Reserve Classification	Ktonnes	Tot Cu (%)	Sol Cu (%)	Gold (g/t)	NSR (US$)
Proven Mineral Reserve					
Heap Leach Ore	17,791	0.340	0.197	0.132	16.53
Oxide Mill Ore	21,674	0.514	0.328	0.849	45.36
Sulfide Mill Ore	52,942	0.305	0.044	0.543	24.92
Low Grade Mill Ore	6,734	0.184	0.027	0.218	10.80
Total Proven Reserve	99,141	0.349	0.132	0.514	26.92
Probable Mineral Reserve					
Heap Leach Ore	77,373	0.305	0.172	0.145	14.81
Oxide Mill Ore	45,440	0.393	0.259	0.745	35.30
Sulfide Mill Ore	345,715	0.288	0.037	0.398	20.48
Low Grade Mill Ore	50,247	0.191	0.023	0.211	10.93
Total Probable Reserve	518,775	0.290	0.075	0.373	20.01
Proven/Probable Mineral Reserve					
Heap Leach Ore	95,164	0.311	0.177	0.143	15.13
Oxide Mill Ore	67,114	0.432	0.281	0.779	38.55
Sulfide Mill Ore	398,657	0.290	0.038	0.417	21.07
Low Grade Mill Ore	56,981	0.190	0.023	0.212	10.91
Total Prov/Prob Reserve	617,916	0.300	0.084	0.395	21.12

15.2 ECONOMICS PARAMETERS FOR PIT DESIGN

Table 15-2 summarizes the economic parameters for mine design and scheduling. It should be noted that these are initial estimates used to initiate mine design and scheduling and are not the final economics developed for this NI 43-101 Technical Report. Note that two sets of copper/gold commodity prices are shown on Table 15-2. The final pit design was based on commodity prices of $2.50 per pound copper and $833 per troy ounce gold. Commodity prices for mine production scheduling and ore routing are $3.00 per pound copper and $1,000 per troy ounce gold. A mine design was completed at prices of $3.00 copper and $1,000 gold, but the incremental waste to ore ratio was high, and a slightly smaller pit was chosen for this study.

Mining costs are estimated by IMC based on a review of actual current costs at four large open pits, two in Mexico and two in the southwest US. The $1.00 base case valueless rock haulage cost is based on the perceived high cost of hauling to the west Sagittarius Alpha Realty

Corporation (SARC) VRMA. A mine replacement capital cost of $0.15 was also included in the mining costs for design purposes. This is relatively a common practice in base metal operations to prevent mining of large quantities of marginal material that will not recover the cost of equipment consumed in the mining.

Process and G&A costs were provided to IMC by SAGC personnel. The G&A cost is based on a fixed cost of US $10 million per year and 21.9 million ore tons per year. Table 15-3 shows that the smelting, refining, and freight cost per pound of copper in concentrate is approximately $0.35 per pound. This is based on a concentrate grade of 22%, smelting charges of $80 per ton and refining charges of $0.08 per pound. Transport is $40 per wet ton. The SX-EW cost for cathode copper produced on site is estimated at $0.12 per pound.

Table 15-2: Economic Parameters for Pit Design

Parameter	Units	Oxide/Mix Heap Leach	Oxide/Mix Float/Agit	Sulfide Float/Agit	Waste
Copper/Gold Price - Mine Design	(US$)	2.50 / 833	2.50 / 833	2.50 / 833	
Copper/Gold Price - Production Scheduling	(US$)	3.00/ 1000	3.00 / 1000	3.00 / 1000	
Mining Cost Per Tonne					
Base Mining Cost Without Haulage	(US$)	0.750	0.750	0.750	0.750
Base Haulage Cost	(US$)	0.750	0.500	0.500	1.000
Mine Replacement Capital Per Tonne	(US$)	0.150	0.150	0.150	0.150
Subtotal Mining	(US$)	1.650	1.400	1.400	1.900
Incremental Haulage Per Bench Below 300m	(US$)	0.025	0.025	0.025	0.025
Process Cost Per Ore Tonne					
Crushing, Grinding, Flotation	(US$)	N.A.	5.000	5.000	
Agitated Leach	(US$)	N.A.	2.300	2.300	
Tailings	(US$)	N.A.	0.840	0.840	
Heap Leach	(US$)	2.737	N.A.	N.A.	
Processing Cost Per Ore Tonne	(US$)	2.737	8.140	8.140	
G&A Cost Per Ore Tonne	(US$)	0.100	0.460	0.460	
Process Recoveries					
Process Recovery of Copper (Average)	(%)	73.7%	81.7%	74.2%	
Process Recovery of Gold (Average)	(%)	N.A.	74.2%	70.6%	
Conventional Smelting/Refining					
Smelting/Refining Payable for Copper	(%)	N.A.	95.5%	95.5%	
Smelting/Refining Payable for Gold	(%)	N.A.	97.0%	97.0%	
SRF Cost Per Pound Copper	(US$)	N.A.	0.348	0.348	
Site Solvent Extraction/Electrowinning					
Payable Copper	(%)	100%	100%	100%	
SXEW Per Pound Copper	(US$)	0.120	0.120	0.120	
Cathode Freight/Insurance	(US$)	0.040	0.040	0.040	
Gross Royalty	(%)	3.0%	3.0%	3.0%	

Table 15-3: Smelting, Refining, and Freight

Parameter	Units	Oxide	Sulfide
Concentrate Copper Grade	(%)	22.0	22.0
Payable Pounds/Tonne (1% Deduct)	(lbs)	463.0	463.0
Treatment Per Dry Tonne	(US$)	80.00	80.00
Refining Per Pound	(US$)	0.08	0.08
Transport Per Wet Tonne	(US$)	40.00	40.00
Moisture Content	(%)	10.0%	10.0%
Payable Percentage of Copper	(%)	95.5%	95.5%
Treatment Per Pound	(US$)	0.173	0.173
Refining Per Pound	(US$)	0.080	0.080
Transport Per Pound	(US$)	0.095	0.095
Total SRF Per Pound Copper	(US$)	0.348	0.348

Plant recoveries were incorporated into the model on a block by block basis based on equations provided by AMEC.

The copper flotation recovery was estimated as:

$$\text{Flotation recovery} = (\text{nsol}/\text{totcu})\text{x}(151.02\% + 36.0385\% \text{ x } \ln(\text{nsol})) - 14.442\% \text{ x } \ln(22) + 12.63\%$$

Where:

- Totcu = total copper
- Solcu = soluble copper
- Nsol = totcu – solcu = non-soluble copper
- Ln is natural log
- 22 is estimated concentrate grade
- The flotation recovery was capped at 95%.

Copper recovery in the agitated leach circuit was then estimated as:

$$\text{Agitation recovery} = 99.77\% \text{ x solcu/totcu} + 10.7\% \text{ x } (1 - \text{Flotation Recovery}/100\% - \text{solcu/totcu})$$

Total recovery is the sum of flotation and agitation recovery. Flotation, agitation, and total recovery are stored in the block model.

Gold recovery from flotation and gravity was estimated as:

$$\text{Gold recovery} = 75\% + 5.98\% \text{ x } \ln(\text{Gold})$$

Where:

- Gold is the head grade in grams per ton.
- This was also capped at 95%.

Heap leach recovery of copper for oxide ores was based on a constant tail of 0.08% total copper, i.e.:

Leach Recovery = 100% (totcu – 0.08) / totcu, (capped at 85%)

The recoveries shown on Table 15-2 are averages for the ore types in the plant production schedule.

To facilitate economic calculations, an NSR value was calculated and stored in the model on a block by block basis. NSR is gross revenue less smelting, refining, freight, and also the royalty and is in US$ per ore ton. For cathode copper the NSR is gross revenue less SX-EW and the royalty. For metal in concentrate, the NSR calculations are as follows:

NSR_cu_conc = ($3.00-$0.348) x 0.955 x 0.97 x 22.046 x totcu x (flot recov/100)

NSR_au_conc = $1000 x 0.97 x 0.97 x gold x (gold flot recov/100) / 31.103

The 0.97 term in both equations (second one in the gold equation) accounts for the royalty.

For copper in cathode:

NSR_cu_agit = ($3.00 - $0.16) x 1.0 x 0.97 x 22.046 x copper x (agit recov/100)

The final block NSR for mill ore is the sum of the terms:

NSR Mill = NSR_cu_conc + NSR_cu_agit + NSR_au_conc

The NSR for potential heap leach ore is:

NSR Leach = ($3.00 – $0.16) x 0.97 x 22.046 x totcu x (leach recovery/100)

Oxide ores were routed to the mill or heap leach based on maximum profit after processing and G&A, i.e. for oxide to be routed to the mill:

NSR Mill - $8.60 > NSR Leach - $2.837

Since all the recoveries and all costs except mining, processing, and G&A are incorporated into the NSR calculation, the internal cutoff grade for mill ore is the processing plus G&A cost of $8.60 per ton and the internal cutoff grade for heap leach ore is $2.837 per ton. This assumes mining is a sunk cost for blocks that have to be removed from the pit. Breakeven cutoff is the processing, G&A, and mining cost of approximately $10.00 per ton for mill ore.

It should be noted that the above formulas were used for the pit design parameters. Updated metal recovery formulas were obtained recently after completion of the pit design and scheduling. The updated metal recoveries are for the copper concentrate, gold and copper agitated leach process. Metal recovery formula for heap leach process remains the same. The updated metal recovery formulas are applied to the metal production schedule presented in Section 16.

16 MINING METHODS

16.1 OPERATING PARAMETERS AND CRITERIA

The King-king mine will be a conventional open pit mine. Mine operations will consist of drilling holes with large diameter (32 to 46 cm) blast holes, blasting with either explosive slurries or ANFO (ammonium nitrate/fuel oil) depending on water conditions, and loading the ore onto large off-road trucks with large cable shovels and wheel loaders. Ore will be delivered to the primary crusher and valueless rock to the Valueless Rock Management Area (VRMA) facilities. There will also be a low-grade stockpile facility to store marginal ore material for processing at the end of commercial pit operations. There will also be a fleet of track dozers, rubber tired dozers, motor graders, and water trucks to maintain the working areas in the pit, VRMA area, and the roads.

The mine plan was developed to deliver ore at the rate of 100,000 tons per day, split between 40,000 tpd to the heap leach and 60,000 tpd to the mill. The mining rate will be approximately 178,000 tons per day. The heap leach process is expected to start 9-12 months before the mill starts, and finishes approximately 13 years into the project. The mill continues to process sulfide predominant ore until the end of the mine life.

16.2 GEOTECHNICAL AND HYDROLOGICAL CONSIDERATIONS

16.2.1 Slope Angles

AMEC Environment & Infrastructure (AMEC) has developed a range of credible overall slope angles for pit development at the King-king Project, which are commensurate with a scoping-level study. The slope study used information from drillhole data collected from five oriented core drillholes and three geohydrology drillholes placed in the predicted final pit walls. This study also used results from unconfined compressive strength (UCS) tests conducted on thirty selected intervals of oriented core from these five holes. Bench design and kinematic analyses are not included as part of the present study. A detailed open pit design and recommendations report, including bench design parameters, will be provided in a later phase to support the King-king Project Feasibility study. However, it should be noted that the interaction of the pit walls with major geologic structures such as faults and shear zones is not included in the present study, as the structural model for the King-king Project is still under development. The incorporation of such structures in the geotechnical pit design will be included for the Feasibility Study report. Therefore, the overall slope angles provided herein will be adjusted as needed upon completion of additional slope study.

The geotechnical design presented herein is based on analyses of the geotechnical drilling program completed in 2011 and the “Technical Report – Pursuant to National Instrument 43-101 of The Canadian Securities Administrators” prepared by IMC (King-king Copper-Gold Project – Mindanao, Philippines. Technical Report – Pursuant to National Instrument 43-101 of The Canadian Securities Administrators. Independent Mining Consultants Inc., 12 October 2010).

Rock mass classification based on RMR’$_{76}$ suggests that the proposed open pit will be excavated in a highly variable poor to fair rock mass. Borehole logs indicate several “no core” recovery zones which have not been associated with faults or shear zones. It is strongly recommended to try to reconcile these zones with the structural model.

In terms of rock strength, the rock can be classified as Strong to Very Strong as the average unconfined compressive strength grouped by geotechnical domain ranged between 100 MPa and 150 MPa. However, the results are quite variable as demonstrated by relatively high coefficients of variation. In order to reduce the uncertainty related to the rock strength, additional unconfined compressive strength testing is strongly recommended. The number of samples that would need to be tested in order to attain the level of confidence required cannot be determined beforehand, as the number of test depends on the variability of the results; however, based on the statistical analysis performed by AMEC, the following minimum number of additional tests is suggested:

- Host Rock
  - Andesite: 6 to 10 tests;
  - Metavolcanic: 10 to 15 tests; and
  - Metasediment: 8 to 12 tests.
- Intrusive Rocks
  - Pre-Mineral Intrusions (Biotite Diorite Porphyry, Intra-Mineral Dacite Porphyry, Intra-Mineral Diorite Porphyry and Intra-Mineral Hornblende Diortie Porphyry: 10 to 15 tests
  - Post Mineral Intrusions (Dacite Porphyry, Diorite Porphyry and Hornblende Diorite Porphyry): 10 to 15 tests

Three geotechnical domains were defined based on the lithology model and the subsurface conditions:

- Overburden;
- Host Rocks; and
- Pre- and Post-Mineral Intrusions and Breccias.

The overburden includes cohesive, non-cemented, loose and granular material present near the surface. The Host Rocks include andesites, metavolcanics, and metasediments. The Pre- and Post-Mineral Intrusions and Breccias include Dacite Porphyry, Diorite Porphyry, Hornblende Diorite Porphyry, Biotite Diorite Porphyry, Intra-Mineral Dacite Porphyry, Intra-Mineral Diorite Porphyry and Intra-Mineral Hornblende Diorite Porphyry. For further development phases of the project, it is recommended to differentiate the Host Rocks by major rock types, as the average intact rock strengths for these rock types are quite different.

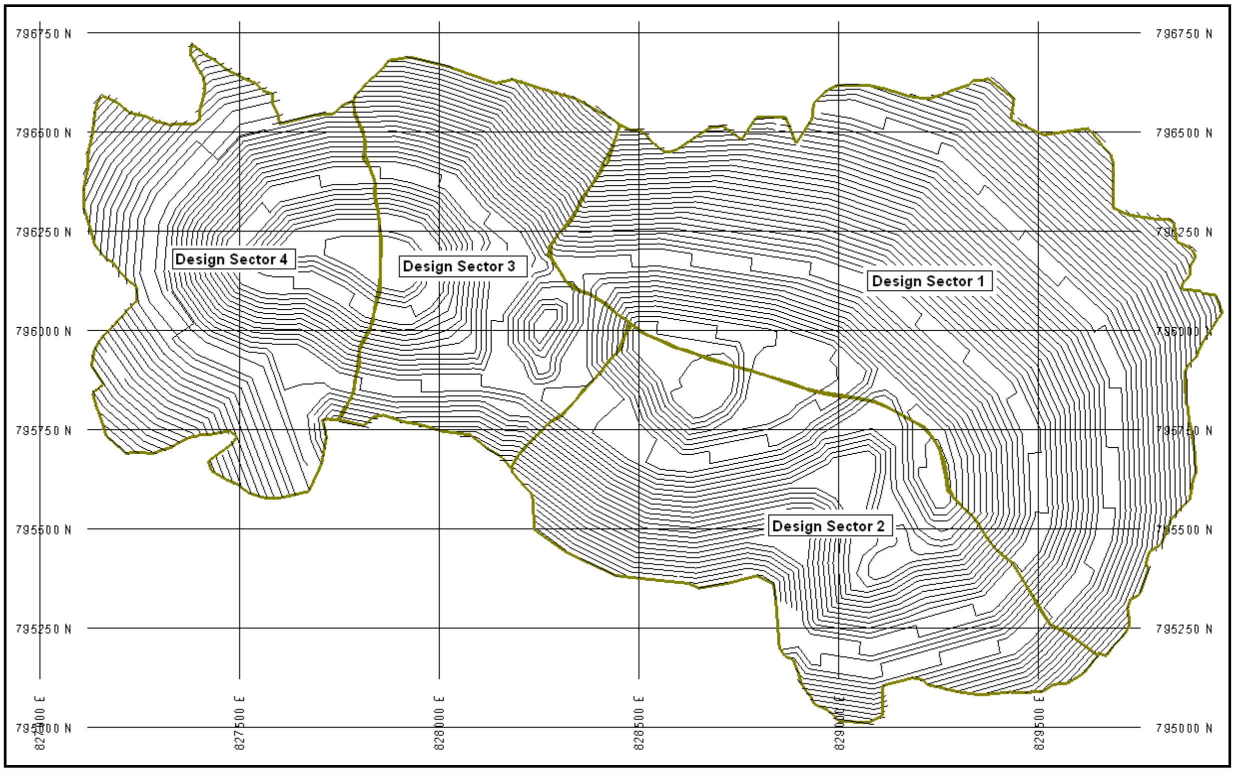
The preliminary nature of the scoping-level study limits the detail that can be incorporated into the geotechnical assessment at this time. The uncertainties related to the joint pattern and the structural model (neither of which have been considered in this study) suggest that a conservative factor of safety is appropriate; i.e. a static factor of safety between 1.3 and 1.5 with a probability of failure ≤5%, which corresponds to a High consequence of failure according to the acceptance criteria proposed by Read and Stacey (2009). As the geotechnical, structural and groundwater models are better defined by additional site investigation and laboratory tests, the factor of safety can be decreased to reflect the level of confidence attained. Moreover, once the mine plan is better developed and the locations of planned infrastructure and ramps related to the pit are defined, the acceptance criteria can be tailored to specific sectors of the pit. Of note, boreholes GT-05 and GT-06, which were not drilled, were located to cover critical project areas (the west end and southwest side of the pit). Thus, critical geotechnical information was not collected in these sectors and therefore geotechnical parameters had to be assumed by projection from areas where subsurface information was available. Further development phases of the project should target drilling in these sectors to validate the assumptions put forward in this section.

The proposed preliminary pit slope configuration and design sectors for which they apply are presented respectively in Table 16-1 and Figure 16-1.

Table 16-1: Proposed Preliminary Pit Slope Configuration

Design Sector	Maximum Inter-ramp Slope Angle (degrees)	Maximum Inter-ramp Slope Height (m)	Maximum Overall Slope Angle (Degrees)	Maximum Overall Slope Height (m)	Geotechnical Berm	Static Factor of Safety
1	**44**	**200**	40 Note 1	750	**A 20 m wide berm at each 200 m vertical interval**	**1.3 to 1.5 Note 4**
2			40 Note 2	500		
3			42 Note 3	600		
4			43	500		

Note 1: Provided that the water table is lowered below elevation 320 m
Note 2: Provided that the water table is lowered below elevation 250 m
Note 3: Provided that the water table is lowered below elevation 180 m
Note 4: For all sectors the probability of failure was less than 5%



Note: The pit shell was provided by IMC and is shown for reference only as it does not reflect the proposed pit slope angles

Figure 16-1: Design Sectors

16.2.2 Pit Diversion Design

A significant aspect of the King-king final pit design is that the Kingking River diversion will be integrated into the final pit wall. During mining phase 4, the first west mining phase, a temporary diversion channel will be developed during Year 8 of commercial operations. During mining phase 5, about Year 16, the final diversion channel will be constructed.

Hydrology and hydraulic calculations were performed to determine the required size for the Kingking River diversion. The Kingking River final diversion will be required at approximately Year 17 when the pit is expected to encroach on the river. The diversion will be constructed within a 75 meter wide pit bench. The 75 meter width will provide room for the diversion, safety berm(s) and access road(s). The layout of the channel should fit inside the pit bench as illustrated in Figure 16-2.

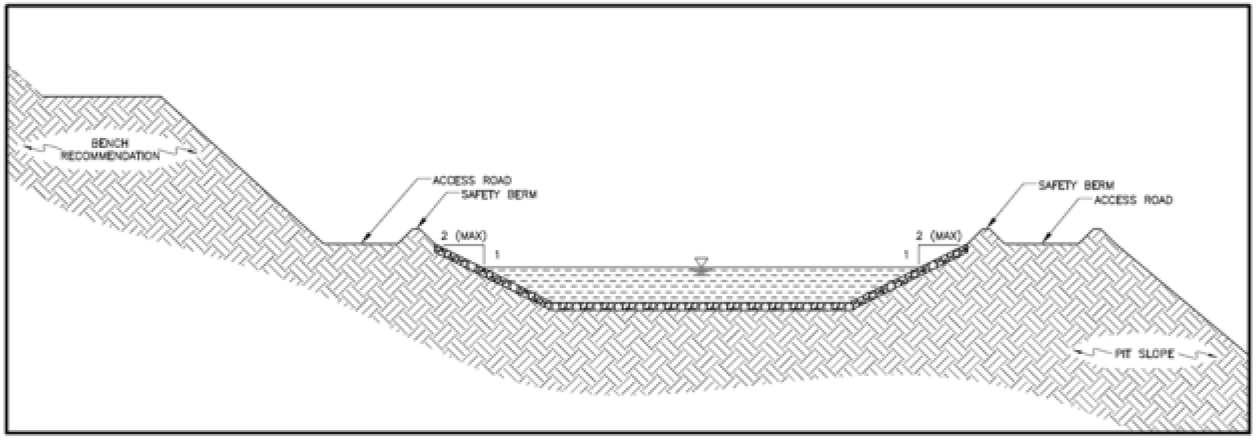


Figure 16-2: Pit Diversion Schematic

The diversion was designed to convey the peak flows associated with the 100-year, 24-hour storm event with one meter of freeboard. A design rainfall of 310 mm with a Natural Resource Conservation Service (NRCS) Type I storm distribution was used (AMEC 2012). A rainfall/runoff model was created using the software HEC-HMS to estimate the peak runoff resulting from the design rainfall. Extents of the Kingking River drainage tributary to the pit were delineated based on available topographic mapping and the total area contributing flow at the pit was found to be 32.28 square kilometers. An average NRCS Curve Number of 62 with hydrologic soil group C was assumed for the basin. One percent of the basin was assumed to be impervious and a lag time of 3.27 hours was used in the model based on an evaluation of the drainage. Results produced a peak flow rate of 181 m^3/s and a total flow volume of 5,819,000 m^3 for the 100-year storm.

A diversion channel was then sized to convey the design flow of 181 m^3/s. The diversion was assumed to be trapezoidal with 2.5H:1V (horizontal:vertical) side slopes. The overall change in elevation of the existing Kingking River channel from a point upstream of the proposed pit to downstream of the proposed pit is approximately 45 meters. The preliminary diversion channel layout provided to AMEC by IMC on 11 November 2011 shows an overall channel length of approximately 1,300 meters, producing a longitudinal slope of approximately 3.5%, which is slightly flatter than the natural channel slope in this area. The diversion is therefore assumed to be constructed at a uniform slope of approximately 3.5% for a length of 1,300 meters based on the pit layout. The channel was sized assuming uniform flow and a Manning's "n" value of 0.07. This relatively high Manning's "n" reflects a channel that is in irregular bedrock or riprap. One meter of freeboard is recommended. This additional freeboard will protect the pit from even higher flow events and provide protection from nuisance water that may result from waves or other flow irregularities.

Hydraulic calculations indicate that the diversion channel will require a base width of 20 meters and have a flow depth of 2.0 meters at the design flow rate. Including the recommended 1 meter of freeboard, the design results in a total channel depth of three (3) meters and a top width of 35 meters. With this geometry, 40 meters of the bench will be available for berms and access.

Due to the large peak discharge from the design storm, the channel is expected to experience high flow velocities. Velocities of approximately 3.5 m/s are predicted for the 100-year flow, which will be erosive. It is anticipated that the channel will be excavated into bedrock, which will prevent erosion.

16.3 MINING PHASES

The final pit design was based on a floating cone run at $2.50 per pound copper and $833 per troy ounce gold. Six mining phases were also designed to mine the pit from the initial starter pit to the final pit limits. The phase designs include haul roads and adequate working room for large mining equipment. The in-pit roads are 33m wide at a maximum grade of 10%. The width will accommodate trucks up to the 230 ton class such as Caterpillar 793 trucks. The designs were also based on the slope angle recommendations reported in the previous section 16.2.1.

Table 16-2 summarizes the proven and probable mineral reserve by mining phase. Heap leach ore is tabulated at an NSR cutoff grade of $2.84, internal cutoff grade based on the parameters on Table 15-2. The NSR cutoff grade for oxide and sulfide mill ore varies by year according to the cutoff grade strategy that will be discussed in Section 16.4.

Figure 16-3 shows the final pit design. It should be noted that it is smaller than Figure 16-1 from the AMEC slope stability design work in the northeast and east. There was originally a phase 7 that is the basis for Figure 16-1. Phase 7 was based on design prices of $3.00/lb copper and $1,000/troy ounce gold. Phase 7 also had a relatively high waste to ore ratio, so it has been excluded from this present study.

Figure 16-4 shows mining phases 1 and 2. Phase 1 is the larger, more central phase, and phase 2 is to the southeast of phase 1. The phases are based on a floating cone run at commodity prices of $1.10 copper and $415 gold. Table 16-2 shows phase 2 has a significant amount of high grade oxide mill ore.

Table 16-2: Mining Phases by Ore Type

Mining Phase	Heap Leach Ktonnes	Heap Leach Tot Cu (%)	Heap Leach Sol Cu (%)	Heap Leach Gold (g/t)	Oxide Mill Ktonnes	Oxide Mill Tot Cu (%)	Oxide Mill Sol Cu (%)	Oxide Mill Gold (g/t)	Sulfide Mill Ktonnes	Sulfide Mill Tot Cu (%)	Sulfide Mill Sol Cu (%)	Sulfide Mill Gold (g/t)	Waste Ktonnes	Total Ktonnes	Waste: Ore Ratio
1	43,181	0.332	0.186	0.137	38,576	0.396	0.278	0.804	91,572	0.320	0.042	0.467	86,399	259,728	0.50
2	17,698	0.358	0.244	0.120	9,368	0.949	0.633	0.764	6,216	0.531	0.051	0.553	8,478	41,760	0.25
3	17,560	0.318	0.166	0.120	727	0.631	0.143	0.402	139,010	0.325	0.035	0.293	127,618	284,915	0.81
4	14,160	0.188	0.091	0.217	20,110	0.224	0.109	0.714	67,675	0.221	0.035	0.507	98,869	200,814	0.97
5	33	0.280	0.135	0.282	367	0.173	0.062	0.670	55,196	0.182	0.035	0.449	167,722	223,318	3.02
6	2,532	0.271	0.096	0.138	3	0.420	0.338	0.281	93,932	0.251	0.032	0.335	167,470	263,937	1.74
TOTAL	95,164	0.311	0.177	0.143	69,151	0.422	0.274	0.767	453,601	0.278	0.036	0.392	656,556	1,274,472	1.06

Figure 16-3: Final Pit Design

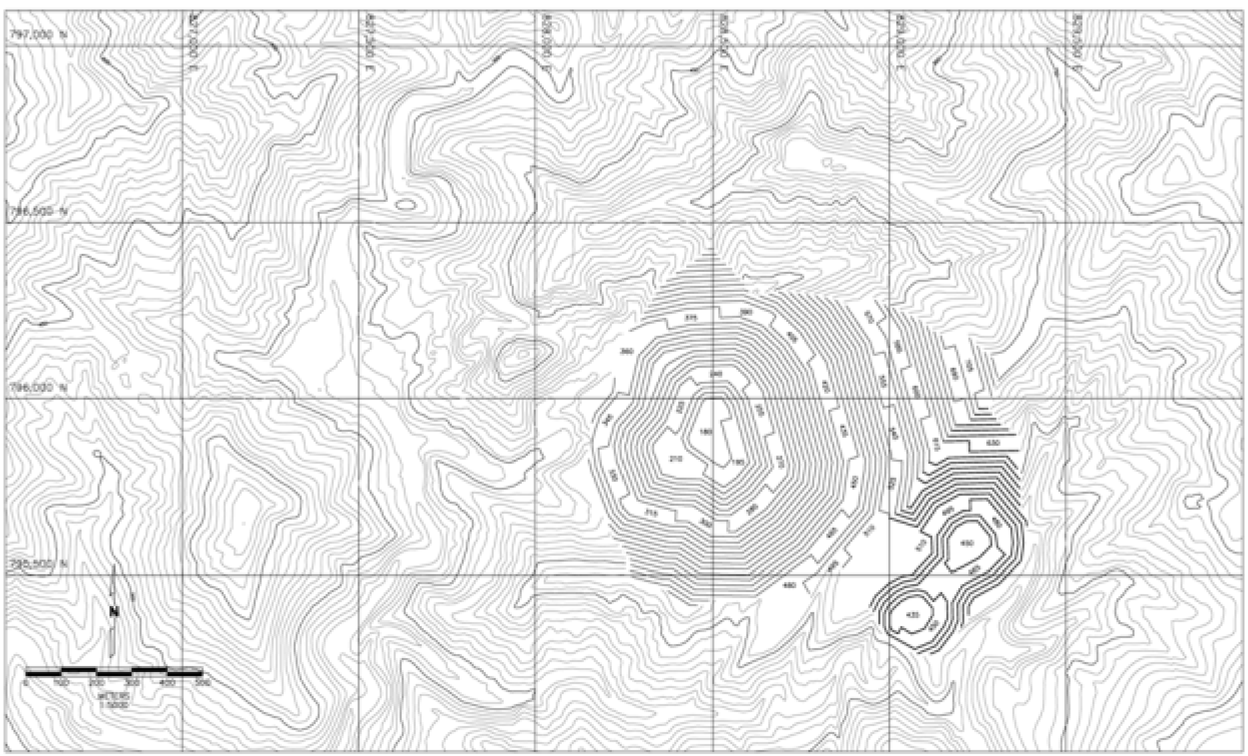


Figure 16-4: Phases 1 and 2

Figure 16-5 shows phases 3 and 4. Phase 3 is on the east side and phase 4 is the first phase on the west side of the pit. The phases are based on floating cone runs at commodity prices of $1.25 copper and $475 gold. Phase 3 includes significant amounts of heap leach ore and sulfide mill ore, but insignificant amounts of oxide mill ore. Phase 4 has significant amounts of heap leach and oxide mill ore, as well as sulfide mill ore.

Phase 5 is the west side of Figure 16-3 and phase 6 is the east side of Figure 16-3. These are based on the floating cone run at commodity prices of $2.50 copper and $833 gold.

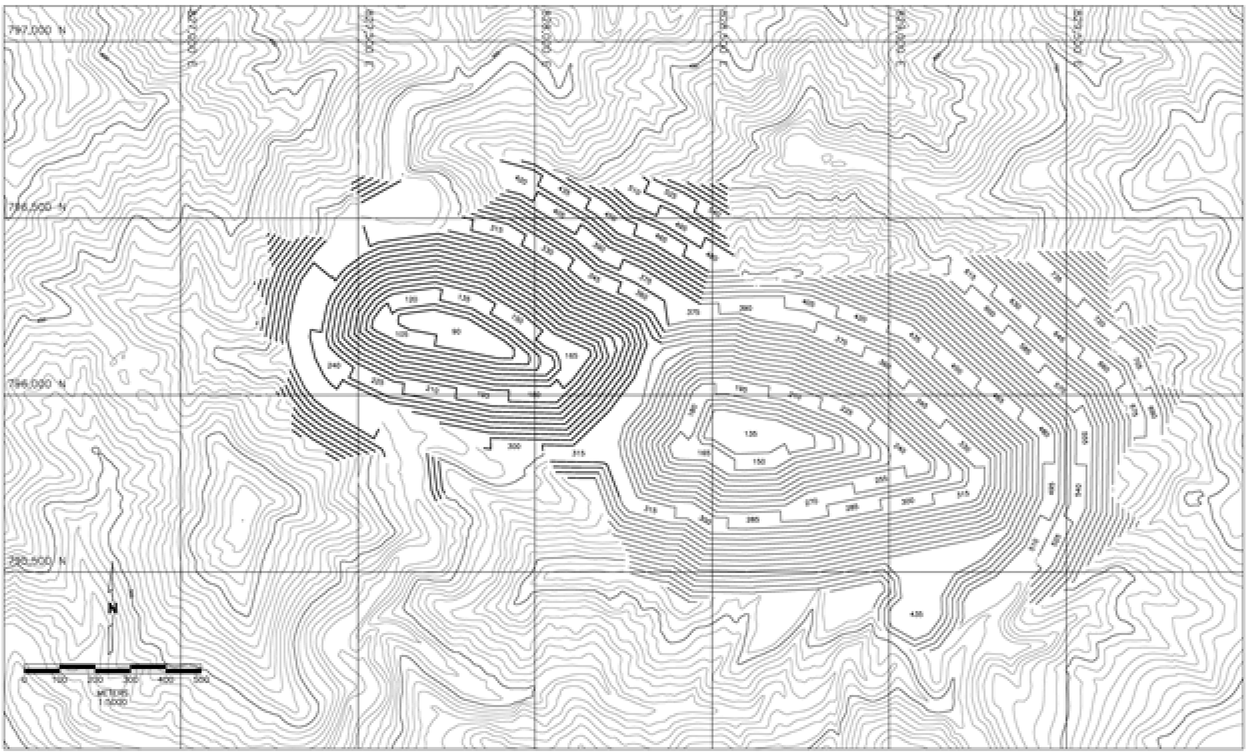


Figure 16-5: Mining Phases 3 and 4

16.4 MINE PRODUCTION SCHEDULE

A mine and plant production schedule was developed for the King-king Project based on the six mining phases. Ore is scheduled at a nominal rate of 60,000 tons per day or 21,900 ktpa though the rate varies by year according to ore hardness. Also, the lower gold grade oxide ores are routed to a heap leach facility that will recover copper only. This facility operates at 40,000 tpd or 14,600 ktpa for 13 years.

As previously discussed, for ore routing purposes, copper and gold plant recoveries and ore values expressed as NSR values were incorporated into the block model on a block by block basis. NSR is in US$ per ore ton and is calculated as revenue less smelting, refining, freight, and royalties for mill ore.

Table 16-3 shows the mine production schedule. The top section of the table shows potential heap leach ore by year. This is tabulated at an NSR cutoff grade of $2.84 per ton that covers direct processing costs. This potential ore amounts to 95.2 million tons at 0.311% total copper, 0.177% soluble copper and 0.143 g/t gold, so as expected the gold grade is low for heap leach ore. The gold will not be recovered on this facility.

The next three sections of the table show total mill ore from the mine by year and the oxide and sulfide portions of this ore. The schedule assumes co-mingling of oxide and sulfide ores in the plant. Oxide plant feed is 67.1 million tons at 0.432% total copper, 0.281% soluble copper, and 0.779 g/t gold, so as expected the gold grade is relatively high for this material. Sulfide plant feed amounts to 398.7 million tons at 0.290% total copper, 0.038% soluble copper, and 0.417 g/t gold. Total mill ore is 465.8 million tons at 0.311% total copper, 0.073% soluble copper, and 0.469 g/t gold. All ore production figures are based on measured and indicated mineral resources only, inferred resource is considered valueless rock.

Preproduction stripping during Years -2 and -1 is 46.2 million tons, though this contains 28.4 million tons of heap leach ore and 3.8 million tons of mill ore. Another significant activity during the preproduction period will be the construction of access roads to the mine, VRMA, crusher, and low grade stockpile areas.

The bottom of the table shows total mine production by year and waste. The peak annual total material movement is 65 million tpa from Years 1 through 17. Total material is 1.27 billion tons and total valueless rock is 656.6 million tons.

The mill NSR cutoff grade varies by year to balance the mine and plant production rates. It starts at $8.60 per ton during preproduction and Year 1 and increases to $14.00 per ton during Years 2 through 5. It then decreases to $12.75 per ton for Years 6 through 9 then declines gradually to the internal cutoff grade of $8.60 per ton at the end of the mine life. The mill cutoff grade is low during Year 1 due to relatively low initial tonnage movements and much of the oxide ore going to the heap leach facility.

Between a potential low-grade NSR cutoff of $9.25 per ton and the operating cutoff grade for each year there is the potential to stockpile approximately 57 million tons (55 million sulfide and 2 million oxide) for potential processing at the end of commercial open pit mining. Table 16-3 also shows this material by year.

Table 16-4 shows a proposed plant production schedule for mill and heap leach ore. For mill ore Year 1 is shown as the ore mined during preproduction and Year 1. Years 20 through 23 include the reclaim of the low grade ore stockpile. Including the low grade, total plant production amounts to 522.8 million ore tons at 0.297% total copper, 0.067% soluble copper, and 0.441 g/t gold. Updated metal recovery formulas, compared to those presented with the design economics presented in Section 15.2 and Table 15-2, were developed after the pit design and scheduling. The updated metal recoveries are for the copper concentrate, gold, and copper agitated leach process. Metal recovery formula for the heap leach process remains the same. The updated metal recovery formulas are applied to the metal production schedule as shown in Table 16-3 and Table 16-4. The updated metal recovery formulas are shown in the formulas below.

Copper flotation recovery, $CuRec_{CON}$ (%) = 95 x ((totcu – solcu) / totcu) x (0.99 – EXP (-20 x (totcu-solcu – 0.1)))

Copper concentrate grade (%) = 34.5 x (0.99 – EXP (-7 x (totcu – solcu – 0.05)))

Agitation leach copper recovery (%) = 100 x (0.9977 x solcu + 0.107 x (totcu x (1 – $CuRec_{CON}$ /100) – solcu) / totcu

Gold recovery (%) = 80 x (1.0 – EXP (-6.0 x (gold)))

Where:

EXP = exponent of the natural logarithm

Recovery capped at 95%

Agitation leach wash efficiency of 98% is not included in this calculation. It is incorporated in the financial yearly model.

Indicated mill recoveries, based on applying the recovery equations on a block by block basis are 86.7% for copper and 73.2% for gold. Indicated heap leach copper recovery on a block by block basis is 73.7%. The table also shows a proposed schedule for the heap leach ore. During early years, the mine will produce heap leach ore much faster than the 14.6 million tpa processing rate. Excess ore will be stored in the low grade stockpile area and re-handled when needed. This ore will be removed before it interferes with low grade mill ore.

Table 16-3: Mine Production Schedule – Nominal 60,000 TPD Plant with Heap Leach

	Units	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	TOTAL
Heap Leach Ore:																											
Ore Ktonnes	(kt)	6,422	21,949	21,182	8,838	3,684	6,515	5,568	4,654	5,280	4,674	2,841	944	24	2,373	182	10	12		12							95,164
Total Copper	(%)	0.242	0.298	0.384	0.387	0.329	0.316	0.343	0.238	0.240	0.186	0.220	0.228	0.201	0.274	0.214	0.440	0.279		0.150							0.311
Soluble Copper	(%)	0.132	0.190	0.218	0.251	0.168	0.184	0.198	0.119	0.099	0.080	0.093	0.081	0.065	0.099	0.058	0.217	0.139		0.064							0.177
Gold	(g/t)	0.085	0.109	0.139	0.165	0.172	0.120	0.130	0.171	0.204	0.243	0.183	0.222	0.341	0.139	0.157	0.333	0.403		0.119							0.143
Total Mill Ore:																											
Ore Ktonnes	(kt)	32	3,793	15,518	25,272	26,193	26,193	26,146	26,070	23,118	22,160	22,652	22,812	23,181	23,300	23,680	23,843	23,801	23,171	23,217	23,137	22,891	15,593				465,773
Total Copper	(%)	0.298	1.000	0.712	0.442	0.343	0.309	0.323	0.324	0.341	0.282	0.339	0.298	0.283	0.256	0.296	0.292	0.270	0.235	0.219	0.220	0.206	0.178				0.311
Soluble Copper	(%)	0.227	0.744	0.330	0.212	0.134	0.074	0.046	0.062	0.059	0.051	0.044	0.040	0.038	0.039	0.030	0.036	0.036	0.031	0.031	0.035	0.037	0.039				0.073
Gold	(g/t)	0.392	0.603	0.528	0.562	0.667	0.659	0.540	0.493	0.336	0.501	0.466	0.512	0.456	0.477	0.302	0.343	0.368	0.415	0.425	0.403	0.374	0.512				0.469
Oxide Mill Ore:																											
Ore Ktonnes	(kt)	32	3,098	9,574	13,002	12,880	8,189	1,016	3,953	4,643	6,490	1,599	1,361	670	242	11	65	292									67,117
Total Copper	(%)	0.298	1.126	0.833	0.457	0.348	0.253	0.202	0.230	0.245	0.196	0.386	0.381	0.354	0.294	0.411	0.261	0.145									0.432
Soluble Copper	(%)	0.227	0.904	0.507	0.375	0.229	0.134	0.076	0.139	0.139	0.082	0.118	0.121	0.122	0.113	0.240	0.113	0.044									0.281
Gold	(g/t)	0.392	0.705	0.648	0.779	0.924	0.851	0.692	0.627	0.636	0.752	0.971	0.962	0.868	0.646	0.549	0.554	0.701									0.779
Sulfide Mill Ore:																											
Ore Ktonnes	(kt)		695	5,944	12,270	13,313	18,004	25,130	22,117	18,475	15,670	21,053	21,451	22,511	23,058	23,669	23,778	23,509	23,171	23,217	23,137	22,891	15,593				398,656
Total Copper	(%)		0.436	0.518	0.426	0.338	0.334	0.328	0.341	0.365	0.317	0.335	0.293	0.281	0.256	0.296	0.292	0.272	0.235	0.219	0.220	0.206	0.178				0.290
Soluble Copper	(%)		0.031	0.045	0.040	0.043	0.047	0.045	0.048	0.039	0.038	0.038	0.035	0.036	0.038	0.030	0.036	0.036	0.031	0.031	0.035	0.037	0.039				0.038
Gold	(g/t)		0.147	0.334	0.332	0.418	0.571	0.534	0.469	0.261	0.397	0.428	0.483	0.444	0.475	0.302	0.342	0.364	0.415	0.425	0.403	0.374	0.512				0.417
Low Grade Stockpile (Oxide):																											
Ore Ktonnes	(kt)				17	72	191	50	326	410	905	55	8														2,034
Total Copper	(%)				0.136	0.120	0.094	0.075	0.097	0.090	0.094	0.087	0.094														0.094
Soluble Copper	(%)				0.061	0.053	0.051	0.035	0.051	0.060	0.047	0.027	0.027														0.050
Gold	(g/t)				0.408	0.415	0.421	0.433	0.380	0.355	0.400	0.400	0.402														0.391
Low Grade Stockpile (Sulfide):																											
Ore Ktonnes	(kt)				4,080	2,205	2,574	4,555	5,425	6,623	4,899	6,900	5,601	3,590	2,410	2,481	518	791	1,037	1,256							54,945
Total Copper	(%)				0.236	0.230	0.209	0.202	0.189	0.222	0.193	0.183	0.167	0.163	0.181	0.191	0.166	0.177	0.163	0.133							0.193
Soluble Copper	(%)				0.022	0.025	0.026	0.024	0.023	0.024	0.021	0.020	0.022	0.023	0.022	0.019	0.020	0.017	0.017	0.025							0.022
Gold	(g/t)				0.153	0.183	0.212	0.221	0.215	0.150	0.209	0.231	0.249	0.241	0.188	0.169	0.204	0.183	0.212	0.272							0.206
Total Material, Waste, W:O																											
Total Material	(kt)	11,236	35,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	49,199	49,281	24,756				1,274,472
Waste (Low Grade as Ore)	(kt)	4,782	9,258	28,300	26,793	32,846	29,527	28,681	28,525	29,569	32,362	32,552	35,635	38,205	36,917	38,657	40,629	40,396	40,792	40,515	26,062	26,390	9,163				656,556
Waste:Ore Ratio (LG as Ore)	(none)	0.74	0.36	0.77	0.70	1.02	0.83	0.79	0.78	0.83	0.99	1.00	1.21	1.43	1.31	1.47	1.67	1.64	1.69	1.65	1.13	1.15	0.59				1.06
Stockpile Rehandle	(kt)			3,825	5,762	10,916	75																6,607	22,200	22,200	5,972	77,557

Table 16-4: Proposed Plant Production Schedule – Nominal 60,000 TPD Plant with Heap Leach

	Units	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	TOTAL
Proposed Heap Schedule:																											
Ore Ktonnes	(kt)	3,600	14,600	14,600	14,600	14,600	6,590	5,568	4,654	5,280	4,674	2,841	944	24	2,373												94,948
Total Copper	(%)	0.242	0.298	0.384	0.386	0.300	0.315	0.343	0.238	0.240	0.186	0.220	0.228	0.201	0.274												0.311
Soluble Copper	(%)	0.132	0.190	0.218	0.238	0.175	0.183	0.198	0.119	0.099	0.080	0.093	0.081	0.065	0.099												0.177
Gold	(g/t)	0.085	0.109	0.139	0.155	0.122	0.120	0.130	0.171	0.204	0.243	0.183	0.222	0.341	0.139												0.143
Proposed Mill Schedule																											
Ore Ktonnes	(kt)			19,343	25,272	26,193	26,193	26,146	26,070	23,118	22,160	22,652	22,812	23,181	23,300	23,680	23,843	23,801	23,171	23,217	23,137	22,891	22,200	22,200	22,200	5,972	522,752
Total Copper	(%)			0.768	0.442	0.343	0.309	0.323	0.324	0.341	0.282	0.339	0.298	0.283	0.256	0.296	0.292	0.270	0.235	0.219	0.220	0.206	0.176	0.173	0.200	0.234	0.297
Soluble Copper	(%)			0.411	0.212	0.134	0.074	0.046	0.062	0.059	0.051	0.044	0.040	0.038	0.039	0.030	0.036	0.036	0.031	0.031	0.035	0.037	0.033	0.022	0.025	0.023	0.067
Gold	(g/t)			0.542	0.562	0.667	0.659	0.540	0.493	0.336	0.501	0.466	0.512	0.456	0.477	0.302	0.343	0.368	0.415	0.425	0.403	0.374	0.420	0.238	0.203	0.163	0.441
Oxide Mill Ore:																											
Ore Ktonnes	(kt)			12,704	13,002	12,880	8,189	1,016	3,953	4,643	6,490	1,599	1,361	670	242	11	65	292						968	1,049	17	69,151
Total Copper	(%)			0.903	0.457	0.348	0.253	0.202	0.230	0.245	0.196	0.386	0.381	0.354	0.294	0.411	0.261	0.145						0.094	0.094	0.136	0.422
Soluble Copper	(%)			0.603	0.375	0.229	0.134	0.076	0.139	0.139	0.082	0.118	0.121	0.122	0.113	0.240	0.113	0.044						0.046	0.054	0.061	0.274
Gold	(g/t)			0.661	0.779	0.924	0.851	0.692	0.627	0.636	0.752	0.971	0.962	0.868	0.646	0.549	0.554	0.701						0.400	0.383	0.408	0.767
Sulfide Mill Ore:																											
Ore Ktonnes	(kt)			6,639	12,270	13,313	18,004	25,130	22,117	18,475	15,670	21,053	21,451	22,511	23,058	23,669	23,778	23,509	23,171	23,217	23,137	22,891	22,200	21,232	21,151	5,955	453,601
Total Copper	(%)			0.509	0.426	0.338	0.334	0.328	0.341	0.365	0.317	0.335	0.293	0.281	0.256	0.296	0.292	0.272	0.235	0.219	0.220	0.206	0.176	0.177	0.206	0.234	0.278
Soluble Copper	(%)			0.044	0.040	0.043	0.047	0.045	0.048	0.039	0.038	0.038	0.035	0.036	0.038	0.030	0.036	0.036	0.031	0.031	0.035	0.037	0.033	0.021	0.024	0.023	0.036
Gold	(g/t)			0.314	0.332	0.418	0.571	0.534	0.469	0.261	0.397	0.428	0.483	0.444	0.475	0.302	0.342	0.364	0.415	0.425	0.403	0.374	0.420	0.230	0.195	0.162	0.392

16.5 MINING EQUIPMENT

This Preliminary Feasibility Study was based on an assumption that the mining will be performed by a contractor throughout the life of the mine (contract mining). Mine equipment requirements were calculated on a first principles basis, based on the annual mine production schedule, the mine work schedule, and estimated equipment productivity per shift. The size and type of mining equipment is consistent with the size of the project, i.e. peak material movements of approximately 65 million tpa.

Table 16-5 shows the major mining equipment required by year. During peak production years, three large drills (P&H 320XPC class) and three large cable shovels (P&H 2800XPC class) will be required. The required truck fleet (Caterpillar 793 class trucks) peaks at 29 units during Years 3 through 6.

Table 16-5: Mine Major Equipment Fleet Requirement

Table 16-5: Mine Major Equipment Fleet Requirement																											
	Capacity/	Time Period																									
Equipment Type	Power	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24
P&H 320XPC Drill	(457 mm)	1	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1	0	0	0	0
P&H 2800XPC Cable Shovel	(36.6 cu m)	0	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	1	1	1	0
Cat 994F Wheel Loader	(17 cu m)	2	2	1	2	2	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0	0
Cat 793F Truck	(230 mt)	6	14	25	23	29	28	27	29	20	21	24	25	28	27	27	27	27	24	26	21	23	14	5	6	2	0
Cat D10T Track Dozer	(433 kw)	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	0
Cat D9T Track Dozer	(306 kw)	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	0
Cat 844H Wheel Dozer	(468 kw)	1	1	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	0
Cat 16M Motor Grader	(221 kw)	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	0
Water Truck - 30,000 gal	(113,562 l)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	0
Cat 345D Excavator	(2.7 cum)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	0
Atlas Copco ECM 720 Drill	(140 mm)	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	0	0	0	0
Cat 992K Wheel Loader	(10.7 cu m)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	0
Cat 777F Truck	(90 mt)	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	3	3	3	3	3	0
TOTAL		28	40	56	55	61	59	58	60	51	52	55	56	59	58	58	58	58	55	57	50	44	34	22	23	18	0

17 RECOVERY METHODS

17.1 PROCESS DESCRIPTION

The King-king processing facility will recover copper by conventional flotation, by agitated leach of the flotation tails, and by heap leaching of oxide copper ores. Leached copper will be processed through solvent extraction and electrowinning (SX-EW) to produce copper cathodes. Gold will primarily be recovered in the copper flotation concentrate with a fraction recovered in bullions produced by gravity concentration in the grinding circuit. The process design will be based on metallurgical tests results from AMEC Australia and column heap leach tests performed by Leach, Inc. (Tucson, AZ).

Figure 17-1 is a simplified schematic of the process for the sulfide plant. Figure 17-2 is a simplified schematic of the heap leach operation. These provide the basis for the process description that follows.

17.2 PROCESS DESIGN CRITERIA

SAGC tasked M3 Engineering to design a process plant for the King-king project with a nameplate capacity of 60,000 tpd. For the design, M3 used an availability factor of 92%, except for the primary crushing area where an availability factor of 75% was used. These design availability factors are common for current and recent projects at M3.

The current mine plan developed for the project is based on a 365-day calendar year. The yearly ore tonnage is nominally 21.9 million metric tons, with the tonnage variations arising from variations in the hardness of the ore being mined.

Table 17-1 is a summary of the main components of the process design criteria used for the study.

The mass balance was developed for the King-king process using MetSim software. The process simulation assumed overall recoveries shown on Table 17-2 for gold, sulfide copper and oxide copper.

These recoveries are based on the recovery equations supplied by AMEC Australia and provided to M3 in April of 2012. The average grades used for the MetSim simulation were 0.297% total copper and 0.371 g/t gold, which were the average grades originally reported by IMC in their NI 43-101 report (October 12, 2010). Newer recoveries, including copper oxide recovery to cathodes, have since become available. In addition, IMC has revised the mine plan with new head grade predictions. The new head grades and recoveries will be used in the next study.

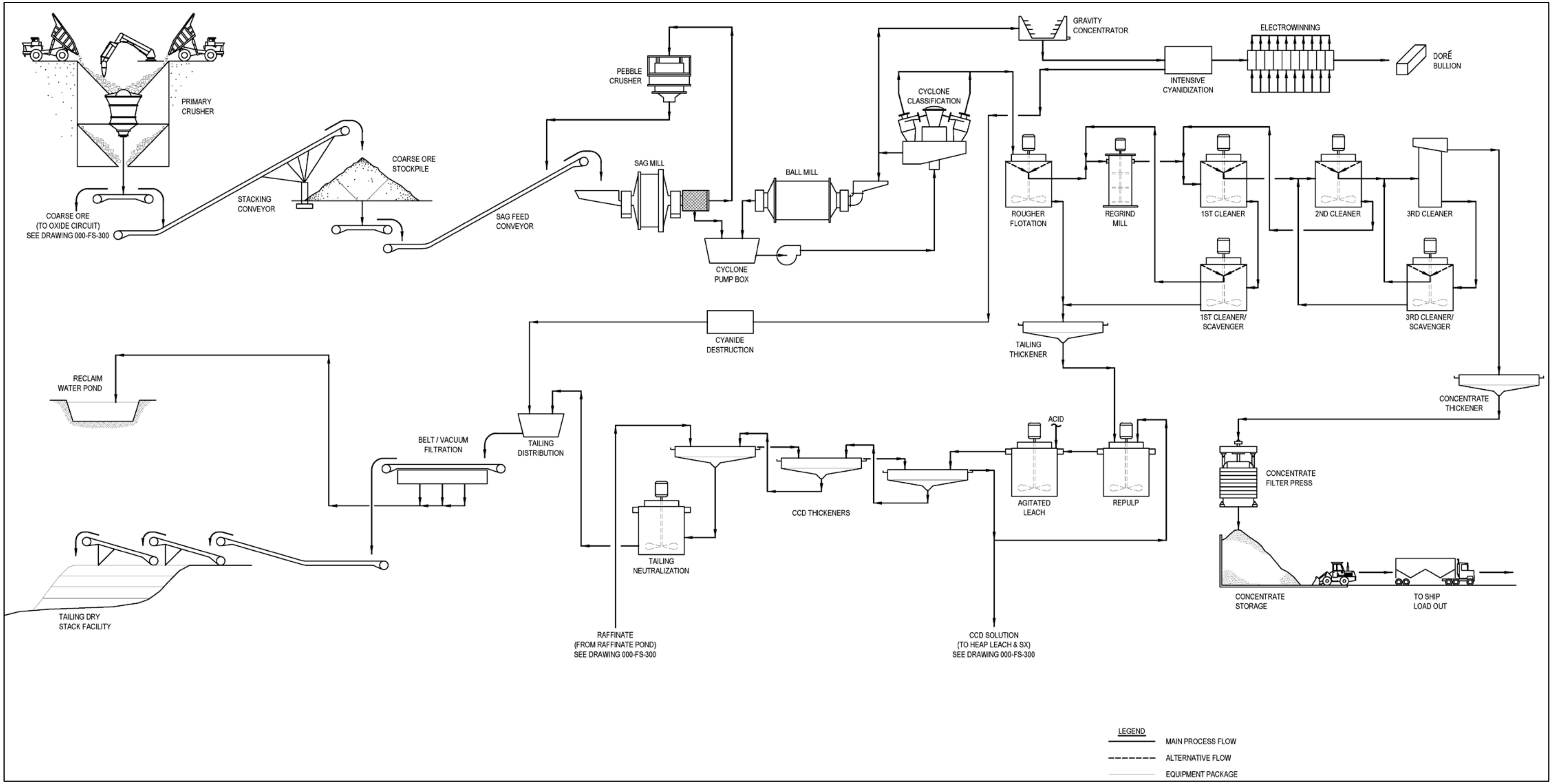


Figure 17-1: Simplified Process Flow Diagram for the King-king Sulfide Plant

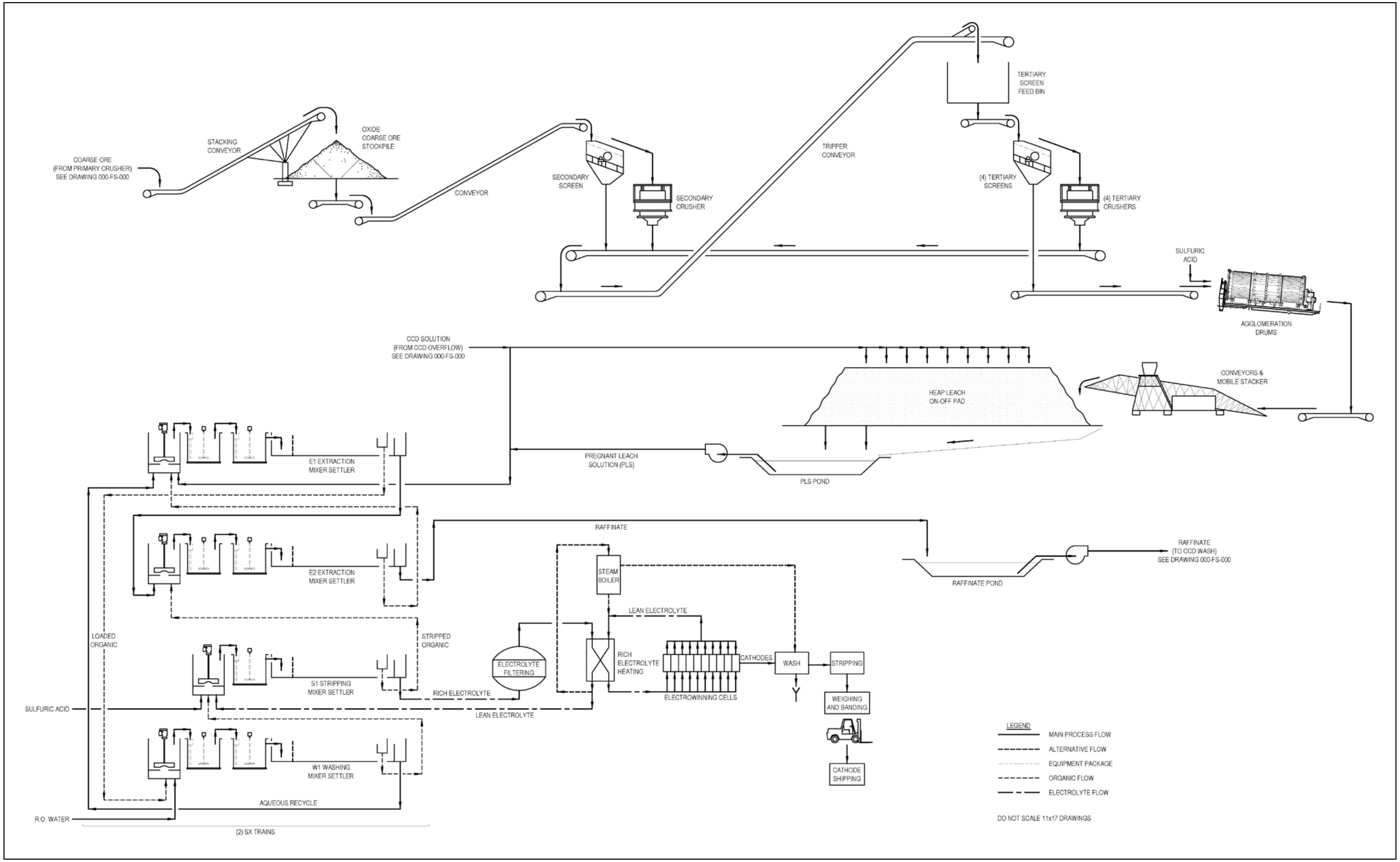


Figure 17-2: Simplified Process Flow Diagram for the King-king Heap Leach Operation

Table 17-1: Process Design Criteria

DESCRIPTION	DESIGN
Primary Crusher	
Feed F80, mm	400
Product P80, mm	120
Crushing work index, kWh/t	11
SAG Mill Grinding	
Feed F80, mm	120
Product P80, mm	11.123
SAG Mill Axb Parameter	40
Ball Mill Grinding	
Feed F80, microns	3,105
Product P80, microns	106
Ball Mill Work Index, kWh/t	16.1
Flotation	
Rougher Flotation Time, min	35
First Cleaner Flotation Time, min	12
Cleaner Scavenger Flotation Time, min	12
Second Cleaner Flotation Time, min	10
Third Cleaner Flotation Time, min	5
Agitated Leach	
Leach Time, h	4
% Solids	35 - 40
Acid consumption, kg/t	25
Leach Temperature, ^{o}C	50

*Owner-supplied information

Table 17-2: Metal Recoveries Used for Mass Balance Simulation

Metal	Head Grade (AMEC)	Product	Recovery, %
Cu, Total	0.39 – 0.62 %		
Cu, in sulfides	0.14 – 0.24	Cu Concentrate	86
Au	0.39 – 0.61 g/t	Cu Concentrate	60
		Au Bullion	17

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 60” x 110” gyratory crusher, with an open side setting of 180 mm (7 inches) and a feed opening of 1,524 mm (60 inches). It will be powered by a 1,000-kW motor. The crushed ore will drop into a discharge bin equipped with an apron feeder.

The apron feeder will meter ore onto a transfer conveyor, which will deliver the crushed ore to a 3.4-km aerial conveyor that discharges via a tripper arrangement onto one of two overland conveyors feeding oxide and sulfide coarse ore stockpiles. The oxide and sulfide coarse ore stockpiles will have equivalent total capacities of 200,000 tons and live capacities of 60,000 tons. This is equivalent to about 24 hours of SAG mill feed and about 36 hours of secondary/tertiary crushing plant feed. Three belt feeders (two operating and one standby) will reclaim crushed ore from each stockpile and transfer it onto the reclaim belts feeding, respectively, the SAG feed and secondary/tertiary crusher feed conveyors.

17.4 GRINDING

The grinding circuit for the King-king Project will be a conventional semi-autogenous grinding (SAG) mill-ball mill-pebble crusher (SABC) system. The SAG mill will be in a closed circuit with a pebble screen and a pebble crusher. The ball mills will be in a closed circuit with hydrocyclone clusters.

The SAG feed conveyor will feed ore to the SAG mill (12.2-m diameter by 7.3-m effective grinding length (EGL), 28-MW gearless drive). The SAG mill product will discharge to a trommel and then to a pebble wash screen. The undersize of the trommel and pebble wash screen drops into the cyclone feed pump box. This will constitute fresh feed to two ball mills. It will mix with the discharge from the ball mills and be pumped to two primary cyclone clusters. Pumping will be by two 30x26 cyclone feed pumps with 2,250-kW variable frequency drives (VFD), with a third 30x26 pump as operating spare. The cyclone underflows will be fed to two ball mills (8.23-m diameter by 12.8-m length, 20-MW gearless drives). The cyclone overflows will constitute the product of the grinding circuit and will be fed to the flotation circuit. The target size distribution is 80 percent finer than 106 microns. A bleed from the cyclone underflow will be processed for recovery of free gold by gravity concentration and intensive cyanidation. This part of the process will be discussed further under the section for gold bullion production.

The pebbles separated by the pebble wash screen will be collected on the pebble crusher feed conveyor, transported to the pebble crusher feed bins, crushed by a single MP800-type cone crusher (1” closed-side setting), and returned to the SAG mill via the SAG feed conveyor. The pebbles may also bypass the pebble crusher onto a pebble stockpile for further handling, as deemed appropriate.

17.5 FLOTATION

Flotation of copper in the King-king process plant will be accomplished using two banks of rougher flotation cells to achieve recovery, and three stages of cleaning to meet smelter grade requirements.

The cyclone overflow from the grinding circuit will report to the rougher bank feed tanks. Tailing from the rougher banks will report to the flotation tails thickeners.

Rougher concentrates will be sent to one of two 300-kW vertical regrind mills. Both will be in closed circuit with hydrocyclones. Concentrate from each rougher bank will be sent to the corresponding regrind pump box where it will combine with discharge from the regrind mill.

From the pump box, the slurry will be pumped to the hydrocyclones for classification. The hydrocyclone underflow will be returned to the regrind mill, while the overflow will flow to the first cleaner flotation circuit. The target particle size distribution for the reground material is 100 percent finer than 20 microns. A circuit bypass will be included such that rougher concentrates can be pumped directly to the first cleaner cells without first being passed through the regrind step.

Three stages of cleaning will upgrade the reground concentrate to meet smelter specifications. In addition, a first cleaner scavenger stage will be installed to produce tailing that can be forwarded to the final flotation tailing (leach feed) without significant loss of sulfide copper.

The concentrate of the first cleaner cells will be transferred to the second cleaner flotation circuit while the tails will proceed by gravity to the cleaner scavenger flotation circuit. Concentrate from the cleaner scavenger flotation circuit will be sent to the regrind circuit feed. Tailing from the cleaner scavenger circuit will be pumped to the flotation tails (leach feed) thickener.

The concentrate from the second cleaner flotation circuit will be pumped to the third cleaner flotation column. Concentrate from this column will be pumped to the concentrate thickener as final concentrate. The tailing from the second cleaner flotation circuit will be recycled to the first 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 upon due to operator preference. The column flotation cells may be removed from the design altogether if the feed size proves to be too fine for the column to process.

The sizes and numbers of the flotation cells that will be installed in the flotation circuit are shown in Table 17-3.

Table 17-3: Flotation Cells

STAGE	NUMBER OF CELLS	SIZE OF CELLS m^3
Rougher	14	300
First Cleaner	4	100
Cleaner-Scavenger	4	100
2nd Cleaner	6	50
3rd Cleaner Column	1	3.6-m dia.
3rd Cleaner Scavenger	6	30

Reagents to be used in the flotation plant include sodium isobutyl xanthate (SIBX) or potassium amyl xanthate (PAX), or possibly an alkyl dithiophosphate-based reagent as collectors, methyl isobutyl carbinol (MIBC) or equivalent as frother, and milk of lime for pH control.

17.6 CONCENTRATE THICKENING, FILTRATION, AND STORAGE

Concentrate from the third cleaner flotation circuit will be dewatered in the 30-m diameter concentrate thickener. The thickened concentrate will be pumped to two vertical filter presses, operating in parallel. The filtered concentrate will be conveyed to a concentrate stockpile, from where it will be loaded onto trucks by a front-end loader and sent to concentrate storage facility at the port. The port storage facility will consist of a covered stockpile, three sub floors reclaim feeders, and a load out conveyor to the cargo ship. A front-end loader will be required to keep the reclaim feeders full during load out. The load out conveyor will be equipped with a belt scale and sampler, to determine tonnage, moisture and grades for the shipment.

17.7 GRAVITY GOLD RECOVERY AND CYANIDE DESTRUCTION

Slurry bleed streams will be taken from each of the two primary cyclone underflow launders. These will be screened and fed to gravity concentrators (Knelson or Falcon) in two parallel lines to recover free gold from the ore. Concentrate from the gravity concentrator will be passed to an intensive cyanidation unit where it will be leached for gold and silver. Pregnant solution produced by the intensive cyanidation unit will be sent to a single bank of electrowinning (EW) cells. The gold rich cathode slimes harvested from the EW cells will be smelted and poured into doré bullions.

A bleed of the barren cyanide solution will be taken for disposal through a cyanide destruction system. This will be a SO_2-air system that will reduce the weak-acid dissociable (WAD) cyanide down to <50 ppm, before disposal into tailing handling facility.

Reagents for this section of the mill will include sodium cyanide, an oxidizer, milk of lime, sodium metabisulfite or ammonium bisulfite, and small amounts of borax, soda ash, niter and silica for gold smelting.

17.8 AGITATED COPPER LEACHING

During the treatment of oxide dominant ores, rougher and cleaner scavenger flotation tails will be processed by agitated acid leach to recover acid soluble copper. The tails will be leached at 35% solids with 25 kg of sulfuric acid per ton of ore for four hours at 50^{o}C. Pregnant solution will be recovered in a counter-current decantation (CCD) circuit using raffinate from the SX-EW plant as the wash solution.

The flotation tailing feed to the agitated copper leach plant is expected to be at a pulp density of about 30% solids. This will be thickened to 55% solids in two parallel 55-m diameter high rate thickeners. This thickening step prior to leaching enables the use of CCD overflow PLS to dilute the leach feed slurry back to 35% solids thereby conserving acid. In order to attain the 50^{o}C leach temperature target, the PLS for leach dilution will be preheated to 62^{o}C by hot water (80^{o}C) bringing waste heat from the power plant.

Agitated leaching will be carried out in 5 agitated tanks in series, each with a residence time of one hour. A bypass line between the tanks will allow bypassing of any tank for maintenance. Thus, at any time four tanks will be in use and one will be on standby. After leaching, the slurry will go through six stages of counter current decantation (CCD) using two trains of high-rate thickeners (55-m diameter) with six thickeners per train. The clear solution overflow from the CCD trains are split four ways as follows: (1) tails leach repulp tank, (2) heap leach pad, (3) agglomeration drum and (4) the rest to two SX trains. The final CCD train underflows will be neutralized in two separate stages, first with limestone slurry followed by milk of lime. The neutralized tails slurry will then be sent to the tailing dewatering facility.

17.9 COPPER OXIDE ORE HEAP LEACHING

17.9.1 Crushing, Agglomeration, Stacking and Leaching

Coarse ore from the primary crusher will be transferred by aerial and overland conveyors to a stockpile that will feed the secondary/tertiary crushing plant. The ore will be crushed and screened to 80% minus 20 mm. The fine ore will then be fed to a rotary drum agglomerator where it will be agglomerated with 12.8 kg/t of sulfuric acid and water (CCD overflow solution) to a moisture content of about 8%, excluding acid. The agglomerated ore will then be delivered to a stacking conveyor on the heap leach pad. A tripper diverts the ore into a hopper that feeds a line of portable conveyors terminating in a stacker that distributes the ore onto one of the cells in the lined heap leach pad.

Once the cell is fully loaded, the ore is irrigated through a system of drip irrigator lines with a sulfuric acid solution that percolates through the ore and dissolves the copper. The solution containing the dissolved copper (pregnant leach solution or PLS) is collected at the base of the ore stack in a gravel drainage layer and perforated piping system and directed into a geosynthetically lined pond. The PLS is then pumped to the SX-EW plant.

The ore must be leached for a specific period of time to optimize the recovery of the copper. At King-king, the leach cycle duration is estimated to be 67 days (60 days of leach and 7 days of rinse to remove entrained copper solution). The heap leach pad currently planned for King-king will be an “on-off” pad. When the ore has completed its 67-day leach cycle, it is removed from the pad using a second system of portable conveyors and a bucket wheel excavator. This conveyor line feeds a hopper that loads the spent ore onto a dedicated conveyor for delivery to the Spent Ore Storage Facility (SOSF). There it is stacked for permanent storage and ultimately reclamation. Conveyor corridors are provided at the top and bottom of the cells that include space for a 10 m wide access road, a 5 m corridor for the main conveyor, and a 25 m corridor for moving hoppers and other equipment required for the loading of the portable conveyor lines. During heap operation, the loading/stacking and the reclaiming/unloading occur within a moving window that migrates across the heap cells. Stacking equipment is anticipated to have an operating capacity of 2,000 tons per hour (tph) and reclaiming equipment a capacity of 2,500 tph. Ore will be stacked on the heap leach pads in 6 m lift heights.

17.9.2 Slope Stability

The ore stacks on the on-off HLP will be placed within cell limits to heights of 6 m on grades of less than 5% so as to achieve stability. The edges of the ore stack will be at the natural angle of repose. The SOSF will be designed with a minimum static Factor of Safety (FOS) of 1.3. Mass stability can be controlled by any one of several potential failure mechanisms including circular failure modes through the ore mass itself, block type failure modes along linear planes at the liner-soil interface, and failure modes that penetrate the native foundation soil and weathered rock beneath the liner system. All of these potential failure modes will be investigated using a computer assisted limit equilibrium stability model (Slide 6.0) once needed material properties have been acquired from a planned program of site-specific geotechnical investigation, sampling, and testing.

The King-king site is located in an area of active seismicity and therefore facilities will be designed to resist seismic (earthquake) loads. Peak ground accelerations (PGA) have been estimated for the project site for the operating basis earthquake (OBE) having a 10% probability of occurrence in 50 years (recurrence interval of 475 years) and for the Maximum Design Earthquake (or Maximum Credible Earthquake (MCE)) having a 2% probability of occurrence in 50 years (recurrence interval of 2,475 years). The estimated PGAs are 0.6g and 1.01g respectively. Common practice has been to estimate the pseudostatic acceleration coefficient used in stability analysis by using half of the actual estimated PGA. For the King-king site, this is 0.3g (OBE) and 0.505g (MCE). Pseudostatic analysis and design will use a target FOS of 1.0 for the non-impounding earthworks represented by the HLP and SOSF.

17.9.3 Water Balance and Solution Management

The King-king site is in an area of high precipitation and moderately high evaporation resulting in a net precipitation environment. Geosynthetic liner systems are used for environmental containment to prevent contamination of surface or groundwater by acid solutions used in the copper leaching process. The pond system for the HLP is designed to store runoff from a 100-yr 24-hr storm event (310 mm) plus the expected drain down volume from a 12-hr power outage. Similarly, the ponds for the SOSF are designed to store the runoff from a 100-yr 24-hr storm event.

However, storage of the runoff volume from an extreme storm event alone is not sufficient to assure an acceptable level of containment. Another method of reducing the buildup of water is through the use of temporary removable liners on the surface of the ore to exclude meteoric water.

17.10 SOLVENT EXTRACTION AND ELECTROWINNING

The SX-EW plant will consist of two lines of mixer-settler trains that will run in parallel, and 4 banks of electrowinning cells.

A portion of the PLS from the flotation tailing leach and all the PLS from the heap leach will be pumped to the solvent extraction feed tank, which feeds the solvent extraction train by gravity.

Most of the raffinate solution leaving the electrowinning plant will be used as wash solution in the flotation tailing leach CCD circuit. A smaller portion will be used in the rinse cycle of heap leach cells. A bleed stream will be neutralized with the CCD tailing stream to control the buildup of impurity elements.

17.11 TAILING SLURRY TRANSPORT

Thickened tailing slurry will be piped to the tailing dewatering plant near the dry-stacked tailing storage facility. The pipeline system will consist of two 0.71-m diameter high-density polyethylene (HDPE) pipes (both lines in service). The pipelines will be 2,300 meters long and will require pumping. The slurry pipes will discharge into agitated surge tanks, from where the slurry will be distributed to the dewatering belt filters.

17.12 TAILING DEWATERING

Mill tailing will be dewatered in two stages, first by high-rate thickeners to about 55% solids then by dewatering belt filters to about 15% moisture. The dewatered tailing will be transported by conveyors and trucks and deposited in a dry-stack tailing storage.

17.13 REAGENTS AND CONSUMABLES

Reagent storage, mixing and pumping facilities will be provided for all of the reagents used in the processing circuits. Table 17-4 below is a summary of reagents used in the process plant.

17.14 WATER REQUIREMENT

A water balance was developed for the King-king project using MetSim modeling software. The King-king process plant and heap leaching are projected to require 673.6 m^3/h of fresh water makeup to sustain its operation. In addition, an average of 150 m^3/h of fresh water is estimated for mine dust control and another 32 m^3/h for potable water. The total fresh water requirement will then be 855.6 m^3/h. If the consumption by heap leaching is excluded, the equivalent fresh water consumption is 0.28 m^3/ton. This is lower than the water consumption from typical operations (approximately 0.4 to 0.5 m^3/ton) because of better water recovery from tails dewatering.

17.15 MILL POWER CONSUMPTION

The power consumption in the process plant for a typical year is tabulated in Table 17-5 with a total consumption of 1.051 billion kWh. This translates to about 26.4 kWh/ton or US$1.48/ton of ore processed.

Table 17-4: Process Reagents and Consumption Rates

Reagent	Consumption, g/t
Sodium Isobutyl Xanthate (SIBX)	38
Lime	1,360
Limestone	3,600
Methyl Isobutyl Carbinol (MIBC)	25
Sodium Cyanide	0.11
Sodium Hydroxide	0.037
Oxidizer	0.11
LIX Reagent, kg/t cathode	2.2
LIX diluent, kg/t cathode	20
Cobalt Sulfate, kg/t cathode	0.51
Guar Gum, kg/t cathode	0.2
Mist Supressor, kg/t cathode	0.02
D.E. filter precoat, kg/t cathode	0.75
Clay, kg/t cathode	0.45
Flocculant	90
Antiscalant	1
Grinding Balls, 125 mm kg/ton	0.345
Grinding Balls, 75 mm kg/ton	0.241

Table 17-5: Summary of Mill Power Consumption in a Typical Year

Area	Total kWh/Year	Total Cost/year
Concentrator		
Primary Crushing /Coarse Ore Storage & Reclaim	32,367,128	$ 1,814,427
Grinding	526,455,372	$ 29,511,889
Flotation	55,512,732	$ 3,111,917
Concentrate Thickening and Filtration	7,426,064	$ 416,288
Agitated Leach	95,656,233	$ 5,362,270
Tailing Disposal	41,910,719	$ 2,349,419
Water Systems	1,693,378	$ 94,927
Ancillary Facilities	3,163,778	$ 177,354
Gravity Gold Circuit and Intensive Leach	825,929	$ 46,300
Gold Refinery	1,136,414	$ 63,705
Subtotal	766,147,748	$ 42,948,497
Heap Leach		
Crusher	53,540,828	$ 3,001,377
Heap Leach	55,578,492	$ 3,115,604
Solvent Extraction	2,442,275	$ 136,908
Tank Farm	2,954,790	$ 165,639
Electrowinning	170,754,194	$ 9,572,091
Subtotal	285,270,579	$ 15,991,619
Grand Total	1,051,418,327	$ 58,940,116
Cost per kWh		$ 0.056

17.15.1 Control Systems

A crusher control room, located in the primary crusher area at the mine, will be the operating and control center for the crusher and coarse ore transport conveyors.

A central control room (CCR) will be provided in the concentrator facility core, which will be the main operating control center for the complex. From the CCR control consoles, primary crushing, material handling systems, grinding and flotation, reagents, tailing, and utility systems will be monitored and/or controlled.

A computer room, located adjacent to the CCR will contain engineering workstations (EWS), a supervisory computer, historical trend system, management information systems (MIS) server, programming terminal, network and communications equipment, and documentation printers. This will be primarily used for Distributed Control System (DCS) development and support activities by plant and control systems engineers.

Although the facilities will normally be controlled from the CCR, local video display terminals 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.

17.16 PLANT SERVICES

17.16.1 Mobile Equipment

Table 17-6 lists the mobile equipment that is provided in the project capital cost estimate.

Table 17-6: Mobile Equipment List

DESCRIPTION	QTY	DUTY
CAT 966 Front-End Loader	7	COS, Concentrate, Heap Leach
Pick-Up Truck	25	Utility
Boom Truck, 45’ 10T	1	General Maintenance
Boom Truck, 50’ 15T	1	Water System Maintenance
Bob Cats	3	General Clean-up
Fork Lifts	5	Warehouse & SX/EW, General
Mobile HDPE Welder	1	30” pipe maximum
Mobile Hydraulic Crane, 25T	1	General Maintenance
Mobile Hydraulic Crane, 60T	1	General Maintenance
Dump Trucks	2	General Maintenance
Flat-Bed Trucks, 2T	2	General Maintenance
Track-Type Tractor, CAT D8/D9	6	General Maintenance, Stockpiles, Stacking
Concentrate Trucks	3	Concentrate Delivery
Fuel Trucks	2	Fuel Delivery
Short Bed Cathode Trucks	2	Cathode Delivery
Delivery Trucks	2	Miscellaneous
Acid Trucks, 25-ton capacity	4	Acid Delivery
Lime or Limestone Truck	2	Lime or Limestone Delivery
Motor Grader, CAT 140M	2	Maintenance & Stacking
Ambulance, 4WD	1	Emergency
Water Truck	1	Road Maintenance

17.16.2 Assay / Metallurgical Laboratories

A 16-meter wide by 60-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 laboratory, wet laboratory, and fire assay for mine and process plant samples. In addition, a small metallurgical laboratory at the mill has been designed for quick tests and measurements. The floor area for this laboratory is 3.95 meters wide and 7.6 meters long.

17.17 PRODUCTION ESTIMATE

Production by project year is tabulated in Table 17-7, showing production from the four operations described above, that is, copper, silver and gold in copper concentrate (and concentrate production), doré bullions, copper cathode from tailing leach and copper cathode from heap leach. These are illustrated in Figure 17-3 through Figure 17-6.

Table 17-7: Metal Production

PROJECT YEAR	CONCENTRATOR							HEAP LEACH	
		Flotation				Agitated Leach	Gravity Gold		
	Production	Concen-trate	Recovered Copper	Recovered Gold	Recovered Silver	Cathode Copper	Gold Bullion	Production	Cathode Copper
	kt	kt	klbs	kozs	kozs	klbs	kozs	kt	klbs
-1	-	-	-	-	-	-	-	18,200	82,704
1	19,343	191	130,051	228	593	173,580	25	14,600	96,240
2	25,272	157	101,356	310	486	118,427	34	14,600	97,410
3	26,193	163	97,265	388	507	78,348	43	14,600	70,516
4	26,193	181	110,358	385	563	44,219	43	6,590	34,021
5	26,146	225	137,958	310	696	28,404	34	5,568	32,039
6	26,070	209	127,522	279	649	37,317	31	4,654	16,211
7	23,118	199	125,372	157	617	31,310	17	5,280	18,625
8	22,160	157	91,630	240	485	26,559	27	4,674	10,923
9	22,652	213	130,290	225	660	-	25	2,841	8,769
10	22,812	192	110,541	252	594	-	28	944	3,080
11	23,181	184	104,676	225	571	-	25	24	64
12	23,300	165	88,446	240	512	-	27	2,373	10,149
13	23,680	210	122,289	141	651	-	16	-	-
14	23,843	203	116,809	165	630	-	18	-	-
15	23,801	188	103,462	179	583	-	20	-	-
16	23,171	160	81,938	202	497	-	22	-	-
17	23,217	141	70,481	209	436	-	23	-	-
18	23,137	134	68,708	195	415	-	22	-	-
19	22,891	120	57,279	176	373	-	20	-	-
20	22,200	88	36,606	197	274	-	22	-	-
21	22,200	104	44,832	95	322	-	11	-	-
22	22,200	129	62,938	78	399	-	9	-	-
23	5,972	44	23,039	15	138	-	2	-	-
Total	522,752	3,758	2,143,846	4,889	11,650	538,163	543	94,948	480,751

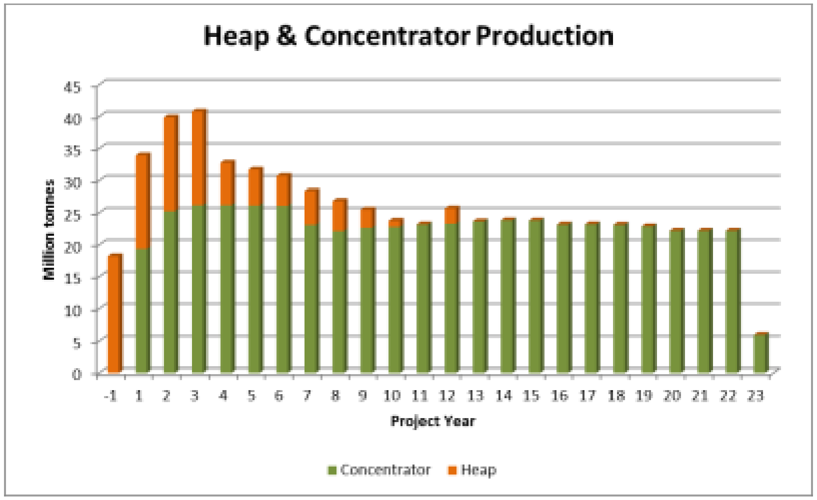


Figure 17-3: Heap and Mill Production by Project Year

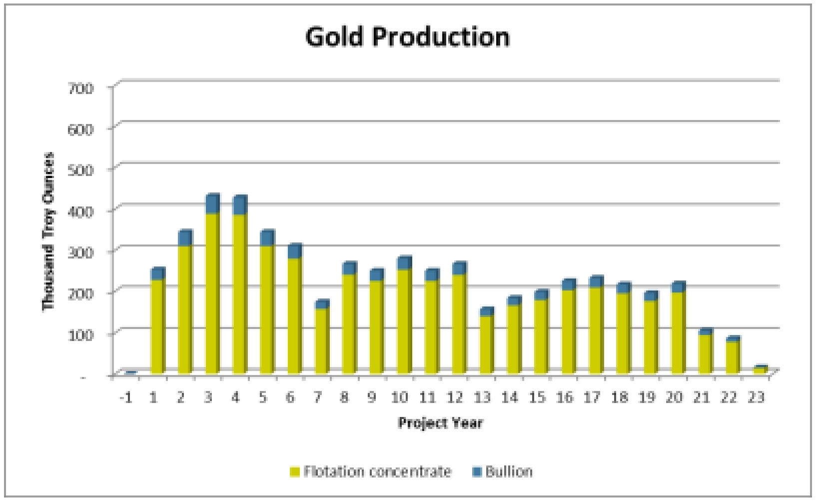


Figure 17-5: Gold Production by Product Year

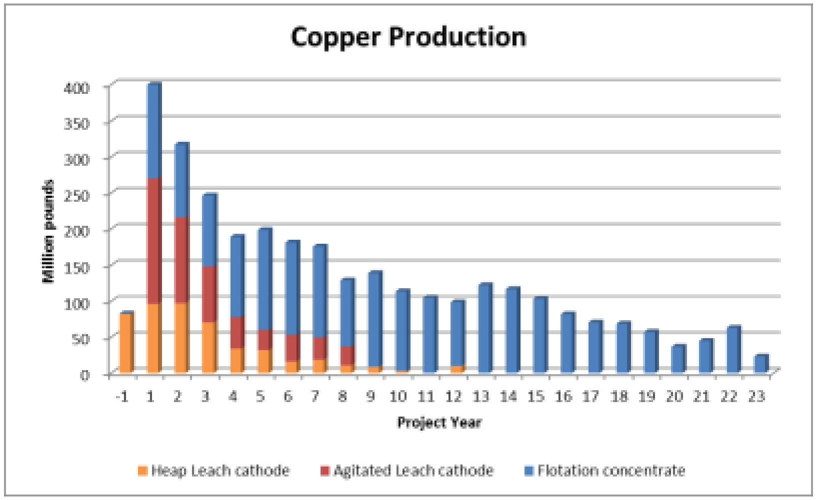


Figure 17-4: Copper Production by Project Year

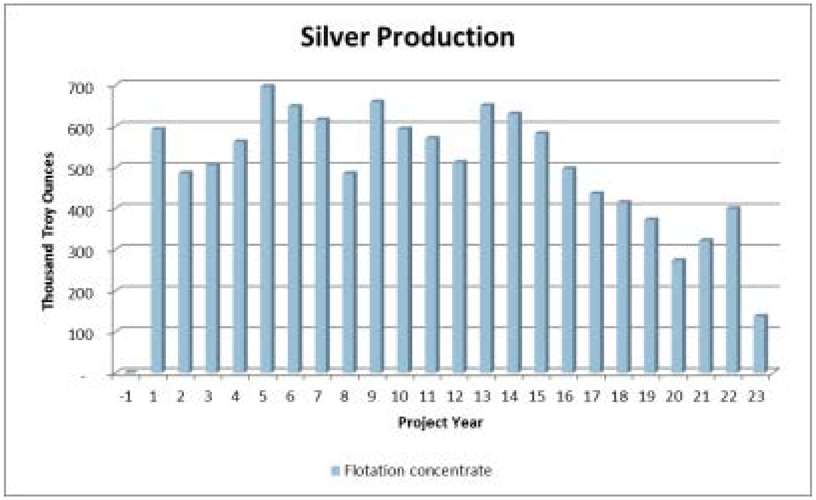


Figure 17-6: Silver Production by Project Year

18 PROJECT INFRASTRUCTURE

18.1 ACCESS ROAD

18.1.1 Main Access Road

The 13.5 km long main access road will be built from the port complex at Pantukan to the project site. The road will be unpaved and will have two lanes and widened sections for passing. The primary function of the road is to transport materials and people from the port complex to the plant, crusher and mine sites. The road will have a nominal width of 8 meters (10.8 meters at passing areas) and be constructed with a center crown and safety berms as needed on the sides of the roadway. The maximum design grade is 10%. An overpass will be constructed where the road crosses the National Highway at Pantukan.

A guard house will be located at the mill property boundary approximately 5.5 km from the port facility along the access road. The guard house will be used for security and for weighing vehicles entering and leaving the property.

18.1.2 Haul Road

Haul roads will nominally be 33 m wide and will include safety berms and ditches. The maximum gradient will be 10% per the mine plan. The roads will be constructed according to safety standards which include a berm height of half the wheel height of the largest vehicle using the road.

18.1.3 Secondary Access Roads

Secondary roads will generally be approximately 6 m wide with safety berms. Grades will vary according to use and terrain. The roads will provide access to less-frequented areas such as the explosives magazine area, power facilities, water pumping stations, pipelines, VRMA, etc.

18.1.4 Access Road along Tailing Line

A 2-km long road will be constructed from the main process plant location to the tailing dewatering building. Parallel to the road will be three 710-mm HDPE tailing lines in a trench (two active and one spare) as well as one 710-mm HDPE process water return line. The road will be used for access and inspection of the tailing lines. The road route crosses the Kingking River over a 50 m long culvert and concrete roadway section to support vehicle traffic and an independent steel structure with concrete footings to support piping.

18.2 POWER PLANT

To support the operations of the Project, a captive power generation facility shall be established. The power plant will be designed with two 80-MW coal-fired units. Four 7.25-MW (29 MW total) heavy fuel oil (HFO)-fired units are included to serve as back up and to ensure adequate availability of power supply to operations. These units will utilize the most advanced coal-fired

circulating fluidized technology and HFO-fired diesel engines in order to mitigate against environmental issues generally associated with the burning of coal and HFO.
The power generation facility will be designed in accordance with general design codes and standards. The plant systems will be designed to achieve the best possible efficiency under the specified operating conditions.

Figure 18-1 and Figure 18-2 show the power plant process diagrams. Table 18-1, Table 18-2 and Table 18-3 are summaries of power plant predicted performance data.

Table 18-1: Boiler Performance Data

Description	Unit	Design Point	Performance Test
Fuel Type	-	Coal 100%	Coal 100%
Load / Operation Condition	-	100% MCR	100% MCR
Steam Flow Leaving SH	t/h	360.00	360.00
Steam Pressure Leaving SH	bar (a)	128.50	128.50
Steam / Water Temperature			
Spray Water	°C	215.00	215
Entering Economizer	°C	215.00	215
Superheater Outlet	°C	541.00	541
Continuous Blowdown	%	1.00	-
Boiler Thermal Output	MWth	254	252
Ambient Air Condition			
Entering Fans	ºC	30.00	30.00
Relative Humidity	%	80.00	80.00
Excess Air	%	20.00	20.00
Quantity			
Coal	t/h	53.27	53.14
Limestone	t/h	1.31	1.31
Sand	t/h	0.10	0.10
Fly Ash	t/h	4.91	4.90
Bottom Ash	t/h	0.86	0.86
Air	Nm3-wet	322,338	321,508
Flue Gas	Nm3-wet	355,137	354,218
Flue Gas Temperature			
Furnace Inside (Average)	ºC	895.00	895.00
Leaving Air heater	ºC	145.00	145.00
Make-Up Water Condition			
Flow	t/h	3.25	3.25
Temperature	°C	25.00	25.00

Table 18-2: Turbine Performance Data

Description	Unit	STG Rating
Load Condition	MW	86
Main Steam Condition		
Flow	t/h	353
Pressure	bar (a)	123.56
Temperature	°C	538.00
Enthalpy	kJ/kg	3,445
Turbine Exhaust		
Pressure	bar (a)	0.106
Temperature	°C	48.37
Speed	rpm	3600

Table 18-3: Predicted Main Generator Performance Data

Capacity	86 MW (gross at full condensing mode)
Power Factor (lagging)	0.8
Generation Voltage (kV)	24
Ambient Temperature	40 ºC (minimum for electrical equipment design)
Parallel Operation with Grid	138 kV
Generator Voltage Ratings	
Nominal	24 kV, 3 phase
Operating Range	10% (+/-)
Steady State Stability	5% (+/-)
Generator Frequency Ratings	
Nominal	60 Hz
Operating Range	10% (+/-)
Steady State Stability	2.5% (+/-)

The power plant will be complete with the following sub-systems:

- Boiler system
- Steam turbine system
- Other plant equipment
- Controls and instrumentation
- Electrical system
- Material handling system
- Civil works
- Switchyard and transmission line system
- Water supply and treatment system
- Tank farm
- Waste treatment plant
- Load end substations

Power transmission from the power plant to the load end areas is planned at three voltage levels:

- 138 kV from the power plant to the concentrator area;

- 39.5 kV from the mill area to the rest of the mine operation sites, such as open pit, crusher area, TSF, water treatment and heap leach.
- 13.8 kV from the power plant to the coastal complex. Areas are planned for power supply at a 13.8 kV level from the power plant, and will be appropriately managed by the respective demand and consumer areas.

Fuel (coal) unloading facility will be shared with other port requirements of mine operations. Coal handling and processing shall include the coal unloader, conveyors, transfer stations, coal yard, stacker/reclaimer and coal crusher plant and coal feeder systems.

Figure 18-3 shows the general layout of the power plant.

Boiler start-up fuel is also required using light fuel oil (Distillate oil #6). Heavy fuel oil (HFO) is required for the 29 MW HFO Power Plant. Table 18-4 provides the estimated fuel and lube oil balance.

Table 18-4: Fuel and Lube Oil Balance

	Coal Plant	HFO Plant	Total
Coal, tons	500,000		500,000
HFO, tons		12,500	12,500
LFO, tons	500	400	900
Steam Turbine Oil, tons	80		80
Generator Lube Oil, tons	80	5	85
Engine Oil, tons	40	65	105

Raw water shall be taken and piped from the mine raw water pond to the power plant water supply and treatment plant. The raw water tank system is anticipated to include two 5000-ton concrete tanks with capacity for seven (7) days.

Table 18-5 presents the anticipated water supply balance for power plant operation for 60,000 TPD mine processing capacity.

Table 18-5: Power Plant Fresh Water Balance

Boiler Make Up Water	tph	16.87
Demineralization Plant Loss	tph	0.34
Plant Service Water	tph	3.00
Plant Domestic Water	tph	2.00
Ash Handling Plant Wash	tph	2.00
Total Filtered Water Flow	tph	24.21
Filtering Loss	tph	0.24
Total Pre-Filtering Water Flow	tph	24.45
Clarifier Loss	tph	0.24
Other Users	tph	4.19
Total Raw Water Flow	tph	28.88

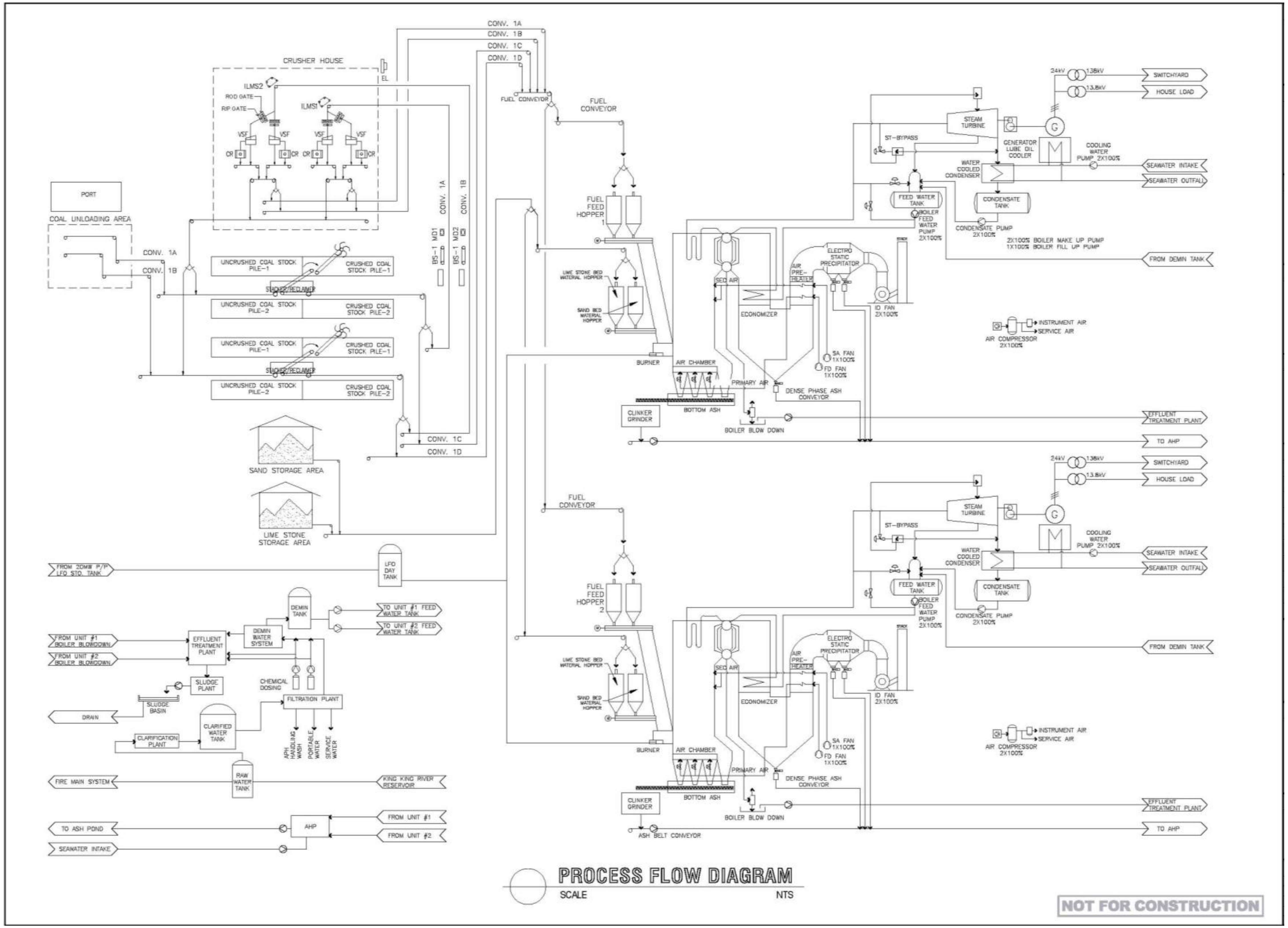


Figure 18-1: Coal Power Plant Process Diagram

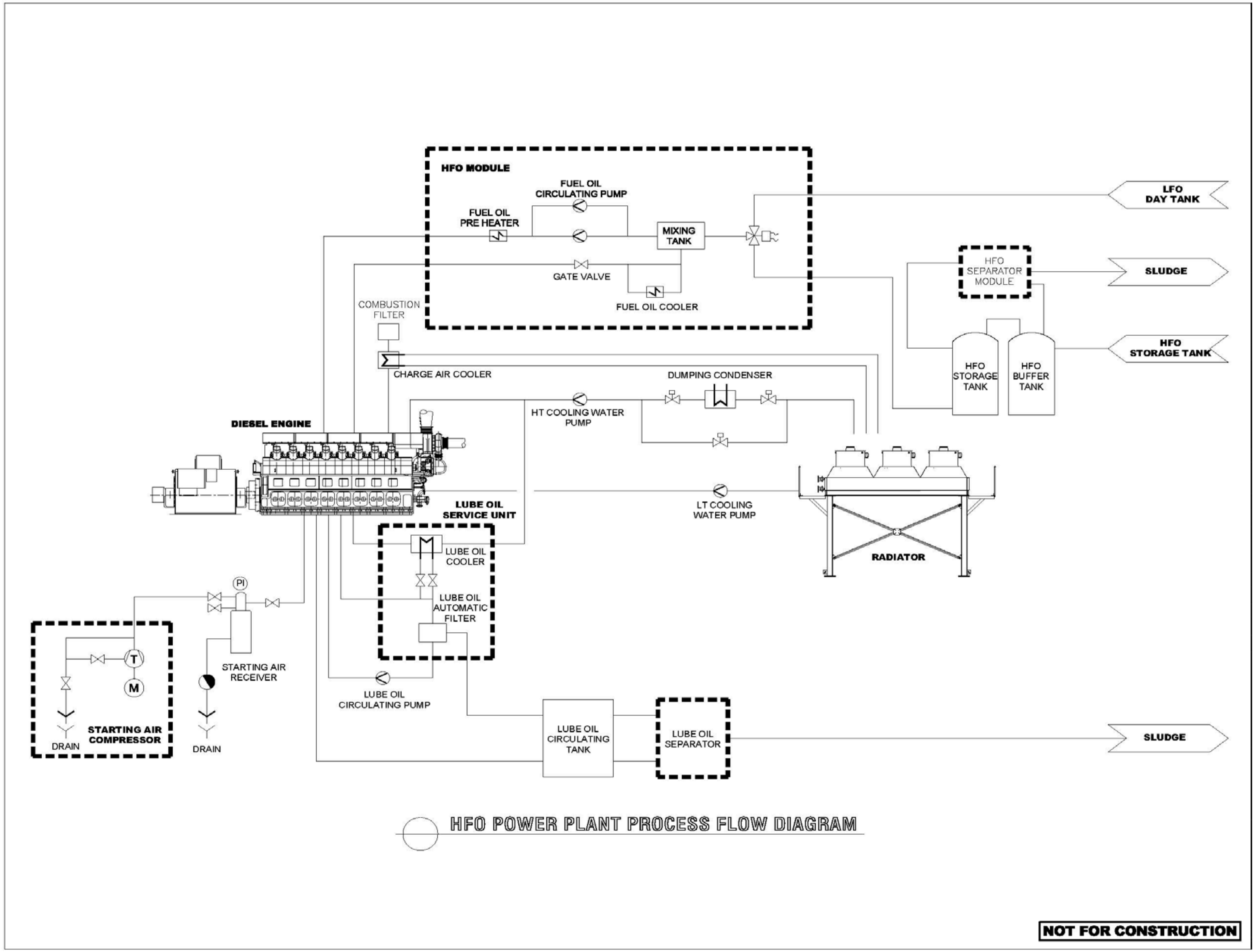


Figure 18-2: HFO Power Plant Process Flow Diagram

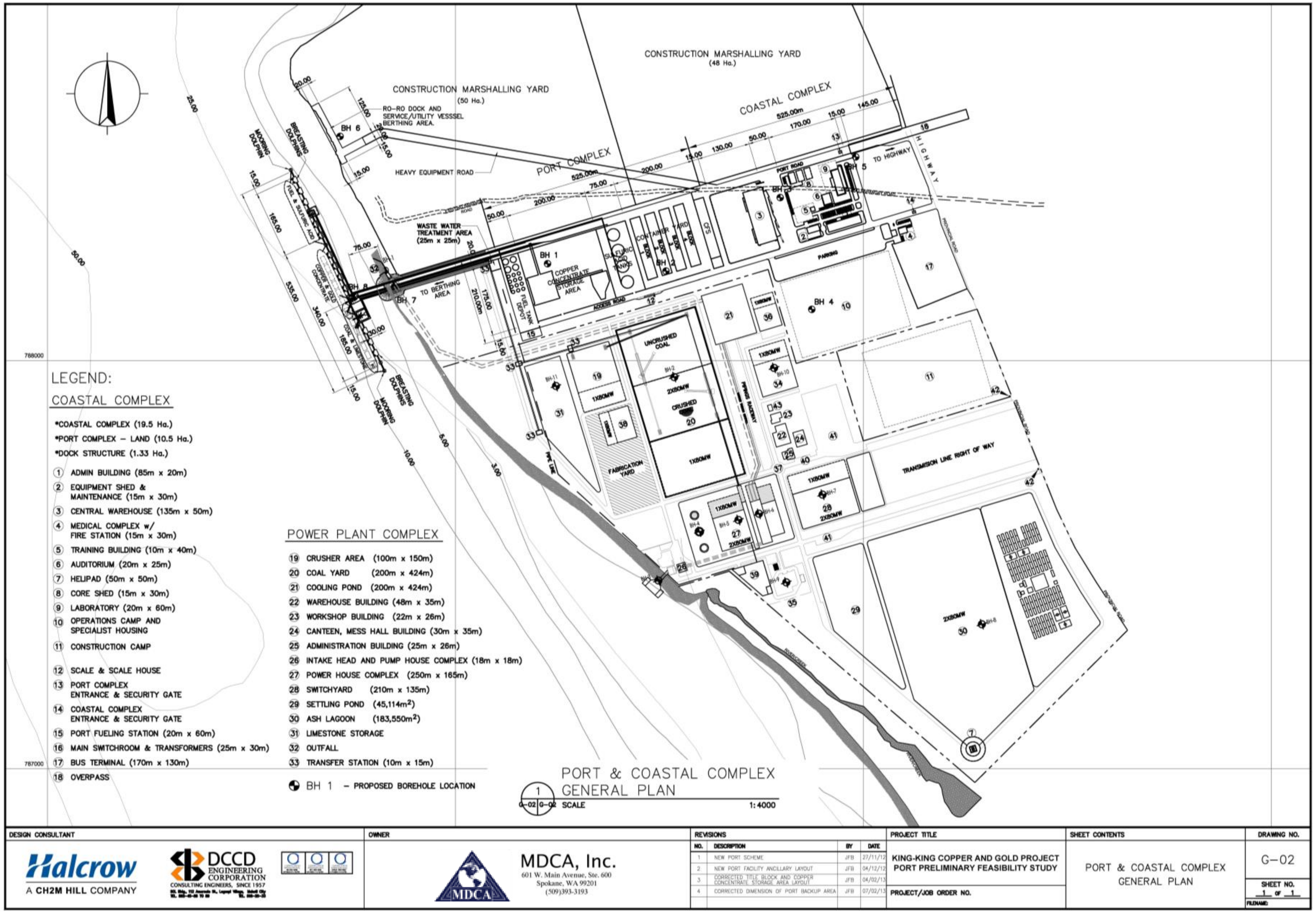


Figure 18-3: Port & Coastal Complex General Plan

The power plant shall be located near the shore to take advantage of the lower cost of seawater cooling and coal handling infrastructure. Cooling water will be taken from Davao Gulf through the intake water head and cooling water canal. Seawater will be treated for algae by chlorination. The approximate chlorine requirement shall be 2 TPD. The cooling water intake house shall be equipped with rotating and fixed trash racks and screen to prevent ingress of foreign objects into the cooling water channel. Table 18-6 provides the approximate seawater balance for the plant.

Table 18-6: Seawater Demand Balance

Intake Flow	tph	25,716
Condenser Cooling Water Flow	tph	24,016
Condenser Auxiliary Cooling Flow	tph	1,300
Ash Handling Flow	tph	400
Ash Handling Evaporation Loss	tph	120
Outfall Flow	tph	25,596

18.3 POWER DISTRIBUTION

Electrical power for the plant and mine will be transmitted from the power plant via a 138 kV transmission line over a distance of approximately 7 km.

At the plant site the 138kV transmission line will terminate at a substation. Two 70/93/116 MVA transformers in the substation will transform voltages to 34.5 kV for distribution to the various processing areas. A harmonic filter/capacitor bank is added at the substation to correct power factor to 95% and mitigate the effects of harmonics that will be generated by the cyclo-converters used to operate the grinding mills. A 2-MW diesel generator is also added at the substation to provide power to essential services in the event of an extended power outage. The estimated load for the plant including mine and well field after power factor correction is 90 MVA. One transformer will be capable of supplying power to the facility should one of the two transformers fail. The 34.5-kV switchgear is arranged so that both transformers can operate to share the load without being paralleled. The substation is monitored by a PLC connected to the process control system via fiber optics to provide status indications and alarms.

Power to the ancillary buildings, the primary crusher, the mine, water wells and booster stations, tailing dewatering and leaching facility will be by 34.5-kV overhead power lines on wood poles. Power to processing areas such as grinding, flotation, etc. will be by cables in underground duct banks. Transformers will be provided at the various processing areas to reduce voltages to the appropriate utilization voltages. Switchgears and motor control centers in all areas will control power, provide protection for equipment as required, and will be connected to PLCs and to the plant process control system via fiber optics for monitoring and control.

18.4 PORT FACILITY

A dock facility will need to be constructed in order to support the Project.

The new dock facility will be located on the northeast coast of the Davao Gulf where the Kingking River flows into the gulf. The dock will be a part of the coastal complex which will comprise a power plant, a concentrate storage facility, operations and construction camp, and access roads supporting the gold and copper open pit mine. When completed, the dock facility will handle the loading of ore concentrate via a conveyor system, unloading of coal and limestone from Panamax and Handymax vessels, unloading of sulfuric acid, fuel and heavy cargo, and a dock for service or utility vessels.

The dock will comprise three systems, namely, berth, transfer and delivery systems. In each system, there are facilities to be provided and corresponding operations to be undertaken. The facilities under the berth system are the berthing spaces and apron where the vessels will dock and where the unloading and loading of cargoes to and from the vessels take place. Facilities under the transfer system are the open or closed storage areas as part of the backup area of the port where cargoes coming from the apron and from the yard gates are shuttled or transported to be stacked at the storage facilities. The facilities under the delivery system are the yard gates where the outgoing cargoes are brought out of the port to be delivered to the consignees and where the incoming cargoes are brought into the port by the shippers. The support facilities are also usually located at the back-up area of the port. These facilities are the administrative office building, maintenance shop with a wash area and refueling station, equipment shed, stevedore's lounge, control room, power station and amenities.

18.4.1 Dock

18.4.1.1 Berthing Facility and Causeway

The apron will require a platform where the two plants (one loading and one unloading) shall be placed. The platform length is 70 m and the two plants will be placed at each end. The width of the platform is 30 m.

On the left side of the platform are three breasting dolphins with clear distances of 15 m between dolphins. On the right side of the platform are four breasting dolphins with the same 15 m clear distance between dolphins. Hence, on each side of the platform are berthing facilities with breasting dolphins. The total length then of the berthing facility is 330 m. This length is based on the LOA (length over all) of the vessel of up to 185 m. Loading and unloading of dry bulk cargo will share this berth and will therefore require scheduling of ship calls.

The deck elevation is at 4.50 m. The full load draft of the controlling vessels is 10.50 m to include the clearance and the fluctuation. The design water depth is 12.00 m below the MLLW (mean lower low water).

The length of the liquid bulk berthing facility is 165 m and is equal to the overall length of the vessels that would deliver sulfuric acid. Since liquid cargoes will be unloaded by pumping, the apron will only require accommodating the unloading pumps for sulfuric acid and fuel. The apron will be 20 m x 20 m and will have two breasting dolphins on each side. There will be 4 dolphins with a clear distance of 15 m between them. The overall distance is 165 m.

The two dedicated berths to be provided at the project site are a berth for the dry bulk cargoes and a berth for the liquid bulk cargoes. Except for the platform for the loader/unloader plant and for the pumps, the berthing facilities for both berths are composed of breasting dolphins. The platform shall be connected to the port back-up area by means of a trestle/causeway bridge that is

75 m long and 20 m wide. The width of the bridge is allocated as follows: 11 m for conveyor lines, 8 m for vehicle access, and 1 m for a pipe rack.

18.4.1.2 Loading and Unloading

The dry bulk loading and unloading operation is based on a swinging fixed loader and unloader, and this would mean there is no need for an apron for the entire length of the berth. The disadvantage with the fixed loader and unloader scheme is that this would require the vessel to be shifted forward and backwards to facilitate loading or unloading. The advantage of this, however, is that a shorter apron accommodates the loading and unloading plants, thus minimizing the cost of the structure.

The liquid bulk unloading will be through the liquid bulk apron and this will be conveyed through pipes and pumps. Separate pumps and piping will be utilized for sulfuric acid and for fuel.

The containers will be handled via the container Port of Sasa-Davao and there will be no container berth at the project site. Another option would be for the container vessels to anchor near the King-king Port and to transfer the containers in barges and unload at the roll-on/roll-off (RORO) berth.

Table 18-7: Expected Cargo Traffic

COMMODITIES	VOLUME PER YR. (MT)	PACKAGING
1. INCOMING		
1. Coal	600,000	Dry bulk
2. Limestone	88,000	Dry bulk
3. Sulfuric Acid	940,000	Liquid bulk
4. Diesel	47,600	Liquid bulk
5. Diluent	2,332	Liquid bulk
6. HFO/LFO	8,905	Liquid bulk
7. Lime	52,000	Containerize
8. Concentrator Reagents	2,695	Containerize
9. Concentrator Spare Parts	varies	Containerize
10. Equipment Spare Parts	varies	Containerize
11. SX-EW Reagents	76	Containerize
12. SX-EW Spare Parts	varies	Containerize
13. Explosives	21,500	Containerize
14. Grinding Balls	28,000	Containerize
15. Extractant	210	Containerize
2. OUTGOING		
1. Gold/Copper Concentrate	250,000	Dry bulk
2. Cathodes	120,000	Containerize

18.4.2 Conveyance Systems

18.4.2.1 Containers

The containers will be handled via the Port of Sasa-Davao and from there will be transferred to the port facility by road. Trucks will be used to transfer the containers to the container yard if these were handled at the RORO berth of the port facility.

18.4.2.2 Dry Bulk

Dry bulk will be transferred by conveyor systems. Coal and limestone will have a dedicated unloading conveyors and concentrate will also have a dedicated loading conveyor.

18.4.2.3 Liquid Bulk

The liquid bulk will be handled through pipes and pumps. Separate pumps and piping will be utilized for sulfuric acid which will be from the liquid bulk platform and will connect to the sulfuric acid tank area. Fuel piping will also connect from the liquid bulk platform all the way to the fuel tank depot area.

18.4.3 Storage

18.4.3.1 Containers

The container yard will comprise 4 stacking blocks, with a total capacity of 1,920 twenty-foot equivalent units (TEU). Each block will have eight container rows, 15 container columns and up to four containers high. Hence, a block could accommodate 480 TEUs (8 x 15 x 4) with 120 TEU ground slots (8 x 15). Four container stacking blocks would be required to accommodate 1,800 TEUs (1800/480).

18.4.3.2 Dry Bulk

The facilities to be provided at the coastal complex copper concentrate storage area are the following: a copper concentrate truck unloading and concentrate stacking system, a copper concentrate storage building (62 x 69 meters), a copper concentrate underground draw point system and conveyor to transfer copper concentrate to the ship loading copper concentrate conveyor.

18.4.3.3 Liquid Bulk

The capacity of the fuel tanks must be able to at least handle the volume to be delivered at each ship call. The number and dimensions of the tanks provided per type of fuel are as follows:

1. Diesel = 4 units x 15 m diameter x 4 m high
2. HFO, LGO and Diluent = 12 Units x 8 m diameter x 4 m high

Four tanks, 30 m in diameter and 15 m high, will be provided for sulfuric acid.

18.4.4 Weighing and Access Control

A scale and a scale house with communications and computing facility will be included as part of the port upland area. Also included in the upland area is the entrance/security gate, also with communications and computer systems to provide security and access control for the entire coastal complex.

The port facility general plan is shown in Figure 18-3 above (Port and Coastal Complex – General Plan).

18.5 TAILING STORAGE FACILITY

18.5.1 Background

The King-king Project is situated in a region with few traditional slurried tailing options for on-land storage. Given the seismicity and hydrology, a traditional upstream constructed dam is not recommended. Furthermore, a rockfill dam will require approximately the same volume of rockfill as the volume of tailing given the nature of the topography in the potential Tailing Storage Facility (TSF) areas. The proposed project throughput is considerable and a robust TSF that is compatible with the challenges of the topography and the environmental setting is required. Therefore, after performing an options evaluation (AMEC, 2011a; 2011b), a dewatered drystack tailing option is being proposed as the preferred alternative for the project at this stage. Deep sea tailing placement would be an attractive alternative, but social and political opposition to this technology would probably prevent it from being permitted in this region.

18.5.2 Design Criteria

Tailing will be delivered at approximately 55% solids (by weight) to a dewatering plant located near the drystack facility where the flotation tailing will be dewatered to reduce the moisture content at or near its optimum moisture content for compaction. Laboratory bench-scale vacuum and pressure filtration tests on representative tailing from the proposed process were performed at AMDEL Labs (Adelaide) under the guidance of AMEC Australia and at Pocock Labs (Salt Lake City). The results were reviewed and accepted by a qualified supplier, which sized and quoted horizontal-belt vacuum filters. Given the topography and throughput, a fleet of articulated trucks were chosen as the preferred method for delivering the dewatered tailing to the TSF.

The TSF design is currently based on the following production schedule:

- Average mill production rate is 60,000 tpd with an expected life of approximately 23 years.
- The total tailing tonnage to the drystack facility is 523 Mt.

The drystack facility will be constructed with dewatered tailing being placed as either structural shell tailing or general placement tailing, with the former being placed when conditions are the most favorable. Table 18-8 presents the representative volumes for each zone. The ratio of general to shell tailing is approximately 50:50 for the present configuration.

Table 18-8: Drystack Storage Requirements

Production Plan	Total Tonnage (Mt)	Ultimate Height (m)	Shell Tailing (~50% of total)		General Placement Tailing (50% of total)	
			Mt	Mm^3	Mt	Mm^3
Base Case	537.5	222	261.3	153.7	276.2	172.6

The geotechnical nature of the design is based on the sufficient strength of the structural shell, which eliminates the need for external buttressing. The benefits of this design are lower initial and sustaining capital costs.

18.5.3 General Facility Design Concepts

Dewatered tailing will be loaded by excavators onto articulated haul trucks and transported from the dewatering plant to the drystack storage facility, spread with dozers and compacted. Test work indicates that the optimum moisture content determined by a standard Proctor density test will be an appropriate target moisture content for the dewatered tailing. This should allow the material to be placed and compacted near that moisture content if time-appropriate spreading and compaction are carried out in an integrated deposition management scheme.

Lift thicknesses for the structural shell (downstream shell and on the flanks of the drystack facility) is best determined via field trials with actual site spreading and compaction equipment. For this level of study, it is assumed that the structural shell will be developed in 0.6 m-thick loose lifts. The tailing will be stacked, loaded, hauled and then spread to the target lift thickness and then compacted with appropriate compactive effort by dedicated equipment.

The general placement area will be used year-round, but exclusively during periods when conditions are such that the placement and compaction of these materials may be less than optimal (e.g. heavier rainfall). Similar compactive effort and placement procedures will be used in the development of the structural shell and for the general placement area. Lift thickness in the general placement area may be permitted to be moderately thicker than the structural shell; however, adequate compaction of the general placement area is still required to ensure trafficability of the haulage equipment.

A plan view of the drystack facility is shown in Figure 18-4 with a cut-away on top to show some of the underlying drainage and decant systems more clearly (would be all grey at very top otherwise).

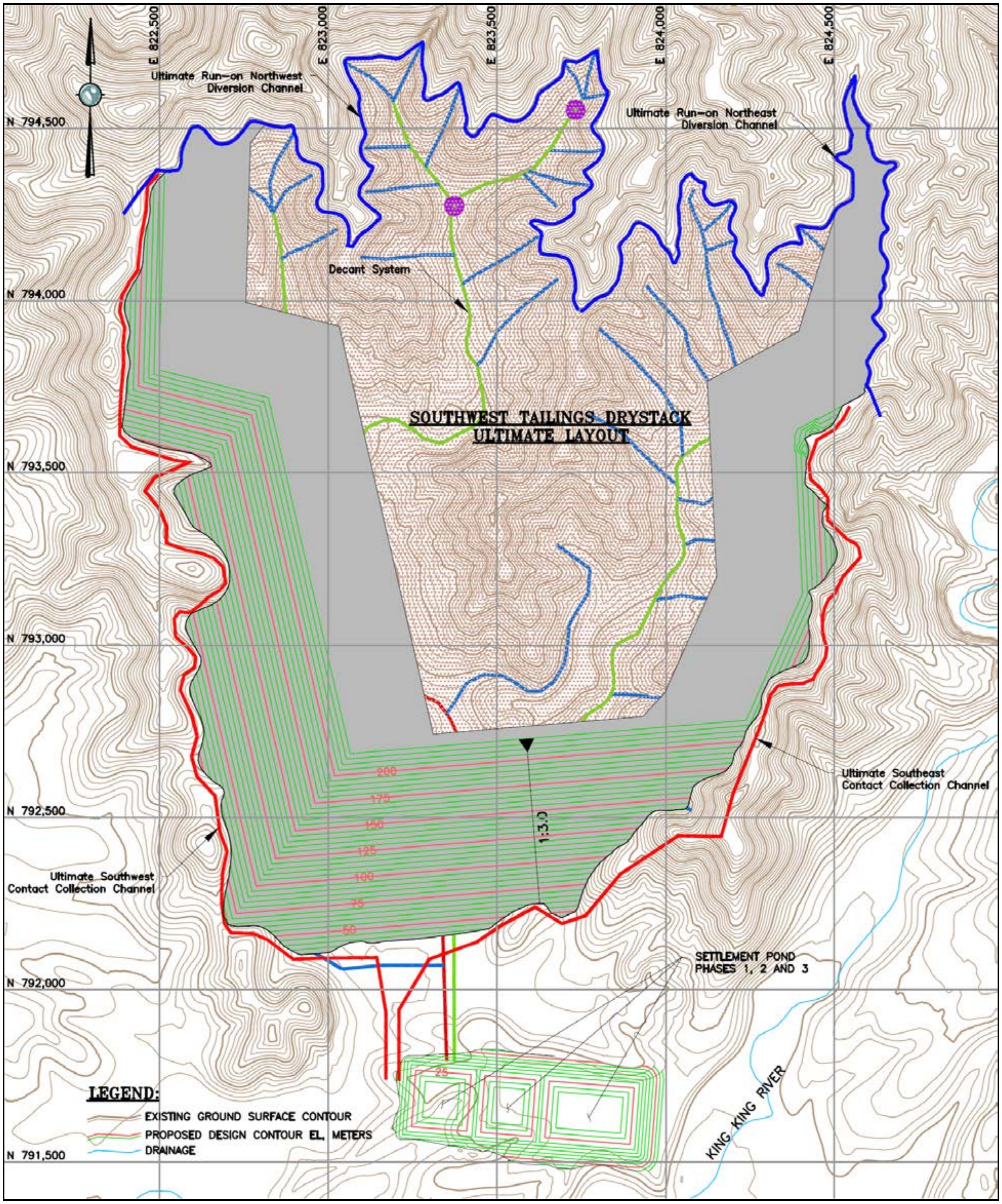


Figure 18-4: Southwest Tailing Drystack Facility Ultimate Layout

The drystack facility will be designed to include flow-through drains constructed of non-mineralized rock to manage potential foundation seeps. The drains will be situated in the valley bottoms to limit moisture ingress into the drystack facility from groundwater sources. The flow-through drains will also require filter materials for transition between the rockfill and overlying tailing, which could be provided by the use of suitably graded granular material or with a geosynthetic filter layer. Rockfill material may also be used within the drystack facility to improve trafficability and to facilitate progressive reclamation of the exterior slopes of the drystack facility that have been constructed to their final configuration.

18.5.4 Surface Water Management

One of the most significant issues of concern with drystacking is surface erosion from unmanaged runoff. Surface runoff on the face of the drystack facility shell has to be aggressively controlled at all stages of facility development. Surface flows on the general placement area are less likely to cause erosion since the slopes in this area are generally flat. Erosion of the dewatered tailing pile will become a concern if it is allowed to develop to the point where it compromises the structural integrity of the compacted outer shell zone and if high sediment loads generated by erosion are not properly managed (i.e. impact receiving waters).

Two primary types of diversion channels are planned for the Southwest Tailing Drystack Facility: (i) run-on channels; and (ii) contact water collection channels. Run-on channels will be used to divert non-contact storm water around the Southwest Tailing Drystack Facility and prevent flow onto the filtered tailing surface. Contact collection channels will collect contact runoff from impacted surfaces and convey it to sediment control storm water ponds. Figure 18-4 shows the locations of the ultimate run-on channels and contact water collection channels. Temporary run-on channels will be constructed during the life of the facility. The ultimate diversion ditches will be sized to accommodate the peak runoff associated with a 100-year return period, 24-hour rainfall event without failing.

The drystack facility will also require a drainage system to collect runoff from the top of the stack and convey it to the collection pond located at the toe of the facility. Surface runoff from the drystack facility will be routed to a storm water collection pond through a subsurface decant system. For preliminary sizing of the decant system, it is assumed that the surface of the stack will be graded at an approximate 0.5 percent average slope to the north. At the north end of the stack, storm water will be routed to two designated collection locations where it will enter the decant system.

The proposed storm water collection pond will be located south of the drystack facility. Water from the contact collection channels and runoff from the surface of the drystack facility will be routed to the sediment control pond where suspended solids will drop out. The sediment control pond will have to be constructed such that periodic removal of accumulated sediment can take place. The pond will also contain flows from the underdrain system, which is anticipated to be a small contributing volume, and not considered explicitly for this level of design.

18.5.5 TSF Slope Stability Evaluation

Stability analyses were performed on two sections of the proposed facility, conservatively based on the larger expanded facility. Section A-A (Figure 18-5) considered the maximum cross-section through the downstream shell of the Southwest Tailing Drystack Facility. Section B-B (Figure 18-6) considered a cross valley section through the general placement tailing. Static and pseudostatic slope stability analyses were conducted under effective stress and total stress conditions using the computer program *SLIDE 5.0* (Rocscience, 2007) to estimate the least stable failure via a critical surface search routine. The design criteria pertinent to the stability requirements are:

- Minimum factor of safety under static conditions = 1.2 (DENR, 1999), with a desired minimum static factor of safety in excess of 1.5;
- Minimum factor of safety under seismic (pseudostatic) conditions = 1.0, with deformation analyses to be performed for pseudo-static factors of safety less than 1.0 to confirm acceptable deformation. DENR Memorandum Order No. 99-32 indicates that the factor of safety should be a minimum of 0.98 (DENR, 1999);
- Operations Basis Earthquake (OBE) peak ground acceleration (PGA) = 0.6g (AMEC, 2011c); and
- Maximum Design Earthquake (MDE)[2] PGA = 1.01g (AMEC, 2011c).

18.5.5.1 Method

For the failure mechanisms considered in the analyses, slope stability was evaluated using limit equilibrium methods based on Spencer's method of analysis (Spencer, 1967). The pseudostatic analyses conservatively model seismic events as constant acceleration and direction. Therefore, it is customary for geotechnical engineers to take only a fraction of the predicted peak maximum acceleration when modeling seismic events using pseudostatic analyses. For this analysis, a seismic coefficient of half the horizontal peak ground acceleration (PGA) was used to evaluate the facility under seismic loading, which is equal to 0.30 and 0.505 for the OBE and MCE, respectively, representing a conservative approach.

18.5.5.2 Material Properties

For purposes of this analysis, the drystack system is assumed to be composed of two main material types:

- *Foundation* – Foundation materials are the original ground the drystack facility will rest on after it has been stripped and prepared for material placement. It is assumed that bedrock is relatively shallow in the proposed drystack facility area and that the structural

[2]The median MCE has a PGA of 1.15g. Because a PGA of 1.01g is associated with a return period of 2475 years, ground motions associated with the MDE could be selected as the median MCE, as they are likely greater than the 3,000 year ground motions, as recommended by ICOLD (1995).

shell of the drystack facility would be founded on bedrock (any foundation soils removed).

- *Filtered Tailing* – Filtered tailing will be managed as two zones, shell and general placement tailing. Parameters for the filtered tailing were derived from the limited testing results and experience from similar projects.

Material parameters used in the stability analyses are presented in Table 18-9.

Table 18-9: Material Parameters used for Stability Analyses

Zone	Bulk Density	Strength	
		Static	Psuedo-static
Foundation Bedrock	19.5 kN/m^3	ϕ' = 40°, c' = 25 kN/m^2	ϕ' = 40°, c' = 25 kN/m^2
Shell (Compacted Tailing)	16.5 kN/m^3	ϕ' = 32° c' = 0 kN/m^2	ϕ' = 32° c' = 0 kN/m^2
General Placement (Tailing)	15.7 kN/m^3	ϕ' = 31° c' = 0 kN/m^2	ϕ' = 26.6°* c' = 0 kN/m^2

*Shear strength of general placement tailing was reduced 30% to conservatively account for localized zones of general placement tailing that may have liquefied during the seismic event.

18.5.5.3 Preliminary Results

Circular failure surfaces were modeled under static and seismic conditions for the cross-sections considered. Figure 18-5 and Figure 18-6 illustrate the evaluated cross-sections A-A and B-B, respectively, showing the circular failure surface with the minimum factor of safety for the static loading condition.

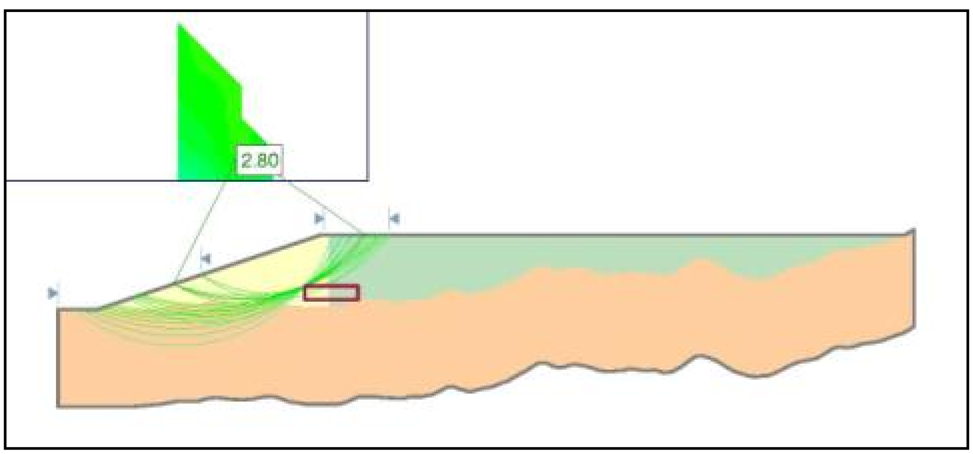


Figure 18-5: Cross Section A-A- Stability Evaluation

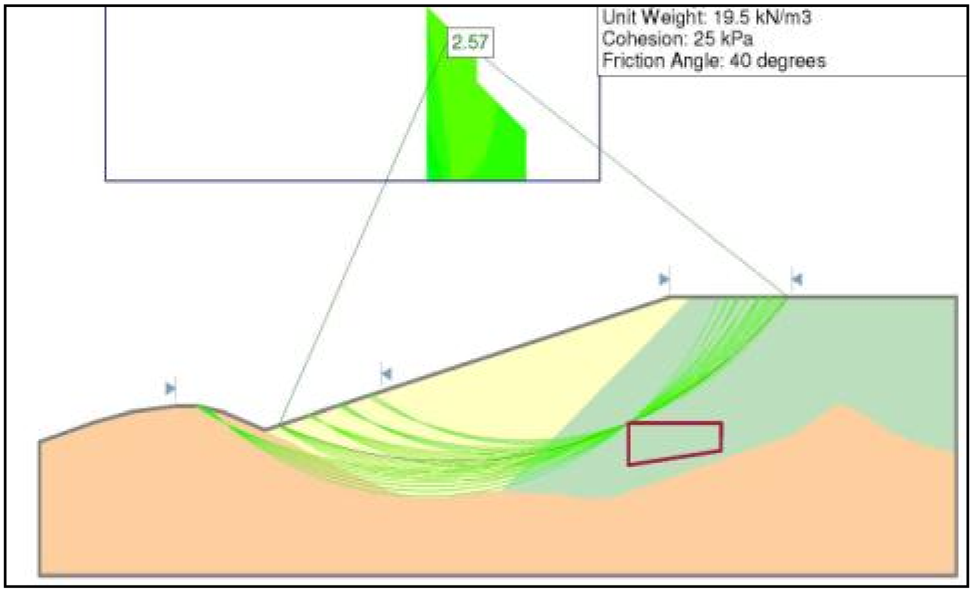


Figure 18-6: Cross Section B-B- Stability Evaluation

18.5.5.4 Results – Discussion

Philippine regulations (DENR, 1999) require that the minimum factor of safety be within the range of 0.98 to 1.2 under seismic loading conditions, with deformation analyses required for lower factors of safety. Considering the static and OBE loading conditions, the stability analyses results satisfy the required minimum factor of safety for the drystack facility. The results of the stability evaluation are presented in Table 18-10.

Table 18-10: Stability Analyses Results

Section	Factor of Safety		
	Static	Psuedo-Static (OBE)	Psuedo-Static (MCE)
A-A	2.80	1.25	0.94
B-B	2.57	1.21	0.91

Notes:
1. Assumes no development of a phreatic surface within the drystack facility.
2. For the OBE, a pseudostatic load coefficient of 0.3 was assumed. For the MCE, a pseudostatic load coefficient of 0.505 was assumed.

Under the MCE, the required factor of safety is not satisfied. However, the values are still close to unity and experience with non-brittle materials has shown values close to unity will lead to deformations that will be manageable by the operation. Section 18.5.5.5 presents a summary of the “first-cut” deformation analyses that have been carried out for the MCE.

18.5.5.5 Evaluation of Seismically-Induced Permanent Deformations

The drystack facility will likely experience some deformation during larger earthquakes. As a result, simplified approaches for evaluation of seismically-induced permanent deformations can be used. In this case, Makdisi and Seed (1978) and Bray and Travasarou (2007), both Newmark-based simplified slope displacement procedures, were used.

Based on the existing seismicity information at the site, it is anticipated that the seismically-induced permanent deformations resulting from the 2,475-year recurrence interval earthquake event will be less than a meter.

18.6 VALUELESS ROCK STORAGE

18.6.1 VRMA

Based on the current mine production schedule, the valueless rock production for the project is anticipated to be 657 million tons. Most valueless rock will be deposited in VRMA, and some of the suitable material in the early years will be used for construction. In the later years of the mine life, the mine plan will be evaluated to determine the potential for partial pit backfill. If the remaining resource and potential for future development along with operational considerations allow partial backfill, the potential for reduced environmental impacts will be realized through reduced tons placed in the VRMA. Previous mine schedules have shown that partial backfill can be a viable option.

Two VRMA facilities were advanced to preliminary feasibility level design. The preferred location, referred to as the Southwest VRMA, is located south and west of the pit. The alternative option, referred to as the West VRMA, is located directly west of the pit northwest of the Kingking River. The current VRMA design utilizes portions of both locations.

18.6.2 Design Criteria

The VRMA is anticipated to be constructed in 30-meter lifts with initial slopes of 1.5H:1V (horizontal:vertical). The lifts will be constructed with appropriate set-back benches to achieve an overall 3H:1V slope. The lift slopes will be reduced to 2.5H:1V during progressive reclamation.

The valueless rock properties that were used in design include:

- Bulk Densities:
  - Oxide – 2.4 t/m^3;
  - Sulfide – 2.5 t/m^3; and
  - Overburden – 2.0 t/m^3.
- 56 percent of sulfide valueless rock and 46 percent of the oxide valueless rock are reported to be Acid Generating (AG) (AATA, 2011); and
- Delivered moisture content of 3% (KP, 1996).

While most of the valueless rock materials will be deposited at VRMA facilities, some of the suitable material in the early years will be used for construction.

18.6.2.1 Surface Water Management

Surface water management of the VRMA has three components: impacted diversion channels, underdrains, and a collection pond. Both stacking areas of the VRMA are constructed in distinct natural drainage basins. Because of this, any precipitation that falls within these drainage basin areas is considered impacted water (water that has come into contact with the valueless rock). Only the southwestern stacking area will require the construction of downstream collection channels to collect impacted water and convey it to a pond for testing and potential treatment prior to discharge.

Downstream collection channels were based on design criteria to contain flows resulting from the 100-year, 24-hour design storm. The magnitude and timing of the peak discharge resulting from this storm was calculated using the hydrological modeling system *HEC-HMS*, version 3.4. The precipitation depth associated with the 100-year, 24-hour design storm event was assumed to be 310 mm based on revised climate data (AMEC, 2012). The USACE HEC-RAS hydraulic modeling software, version 4.1, was utilized to size the diversion channels to effectively transport the discharge from the design precipitation event. For this preliminary design, it was assumed that a trapezoidal channel would be built with 2.5H:1V side slopes and a minimum 300 mm of required freeboard. In areas where high velocities occur grouted riprap may be utilized (if the channel is not in bedrock) to assure the channel is stable during high flow periods.

Foundation underdrains will be installed in the major natural drainages beneath the VRMA facility to assist in controlling surface water that has filtered through the valueless rock. Foundation drains will only be constructed in approximately the first 500 m of the stack and will release any captured precipitation to the downstream collection channels (for the southwest stacking area) or directly into the collection pond (for the west stacking area).

The collection ponds act to store the impacted water before treatment (if required) or controlled release. Collection ponds were sized sufficiently to store the 100-year, 24 hour storm. The Southwest VRMA Pond 1 will be constructed in two phases to accommodate the expansion of the stack.

The VRMA is shown in Figure 18-7 with associated diversion channels and ponds.

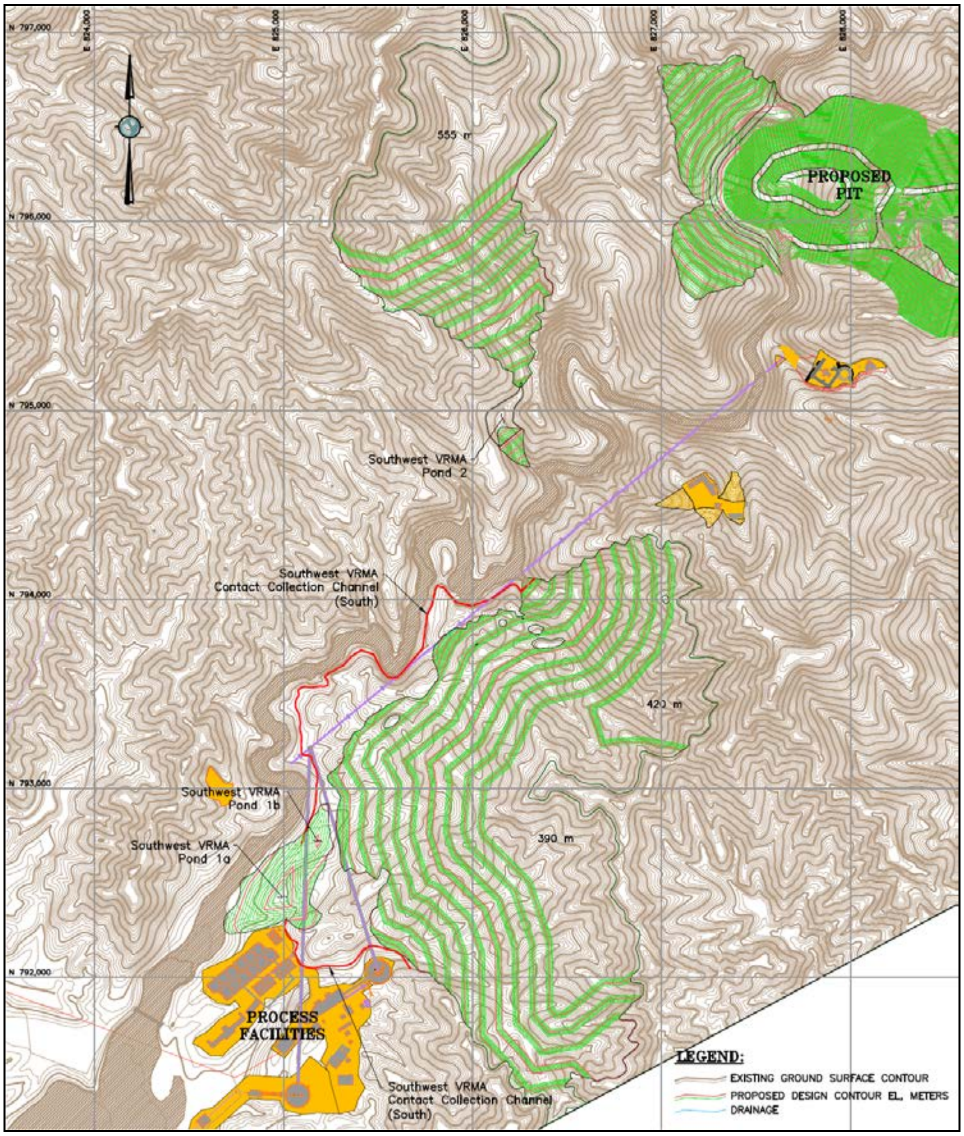


Figure 18-7: Proposed VRMA Ultimate Layout

18.6.2.2 Concurrent and Final Facility Closure

As a significant portion of the VR material is anticipated to consist of AG material, the concept being advanced considers reclamation of the exposed slopes of the VRMA concurrently during VRMA development to limit the amount of VR exposed. Overall reclamation slopes are anticipated to be achieved by a dozer working on the slope. A reclamation cover system that will limit the degree of saturation of the VR has been included in the sustaining capital of the VRMA to reduce the potential for ARD.

18.6.3 VRMA Slope Stability Evaluation

Slope stability analyses were conducted using the computer program *SLIDE 5.0* (Rocscience, 2007) to estimate the least stable failure surface via a critical surface search routine. The maximum cross-section through the Southwest VRMA with a downstream slope of 3H:1V was considered under seismic loading, as this is considered the most critical situation for stability of the VRMA for the King-king Project. The design criteria pertinent to the stability requirements for the VRMA include:

- Minimum factor of safety under seismic (pseudo-static) conditions = 1.0, with deformation analyses to be performed for pseudo-static factors of safety less than 1.0 to confirm acceptable deformation. DENR Memorandum Order No. 99-32 indicates that the factor of safety should be a minimum of 0.98;
- Operations Basis Earthquake (OBE) peak ground acceleration (PGA) = 0.60g; and
- Maximum Design Earthquake (MDE) PGA = 1.01g.

18.6.3.1 Method

For the failure mechanisms considered in the analyses, slope stability was evaluated using limit equilibrium methods based on Spencer's method of analysis (Spencer, 1967). The pseudostatic analyses conservatively model seismic events as constant acceleration and direction. Therefore, it is customary for geotechnical engineers to take only a fraction of the predicted peak maximum acceleration when modeling seismic events using pseudostatic analyses. For this analysis, a seismic coefficient of half the horizontal peak ground acceleration (PGA) was used to evaluate the facility under seismic loading, which is equal to 0.30 and 0.505 for the OBE and MCE, respectively, representing a conservative approach.

18.6.3.2 Material Properties

For purposes of this analysis, the VRMA is assumed to be composed of two main material types:

- *Valueless Rock* – Valueless rock material is rock that is mined from the pit but contains no valuable ore. Sensitivity analyses were performed to estimate the minimum required effective friction angle (Φ') to achieve stability. Other values were assumed, including a bulk density (γ) of 18.5 kN/m³ with no cohesion. A thorough geotechnical investigation is planned to provide additional information on the valueless rock materials to support the feasibility-level design.
- *Foundation* – Foundation materials are the original soils the facility will rest on after it has been stripped and prepared for material placement. Assumed values used for this analysis include a bulk density (γ) of 19.5 kN/m^3, effective cohesion (c') of 25 kN/m^2, and an effective friction angle (Φ') of 40 degrees. It is anticipated that bedrock is relatively shallow in the proposed VRMA area, and have currently assumed that the VRMA will be founded on bedrock (stripped to bedrock).

18.6.3.3 Preliminary Results

Assuming the material parameters presented in Section 18.6.3.2, sensitivity analyses were performed to evaluate the minimum effective stress friction angle requirements for the valueless rock material to achieve stability under OBE and MCE loading conditions assuming downstream slopes of 3H:1V. Figure 18-8 presents the results of the sensitivity analysis. The results show that under OBE and MCE loading conditions, the valueless rock material requires an effective stress friction angle of 35 degrees and 45 degrees, respectively, to achieve a factor of safety of 1.0 (assuming no cohesion).

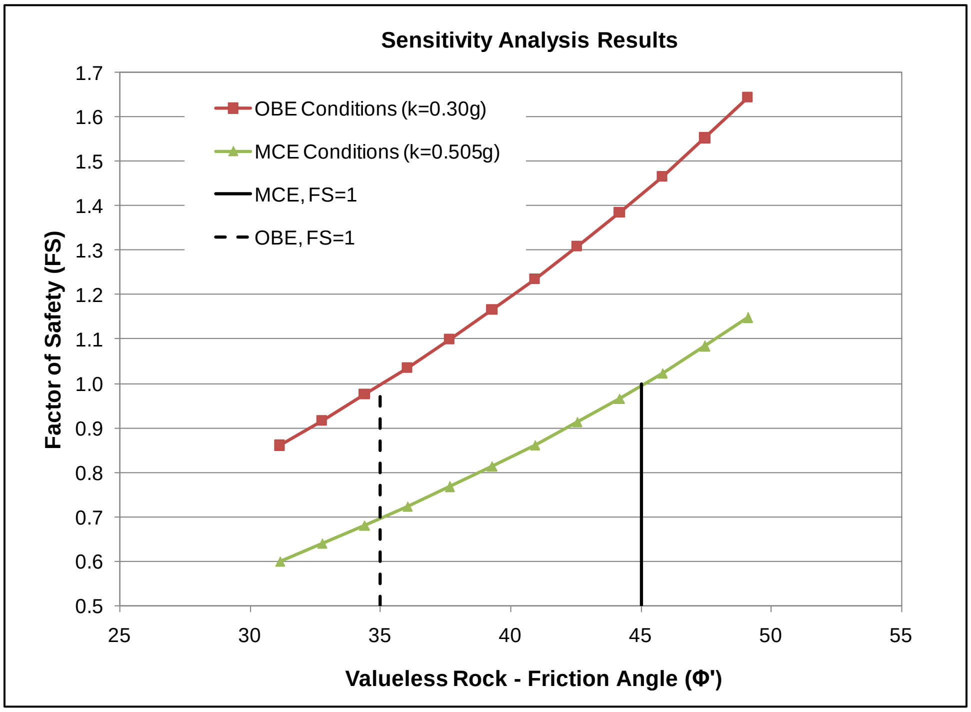


Figure 18-8: Sensitivity of Valueless Rock Material Friction Angle on Factor of Safety

Figure 18-9 illustrates the evaluated cross-section and the circular failure surface with the minimum factor of safety evaluated for the King-king Project. Based on the current configuration with 3H:1V downstream slopes, veneer failure surfaces through the downstream shell appear to control stability.

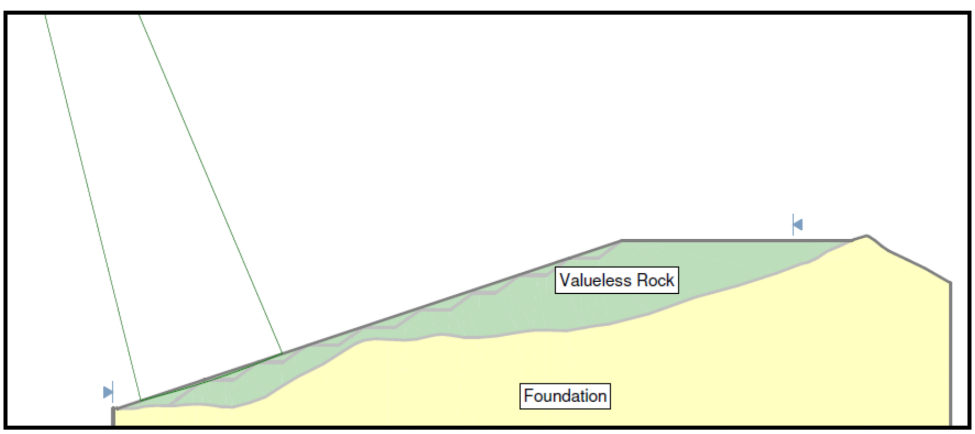


Figure 18-9: Assumed Cross Section for Stability Evaluation

Philippine regulations (DENR Memorandum Order No. 99-32) require that the minimum factor of safety be within the range of 0.98 to 1.2 under seismic loading conditions, with deformation analyses required for lower factors of safety. Deformation analysis may need to be completed during the feasibility-level design to confirm that significant damage to the VRMA would not occur as the factor of safety approximates unity, or the facility configuration may need to be modified to enhance structural stability.

18.7 WATER SYSTEMS

A well field will supply groundwater for processing for the King-king project. The process water demand is estimated to be approximately 600,000 m^3/month, or 20,000 m^3/d.

The well field would be located in the area to the west of the leach pad and to the south of the Kingking River. This area is underlain by alluvial sediments (deltaic deposits associated with the King-king and other rivers draining the highland area to the east) consisting of fine to coarse grained sand and gravel with interbedded silt and clay. Two monitoring wells (MW-14 and MW-17) drilled in this area have encountered the alluvial aquifer. The alluvial sediments thicken to the west towards the ocean and have been identified to depths of at least 120 m. While no hydrologic testing has been completed in the monitoring wells, the permeability of the alluvial deposits is assumed to be moderate to high based on the description of the geological conditions. The aquifer is recharged by the infiltration of precipitation over the alluvial aquifer and likely also by infiltration from the King-king and other rivers as they flow across the deltaic deposits. Groundwater in the aquifer discharges to the ocean at the coastline. There is no information on the quality of the groundwater.

The number of production wells and their capacity was determined using analytical methods and based on assumed hydraulic parameters for the aquifer drawn from literature on similar materials. The analyses indicate that six to eight wells in an approximate east-west line or grid layout could provide 600,000 m^3/month. The pumping rate in individual wells could range from 4,000 to 6,000 m^3/d, and the drawdown in individual wells could range from approximately 5 to 28 m. If the aquifer is of lower permeability than estimated, several additional wells would be required to meet the monthly demand of 600,000 m^3/month. The wells would draw groundwater from the alluvial aquifer with additional recharge induced from the Kingking River as a result of a lowering of the water table around the well field.

The wells would be 125 m deep and completed with 305-mm diameter stainless or carbon steel well screen and casing inside a 406-mm borehole drilled by air or fluid rotary methods. Vertical line-shaft turbine pumps would be installed in each well and would pump into one or several pipelines to convey the water to the process plant (and other potential facilities that need water). The effects of the well field would be monitored using a network of groundwater monitoring wells surrounding the well field.

In order to design the well field, a feasibility-level hydrogeological investigation is needed in the area proposed for the well field. The purpose of this investigation would be to confirm the hydrogeological conditions and provide well design parameters. The investigation would involve geophysical surveys, followed by drilling of exploration boreholes (completed with the installation of piezometers) and then by several test wells. The test wells would be pump tested to determine aquifer hydraulic parameters, groundwater quality and the effects of the well field on local ground and surface water resources. One or more of the test wells could be used as production wells depending on the method of construction and the results of the testing.

A preliminary feasibility study cost estimate for the drilling and testing of six production and three monitoring wells, and for completion of the production wells with pumps, well house infrastructure and pipelines to the process plant has been completed.

The cost estimate includes the cost for hydrogeological services for final well design and construction oversight, and engineering design and construction of the well facilities and pipelines as well as annual operations and maintenance costs.

18.7.1 Pit Dewatering

Surface water collecting in the open pit will flow by gravity in various man made channels and natural drainages to two unlined sediment control ponds during mine operating years -2,-1 and 1-4. After year 4-5, some dewatering pumps are needed to pump out the pits that form after year 4, to lift excess water into the drainages. As the pit size increases, pumps will be added to handle the increased volume and vertical pumping height. The surface water will either be collected in sediment control ponds, without further treatment other than solids removal by settling and a screen on each pond’s decant system or piped to the water treatment plant for discharge. Diversion channels around the pit will prevent as much natural runoff from entering the pit as possible.

Depressurization of groundwater around the pit will be carried out using horizontal drains bored into the pit walls in permeable layers which occur throughout the pit area. This water will also be directed to the collection ponds within the pit. Collected water will be pumped out to a surface settling pond south of the pit and decanted water will be returned to the Kingking River or piped to the water treatment plant to dilute treated effluents.

18.7.2 VRMA Runoff

VRMA water through approximately year 5 will be collected via channels and directed into a sediment control pond and then delivered by pipe or channel to the Kingking River untreated. During Years 4 and 5, a treatment plant will be constructed as the VRMA becomes PAG. VRMA water will be directed to the treatment location via lined channels (or pipeline), treated, diluted with pit surface or ground water, and discharged to the Kingking River. This will continue for the remainder of mine life. Additional lined channels will be constructed along with the continual expansion of the VRMA directing water to the single point of treatment.

18.7.3 Tailing Storage Runoff

Water from the TSF is considered non-acid generating and will be collected in sediment/storm water ponds and allowed to discharge into the Kingking River without further treatment. A small pond will be constructed sufficient for the first few years of mine life. As the TSF grows, additional ponds will be added.

18.7.4 Heap Leach Runoff

Run off and collected water from the heap leach area(s) will require treatment. A separate treatment plant will be built to serve the heap leach area only and will discharge treated water to the Kingking River. Initial assumption is discharge pipe from heap leach to Kingking River will be at most 1.9 km long.

18.7.5 Potable Water

Groundwater from the pit depressurization system will be used for potable water at the mine and process facilities. Current projections indicate it to be of reasonably good quality and available in sufficient and consistent volumes to fill the requirements initially and throughout the rest of mine life. Potable water for the coastal complex will be supplied by local wells.

Treatment will be performed in two potable water treatment plants, one located at the port facility and one located at the processing facility. Excess potable water treatment capacity could be designed to supplement the water supply of the nearby community off-site.

18.8 WATER TREATMENT

The preliminary level evaluation of water treatment requirements for the Project has been conducted to provide estimates of capital, operations and maintenance (O&M), and sustaining capital costs. The preliminary feasibility study for site-wide water treatment includes the following:

- Influent design basis water quality characterizations and flow rates for all sources of mine-influenced water (MIW);
- Treated effluent target values, based on Philippines Drinking Water standards, Inland and Marine discharge standards (2008 draft), and International Finance Corporation (IFC) guidelines;
- Source evaluation and conceptual treatment process for site potable water supply; and
- MIW commingling and treatment options to conceptually establish an optimized water treatment/management strategy.

The preliminary feasibility study for water treatment options relied on data provided by others for water quality characterization and water flow projections from each source. A deterministic site-wide water balance was provided by AMEC. Water quality data were provided by AATA for the six sources including the Drystack TSF, VRMA, HLP, SOSF, pit groundwater, and pit runoff. Assumptions affecting the development of the conceptual water treatment and management strategy include:

- Water Quantity: The AMEC deterministic site-wide water balance was used to determine source flow rates. From these, a source blending and water characterization study was conducted, taking into account monthly maximum flow rates from the sources. The needs for water treatment were based on the study projections for water quality. Average annual flows were used in the estimation of operations and maintenance costs.

- The “TSF Water Quality Estimate” Technical Memorandum (AATA, February 13, 2012) provided the basis of ten percent oxide ore and ninety percent sulfide ore in the drystack facility, and associated TSF runoff water quality.
- VRMA: VRMA water quality accounted for material encapsulation and for the proportion of acid-generating material placed, based on humidity cell and barrel test data (AATA, February 13, 2012).
- Heap Leach: Averaged analytical data provided by AATA on February 29, 2012 was used.
- Heap Leach Spent Ore Storage Facility: An assumption was made based on expected ore sent to the leach pad. A ninety percent oxide and ten percent sulfide ore quality was used based on “TSF Water Quality Estimate” Technical Memorandum (AATA, February 13, 2012). Current barrel test and humidity cell test data were also utilized.
- Pit Groundwater: Water quality characterization was estimated from analytical data provided for six (6) seeps and springs within and near pit area, and three (3) boreholes within pit area. A 35 percent contribution from seeps and springs was assumed.
- Pit Runoff: Water quality characterization was based on "VRMA Water Quality Estimate" Technical Memorandum, Table 1, "Higher Quality," (AATA, February 13, 2012).

The overall water treatment and management strategy for the site will include a multi-pronged approach that includes some water treatment and discharge of MIW, and other options such as commingling the various waters or reuse options. It is assumed that pit water treatment will not be required other than for potential use as potable water, and the heap leach water will be segregated and managed separately.

Two treatment alternatives have been developed for compliance with varying levels of treated effluent standards and discharge strategies:

- Standards for treated effluent quality based on 2008 draft Philippines effluent standards (DENR, 2008) for discharge to Class C inland and/or Class SC coastal waters focusing on both metals and total dissolved solids (TDS) treatment.
- Standards for treated effluent quality based on 2008 draft Philippines effluent standards (DENR, 2008) for discharge to Class C inland and/or Class SC coastal waters focusing on only metals treatment, assuming that a variance for sulfate and TDS can be permitted, or that a mixing zone point of compliance is acceptable to regulators.

The treatment concept for conformance with discharge standards for inland Class C or coastal Class SC waters includes the following:

- Pretreatment by oxidation and pH adjustment for iron, aluminum and manganese removal;

- Reverse osmosis (RO) treatment for concentration of metals, selenium and TDS (including sulfate). RO permeate (clean water stream) will be nearly demineralized, high-quality water. It is assumed that an 80% permeate recovery can be achieved. The secondary waste or brine stream is 20% of the influent flow, approximately 45 m^3/hr.
- RO brine treatment by chemical precipitation for metals and sulfate removal. RO brine will carry a sulfate concentration of approximately 8,500 mg/L which can be reduced to approximately 1,700 mg/L (sulfate solubility limit) by lime precipitation. The sludge produced will contain calcium sulfate and metal hydroxide.

If a mixing zone approach can be used to achieve sulfate compliance, then the treatment approach described above can be simplified to a conventional lime treatment system. Commingling of lime-treated VRMA water with the TSF or pit waters may reduce the sulfate concentration to a point where no mixing zone, or only a relatively small mixing zone, will be required to achieve compliant discharge.

The currently projected water quality for the combined waste streams (VRMA, SOSF, and TSF) meets Philippine inland and marine discharge standards. Construction and operation of a wastewater treatment facility may be deferred to later years based on current projections. The preliminary feasibility prediction that water treatment may be deferred should be revisited as additional geochemical and water balance data, developed to the feasibility level of detail, become available.

Water treatment for potable supply will be achieved with pre-engineered package systems. Two stand-alone potable treatment plants are anticipated, one at the mill site and one at the port facility. They will be designed to supply water to 4,000 people on-site initially (SAGC, 2012) and 2,000 people on-site permanently.

Sewage treatment is also expected to use pre-engineered systems. It is anticipated that two sewage treatment plants will be required at the site, one at the mill and the other at the port. Sewage treatment design flow at each site is sufficient to support a construction camp inhabited by 4000 people.

19 MARKET STUDIES AND CONTRACTS

19.1 COPPER MARKET

Copper is characterized as a ductile metal with very high thermal and electrical conductivity and is used for a wide variety of applications, primarily as a conductor of heat and electricity, a building material, and a constituent of various metal alloys. The world consumption of new copper is about 60% electrical wiring, 20% roofing and plumbing, 15% industrial machinery, and 5% alloy production, such as brass and bronze.

19.2 SALE OF MINED COPPER

Copper Mining companies traditionally produce copper in two forms: copper concentrate and pure copper cathodes. The King-king project will produce both forms.

The copper concentrate characteristics for King-king based on flotation test work at Amdel's Adelaide Lab are shown below:

- 25-75 Gold g/t
- 22-25% Copper
- 35-37% Sulfur
- 25-32% Iron
- 1000 – 3700 ppm Arsenic
- 200 – 900 ppm Antimony
- 350 – 600 ppm Selenium

Smelter and refining terms for King-king concentrate are discussed in Section 19.7.

19.3 COPPER PRICE

At the time of this study (end of November 2012), the Spot Price of Copper was US $3.62 per pound and London Metal Exchange (LME) reports a 36-month historical price average of $3.56. Futures price forecast for copper through November 2014 is estimated at $3.62 per pound based on the CME Group and the LME Futures.

For this study, M3 has used a Base Case copper price for all production years of $3.00 per pound. This conservative estimate is 17% below the November 2012 spot copper price.

19.4 GOLD MARKET

Gold is used for a wide variety of applications, ranging from jewelry and the arts to dentistry, electronics, and diverse industrial applications. It has also traditionally been used as a backing for paper currency systems and continues to be used as a hedge against inflation. In the current market, investment demand has been driving gold prices higher. Many economists have forecasted that this trend is likely to continue among several major countries of the world for the foreseeable future.

19.5 SALE OF MINED GOLD

Mining companies traditionally use Merrill-Crowe or electrowinning processes for final recovery of gold precipitates or sludge which are smelted to produce doré bars. These bars typically have gold and silver with other possible impurities and typically contain more than 95% precious metal. Doré bars require further refining to produce high-quality gold bars.

The refinery pays the mining company for the contained precious metals, charges refining fees, and sells the gold on the open market or deposits the gold in the mining company's account and bills the mining company for the refining charges.

Present day typical precious metal transportation/insurance and refining costs were used in the study:

- Transportation and Insurance: 1% of gross metal revenue
- Refining Charge: $2.00 per troy ounce of gold contained

19.6 GOLD PRICE

At the time of this study (end of November 2012), gold was trading at USD $1,762 per ounce. The 36-month historical average gold price was $1,474.39, based on London Bullion Market Association and Kitco Gold Index. Futures price forecast for gold through November 2014 is estimated at $1,728.63 per troy ounce based on the CME Group Futures.

For this study, M3 has used a Base Case gold price in all production years of $1,250 per troy ounce. This conservative estimate is 29% below the November 2012 spot gold price.

19.7 SMELTERS AND REFINING TERMS

Based on a 2012 study commissioned by SAGC and performed by Simon Hunt Strategic Services (SHSS), the smelters which are most-likely to accept the King-king concentrate are located in Japan, South Korea and India. With flexibility in gold payments and recovery rates, Chinese smelters may also be buyers of King-king concentrate.

SHSS provided the following general comments and smelter and refining terms applicable to the King-king project. These were utilized in developing the project economics:

- Smelter Payable Copper: 96.5% subject to minimum deduction of 1 unit
- Smelter Charge: Annual benchmark, Treatment Charge at the time of the study was US $63.50/ton
- Gold Deduction: 1 gm per metric ton of concentrate

- Payable Gold Recovery: From 90% - 97.50% with rare cases to 98.5%, such as for Japanese smelters, but other terms may then apply. For the purposes of this report 97.5% is assumed.
- Gold Refining Costs: Annual benchmark, but cost at the time of the study was approximately US$5.00/oz.
- Silver Deduction: 30 gm per metric ton of concentrate
- Payable Silver Recovery: 92%
- Silver Refining Cost: Annual benchmark, but cost at the time of the study was approximately US$ 0.40/oz.
- Copper Refining to Cathode: Annual benchmark, now 6.35 USC/lb

For the purposes of this report, copper concentrate grade is assumed to be at or above 22% and additional deductions are not applied.

19.8 CONCENTRATE SHIPPING COSTS

The shipping cost from the King-king mine port to Asian smelters is calculated on a wet weight basis with moisture specified at six to ten percent upon arrival. Current indicative shipping costs ex Philippine port in parcels of 11kt were provided by SHSS. Pricing per wet metric ton (WMT) falls within the range $23 - $44/ WMT, depending on destination. Examples from the SHSS report are shown below:

Table 19-1: Concentrate Shipping Costs

Destination	Shipping Cost (USD / WMT)
Tuticorin (East Coast, India)	$35 - $38
Dahej (West Coast, India)	$40 - $44
Qingdao (China)	$23 - $28
Japan	$23 - $28

Based on the indicative shipping cost data, the King-king study uses a shipping cost of $30 per WMT to move concentrate by vessel from the company port to a copper smelter located at a port in Japan or China.

19.9 SULFURIC ACID MARKET ANALYSIS

Sulfuric acid is used as a reagent in both the milling processing and heap leach operations. The Fertecon Research Center (Fertecon) was commissioned by SAGC to report on Sulfur and Sulfuric Acid Markets. In their report (March 2012), Fertecon cited competitive sources of sulfuric acid supply in the Philippine market. The two major exporters of smelter acid are Japan and South Korea, with some smaller quantities of smelter acid available from the PASAR smelter within the Philippines. There is also some supply of acid exported from Indian smelters. It is expected that in the next three years there will be development of smelter acid capacity in China which should likely result in acid exports. There is also the possibility of some investment in sulfur-burning acid production capacity in the Philippines. These should further increase the acid inventory to the local market.

Fertecon concluded that the Philippines region is currently in a state of surplus for sulfuric acid availability, and is expected to remain so. Fertecon estimates the 2013 sulfuric acid price for product from Japan at approximately $38 per ton.

19.9.1 Purchase of Sulfuric Acid

According to Fertecon, trade aggregators operating in Southeast Asia account for most sulfuric acid exports in the region. Cargoes are sold individually through sales tenders or certain tonnages are allocated to individual traders over a period. New entrants into the market would likely be supplied by one or more trade aggregators until consistency of offtake can be demonstrated to encourage a supplier to offer a long-term contract.

19.9.2 Sulfuric Acid Pricing

The Fertecon study concluded that a sulfuric acid consumer in the Philippines should be able to purchase imported acid at close to the cost of producing the acid from sulfur. The table below captures recent historical acid prices and estimates for the near future for acid originating from South Korea and Japan.

Table 19-2: Sulfuric Acid Price Data

Korean and Japanese Sulfuric Acid Prices, $/ton						
	2009	**2010**	**2011**	**2012**	**2013**	**Average**
Korean	$ 15.00	$ 36.00	$ 66.00	$ 47.00	$ 39.00	**$ 40.60**
Japanese	$ 16.00	$ 36.00	$ 62.00	$ 43.00	$ 38.00	**$ 39.00**
Average	**$ 15.50**	**$ 36.00**	**$ 64.00**	**$ 45.00**	**$ 38.50**	**$ 39.80**

This study assumes a cost of $40/ton for sulfuric acid received by vessel at the port based on historical averages.

19.9.3 Sulfuric Acid Transport

Freight costs for acid are relatively high due to its corrosive characteristics requiring special shipping vessels. Costs are currently estimated by Fertecon at approximately $30/ton. This study assumes a shipping cost of $30/ton for sulfuric acid.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section summarizes the current understanding of the Project's environment and the corporate, regulatory, and international framework within which the Project is being developed. SAGC has prepared or is preparing environmental reports and programs to meet municipal, provincial and national regulatory requirements as well as international standards.

The description of the Project's current status on permitting, environmental and social considerations is based on:

- the Project Environmental Impact Statement (SAGC, 2012) submitted to DNR for comments in February 2012;
- the DMPF submitted to MGB in May 2012;
- field data collected in 2010, 2011 and 2012 and baseline environmental and social studies reports prepared for the EIS
- the September 2010 King-king Copper-Gold Project Technical Report (IMC)
- the December 2011 Project Description Report (AMEC Environment & Infrastructure Inc., 2011); and
- ongoing environmental and social data collection.

20.1 PHYSICAL ENVIRONMENT

The topography in the immediate Project area is steep and rugged with elevations ranging from 260 to 950 meters above mean sea level (amsl) (SAGC, 2012). Moving west from the proposed mine site, the terrain gradually transitions from the mountains through rolling hills to coastal plains ending at the Davao Gulf. The climate is tropical with no pronounced wet and dry seasons and daytime temperatures range from 18 to 35ºC. The annual precipitation ranges from 2,000 to 3,200 mm per year in the mountains and 1,800 to 2,000 mm per year along the coastal plain (SAGC, 2012).

The main structural feature of the Philippines is the Philippine Fault, which extends over the entire length of the archipelago. This fault controls the structural fabric resulting in the emplacement of intrusive rocks and the associated mineralization in the King-king deposit (Kilborn International Inc. [Kilborn], 1997). Copper and gold mineralization occurs at or near the apex of the composite diorite intrusive complex within the intrusive rocks and extending well into the surrounding wall rocks (IMC, 2010).

Several soil types have been identified in the region, including Banhigan, Camansa, Umingan, San Manuel, and Catanauan, each of which is a mix of silt, clay, and loam. Topsoil thickness, averaged at 50 cm, varies from very thin (<10 cm) in the mountainous areas with steep slopes to very thick (> 100 cm) in the coastal plain area (SAGC, 2012).

The dominant drainage system in the area is dendritic. The Project area itself is located largely within the Kingking watershed. The Kingking watershed is nearly 20 km long with an average slope of 0.058 meters per meter (SAGC, 2012). The small-scale mining activity within the Project area has significantly changed the erosion and sedimentation rates of the lower Kingking watershed. Project activities are also planned to occur within the Lahi and Matiao watersheds. The Lahi and Matiao Rivers also originate in the mountains and flow southwest into the Davao Gulf.

Based upon project related monitoring efforts, the flow rate of the Kingking River has ranged from 1.20 to 5.79 cubic meters per second (m^3/s). The highest flow rates were measured in the Matiao River, with a maximum rate of 10.49 m^3/s. The flow rate of the Lahi River has been measured between 0.50 and 2.71 m^3/s. Much higher flows than those measured can be expected based upon local observations following major rainfall events.

There are two groundwater regimes within the Project area, an alluvial aquifer along the coastal plain and a bedrock, fracture-controlled aquifer in the mountains. The alluvial aquifer is a major drinking water source for residents. Residents also use groundwater sources (springs and artesian boreholes) for their water supply.

Two Class A meteorological stations were installed; one in the lowland and one in the highland regions of the Project Area in March, 2011. Two stations were necessary due to the different site specific conditions between the highland and lowland regions and thus improved the ability to fully characterize the climatic differences within the entire Project Area. The stations were installed to collect baseline meteorology data in support of the Social and Environmental Impact Assessment (SEIA) effort.

Data are continuously collected with hourly and daily averages being measured, calculated and stored in the on-site data loggers. The parameters being measured at the stations include: (1) air temperature, (2) relative humidity, (3) wind speed, (4) wind direction, (5) evaporation (only at Lowland station), (6) solar radiation and (7) precipitation. Some values such as 2m and 10m temperature differential and dew point are calculated. Data loggers are programmed to be automatically down loaded for data and instrument QA/QC via a remote satellite transmission.

Noise level surveys were conducted throughout the project area in 17 strategic locations, and showed that non-residential areas were below National Pollution Control Commission (NPCC) limits; but they were over the NPCC limits at the community centers surveyed (SAGC, 2012). Eight sample locations all exceeded Philippines DENR noise standards. One additional site exceeded DENR standards during the day, two exceeded standards during the evening and one sample location exceeded DENR standards during the daytime and at night. Only one population center did not exceed DENR noise standards. The proposed locations for the crusher and drystack tailing facility exceeded all noise standards while the construction camp site exceeded standards in the evening. All other noise sampling locations were within DENR noise standards.

20.2 CHEMICAL ENVIRONMENT

The chemical environment was assessed by examining the chemical properties and quality of the air, soil, sediment, surface water, and groundwater. The information below is based on the samples that have been analyzed to date and is subject to change when more laboratory results are received.

Air quality baseline studies show that none of the parameters measured exceeded Philippine standards. The measured parameters include particulate matter, total suspended particles, nitrogen dioxide (NO_2), sulfur dioxide (SO_2), carbon monoxide (CO), and metals.

The soil sample results received to date represent several different soil types within the area. The soils are slightly acidic, with pH results between 5.4 and 6.1 standard units. The soils analyzed contain high aluminum and iron; however, no parameters measured exceeded Philippine environmental standards.

Among the surface water bodies in the proposed Project area, samples collected from the Kingking River showed the highest concentrations of copper and mercury, as well as total suspended solids (SAGC, 2012). At all but three sample sites along the Kingking River, dissolved copper exceeded Philippines water quality standards for class “C” water bodies. United States Environmental Protection Agency (EPA) aquatic life standards for dissolved mercury were exceeded at one site and exceeded both EPA and Philippines DENR standards at three sampling sites. Total mercury and cyanide exceeded both EPA and DENR standards at all sampling sites. High total coliform concentrations were found in all water bodies sampled and exceeded DENR standards at all sampling sites.

Current sediment chemistry data indicate high concentrations of aluminum, chromium, copper, iron, and manganese. Mercury has been visually observed in the sediments at the mouth of the Kingking River due to its use by small-scale mining operators.

Groundwater was sampled through existing boreholes, domestic wells, and seeps and springs. Groundwater quality is generally good within the mountains; while unsanitary sewage disposal has directly impacted portions of the alluvial aquifer in the lowland areas (SAGC, 2012).

20.3 BIOLOGICAL ENVIRONMENT

The biological environment was surveyed by examining the terrestrial vegetation and wildlife in the proposed Project area. Marine and aquatic environments were also examined for life ranging from phytoplankton to whales.

The natural vegetation has been altered or removed by past logging operations, as well as small-scale mining and agricultural activities in the proposed Project region. Six general types of vegetation were recognized by the Project environmental team: open-canopy mid-mountain forest, brushland, wooded grassland, agricultural plantations (coconut and banana), riparian-riverine vegetation, and coastal vegetation. A total of 301 plant species were recorded in the survey, with over half of the species being trees. Exotic or introduced species account for 48 of the total 301 species. Twelve species are considered vulnerable or critically endangered, as defined by the International Union for Conservation of Nature (IUCN) Red List and the Philippine National Red List.

Wildlife surveys were conducted in 2011 to identify rare, endangered, and threatened species in conjunction with baseline studies associated with the Project. A total of 74 bird species were observed in the Project area, most of which can be found throughout Southeast Asia with only 8 percent of the species being endemic to Mindanao. A total of 17 mammal species and 10 reptilian species were identified in the region. Several of the species found in the region are listed as near-threatened or vulnerable by the IUCN, while others are protected by Convention on International Trade in Endangered Species (CITES), including 11 bird species, two (2) mammal species and five (5) reptile and amphibian species (SAGC, 2012).

Marine studies showed that several species of sea turtles, dolphins, whales, and seabirds live in the area. Sea cows and whale sharks also live in the region. The sea cow species and all species of sea turtle found in the region are listed as endangered. Phyto-, nano-, zoo-, and ichthyoplankton, as well as coral and benthic species were found in abundance during oceanographic surveys which included diving surveys. The sea grass density ranged from 772.0 to 3,174.2 shoots per square meter.

A significant fishery exists in the Davao Gulf; however, the quality of fish caught has decreased over the years and, as a result, so have the number of fishermen.

Because various sensitive species with special conservation status have been identified in the proposed Project area during the baseline studies, it will likely be necessary to implement ongoing monitoring for these species and modify Project activities accordingly to avoid habitat disturbance. A comprehensive Biodiversity Action Plan, including a well-designed biodiversity offset program, will be developed and implemented with full consideration of all threatened, endangered, and vulnerable species.

20.4 SOCIAL ENVIRONMENT

Compostela Valley Province, once part of Davao del Norte Province, was created in 1998. Pantukan is one of the eleven municipalities of the Province. Based on the 2000 Census, Pantukan has a population of 61,801 people in 13,311 households (69,656 people in 2007).

The majority of inhabitants are migrants from Cebu, Samar, Bohol, and other Visayan provinces. The minorities in the Province include the Mansaka, Mandaya, Dibabawons, Mangguangans, and Aeta groups, such as the Talaingod, Langilan, and Matigsalug.

The main source of livelihood in Compostela Valley is the production of agricultural products, such as rice, coconut, cacao, coffee, papaya, mango, pineapple, durian, and banana. Some residents have fishponds and culture their own fish, including tilapia, milkfish, and possibly other species. Mining, mostly small-scale, is also a major source of livelihood. The unemployment rate (12.6%) of Pantukan is relatively high compared to the provincial, regional and national average of 6.3% (IMC, 2010).

Pantukan is divided into 13 barangays. Barangays Bongbong, Kingking, Magnaga, Napnapan, and Tagdangua may be directly impacted by the proposed Project. An estimated 4,000 families (17,000 people) live on-site within the tenement (IMC, 2010; Personal Communication, 2012).

According to the National Statistics Office of the Philippines, the 2007 populations of the five barangays directly impacted by the Project were:

- Barangay Bongbong: 2,812 people
- Barangay Kingking: 21,444 people;
- Barangay Magnaga: 7,743 people;
- Barangay Napnapan: 9,983 people; and
- Barangay Tagdangua: 3,928 people.

Of the barangays surveyed, about three-quarters of the population are of Visayan origin. Indigenous people account for seven to thirty-two percent of each barangay's population. In general, the indigenous people of the area belong to the Mansaka, Mandaya, Manobo and Bagobo Tribes. The indigenous people largely practice agriculture. Nearly all people in the region speak Cebuano.

In most barangays, Catholicism is the religion of more than three-quarters of the population. Other denominations of Christianity are commonly practiced as well. About one to two percent of the survey participants were Muslim. Barangay Bongbong has a more significant Islamic population of 45% of survey participants, and a Catholic population of 52%.

College graduates account for one to six percent of the population, while high school graduates account for ten to 12% of the population.

The surveys conducted in 2011 suggest that employment is specific to each barangay. Mining is the main source of income for 33% of the surveyed participants in Kingking, 6% of Magnaga survey participants, and 12% of Tagdangua. Housekeeping, day laboring, and farming account for a larger percentage of employment in Magnaga and Tagdangua. Housekeeper, daily wage laborer, and self-employed are the main employment types held in Napnapan and Bongbong. About 50 to 90 percent of each barangay's population earns less than 5,000 Philippine pesos (PhP) per month. A few individuals in Magnaga and Tagdangua reported to earn more than PhP 50,000 per month; three percent of individuals in King-king reported wages above PhP 50,000.

Electric lighting is used by more than two-thirds of the households. The remainder of the households generally uses gas lamps for lighting. Wood and charcoal are used as cooking fuel by more than three-quarters of the households. The majority of households use streams, springs, or wells for their water supply. Some households use personal or communal faucets for their water supply. There are reportedly brownouts that occur in the region with some regularity.

Markets are supplied with local produce, rice, and other crops, as well as other domestic, agricultural, and industrial supplies from Davao City, and other localities in Mindanao. The nearest major airport is in Davao City. A hard surface highway connects Pantukan with Davao City and other localities in Southern Mindanao.

20.5 PERMITTING: REGULATORY APPROVAL PROCESS

Development of the Project will require compliance with environmental laws and local requirements. Philippine environmental laws regulate emissions and discharges to the environment, and also specify the manner in which mines are operated. Environmental laws are promulgated and administered at the national level. Environmental regulation and enforcement of the mining industry is mainly performed by bureaus within the DENR. Within DENR, MGB and the EMB possess the most authority in regulating the mining industry. The Project submitted the draft EIS to EMB in February 2012 and the DMPF application to MGB in May 2012.

At the local level, the key governmental units that will oversee the Project include the Compostela Valley Provincial Government, the Pantukan Mayor's Office, the Pantukan Municipal Planning and Development Department, and barangay captains. An important, but unquantifiable, aspect of the Project permitting will be the social acceptability required by the DMPF process and will also be addressed in the Environmental Impact Assessment (EIA) process, which provides for the development of the Environmental Impact Statement (EIS) of the Project. The endorsements required by the DMPF have been obtained from the five Barangays affected by the project, the Municipality of Pantukan and the Compostela Valley Province.

All large-scale mine developments in the Philippines are required to secure an Environmental Compliance Certificate (ECC). The ECC is required before numerous other authorizations are granted. The ECC is issued after completion of the EIA process. The EIA process consists of six stages: 1) project screening; 2) EIA study scoping; 3) conduct of EIA study and report preparation; 4) review and evaluation; 5) decision-making; and 6) environmental impact and monitoring. To complete the EIA process, and before ECC issuance, local government and non-governmental units must endorse the project as being in the best interest of the community while balancing environmental impact. Written project endorsements required by MGB have been obtained from these groups for the DMPF application process.

When the ECC has been issued, the project may continue planning and permitting with other government agencies and local government units, after which the project may commence construction, development, and operation. Table 20-1 lists the major pre-construction permits required for the project, submission dates, and the anticipated dates of receipt.

Table 20-1: Major Permits Needed, Submission Dates, and Expected Dates of Receipt

Name of the Permit	Submission Date	Expected Date of Receipt
Environmental Compliance Certificate (ECC)	N/A*	3rd Quarter 2013
Declaration of Mine Project Feasibility (DMPF)	May 2012	3rd Quarter 2013
Foreshore Lease	4th Quarter 2013	2nd Quarter 2014
Road Right of Way Permit (RROW)	4th Quarter 2013	2nd Quarter 2014
Tree Cutting Permit	4th Quarter 2013	2nd Quarter 2014
Sanitary Landfill Permit	4th Quarter 2013	2nd Quarter 2014
Water Rights Permit	4th Quarter 2013	2nd Quarter 2014
Quarry Permit	4th Quarter 2013	2nd Quarter 2014
Building Permits	4th Quarter 2013	2nd Quarter 2014

*EIS Submitted in February 2012 for comments from the DENR is the application for the ECC

20.6 INTERNATIONAL STANDARDS

There are a number of international standards and guidelines that will also be employed by SAGC in the design, construction, operation, and closure of the Project. These international guidelines and standards include the following: the Performance Standards of the International Finance Corporation, World Bank Group (IFC PS) on Environmental and Social Sustainability (January 2012), including IFC PS Guidance Notes (2007a); IFC's General Environmental, Health, and Safety Guidelines (2007b); IFC's Environmental, Health, and Safety Guidelines for Mining (2007c); IFC's Policy on Disclosure of Information (2006); the World Bank's Anti-Corruption Strategy (2011); the Voluntary Principles on Security and Human Rights (2000); and the Equator Principles (2006).

In summary, these international guidelines and standards provide a project owner with: guidelines for conducting an I-SEIA; a set of specific environmental quality standards, including both "end of pipe" discharge limits and acceptable ambient levels for various parameters; extensive operating management practices (known as "good international industry practices" or GIIP); standards of performance for the design, construction, operation, and closure of a mine project standards; and a system for formal documentation of social and environmental studies, programs and practices. SAGC is committed to voluntary conformance with these international guidelines and standards for the Project.

20.7 ENVIRONMENTAL IMPACT ASSESSMENT

SAGC is engaged in the EIA process for the Project with the Draft EIS report submitted to DENR in February 2012.

20.8 ENVIRONMENTAL AND SOCIAL MANAGEMENT

SAGC is currently developing environmental and social management plans and programs for the Project, which includes the Environmental Management Plan, Environmental Monitoring Plan, Environmental Health Impact Assessment, Environmental Risk Assessment, Resettlement Action Plan, Biodiversity Action Plan, Social Development and Management Program, Environmental Protection and Enhancement Program, Safety and Health Program, Final Mine Rehabilitation and/or Decommissioning Plan, and Environmental Resolution Assessment.

21 CAPITAL AND OPERATING COSTS

M3 Engineering & Technology compiled cost data into a master capital cost estimate and master operating cost estimate. The following organizations provided data for the estimates in their respective areas of expertise:

- *M3 Engineering & Technology (M3)* – Tucson, AZ – Process plant overall project site layout infrastructure.
- *Independent Mining Consultants (IMC)* – Tucson, AZ – Mine and contract mining.
- *AMEC Earth & Environmental, Inc. (AMEC)* – Denver, CO – Drystack tailing facility, VRMA and pit dewatering.
- *A.V. Garcia* – Quezon City, Philippines –Power.
- *Halcrow Group (Halcrow)* – Manila, Philippines –Port facilities.
- *Landgon & Seah Philippines Inc.* – Manila, Philippines – Provided the Asian material unit rates.
- *The Mines Group* – Reno, NV – Leach pad design and cost estimate
- *St Augustine Gold and Copper (SAGC)* – Spokane, WA – owner’s costs, consumables etc.

All rates and costs are stated in US dollars (USD) unless noted otherwise.

21.1 CAPITAL COST SUMMARY

The detailed initial capital cost estimate is summarized by area in Table 21-3 below. Designs were completed to a preliminary feasibility level. Process flow diagrams were developed, equipment was identified, and construction costs were developed based on general arrangement drawings. Major equipment costs were obtained from equipment vendors and material unit rates were obtained from local and international quotations.

The estimate was prepared in Q3 2012 dollars. No escalation has been assumed.

Table 21-1 Summary of Capital Costs

Area	Description	($ Millions)
Process Plant and General Infrastructure	General Site, Mine support infrastructure, waste disposal, primary crushing, aerial conveyors, heap leach, grinding, flotation, SX-EW, Agitated leach, tailing dewatering, drystack tailing, water systems, water treatment, on site power distribution, ancillary facilities, EPCM, freight, import duties	$1,082.30
Mine	Contract mining operating costs before the start of production.	$114.90
Power Plant	Power plant and support facilities. Two 80 MW coal fired power generators and 29 MW of HFO, and power line to main project substation.	$320.00
Port Facility	Dock Facility, Coal unloading, Concentrate loading, Coastal Complex	$108.80
Owners Costs	Land Acquisition, Construction/ Operating Camps, Environmental Permits, Initial Fills, Owner's Project Management, Security, Early Staffing, Community relations	$175.80
Contingency	Contingency on all parts of the project	$240.10
Escalation	Not included in this estimate	$0
Total Before VAT		$2,041.90
Value Added Tax (VAT)		$167.20

21.1.1 Mine Capital Basis

This Preliminary Feasibility Study was based on an assumption that the mining will be performed by a contractor throughout the life of the mine (contract mining). For modeling purposes, the "cost plus" methodology was applied to estimate the total mining cost. The contractor's cost was estimated by IMC based on the bottom-up approach with considerations of direct mining costs, contractor overhead and profit, and estimated equipment depreciation costs incurred by the contractor. The estimate is not based on contractor mining quotes.

Table 21-1 shows summary of Mine Capital for the life of mine for contract mining and includes the following:

- Mobilization – Initial mining equipment, both major and support equipment contractor mobilization was estimated at 3% of owner's capital in year -2 and -1, or US$ 3.8 million. This assumes most of the equipment will be purchased new and delivery to site will be in the purchase price. The equipment depreciation charge is consistent with this assumption. The 3% mobilization charge is to cover logistics, hiring of personnel, procuring supplies, additional small equipment, etc.
- Initial Access Road and Mine Development Contingency – Estimated at US$ 16.0 million during Years -2 and -1, the same as the owner operation case.
- Mine Development – US$ 109.9 million is the estimated operating cost to mine 46.2 million tons of material during the preproduction period by the contractor.

- Owner Equipment - An allowance for owner equipment is estimated at 1% of the value of mine major equipment purchases for the owner operation case. This includes pickup trucks for mine technical services staff, computer equipment, surveying equipment, etc.

Table 21-1: Summary of Mine Capital – Contract Mining

MINE CAPITAL COSTS:	Units	Capital by Period Yr -2	Capital by Period Yr -1	Initial Capital	Sustaining Capital	Total Capital
Contractor Mobilization	($x1000)	1,955	1,867	3,823		3,823
Initial Access and Development Contingency	($x1000)	10,500	5,500	16,000		16,000
Mine Development	($x1000)	37,278	72,638	109,915		109,915
Owner Equipment	($x1000)	593	566	1,158	2,231	3,389
Total	($x1000)	50,326	80,571	130,897	2,231	133,127

21.1.1.1 Mine Capital- Owner Operation

The owner operation case was used to develop the contract mining estimates presented in the previous section. Table 21-2 summarizes the mine capital cost by category for initial and sustaining capital.

The estimated mine capital cost, for the owner operation case, was developed by IMC includes the following items:

- Mine major equipment
- Mine support equipment and initial spare parts
- Mine preproduction development expense

The estimated cost of the following facilities was developed by others and is included in the infrastructure capital budget:

- The mine shop and warehouse
- Fuel and lubricant storage facilities
- Explosive storage facilities
- Electrification of the pit
- Office facilities

Table 21-2: Mining Capital – Mine Equipment and Mine Development (US $x1000) – Owner Operation

Category	Initial Capital by Time Period Yr -2	Yr -1	Year 1	Initial Capital	Sustaining Capital	Total Capital
Major Equipment	59,254	56,590	101,622	217,466	121,445	338,911
Support Equipment	5,925	5,659	10,162	21,747	12,145	33,891
Equipment Subtotal	65,179	62,249	111,784	239,212	133,590	372,802
Equipment Contingency	6,518	6,225	11,178	23,921	0	23,921
Mine Development	37,057	58,990	0	96,047	0	96,047
TOTAL MINE CAPITAL	108,754	127,463	122,963	359,181	133,590	492,770
Exclusions: Mine shop and warehouse, fuel and lubricant storage, explosives storage, and offices.						

An allowance for support equipment is based on 10% of the major equipment purchases for each year. Support equipment includes items such as fuel and lube trucks, tire handlers, mechanics trucks, welding trucks, cranes, shop forklifts, pickup trucks, etc. This also includes mine engineering and safety equipment such as a GPS system, surveying equipment, computers, etc. This allowance is also assumed to cover initial spare parts inventory.

A contingency of 10% is added to the equipment cost estimate during the initial capital period.

21.1.2 Power Plant

The power plant capital cost includes the following: fuel and other input media handling and storage system; boiler island system; boiler feed water circulation and make up system; air handling system; ash handling system; exhaust gas system; steam turbine-generator island system; fresh water supply, storage and treatment system; sea water cooling system; fire alarm and protection system; plant controls, instrumentation, metering and protection and data management system; electrical switchgears, control panels and distribution board; power, control and instrumentation and data cabling system; waste treatment systems; all associated site development and formation, civil works, foundation, buildings, roads and underground utilities; liquid fuel handling and storage system, grid interface switchyard and transmission line to the mill and port complex. It also includes a heat exchanger system to provide hot water to the mill process.

The power plant capital cost estimate were based on a combination of vendors' preliminary equipment proposals (Sumitomo Heavy Industries of Japan for Boiler and associated plant and equipment; GE-Triveni from India/Italy for the Steam Turbine-Generator Island) and historical data from AVGPSC's most recent similar project studies for construction and related works.

Other power plant equipment (heat exchangers and condensers) costs are based on GEA historical price data, for equipment made in India and China. All electrical equipment (high and low switchgears, power transformers and control center) cost estimates are based on proposals from the local Philippines offices of Siemens, ABB and Schneider.

21.1.3 Port Facility

The capital basis for the port facilities involves consolidation of unit construction costs from recently completed port projects and past cost data from similar undertakings into a parametric form to simplify derivation of cost. These were then adjusted to reflect current trends and actual project economic conditions.

In absence of detailed design drawings, parametric costing has been used to derive the unit cost of a particular works item such as the cost of a bridge overpass structure across the National Highway serving as an access from the mine site to the port area. The cost for this item was treated as on a square meter or linear meter basis. Parametric costs for pavement structures have been analyzed and established based on the recommended typical roadway sections for an average carriageway at a per kilometer cost.

21.1.4 Tailing and VRMA Facilities

The cost estimate includes site preparation, underdrains, decants, diversion channels and ponds including liners. In addition, mobile equipment required to place tailing material was developed.

21.1.5 Infrastructure and Process Plant

The larger number of general arrangement drawings than typical for a PFS allowed area costs for concrete, structural steel, architectural and equipment to be estimated to a feasibility level. These more accurate estimates allowed an overall lower contingency and tighter level of accuracy than typical for a preliminary feasibility study to be applied to the process plant capital costs. Estimates were based on these general arrangement civil and architectural drawings, a detailed equipment list, process design criteria and flowsheets.

21.1.6 Pit Diversions

The proposed system consists primarily of run-on channels used to divert non-contact storm water around mine facilities. In addition to these channels, a major diversion will be required to reroute the Kingking River through the pit to allow uninterrupted mining in the pit. Costs include the run-on diversion channels as well as the Kingking River diversion.

21.1.7 Owner's Costs

SAGC developed the owner's costs for the project. These costs include land acquisition, permitting, first fills, early staffing, legal costs, construction and operations camp, owner's team, site security and other miscellaneous cost.

21.2 OPERATING COST SUMMARY

Life of mine (23 years) consolidated net cash costs (net of by-product credits) per pound of payable copper are US $0.40. Life of mine payable copper is 3,079.5 million pounds. The consolidated net cash costs for the first 10 years of full production are US $0.19 per pound of payable copper. Payable copper in the first 10 years of full production is 2,046.3 million pounds.

Life of mine consolidated net cash costs per ton of ore processed (heap leach and mill ores combined) are $2.00. Life of mine processed ore is 617.7 million tons. Consolidated net cash costs per ton of ore processed in the first 10 years of full production are $1.23. Ore processed in the first 10 years of full production is 314.3 million tons.

Life of mine and first 10 years of full production cash costs per ton of ore processed without by-product credits are $12.80 and $13.27, respectively.

Table 21-3: Summary of Operating Cost by Year ($000)

Year	Mining (incl. pit dewatering)	Concentrator	Gravity Gold Circuit	Agitated Tailing Leach	SX-EW - Tailing Leach	Tailing Disposal	Heap Leach & SX-EW	G&A	Laboratory	Port	Total
-2	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -
-1	$ -	$ -	$ -	$ -	$ -	$ -	$85,248	$25,048	$469	$1,876	$112,641
1	$123,067	$66,395	$403	$56,561	$15,066	$17,716	$43,747	$32,835	$760	$1,909	$358,459
2	$117,796	$78,935	$415	$70,861	$11,163	$22,090	$44,903	$32,458	$760	$1,904	$381,283
3	$133,465	$78,333	$413	$72,802	$8,184	$22,647	$42,924	$26,638	$760	$1,921	$388,088
4	$128,238	$82,114	$423	$73,283	$6,106	$22,858	$23,421	$26,639	$760	$1,937	$365,780
5	$126,257	$85,771	$432	$73,610	$4,565	$23,016	$21,842	$25,747	$760	$1,907	$363,907
6	$130,250	$81,337	$415	$72,622	$6,138	$22,608	$17,048	$25,744	$760	$1,921	$358,844
7	$109,537	$80,406	$419	$66,313	$5,461	$20,785	$19,124	$25,746	$760	$1,913	$330,464
8	$110,438	$77,927	$414	$63,916	$5,469	$20,040	$16,727	$25,745	$760	$1,935	$323,371
9	$118,238	$83,454	$426	$ -	$ -	$20,631	$12,731	$25,747	$760	$1,953	$263,941
10	$122,164	$86,812	$437	$ -	$ -	$20,957	$7,406	$25,675	$760	$1,918	$266,128
11	$128,196	$80,941	$418	$ -	$ -	$20,790	$72	$25,671	$760	$1,933	$258,780
12	$127,277	$83,420	$426	$ -	$ -	$21,044	$7,567	$25,673	$760	$1,924	$268,090
13	$126,498	$82,308	$421	$ -	$ -	$21,197	$ -	$25,672	$677	$1,941	$258,714
14	$126,938	$85,443	$431	$ -	$ -	$21,509	$ -	$25,674	$677	$1,959	$262,631
15	$126,876	$88,826	$441	$ -	$ -	$21,698	$ -	$25,116	$677	$1,922	$265,557
16	$119,187	$81,460	$420	$ -	$ -	$20,827	$ -	$25,113	$677	$1,939	$249,622
17	$123,810	$84,873	$429	$ -	$ -	$21,062	$ -	$25,114	$677	$1,927	$257,893
18	$101,207	$82,592	$423	$ -	$ -	$20,867	$ -	$25,113	$677	$1,944	$232,822
19	$102,390	$84,554	$432	$ -	$ -	$20,900	$ -	$25,115	$677	$1,962	$236,030
20	$65,972	$87,659	$441	$ -	$ -	$20,652	$ -	$24,548	$677	$1,925	$201,873
21	$32,445	$80,510	$420	$ -	$ -	$20,209	$ -	$24,545	$677	$1,942	$160,747
22	$32,747	$83,609	$430	$ -	$ -	$20,416	$ -	$24,546	$677	$1,765	$164,191
23	$21,110	$23,287	$395	$ -	$ -	$4,351	$ -	$24,543	$280	$1,711	$75,677
Total	$2,454,104	$1,830,965	$9,722	$549,967	$62,153	$468,871	$342,759	$624,466	$16,640	$45,887	$6,405,535

21.2.1 Mine Operating Costs

IMC developed the contract mine operating cost using owner cost as the base with additional markup estimate on contractor overhead, profit, and depreciation cost. Table 21-4 summarizes the contractor mine operating costs. Total cost and the cost per total ton are shown by various time periods.

Table 21-4: Summary of Total and Unit Mining Costs

	Mine Operating Cost ($000)	Total Material (Ktonnes)	Unit Cost $/ton
Year 1 - 5	627,915	345,578	1.82
Year 6-10	588,830	325,000	1.81
Year 11-15	632,938	325,000	1.95
Year 16-20	509,116	259,840	1.96
Year 20-23	83,618	50,372	1.66
Year 1-23	2,442,418	1,305,790	1.87

The markup estimate on contractor overhead, profit and depreciation cost in contractor mine operation cost include the following:

- Contractor Equipment Depreciation Charge – The contract mining cost will include significant charges for equipment depreciation. It is not certain how a specific contractor will calculate this cost. IMC has developed annual depreciation charges to be applied to each equipment type and total annual charges.
- Contractor Overhead/Profit – 15% of Direct Operating Cost.
- Mobilization/Demobilization – After year -1 this cost is part of mine operating cost. Demobilization is about 2% of initial equipment requirements, divided between Years 21 and 23

The overall mining operating cash cost for contract mining is 36% higher than owner operation cost.

21.2.2 Process Plant Operating Costs

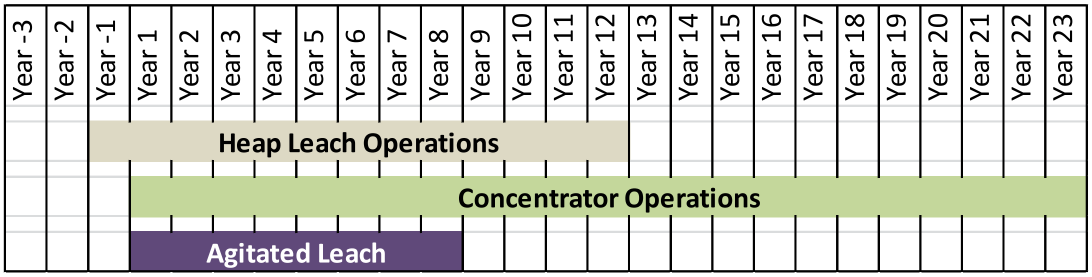
The process plant has three processes they are as follows:

- Heap Leach
- Concentrator
- Agitated Leach

Each process was determined using cost elements which include labor, reagents, electrical power, grinding media and liners, maintenance parts and services, supplies and tools.

These processes are operated for different lengths of time during the mine life; the chart below shows the time periods when these processes are operating for the base case.

Table 21-5: Periods of Operations



21.2.2.1 Heap Leach Operations

The heap leach is showing an operating cost of $3.61 per ton of heap leach ore (95 million tons) during its 13-year operation. The table below shows the operating cost by area.

Table 21-6: Heap Leach Life of Mine Operating Cost

	Total Cost	$/ton processed
Heap Leach & Crushing & Conveying	$273,535,270	$2.88
Solvent Extraction	$14,646,588	$0.15
Tank Farm	$3,032,438	$0.03
Electrowinning	$47,195,538	$0.50
Ancillary Services	$4,348,933	$0.05
Total Heap Leach & SX-EW Plant	$342,758,767	$3.61

21.2.2.2 Concentrator Operations

The concentrator is showing an operating cost of $4.42 per ton of mill ore (523 million tons) during its 23 year operation. The table below displays the operating cost by area.

Table 21-7: Concentrator Life of Mine Operating Cost

Concentrator Operations		
Crushing & Conveying	$119,633,978	$0.23
Grinding & Classification	$1,240,844,839	$2.37
Flotation & Regrind	$272,294,606	$0.52
Concentrate Thickening, Filtration & Dewatering	$85,647,112	$0.16
Tailing Disposal	$468,871,014	$0.90
Gravity Gold Circuit	$4,634,955	$0.01
Gold Refinery	$5,087,115	$0.01
Ancillary Services	$112,544,662	$0.22
Total Concentrator Operations	$2,309,558,281	$4.42

21.2.2.3 Agitated Leach Operations

The agitated leach process is showing an operating cost of $3.15 per ton of mill ore (194 million tons) during its 8-year operation. The table below shows the operating cost by area.

Table 21-8: Total Operating Costs by Area

	Total Cost	$/ton mill ore leached
Agitated Tailing Leach	$549,966,526	$2.83
SX-EW Tailing Leach	$62,153,048	$0.32
Total Agitated Leach Operations	$612,119,574	$3.15

21.2.3 Power Plant Operating Costs

Operating costs for the power plant were based on a blend between coal and heavy fuel power production. The coal power plant is expected to run 8,000 hours per year, with the HFO plant utilized in conjunction with the operating coal plant during outages. The average life of mine power costs are summarized below:

	Annual Cost ($ Millions)	Cost per kWh
Coal Fired Power Plant	$ 37.46	$ 0.0500
Heavy Fuel Oil Plant	$ 8.28	$ 0.0111
Total	$ 45.74	$ 0.0611

21.2.4 Tailing Drystack Placement Costs

The cost for placing the drystack tailing material at a rate of 60,000 tons/day is included in the concentrator costs within the tailing disposal costs of $0.90/ton ore.

21.2.5 General Administration and Laboratory

The operating cost for the General Administration and laboratory were estimated by cost element. The cost elements include labor, supplies, support infrastructure, services, insurances, real property taxes, on-going land acquisition, and other expenses. The departments included are as follows:

- Administration
- Controllers
- Human Resources
- Purchasing
- Safety & Security
- Environmental

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

The economic analysis in this study included PFS compliant modeling of the annual cash flows based on projected production volume, sales revenue, initial capital, operating cost, and sustaining capital with resulting evaluation of the key economic indicators such as Internal Rate of Return (IRR), the Net Present Value (NPV), and payback period (time in years to recapture the initial capital investment) for the Project. The sales revenue was based on the production of copper concentrate containing gold, gold doré bullions and copper cathode. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in earlier sections of this report.

22.2 MINE PRODUCTION STATISTICS

Mine production is reported as ore and valueless rock 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

	Tons (kt)	Copper (%)	Gold (g/t)
Heap Leach Ore	95,164	0.311%	0.143
Mill Ore	465,773	0.311%	0.469
Mill Low Grade Sulfide Ore	54,945	0.193%	0.206
Mill Low Grade Oxide Ore	2,034	0.094%	0.391
Waste	656,556		
Total Material Mined	1,274,472		

22.3 PROCESS PLANT PRODUCTION STATISTICS

The mill ore will be processed through a concentrator, gold gravity circuit and an agitated leach process for the tailing. This will result in three products: a copper concentrate containing gold, gold bullion, and copper cathodes. The heap leach ore will be processed using a SX-EW Process. The metal recoveries over LOM are projected as follows:

Table 22-2: Metal Recoveries

Recovery	Copper Concentrate	Copper Cathode	Gold Bullion
Mill Ore Copper	62.6%	15.7%	
Mill Ore Gold	65.9%		7.3%
Heap Leach Ore Copper		73.8%	

Table 22-3: Metal Production

Copper Concentrate (kt)	3,758
Copper (klbs)	2,143,846
Gold (kozs)	4,889
Silver (kozs)	11,650
Gold Dore' (kozs)	543
Mill Ore Copper Cathode (klbs)	538,163
Heap Leach Ore Copper Cathode (klbs)	481, 751

22.4 SMELTER AND REFINERY RETURN FACTORS

The copper concentrates will be shipped from the site to a smelting company. Smelter treatment charges and refining charges will be negotiated at the time of the finalization of the sales agreements.

Asian market smelter charges, as calculated in the financial evaluation research by Simon Hunt Strategic Services, are presented in the table below.

Table 22-4: Smelter Treatment Factors

Copper Concentrate Terms	
Payable copper (%)	96.5%
Cu Minimum Deduction (%)	1.0%
Payable gold (%)	97.5%
Au Minimum Deduction (g)	1.0
Payable Silver	92.0%
Ag Minimum Deduction (g)	30.0
Treatment charge ($/dmt)	$63.50
Refining charge – Cu ($/lb.)	$0.06
Refining charge – Au ($/payable oz.)	$5.00
Refining charge – Ag ($/payable oz)	$0.40
Gold Insurance (% of gross revenue)	0.4%
Copper Concentrate Transportation ($/wmt)	$30.00
Moisture	10%
Cathode Terms	
Payable copper (%)	100.0%
Transportation ($/lb.)	$0.01
Gold Bullion Terms	
Payable Gold (%)	99.9%
Refining Charge ($.oz.)	$2.00
Gold transportation/insurance (% of gross revenues)	1.0%

22.5 CAPITAL EXPENDITURE

22.5.1 Initial Capital

The base case financial indicators have been determined with 100% equity financing of the initial capital. The total initial capital carried in the financial model for new construction and pre-production mine development is $2,041.9 million expended over a 5 year period. The initial capital includes all required cost categories including Owner's costs and contingency. The initial capital cash flow is estimated to be expended in the years before production with a percentage carried over into the first production year. Presented below is the initial capital summary.

Table 22-5: Initial Capital

	$ in millions
Mining*	130.9
Process Plant	1,244.6
Power Plant	350.4
Port	119.0
Owner's Cost	197.0
Total	2,041.9

*The mining capital cost reflects contract mining

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 $248.6 million. This capital will be expended during a 21 year period.

Table 22-6: Sustaining Capital ($ million)

1000	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Mine Equipment	1.0	-	0.2	-	-	-	0.0	-	0.1	0.0	0.0
Pit Diversions	0.8	1.1	2.8	0.6	2.3	1.2	-	1.3	0.6	5.0	0.6
Dry Stack Tailings	19.6	-	-	55.0	-	-	-	-	18.8	-	-
Southwest VRMA	-	6.4	0.4	14.1	0.0	-	-	-	-	5.9	-
Tailings Stacking Conveyo	-	-	2.1	-	-	2.1	-	-	2.1	-	-
Site General	-	-	-	-	5.7	0.1	6.3	4.6	1.2	6.1	0.1
Heap Leach	7.7	-	0.1	0.8	-	-	-	-	-	-	-
Total	29.0	7.5	5.5	70.5	8.1	3.3	6.3	5.8	22.7	17.0	0.7

	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Total
Mine Equipment	0.1	-	0.3	0.3	0.2	-	-	-	-	-	2.2
Pit Diversions	-	0.4	0.3	4.4	0.2	0.5	0.5	1.0	0.9	-	24.2
Dry Stack Tailings	-	-	17.4	-	-	-	-	18.5	-	-	129.2
Southwest VRMA	-	-	-	-	-	-	-	-	-	-	26.8
Tailings Stacking Conveyo	-	-	-	-	-	-	-	-	-	-	6.2
Site General	0.1	0.1	5.9	10.3	1.2	0.4	0.1	0.1	5.9	3.3	51.4
Heap Leach	-	-	-	-	-	-	-	-	-	-	8.6
Total	0.2	0.5	23.8	15.0	1.6	0.9	0.6	19.6	6.7	3.3	248.6

22.5.3 Working Capital

A 60 day delay of receipt of revenue from sales is used for accounts receivables. A delay of payment for accounts payable of 30 days is also incorporated into the financial model. In addition, working capital allowance of $35.0 million for plant consumable inventory is estimated in year -1 and year 1. 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. The copper concentrate revenues are based on the value of the payable metals sold less transportation and smelter treatment and refining charges. While the copper cathode and gold bullion revenue is based on the gross value of the payable metals sold before refining and transportation charges.

Copper	$3.00 per pound
Gold	$1,250.00 per troy ounce
Silver	$25.00 per troy ounce

The smelting, refining and shipping charges for copper concentrate were deducted from gross revenue to calculate the net smelter returns (NSR) which are shown in revenue section of the economic model in Table 22-10.

22.5.5 Total Production Cost

Total Production Cost includes mine operations, process plant operations, general administrative cost, reclamation and closure cost, government fees, smelting, refining charges and shipping charges, and a by-product credit for gold and silver. The table below shows the estimated operating cost by area based on payable copper pounds for three time periods (5 year, 10 years, and LOM averages).

Table 22-7: Production Cost

Cost Component	Units	Time Period		
		Years 1-5	Years 1-10	LOM[1]
Payable Pounds of Copper	000'000	1,330	2,046	3,079
Mining	$/lb Cu	$0.47	$0.60	$0.80
Processing	$/lb Cu	$0.81	$0.92	$1.06
Operating Costs	**$/lb Cu**	**$1.28**	**$1.52**	**$1.86**
G&A	$/lb Cu	$0.13	$0.16	$0.25
Reclamation & Closure	$/lb Cu	$0.00	$0.00	$0.02
Cash Costs at Mine	**$/lb Cu**	**$1.41**	**$1.68**	**$2.13**
Government Fees	$/lb Cu	$0.17	$0.22	$0.26
Total Cash Costs at Mine	**$/lb Cu**	**$1.58**	**$1.89**	**$2.38**
Shipping, Smelting & Refining	$/lb Cu	$0.11	$0.15	$0.18
Total Costs	**$/lb Cu**	**$1.69**	**$2.04**	**$2.57**
By-Product Credits	$/lb Cu	-$1.66	-$1.85	-$2.17
Consolidated Net Cash Costs	**$/lb Cu**	**$0.03**	**$0.19**	**$0.40**

[1] Includes year -1 heap leach production

22.6 DEPRECIATION

The depreciation was calculated using 15-year straight line method following assumptions for both initial and sustaining capital. Last year of production is the catch up year if assets are not fully depreciated.

Depreciation will be further refined during project feasibility level study.

22.7 GOVERNMENT FEES

The following government payments were estimated in the cash flow analysis:

- Excise Tax – 2% of gross revenue, starting from production start; estimated life of mine cost is $307.3 million.
- Local Business Tax – 2% of gross revenue, starting after Income Tax Holiday; estimated life of mine cost is $181.2 million.
- Royalty – ICC - 1% of gross revenue (w/ credit for Community Development); estimated life of mine cost is $153.6 million.
- Development Mining Technology (ComDev) - Min of 1.5% of total operating cost (incl. depreciation, excise tax); estimated life of mine cost is $142.2 million.
- Monitoring Trust Fund (MTF) - PhP150,000/QTR as determined by MRF Committee; estimated life of mine cost is $0.3 million.

- Mine Waste and Tailing (MWT) Fees – Waste - PhP0.05 per MT of waste; estimated life of mine cost is $0.7 million.
- Mine Waste and Tailing (MWT) Fees – Tailing - PhP0.10 per MT of waste; estimated life of mine cost is $1.4 million.
- Occupational Fees - PhP75/he/yr post MPSA renewal; PhP50/he/yr prior to MPSA renewal; estimated life of mine cost is $0.07 million.

22.7.1 Income Tax

Income taxes will be paid at a rate of 30% based on operating profits. No taxes were applied for the first six years of operation, starting from initial heap leach production, taking advantage of the Philippine government's income tax holiday (ITH) incentive for investments in key industries. Total income taxes paid during the life of the mine is estimated to be $745.4 million.

Due to the recent move by the Philippine Board of Investments (BOI), it could be more challenging to obtain the tax holiday for companies under the MPSA. SAGC and Nadecor are currently working with appropriate Philippine agencies to address this incentive. The economics of the project would substantially change without the tax holiday.

22.8 Net Income After Tax

Net Income after Tax amounts to $4,967.7 million.

22.9 Project Financing

It is assumed the project will be all equity financed.

22.10 Net Present Value, Internal Rate of Return, Payback

The economic analyses for the project are summarized below in Table 22-8.

Table 22-8: Key Economic Results

	After Tax
NPV @ 8% (billions)	$1.76
IRR	24.0%
Payback (Years)	2.4

22.11 Sensitivity Analysis

Figure 1-6 and Table 22-9 below shows the sensitivity analysis of the key economic indicators (NPV, IRR, and Payback) from changes in key input variables by +/-10% and +/-20% (Metal Prices, Initial Capital, Operating Cost).

The sensitivity analysis illustrates NPV sensitivity to metals prices, initial capital, and operating cost. This graph indicates that NPV is most sensitive to the metal prices and much less sensitive to initial capital and operating cost. As stated above, the base case of the project was estimated at conservative metal prices.

Table 22-9: After Tax Sensitivity Analysis

	NPV 8% ($000)	IRR	Payback
Base Case Metal Prices	$1,757,074	24.0%	2.4
+20%	$2,954,347	33.0%	1.8
+10%	$2,356,012	28.6%	2.0
-10%	$1,157,336	19.0%	2.9
-20%	$551,133	13.6%	3.8
Initial Capital			
+20%	$1,438,076	19.2%	2.9
+10%	$1,597,575	21.4%	2.6
-10%	$1,916,574	27.0%	2.1
-20%	$2,076,073	30.5%	1.9
Operating Cost			
+20%	$1,305,153	20.6%	2.7
+10%	$1,531,174	22.3%	2.5
-10%	$1,982,975	25.6%	2.3
-20%	$2,208,876	27.1%	2.2

22.12 DETAILED FINANCIAL MODEL

The detailed bottom-up financial model, shown in Table 22-10 below, was developed in compliance with the preliminary feasibility study requirement. This model has captured all the parameters of the mine production volume, annual sales revenue, and all the associated costs. This model was also used to calculate the economics of the project as well as for sensitivity analysis.

Table 22-10: Detailed Financial Model

Flotation, Tailings and Heap Leach		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035	2036	2036	2037	2038	2039	2040	2041	2042
	Total	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27
Mining Operations																																
Heap Leach Ore																																
Beginning Inventory (kt)	95,164	95,164	95,164	95,164	88,742	66,793	45,611	36,773	33,089	26,574	21,006	16,352	11,072	6,398	3,557	2,613	2,589	216	34	24	12	12	-	-	-	-	-	-	-	-	-	-
Mined (kt)	95,164	-	-	6,422	21,949	21,182	8,838	3,684	6,515	5,568	4,654	5,280	4,674	2,841	944	24	2,373	182	10	12	-	12	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	95,164	95,164	88,742	66,793	45,611	36,773	33,089	26,574	21,006	16,352	11,072	6,398	3,557	2,613	2,589	216	34	24	12	12	-	-	-	-	-	-	-	-	-	-	-
Gold Grade (g/t)	0.143	-	-	0.085	0.109	0.139	0.165	0.172	0.120	0.130	0.171	0.204	0.243	0.183	0.222	0.341	0.139	0.157	0.333	0.403	-	0.119	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.311%	0.000%	0.000%	0.242%	0.298%	0.384%	0.387%	0.329%	0.316%	0.343%	0.238%	0.240%	0.186%	0.220%	0.228%	0.201%	0.274%	0.214%	0.440%	0.279%	0.000%	0.150%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Contained Gold (kozs)	437	-	-	18	77	95	47	20	25	23	26	35	37	17	7	0	11	1	0	0	-	0	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	652,958	-	-	34,263	144,200	179,321	75,405	26,721	45,387	42,104	24,420	27,937	19,166	13,779	4,745	106	14,334	859	97	74	-	40	-	-	-	-	-	-	-	-	-	-
Mill Oxide Ore - Low Grade Stockpile																																
Beginning Inventory (kt)	2,034	2,034	2,034	2,034	2,034	2,034	2,034	2,017	1,945	1,754	1,704	1,378	968	63	8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mined (kt)	2,034	-	-	-	-	-	17	72	191	50	326	410	905	55	8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	2,034	2,034	2,034	2,034	2,034	2,017	1,945	1,754	1,704	1,378	968	63	8	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Grade (g/t)	0.391	-	-	-	-	-	0.408	0.415	0.421	0.433	0.380	0.355	0.400	0.400	0.402	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.094%	0.000%	0.000%	0.000%	0.000%	0.000%	0.136%	0.120%	0.094%	0.075%	0.097%	0.090%	0.094%	0.087%	0.094%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Contained Gold (kozs)	26	-	-	-	-	-	0	1	3	1	4	5	12	1	0	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	4,228	-	-	-	-	-	51	190	396	83	697	814	1,875	105	17	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mill Ore																																
Beginning Inventory (kt)	465,773	465,773	465,773	465,773	465,741	461,948	446,430	421,158	394,965	368,772	342,626	316,556	293,438	271,278	248,626	225,814	202,633	179,333	155,653	131,810	108,009	84,838	61,621	38,484	15,593	-	-	-	-	-	-	-
Mined (kt)	465,773	-	-	32	3,793	15,518	25,272	26,193	26,193	26,146	26,070	23,118	22,160	22,652	22,812	23,181	23,300	23,680	23,843	23,801	23,171	23,217	23,137	22,891	15,593	-	-	-	-	-	-	-
Ending Inventory (kt)	-	465,773	465,773	465,741	461,948	446,430	421,158	394,965	368,772	342,626	316,556	293,438	271,278	248,626	225,814	202,633	179,333	155,653	131,810	108,009	84,838	61,621	38,484	15,593	-	-	-	-	-	-	-	-
Gold Grade (g/t)	0.469	-	-	0.392	0.603	0.528	0.562	0.667	0.659	0.540	0.493	0.336	0.501	0.466	0.512	0.456	0.477	0.302	0.343	0.368	0.415	0.425	0.403	0.374	0.512	-	-	-	-	-	-	-
Copper Grade (%)	0.311%	0.000%	0.000%	0.298%	1.000%	0.712%	0.442%	0.343%	0.309%	0.323%	0.324%	0.341%	0.282%	0.339%	0.298%	0.283%	0.256%	0.296%	0.292%	0.270%	0.235%	0.219%	0.220%	0.206%	0.178%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Contained Gold (kozs)	7,028	-	-	0	74	263	457	562	555	454	413	250	357	340	375	340	357	230	263	282	309	317	300	275	257	-	-	-	-	-	-	-
Contained Copper (klbs)	3,188,747	-	-	210	83,585	243,702	246,233	198,020	178,247	186,243	186,314	173,744	137,556	169,094	149,995	144,684	131,704	154,556	153,445	141,907	120,046	112,094	112,218	103,960	61,190	-	-	-	-	-	-	-
Mill Sulfide Ore - Low Grade Stockpile																																
Beginning Inventory (kt)	54,945	54,945	54,945	54,945	54,945	54,945	54,945	50,865	48,660	46,086	41,531	36,106	29,483	24,584	17,684	12,083	8,493	6,083	3,602	3,084	2,293	1,256	-	-	-	-	-	-	-	-	-	-
Mined (kt)	54,945	-	-	-	-	-	4,080	2,205	2,574	4,555	5,425	6,623	4,899	6,900	5,601	3,590	2,410	2,481	518	791	1,037	1,256	-	-	-	-	-	-	-	-	-	-
Ending Inventory (kt)	-	54,945	54,945	54,945	54,945	54,945	50,865	48,660	46,086	41,531	36,106	29,483	24,584	17,684	12,083	8,493	6,083	3,602	3,084	2,293	1,256	-	-	-	-	-	-	-	-	-	-	-
Gold Grade (g/t)	0.206	-	-	-	-	-	0.153	0.183	0.212	0.221	0.215	0.150	0.209	0.231	0.249	0.241	0.188	0.169	0.204	0.183	0.212	0.272	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.193%	0.000%	0.000%	0.000%	0.000%	0.000%	0.236%	0.230%	0.209%	0.202%	0.189%	0.222%	0.193%	0.183%	0.167%	0.163%	0.181%	0.191%	0.166%	0.177%	0.163%	0.133%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Contained Gold (kozs)	363	-	-	-	-	-	20	13	18	32	37	32	33	51	45	28	15	13	3	5	7	11	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	234,233	-	-	-	-	-	21,228	11,181	11,860	20,285	22,605	32,415	20,845	27,838	20,621	12,901	9,617	10,447	1,896	3,087	3,726	3,683	-	-	-	-	-	-	-	-	-	-
All Ore																																
Beginning Inventory (kt)	617,916	617,916	617,916	617,916	611,462	585,720	549,020	510,813	478,659	443,186	406,867	370,392	334,961	302,323	269,875	240,510	213,715	185,632	159,289	134,918	110,314	86,106	61,621	38,484	15,593	-	-	-	-	-	-	-
Mined (kt)	617,916	-	-	6,454	25,742	36,700	38,207	32,154	35,473	36,319	36,475	35,431	32,638	32,448	29,365	26,795	28,083	26,343	24,371	24,604	24,208	24,485	23,137	22,891	15,593	-	-	-	-	-	-	-
Ending Inventory (kt)	-	617,916	617,916	611,462	585,720	549,020	510,813	478,659	443,186	406,867	370,392	334,961	302,323	269,875	240,510	213,715	185,632	159,289	134,918	110,314	86,106	61,621	38,484	15,593	-	-	-	-	-	-	-	-
Gold Grade (g/t)	0.395	-	-	0.087	0.182	0.303	0.426	0.576	0.526	0.437	0.410	0.282	0.417	0.391	0.452	0.427	0.423	0.289	0.340	0.362	0.406	0.417	0.403	0.374	0.512	-	-	-	-	-	-	-
Copper Grade (%)	0.300%	0.000%	0.000%	0.242%	0.401%	0.523%	0.407%	0.333%	0.302%	0.311%	0.291%	0.301%	0.249%	0.295%	0.271%	0.267%	0.251%	0.286%	0.289%	0.267%	0.232%	0.215%	0.220%	0.206%	0.178%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Contained Gold (kozs)	7,855	-	-	18	150	358	524	596	600	510	480	321	438	408	427	368	382	244	266	287	316	328	300	275	257	-	-	-	-	-	-	-
Contained Copper (klbs)	4,080,166	-	-	34,473	227,785	423,023	342,916	236,112	235,890	248,715	234,036	234,909	179,442	210,816	175,378	157,691	155,655	165,862	155,437	145,067	123,772	115,817	112,218	103,960	61,190	-	-	-	-	-	-	-
Waste																																
Beginning Inventory(kt)	656,556	656,556	656,556	656,556	651,774	642,516	614,216	587,423	554,577	525,050	496,369	467,844	438,275	405,913	373,361	337,726	299,521	262,604	223,947	183,318	142,922	102,130	61,615	35,553	9,163	-	-	-	-	-	-	-
Mined (kt)	656,556	-	-	4,782	9,258	28,300	26,793	32,846	29,527	28,681	28,525	29,569	32,362	32,552	35,635	38,205	36,917	38,657	40,629	40,396	40,792	40,515	26,062	26,390	9,163	-	-	-	-	-	-	-
Ending Inventory (kt)	-	656,556	656,556	651,774	642,516	614,216	587,423	554,577	525,050	496,369	467,844	438,275	405,913	373,361	337,726	299,521	262,604	223,947	183,318	142,922	102,130	61,615	35,553	9,163	-	-	-	-	-	-	-	-
Total Material Mined (kt)	1,274,472	-	-	11,236	35,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	65,000	49,199	49,281	24,756	-	-	-	-	-	-	-
Process Plant Operations																																
Concentrator																																
Milled Ore - Processed (kt)	522,752	-	-	-	-	19,343	25,272	26,193	26,193	26,146	26,070	23,118	22,160	22,652	22,812	23,181	23,300	23,680	23,843	23,801	23,171	23,217	23,137	22,891	22,200	22,200	22,200	5,972	-	-	-	-
Gold Grade (g/t)	0.441	-	-	-	-	0.542	0.562	0.667	0.659	0.540	0.493	0.336	0.501	0.466	0.512	0.456	0.477	0.302	0.343	0.368	0.415	0.425	0.403	0.374	0.420	0.238	0.203	0.163	-	-	-	-
Copper Grade (%)	0.297%	0.000%	0.000%	0.000%	0.000%	0.768%	0.442%	0.343%	0.309%	0.323%	0.324%	0.341%	0.282%	0.339%	0.298%	0.283%	0.256%	0.296%	0.292%	0.270%	0.235%	0.219%	0.220%	0.206%	0.176%	0.173%	0.200%	0.234%	0.000%	0.000%	0.000%	0.000%
Contained Gold (kozs)	7,417	-	-	-	-	337	457	562	555	454	413	250	357	340	375	340	357	230	263	282	309	317	300	275	299	170	145	31	-	-	-	-
Contained Copper (klbs)	3,427,208	-	-	-	-	327,497	246,233	198,020	178,247	186,243	186,314	173,744	137,556	169,094	149,995	144,684	131,704	154,556	153,445	141,907	120,046	112,094	112,218	103,960	86,120	84,733	98,012	30,786	-	-	-	-
Recovery Gold to Conc. (%)	65.91%	0.00%	0.00%	0.00%	0.00%	67.48%	67.83%	69.16%	69.48%	68.20%	67.54%	62.64%	67.20%	66.22%	67.22%	66.17%	67.12%	61.10%	62.92%	63.46%	65.49%	65.86%	64.99%	64.01%	65.62%	55.83%	53.69%	46.60%	0.00%	0.00%	0.00%	0.00%
Recovery Gold to Bullion (%)	7.32%	0.00%	0.00%	0.00%	0.00%	7.50%	7.54%	7.68%	7.72%	7.58%	7.50%	6.96%	7.47%	7.36%	7.47%	7.35%	7.46%	6.79%	6.99%	7.05%	7.28%	7.32%	7.22%	7.11%	7.29%	6.20%	5.97%	5.18%	0.00%	0.00%	0.00%	0.00%
Recovery Copper to Conc. (%)	62.55%	0.00%	0.00%	0.00%	0.00%	39.71%	41.16%	49.12%	61.91%	74.07%	68.44%	72.16%	66.61%	77.05%	73.70%	72.35%	67.16%	79.12%	76.12%	72.91%	68.26%	62.88%	61.23%	55.10%	42.51%	52.91%	64.21%	74.84%	0.00%	0.00%	0.00%	0.00%
Recovery Copper to Cathode (%)	15.70%	0.00%	0.00%	0.00%	0.00%	53.00%	48.10%	39.57%	24.81%	15.25%	20.03%	18.02%	19.31%	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%
Copper Concentrate (kt)	3,758	-	-	-	-	191	157	163	181	225	209	199	157	213	192	184	165	210	203	188	160	141	134	120	88	104	129	44	-	-	-	-
Copper Concentrate Grade (%)	25.88%	0.00%	0.00%	0.00%	0.00%	30.84%	29.31%	27.00%	27.59%	27.86%	27.65%	28.59%	26.55%	27.77%	26.16%	25.78%	24.28%	26.41%	26.05%	24.96%	23.18%	22.72%	23.30%	21.62%	18.80%	19.61%	22.15%	23.49%	0.00%	0.00%	0.00%	0.00%
Recovered Gold (kozs)	4,889	-	-	-	-	228	310	388	385	310	279	157	240	225	252	225	240	141	165	179	202	209	195	176	197	95	78	15	-	-	-	-
Recovered Copper (klbs)	2,143,846	-	-	-	-	130,051	101,356	97,265	110,358	137,958	127,522	125,372	91,630	130,290	110,541	104,676	88,446	122,289	116,809	103,462	81,938	70,481	68,708	57,279	36,606	44,832	62,938	23,039	-	-	-	-
Recovered Silver (kozs)	11,650	-	-	-	-	593	486	507	563	696	649	617	485	660	594	571	512	651	630	583	497	436	415	373	274	322	399	138	-	-	-	-
Cathode Copper (klbs)	538,163	-	-	-	-	173,580	118,427	78,348	44,219	28,404	37,317	31,310	26,559	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Bullion (kozs)	543	-	-	-	-	25	34	43	43	34	31	17	27	25	28	25	27	16	18	20	22	23	22	20	22	11	9	2	-	-	-	-
Heap Leach																																
Heap Leach Ore - Processed (kt)	94,948	-	-	-	18,200	14,600	14,600	14,600	6,590	5,568	4,654	5,280	4,674	2,841	944	24	2,373	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Grade (g/t)	0.143	-	-	-	0.104	0.139	0.155	0.122	0.120	0.130	0.171	0.204	0.243	0.183	0.222	0.341	0.139	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Copper Grade (%)	0.311%	0.000%	0.000%	0.000%	0.287%	0.384%	0.386%	0.300%	0.315%	0.343%	0.238%	0.240%	0.186%	0.220%	0.228%	0.201%	0.274%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%	0.000%
Contained Gold (kozs)	436	-	-	-	61	65	73	57	25	23	26	35	37	17	7	0	11	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Contained Copper (klbs)	651,889	-	-	-	115,125	123,600	124,184	96,600	45,788	42,104	24,420	27,937	19,166	13,779	4,745	106	14,334	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Recovery Copper to Cathode (%)	73.75%	0.00%	0.00%	0.00%	71.84%	77.86%	78.44%	73.00%	74.30%	76.09%	66.39%	66.67%	56.99%	63.64%	64.91%	60.20%	70.80%	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%
Cathode Copper (klbs)	480,751	-	-		82,704	96,240	97,410	70,516	34,021	32,039	16,211	18,625	10,923	8,769	3,080	64	10,149	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Payable Metals																																
Copper Concentrate																																
Payable Gold (kozs)	4,649	-	-	-	-	216	297	374	370	295	266	146	229	213	240	214	229	130	155	168	192	199	186	168	189	89	72	13	-	-	-	-
Payable Copper (klbs)	2,060,571	-	-	-	-	125,499	97,809	93,662	106,358	133,006	122,910	120,984	88,179	125,598	106,315	100,615	84,803	117,659	112,326	99,317	78,403	67,379	65,760	54,629	34,659	42,545	60,097	22,058	-	-	-	-
Payable Silver (kozs)	7,383	-	-	-	-	376	308	321	357	441	411	391	308	418	377	362	325	413	400	369	315	276	263	236	173	204	253	87	-	-	-	-
Cathode Copper																																
Heap Leach - Payable Copper (klbs)	480,751	-	-	-	82,704	96,240	97,410	70,516	34,021	32,039	16,211	18,625	10,923	8,769	3,080	64	10,149	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Leach - Payable Copper (klbs)	538,163	-	-	-	-	173,580	118,427	78,348	44,219	28,404	37,317	31,310	26,559	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Bullion																																
Payable Gold (kozs)	543	-	-	-	-	25	34	43	43	34	31	17	27	25	28	25	27	16	18	20	22	23	22	20	22	11	9	2	-	-	-	-
Income Statement ($000)																																
Metal Prices																																
Gold ($/oz)	$ 1,250.00			$ -	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ 1,250.00	$ -	$ -
Silver ($/oz)	$ 25.00			$ -	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ 25.00	$ -	$ -
Copper ($/lb)	$ 3.00			$ -	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ 3.00	$ -	$ -

Table 22-10: Detailed Financial Model (Continued)

Revenues																																
Copper Concentrate																																
Gold Revenue ($ 000)	5,810,844	-	-	-	-	269,818	371,317	466,945	462,509	368,614	331,899	183,036	286,166	265,753	299,882	266,996	285,690	163,048	193,419	210,504	240,491	249,120	232,197	210,022	236,025	111,352	89,992	16,049	-	-	-	-
Silver Revenue ($ 000)	184,575	-	-	-	-	9,396	7,704	8,027	8,913	11,033	10,276	9,770	7,688	10,454	9,414	9,046	8,115	10,315	9,989	9,234	7,875	6,910	6,568	5,903	4,337	5,094	6,329	2,185	-	-	-	-
Copper Revenue ($ 000)	6,181,714	-	-	-	-	376,497	293,427	280,985	319,073	399,019	368,729	362,951	264,537	376,794	318,945	301,846	254,410	352,978	336,978	297,952	235,208	202,138	197,280	163,888	103,978	127,636	180,291	66,175	-	-	-	-
Less: Treatment & Refining Charges																																
Copper Concentrate																																
Treatment Charges	(238,629)			-	-	(12,148)	(9,960)	(10,378)	(11,523)	(14,264)	(13,285)	(12,632)	(9,939)	(13,516)	(12,171)	(11,695)	(10,492)	(13,335)	(12,914)	(11,938)	(10,182)	(8,934)	(8,492)	(7,631)	(5,607)	(6,586)	(8,183)	(2,825)	-	-	-	-
Copper Refining Charges	(136,134)			-	-	(8,258)	(6,436)	(6,176)	(7,008)	(8,760)	(8,098)	(7,961)	(5,818)	(8,273)	(7,019)	(6,647)	(5,616)	(7,765)	(7,417)	(6,570)	(5,203)	(4,476)	(4,363)	(3,637)	(2,324)	(2,847)	(3,997)	(1,463)	-	-	-	-
Gold Refining Charges	(23,243)			-	-	(1,079)	(1,485)	(1,868)	(1,850)	(1,474)	(1,328)	(732)	(1,145)	(1,063)	(1,200)	(1,068)	(1,143)	(652)	(774)	(842)	(962)	(996)	(929)	(840)	(944)	(445)	(360)	(64)	-	-	-	-
Silver Refining Charges	(2,953)			-	-	(150)	(123)	(128)	(143)	(177)	(164)	(156)	(123)	(167)	(151)	(145)	(130)	(165)	(160)	(148)	(126)	(111)	(105)	(94)	(69)	(82)	(101)	(35)	-	-	-	-
Gold Insurance Cost	(23,243)			-	-	(1,079)	(1,485)	(1,868)	(1,850)	(1,474)	(1,328)	(732)	(1,145)	(1,063)	(1,200)	(1,068)	(1,143)	(652)	(774)	(842)	(962)	(996)	(929)	(840)	(944)	(445)	(360)	(64)	-	-	-	-
Transportation	(124,012)			-	-	(6,313)	(5,176)	(5,393)	(5,988)	(7,413)	(6,904)	(6,564)	(5,165)	(7,024)	(6,325)	(6,078)	(5,453)	(6,930)	(6,711)	(6,204)	(5,291)	(4,643)	(4,413)	(3,966)	(2,914)	(3,423)	(4,252)	(1,468)	-	-	-	-
Net Smelter Return				-	-	626,683	647,781	730,146	762,133	745,103	679,797	526,980	535,056	621,895	600,176	551,187	524,239	496,841	511,636	491,146	460,848	438,013	416,814	362,804	331,537	230,255	259,359	78,489	-	-	-	-
Copper Cathode Revenue ($ 000)	-	-																														
Heap Leach	1,442,252	-	-	-	248,112	288,721	292,229	211,548	102,063	96,116	48,634	55,874	32,768	26,306	9,240	192	30,448	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Leach	1,614,488	-	-	-	-	520,740	355,280	235,043	132,656	85,212	111,951	93,930	79,676	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Bullion	678,306	-	-	-	-	31,571	42,973	53,889	53,464	42,967	38,719	21,725	33,277	31,204	34,995	31,218	33,262	19,499	22,927	24,804	28,094	28,989	27,031	24,446	27,264	13,140	10,820	2,026	-	-	-	-
Total Revenues (NSR)	15,363,963	-	-	-	248,112	1,467,716	1,338,262	1,230,626	1,050,317	969,399	879,101	698,509	680,776	679,406	644,412	582,597	587,949	516,340	534,563	515,950	488,943	467,002	443,845	387,250	358,801	243,395	270,179	80,515	-	-	-	-
Operating Cost																																
Mining (included pit dewatering)	2,454,104			-	-	123,067	117,796	133,465	128,238	126,257	130,250	109,537	110,438	118,238	122,164	128,196	127,277	126,498	126,938	126,876	119,187	123,810	101,207	102,390	65,972	32,445	32,747	21,110	-	-	-	-
Concentrator	1,830,965			-	-	66,395	78,935	78,333	82,114	85,771	81,337	80,406	77,927	83,454	86,812	80,941	83,420	82,308	85,443	88,826	81,460	84,873	82,592	84,554	87,659	80,510	83,609	23,287	-	-	-	-
Gravity Gold Circuit	9,722			-	-	403	415	413	423	432	415	419	414	426	437	418	426	421	431	441	420	429	423	432	441	420	430	395	-	-	-	-
Agitated Tailings Leach	549,967			-	-	56,561	70,861	72,802	73,283	73,610	72,622	66,313	63,916	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
SXEW - Tailings Leach	62,153			-	-	15,066	11,163	8,184	6,106	4,565	6,138	5,461	5,469	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Disposal	468,871			-	-	17,716	22,090	22,647	22,858	23,016	22,608	20,785	20,040	20,631	20,957	20,790	21,044	21,197	21,509	21,698	20,827	21,062	20,867	20,900	20,652	20,209	20,416	4,351	-	-	-	-
Heap Leach & SXEW	342,759			-	85,248	43,747	44,903	42,924	23,421	21,842	17,048	19,124	16,727	12,731	7,406	72	7,567	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
General Administration	624,466			-	25,048	32,835	32,458	26,638	26,639	25,747	25,744	25,746	25,745	25,747	25,675	25,671	25,673	25,672	25,674	25,116	25,113	25,114	25,113	25,115	24,548	24,545	24,546	24,543	-	-	-	-
Laboratory	16,640			-	469	760	760	760	760	760	760	760	760	760	760	760	760	677	677	677	677	677	677	677	677	677	677	280	-	-	-	-
Port	45,887			-	1,876	1,909	1,904	1,921	1,937	1,907	1,921	1,913	1,935	1,953	1,918	1,933	1,924	1,941	1,959	1,922	1,939	1,927	1,944	1,962	1,925	1,942	1,765	1,711	-	-	-	-
Custom Duties	75,465				1,606	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	3,211	-	-	-	-
Copper Cathode																																
Transportation	10,189			-	827	2,698	2,158	1,489	782	604	535	499	375	88	31	1	101	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gold Bullion																																
Gold Refining Charges	1,085			-	-	51	69	86	86	69	62	35	53	50	56	50	53	31	37	40	45	46	43	39	44	21	17	3	-	-	-	-
Gold Transportation/Insurance Cost	6,783			-	-	316	430	539	535	430	387	217	333	312	350	312	333	195	229	248	281	290	270	244	273	131	108	20	-	-	-	-
Total Operating Cost	6,499,058			-	115,074	364,735	387,151	393,413	370,394	368,221	363,040	334,427	327,343	267,601	269,776	262,354	271,788	262,152	266,109	269,056	253,160	261,440	236,347	239,525	205,401	164,111	167,527	78,911	-	-	-	-
Salvage Value	-				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reclamation & Closure	74,458				-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	18,614	18,614	18,614	18,614	-	-	-
Total Production Cost	6,573,516		-	-	115,074	364,735	387,151	393,413	370,394	368,221	363,040	334,427	327,343	267,601	269,776	262,354	271,788	262,152	266,109	269,056	253,160	261,440	236,347	239,525	205,401	182,725	186,142	97,526	18,614	-	-	-
Operating Income	8,790,448		-	-	133,038	1,102,981	951,110	837,213	679,923	601,177	516,061	364,082	353,433	411,804	374,635	320,243	316,160	254,188	268,454	246,894	235,783	205,561	207,499	147,725	153,400	60,669	84,037	(17,011)	(18,614)	-	-	-
Depreciation	2,290,510				83,782	138,063	138,564	138,928	143,625	144,163	144,386	144,806	145,195	146,707	147,922	147,977	147,996	148,040	150,424	68,307	14,220	13,846	13,578	12,803	13,952	14,818	14,398	14,009	-	-	-	-
Total Depreciation	2,290,510		-	-	83,782	138,063	138,564	138,928	143,625	144,163	144,386	144,806	145,195	146,707	147,922	147,977	147,996	148,040	150,424	68,307	14,220	13,846	13,578	12,803	13,952	14,818	14,398	14,009	-	-	-	-
Net Income After Depreciation	6,499,937		-	-	49,256	964,918	812,547	698,285	536,298	457,015	371,676	219,276	208,238	265,097	226,713	172,266	168,164	106,148	118,030	178,587	221,563	191,715	193,920	134,922	139,448	45,851	69,640	(31,020)	(18,614)	-	-	-
Government Fees																																
Excise Tax	307,279		-	-	4,962	29,354	26,765	24,613	21,006	19,388	17,582	13,970	13,616	13,588	12,888	11,652	11,759	10,327	10,691	10,319	9,779	9,340	8,877	7,745	7,176	4,868	5,404	1,610	-	-	-	-
Local Business Tax	181,191		-	-	-	-	-	-	-	-	17,582	13,970	13,616	13,588	12,888	11,652	11,759	10,327	10,691	10,319	9,779	9,340	8,877	7,745	7,176	4,868	5,404	1,610	-	-	-	-
Royalty - ICC	153,640		-	-	2,481	14,677	13,383	12,306	10,503	9,694	8,791	6,985	6,808	6,794	6,444	5,826	5,879	5,163	5,346	5,159	4,889	4,670	4,438	3,873	3,588	2,434	2,702	805	-	-	-	-
Development Mining Technology (ComDev)	142,226		-	-	3,071	8,250	8,518	8,573	8,225	8,186	8,327	7,764	7,629	6,782	6,796	6,636	6,772	6,597	6,703	5,493	4,410	4,502	4,101	4,088	3,555	2,870	2,948	1,428	-	-	-	-
Annual EPEP	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Monitoring Trust Fund (MTF)	331		-	-	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	-	-	-	-
Rehabilitation Cash Fund (RCF)	-		-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
MWT - Waste	749			-	11	33	31	38	34	33	33	34	37	37	41	44	42	44	47	46	47	47	30	30	11	-	-	-	-	-	-	-
MWT - Tailings	1,420			-	42	78	92	94	75	73	71	65	62	59	55	53	59	54	55	55	53	53	53	53	51	51	51	14	-	-	-	-
Occupational Fees	67			-	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	-	-	-	-
Net Income After Government Expenses	5,713,035	-	-	-	38,673	912,510	763,741	652,646	496,437	419,624	319,274	176,471	166,455	224,232	187,584	136,386	131,877	73,619	84,481	147,178	192,589	163,746	167,528	111,372	117,875	30,743	53,115	(36,505)	(18,614)	-	-	-
Income Taxes	745,357	-	-	-	-	-	-	-	-	-	95,782	52,941	49,936	67,270	56,275	40,916	39,563	22,086	25,344	44,153	57,777	49,124	50,258	33,412	35,362	9,223	15,935	-	-	-	-	-
Net Income After Taxes	4,967,678		-	-	38,673	912,510	763,741	652,646	496,437	419,624	223,492	123,529	116,518	156,962	131,309	95,470	92,314	51,533	59,137	103,024	134,812	114,622	117,269	77,960	82,512	21,520	37,181	(36,505)	(18,614)	-	-	-
Cash Flow																																
Operating Income & Government Expenses	8,003,546	-	-	-	122,455	1,050,573	902,305	791,573	640,062	563,787	463,659	321,277	311,650	370,939	335,506	284,364	279,872	221,658	234,905	215,485	206,809	177,592	181,106	124,175	131,826	45,562	67,513	(22,496)	(18,614)	-	-	-
Working Capital																																
Account Recievable (60 days)	-	-	-	-	(82,704)	(158,564)	21,280	17,694	29,640	13,302	14,843	29,686	2,915	225	5,752	10,161	(880)	11,771	(2,996)	3,060	4,440	3,607	3,807	9,303	4,677	18,971	(4,403)	31,178	13,235	-	-	-
Accounts Payable (30 days)	-	-	-	-	19,179	10,799	1,842	515	(1,892)	(179)	(426)	(2,352)	(582)	(4,910)	179	(610)	775	(792)	325	242	(1,307)	681	(2,062)	261	(2,805)	(3,394)	281	(7,284)	(6,486)	-	-	-
Inventory - Parts, Supplies	-	-	-	-	(17,500)	(17,500)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	35,000	-	-	-	-	-
Total Working Capital	0	-	-	-	(81,025)	(165,265)	23,123	18,208	27,748	13,123	14,418	27,335	2,333	(4,685)	5,931	9,551	(104)	10,979	(2,670)	3,302	3,133	4,287	1,744	9,565	1,872	15,577	30,878	23,894	6,749	-	-	-
Contingent Liability and Reclamation Fund	(0)					15,413	13,402	11,169	9,680	7,446	5,957	4,691	3,723	1,489	745	745	-	-	-	-	-	-	-	-	-	(18,614)	(18,614)	(18,614)	(18,614)	-	-	-
Initial Capital VAT - Paid	(167,193)	-	(8,360)	(66,877)	(83,596)	(8,360)																										
Initial Capital VAT - Recovered	167,193	-	-	-	-	52,356	44,007	33,262	20,499	16,036	1,033	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	
Capital Expenditures																																
Initial Capital																																
Mine Equipment	130,897	-	-	50,326	80,571	-																										
Process Plant	1,244,648	-	124,465	560,092	497,859	62,232																										
Power Plant	350,391	-	105,117	140,156	105,117	-																										
Port	118,968	-	47,587	71,381	-	-																										
Owner's Cost	197,015	19,701	78,806	59,104	39,403	-																										
Sustaining Capital																																
Mine Equipment	2,231	-	-	-	-	1,016	-	157	-	-	-	18	-	68	42	31	107	-	303	333	155	-	-	-	-	-	-	-	-	-	-	-
Pit Diversions	24,236	-	-	-	-	762	1,116	2,758	623	2,317	1,162	-	1,264	556	4,988	595	-	392	300	4,365	198	508	458	996	878	-	-	-	-	-	-	-
Dry Stack Tailings	129,204	-	-	-	-	19,554	-	-	54,998	-	-	-	-	18,763	-	-	-	-	17,365	-	-	-	-	18,525	-	-	-	-	-	-	-	-
Southwest VRMA	26,763	-	-	-	-	-	6,390	378	14,050	43	-	-	-	-	5,902	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings Stacking Conveyor	6,201	-	-	-	-	-	-	2,067	-	-	2,067	-	-	2,067	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Site General	51,366	-	-	-	-	-	-	-	-	5,701	116	6,293	4,565	1,225	6,079	92	116	92	5,873	10,291	1,201	378	116	92	5,865	3,269	-	-	-	-	-	-
Heap Leach	8,591	-	-	-	-	7,700	-	100	791	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital Expenditures	2,290,510	19,701	355,975	881,059	722,951	91,265	7,506	5,460	70,462	8,061	3,345	6,312	5,829	22,679	17,011	719	223	484	23,841	14,989	1,554	886	575	19,613	6,743	3,269	-	-	-	-	-	-
Cash Flow before Taxes	5,713,035	(19,701)	(364,335)	(947,936)	(765,117)	822,627	948,527	826,415	608,168	577,439	469,808	337,609	304,431	342,086	323,682	292,451	279,545	232,153	208,394	203,798	208,389	180,994	182,275	114,127	126,955	76,484	117,005	20,013	6,749	-	-	-
Cummulative Cash Flow before Taxes		(19,701)	(384,036)	(1,331,973)	(2,097,089)	(1,274,462)	(325,935)	500,480	1,108,648	1,686,087	2,155,894	2,493,504	2,797,934	3,140,020	3,463,702	3,756,154	4,035,698	4,267,852	4,476,246	4,680,044	4,888,433	5,069,427	5,251,702	5,365,829	5,492,784	5,569,268	5,686,273	5,706,286	5,713,035	5,713,035	5,713,035	5,713,035
						1.0	1.0	0.4	-	-	-	-	-	-	-	-	-															
Taxes																																
Income Taxes	745,357	-	-	-	-	-	-	-	-	-	95,782	52,941	49,936	67,270	56,275	40,916	39,563	22,086	25,344	44,153	57,777	49,124	50,258	33,412	35,362	9,223	15,935	-	-	-	-	-
Cash Flow after Taxes	4,967,678	(19,701)	(364,335)	(947,936)	(765,117)	822,627	948,527	826,415	608,168	577,439	374,026	284,668	254,494	274,816	267,407	251,536	239,982	210,068	183,049	159,645	150,612	131,870	132,017	80,715	91,592	67,261	101,071	20,013	6,749	-	-	-
Cummulative Cash Flow after Taxes		(19,701)	(384,036)	(1,331,973)	(2,097,089)	(1,274,462)	(325,935)	500,480	1,108,648	1,686,087	2,060,112	2,344,780	2,599,275	2,874,091	3,141,498	3,393,033	3,633,015	3,843,083	4,026,132	4,185,777	4,336,389	4,468,260	4,600,276	4,680,992	4,772,584	4,839,845	4,940,916	4,960,929	4,967,678	4,967,678	4,967,678	4,967,678
						1.0	1.0	0.4	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

NPV @ 0%	0%	$ 4,967,678
NPV @ 5%	7%	$ 2,001,104
NPV @ 8%	8%	$ 1,757,074
NPV @ 10%	10%	$ 1,347,097
IRR		24.0%
Payback	Years	2.4

23 ADJACENT PROPERTIES

The project will be primarily located at the west-central section of the Municipality of Pantukan in the Province of Compostela Valley. There are currently no active mineral projects that are adjacent to the King-king Deposit.

There is significant artisanal mining for gold in the King-king Mineral Property Area and in adjacent mining tenements surrounding the King-king claims. These areas are north and northeast of the King-king Deposit. To the best of SAGC's knowledge no reserve estimates have been compiled for any of the adjacent properties. The properties are primarily used for small scale mining operations and do not have reserve estimates.

24 OTHER RELEVANT DATA AND INFORMATION

No other relevant data is presented in this document.

25 INTERPRETATION AND CONCLUSIONS

25.1 PROJECT ECONOMICS

This project shows robust economic returns at conservative metal prices. These favorable economics are resulted from the following characteristics:

- Optimized mine plan with the highest returns in initial 5 years
- Significant economies of scale due to large scale production
- Low labor cost and low cost of key raw materials in the region which had a positive effect on initial capital and operating cost
- Low operating cost (net of by-product) of $0.40 per pound of copper over LOM and $0.03 per pound of copper in initial 5 years
- Low cost heap leach production starting earlier than concentrator and generating early cash flow
- Six year income tax holiday

The table below outlines the base case key economic results at the following metal prices:

- Gold = $1,250 per troy ounce
- Copper = $3.00 per pound
- Silver = $25.00 per troy ounce

Table 25-1: Economic Indicators After Tax

NPV @ 0% ($000)	$4,967,678
NPV @ 5% ($000)	$2,588,925
NPV @ 8% ($000)	$1,757,074
NPV @ 10% ($000)	$1,347,097
IRR %	24.0%
Payback - years	2.4

The results further indicate that if the prices in the future drop by 20% from the base case assumptions shown above ($1,000/oz gold and $2.40/lb copper), the project would still produce positive economics ($0.6 billion NPV and 14% IRR).

As was stated above, the project’s economics are much less sensitive to the initial capital. Thus, if the capital would increase from the base case of $2.04 billion by 20% to $2.45 billion, the project would still remain robust at $1.4 billion NPV and 19.2% IRR.

25.2 EXPLORATION AND GEOLOGY

The interpretation of the exploration work performed to date is that the King-king deposit is a significant copper-gold porphyry system with the potential to become an economic project. The drilling performed through 1998 has also been used to develop an NI 43-101 compliant mineral resource for the deposit, as presented in Section 14.

Fourteen (14) drillholes were completed in 2011 and 2012 with a total depth of 5,980 meters. New information from these drillholes along with structural geology data should be included in the geology model update at the feasibility level. The inclusion of this information is expected to increase the confidence level of the resource and reserve estimate for the deposit. Initial analysis showed that the new drillhole composites have higher copper and gold assays compared to the model used in the resource as discussed in Section 10.1.1.

SAGC reviewed all exploration data in detail and determined that there are significant copper and gold values in drill intercepts in three exploration areas of the King-king Project, based on a review of data recently recovered from historic files. The most notable intercept in these data is a hole (DD-1) located approximately 4 km north of the current pit area. The results from this drillhole (a 237-meter-deep core hole which intercepted 81 continuous meters averaging 0.44% total copper and 0.34 g/t gold) confirm wide intervals of porphyry Au-Cu mineralization intersected in the historic initial drill tests of two areas located 1 km and 4 km northeast of the King-king deposit.

25.3 MINING

The results of this study indicate that the King-king Project has the potential to become an economic producer of copper and gold.

This study has developed a proven and probable mineral reserve of 617.9 million ore tons at 0.300% total copper and 0.395 g/t gold. This amounts to 4.1 billion pounds of contained copper and 7.8 million ounces of contained gold.

There is potential to add resource and reserve tonnage to the King-king deposit as there are significant quantities of inferred resource where drilling has not found the limits of the mineralization.

The mining methods proposed for King-king are conventional open pit methods for bulk mining. There are no significant technical challenges to mining at King-king.

One of the goals of this study was to define an NI 43-101 compliant mineral reserve for the project. The study has met this goal.

25.4 TAILING AND GEOTECHNICAL

25.4.1 Tailing Testwork

25.4.1.1 Geotechnical Testwork

Two samples of tailing have been prepared by metallurgical testing. One sample was characterized as an oxide and the other as a sulfide. ABA results for both these materials indicated both low acid generating potential (AGP) as well as acid neutralizing potential (ANP).

Oxide tailing have 6.3 kg $CaCO_3$/ton ANP and 1.7 kg $CaCO_3$/ton AGP. By comparison, sulfide tailing have 12 kg $CaCO_3$/ton ANP and 4.2 kg $CaCO_3$/ton AGP. Tailing are not currently anticipated to have the potential to generate acidic drainage.

The TSF will be managed to maintain water quality via diversion of upstream unimpacted surface water, and through collection, testing, and potential treatment of impacted surface water. The dewatered tailing is anticipated to be neutralized prior to placement in the TSF. Further, the dewatered tailing is anticipated to exhibit a low permeability upon placement, similar to that of a constructed liner system. As such, an engineered liner system is not considered appropriate for construction beneath the TSF.

25.4.1.2 Geotechnical Testwork

Two samples of tailing were made available for geotechnical testwork, i.e., an oxide sample and a sulfide sample. Geotechnical testing has included: (i) particle size distribution; (ii) solids specific gravity; (iii) Atterberg limits; (iv) one-dimensional settling tests under drained and undrained conditions; (v) triaxial testing (including estimation of permeability); (vi) standard Proctor compaction testing; and (vii) slurry consolidation testing. Some of the testwork was commissioned prior to the selection of the dewatered tailing option.

The tailing are classified as low plasticity silt (ML) with 70 to 74 percent by weight passing the No. 200 sieve (finer than 75 µm), with tailing solids specific gravities of 2.72 (oxide ore composite) and 2.77 (sulfide ore composite).

The effective stress parameters obtained by the triaxial testing indicate no effective cohesion and an effective angle of internal friction of 31 to 32 degrees. Proctor compaction testing is an important test for initial evaluation of behavior for a dewatered tailing material. Results of the standard Proctor compaction test indicate a maximum dry density of 1.8 t/m^3 at an optimum moisture content of 14.0%. The shape of the Proctor curve is relatively flat, showing that the tailing material tested is not highly sensitive to moisture content. Prior to placement in the TSF, the tailing shall be dewatered via filters to approximately the optimum moisture content.

25.4.2 Tailing Design Conclusions

Based on the stability analyses performed on two sections of the proposed facility (refer to Section 18.5) and the tailing testwork completed to date, the design of the Southwest Tailing Drystack facility appears to be feasible. However, because the stability analyses were largely based on assumed parameters relating to the foundation bedrock, completion of the geotechnical exploration program is essential to support the feasibility design. Also, additional tailing testwork will be performed during the feasibility-level study to provide a more comprehensive understanding of how the material will behave. The recommendations regarding the geotechnical exploration program and required tailing testwork for the next phase of design are listed in Section 26.4.

25.5 PROCESS FACILITIES

The main challenge to processing King-king ore is the presence of a significant amount of copper oxide intermixed with copper sulfide in the deposit. Moreover, some of the oxide-dominant materials have gold grades that merit routing to the flotation plant. Once the ore classification and routing scheme was developed, the resulting process plant is more complicated and larger, but the technologies required are conventional. These are:

- Sulfide flotation to produce copper concentrate containing gold.
- Heap leaching of copper oxide minerals with sulfuric acid followed by SX-EW to produce copper cathodes
- Agitated leaching of flotation tailing with sulfuric acid followed by SX-EW to produce copper cathodes
- Gravity concentration and intensive cyanidation of coarse free gold to produce gold doré bullions.

The project economics are affected by gold recovery and the price of acid. Gold recoveries will be further studied with additional locked cycle tests. Given the copper grades in the oxide ores, the price of acid impacts the extent of heap leaching that will be done and the length of time agitated leach will be conducted.

Campaigning of mill ore containing high or low levels of oxide copper minerals would be an attractive practice. At the preliminary feasibility level of study this was only considered to predict when to stop the tailing leach process on mill ore. Campaigning sulfide or oxide dominant ore through the mill via a large stockpile ahead of the primary crusher would enhance process economics due to maximizing the cost and benefit of applying acid. It may also prove practical to optimize the acid cost-benefit by bypassing and resuming the tailing leach process based on predictive controls via ore control and utilization of suitably sized ore stockpiles.

Heap leaching of copper ore in the Philippines will be performed for the first time. While measures have been designed to deal with positive water balances during heavy rainfall periods, negative impacts to operations remain. PLS dilution will reduce copper production in the SX-EW plant, and containment and treatment of excess solution volumes will be required.

25.6 POWER PLANT OPPORTUNITIES

The power plant infrastructure may present some upside opportunities for the mine operations. It assumed that heap leach operation will start from Operating Year-1, which at current power plant availability schedules, will be supported by the heavy oil fired diesel generating units.

The normal construction schedule of the 2 x 80-MW coal-fired power plant is 36 months from start to COD (commercial operation date), with the 1st unit operational by the 30th month. The normal construction schedule for the 4 x 7.5-MW HFO-fired power plant is 16 months, with the first two units (15 MW) operational by 13th month.

Current schedule and cost assumptions projects that the coal plant to be operational on Day 1 of Operating Year 1, while the HFO plant will be operational as early as middle of Operating Year 2. This means that the coal plant construction must start, at the latest, 36 months before Operating Year 1, that is Day 1 of Operating Year -3 (1Q14).

On this assumption, the 1st unit of the coal-fired power plant can be pushed to be operational by middle of Operating Year -1 to support heap leach operations, reducing electricity cost tremendously.

The HFO plant will also be able to support mine construction power as early as second quarter of Operating Year -2, reducing construction power cost from light fuel cost to heavy fuel cost basis.

The technology assumed in the power plant infrastructure is a circulating fluidized bed coal fired boiler, providing a wider range and more flexibility to accept coal from various sources, which may have different specifications.

A major factor in the power plant operating costs is fuel. Current assumption built into the model is the use of low-quality coal from Kalimantan, Indonesia. The fast growing economies of China, India and ASEAN countries will put pressure on the coal supply from Indonesia and Australia, thus coal exports from the USA is very likely to become economically feasible. The driving factor will be the effective cost of coal per unit heat value, currently assumed at US $ 3.027 per metric ton CIF Davao, with a heat value of 5,500 kcal/kg. Any supply better than this will reduce the power plant operating cost.

25.7 VRMA

25.7.1 Valueless Rock Material Testing

Acid-Base Accounting (ABA) data for valueless rock have been collected for 180 samples, comprising the four major rock types at the site. The data show a relatively wide range of Acid Generating Potential (AGP) and a limited range of Acid Neutralizing Potential (ANP). Sampling to date shows AGP ranges from a minimum of zero to a maximum of approximately 220 tons $CaCO_3$/1000 ton (t/kt) rock, with an average of approximately 23 t/kt. ANP ranges from a minimum of near zero to a maximum of 160 t/kt, with an average of about 16 t/kt. Approximately 42% of samples have a Neutralization Potential Ratio (NPR = ANP/AGP) ratio less than 1 and on that basis are anticipated to produce acidic drainage (defined as pH below rain at about 5.2). For 37% of the samples, the NPR is greater than 3 and not anticipated to produce acidic drainage. The NPR of the remaining samples, approximately 20%, is uncertain. The relative proportions of site rock types anticipated to be mined, as estimated by a block model, suggest that approximately 35% of valueless rock will be acid-producing.

Humidity cell leach testing (HCT) of valueless rock is currently in progress, and several field-deployed barrel leach tests of near-surface rock have been completed. HCT data are available for over 52 weeks of testing and field barrel leachate data were collected monthly for approximately one year. HCT results indicate that approximately 67% all rock types will likely produce leachates with mildly acidic pH (4.2 - 6) while approximately 33% will produce leachates with pH below 4. In general, lower pH is also associated with leachates having higher total dissolved solids (TDS) and metals (e.g. copper).

Seven field barrels samples were collected, each within 10 meters of the ground surface. Three of the seven samples had Net Neutralization Potential (NNP = ANP-AGP) < 0 and NPR < 1. They produced acidic leachate (pH 2.4 - 3.8) with relatively high TDS and metals. The remainder of the barrels had NNP > 0 and NPR > 1 and produced mildly acidic leachate (pH 5.6 – 6.8) with limited TDS.

25.7.2 VRMA Design Conclusions

Based on the stability analyses performed on the maximum cross-section (Section 18.6) and the valueless rock testwork that has been completed, the design of the Southwest VRMA appears to be feasible. However, because the stability analyses were based on assumed parameters relating to the foundation bedrock, completion of the geotechnical exploration program is essential to support the feasibility design. Supplemental valueless rock testwork will also need to be completed to confirm that the material meets the minimum effective stress friction angle requirement to achieve stability under OBE or MCE loading conditions assuming an overall downstream slope of 3H:1V. The recommendations regarding the geotechnical exploration program and required valueless rock testwork for the next phase of design are listed in Section 26.4.

25.8 LAND ACQUISITION AND RELOCATION EXPENSES

Land acquisition and relocation expenses could be higher than currently expected. In the next phase of the project different alternatives will be evaluated to reduce costs such as co-mingling valueless rock with tailing. This process would much reduce the land area required to handle tailing and valueless rock by placing them together rather than handling in two different places.

25.9 MINING AND PROCESSING THROUGHPUTS

Several mine schedules were developed over the course of the preliminary feasibility studies for the evaluation of the project. Some examples were:

- 40 ktpd heap leach only
- 30 ktpd mill with tailing leach
- 60 ktpd mill with tailing leach
- 60 ktpd mill with tailing leach and 40 ktpd heap leach
- 100 ktpd mill with no tailing leach
- 120 ktpd mill with no tailing leach
- 120 ktpd mill with tailing leach
- 60 ktpd mill with tailing leach and 40 ktpd heap leach for 3 years and expanded mill to 120 ktpd by year 4

Estimates of capital and operating costs were determined by the technical team at MDCA for various scenarios and mine schedules based on internal experience, discussions with consultants and on reported costs for other recently published feasibility studies and preliminary feasibility studies for similar sized mine projects as King-king. Practical mine schedules were narrowed as metallurgical and mining studies were completed. A final review of the economics of the remaining practical mine schedules and their associated operating and capital costs was performed. The finding was that the case of a 60 ktpd concentrator with tailing leach and a 40 ktpd heap leach for copper oxide ore with low grade gold content (less than 0.15 g/t Au on average) had the best economics by a significant margin compared to the other scenarios. Therefore, this design was chosen for developing the Preliminary Feasibility Study. This analysis was completed in January 2012.

This analysis showed there is potential for higher mining and processing rates if certain conditions are met:

- Project economics were to favor dropping the mine cutoff grade, such as:
  - Higher metal prices
  - Technological improvements that lower the cost of mining or processing
  - Lower delivered cost of sulfuric acid
  - Application of other or new processing methods that produce higher gold and/or copper recoveries
  - Discovery of another or other recoverable by-products from the ore that increase project revenue
- Exploration activities discover additional ore that adds to ore reserves at grades that favor expanding the operations
- Lower cost of capital due to:
  - Changes in the market for mining and processing equipment and materials
  - Utilization of heap leach crushers, conveyors, stockpile, etc. by timing the expansion to occur when this system is no longer a significant cash flow contributor

The current mineral reserve has six pit phase developments. Another phase (Phase 7) was identified for potential expansion. Analysis was performed on the potential of an expansion utilizing the economic and recovery parameters used to produce the PFS mine schedule, which is also the mine reserve (617.9 million tons of ore). A pit expansion would require additional reserve tonnage and this would come from mining Phase 7. The maximum additional ore from Phase 7, under the PFS parameters, would be 83 million tons and require considerably higher stripping. These conditions would not support a pit expansion at this time.

26 RECOMMENDATIONS

26.1 ECONOMICS

The positive result of the economic analysis warrants advancing this project into the next phase of development and construction, subject to completion of the feasibility study. The availability and sources of strategic material and equipment will need to be evaluated further. Additional trade-off studies will be conducted in the next phase in collaboration with the industry experts in the US, Europe, and Asia.

The engineering cost of completion of the feasibility study is estimated at US $12.4 million.

26.2 EXPLORATION AND GEOLOGY

The geologic interpretation of the King-king deposit would be significantly enhanced with the completion of a 3-D structural model. Some work has been performed to date by Fisher & Strickler Rock Engineering, LLC (FSRE). REI recommends that the suggestions presented in FSRE's March 6, 2012 Report should be undertaken. These include:

- Compilation of Rock Quality Designation (RQD), Total Core Recovery Percentage, Bedding and Discontinuity Orientations (smaller-scale faults and large scale joints) from the Benguet and Echo Bay core logs and from re-logging of select drill core. Third party consultant oversight for this internally completed work would be approximately US $100,000 (included in $12.4 million aforementioned above).
- As part of the upcoming feasibility study, the resulting structural model should be incorporated in the mineral resource and mineral reserve model updates. Estimated consultant fees for this work are $150,000 and are included in the aforementioned $12.4 million.

A number of drilling intervals were not assayed in earlier drilling campaigns due the apparent absence of mineralization. However, some copper oxide minerals may have been missed during core logging because they are not easily visible by eye or optical microscopy. A campaign to assay these intervals, where samples still exist (pulps, core or core halves), is recommended. Similarly, all unsampled 3-meter drill core intervals from the 2011 and 2012 SAGC drilling campaigns should be split and sent for assays. Estimated shipping and laboratory fees for this program total $55,000 (included in $12.4 million aforementioned above).

Fourteen drillholes were completed in 2011 and 2012 from the new drilling program as shown in Table 10-4 with a total depth of 5,980 meters. The assay results from these drillholes have not been added to the drillhole data base used for the prefeasibility reserve and resource estimates. Adding these results would improve the accuracy of the resource models. It is recommended to update the drillhole data base with this new data prior to updating the mine block model.

The current drillhole data base excludes gold assay results from the Benguet drilling program, except for the assays that were re-done by Echo Bay, due to a bias discovered by Echo Bay and confirmed by IMC. New gold assays of the excluded intervals will supplement the existing database and improve the confidence of the mineral resource and mineral reserve estimates.

The analytical procedure used by Benguet was a complicated wet assay procedure that was more appropriate for high-grade underground samples. Most of the drillhole interval pulps for the Benguet drilling programs were obtained during the negotiations with Benguet in 2011. IMC recommends assaying these pulps for gold using fire assaying with an atomic absorption finish (FA-AAS). Estimated shipping and laboratory fees for this program are $60,000 (included in $12.4 million aforementioned above).

As part of the October 2010 Technical Report, IMC and REI recommended additional diamond drillholes to provide the following:

- Increased confidence in the current indicated resource estimate in areas where current drillhole spacing is wider than average;
- Additional gold data in areas where drilling currently consists of mostly pre-Echo Bay holes that do not have reliable gold assays. This should also upgrade some inferred resource inside the current pit design to indicated resource;
- Better definition of lithology contacts and interpretation in certain areas of the deposit.

Ten (10) drillholes are budgeted for additional drillhole information requested above. This should amount to approximately 4,700 meters of drilling. The estimated cost of this drilling program is $2 million and is included in the exploration budget and is not part of the $12.4 million mentioned above.

26.3 MINING

It is recommended that the pit slope angle study be updated to feasibility study level. This will include compilation of joint orientation data using stereographic projection, statistical analysis of kinematic modes of failure (for bench face design), validation of the design sector (accounting for the bench face design), identification of significant structural features and their potential interaction with developing and final pit walls, completion of boreholes GT-05 and GT-06 (located at the west sector of the pit) and additional laboratory testing on Host Rock and Intrusive Rocks.

The hydrogeology model, pit dewatering model, alteration model, and structural model will need to be completed and finalized. Differentiating the Host Rocks by major rock type would also allow for pit optimization based on the spatial distribution of the andesites. Laboratory results suggest that this rock type is stronger than the other rock types grouped with the Host Rock group.

AMEC is confident that the higher level of detail from these updates and programs will eliminate some of the conservative assumptions made thus far. As a result, slope angles for the pit walls should increase from those used in the Preliminary Feasibility Study. This should improve project economics by decreasing the waste to ore ratio for the mine, thereby reducing waste mining costs and potentially increasing the mine reserve.

The estimated contractor and consultant fees (including drilling and site maintenance) are $4.1 million that is included in the $12.4 million mentioned above in section 26.1.

Evaluate mine scheduling to optimize acid consumption in tailing leaching. Estimated cost $22,000.

26.4 TSF GEOTECHNICAL INVESTIGATION

26.4.1 Tailing Testwork

The testwork required to provide sufficiently detailed engineering decisions at the feasibility stage is relatively modest for the filtered tailing. The recommended tailing testing requirements include cyclic triaxial testing (or cyclic simple shear) on compacted shell and general placement tailing materials, advanced triaxial testing, geochemical testwork, bench-scale filtration testing, extended moisture density work, Soil Water Characteristic Curve (SWCC) testing, variable moisture testwork, and possible field compaction trial.

Additional tailing materials are required to facilitate the tailing geotechnical testwork to support the feasibility level designs. The tailing samples will need to span the range of anticipated ore lithologies.

26.4.2 Geotechnical Investigation Program

The first phase geotechnical field investigation for the proposed Southwest Tailing Drystack facility is anticipated to include drilling of eight (8) geotechnical drillholes, excavation of nineteen (19) test pits, performing two (2) cone penetration tests (CPTs), and performing geophysics surveys along three profile lines.

26.4.2.1 Drill, Log, Sample, and Test Drillholes

AMEC has selected drillhole locations within the footprint of the proposed Southwest Tailing Drystack facility to support the feasibility-level design. The primary objectives of the geotechnical drillholes are to provide empirical strength data for the overburden materials, provide depths to bedrock at the drillhole locations, provide empirical strength data for the soil/bedrock contact zone (if applicable), observe and log the bedrock conditions from rock cores, provide information on groundwater levels, obtain permeability of the bedrock materials underlying the proposed drystack facility, and supplement previous fieldwork and laboratory testing information. By achieving these objectives, the drillholes will help to provide subsurface data for design decisions within the proposed Southwest Tailing Drystack facility footprint.

A total of eight (8) drillholes are proposed for the first phase geotechnical investigation. Soil types, moisture, density, color, weathering, grain size, grain angularity, grain lithology, gradation, plasticity, structure, and other noteworthy characteristics will be logged for each of the soil samples obtained from the drillhole.

26.4.2.2 Excavate, Log and Sample Test Pits

The first phase of test pits may be completed before, during, or after the drilling program. The primary objectives of the test pit program are to observe and log the existing subsoil conditions at each test pit location, collect representative disturbed bulk samples of the subsurface soils for laboratory testing, collect relatively undisturbed tube or block samples of the soils (if deemed necessary), measure the depth to bedrock (if encountered), observe the nature of the soil-bedrock interface (if encountered), and observe the groundwater conditions in each test pit (if encountered).

The results of the test pits will be used to make design decisions regarding subsurface excavation depths within the footprint of the proposed TSF. Test pits in areas outside of proposed drystack facility are provided for the purpose of initial reconnaissance for potential soil and rock borrow for embankment construction. A second phase investigation will then be developed to assess potential construction borrow sources identified within the TSF impoundment limits.

26.4.3 VRMA

26.4.3.1 Geotechnical Exploration Program

The geotechnical field investigation for the selected VRMA will support the feasibility-level design. Approximately seven geotechnical drillholes and seven test pits excavations will be required to investigate the foundation of the VRMA area.

It is anticipated that drillholes will be advanced to a depth equivalent to the proposed ultimate VRMA height, or a nominal distance into competent bedrock (whichever is shallower). Undisturbed and disturbed geotechnical samples will be recovered from each drillhole, and *in situ* permeability testing is proposed in the bedrock materials using the Packer test method. If applicable on completion, each borehole will be screened and retained for future groundwater monitoring.

Representative samples of the various materials will be subjected to geotechnical tests to establish their engineering properties for design. The number of tests will be tailored to the results of the geotechnical field exploration, based in part on the recovery of materials. The proposed testing will include moisture content, dry density of undisturbed samples, particle size distribution (gradation and hydrometer), Atterberg limits, specific gravity, soil water characteristics curve (SWCC) (for cover design), Standard Proctor moisture/density relationship (potential borrow materials), triaxial shear tests (undisturbed samples), remolded triaxial shear tests, direct shear interface tests (for geomembrane liner system, if and when appropriate), permeability, and consolidation.

26.4.3.2 Valueless Rock Material Testing

Samples of the anticipated valueless rock (i.e., waste rock) should be tested for geotechnical parameters in support of the VRMA design. This proposed testing includes large-scale direct shear testing, unconfined compressive strength, and density testing. This may include testing of crushed core materials from the pit slope stability investigation.

26.4.4 Pit Diversions

Completion of boreholes GT-05 and GT-06 is required for the feasibility design of the Kingking River diversion. Currently, it is assumed that channels will be excavated into bedrock; however, if poor quality rock is encountered, a low permeability armoring layer will be required.

26.4.5 Pit Dewatering

To support the feasibility-level pit dewatering design, compilation and evaluation of hydrology information will be required. This information will provide the pumping rate required to remove storm water that has collected in the pit, as well as determine peak flow rates (such as that associated with the 25-year, 24-hour storm event), which will be used for sizing pumps, trenches, swales and sumps. Compilation and evaluation of geology and hydrogeology information will advance the understanding of groundwater flow in the region, which will be used to calibrate the dewatering numerical model. The dewatering numerical model will incorporate hydrogeological parameters that will be obtained by the drilling and testing of wells around the pit. Finally, to understand the general competency of rock masses, compilation and evaluation of geologic structures must be completed. This information will be used in designing horizontal drains and evaluating the pit wall slope stability.

26.4.6 Costs (Tailing, VRMA, Pit Diversions, Pit Design, Pit Dewatering)

To accompany the recommendations put forth for the tailing facility, VRMA, pit diversions, pit design, and pit dewatering, the following costs have been estimated:

- Drilling of approximately 2,000 linear meters at various sites: $650,000
- Cone Penetration Testing (CPT) at the Port, Tailing and VRMA facilities: $200,000
- Performing subsurface geophysics at the Tailing and VRMA facilities: $80,000
- Completion of test pits at various sites: $50,000
- Laboratory testing of collected samples: $75,000
- Completion of pumping tests at the site of the pit: $50,000

26.5 METALLURGICAL

Additional lock cycle flotation tests on low-grade gold-containing ore (0.1 to 0.4 gram Au per ton) are required to develop a feasibility level recovery algorithm. Estimated cost including shipping is $61,000 (included in $12.4 million aforementioned above).

Cleaner tailing mineralogy and liberation should be further studied to characterize and potentially improve recovery. Estimated cost is $11,000 (included in $12.4 million aforementioned above).

Acid consumption in the tailing leach may be reduced by removal of magnetics with the potential to produce salable magnetite concentrate. Echo Bay studied this in 1997 and showed that this may be achievable. Additional annual revenue of US 50-100 million may be possible from the sale of magnetite concentrate. The cost estimate is an estimated $55,000 including sample shipment (included in $12.4 million aforementioned above).

Actual test work, using a laboratory-scale Knelson or Falcon concentrator, is necessary to confirm the gravity concentration assumptions used in the mill design. Half core samples are still available at the JKTech lab and also from core house. The cost estimate is $35,000 (included in $12.4 million aforementioned above).

More column leach tests and studies are recommended to confirm column leach results. These include (1) large column tests (6-meter high columns), (2) closed cycle tests to study build up deleterious elements or compounds in the PLS, (3) proof of cathode quality tests with closed cycle SX-EW pilot plant attached to the column leach confirmatory test, (4) SX PLS isotherm work by organic vendor to test for the extraction, scrubbing and stripping phase of solvent extraction, (5) McCabe-Thiele determination of the number of stages for each phase from individual isotherms. Estimated cost for these programs is $35,000 (included in $12.4 million aforementioned above).

Seventy three percent of Axb results obtained for the King-king samples were below 50, which suggest that HPGR may be a viable option. If trade-off studies on HPGRs or tertiary crushing v. SAG milling are positive, samples should be sent to vendors (Koppern, Polysius or Weir) for HPGR and abrasion testing. A trade-off study (including power plant size reduction) is estimated at $50,000 and metallurgical tests (including sample shipment) at a qualified vendor are estimated at $50,000 (included in $12.4 million aforementioned above).

The large variation in Axb values for samples within the nominated ore types calls for further test work on more samples to quantify the extent of this variation. Primary crusher capacity checks should be carried out, which will require more uniaxial compressive strength (UCS) and Bond crushing index (CWI) measurements. These test results will also be useful in any trade-of studies between conventional secondary and tertiary crushing v. SAG milling. Estimated costs of these tests at vendor labs are $80,000 (*not* included in $12.4 million aforementioned above).

The processing, handling and deposition of tailing will have to be revisited to confirm applicability of conventional equipment and to optimize capital and operating costs. Estimated costs of these studies at equipment vendors are $10,000 (included in $12.4 million aforementioned above).

26.6 PROCESS FACILITIES

Co-mingling valueless rock, spent heap leach ore and tailing has been considered in other studies that involve stacking of filtered tailing. Valueless rock can provide a large stable buttress to the tailing storage facility. In the years when no tailing leach is performed, mill tailing can add positively to the acid-base balance of the valueless rock. Co-mingling may also reduce the cost of land acquisition and people relocation, as well as operating costs. The starting time of co-depositing crushed and conveyed valueless rock with the other two materials may be an important factor to consider in this analysis. The short valueless rock hauls in the early mining years might make it more economical to start crushing and conveying at a later time when hauls are longer. Estimated cost of studies is $100,000.

The low Axb parameter (high hardness) results for majority of the King-king samples tested signify the need to consider other crushing options, namely HPGR and conventional secondary and tertiary crushing. These options may provide capital and operating cost savings. In addition, the potential reduction of comminution power demand can reduce the size, and consequently the capital cost of the power plant. The cost is estimated above in the metallurgy recommendation.

The large variation in Axb values for samples within the nominated ore types is considered a risk, and would lead to significant milling circuit and operational problems if hard ore were processed without blending with softer ore. Mine planning must include Axb as a factor to even out the variation as much as possible.

26.7 INFRASTRUCTURE

Continued evaluation of electric power supply options to improve project capital and operating costs. These include discussions with independent Philippine power providers about long term power supply costs. Estimated cost is in owners' costs and is $10,000.

Develop relationships with North American coal companies currently shipping and expected to ship North American mined coal to international clients. This could lead to future supplies of low-cost coal for the power plant. Estimated cost to travel to meetings is $4,000.

Keep power plant geotechnical studies and basic engineering at the forefront of project development schedules so low cost power is available to the heap leach as soon as practical. Estimated cost is part of aforementioned above $12.4 million and is $100,000 for the site geotechnical work, offsite lab analysis and reporting.

26.8 LAND ACQUISITION AND RELOCATION

It is recommended to proceed with land acquisition through options agreements as soon as areas for various facilities are confirmed by geotechnical programs and completion of studies for co-mingling of valueless materials together in a common storage facility. Lower acquisition costs are more likely to occur if these agreements are in place well before approval to proceed with the project construction is received. Estimated cost for these option agreements is $3.1 million.

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APPENDIX A: FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

CERTIFICATE OF QUALIFIED PERSON

I, Joshua W. Snider, P.E., do hereby certify that:

1. I am currently employed as an Engineer and Project Manager by:

   M3 Engineering & Technology Corporation
   2051 W. Sunset Road, St 101
   Tucson, Arizona 85704
   U.S.A.

2. I am a graduate of the University of Arizona and received a Bachelor of Civil Engineering in 1996.

3. I am a:

   - Registered Professional Engineer in the State of Arizona (No.41971)

4. I have practiced engineering and project management at M3 Engineering for 16 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 report coordination and overall project authorship of the technical report titled "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study, Mindanao, Philippines" dated effective February 25, 2013 (the "Technical Report"). More specifically I am responsible for sections 1.1, 1.2, 1.3, 1.10, 1.12, 1.13, 1.14, 1.15, 2, 3, 4, 5, 6, 18, 19, 21, 22, 23, 24, 25.1, 25.6, 25.8, 25.9, 26.1, 26.7, 26.8, 27. I am also responsible for interpretations, conclusions, and recommendations that relate the sections that I am responsible for.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have not visited the King-King Site.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests of National Instrument 43-101.

11. I have read National Instrument 43-101, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 25th day of February, 2013.

____
Signature of Qualified Person

Joshua W. Snider
Print name of Qualified Person

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 Sections 1.7, 17, 25.5, 26.5 and 26.6 of the technical report entitled "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study, Mindanao, Philippines" dated effective February 25, 2013 (the "Technical Report"). I visited the King-king property on January 25, 2011.

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 St. Augustine Gold and Copper Ltd. and do not own any of their stocks or shares.

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.

Dated this 25th Day of February, 2013.

Art S. Ibrado, Ph.D.

Art S Ibrado
01395QP
Mining & Metallurgical Society of America

CERTIFICATE OF QUALIFIED PERSON

I, Michael G. Hester, do hereby certify that:

1. I am currently employed as Vice President and Principal Mining Engineer by Independent Mining Consultants, Inc. ("IMC") of 3560 East Gas Road, Tucson, Arizona, 85714, USA, phone number (520) 294-9861.

2. I hold the following academic qualifications:

B.S. (Mining Engineering)	University of Arizona	1979
M.S. (Mining Engineering)	University of Arizona	1982

3. I am a Fellow of the Australian Institute of Mining and Metallurgy (FAusIMM #221108), a professional association as defined by National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101"). As well, I am a member in good standing of the following technical associations and societies:

   Society for Mining, Metallurgy, and Exploration, Inc. (SME Member # 1423200)

   Member of Resources and Reserves Committee of the Society of Mining, Metallurgy, and Exploration.

   The Canadian Institute of Mining, Metallurgy and Petroleum (CIM Member #100809)

4. I have worked in the minerals industry as an engineer continuously since 1979, a period of 34 years. I am a founding partner, Vice President, and Principal Mining Engineer for Independent Mining Consultants, Inc. ("IMC"), a position I have held since 1983. I have been employed as an Adjunct Lecturer at the University of Arizona (1997-1998) where I taught classes in open pit mine planning and mine economic analysis. I am also a member of the Resources and Reserves Committee of the Society of Mining, Metallurgy, and Exploration since March 2012. I was employed as a staff engineer for Pincock, Allen & Holt, Inc. from 1979 to 1983. During my career I have had extensive experience reviewing and auditing deposit sampling methods, analytical procedures, and QA/QC analysis. I also have many years of experience developing mineral resource models, developing open pit mine plans and production schedules, calculating equipment requirements for open pit mining operations, developing mine capital and operating cost estimates, performing economic analysis of mining operations and managing various PEA, Pre-Feasibility, and Feasibility Studies.

5. This certificate applies to the Technical Report titled "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study, Mindanao, Philippines", dated effective February 25, 2013 (the "Technical Report"). I have not inspected the property.

6. I have read the definition of "Qualified Person" ("QP") set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association and past relevant

experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

7. I am the responsible QP for Sections 1.8, 1.9, 12, 14, 15, 16, 25.3 and 26.3 of the Technical Report.

8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

9. I am independent of St. Augustine Gold and Copper Ltd. as described in Section 1.5 of the NI 43-101.

10. My prior involvement with the property includes work on the October 2010 Technical Report for Russell Mining and Minerals, Inc. and Ratel Gold.

11. I have read National Instrument 43-101, and the Technical Report has been prepared in compliance with that instrument.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 25th day of February, 2013

Michael G. Hester, FAusIMM

CERTIFICATE OF QUALIFIED PERSON

I, Donald F. Earnest, do hereby certify that:

1. I am the President of Resource Evaluation, Inc., 1955 W. Grant Rd., Suite 125X, Tucson, AZ 85745, USA;
2. I am a graduate of The Ohio State University with the degree of Bachelor of Science, Geology, 1973;
3. I have worked as a Mine Geologist, Senior Mine Geologist, Resident Manager of an operating mine, Manager, Exploration, Vice President, Geology for a major mining consulting firm, and President of a mining geology consulting firm;
4. I have read the definition of a "Qualified Person" set out in Canada National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with the Society for Mining, Metallurgy, and Exploration, Inc. (SME) as a Registered Member (No.883600RM) and past relevant professional experience, I fulfill the requirements to be a "Qualified Person" (QP) for this Technical Report;
5. I am responsible for Sections 1.4, 1.5, 7, 8, 9, 10 (portions), 11, 25.2, and 26.2 of the Technical Report titled "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study, Mindanao, Philippines' dated February 25, 2013 (the "Technical Report"). I visited the King-king property on June 5, 2010, and again on March 23, 2011;
6. 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;
7. I am Independent of St. Augustine Gold and Copper Ltd. and do not own any stock shares or other securities in the company;
8. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with the instrument and form.
9. I consent to the filing of the Technical Report with any stock exchange or other regulatory authority and any publication by these authorities for regulatory purposes, including electronic publication in the public company files on the SEDAR website, and publication by St. Augustine Gold and Copper Ltd. on the company's website.

Dated this 25th day of February, 2013.

Donald F. Earnest

CERTIFICATE OF QUALIFIED PERSON

I, John Gerald Aronson, do hereby certify that:

1. I am currently employed as President, Principal-In-Charge, by:

   AATA International, Inc.
   1400 Wewatta Street
   Suite 310
   Denver, Colorado, USA 80202

2. I am a graduate of Nebraska Wesleyan University (1971), University of Nebraska Lincoln (1973), and have taken post graduate training at Colorado State University, 1974-1984.

3. I am a Certified Fisheries Scientist with the American Fisheries Society, and a Certified Senior Ecologist with the Ecological Society of America. I am a member in good standing with various professional organizations.

4. I have been a professional environmental consultant for 38 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 report coordination and overall project authorship of the environmental and social report sections of the report titled “King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study, Mindanao, Philippines” dated February 25, 2013 (the "Technical Report"). More specifically I am responsible for Sections 1.11 and 20. I am also responsible for interpretations, conclusions, and recommendations that relate the sections that I am responsible for.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have visited the King-King Site on several occasions.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests of National Instrument 43-101.

11. I have read National Instrument 43-101, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic

publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 25th day of February, 2013.

John G Aronson

____

Signature of Qualified Person
John G. Aronson
AATA International, Inc.

CERTIFICATE OF QUALIFIED PERSON

I, Ronald J. Roman, do hereby certify that:

1. I am currently employed as a Metallurgical Engineer by:

   Leach, Inc.
   4741 N. Placita del Sol
   Tucson, AZ, 85749

2. I am a graduate of the Colorado School of Mines and received a Doctor of Science in June 1966.

3. I am a:

   - Registered Professional Engineer in the State of Arizona (No.25799)

4. I have practiced engineering and project management at Leach, Inc. for 21 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 report coordination and overall project authorship of the technical report titled "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study, Mindanao, Philippines" dated February 25, 2013 (the "Technical Report"). More specifically I am responsible for sections 1.6.6 and 13.6. I am also responsible for interpretations, conclusions, and recommendations that relate the sections that I am responsible for.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I have not visited the King-King Site.

9. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information required to be disclosed to make the report not misleading.

10. I am independent of the issuer applying all of the tests of National Instrument 43-101.

11. I have read National Instrument 43-101, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 25th day of February, 2013.

Signature of Qualified Person

Ronald J. Roman

Print name of Qualified Person

CERTIFICATE OF QUALIFIED PERSON

I, Charles C. Rehn, P.E., do hereby certify that:

1. I am employed as the Salt Lake City/Elko Unit Manager at AMEC, 9865 South, 500 West, Sandy, Utah 84070, USA
2. I graduated with the following degrees:
   Bachelor of Science in Civil Engineering, Iowa State University, 1975
3. I am a Professional Member of the Society of Mining, Metallurgy, and Exploration, Inc. (SME).
4. I have worked as a geotechnical engineer for projects in the mining industry for 37 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 experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for Sections 18.5, 18.9, 21.1.4, 21.1.6, 21.1.7, 25.4, 25.7, and 26.4 of the technical report entitled "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study, Mindanao, Philippines" dated February 25, 2013 (the "Technical Report"). I have not visited the site as of the publication date of this Technical Report.
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 St. Augustine Gold and Copper Ltd. and do not own any of their stocks or shares.
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.

Dated this 30th Day of October, 2013.

Charles C. Rehn, P.E.

CERTIFICATE OF QUALIFIED PERSON

I, Gregory J Harbort, Ph.D., do hereby certify that:

1. I am employed as a Process Manager at Amec Australia, Level 4, 144 Edward Street, Brisbane, 4000, Queensland, Australia.

2. I graduated with the following degrees:
   Bachelor of Metallurgical Engineering, University of Queensland, 1985
   Doctor of Philosophy (Mineral Processing), University of Queensland, 2005

3. I am a Qualified Professional (QP) fellow of the Australasian Institute of Mining and Metallurgy (AusIMM)

4. I have worked as a professional and in the mining industry for 27 years.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for Sections 1.6.1-1.6.5, 13.1-13.5 of the technical report entitled "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study, Mindanao, Philippines" dated February 25, 2013 (the "Technical Report"). I have not visited the King-king property.

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 St. Augustine Gold and Copper Ltd. and do not own any of their stocks or shares.

9. I have read National Instrument 43-101 and Form 43-101FI, 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.

Dated this 25th Day of February, 2013

Gregory J Harbort