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# Kingking Copper-Gold Project

## NI 43-101 Technical Report

## Preliminary Feasibility Study

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The effective date of this Technical Report is July 10, 2025. The issue date of this Technical Report is July 29, 2025. See Appendix A, Preliminary Feasibility Study Contributors and Professional Qualifications, for certificates of qualified persons. These certificates are considered the date and signature of this Technical Report in accordance with Form 43-101F1.

KINGKING COPPER-GOLD PROJECT
FORM 43-101F1 TECHNICAL REPORT

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## List of Appendices

| APPENDIX | DESCRIPTION |
|---|---|
| A | Preliminary Feasibility Study Contributors and Professional Qualifications<br>• Certificate of Qualified Person ("QP") |

# 1 SUMMARY

M3 Engineering and Technology Corporation ("M3") in Tucson, AZ was contracted by St. Augustine Gold & Copper, Ltd. ("SAGC") of Makati, Philippines, to prepare a Preliminary Feasibility Study (the "PFS") Technical Report compliant with National Instrument 43-101 ("NI 43-101") on the Kingking Copper-Gold Project (the "Kingking Project"). This section briefly summarizes the findings of the Preliminary Feasibility Study.

The Kingking Project is located near Davao City, in Mindanao, Philippines. The mining rate will be approximately 178,000 tonnes per day (tpd) utilizing contract mining. Over the life of the Kingking Project, 4.55 billion pounds of copper and 7.10 million ounces of gold 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 mill facility treats the mill ore with primary crushing, grinding, flotation, tailing agitated leach (with Solvent Extraction and Electrowinning [SX-EW]), tailing neutralization followed by dry stack tailing placement. The heap leach facility treats the heap leach ore with primary through tertiary crushing, agglomeration, placement on an on-off leach pad that produces pregnant leach solution (PLS) that will be processed in an SX-EW facility.

Other key components of the Kingking Project include a 32-kilometer power line and port facility.

SAGC selected 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 Technical Report. All consultants have the capability to support the Kinking Project, as required and within the confines of expertise, from feasibility study to full operation. The costs are based on fourth quarter (Q4) 2024 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) | 38 years | | |
| Mine Type: | Open Pit | | |
| Process Description: | Crushing, grinding, flotation, flotation tail leach, dry-stack tailing deposition, SX-EW, On/off Leach pad | | |
| Total Material Mined (Tonnes per day) | 178,000 | | |
| Design Mill Throughput (Tonnes per day) | 60,000 | | |
| Design On/Off Leach Pad Throughput (Tonnes per day) | 40,000 | | |
| LOM Copper Ore Grade | 0.26% | | |
| LOM Gold Ore Grade | 0.32 g/t | | |
| Initial Capital Costs ($US Millions) | $2,373.8 | | |
| Sustaining Capital Costs ($US Millions) | $798.4 | | |
| Adjustment for Escalation | None – Assumed 2024 dollars | | |
| **Payable Metals** | | | |
| Copper (Billion Pounds) | 4.4 | | |
| Gold (Million troy ounces) | 6.9 | | |
| **Unit Operating Cost: (per payable pound of Copper Equivalent)** | **Years 1-5** | **Years 1-10** | **LOM** |
| Mining Cost | $0.30 | $0.38 | $0.53 |
| Processing Cost | $0.68 | $0.82 | $1.27 |
| G&A Costs | $0.11 | $0.11 | $0.14 |
| Shipping, Smelting and Refining Costs | $0.09 | $0.11 | $0.12 |
| Government Fees | $0.30 | $0.31 | $0.39 |
| Total Cost | $1.47 | $1.73 | $2.46 |
| By-Product Credits (Gold) | ($0.74) | ($0.76) | ($0.87) |
| Total Consolidated Net Cash Cost | $0.73 | $0.97 | $1.59 |
| **Financial Indicators** | **Base Case** | **High Metal Price (+20%)** | **Low Metal Price (-20%)** |
| Gold Price (price per troy ounce) | $2,150 | $2,580 | $1,720 |
| Copper Price (price per pound) | $4.30 | $5.16 | $3.44 |
| After-Tax Project Internal Rate of Return (IRR) | 34.2% | 44.5% | 22.0% |
| After-Tax NPV at 7% Discount Rate ($ Billions) | $4.2 | $6.5 | $1.8 |
| After-Tax Payback (Years) | 1.9 | 1.5 | 2.6 |
| **Major Permit Status** | | | |
| Environmental Impact Statement Approved/ECC Issued | February 26, 2015 | | |
| Declaration of Mine Project Feasibility Approved | December 3, 2014 | | |

## 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 Kingking Mining Corporation (KMC or the “Company”). It covers a total area of approximately two thousand nine hundred seventy-six (2,976) hectares situated in Sitio Lumanggang, Pantukan.

The draft EIS was submitted to the Environmental Management Bureau (EMB) in February 2012. The DMPF (Declaration of Mining Project Feasibility) including the relocation plan was submitted on May 4, 2012 to the Mines and

Geosciences Bureau (MGB). The endorsements required by the DMPF have been obtained from the Local Government Units (LGU). The Kingking Project has received approval of the Environmental Impact Statement (EIS) from the Philippine Environmental Management Bureau (EMB) on February 26, 2015. In connection with the EIS approval, the EMB has issued the Environmental Compliance Certificate (ECC). The Declaration of Mine Project Feasibility (DMPF) was approved on December 3, 2014 and the Amended Declaration of Mine Project Feasibility (DMPF) was approved on May 16, 2016.

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 on September 2, 2008 and covers a total area of approximately 141,773.3097 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 facilities areas are within 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 is an initial project expense necessary to secure the areas intended to be used for project facilities. The acquisition costs may be spread throughout the life of the Kingking Project. The Kingking Project intends to use an option agreement as its preferred instrument in securing its hold on the project facilities areas.

Initial estimates of potential project-affected people (PAP) and households peg 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 facilities outside the MPSA area will need to be reclassified as heavy industrial, within the Municipality of Pantukan. 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 260-950 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. The physical, chemical, biological, and social environment of the Kingking Project area is summarized in Section 1.11 and detailed in Section 20. 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 Kingking Project is located in the Philippines on the island of Mindanao, in the Municipality of Pantukan, Province of Davao de Oro. It is on the eastern side of the Gulf of Davao, 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 Kingking Project.


**Figure 1-2: Project Location (M3, 2013)**

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.

A number of locations at the Kingking Project site were evaluated for the Tailing Storage Facility (TSF), Valueless Rock Management Area (VRMA) and Mill facilities. The preferred locations of the facilities for the PFS are shown in Figure 1-3.

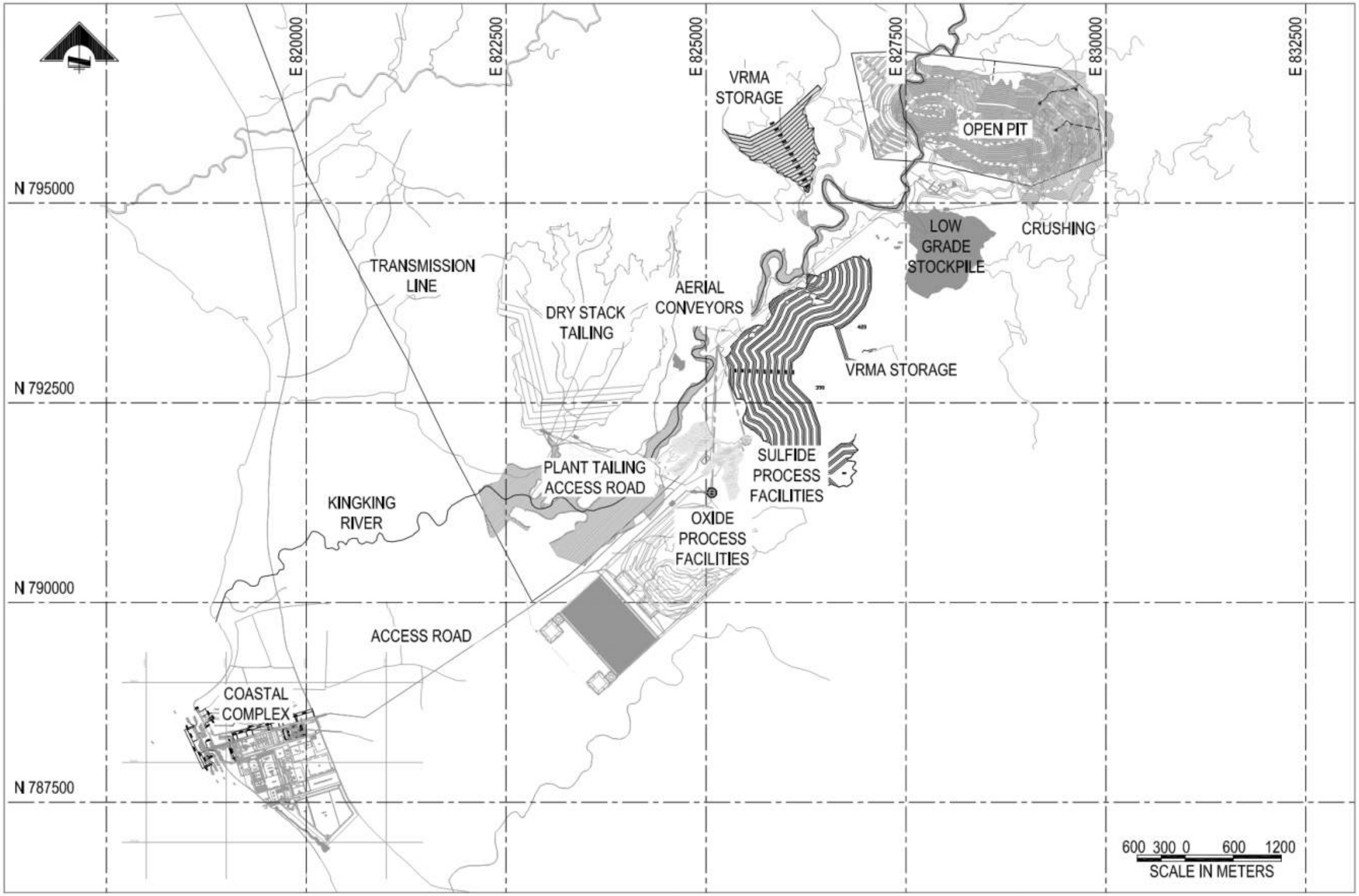



Figure 1-3: Overall Site Facility Map (M3, 2013)

## 1.3 HISTORY

| | |
|---|---|
| 1966-1968 | NADECOR discovered the Kingking mineralization anomaly; |
| 1969-1972 | Mitsubishi Mining Corporation drilled 54 surface diamond drill holes 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 Mineral Production Sharing Agreement (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 Kingking 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 Kingking Project; |
| 2009 | NADECOR and Russell Mining and Minerals, Inc. (RMMI) signed a letter of intent (LOI) to work together to develop the Kingking Project. All Echo Bay data were acquired by Kinross Gold, which waived its option to proceed with the Kingking 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 Kingking Project. NADECOR and RMMI signed a memorandum of understanding (MOU) to advance the Kingking 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 Kingking Project. Ratel Gold (CGA Mining spinoff) and RMMI published a NI 43-101-compliant resource estimate for the Kingking Project. RMMI, NADECOR and Benguet reached a settlement agreement, wherein Benguet relinquished their right, title and interest in the Kingking Project and in the Operating Agreement. |
| 2011 | RMMI assigned its interests in the Kingking 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 Kingking 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 Kingking 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 Kingking Project area's indigenous people was signed. |
| 2012 | The Preliminary EIS was submitted for comments to the DENR (EMB). The Declaration of Mine Project Feasibility (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. |
| 2013 | NADECOR assigned its rights, title and interest over the MPSA in favor of KMC which assumed all obligations and responsibilities thereto. |
| 2014 | December 3, 2014, the Declaration for Mining Project Feasibility (DMPF) for the extraction and commercial disposition of copper, gold, silver and other associated minerals in the contract area of the amended MPSA was filed by NADECOR in MGB. |
| 2015 | February 26, 2015, the Kingking Project has received approval of the NADECOR's Environmental Impact Statement (EIS) from the Philippine Environmental Management Bureau (EMB). In connection with the EIS approval, the EMB has issued the Environmental Compliance Certificate (ECC). |
| 2016 | February 19, 2016, the Certificate Precondition (CP) was secured from National Commission on Indigenous People (NCIP); one of the major requirements for the extension of the amended MPSA.<br><br>The Amended Declaration of Mine Project Feasibility (DMPF) was approved on May 16, 2016.<br><br>MPSA renewed term for another twenty-five (25) years to expire on May 22, 2041. On June 2, 2016, the Department of Environment and Natural Resources approved the expansion of the area covered by the MPSA by annexing Parcels III and IV the 1,767.1641 hectare portion of the area covered by NADECOR's mining application denominated as Application for Mineral Production Sharing Agreement (APSA) No. 000026-IX which was renamed to MPSA No. 009-92-XI Amended II.<br><br>The MGB approved the assignment of the MPSA by BC in favor of NADECOR and the assignment of NADECOR in favor of KMC on June 27, 2016.<br><br>In 2016, the MGB issued the Mineral Processing Permit (MPP) No. 16-2016-XI in favor of NADECOR. However, the imposition of the open-pit mining ban that same year, coupled with the impact of the COVID-19 pandemic from 2020 to 2022, constrained the operational and economic viability of the Kingking Project and hindering compliance with MPP requirements. With the lifting of the open-pit ban in late 2021, the easing of pandemic restrictions, and improved metal prices, NADECOR is now in a position to move forward and is currently completing the requirements to renew the MPP and advance the Kingking Project. |

## 1.4 GEOLOGICAL SETTING AND DEPOSIT TYPE

The Kingking 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 extends well into the surrounding wall rocks.

Much of the sulfide copper mineralization in the Kingking 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 oxidized. 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 Kingking 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 Kingking 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 Kingking 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.

Three companies completed exploration-level drilling campaigns on the Kingking 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 drill holes totalling 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 hydrogeologic 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 Kingking deposit have historically been based upon assays derived from drilled intercepts. Approximately 33,660 samples were collected over the course of the Kingking 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 Kingking Project.

Sample preparation and analysis procedures for the Benguet, Echo Bay, and SAGC drilling campaigns were 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 was accomplished by various laboratories under contract with Benguet and Echo Bay and 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 AMDEL Limited in Australia, JK Tech, and University of South Australia under the supervision AMEC Australia, and by 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 leach drill core samples.

Core samples for metallurgical testing were selected to represent the ore body based on the resource and mining schedule developed in November 2010. Samples were classified as sulfide ore or oxide ore, where oxide ore has an acid-soluble copper content more than 35% of the total copper content.

### 1.6.1 Comminution Tests

The Kingking rock mineralization exhibits variable rock competency and ball mill grindability. Sulfide samples had the lowest Axb, with a median value of 36.0 and The Bond ball mill work indices 13.8 kWh/t. Oxide samples, scheduled for early processing, were less competent, exhibiting a median Axb and an average Bond ball mill work indices of 10.8 kWh/t.

### 1.6.2 Primary Grind and Regrind Sizes

The grind size optimization flotation tests were conducted at $P_{80}$ values of 150 micrometers (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 as grind size decreased from 150 µm to 106 µm. 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 re-grind improves the cleaner concentrate copper grade but at a lower copper recovery.

### 1.6.3 Flotation

The collectors PAX, SIBX, Aero 404, and A3302 were tested at the optimum grind, all at a dosage of 40 g/t. The results demonstrated that the PAX and SIBX and Aero 404 collectors yielded similarly high total copper and gold recoveries. The recovery was slightly lower with Aero 404 but it had a better (lower) mass recovery than the xanthates. During testing, SIBX was chosen as the preferred collector, for which the optimum dosage for SIBX was found to be 30 g/t in the rougher stage. Note that collector and frother dosages, could be reduced in actual operations due to the return of reagents with reclaim water.

Flotation recoveries for copper and gold increased by raising the pH to 10. Raising the pH further to 11 did not significantly improve copper recovery and it depressed gold recovery. 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. While acceptable concentrate grades were attained, the overall recoveries were inadequate. These may be due to the low pH maintained in the roug her stage and the use of SIBX, which is not a selective collector. Further locked-cycle flotation tests should increase the rougher pH to 10 to 10.5 and use a more selective reagent like dialkyl dithiophosphate (Aero 3477) for sulfide ores.

**Table 1-2: Steady State Results of Locked Cycle Test Work**

| Composite | Re-Cleaner 2 Conc. Grade | | Re-Cleaner 2 Conc. Recovery | |
|---|---|---|---|---|
| | Copper (%) | Gold (ppm) | Copper (%) | Gold (%) |
| Oxide | 23 | 61 | 25 | 53 |
| Sulfide | 25 | 39 | 71 | 51 |

Detailed analyses of the concentrates show that arsenic may be a penalty concern over the life of mine composites, reaching a high of 3,700 ppm. 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.

### 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 tonne of ore, for 12 hours. The results show that the dissolution of copper (total or weak-acid soluble (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 was 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 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 over the first thirteen years of the life of mine, and in Years 15, 18, 19, 24 and 25. Extraction of copper from column leach tests were expressed as percentages of hot acid-soluble copper (Cu(AS)hot) in the samples. Recoveries ranging 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.

## 1.7 MINERAL PROCESSING AND RECOVERY METHODS

The Kingking 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 solvent extraction and electrowinning (SX-EW) into copper cathodes. Much of the recovered gold will report to copper concentrate, and a portion will be in bullion form produced by gravity concentration, intensive leaching, electrowinning, and smelting.

Figure 1-4 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%. The heap leach operation will have a capacity of 40,000 tpd.

Run-of-mine (ROM) ore will be crushed to 5 ½ 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 percent 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 Kingking Project 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 percent 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 tonne of ore, at 50°C. After going through a counter-current decantation (CCD) wash, the dissolved copper will be recovered by solvent extraction and electrowinning (SX-EW).

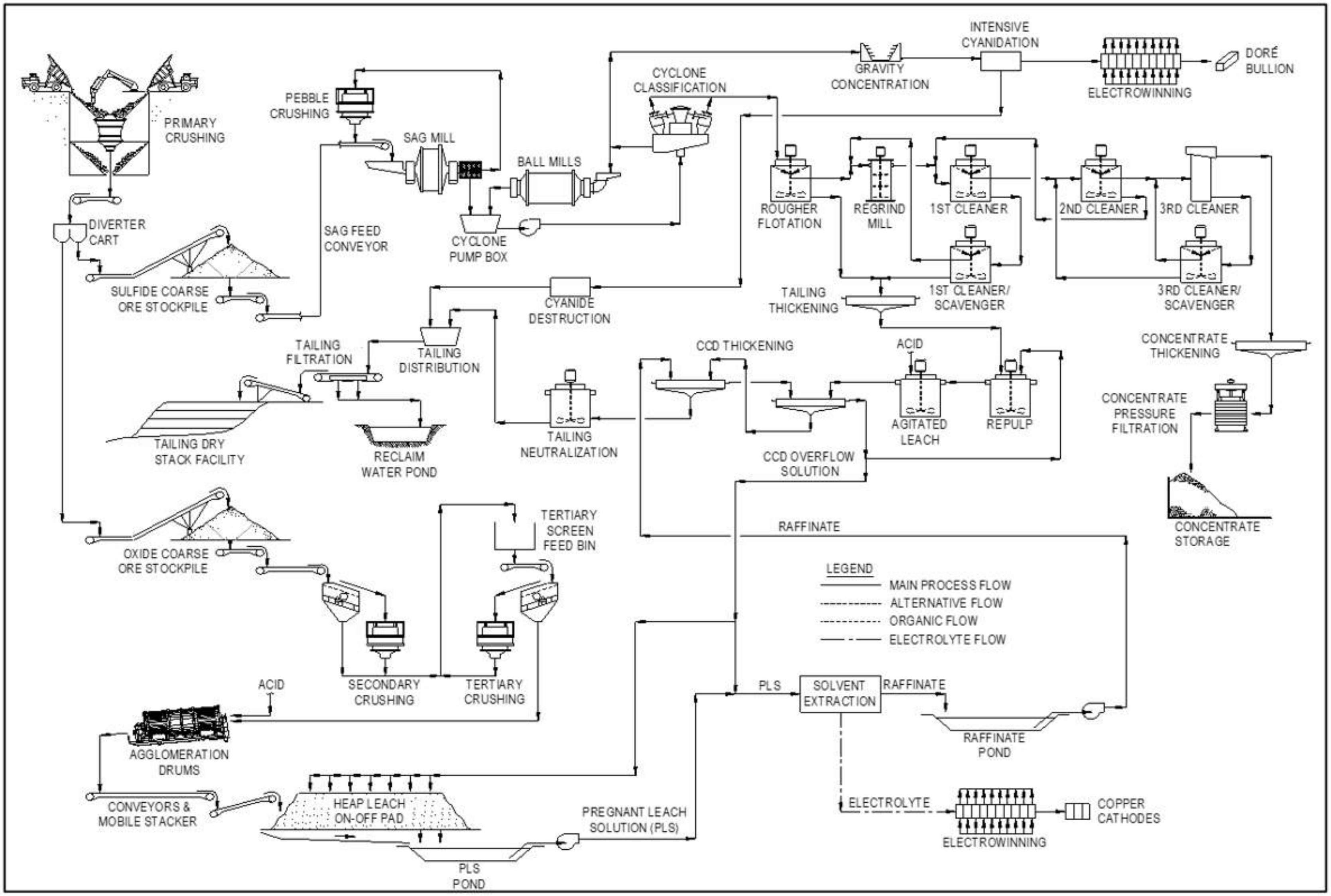



**Figure 1-4: Simplified Process Flow Diagram for the Kingking Process Facility (M3, 2013)**

### 1.7.1 Copper Oxide Ore Heap Leaching

Copper oxide ore (with low gold content) will be mined and leached for copper in an on-off leach pad. 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 6 kg/t of sulfuric acid and water (CCD overflow solution). The agglomerated ore will then be delivered to a stacking conveyor to the heap leach pad and leached with SX-EW raffinate 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 Kingking site is located in an area of active seismicity; therefore, facilities will be designed to resist seismic (earthquake) loads. The Kingking 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.

### 1.7.2 Tailing and Water Systems

Final mill tailing will be thickened and filtered using vacuum belt filters. 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 thickeners. A balance of fresh water at a rate of 674 $m^3$/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 1-3 presents the Mineral Resources for the Kingking Project. The Mineral Resources include Mineral Resources amenable to milling and flotation concentration methods (mill material) and Mineral Resources amenable to heap leach recovery methods (leach material). The upper portion of the table presents the Mineral Resources for mill material. Measured and Indicated Mineral Resources amount to 1.03 billion tonnes at 0.24% total copper and 0.34 g/t gold for 5.55 billion pounds of contained copper and 11.1 million ounces contained gold. Inferred Mineral Resource is an additional 640.5 million tonnes at 0.20% total copper and 0.27 g/t gold for 2.85 billion pounds of contained copper and 5.46 million ounces of contained gold.

The middle portion of the table presents the Mineral Resource for heap leach material. Leach material is oxide/mixed dominant mineralization. Measured and Indicated Mineral Resources amount to 85.0 million tonnes at 0.23% total copper and 0.15% soluble copper and contained metal amounts to 430.0 million pounds of copper. Gold is not recovered in the heap leach process. Inferred Mineral Resource is an additional 34.7 million tonnes at 0.21% total copper and 0.12% soluble copper and contained metal amounts to 159.0 million pounds of copper.

The bottom portion of the table presents the Mineral Resource for combined mill and leach material for copper. Measured and Indicated Mineral Resources amount to 1.12 billion tonnes at 0.24% total copper for 5.98 billion pounds of contained copper. Inferred Mineral Resource is an additional 675.2 million tonnes at 0.20% total copper for

3.01 billion pounds of contained copper. The Mineral Resource for gold is as shown with mill resource since it will not be recovered for leach material.

The Measured, Indicated, and Inferred Mineral Resources reported herein are contained within a constraining pit shell to demonstrate "reasonable prospects for eventual economic extraction" to meet the definition of Mineral Resources in NI 43-101.

The Mineral Resource is reported inclusive of the Mineral Reserve presented in Section 1.8.2.

**Table 1-3: Kingking Mineral Resource**

| Mineral Resource (Milling) | Tonnes Mt | NSR ($/t) | Tot Cu (%) | Sol Cu (%) | Gold (g/t) | Copper (Mlbs) | Gold (Koz) |
|---|---|---|---|---|---|---|---|
| **Measured Mineral Resource:** | **157.4** | **36.01** | **0.32** | **0.12** | **0.47** | **1,102** | **2,360** |
| Mixed Oxide/Sulfide | 56.3 | 51.34 | 0.46 | 0.26 | 0.62 | 570 | 1,115 |
| Sulfide | 101.1 | 27.47 | 0.24 | 0.03 | 0.38 | 533 | 1,245 |
| **Indicated Mineral Resource:** | **877.3** | **23.87** | **0.23** | **0.04** | **0.31** | **4,450** | **8,787** |
| Mixed Oxide/Sulfide | 67.7 | 38.29 | 0.33 | 0.18 | 0.51 | 488 | 1,109 |
| Sulfide | 809.5 | 22.66 | 0.22 | 0.03 | 0.30 | 3,962 | 7,678 |
| **Measured/Indicated Resource:** | **1,034.7** | **25.71** | **0.24** | **0.05** | **0.34** | **5,553** | **11,147** |
| Mixed Oxide/Sulfide | 124.1 | 44.21 | 0.39 | 0.22 | 0.56 | 1,058 | 2,224 |
| Sulfide | 910.6 | 23.19 | 0.22 | 0.03 | 0.30 | 4,495 | 8,923 |
| **Inferred Mineral Resource:** | **640.5** | **20.17** | **0.20** | **0.03** | **0.27** | **2,854** | **5,464** |
| Mixed Oxide/Sulfide | 19.6 | 30.82 | 0.24 | 0.12 | 0.50 | 103 | 313 |
| Sulfide | 620.9 | 19.83 | 0.20 | 0.02 | 0.26 | 2,751 | 5,151 |
| **Mineral Resource (Leaching)** | **Tonnes Mt** | **NSR ($/t)** | **Tot Cu (%)** | **Sol Cu (%)** | **Gold (g/t)** | **Copper (Mlbs)** | **Gold (Koz)** |
| Measured Mineral Resource | 39.3 | 16.28 | 0.25 | 0.18 | N.A. | 220 | N.A. |
| Indicated Mineral Resource | 45.7 | 12.74 | 0.21 | 0.13 | N.A. | 210 | N.A. |
| **Measured/Indicated Resource** | **85.0** | **14.38** | **0.23** | **0.15** | **N.A.** | **430** | **N.A.** |
| Inferred Mineral Resource | 34.7 | 12.12 | 0.21 | 0.12 | N.A. | 159 | N.A. |
| **Copper Mineral Resource Milling and Leaching** | **Tonnes Mt** | **NSR ($/t)** | **Tot Cu (%)** | **Sol Cu (%)** | **Gold (g/t)** | **Copper (Mlbs)** | **Gold (Koz)** |
| Measured Mineral Resource | 196.7 | 32.07 | 0.30 | 0.13 | N.A. | 1,322 | N.A. |
| Indicated Mineral Resource | 923.0 | 23.32 | 0.23 | 0.04 | N.A. | 4,660 | N.A. |
| **Measured/Indicated Resource** | **1,119.7** | **24.85** | **0.24** | **0.06** | **N.A.** | **5,982** | **N.A.** |
| Inferred Mineral Resource | 675.2 | 19.75 | 0.20 | 0.03 | N.A. | 3,013 | N.A. |

Notes:

1. The Mineral Resources have an effective date of February 6, 2024, and the estimate was prepared using the definitions in CIM Definition Standards (10 May 2014).
2. All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.
3. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources are based on prices of $3.75/lb copper and $1800/oz gold.
5. Mineral Resources for leach material are based on an NSR Cut-off of $3.26/t. For leach material NSR($/t) = $74.85 x copper (%) x recovery where recovery is variable but averages 82.1%.
6. The NSR value due to mill copper is based on a flotation component (concentrates) and an agitation tail leach component (cathode). For flotation NSR($/t) = $70.10 x copper (%) x recovery where recovery is variable but averages 36.5% for oxides and 67.9% for sulfides. For tails agitation NSR($/t) = $74.85 x copper (%) x recovery where recovery averages 55.6% for oxide and 14.0% for sulfide.
7. The NSR value for mill gold is based on a flotation component (concentrates) and a gravity component. For flotation NSR($/t) = $52.08 x gold (g/t) x recovery where recovery averages 38.6% for oxides and 55.1% for sulfides. For gravity NSR($/t) = $54.31 x gold (g/t) x recovery where recovery averages 23.5% for oxides and 19.3% for sulfides.
8. Total NSR for milling is the sum of the copper and gold components. The NSR calculations account for smelter/refinery treatment charges and payables.

9. Mineral Resources for mill material are based on NSR Cut-offs of $11.20/t for oxide material and $11.43/t for sulfide material due to slight differences in crushing and grinding costs.
10. Table 14-2 accompanies this Mineral Resource and shows all relevant parameters.
11. Mineral Resources are reported in relation to a conceptual constraining pit shell in order to demonstrate reasonable prospects for eventual economic extraction, as required by the definition of Mineral Resource in NI 43-101; mineralization lying outside of the pit shell is excluded from the Mineral Resource.
12. The Mineral Resource is reported inclusive of the Mineral Reserve.
13. All the mineralization comprised in the Mineral Resource estimate is contained on mineral titles controlled by St. Augustine and NADECOR. The constraining pit shell that defines the Mineral Resource includes small amounts of waste material on mineral titles controlled by others. The extraction of the entire Mineral Resource will require agreements with other mineral title owners to mine waste on their mineral titles.

### 1.8.2 Mineral Reserve

Table 1-4 presents the Mineral Reserve estimate for the Kingking Project. There are Mineral Reserves amenable to milling and Mineral Reserves amenable to heap leaching. The Proven and Probable Mineral Reserves amenable to milling amount to 848.9 million tonnes at 0.26% total copper and 0.36 g/t gold for 4.84 billion pounds of contained copper and 9.77 million ounces of contained gold. The Proven and Probable Mineral Reserve amenable to heap leaching amounts to 110.5 million tonnes at 0.23% copper for 555 million pounds of contained copper. The effective date of this Mineral Reserve estimate is April 15, 2024. The low-grade stockpile portion of the Mineral Reserve is economic material, but lower grade, that will be stockpiled and processed at the end of open-pit operations. The bottom portion of the table shows the total copper Mineral Reserve for milling and leaching. The gold Mineral Reserve is as shown for Mineral Reserves amenable to milling.

The Mineral Reserve estimate is based on an open-pit mine plan and mine production schedule developed by IMC under the direction of the QP for this section. The Mineral Reserve estimate is based on commodity prices of $3.75/lb copper and $1,800/oz gold. Measured Mineral Resource in the mine production schedule was converted to Proven Mineral Reserve, and Indicated Mineral Resource in the schedule was converted to Probable Mineral Reserve.

The Mineral Reserves are classified in accordance with the "CIM Definition Standards – For Mineral Resources and Mineral Reserves" adopted May 10, 2014 by the CIM Council (as amended, the "CIM Definition Standards") in accordance with the requirements of NI 43-101. Mineral Reserve estimates reflect the reasonable expectation that all necessary permits and approvals will be obtained and maintained.

The QP for this section does not believe that there are significant risks to the Mineral Reserve estimate based on metallurgical or infrastructure factors or environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors. There has been a significant amount of metallurgical testing, however recoveries lower than forecast would result in loss of revenue for the Kingking Project. Other risks to the Mineral Reserve estimate are related to economic parameters such as prices lower than forecast or costs higher than the current estimates. The impact of these is modeled in the sensitivity study with the economic analysis in Section 22.

All the mineralization comprised in the Mineral Reserve estimate with respect to the Kingking Project is contained on mineral titles controlled by St. Augustine and NADECOR, its partner in the Kingking Project. The current pit design includes small amounts of waste material are located outside of the mineral tenement title. The Mineral Reserve estimate has been prepared based on the Qualified Person's reasoned judgment, in accordance with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practices Guidelines and his professional standards of competence, that there is a reasonable expectation that all the necessary permits, agreements and approvals will be obtained and maintained, including the additional agreements with other parties to allow mining of waste material on their mineral titles.

**Table 1-4: Mineral Reserve**

| Mineral Reserve (Milling): | Tonnes Mt | NSR ($/t) | Tot Cu (%) | Sol Cu (%) | Gold (g/t) | Copper (Mlbs) | Gold (Koz) |
|---|---|---|---|---|---|---|---|
| **Proven Mineral Reserve:** | **142.3** | **37.83** | **0.33** | **0.12** | **0.49** | **1,037** | **2,253** |
| Oxide Mill Ore | 45.4 | 58.56 | 0.52 | 0.30 | 0.71 | 516 | 1,033 |
| Sulfide Mill Ore | 76.5 | 31.90 | 0.26 | 0.04 | 0.45 | 444 | 1,117 |
| Low Grade Stockpile | 20.4 | 13.93 | 0.17 | 0.02 | 0.16 | 78 | 103 |
| **Probable Mineral Reserve:** | **706.5** | **25.51** | **0.24** | **0.04** | **0.33** | **3,803** | **7,519** |
| Oxide Mill Ore | 52.0 | 43.46 | 0.36 | 0.21 | 0.59 | 412 | 986 |
| Sulfide Mill Ore | 499.7 | 27.08 | 0.25 | 0.03 | 0.36 | 2,776 | 5,751 |
| Low Grade Stockpile | 154.9 | 14.43 | 0.18 | 0.02 | 0.16 | 615 | 782 |
| **Proven/Probable Reserve:** | **848.9** | **27.57** | **0.26** | **0.06** | **0.36** | **4,840** | **9,771** |
| Oxide Mill Ore | 97.4 | 50.50 | 0.43 | 0.25 | 0.64 | 928 | 2,018 |
| Sulfide Mill Ore | 576.2 | 27.72 | 0.25 | 0.03 | 0.37 | 3,220 | 6,868 |
| Low Grade Stockpile | 175.3 | 14.37 | 0.18 | 0.02 | 0.16 | 692 | 885 |
| **Mineral Reserve (Leaching):** | **Tonnes Mt** | **NSR ($/t)** | **Tot Cu (%)** | **Sol Cu (%)** | **Gold (g/t)** | **Copper (Mlbs)** | **Gold (Koz)** |
| Proven Mineral Reserve | 50.2 | 14.97 | 0.25 | 0.16 | N.A. | 275 | N.A. |
| Probable Mineral Reserve | 60.4 | 12.22 | 0.21 | 0.12 | N.A. | 280 | N.A. |
| **Prov/Prob Leach Reserve** | **110.5** | **13.47** | **0.23** | **0.14** | **N.A.** | **555** | **N.A.** |
| **Copper Mineral Reserve Milling and Leaching** | **Tonnes Mt** | **NSR ($/t)** | **Tot Cu (%)** | **Sol Cu (%)** | **Gold (g/t)** | **Copper (Mlbs)** | **Gold (Koz)** |
| **Proven/Probable Reserve** | **959.4** | **25.95** | **0.26** | **0.06** | **N.A.** | **5,396** | **N.A.** |
| Proven Mineral Reserve | 192.5 | 31.87 | 0.31 | 0.13 | N.A. | 1,312 | N.A. |
| Probable Mineral Reserve | 766.9 | 24.46 | 0.24 | 0.05 | N.A. | 4,083 | N.A. |

Notes:

1. The Mineral Reserve estimate has an effective date of April 15, 2024 and was prepared using the CIM Definition Standards (10 May 2014).
2. Columns may not sum exactly due to rounding.
3. Mineral Reserves are based on commodity prices of $3.75/lb copper and $1,800/oz gold.
4. Mineral Reserves amenable to milling are based on net of processing (NOP) cut-offs that vary by time period to balance mine and plant production capacities (see Section 16). They range from a high of $21/t to a low of $0.01/t (internal cut-off grade). NOP is calculated as the NSR value minus processing and G&A costs.
5. The NSR value due to mill copper is based on a flotation component (concentrates) and an agitation tail leach component (cathode). For flotation NSR($/t) = $70.10 x copper (%) x recovery where recovery is variable but averages 35.8% for oxides and 68.3% for sulfides. For tails agitation NSR($/t) = $74.85 x copper (%) x recovery where recovery averages 57.4% for oxide and 14.6% for sulfide.
6. The NSR value for mill gold is based on a flotation component (concentrates) and a gravity component. For flotation NSR($/t) = $52.08 x gold (g/t) x recovery where recovery averages 38.6% for oxides and 55.8% for sulfides. For gravity NSR($/t) = $54.31 x gold (g/t) x recovery where recovery averages 23.7% for oxides and 19.0% for sulfides.
7. Total NSR for milling is the sum of the copper and gold components. The NSR calculations account for smelter/refinery treatment charges and payables.
8. Mineral Reserves amenable to heap leaching are based on an NSR cut-off of $3.26/t. For leach material NSR($/t) = $74.85 x copper (%) x recovery where recovery is variable but averages 83.8%.
9. Table 15-2 accompanies this Mineral Reserve estimate and shows all relevant parameters.

## 1.9 MINING METHODS

### 1.9.1 Operating Parameters and Criteria

The Kingking mine will be a conventional open pit mine. Mine operations will consist of drilling holes with large diameter (27.3 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. There will be low-grade stockpile facilities to store marginal ore material for processing at the end of commercial pit operations. There will be a fleet of track dozers, rubber tired dozers, motor graders, and water trucks to maintain the working areas in the pit, VRMA area, stockpiles and the roads.

The mine plan was developed to deliver mill ore at a nominal rate of 60,000 tpd or 21.9 million tonnes per year, though the production rate varies depending on the relative amounts of mixed oxide/sulfide versus sulfide material. Heap leach ore is processed at a maximum rate of 40,000 tpd or 14.6 million tonnes per year. The total mining rate (ore plus waste) will be approximately 178,000 tpd or 65 million tonnes per year. The heap leach process is expected to start about 18 months before the mill and finish approximately 19 years into the Kingking Project. The mill continues to process sulfide predominant ore until the end of the mine life, about 38 years based on the current production schedules.

### 1.9.2 Geotechnical and Hydrological Considerations

AMEC Environment & Infrastructure (AMEC) has developed a range of credible overall slope angles for pit development at the Kingking Project, which will commensurate with a scoping-level study. The slope study used information from drill hole data collected from five oriented core drill holes and from three geohydrology drill holes 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 required to support a feasibility level study for the Kingking Project. 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 Kingking Project is still under development. The incorporation of such structures in the geotechnical pit design is recommended for a feasibility level study report. Therefore, the overall slope angles provided herein will be adjusted as needed upon completion of additional slope studies.

It is the opinion of the QP for this section that the slope design work done to date is adequate for this current Technical Report. Significant additional work is required prior to detailed design and the commencement of mining operations.

A significant aspect of the Kingking 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 12, the final diversion channel will be constructed.

Hydrology and hydraulic calculations were performed to determine the required size for the Kingking River diversion. The diversion will be constructed within a 75-meter-wide pit bench. The 75-meter width will provide room for the diversion channel, safety berm(s) and access road(s).

### 1.9.3 Valueless Rock Management

Figure 1-5 shows the final pit and the various valueless rock management areas (VRMA's) and stockpiles. Total waste in the mine plan is 839.0 million tonnes that consists of 748.1 million tonnes of rock and 90.9 million tonnes of overburden. The facilities to contain this material are:

- The southwest VRMA contains 332.1 million tonnes.
- The west VRMA contains 377.8 million tonnes.
- The pit backfill area in the west pit area contains 129.1 million tonnes.

Low-grade mill ore that is stockpiled during open pit operations and processed after open pit mining is complete is 175.3 million tonnes. This is stored in two facilities:

- The north stockpile (Stage 1) contains 114.4 million tonnes.
- The south stockpile (Stage 2) contains 60.9 million tonnes.

During preproduction and the first four years of commercial production, the north stockpile will be used to stockpile heap leach ore (only). This amounts to 15 million tonnes which will all be processed by the end of Year 4.

The VRMA's and stockpiles are all constructed in lifts from the bottom up. They are developed in 30 m lifts at angle of repose (37°). There is a 50 m setback between lifts so the overall slope angle is 3H:1V, about 18.5°. It is anticipated that this is flat enough to simplify closure.

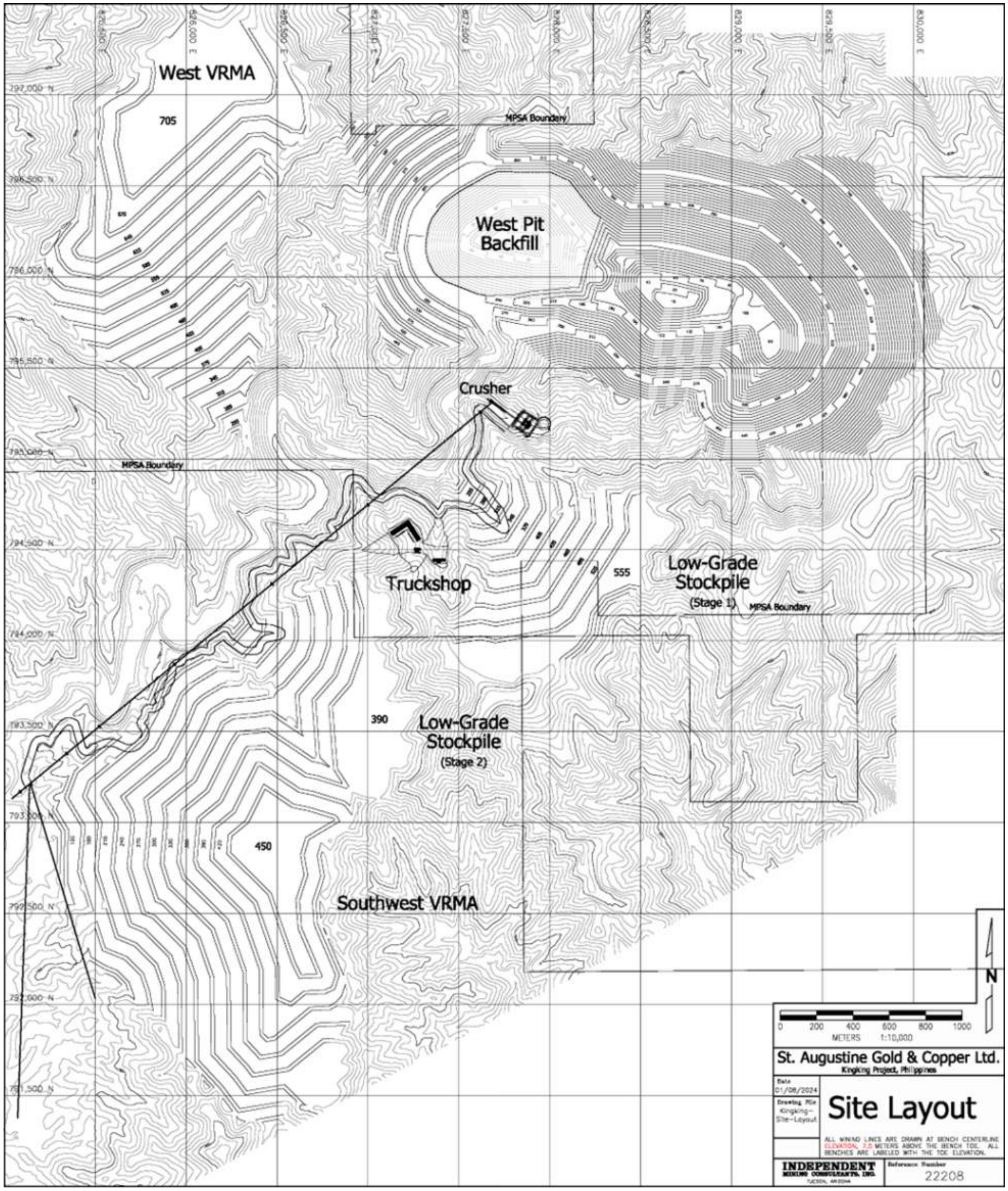



**Figure 1-5: Mine Site Layout (IMC, 2024)**

## 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 Kingking Project requiring this infrastructure are:

- Coastal complex – Includes ship loading/unloading, bulk material storage (reagents and concentrate for example), administration, warehousing and storage, laboratory, medical / fire / rescue, and living quarters.
- Power transmission – Includes 138 kV transmission line to site.
- Mill – Includes 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.
- Dry stack tailing storage facility – Includes tailing dewatering plant.
- 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 process operations 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 (2) lane eight (8) m 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 Transmission and Distribution

The National Grid Corporation of the Philippines (NGCP) identified the Maco substation located in Maco, Davao de Oro for project tie in. A 32 km 138 kV suspended power transmission line with a 60 m right of way between the Maco Substation and the Mill Site Substation is proposed. 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 LAN and microwave transmission from a hub in Davao City.

### 1.10.5 Water Management

Process make up water will be supplied to the Kingking 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. Package wastewater treatment plants will come on line during the construction period and be utilized for on-site treatment. During routine operations wastewater from the camp and other facilities will be treated in these plants prior to discharge.

Water from horizontal pit de-pressurization 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 the 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 sulfuric acid, limestone, grinding media, fuel, reagents and replacement equipment/parts and outgoing concentrate and cathode. 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/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 dissembled 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

The Kingking Project has established a comprehensive and meaningful environmental and social understanding of the Kingking Project and its operational area, including local communities, indigenous peoples, and artisanal mining centers. It has developed detailed environmental and social baseline information, analyses, impact assessment, and documentation for support of an active and comprehensive permitting program. The deep understanding of the site and its environs supports project planning and engineering efforts, allowing for practical and effective management, mitigation, and decision making. As an example, the original onsite coal fired power plant infrastructure has now been replaced with power from the National grid, to be supplied by a new powerline. The Kingking Project has committed to following key international environmental standards and protocols.

Interactions with local communities and stakeholders have been conducted continuously with positive results. The Kingking Project has engaged continued revegetation efforts over the past years and continues to operate its own functional greenhouse to supply vegetative plantings for local environmental improvements, reclamation, and revegetation management applications.

The Kingking Project maintains active compliance with all of its current permits and approvals and is up to date on all required permit compliance inspections and reporting, except for the Mineral Processing Permit which expired in 2021 and is currently in the process of being renewed.

### 1.11.1 Physical Environment

The topography in the Kingking Project area is steep and rugged with elevations ranging from 250 to 950 meters above mean sea level. The climate is tropical with daytime temperatures ranging from 18 to 35ºC, and annual precipitation ranges 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. The Kingking Project also installed and operated digital hydrological gaging stations at key locations in the Kingking River watershed. There is a high sediment transport rate in the Kingking river.

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 especially during daytime. Noise levels in non-residential levels were below the limits.

Heavy rains often cause flooding, and can result in landslides, especially in steep cleared areas.

### 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 high TSS, Cu, Hg, cyanide, and total coliform concentrations, all 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, indicated by elevated coliform bacteria levels. Fugitive mercury from local artisanal gold amalgamation activities has been detected in downstream locations of the Kingking River to the sea. One of the primary goals of the Kingking Project is to significantly improve local water quality by applying modern environmental controls and cleanup protocols.

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

### 1.11.3 Biological Environment

Six general types of vegetation were recognized: open-canopy mid-mountain forest, brushland, wooded grassland, agricultural plantations (coconut and banana), riparian-riverine vegetation, and coastal vegetation. A total of 301 species were recorded in the 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. A comprehensive Biodiversity Action Plan will be developed and engaged.

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 in the marine environment.

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 throughout the project operations.

#### 1.11.4 Social Environment

Based on the 2020 Census, Pantukan's population of 90,786 people were in an estimated 20,400 households. Pantukan is divided into 13 barangays. Barangays Bongbong, Kingking, 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 Davao de Oro 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. Small scale mining is also a source of livelihood throughout the region.

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.

#### 1.11.5 Permitting

The multidisciplinary baseline studies supporting the EIS and SEIA have been completed, and the main permits issued. The Environmental Compliance Certificate was issued in February 2015. An update to the original EIS is anticipated. Monitoring of selected key parameters continues to be performed, and the Kingking Project has established a Permit and Inspection Tracking System. The Kingking Project has achieved ISO 14001:2015 Certification of its Environmental Management System (EMS) which covers all Preliminary Support Activities 2023-2026.

### 1.12 CAPITAL COST SUMMARY

Initial capital costs have been estimated for the Kingking 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 Kingking Project such as the process, tailing, mining facilities and port facilities. The costs also include pre-production mining, owner's costs, and contingency. Table 1-5 shows a summary of the initial capital costs. The estimate was prepared in Q4 2024 US dollars.

**Table 1-5: Initial Capital Cost**

| Area | Description | ($ Millions) |
|---|---|---|
| Process Plant and General Infrastructure | General Site, primary crushing, aerial conveyors, heap leach, grinding, flotation, SX-EW, Agitated leach, tailing dewatering, water systems, power transmission line, on site power distribution, mine support infrastructure, ancillary facilities, EPCM, freight, import duties. | $1,638.25 |
| Mine | Contract mining development costs prior to the start of production. | $131.92 |
| Port Facility | Dock Facility, Concentrate loading, Coastal Complex | $50.00 |
| Owners Costs | Land Acquisition, Construction/ Operating Camps, Environmental Permits, Initial Fills, Owner's Project Management, Security, Early Staffing, Community relations. | $163.48 |
| Contingency | Contingency on all parts of the Kingking Project | $390.13 |
| Escalation | Not included in this estimate | $0 |
| Total Before VAT | | $2,373.78 |
| Value Added Tax (VAT) | | $189.62 |

## 1.13 OPERATING COST SUMMARY

The operating cost was estimated in compliance with PFS requirements based on bottom-up methodology. The operating cost summary is provided in Table 1-6 below, which includes the average cost over initial 5 and 10 years as well as the Life of the Mine (LOM). These operating costs illustrate that Kingking is expected to be a low-cost producer. For instance, the operating cost at Kingking for the first 10 years of full production is expected to be $0.73/lb net of by-products (gold).

**Table 1-6: Operating Cost Summary**

| Time Period | Units | Years 1-5 | Years 1-10 | LOM(1) |
|---|---|---|---|---|
| Payable Pounds of Copper | millions | 1,346 | 2,109 | 4,426 |
| Mining | $/lb Cu | $0.50 | $0.63 | $0.93 |
| Processing | $/lb Cu | $1.14 | $1.36 | $2.26 |
| **Operating Costs** | **$/lb Cu** | **$1.64** | **$2.00** | **$3.19** |
| G&A | $/lb Cu | $0.18 | $0.19 | $0.25 |
| Reclamation & Closure | $/lb Cu | $0.00 | $0.00 | $0.02 |
| **Cash Costs at Mine** | **$/lb Cu** | **$1.82** | **$2.19** | **$3.46** |
| Government Fees | $/lb Cu | $0.50 | $0.52 | $0.70 |
| **Total Cash Costs at Mine** | **$/lb Cu** | **$2.31** | **$2.70** | **$4.16** |
| Shipping, Smelting and Refining | $/lb Cu | $0.15 | $0.18 | $0.22 |
| **Total Costs** | **$/lb Cu** | **$2.46** | **$2.88** | **$4.37** |
| By-Product Credits | $/lb Cu | -$1.24 | -$1.27 | -$1.55 |
| **Consolidated Net Cash Costs** | **$/lb Cu** | **$1.23** | **$1.61** | **$2.82** |
| (1) **Includes Year -1 heap leach production** | | | | |

**Table 1-7: Summary of Unit Operating Costs per tonne of Material Processed**

| | Total Cost (Life of Mine $million) | Total Tonnes Processed (Millions) | $/tonne processed | Years of Operation |
|---|---|---|---|---|
| Contract Mining Costs | $4,137.1 | **959.4** | $4.31 | -2 through 38 |
| Heap Leach & SX-EW | $366.4 | **110.5** | $3.32 | -2 through 19 |
| Concentrator & Tailing | $7,336.3 | **848.9** | $8.64 | 1 through 38 |
| Agitated leach | $2,297.7 | **1,080.2** | $2.13 | 1 through 38 |
| G&A, Lab, Port & Custom Duties | $1,102.4 | **959.4** | $1.15 | -2 through 38 |

## 1.14 ECONOMIC ANALYSIS

The Kingking Project economics were prepared using a discounted cash flow model. The financial indicators examined for the Kingking 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 38 years. Metal price assumptions are $4.30/pound copper, $2,150/ounce gold. The after-tax financial indicators based on a 100% equity case are summarized as follows:

**Table 1-8: Economic Indicators After-Tax**

| Economic Indicator | After-Tax |
|---|---|
| NPV @ 0% ($000) | $10,373,427 |
| NPV @ 5% ($000) | $5,294,176 |
| NPV @ 7% ($000) | $4,181,602 |
| NPV @ 10% ($000) | $2,987,465 |
| IRR % | 34.2% |
| Payback - years | 1.9 |

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 Kingking Project was estimated at conservative metal prices.

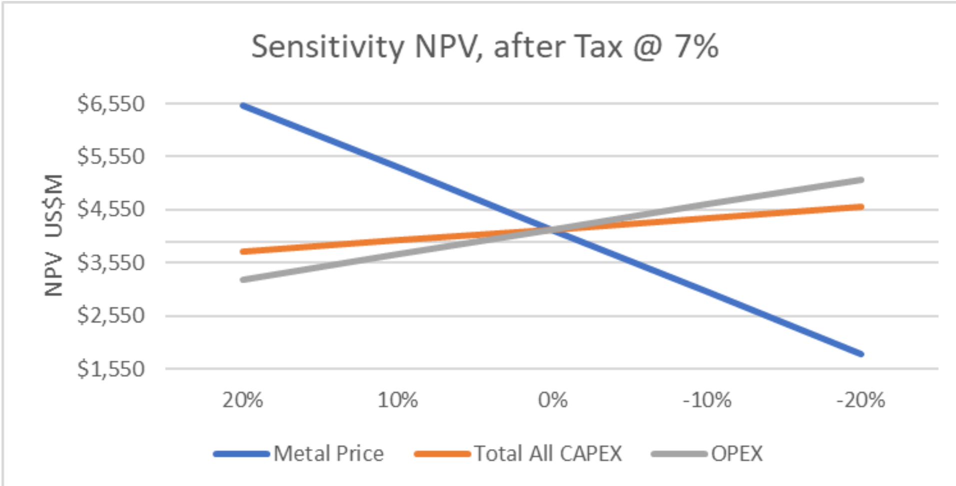



**Figure 1-6: After-Tax NPV (7%) Sensitivities**

## 1.15 CONCLUSIONS AND RECOMMENDATIONS

### 1.15.1 General

The results of this study indicate the Kingking project is both technically and economically feasible and demonstrates robust returns even at conservative prices. Also, there is financial benefit to the Kingking Project by start-up of heap leach operations one year before mill operations.

M3 recommends that the Kingking 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 preliminary feasibility study indicates this project has favorable economics at conservative metal prices. As a result, it is recommended to advance the Kingking Project to the next phase of development which is completion of the feasibility study report.

The project economics are summarized below:

- Gold Price –$2,150 per troy ounce
- Copper Price –$4.30 per pound
- Average Annual Revenue –$1.8 billion (during first five years of full production)
- Net Present Value (NPV after tax) –$4.2 billion at 7% discount rate
- Internal Rate of Return (IRR) After-Taxes – 34.2%
- Payback – 1.9 years
- Initial Capital Cost –$2.4 billion (includes contingency of $0.32 billion)
- LOM operating cost (net of metal credits) - $1.64 per pound of copper ($1.27 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 drill hole data gathered from drilling programs through 1998.

Fourteen (14) drill holes were recently completed with a total depth of 5,980 meters. New data from these drill holes was 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 new information from the new drill holes. The inclusion of this information is expected to increase the confidence level of the resource and reserve estimate for the deposit.

#### 1.15.4 Mining

The results of this study indicate that the Kingking Project has the potential to become an economic producer of copper and gold.

This study has developed a proven and probable mineral reserve amenable to milling of 848.9 million ore tonnes at 0.26% total copper and 0.36 g/t gold. This amounts to 4.84 billion pounds of contained copper and 9.77 million ounces of contained gold. A proven and probable mineral reserve amenable to heap leaching amounts to 110.5 million tonnes at 0.23% copper for 555 million pounds of contained copper.

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

The mining methods proposed for Kingking are conventional open pit methods for bulk mining. There are no significant technical challenges to mining at Kingking.

#### 1.15.5 Tailing / Geotechnical

The results of this study indicate that dry stacking of tailing material is the most economical and least risk option for tailing storage, given the high seismicity and steep topography in the area.

#### 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 saleable copper chalcopyrite/bornite concentrate containing gold.
- Leaching of copper oxide minerals in flotation tailing with sulfuric acid followed by SX-EW to produce saleable copper cathodes.
- Heap leaching of ore containing copper oxide minerals, with very low gold values, using sulfuric acid followed by SX-EW to produce saleable 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 location is advantageous as it is relatively near a large population center, only 10 kilometers from the sea, 32 km from reliable grid power and has a two-lane concrete road within 10 kilometers from the deposit; however, it is challenged with high rainfall and steep terrain. Open pit mining methods used in this study are understood and have been applied extensively in the industry.

The proposed port facility meets the needs of the Kingking Project for import of reagents and other consumables as well as export of products.

#### 1.15.8 Recommendations

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 are recommended in the next phase in collaboration with industry experts in the US, Europe, and Asia.

The engineering cost for geoscience and mining studies, feasibility study (FS) through to 15% of engineering, is estimated at $21.3 million, as shown in Table 1-9.

**Table 1-9: Summary of Cost to Complete a Feasibility Study and Basic Engineering**

| Cost Category | Estimate (US$ millions) |
|---|---:|
| Exploration and Geology | 2.5 |
| Mining | 5.0 |
| TSF, VRMA, Pit Diversions & Dewatering | 1.5 |
| Metallurgical Testing | 0.2 |
| FS and Basic Engineering | 11.6 |
| Due Diligence | 0.5 |
| **Total** | **21.3** |

# 2 INTRODUCTION

## 2.1 PURPOSE

This Technical Report was compiled by M3 Engineering & Technology Corporation ("M3") for SAGC with respect to the Kingking 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 Kingking deposit, and to apply accepted cost 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 Kingking 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 ("QPs") 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 |
|---|---|---|---|---|
| Daniel Roth | M3 Engineering & Technology Corporation – Tucson, AZ | PE, P.Eng. | July 10, 2025 | 1*, 2, 3, 4, 5, 6, 18, 19, 23, 24, 25*, 26*, and 27 |
| Benjamin Bermudez | M3 Engineering & Technology Corporation – Tucson, AZ | PE | N/A | 1.12, 1.13, 1.14, 1.15.2, 21*, 22, 25.1, and 26.1 |
| Art S. Ibrado | Fort Lowell Consulting – Tucson, AZ | PE | January 25, 2011 | 1.6, 1.7, 1.15.6, 13, 17, 25.5 and 26.5 |
| Michael Hester | Independent Mining Consultants (IMC) | FAusIMM | N/A | 1.8, 1.9, 1.15.4, 10, 11, 12, 14, 15, 16, 21.1.1, 21.2.1, 25.3, and 26.3 |
| Donald Earnest | Resource Evaluation Inc. (REI) | P.Geo., SME Registered Member | March 19 - 23, 2011 | 1.4, 1.5, 1.15.3, 7, 8, 9, 25.2, and 26.2 |
| John G. Aronson | ESG Resiliency Plus, LLC. | SME, CEP | October 25-28, 2010; November 30-December 9, 2010; January 13-20, 2011 | 1.11 and 20 |

**Indicates QP is responsible for listed section, except for the sub-sections specifically identified by another QP on this table.*

M3 also relied upon the following consultants for various parts of the Kingking Project:

- AMEC**, Denver, CO, and Salt Lake City, UT, USA - Provided design information and costing data for dry-stack 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.

- Halcrow**, A CH2M Hill Company, Manila, Philippines- Provided the design and capital/ operating costing for the port facilities.
- 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.

** Indicates that the consultant was relied on for the 2013 PFS Technical Report. Information for this updated PFS Technical Report was maintained or referenced regarding the specific area of scope.

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

## 2.3 SITE INSPECTION

Daniel Roth visited the Kingking project site on July 10, 2025 and visually inspected the proposed facilities including the process plant, the tailings storage facility, the heap leach pad and the port area. Existing facilities were also visited, including the core shed on site, and the Maco electrical substation.

Art S. Ibrado visited the Kingking project site on January 25, 2011 and inspected the proposed sites for the process plant, TSF dry stack, heap leach pad, spent ore stockpile, and port.

Donald Earnest visited the Kingking project site from March 19 to 23, 2011, during which he inspected the site locations of the core holes drilled after his prior site visit. He also inspected the core storage facility and found that it continued to be well maintained and accessible.

John G. Aronson visited the Kingking project site multiple times as the Senior Environmental Program manager for the Environmental and Social Impact Assessment (ESIA). Field investigations of the project site covering oceanic, coastal, and upland environments were made October 25-28, 2010; November 30-December 9, 2010; and January 13-20, 2011. The onsite work included detailed studies to support the comprehensive ESIA effort, and included oceanographic, soil, surface and groundwater hydrology, seeps and springs, meteorology and air quality (coastal and mountain monitoring stations), biological resources, reclamation and greenhouse, and local communities.

## 2.4 UNITS AND ABBREVIATIONS

The report considers US Dollars ($) only. Unless otherwise noted, all units are metric units. Salable base metals are described in terms of tonnes 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 Technical Report.

**Table 2-2: Units, Terms and Abbreviations**

| | |
|---|---|
| AATA International, Inc. | AATA |
| Above mean sea level | amsl |
| Acidity | pH |
| AMEC Australia | AMEC Au |
| AMEC USA | AMEC |
| Ammonium nitrate/fuel oil | ANFO |
| Ampere | A |
| Annum (year) | a |
| Benguet Corporation | Benguet |
| Billion pounds | Glb |
| Billion years ago | Ga |
| Billion | G |
| Biotite diorite porphyry | BDP |
| Brinell Hardness Number | BHN |
| Canadian Dam Association | CDA |
| Canadian Institute of Mining | CIM |
| Centimeter | cm |
| Centimeters per second | cm/s |
| Certificate of Ancestral Domain Title | CADT |
| Copper | Cu |
| Crushed Ore Stockpile | COS |
| Cubic centimeter | $cm^3$ |
| Cubic meter | $m^3$ |
| Cubic meters per day | $m^3/d$ |
| Cubic meters per hour | $m^3/hr$ |
| Dacite porphyry | DAP |
| Day | d |
| Days per week | d/wk |
| Days per year (annum) | d/a |
| Dead weight tonnes | DWT |
| Decibel | dB |
| Declaration of Mine Feasibility | DMPF |
| Degree | ° |
| Degrees Celsius | °C |
| Development Rock Stockpile | DRS |
| Dry metric tonne | dmt |
| Echo Bay Mines Ltd. | Echo Bay |
| Electromotive Force | emf |
| Environmental Compliance Certificate | ECC |
| Equivalent (metal grades) | Eq |
| Fisher & Strickler Rock Engineering, LLC | FSRE |
| Foot/feet | ft |
| Gallon | gal |
| Gallons per minute | gpm |
| General & Administration | G&A |
| G-force (seismic) | g |
| Giga (billion) | G |
| Gigajoule | GJ |
| Gold | Au |
| Gram | g |
| Grams per litre | g/L |
| Grams per tonne | g/t |
| Greater than | > |
| Hectare (10,000 $m^2$) | ha |
| Hertz | Hz |
| Horsepower | hp |
| Hour | h |
| Hours per day | h/d |
| Hours per week | h/wk |
| Hours per year | h/a |
| Independent Mining Consultants | IMC |
| Indigenous Peoples Rights Act | IPRA |
| Internal Rate of Return | IRR |
| Intra-mineral dacite porphyry | IDAP |
| Intra-mineral hornblende diorite porphyry | IHDP |
| Joule | J |
| Joules per kilowatt-hour | J/kWh |
| Kelvin | K |
| Kilborn International, Inc. | Kilborn |
| Kilo (thousand) | k |
| Kilobyte | kB |
| Kilogram | kg |
| Kilograms per cubic meter | $kg/m^3$ |
| Kilograms per hour | kg/h |
| Kilograms per year | kg/a |
| Kilojoule | kJ |
| Kilometer | km |
| Kilometer | km |
| Kilometers per hour | km/h |
| Kilonewton | kN |
| Kilopascal gauge | kPa(g) |
| Kilopascal | kPa |
| Kilotonnes | Ktonnes |
| Kilovolt | kV |
| Kilovolt ampere | kVA |
| Kilowatt | kW |
| Kilowatt hour | kWh |
| Kilowatt hours per tonne | kWh/t |
| Kilowatt hours per year | kWh/a |
| Kingking 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 and 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) | µm |
| Microsiemen (electrical) | µs |
| 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 tonnes | Mt |
| Million Years Ago | Ma |
| Million | M |
| Mineral Production Sharing Agreement | MPSA |
| Mines and Geosciences Bureau | MGB |
| 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 and Electrowinning | SX-EW |
| Specific gravity | SG |
| Square centimeter | $cm^2$ |
| Square kilometer | $km^2$ |
| Square meter | $m^2$ |
| St. Augustine Gold & Copper, Ltd. | SAGC |
| Tailing Storage Facility | TSF |
| Thousand tonnes | kt |
| Tonne (metric, 1,000 kg = 2,205 lb) | t |
| Tonnes per cubic meter | $t/m^3$ |
| Tonnes per day | tpd |
| Tonnes per hour | tph |
| Tonnes 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 tonne | 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 Technical Report that are dependent on information provided by Qualified Persons outside the employment of M3 are assumed to be current, accurate and complete as of the effective date of this Technical Report.

M3 relied on other consultants for various parts of the Kingking Project who are 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 Technical 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 TECHNICAL REPORT

This Technical Report is prepared for SAGC ("Client") and 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 Technical 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 Technical Report and that the report complies with published industry standards for such reports.

The recommendations and opinions contained in this Technical Report assume that unknown, unforeseeable or unavoidable events, which may adversely affect the cost, progress, scheduling or ultimate success of the Kingking Project, will not occur. Except as may be expressly stated in writing in the Agreement, the use of this Technical 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 Kingking Project is located on the eastern side of the Gulf of Davao, approximately 92 kilometers (km) from Davao City accessible through the Tagum-Mati National Road and 18 km northeast from Pantukan Town Proper via the Buko-Buko sa Anay – Lawaan dirt road at 7°11'31"N latitude and 125°58'24"E longitude.

The project site is mainly located at Sitio Gumayan, Barangay Kingking, Municipality of Pantukan, Province of Davao de Oro, in Mindanao. While Figure 4-2 shows the Tenement boundaries encompassing the Barangay boundaries only the auxiliary facilities will be situated in other Barangays and will be discussed further in the general arrangement plan. Figure 4-1 shows the project site location.

The Kingking mineral property area falls under Mineral Production Sharing Agreement No. 009-92-XI, Amended II. It covers Two Thousand Nine Hundred Seventy-Six and 71/100 (2,976.071) hectares, located in Sitio Lahi, Barangay Magnaga, and Sitio Gumayan of Barangay Kingking, all within the Municipality of Pantukan, Province of Davao de Oro, on Mindanao Island. The tenement boundaries are defined by two parcels, summarized in Figure 4-2 below. The Kingking tenement, along with the adjacent claims, is also shown in Figure 4-3.

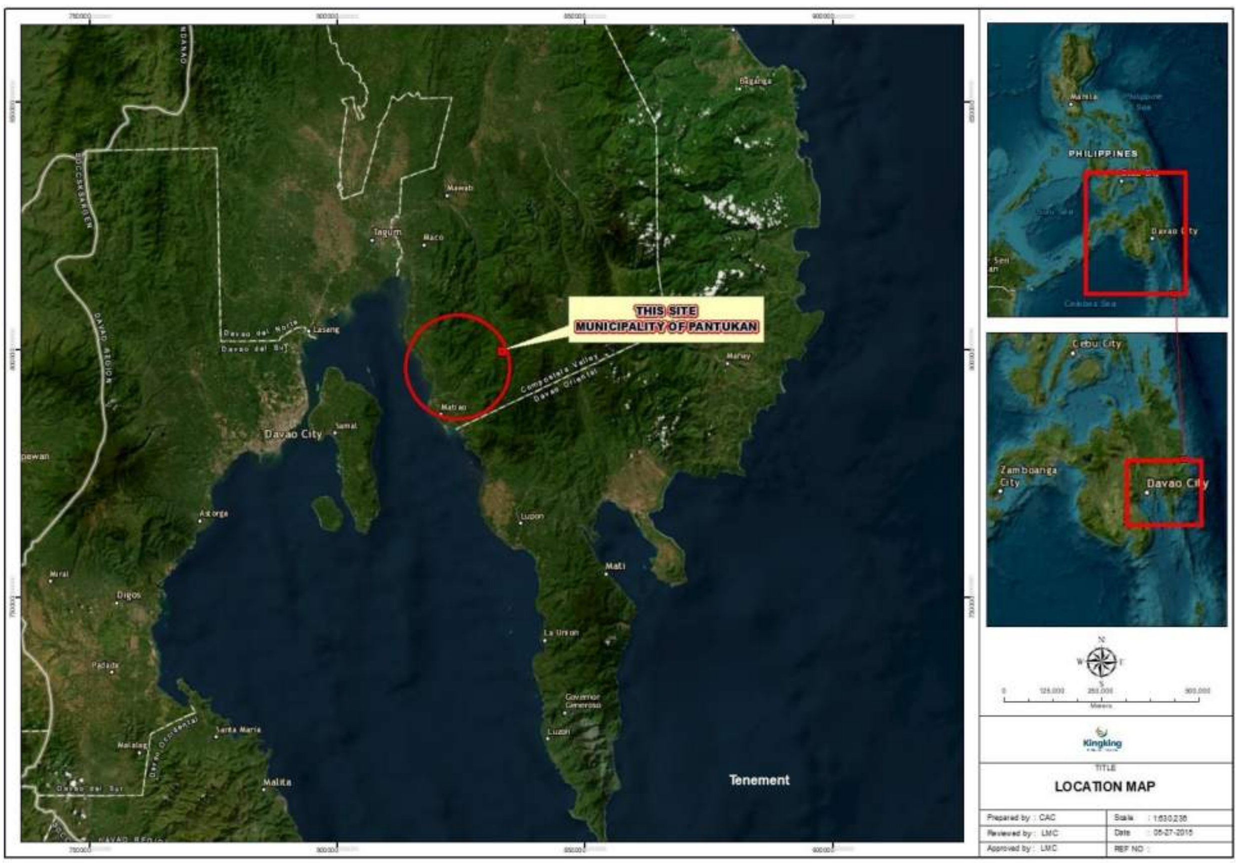



Figure 4-1: Project Location

## 4.2 HISTORY AND LOCATION OF THE MINERAL PRODUCTION SHARING AGREEMENT (MPSA)

### 4.2.1 History of the MPSA

The Kingking Project is covered by Mineral Production Sharing Agreement (MPSA) No. 009-92-XI Amended II approved on May 27, 1992, with an area of two thousand nine hundred seventy-six and 71/100 (2,976.071) hectares.

MPSA No. 009-92-XI (the "MPSA") was approved by the President of the Philippines on May 27, 1992, in favor of Nationwide Development Corporation (NADECOR) as Leaseholder and Benguet Corporation (BC) as Operator.

The Mines and Geosciences Bureau (MGB) approved the amendment of the MPSA on December 11, 2002, to conform to the provisions of Republic Act No. 7942 (Philippine Mining Act) and was renamed to MPSA No. 009-92-IX AMD (the "First Amended MPSA").

BC assigned its rights under the MPSA in favor of NADECOR on October 22, 2010.

Kingking Mining Corporation (KMC or the "Company") is a mining company registered with the Philippine Securities and Exchange Commission (SEC) on October 30, 2013, to engage in and carry on the business of mining exploration, feasibility, development, utilization, extraction of ferrous and non-ferrous mineral ores and quarrying of a mining area; to deal in, purchase, lease, option, locate, apply for, or otherwise acquire, own, exchange, sell, lease, or otherwise dispose of mines, mining claims, mining rights, mining applications, timber rights, water rights, oil, coal and gas rights, all to the extent permitted by law.

Thereafter, on November 25, 2013, NADECOR assigned its rights, title and interest over the MPSA in favor of KMC which assumed all obligations and responsibilities thereto.

The MPSA was further amended on May 23, 2016, to renew its term for another twenty-five (25) years to expire on May 22, 2041.

On June 2, 2016, the Department of Environment and Natural Resources approved the expansion of the area covered by the MPSA by annexing Parcels III and IV the 1,767.1641 hectare portion of the area covered by NADECOR's mining application denominated as Application for Mineral Production Sharing Agreement (APSA) No. 000026-IX which was renamed to MPSA No. 009-92-XI Amended II.

The MGB approved the assignment of the MPSA by BC in favor of NADECOR, and subsequently the assignment by NADECOR in favor of KMC on June 27, 2016.

### 4.2.2 Location of the MPSA

Per the MPSA, the Kingking Project is located at Sitio Lumangang, Municipality of Pantukan, Province of Compostela Valley.

Republic Act No. 11297 renamed the Province of Compostela Valley to Davao de Oro. Consequently, the Kingking Project is now located at Barangay Kingking, Barangay Araibo and Sumlog, Municipality of Pantukan, and Barangay Anitapan, Municipality of Mabini, Province of Davao de Oro.

The tenement boundaries of Kingking Project are defined by 2 Parcels, which are summarized in Figure 4-2 below.

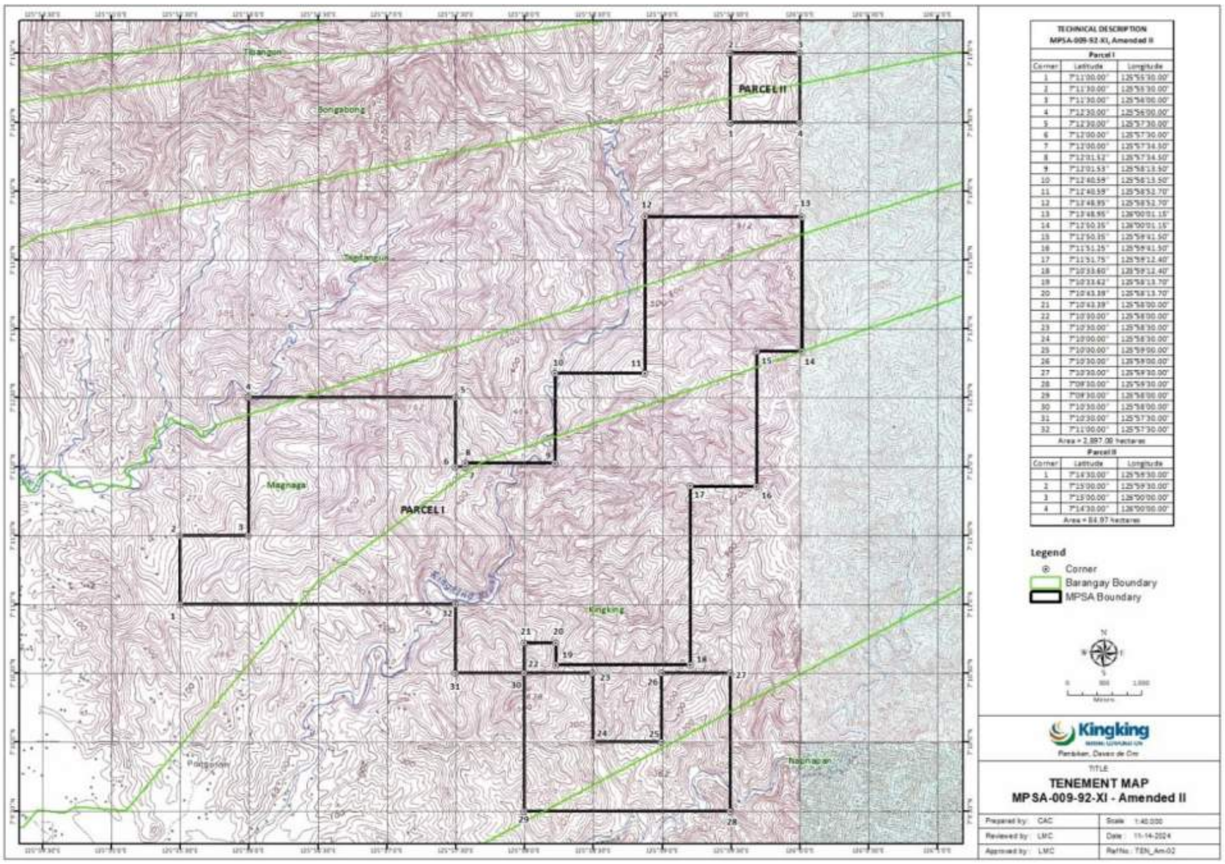



**Figure 4-2: Geographic References for MPSA No. 009-92 XI Amended II Boundaries showing the tenement boundaries encompassing the Barangay**

Moreover, the Kingking Project's tenements with the adjacent claims are shown in Figure 4-3.

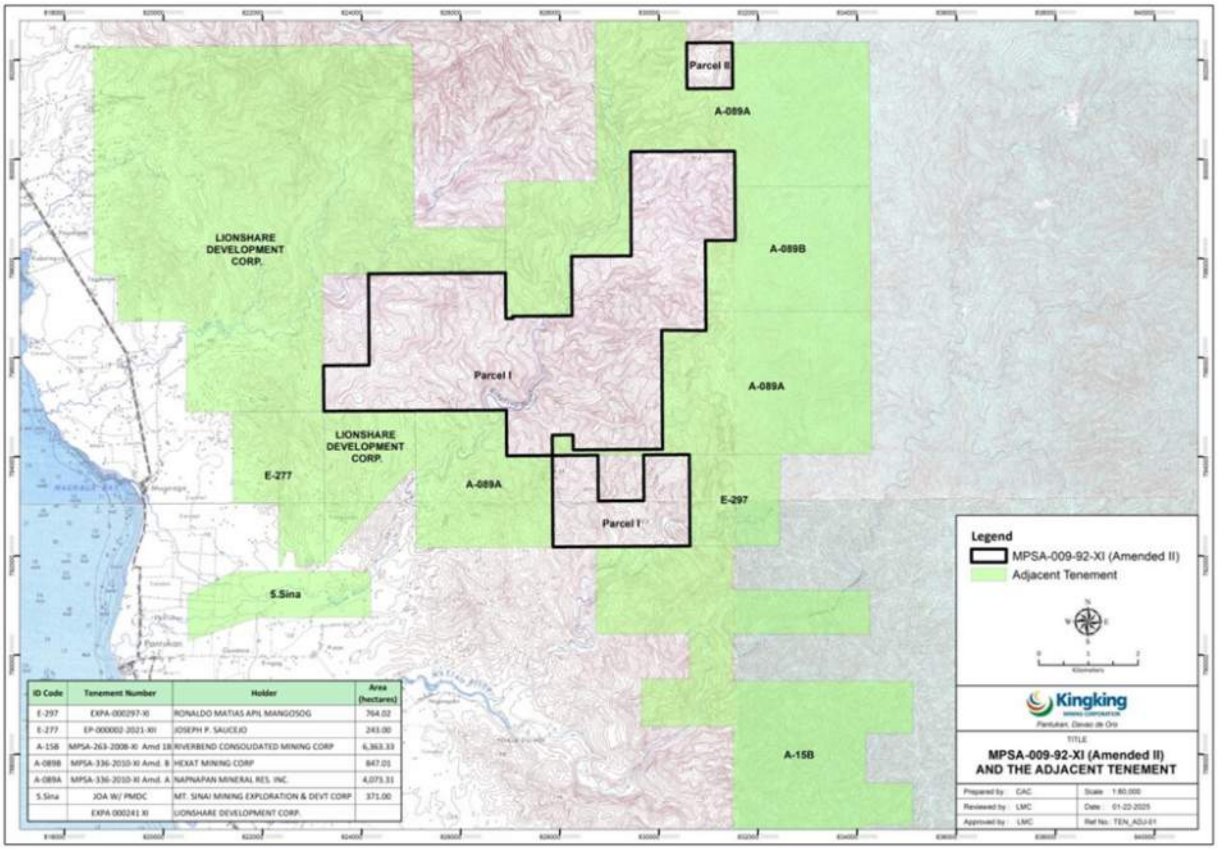



**Figure 4-3: MPSA and Adjacent Tenements**

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

| Parcel 1 | | |
|---|---|---|
| | Latitude | Longitude |
| 1 | 7°11'00.00" | 125°55'30.00" |
| 2 | 7°11'30.00" | 125°55'30.00" |
| 3 | 7°11'30.00" | 125°56'00.00" |
| 4 | 7°12'30.00" | 125°56'00.00" |
| 5 | 7°12'30.00" | 125°57'30.00" |
| 6 | 7°12'00.00" | 125°57'30.00" |
| 7 | 7°12'00.00" | 125°57'34.50" |
| 8 | 7°12'01.52" | 125°57'34.50" |
| 9 | 7°12'01.53" | 125°58'13.50" |
| 10 | 7°12'40.59" | 125°58'13.50" |
| 11 | 7°12'40.59" | 125°58'52.70" |
| 12 | 7°13'48.95" | 125°58'52.70" |
| 13 | 7°13'48.95" | 125°0'01.15" |
| 14 | 7°12'50.35" | 126°0'01.15" |
| 15 | 7°12'50.35" | 126°59'41.50" |
| 16 | 7°11'51.75" | 125°59'41.50" |
| 17 | 7°11'51.75" | 125°59'12.40" |
| 18 | 7°10'33.60" | 125°59'12.40" |
| 19 | 7°10'33.62" | 125°58'13.70" |
| 20 | 7°10'43.39" | 125°58'13.70" |
| 21 | 7°10'43.39" | 125°58'00.00" |
| 22 | 7°10'30.00" | 125°58'00.00" |
| 23 | 7°10'30.00" | 125°58'30.00" |
| 24 | 7°10'00.00" | 125°58'30.00" |
| 25 | 7°10'00.00" | 125°59'00.00" |
| 26 | 7°10'30.00" | 125°59'00.00" |
| 27 | 7°10'30.00" | 125°59'30.00" |
| 28 | 7°9'30.00" | 125°59'30.00" |
| 29 | 7°9'30.00" | 125°58'00.00" |
| 30 | 7°10'30.00" | 125°58'00" |
| 31 | 7°10'30.00" | 125°57'30.00" |
| 32 | 7°11'00.00" | 125°57'30.00" |
| Parcel 2 | | |
| 1 | 7°14'30.00" | 125°59'30.00" |
| 2 | 7°15'00.00" | 125°59'30.00" |
| 3 | 7°15'00.00" | 126°00'00.00" |
| 4 | 7°14'30.00" | 126°00'00.00" |

## 4.3 MPSA TITLE AND SAGC INTEREST

St. Augustine Gold & Copper, Ltd. (SAGC) and its affiliates entered into agreements with NADECOR for development of the Kingking Project as follows:

- In April of 2010, a Memorandum of Understanding between NADECOR and St. Augustine Mining Ltd. (SAML), a subsidiary of SAGC, was signed and 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 Kingking Project in exchange for certain payments, investments, and other deliverables.
- In June 2011, NADECOR and SAGC signed a Technical Services Agreement, and Onshore and Offshore Services Agreements. These agreements allowed wholly owned technical service companies of SAGC (MDCA and SASI) to provide technical services to the Kingking Project including with respect to the feasibility and permitting studies.
- In August 2011, an Interim Funding Agreement was signed between NADECOR and SAGC, setting out the detailed terms of SAGC's investments in the Kingking Project and the protection of said investment. At the same time, NADECOR formed several companies intended to be the joint venture companies for the Kingking Project. The joint venture structure considered Philippine legal requirements (including applicable nationality restrictions) and provided for a legally compliant mechanism under which SAGC can participate in the management and ownership of the Kingking Project. In addition, SAGC and NADECOR then planned to assign the MPSA to one of the joint venture companies mentioned above.
- Thereafter, the law firm of Sycip, Salazar, Hernandez & Gatmaitan (Makati City) reviewed the status of the MPSA, Memorandum of Understanding, Technical Services Agreement, Onshore and Offshore Services Agreement, Interim Funding Agreement and Preferred Shares Investment Agreement between the parties. The Sycip law firm then issued a legal opinion which confirmed that if the MPSA is transferred to one of the joint venture companies, SAGC will have an interest in the MPSA. The legal opinion also verified the agreements stated above.
- From 2016, SAGC acquired an additional 1,320 hectares of mining claims which were adjacent or very near the original mining claims granted to KMC. At the end of 2024, the current inventory of mining claims under the KMC MPSA stood at 2,976.071 hectares.
- On June 2, 2016, the DENR through the MGB approved the expansion of the area of the MPSA by annexing parcels III and IV with an area of 1,761.16 hectares of NADECOR's mining application denominated as APSA No. 000026-XI, and the MPSA was renominated as MPSA No. 009-92-XI Amended II.
- On June 28, 2016, the Mines and Geosciences Bureau (MGB), the division in charge of mining under the Department of Environment & Natural Resources (DENR), approved the assignment of the MPSA to Kingking Mining Corp. (KMC).

### Permits and Licenses

- On December 3, 2014, the Declaration for Mining Project Feasibility (DMPF) for the extraction and commercial disposition of copper, gold, silver and other associated minerals in the contract area of the amended MPSA was filed by NADECOR in MGB.
- On February 26, 2015, the Kingking Project has received approval of NADECOR's Environmental Impact Statement (EIS) from the Philippine Environmental Management Bureau (EMB). In connection with the EIS approval, the EMB has issued the Environmental Compliance Certificate (ECC). With the issuance of the ECC, the proponent is expected to fully implement the measures presented in the EIS which intended to protect and mitigate the Kingking Project's adverse impacts on community health, welfare, and the environment.
- On April 15, 2015, DENR Administrative Order (DAO) No. 2015-07 required all mining companies to secure Environmental Management System (EMS) or ISO 14001:2015 accreditation.

- On February 19, 2016, the Certificate Precondition (CP) was secured from the National Commission on Indigenous People (NCIP); one of the major requirements for the extension of the amended MPSA.
- On May 16, 2016, the MGB approved the Kingking Project's Amended DMPF; therefore, authorizing the proponent to proceed with the development, construction and operation of the Kingking Project by approving plans and strategies.
- On May 23, 2016, the DENR approved the renewal of the amended MPSA for another 25 years term, i.e. from May 23, 2016 to 2041.
- On June 27, 2016, the MGB ordered the approval of the assignment of the amended MPSA from NADECOR to KMC. Consequently, the amended MPSA No. 009-92-XI is now recorded under KMC.

## 4.4 CERTIFICATE OF ANCESTRAL DOMAIN TITLE (CADT)

Republic Act 8371 or the Indigenous Peoples Rights Act (IPRA) provides for the recognition of the ancestral domain and lands of Indigenous People. Areas that are considered as ancestral domains are covered by Certificate of Ancestral Domain Title (CADT).

The MPSA tenement area and all project facilities for the Kingking Project are within the area of the ancestral domain of the Mansaka tribe of the Municipality of Pantukan. The ancestral domain is covered by CADT No. R11-PAN-0908-076) issued by the National Commission on Indigenous People (NCIP) on September 2, 2008, with a total area of approximately 141,773.3097 hectares.

With the exception of alienable and disposable (A&D) lands (Alienable and Disposable), 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 facilities areas are covered by the above CADT.

The CADT grants the Mansaka tribe the legal right over the use of their ancestral land and the authority to allow and consent to the operation of certain economic activities like large-scale mining, provided said economic activities align with their Ancestral Domain Sustainable Development and Protection Plan (ADSDPP). The ADSDPP is a mandatory plan under IPRA. The ADSDPP includes guidelines for negotiating mining and other resource-related ventures, which the Pantukan Federation of Mansaka Tribal Council oversees, along with measures to ensure environmental and cultural sustainability in these projects.

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

The NCIP issued Certification Precondition (Control No. CRXI-16-01-76) on January 14, 2016 which certified the compliance of NADECOR with the requirements for the issuance of the Certification Precondition and Free and Prior Informed Consent for the renewal of the MPSA. The Certification Precondition further certified that NCIP En Banc through Resolution No. 06-03-2016, Series of 2016 approved the issuance of the Certification Precondition for the Kingking Project covered by the MPSA.

## 4.5 LAND TITLES

Project facilities areas encumber approximately 300 parcels of land, most of which are covered by OCTs or TCTs. An Original Certificate Title (OCT) or Transfer Certificate of Title (TCT) evidences the holder's ownership over the land area covered by said document. Project facilities 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 Kingking Project.

## 4.6 LAND ACQUISITION

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

Approximately 300 parcels of land outside of the tenement must be purchased for the Kingking Project. A detailed land acquisition strategy has been developed for the Kingking 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 (now Davao de Oro) 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 COMMITMENTS

Environmental commitments in relation to the Kingking Project 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 the Kingking 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 Kingking Project.

Aside from the mitigating measures enumerated in the ECC, the Kingking 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.

On February 26, 2015, the Kingking Project received approval of the NADECOR's Environmental Impact Statement (EIS) from the Philippine Environmental Management Bureau (EMB). In connection with the EIS approval, the EMB has issued the Environmental Compliance Certificate (ECC). With the issuance of the ECC, the proponent is expected to fully implement the measures presented in the EIS which intended to protect and mitigate the Project's adverse impacts on community health, welfare and the environment.

The EMB approved the transfer of ownership/grantee of the ECC from NADECOR to KMC on October 5, 2021.

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 on May 27, 1992, (Effective Date), and was amended on December 11, 2002 and May 23, 2016. 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. On May 23, 2016, the MPSA term was renewed for another 25 years or until May 22, 2041. It has been a practice in the Philippine Mining Industry to extend the MPSA term.

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 the Kingking Project 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-science. NADECOR/SAGC have 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.

In 2015, NADECOR applied for the renewal of the MPSA for another twenty-five years and was approved by the DENR Secretary through the MGB Director on May 23, 2016, and was renamed as MPSA No. 009-92-XI (Amended II) which includes an additional 1,320 hectares area thus covering a total of Two Thousand Nine Hundred Seventy-Six and 71/100 (2,976.071) hectares.

Thereafter, the Deed of Assignment (DOA) between NADECOR and Kingking Mining Corporation (KMC) was approved on June 27, 2016, which transferred the right to develop the mining property and MPSA No. 009-92-XI Amended II from NADECOR to KMC.

The MGB Region XI issued a Certification dated March 22, 2021 stating that Kingking Mining Corporation (KMC) is a holder of a Mineral Production Sharing Agreement No. 009-92-XI (Amended II) and that KMC is generally compliant with the terms and conditions of MPSA.

## 4.9 PROJECT AFFECTED PEOPLE

Tenured and non-tenured (i.e. informal) people and households will be affected by the Kingking Project.

Initial estimates of potential project-affected people (PAP) and households peg the number of PAP within facilities footprint areas at 7,861 individuals and 1,642 households. A buffer zone of one kilometer from facility boundaries was also included to expand PAP estimates: these 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 – both within the footprint of project facilities and those living within the one-kilometer buffer zone. This buffer zone is for planning purposes only as the mandated buffer zone is 50 to 150 meters for mining facilities. 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 levels (the data gathered from this census was generated for the Kingking Project).

Table 4-2 summarizes the estimated number of PAP and PAP households for each of the Kingking 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 Dry Stack | 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 Kingking 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 facilities areas will need to be reclassified as heavy industrial, as the Municipality of Pantukan has not allocated any land for this 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 Kingking 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 Kingking 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 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 facilities 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 Kingking 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, but 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 showstoppers 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.

At this stage, it is also important to identify the mechanism for consolidating all properties secured by the Company. This is considering 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 Davao de Oro (previously Compostela Valley). Pantukan is approximately 92 km by road from Davao City via the paved Tagum City–Mati National Road. From Pantukan town proper, the Kingking 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. This road will be maintained as a secondary access. 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.

### 5.1.2 Air Access

The nearest commercial airport is Francisco Bangoy International Airport (IATA: DVO, ICAO: RPMD) located in Davao city. The airport has 65,000 sq meter terminal and a 9,800 ft runway capable of supporting all widebody aircraft including the Boeing 747 and the Airbus A380. Daily flights are available to all Philippine major destinations and some international destinations. Air freight services are available.

### 5.1.3 Sea Access

The Kingking Project has acquired property and a foreshore lease for the development of a port facility and pier capable of supporting the mine operations including import of all consumable materials, sulfuric acid, and fuels and export of concentrates (see Section 18.4). The feasibility design work was accomplished by Halcrow (a CH2MHill Company) and updated by M3. In addition, the Port of Davao, consisting of several ports within the Davao Gulf, primarily the Sasa Wharf, can support the operation with 6.4 m and better depths for cargo and bulk carriers.

## 5.2 CLIMATE

The climate at the project site is humid tropical and is considered a Coronas Classification Type 2. The Kingking 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 1961 to 1994 and 2002 to 2010. Additionally, data from Tagum City (approximately 29 km north of the mine site) were reviewed for the period 2020-2025. In evaluation of the earlier climate data, Dames & Moore (1997) applied a 38 percent 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.

Daytime temperatures range from 18°C to 35°C with an average ambient temperature of 27°C.

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

There are no climatic conditions that should cause the Kingking Project great operational difficulty. The greatest climatic issue will be managing storm waters that will result from excessive rainfall at intermittent times during the life of the

Kingking Project. However, this is a common operating issue at many tropical mine sites and will be manageable with proper stormwater management controls and planning.

## 5.3 LOCAL RESOURCES

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 2020 population of communities near the Kingking Project were as follows:

| | |
|---|---|
| Pantukan Municipality | 90,786 |
| Magnaga | 10,766 |
| Napnapan | 12,529 |
| Kingking | 29,317 |
| Davao City | 1,776,949 |

## 5.4 INFRASTRUCTURE

Some of the basic infrastructure is in-place for exploration and development of the Kingking deposit. A paved highway from Davao City runs 10 kilometers southwest of the Kingking 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, and port facility 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 drill holes 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.

Currently, there is a surplus in available power in Mindanao and the assumption is grid-supplied power will be sufficient for the operation of the Kingking 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.

A permanent core house is now in place in Pantukan. 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 Kingking 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. Figure 5-2 provides the facilities locations.

The topography in the immediate project area is steep and rugged with elevations ranging from 250-950 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.

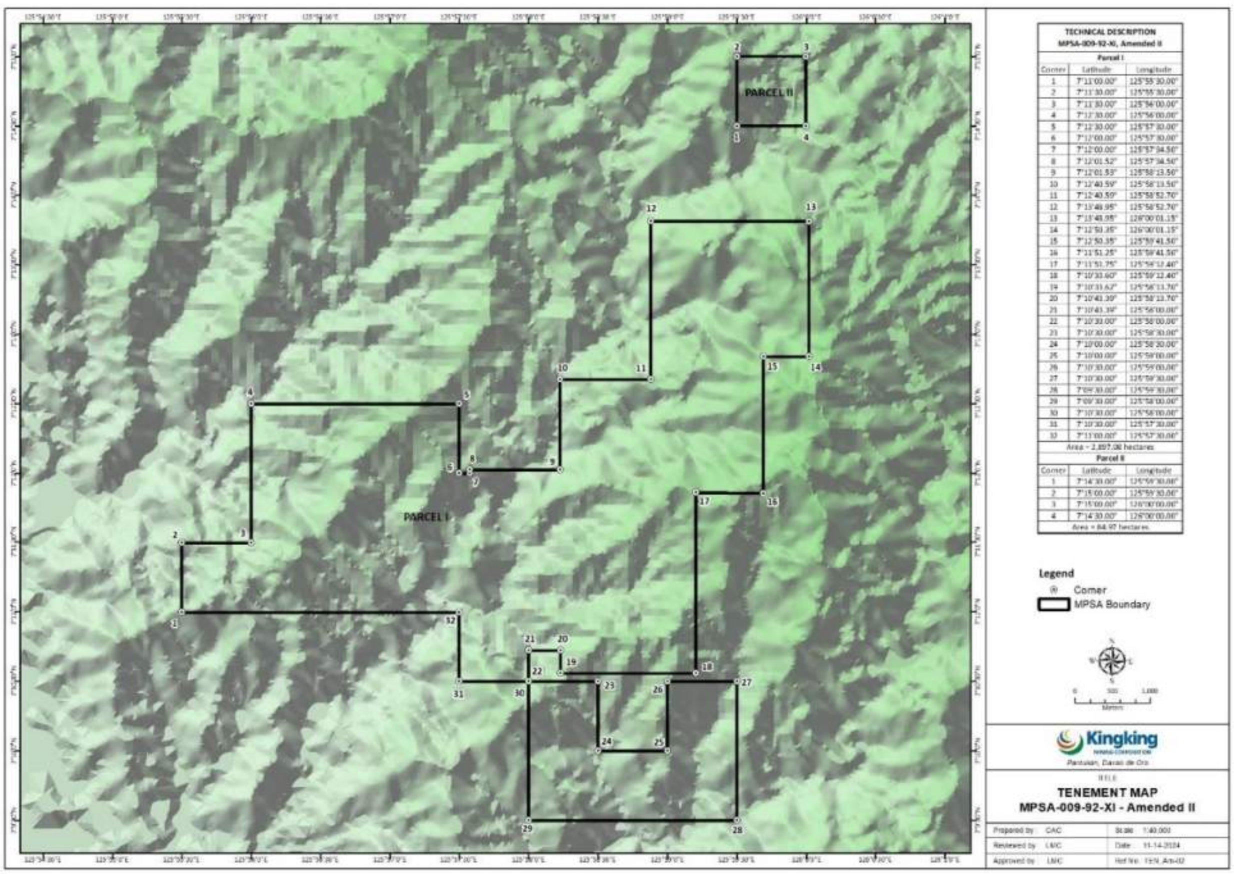



**Figure 5-1: Tenement Map Topography**

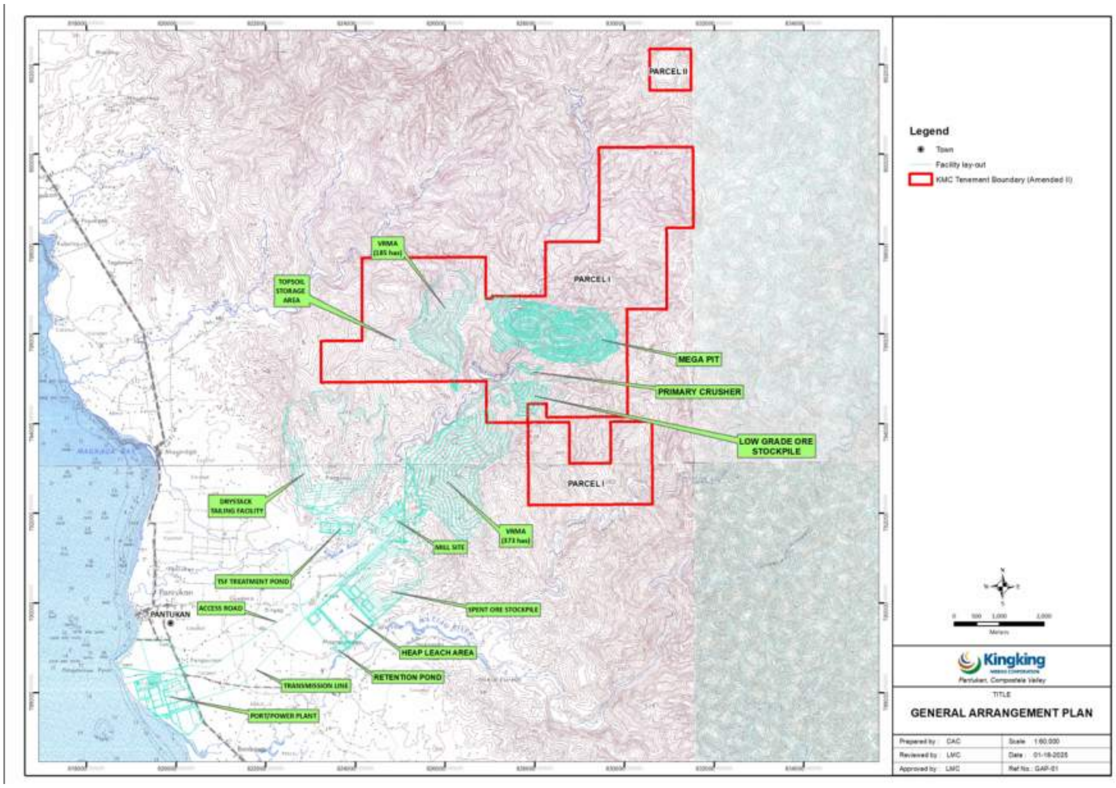



**Figure 5-2: Facilities Locations**

## 5.6 MINING SURFACE RIGHTS AND MINING PERSONNEL

On June 2, 2016, the DENR through the MGB approved the expansion of the area of the MPSA by annexing parcels III and IV with an area of 1,761.16 hectares of NADECOR's mining application denominated as APSA No. 000026-XI, and the MPSA was renominated as MPSA No. 009-92-XI Amended II.

On June 28, 2016, the Mines and Geosciences Bureau (MGB), the division in charge of mining under the Department of Environment & Natural Resources (DENR), approved the assignment of the MPSA to Kingking Mining Corp. (KMC). Moreover, in the same year, the DENR through the MGB approved the renewal of the term of MPSA for another 25 years – valid until 2041.

Under the Philippine Mining Act of 1995, KMC, as a holder of mining rights by virtue of the MPSA No. 009-92-XI Amended II, 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 Amended II. 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 KMC in accordance with applicable regulations, and to guarantee such compensation, KMC 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 (now Davao de Oro), from the island of Mindanao, from the Philippines and from outside of the Philippines. The municipality of Pantukan is home to approximately 90,786 people and approximately 22% are underemployed in this area and in the province. There is a large craft trained work force to draw from in the Davao area. The population of Davao is 1.7 million people. Thus, there is a sizable work force to draw from near the mine site.

# 6 HISTORY

The Kingking 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.

Prior to SAGC's tenure, exploration of the Kingking deposit was conducted intermittently by previous project owner/operators beginning in the late 1960s and continuing until 1997. NADECOR discovered the Kingking mineralization anomaly in 1966 through 1968. From 1969 to 1972, Mitsubishi Mining Corporation undertook initial exploration of the mineral deposit, completing 54 surface diamond drill holes 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 exploration program are known to exist.

Benguet Corporation (Benguet) signed an Operating Agreement with NADECOR on August 21, 1981, for the exploration and development of the Kingking property. However, litigation regarding ownership did not allow any activity within the Kingking Project. In 1988, all legal issues were resolved in favor of NADECOR's rights over the mineral claims and in the same year the appointment of Benguet as Operator was re-ratified by the shareholders of NADECOR.

From 1991 until 1994, Benguet completed 69 diamond core holes (19,247 m), 25 reverse circulation holes (4,926 m), 326 m of confirmatory audits 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 Kingking Mines Inc. (KMI), TVI Pacific (TVI) and Echo Bay Mines, Inc. (Echo Bay), entered into an option agreement with Benguet and NADECOR to develop the Kingking Project. Per the Option Agreement, a non-refundable option amounting to US$10 Million was given to Benguet. Per the Option Agreement, Echo Bay, TVI and KMI would conduct confirmatory exploration work on the properties, and prepare, within 24 months from the signing of the Option Agreement, a bankable feasibility study which would serve as the basis on deciding whether to exercise their option. If Echo Bay, TVI, and KMI exercise the option, an additional payment of US$67 Million would be made to Benguet. A further payment of US$18 Million will also paid to Benguet should the proven and probable gold reserves, as shown in the feasibility study, exceed the reserve indicated in the Benguet's prefeasibility study. Benguet had the option to acquire 20% of KMI at the price of US$20 to retain a 20% interest in the Kingking Project at the time Echo Bay, TVI, and KMI would exercise their option to acquire the Kingking Project. On October 24, 1997, Echo Bay, TVI, and KMI formally informed the Company that they will not exercise their option to acquire the Kingking 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 Kingking Project, the scope of which was based on several specific items and on Kilborn's interpretation of Echo Bay's generic requirements for what was referred by Echo Bay to be a Level I 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 Kingking 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 tailings 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 tonnes containing 0.306% Cu and 0.41 grams Au per tonne for the Kingking deposit. This resource included a "mineable reserve" of 403 million tonnes at 0.332% Cu and 0.488g/t Au. The authors of this Technical 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 Technical Report emphasize, is not compliant with current NI 43-101 guidelines, totaled 749 million tonnes containing 0.387% Cu and 0.433g/t Au.

All Echo Bay data was acquired by Kinross Gold (Kinross) through its merger with Echo Bay in 2002. Kinross subsequently waived its option to proceed with the Kingking 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 Kingking 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 Kingking 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 Kingking resource in October 2010. The resource contained measured and indicated mineralization of 792 million tonnes 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 tonnes 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 Kingking Project to Ratel Gold Limited and took over management of Ratel and changed its name to St. Augustine Gold and Copper, Ltd. (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 Kingking 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 Kingking 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 Kingking 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 Kingking Project.

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 led to an important agreement with the indigenous people that 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 feasibility and permitting studies started in 2011 continued into 2012 and three important large multi-volume reports were completed during the year. Environmental and social studies 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 and social studies 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 report (PFS) 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 the 2013 NI 43-101 Preliminary Feasibility Technical Report which was issued on October 28, 2013.

In 2013, Queensberry Mining & Development Corp. (Queensberry), a company owned by Mr. Paolo Villar, invested in SAGC to provide a stronger local proponent to the Kingking Project. Queensberry and Russell Mining Minerals Inc. (RMMI but now Russell Mining and Minerals, ULC) Groups combined to control 66.29% of SAGC and moved to try to raise funding for the Kingking Project in 2014 and 2015 but copper prices fell below US$3.00 per pound and Gold fell below US$1,200 per ounce in 2015, creating a difficult environment to fund the Kingking Project, especially in the light of the required financial returns and financial structures of prospective investors during that period. Late in 2015, the owners of SAGC and NADECOR agreed to transfer the MPSA into a JV Mining Company to help with the required structure for the funding requirements of the Kingking Project.

SAGC continued as per the agreements to complete project permitting. On December 29, 2015, the MGB approved the Declaration of Mining Project Feasibility of the Kingking Project, and the Environmental Compliance Certification was issued.

From 2016, SAGC acquired an additional 1,320 hectares of mining claims which were adjacent or very near the original mining claims granted to KMC. At the end of 2024, the current inventory of mining claims under the KMC MPSA stood at 2,976 hectares.

On June 2, 2016, the DENR through the MGB approved the expansion of the area of the MPSA by annexing parcels III and IV with an area of 1,761.16 hectares of NADECOR's mining application denominated as APSA No. 000026-XI, and the MPSA was renominated as MPSA No. 009-92-XI Amended II.

On June 28, 2016, the Mines and Geosciences Bureau (MGB), the division in charge of mining under the Department of Environment & Natural Resources (DENR), approved the assignment of the MPSA to Kingking Mining Corp. (KMC).

Moreover, in the same year, the DENR through the MGB approved the renewal of the term of MPSA for another 25 years – valid until 2041.

However, in late 2016, then Philippine President Rodrigo R. Duterte, appointed Regina Lopez as Secretary of the DENR. Lopez was a staunch anti-mining advocate and had been consistently against open-pit mining. On April 27, 2017, Sec. Lopez imposed the open pit mining ban throughout the Philippines and this led to the halt to many mining projects, including development of the Kingking Project. Given the extreme views of Secretary Lopez, Congress refused to confirm her appointment, and she was subsequently replaced. However, her replacement was not able to get the open pit mining ban lifted right away.

While the uncertainty about the lifting of the open pit mining ban existed, SAGC focused on the acquisition of additional mining claims and the maintenance of the original mining claims.

In December 2021, the open pit mining ban was finally lifted, 7 months prior to the end of the Duterte Administration, which had imposed the ban in 2017. While the MPSA for KMC was still in force, the activities to move the Kingking Project could not advance right away as industry players and investors made it clear that starting a large CapEx project required more certainty of the regulatory environment favourability towards the mining industry. With only 6 months remaining in the Duterte administration and an unknown future administration position on mining, the Kingking Project was on hold until clarity could be assured.

In May 2022, Ferdinand R. Marcos Jr. was elected to the Philippine Presidency and by Q4 2022 it was clear that the new Marcos Government was a firm believer of the potential for mining to grow the Philippine Economy faster. Given the huge deficits which the Philippine Government had taken during the COVID years, Government debts had reached 60% of GDP and one of the key strategies to solve the debt issues was to try to grow the economy faster especially with export driven industries. This strategy made mining a priority under the new Marcos Administration. The Kingking Project was listed as a top priority project for the administration.

In early 2023, NADECOR and SAGC evaluated the restart of the Kingking Project and consulted with potential investors, motivated by the pro-mining stance of the Marcos Government and improving prices of both Gold and Copper. The feedback from potential investors and the market turned out to be positive and the owners of both NADECOR and SAGC started to plan and raise funds for a study to re-start the Kingking Project. This meant understanding what the financials of the Kingking Project would be under current market conditions and given the changes in the environment in the last 11 years (from the previous PFS in 2013). In mid-2023, the main shareholders of NADECOR and SAGC decided to take steps to restart the Kingking Project. This mainly meant the updating of the data and the start of consultation with various entities who would help in updating the Prefeasibility Study for the Kingking Project. Independent Mining Consultants (IMC) and M3 were formally engaged to accomplish a revised mine plan and update to the previously completed Pre-Feasibility study published in 2013.

Also in January 2025, KMC started finalizing the key assumptions for the new economic model of the Kingking Project in preparation by M3 for the completion of an Updated Pre-Feasibility Study for the Kingking Project. Consultations with dozens of experts were undertaken to finalize the different parts of the PFS that needed updating. Some of the major changes in the Kingking Project included (but not limited to):

1) Removal of the proposed Power Plant – The Mindanao Grid now has sufficient power supply for the Kingking Project; it will no longer need its own power plant.
2) Provision of Transmission lines from the Grid to the Project Site – The removal of the Project's own power plant will however require a longer transmission facility from Maco, Davao de Oro to Pantukan, Davao de Oro covering approx. 33 kilometers from a primary Maco substation.
3) Assumed Copper and Gold Prices – \$4.30 Cu and \$2,150 Au.

4) Project Resource Base increased due to cut-off grade changes from around 700 million tonnes to just under 1 billion tonnes.
5) The MPSA area substantially increased from 1,656 hectares to 2,976 hectares through the acquisition and consolidation of additional APSA Claims prior to the renewal of the MPSA previously described.

# 7 GEOLOGICAL SETTING AND MINERALIZATION

## 7.1 REGIONAL GEOLOGY

The southeastern Mindanao Peninsula (comprising the mountainous provinces of Davao Oriental, Davao de Oro 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 Kingking 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. 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 Kingking 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 Kingking situated on the upper (over-riding) plate.

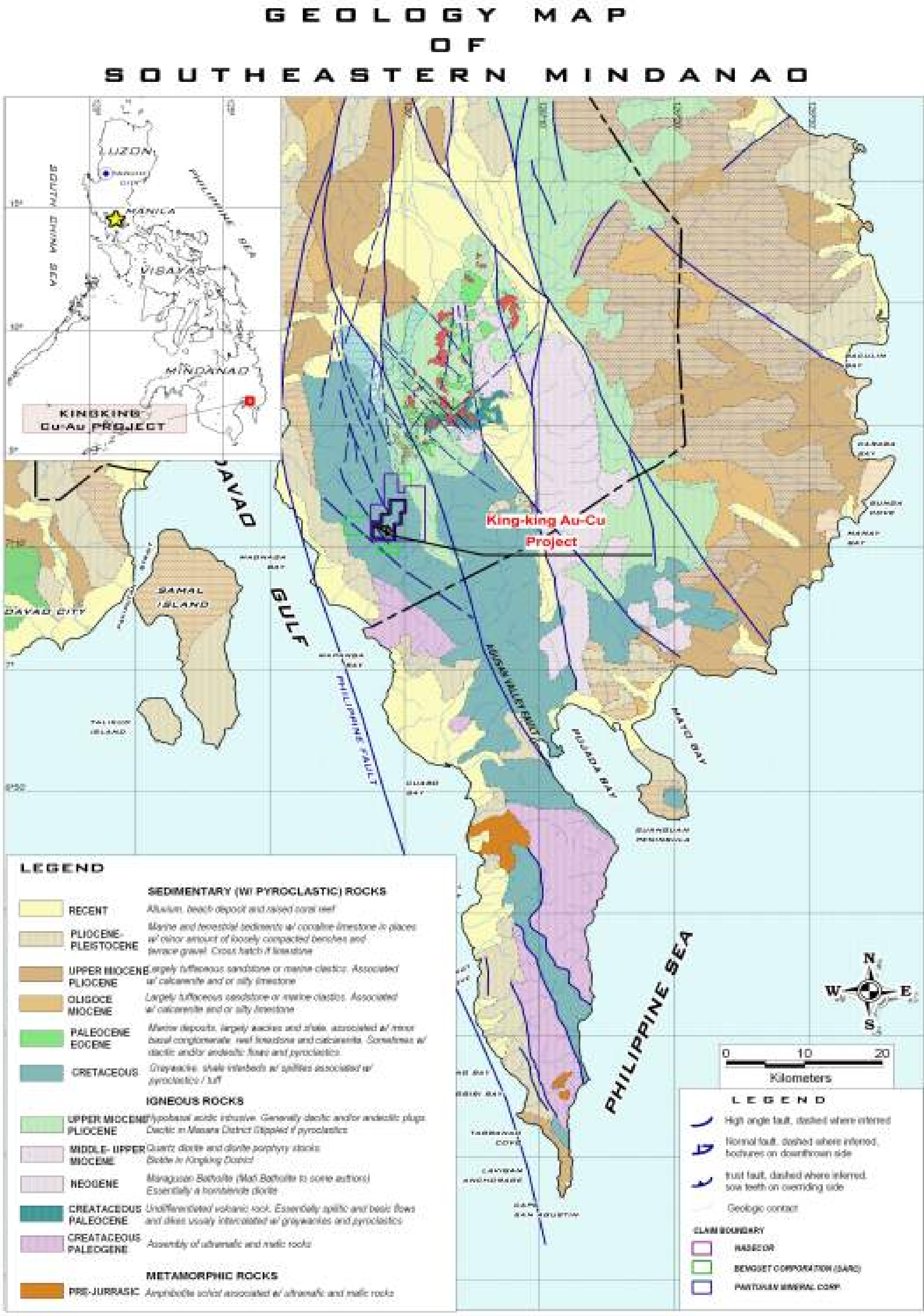



**Figure 7-1: Local Geology**

The Kingking 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 southwest dips. However, local reversals of dip are common, forming minor anticlines and synclines that are evident along roadcuts and gullies south of the main Kingking deposit.

## 7.3 PROPERTY GEOLOGY

### 7.3.1 Lithology

The Kingking 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 average 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 Kingking 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 Kingking 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.

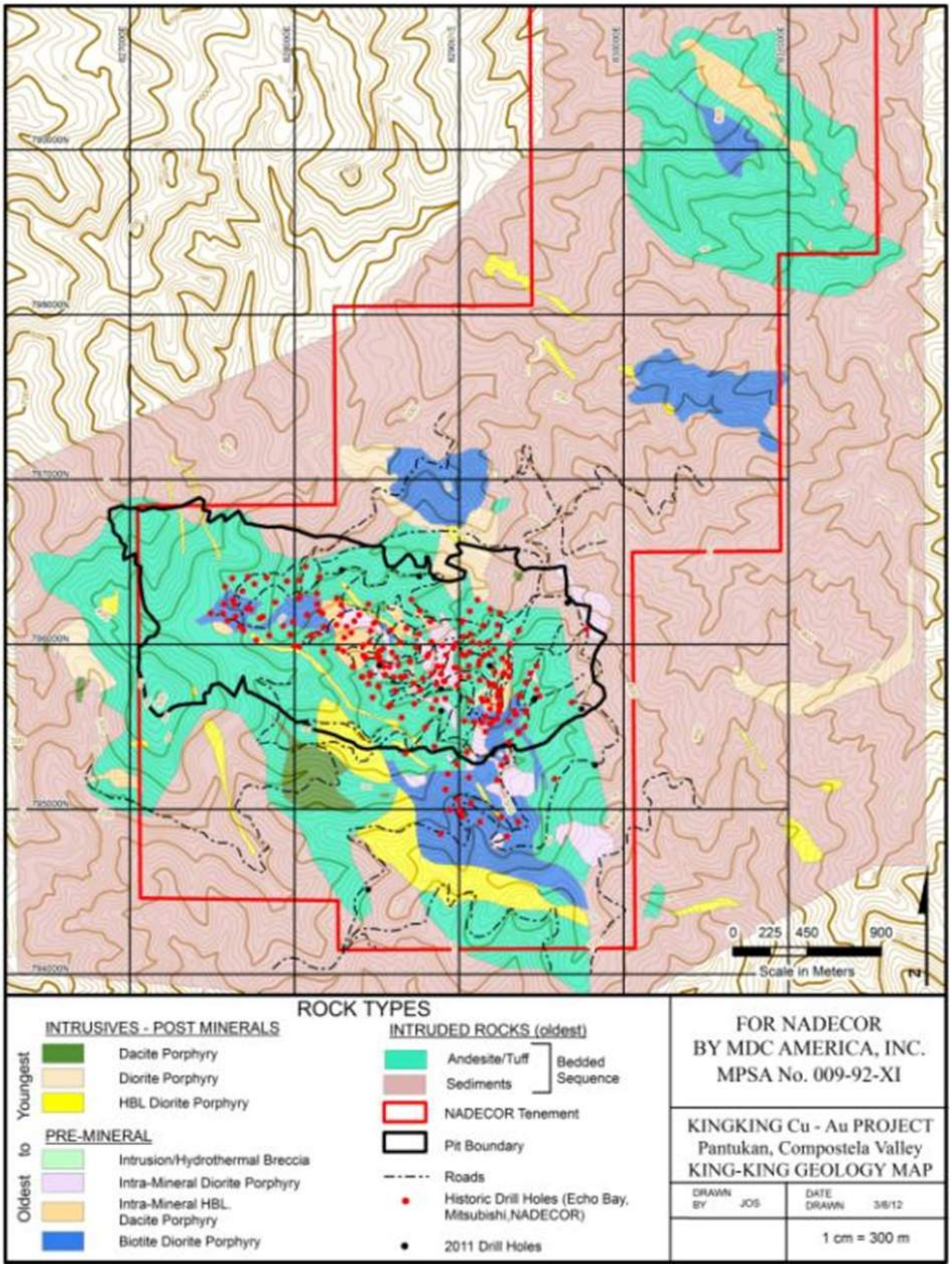



**Figure 7-2: Property Geology Map**

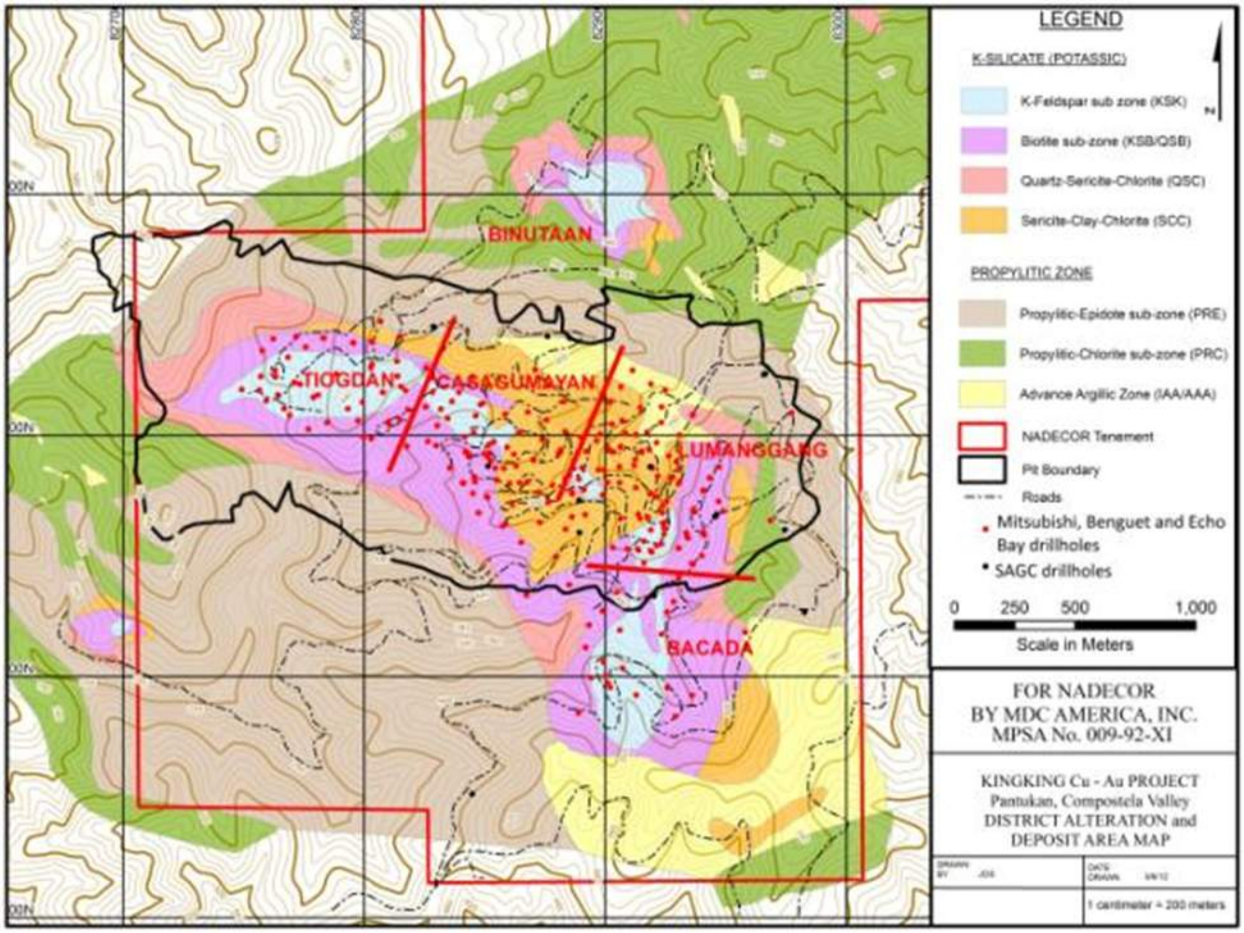



**Figure 7-3: Commonly Reference Deposit Areas**

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 Kingking deposit in the Lahi, Barricade, Buko-buko 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, Kingking 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 Kingking 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 Kingking 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 Kingking 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 Kingking is typical of other porphyry copper deposits in the Philippines and in other parts of the world. However, important differences between Kingking 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 Kingking, 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 Kingking 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 Kingking 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 Kingking 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 to those who pioneered gold panning at Kingking, 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 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 Kingking 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 Kingking 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 Kingking 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 Kingking deposit. These assemblages are 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.

There are three factors that suggest the Kingking deposit is somewhat different from other Philippine porphyry copper deposits:

- 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 Kingking 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 Kingking Project (See Section 6 – History). Since assuming control of the Kingking 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 drill holes 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 Kingking pit. The most notable intercepts from these data are from Benguet drill hole DD-1 in the Diat area, and Echo Bay drill holes 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 Kingking pit limit. Summaries of the significant intervals in these holes are included 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 Kingking deposit.

# 10 DRILLING

Kingking property drilling has been conducted by four companies: Mitsubishi, Benguet, Echo Bay, and St. Augustine from 1969 to 2011. The drilling to date consists of 290 holes and represents 95,909 meters of drilling. Table 10-1 shows the drilling by campaign and drilling type (RC = Reverse Circulation). Figure 10-1 shows the drill holes by drilling campaign. It also shows the location of cross sections that will be presented in Section 14, Mineral Resource Estimates.

**Table 10-1: Drilling by Campaign**

| Campaign Description | No. of Holes | Time Period | Meters |
|---|---|---|---|
| Mitsubishi Core Holes | 54 | 1969-1972 | 13,031 |
| Benguet Core Holes | 69 | 1991-1994 | 19,247 |
| Benguet RC Holes | 25 | 1991-1994 | 4,926 |
| Echo Bay Core Holes | 128 | 1995-1997 | 52,718 |
| St. Augustine Core Holes | 14 | 2011 | 5,987 |
| **Total Drilling** | **290** | **1969-2011** | **95,909** |

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 and having a northeast dip.

The core holes, including the St. Augustine holes, were nominally sampled on 3 m down-hole intervals, though a portion of the early Echo Bay holes were sampled on 2 m intervals. The Benguet RC holes were sampled on 1 m intervals.

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

**Table 10-2: Drilling History by Company**

| No. of Holes | Drill hole Type | Company | Time Period | Hole Series |
|---|---|---|---|---|
| 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 | 1995-1997 | EB 1-126 |
| 14 | DDH Holes | St. Augustine | 2011 | SAG, SAGT, SAM, SAH Series |

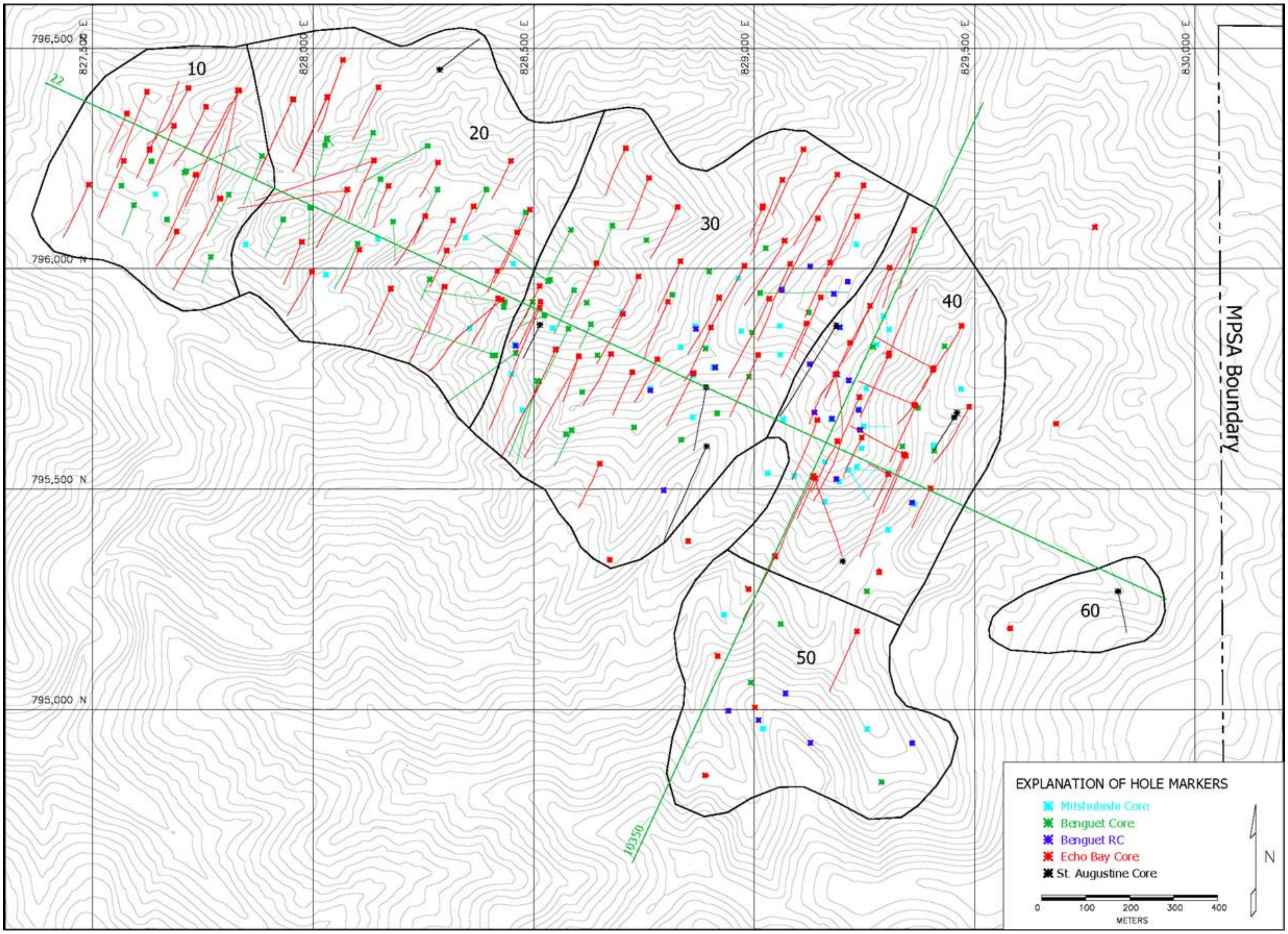



**Figure 10-1: Drill hole Locations by Campaign (IMC, 2023)**

## 10.1 HISTORIC DRILL HOLE COLLAR LOCATION CHECK

During the June 5, 2010 site visit by Don Earnest of Resource Evaluation, Inc. (REI), an attempt was made to locate 21 randomly-selected drill hole collars in the field. Because of dense vegetation overgrowth and sloughing of cutbanks at drill sites, only six drill hole collars were located. Two of the drill holes located contain steel casing with valves and were in use as water wells at that time. These holes were 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 drill holes, an open drill hole collar (no concrete pad) for M31-2R (an RC twin drill hole of the earlier Mitsubishi DDH-31B) was located, as was the collar of the Benguet drill hole NH-4, which contained a cylindrical concrete plug.

In the opinion of Don Earnest (REI), the fact that the majority of the drill hole 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, there was clear evidence from bank sloughing and vegetation removal that a drill site had been constructed. The likelihood that any of the 21 drill holes selected were not drilled is remote.

## 10.2 ST. AUGUSTINE DRILLING

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 hydrogeologic data for open pit dewatering well design. All of these holes, except SAH-02, were diamond core drill holes drilled by either Drillcorp or Indodrill, both of which are Philippine drilling contractors. The total depth for the 14 holes is 5,987 meters. The drill holes are summarized in Table 10-3. As part of its June 5, 2010 site visit, REI examined the two drill rigs that were in operation (one from each contractor) and found the site set-ups and equipment in use to be acceptable.

**Table 10-3: 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 | Metallurgical 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 |

## 10.3 SIGNIFICANT OUTLYING HISTORICAL DRILL HOLES

Section 9 (Exploration) discusses three of five historical holes that have been determined by SAGC to be indicative of additional mineralization that requires additional exploration work to evaluate the potential. The significant intercepts from these holes are summarized in Table 10-4, and Figure 10-2 shows the location of the five historic exploration drill holes.

**Table 10-4: Significant Intercepts from Outlying Historical Drill Holes**

| Location | Drill Hole | 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 |

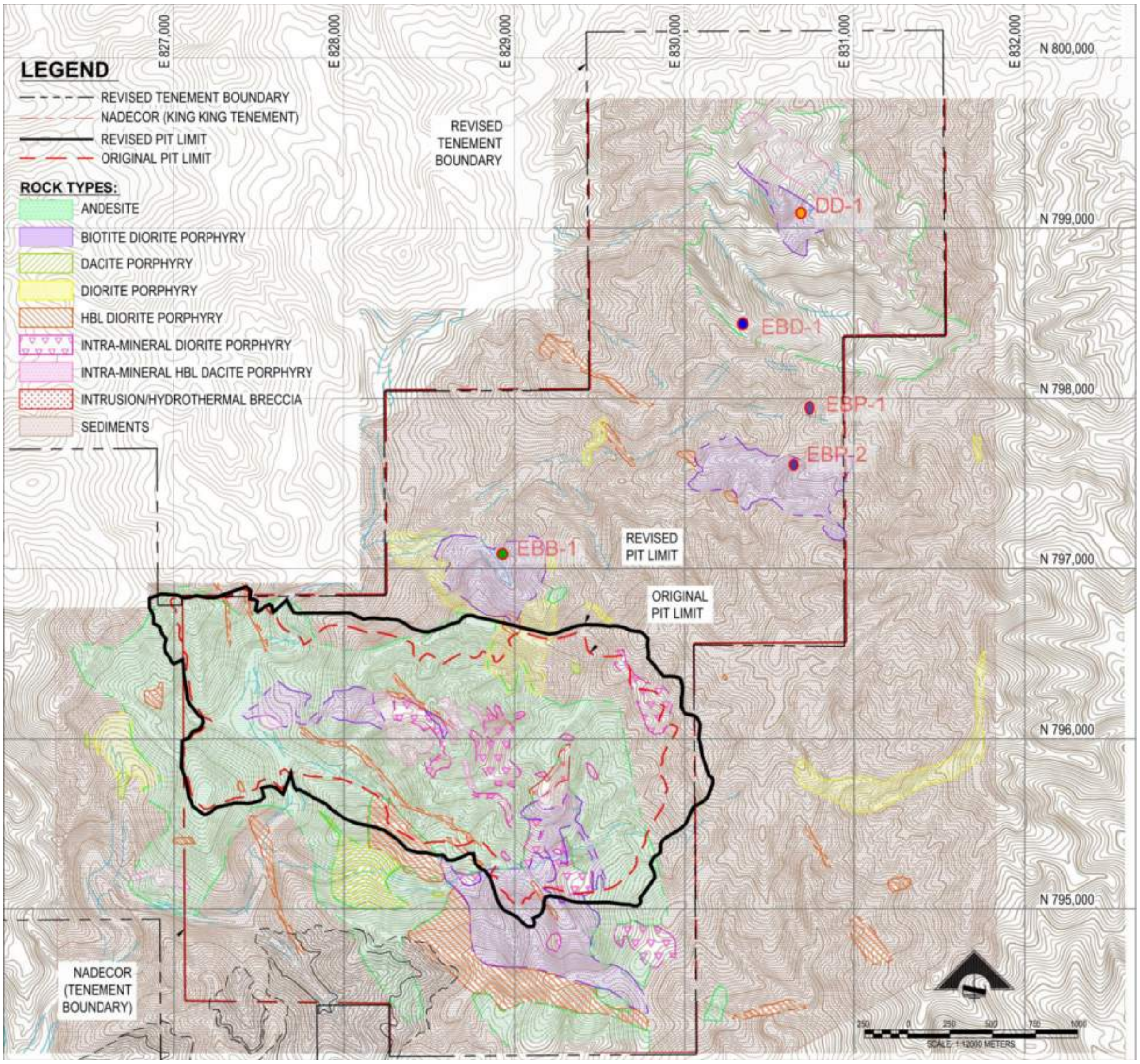



**Figure 10-2: Historical Exploration Drilling Location (SAGC, 2013)**

## 10.4 QP OPINION ON DRILLING DATA

It is the opinion of the QP for this section that the drilling done to date is sufficient to develop an NI 43-101 compliant mineral resource for the Kingking deposit. Based on the structural zones developed to control block grade estimation (see Section 14.4.6), the area represented by the drill hole samples is approximately 1,963,700 square meters ($m^2$), or about 170 hectares. The 400 m bench is a central bench in the deposit. It contains 197 fifteen-meter composites assayed for copper and 112 composites with acceptable gold assays. Dividing the sampled area of 1,963,700 $m^2$ 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 an average sample spacing of 100 m for copper and 132 m for gold.

The QP for this section does not know of any drilling, sampling, or recovery factors that could materially impact the reliability of the results.

# 11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Estimates of mineralized tonnage and grade for the Kingking deposit have historically been based upon assays derived from drilled intercepts. Approximately 33,660 samples were collected over the course of the Kingking 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 Kingking 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, 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 (Knight Piésold, 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 (1) 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 11-1 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.

**Table 11-1: Tolerance Limits**

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

Analytical accuracy and precision are dependent on the following 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 a cumulative 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 for gold in general, actual precision is not as good as the table above based on theoretical "homogenous" samples.

Accuracy was established by using control samples, which 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 were 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 the QA/QC program was to monitor assays on a per batch basis using control samples, duplicates, blanks, replicates and umpire laboratories to ensure assay integrity. KMI monitored seven different types of samples to detect the precision and accuracy of assays provided by the various assay laboratories. These control samples were:

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

##### 11.3.2.2.1 Bulk Pulp Control Samples "A"

During 1996, KMI used six (6) different bulk pulp control samples (KM 1 through KM 6). Control samples KM 1 through KM 3 were submitted randomly in oxidized zones, and KM 4 through KM 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 Kingking Project were composited to obtain a desired grade for copper and gold. Bondar-Clegg Company Ltd. 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 was 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 were compared against the accepted year to date means and global means as established by the round robin analyses for gold and copper. Assays that fall outside these criteria were re-assayed along with the five preceding and five following drill samples in that batch. After the re-assay returned, it was placed in the database as an original assay.

11.3.2.2.2 Bulk Reject Control Samples "B"

The bulk rock control samples used in this program were identified by the letter "B" immediately following the sample number. The material for these samples consisted of composites of coarse reject oxide from drill hole EB-7 having known gold and copper values. Ten samples were initially submitted to Cone Geochemical (Cone) for analysis to set assay ranges for gold and copper. A bulk reject specifically designed to check sample preparation was submitted one per day per hole. Any assays that fell above or below two standard deviations were 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"

Material for the blank control samples was generated by the Inchcape Testing sample prep facility. These samples consisted of ordinary river sand that 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 representing five (5) percent of the assays received were re-submitted to a second lab (Cone). The selection of these samples was random and not biased toward a particular range of concentrations for either gold or copper. The purpose of this type of sample was to provide an outside lab check of the primary lab.

11.3.2.2.6 Lab Duplicate Samples "F"

Inchcape re-assayed 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 assigned the letter "F" to distinguish it from the various other check samples.

11.3.2.2.7 Certified Standards "G"

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

## 11.4 ST. AUGUSTINE DRILLING PROGRAM

### 11.4.1 Core Handling

During St. Augustine's drilling program, the handling of samples was done as follows: Once the inner tube was removed from the core barrel, the actual length of recovered core was measured and recorded. The samples were typically 3 m in length. The core was then transferred from the inner tube of the core barrel and placed in a 3 m PVC pipe that was used as a holder or carrier. The core was then placed in the core box and the top and bottom of the interval were marked by inserting wooden blocks at sample interval breaks and marking the breaks on the core box dividers using black permanent markers. RQD was then measured at the site and recorded.

After the core boxes were filled, they were covered with a stainless steel top and sealed with a crimping tool and plastic strap. The core boxes were then hauled from the drill site to the field camp located in the center of Lumanggang for safe keeping until DrillCorp service vehicles loaded and delivered the boxed core it to the Ayan coreshed in Pantukan.

After delivery to the core shed, the core boxes were opened by personnel that were authorized to break the seals on the boxes, after which the core was cleaned, logged and photographed, two boxes at a time.

Once the core was split into 3 m-length samples, each sample split was placed in double plastic bags and the hole ID and sample number were written on the bag. The bags were then individually weighed, and the sample intervals and lengths recorded in the log book.

A typical dispatch to the laboratories was 100 samples, which were placed in blue plastic barrels bearing the name of the company, number of samples, sample ID's, the name of the destination laboratory, and the name of the person who scheduled to receive the samples at the lab. Pictures were taken of the process, starting with loading of the samples into each drum until each drum was loaded and sealed.

In the opinion of the QP for this section, the documented chain of custody of the samples from the drill rigs in the field to delivery to the assay laboratory is appropriate.

#### 11.4.2 Sample Preparation and Analysis

The samples for the 2011 drilling program were sent to the Intertek Laboratory in Cupang, Muntinlupa City, Philippines. The samples were dried at 105°C. The entire sample was crushed to 95% passing 10 mm and riffle split. An approximate 1.5 kg sample was pulverized to 75% passing 75 microns. The gold analysis was by a 50 g fire assay with AAS finish. Total copper analysis was by 4 acid digestion with AAS finish. These methods are typical and appropriate for copper and gold analysis in the range of grades typical in the Kingking deposit.

The Intertek Laboratory is accredited under ISO/IEC 17025.

#### 11.4.3 QA/QC Program

The QA/QC program for the St. Augustine drilling specified the placement of a blank sample every tenth sample and a standard every 15th sample. The number of blank and standards assays is consistent with this specification. There were three different commercial standards used that were procured from OREAS (Ore Research & Exploration P/L of Australia). There are six samples in the QA/QC database that appear to be from a 4th, much higher-grade standard, but the identity of that standard cannot be determined from the records available to the QP for this section.

There is no record of any pulp or sample duplicate analyses done for the QA/QC program. These need to be added for future drilling programs.

### 11.5 QP OPINIONS OF SAMPLE PREPARATION, SECURITY AND ANALYTICAL PROCEDURES

It is the opinion of the QP for this section that the Echo Bay sample preparation, security, and analytical procedures were adequate for the nature of the relatively low-grade base mineralization and deposit that contain precious metals. In particular, the Echo Bay QA/QC program exceeded industry standards at that time and also exceeds current standards in place at most mining companies. Both QP's of this section worked with Echo Bay's Ken Lovstrom (now deceased) on other Echo Bay projects and have very high regard for his work.

The Benguet sample preparation and analytical procedures, as described in the information provided to the QP for this section, also appear appropriate. The total copper and soluble copper analysis methods are appropriate. The Benguet gold analysis method, however, is complex, and not commonly used. As will be discussed in the next section of this

Technical 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.

It is the opinion of the QP for this section that the sample handling, security, sample preparation, and analytical procedures are adequate for the nature of mineralization at Kingking.

# 12 DATA VERIFICATION

## 12.1 VERIFICATION OF DATA DRILLING

The following data verifications were conducted for the Kingking drill hole 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 assays from the St. Augustine drilling data were verified with the assay certificates.
- The St. Augustine QA/QC procedures were reviewed and verified.

The following sections include the details of the various studies.

## 12.2 COMPARISONS OF ASSAYS WITH ORIGINAL ASSAY CERTIFICATES

### 12.2.1 Echo Bay Assays

Originally, 14 Echo Bay drill holes were selected 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 drill holes, though there was a bias toward selecting more of the higher-grade drill holes.

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, seven additional Echo Bay holes were selected 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.

Holes EB-113, EB-114, EB-117 and EB-118 were audited to bracket the problem holes. Drill holes 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.

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 were checked. 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 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 it would be expected to be approximately half that rate for a verified database. The fact that most of the 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.

The known errors were corrected by replacing the database values with certificate values.

### 12.2.2 Benguet Assays

There were no assay certificates available for the Benguet holes. There were, however, image files from old Benguet drill logs that 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.

Fourteen Benguet holes were selected 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 1 m 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 error rate was 1.2% for total copper, 0.6% for soluble copper, and 1.8% for gold. This is somewhat high, but only 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).

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

### 12.2.3 Mitsubishi Assays

There are no available assay certificates for the Mitsubishi data. Only total copper and soluble copper were assayed for those samples. However, the Mitsubishi copper assays were comparable with Benguet and Echo Bay drilling results.

### 12.2.4 Other Data Checks

Data records with soluble copper greater than or equal to total copper were reviewed with assay 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 review 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 database checks, approximately 132 data records were changed compared with the database used for the 2009 due diligence review.

## 12.3 ECHO BAY RE-ASSAYS OF BENGUET SAMPLES

### 12.3.1 Re-Assayed Holes

Assay certificates for the following 22 Benguet holes that were re-assayed by Echo Bay were received and reviewed:

| | | | | |
|---|---|---|---|---|
| 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. 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.

### 12.3.2 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.

### 12.3.3 Gold

Figure 12-5 shows an xy plot and linear regression for the gold assays. This represents 1,485 assay 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. 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 50g fire assay with an atomic absorption finish.

The Benguet gold assays were not included in the resource model development.

### 12.3.4 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
```

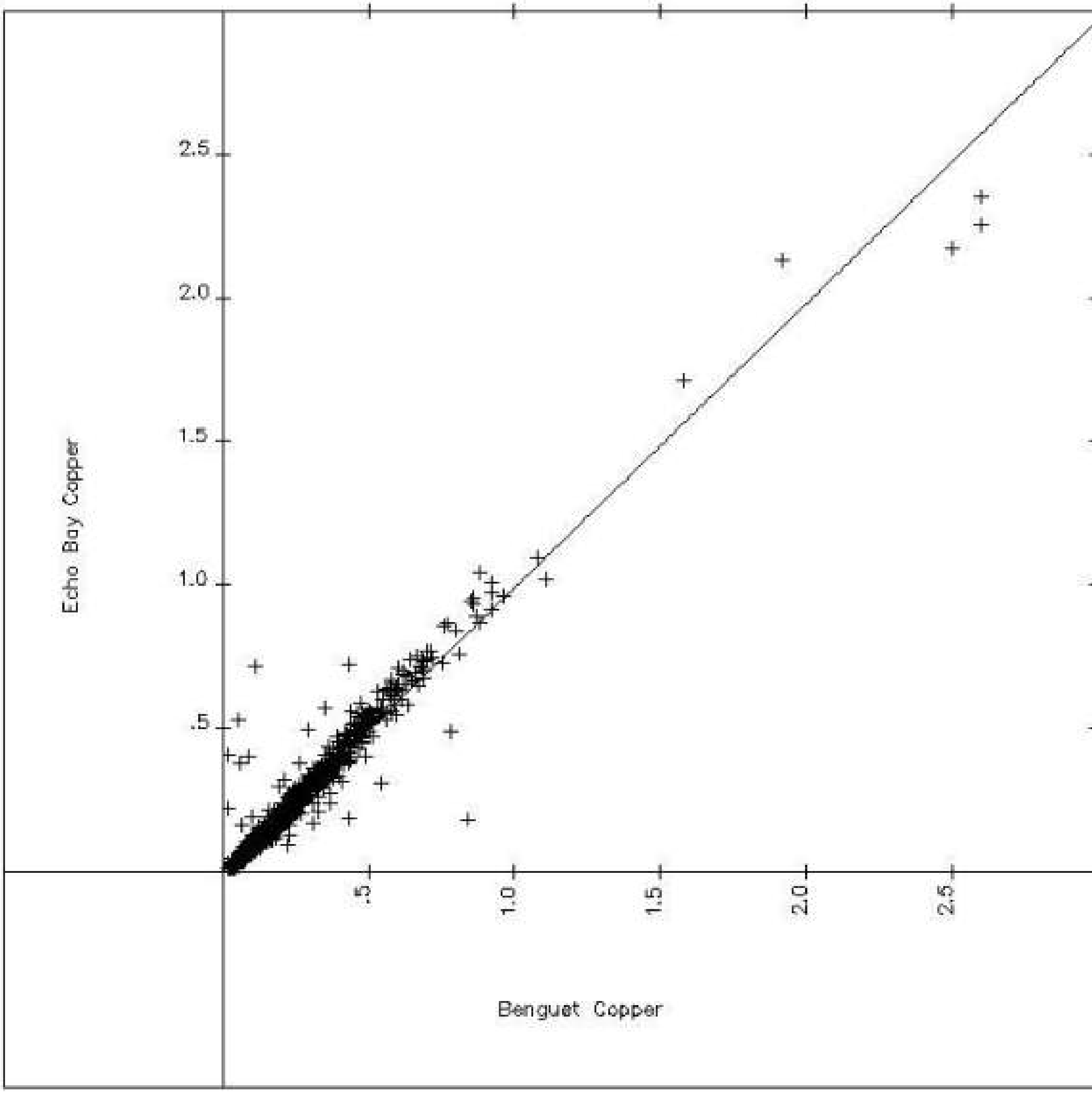



**Figure 12-1: Echo Bay Re-Assays of Benguet Samples Total Copper (IMC, 2010)**

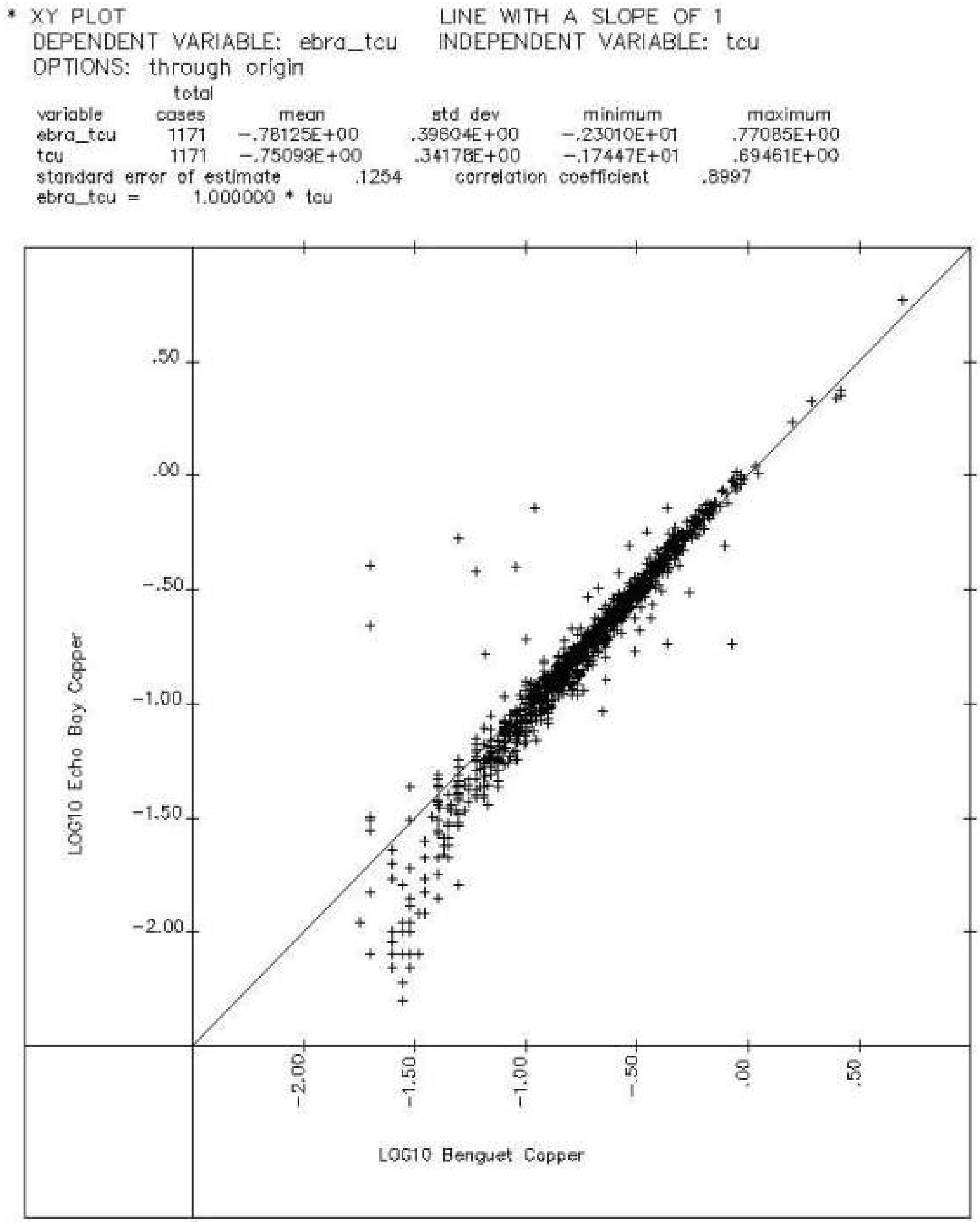
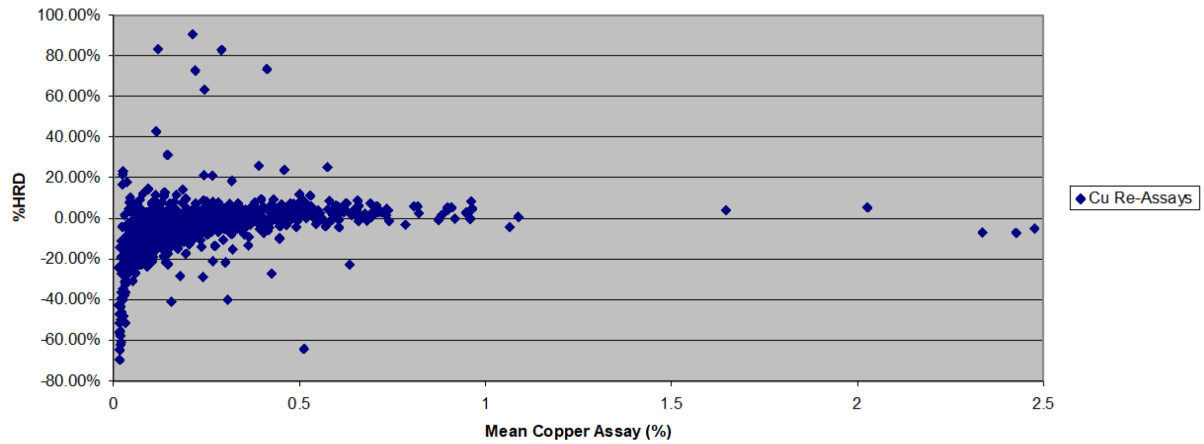
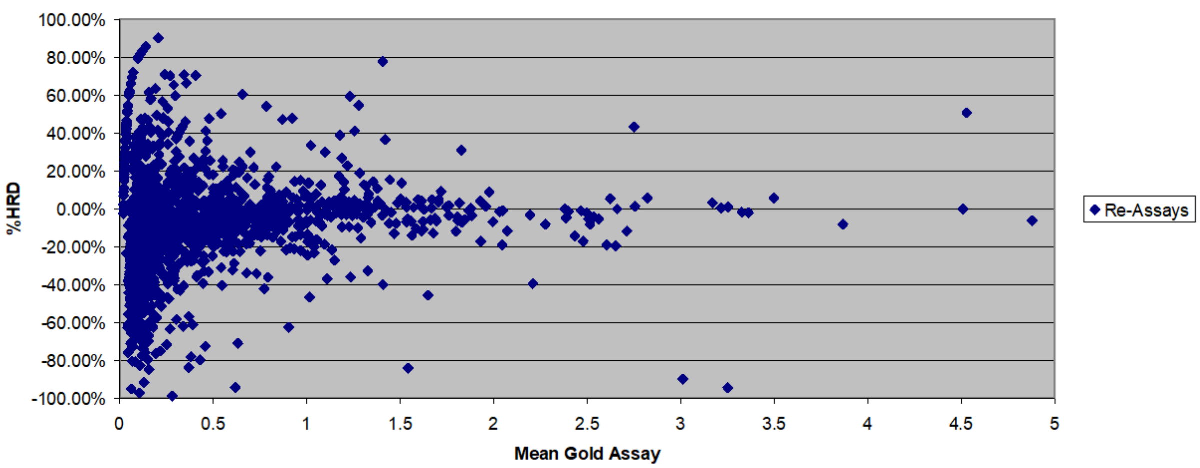



**Figure 12-2: Echo Bay Re-Assays of Benguet Samples Total Copper – Logs Base 10 (IMC, 2010)**




| 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 (IMC, 2010)**




| 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 (IMC, 2010)**

* REGRESSION ANALYSIS LINEAR REGRESSION
DEPENDENT VARIABLE: ebra_au INDEPENDENT VARIABLE: au
OPTIONS: through origin ,biweight

| variable | total cases | non-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

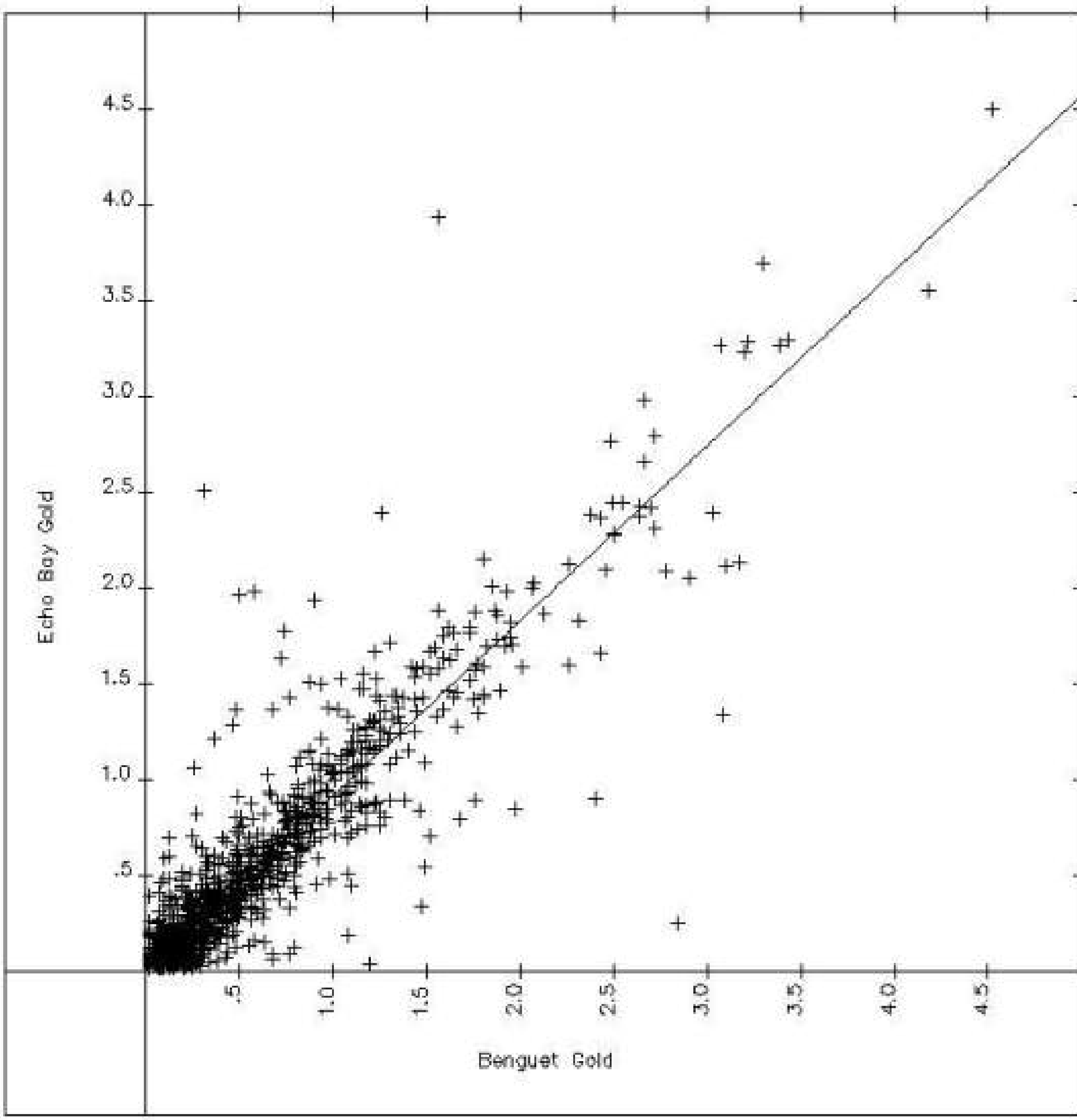



**Figure 12-5: Echo Bay Re-Assays of Benguet Samples Gold (IMC, 2010)**

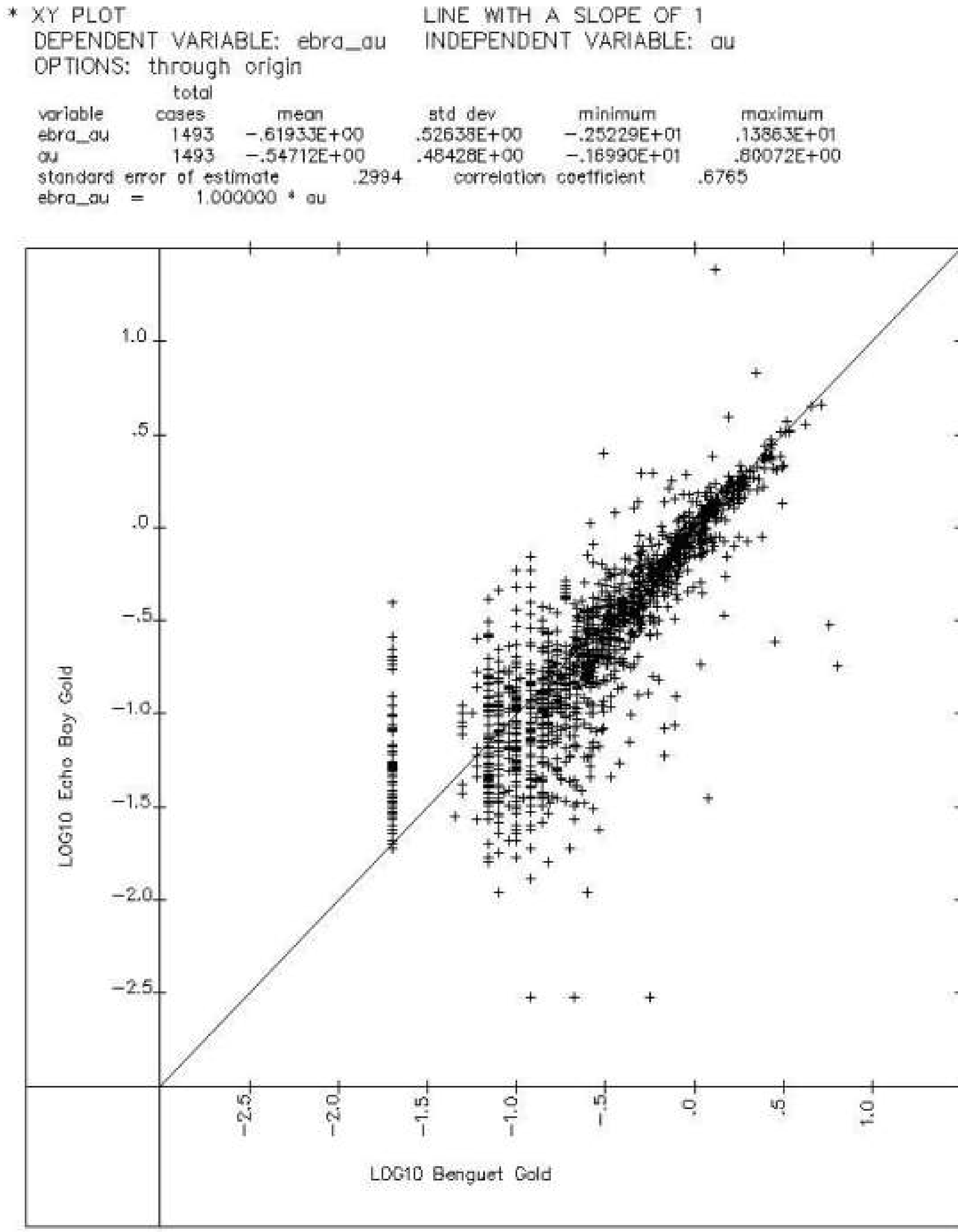



**Figure 12-6: Echo Bay Re-Assays of Benguet Samples Gold – Logs Base 10 (IMC, 2010)**

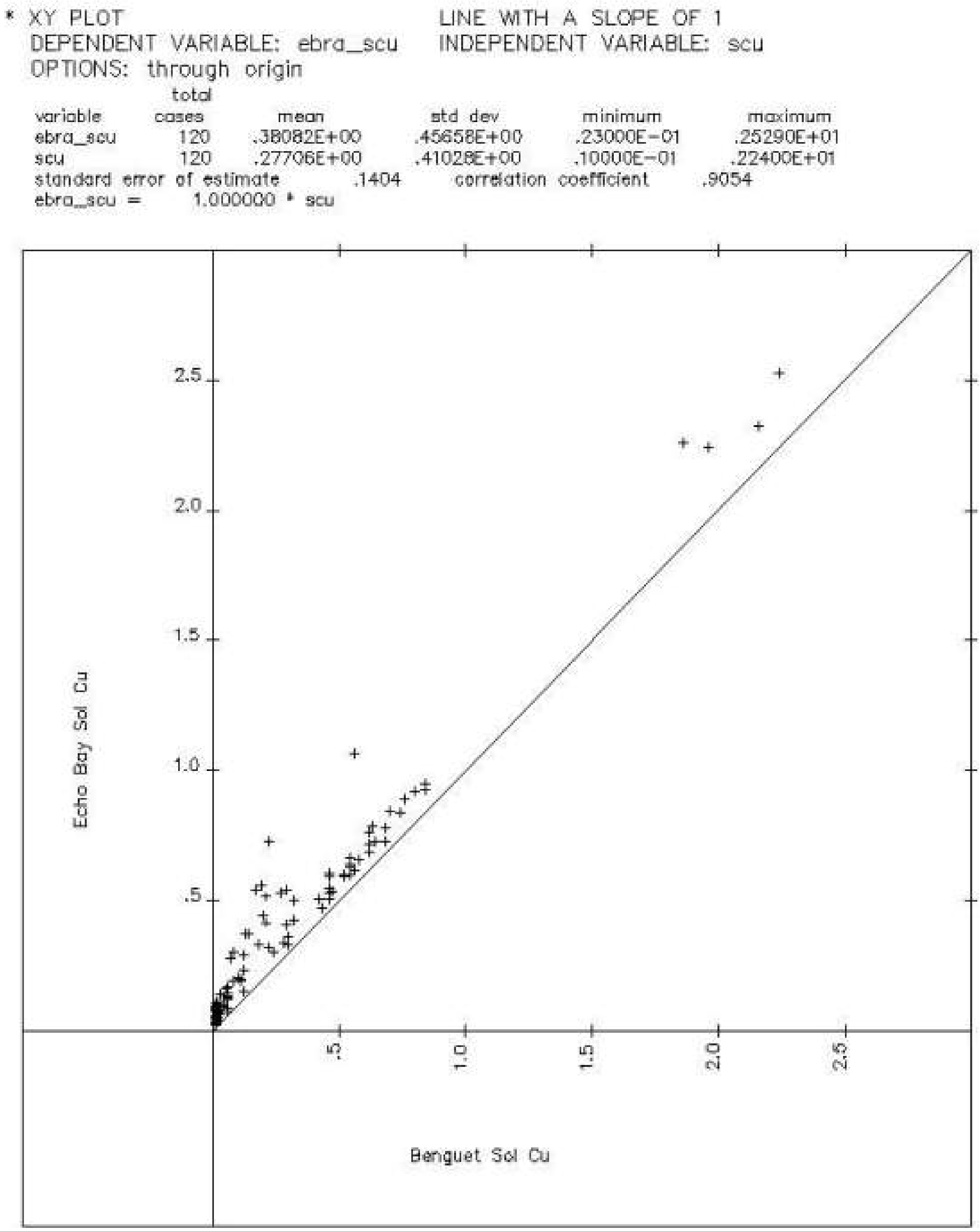



**Figure 12-7: Echo Bay Re-Assay of Benguet Samples Soluble Copper (IMC, 2010)**

## 12.4 ST. AUGUSTINE CHECK ASSAYS

Don 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 Kingking 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).

### 12.4.1 Total Copper

Figure 12-8 shows an xy plot of original total copper assays on the x axis and the SAGC 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 SAGC 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.

### 12.4.2 Gold

Figure 12-10 shows an xy plot of original gold assays on the x axis and the SAGC 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 SAGC 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 > SAGC 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 > SAGC) 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 > SAGC assay). Truncating a few low-grade samples has minimal impact on the results. As with the Echo Bay re-assay program, the SAGC assays indicate the original Benguet gold assays are biased high.

### 12.5 VERIFICATION OF ST. AUGUSTINE DRILLING DATA

Copper and gold assays were done for 9 of the 14 SAGC drill holes. Three of the geotechnical holes and two of the hydrology holes were not assayed. The copper and gold assays in the database were verified with original assay certificates. The database for the SAGC drilling compares well with the certificate data.

SAGC had a QA/QC program in place for the 2011 drilling. Total assays for the drilling included 1,267 copper and 1,267 gold assays. For the QA/QC program there were 149 blank samples submitted, about 1 blank to 8 or 9 samples, and 98 standards, about 1 standard in 13 samples. This is an adequate number of control samples and are higher frequencies than are typical in most exploration drilling. It does not appear, however, that any pulp or sample duplicates were included in the QA/QC program. It is recommended that pulp and sample duplicates be added for future QA/QC programs.

The standards were sourced from OREAS (Ore Research & Exploration P/L of Australia. The standards used were Oreas 151a, 153a, and 161. The certified values for the standards are as follows:

| Standard | Cu (%) | Au (ppm) |
|---|---|---|
| 151A | 0.166 | 0.043 |
| 153A | 0.712 | 0.311 |
| 161 | 0.409 | N.A. |

The copper values for the selected standards bracket appropriate grade ranges for Kingking mineralization. The gold value for standard 153A is also appropriate. The gold value for 151A is lower than desirable for Kingking mineralization.

Six of the standard assays were on a much higher-grade standard, about 1.5% copper and 3 ppm gold. However, the identity of this standard is not known. Control plots for the blanks and standards were developed and reviewed by the QP for this section. There is nothing in the QA/QC samples to indicate any issues with the analysis or that any of the assays need to be repeated.

### 12.6 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 reviews completed, and subsequent corrections, the QP for this section 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 SAGC 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 SAGC 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.

The SAGC drilling was verified with assay certificates and the QA/QC program for the drilling was reviewed. The SAGC assays are acceptable for the estimation of Mineral Resources and Mineral Reserves.

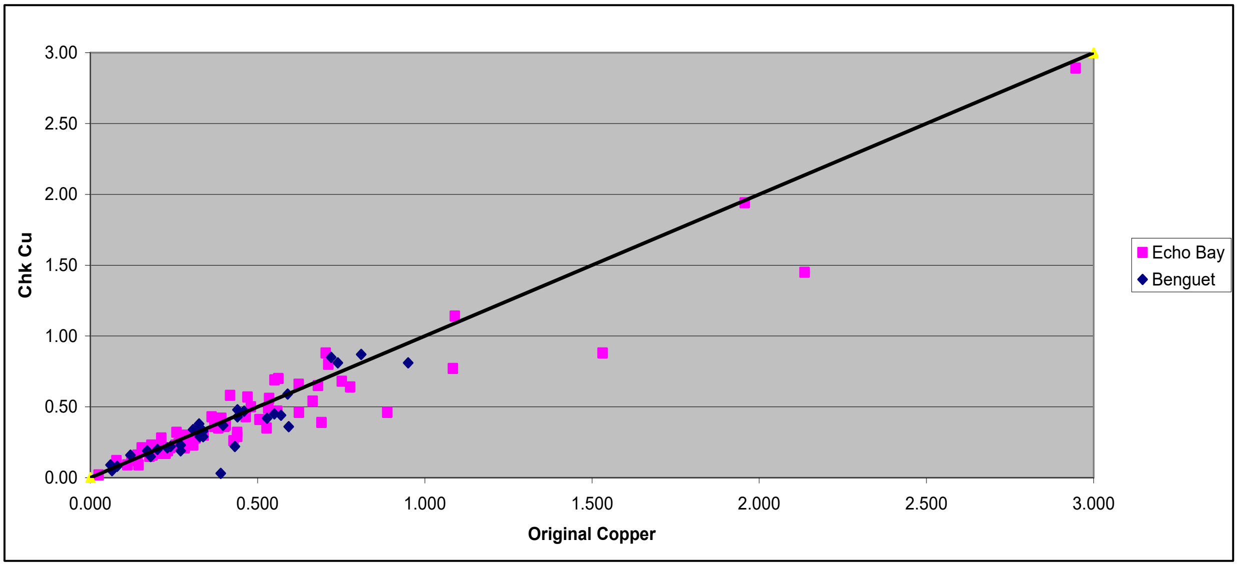



**Figure 12-8: Total Copper – SAGC Check Assays vs Original Assays (IMC, 2010)**

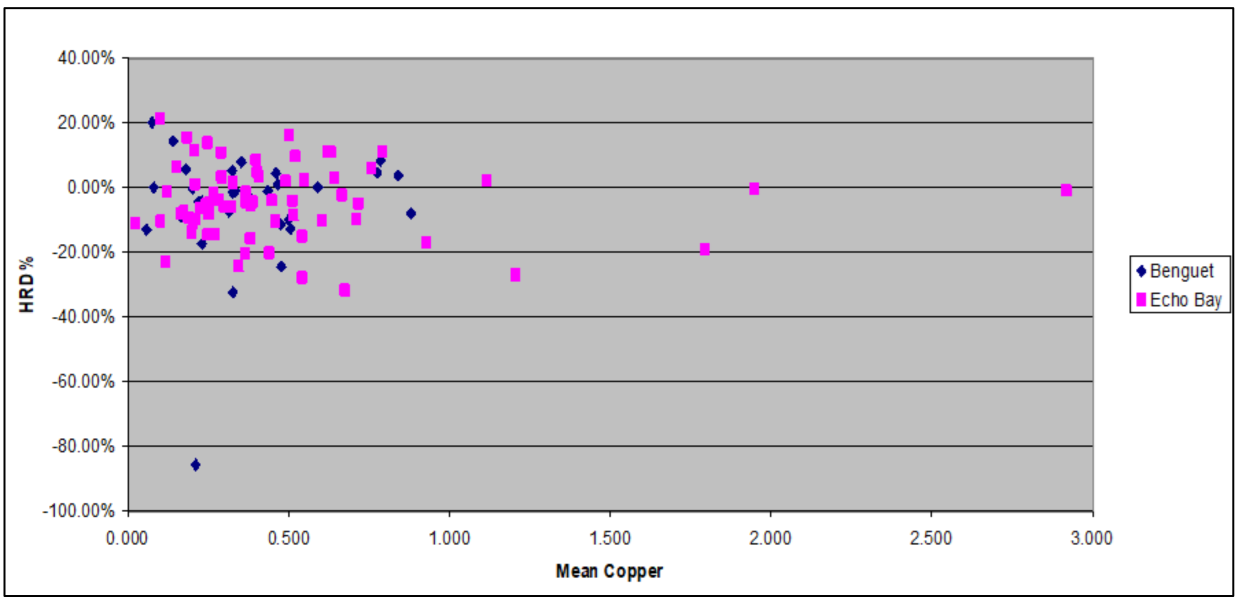



**Figure 12-9: HRD% vs Mean Copper Grade for SAGC Check Assays (IMC, 2010)**

**Table 12-1: SAGC Check Assays versus 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% |

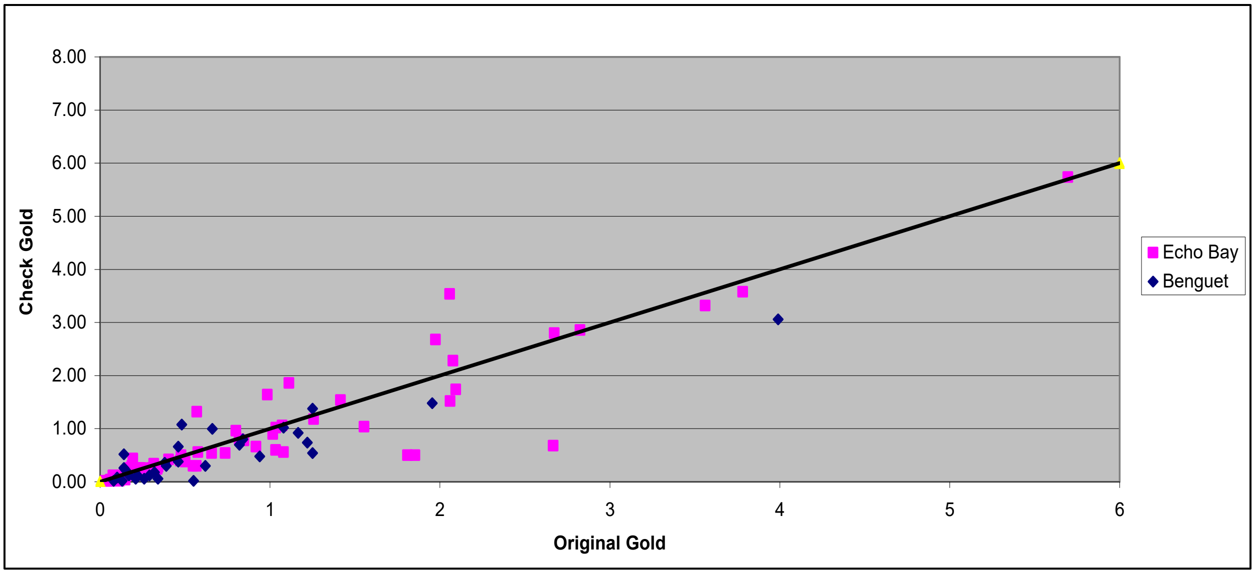



**Figure 12-10: Gold – SAGC Check Assays vs Original Assays (IMC, 2010)**

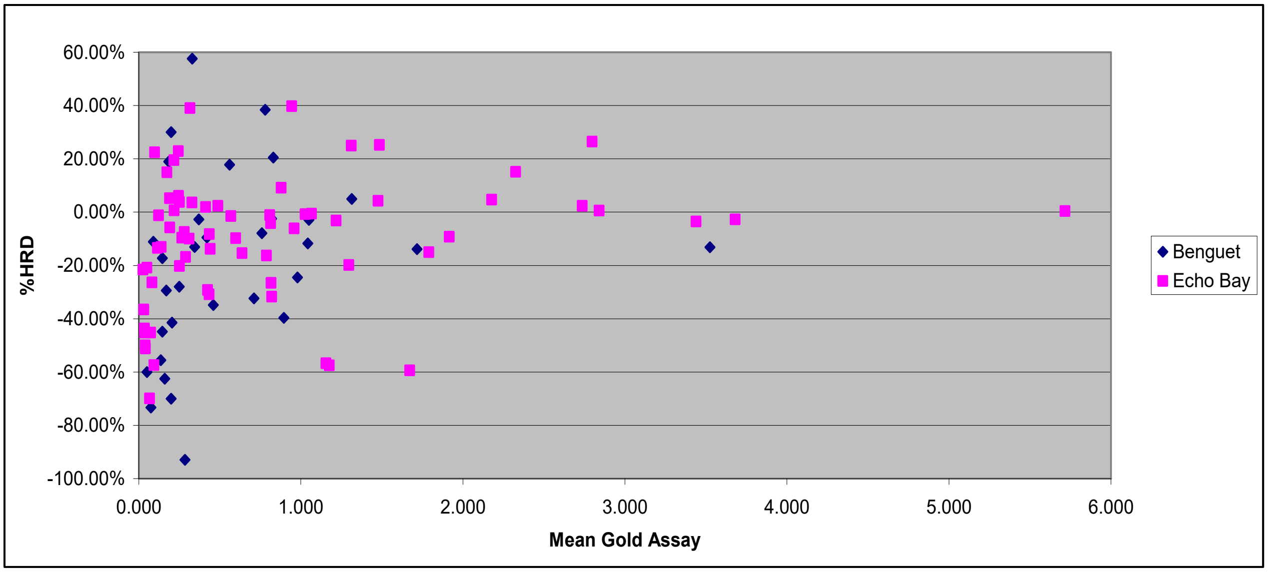



**Figure 12-11: % HRD vs Mean Gold Grade for RMMI Check Assays (IMC, 2010)**

**Table 12-2: SAGC Check Assays versus 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% |

## 12.7 METALLURGICAL TESTING AND INTERPRETATION OF TEST RESULTS

Art Ibrado, PE, the qualified person (QP) for this section, has thoroughly reviewed test work that formed the basis of Section 13 and found them to be compliant with the requirements of this Technical Report in detail and in depth. The work was performed by reputable laboratories with proven reliability in the industry over the years.

The previous QPs for Section 13 of the 2013 Technical Report were (a) Greg Harbort, (then of AMEC Minproc in Australia), who oversaw the metallurgical testing at AMDEL Limited in Australia, JK Tech, and University of South Australia and (b) Ronald Roman of Leach, Inc. in Tucson, AZ, USA. Both wrote metallurgical reports and contributed to the initial writing of this section.

Extensive metallurgical tests were performed to evaluate flotation, heap leaching and flotation tailing leach. More details from the metallurgical reports have been added to provide a clear narrative of how the process was designed. The current QP has interpreted some of the flotation results differently from the previous QP, as indicated by comments inserted in Section 13. For the heap leach work, the current QP agrees with the previous interpretations and conclusions.

# 13 MINERAL PROCESSING AND METALLURGICAL TESTING

*Art Ibrado, PE, of Fort Lowell Consulting PLLC (FLC) is the qualified person (QP) for this section. He has thoroughly reviewed test work that formed the basis of this section. The previous QPs for Section 13 of the 2013 Technical Report were (a) Greg Harbort, (then of AMEC Minproc in Australia), who oversaw the metallurgical testing at AMDEL Limited in Australia, JK Tech, and University of South Australia and (b) Ronald Roman of Leach, Inc. in Tucson, AZ, USA. Both wrote metallurgical reports and contributed to the initial writing of this section.*

***QP Comment:***
*Extensive metallurgical tests were performed to evaluate flotation, heap leaching and flotation tailing leach. More details from the metallurgical reports have been added to provide a clear narrative of how the process was designed. Comments are inserted in this section in reaction to a few methodologies and interpretations given by one of the previous QPs.*

## 13.1 PREVIOUS METALLURGICAL TESTS

Previous metallurgical tests on Kingking ores were carried out by Metcon Research Inc. (Tucson, AZ, USA) in 1993 for Benguet Corp. and by Lakefield Research Inc. (Lakefield, Ontario, Canada) in 1997 for Echo Bay Mines.

### 13.1.1 Metcon Leach and Flotation Tests (1993)

The Metcon work was done to develop column leach parameters for two oxide composites. Lower and Upper oxide materials, with head grades of 0.70 and 0.51% Cu, respectively. A recovery of 90.5% was achieved for the Lower Oxide material, and 84.0% for the Upper Oxide material, both at an acid-cure dosage of 17.5 kg/ton.

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. The collectors investigated were potassium amyl xanthate (PAX), isopropyl ethyl thionocarbamate (IPETC or Z-200), sodium dialkyl dithiophosphate (Aero 3477) and xanthate allyl ester (Aero 3302), with copper sulfate added in one reagent combination, and with none of the collectors tested by itself. For both composites, the best collector combination was 48 g/t 3477 and 25 g/t PAX, resulting in the best recoveries and concentrate grades. This collector combination was used to test the effect of particle size, testing 48, 65, 100 and 150 mesh. For the mixed material, copper recoveries were the same for all sizes, but the best concentrate grade was from the 48-mesh material. However, gold recoveries were better at 100 and 150 mesh. For the sulfide material, the best copper and gold recoveries were obtained from the 150-mesh material with a copper concentrate grade of 6.11.

Locked-cycle tests (LCT) were performed using 3477/PAX reagent scheme on samples ground to 48 mesh and at pH 10 to 10.5. The tests yielded 69.7% recovery of copper and 72.2% recovery of gold from the mixed material, and 77.5% recovery of copper and 75.3% recovery of gold from the sulfide material. Final copper concentrates assayed 11.7% Cu for the mixed material, and 20.9% Cu for the sulfide material. Metcon attributed the poor locked-cycle test results to the accumulation of slimes. However, the use of PAX, a strong collector, may have contributed to the inability to achieve better concentrate grades.

### 13.1.2 Lakefield Flotation Tests (1997)

The Lakefield tests were performed to develop a flowsheet and reagent scheme for two sulfide master composites: Composite 1S (0.72% Cu, 0.44 g/t Au, 1.95% total sulfur) and Composite 2S (0.61% Cu, 1.06 g/t Au and 0.88% total sulfur). Composite 1S was expected to contain more pyrite, as evidenced by its higher sulfur-to-copper ratio.

Collector additions during the locked-cycle tests were rather high at 90 g/t Aero 3477 and at 30 g/t PAX in the roughers, with smaller additions (up to 10 g/t) in the cleaner and cleaner scavenger stages. For Composite 2S, the pH was targeted at about 10.5 in the rougher tests and 11.5 in the cleaner tests. For Composite 1S, the pH was lower in the roughers, averaging 9 in two tests, and 7.0 in another test. This seems to go in the wrong direction as a higher pH may work better in depressing pyrite in the higher-pyrite composite.

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 LCT flotation tests on Composite 2S yielded a copper recovery of 87.6% at a final concentrate grade of 32.5% Cu and 90.5% recovery at a final concentrate grade of 26% Cu. The maximum gold recovery attained was 81%.

For Composite 1S, the LCT flotation performance was less satisfactory as shown in Table 13-1 below. The best copper recovery was from Test 61 but with a lower concentrate grade. A coarser grind (Test 63) resulted in a lower copper recovery but better concentrate grade and better gold recovery. The test done at pH 7 did poorly overall.

**Table 13-1: Results of LCT Flotation on Composite 1S (high pyrite)**

| LCT Test No | Grind, microns | pH | Cu Recovery, % | Final Conc Grade, % Cu | Au Recovery, % |
|---|---|---|---|---|---|
| 61 | 90 | 9.0 | 85.1 | 20.9 | 74.8 |
| 63 | 209 | 9.0 | 80.6 | 33.7 | 80.6 |
| 64 | 90 | 7.0 | 74.7 | 18.1 | 64.2 |

### 13.1.3 Current Tests

Following a preliminary assessment of historical Kingking metallurgical tests, SAGC with AMEC Australia commissioned a series of tests at AMDEL Limited in Australia, JK Tech, and University of South Australia to aid in the understanding of the individual domain characteristics, to assess variability of metallurgical performance, and to predict the performance of the commercial plant based on the current mine plan and production composite samples. Column leach tests were conducted by Leach Inc. in Tucson, Arizona, USA under contract with SAGC.

The metallurgical test work program consisted of a series of comminution and flotation tests, thickener sizing tests, a gold deportment test 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.

## 13.2 SAMPLE SELECTION

### 13.2.1 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). These samples are listed in Table 13-2 and plotted on a map in Figure 13-1.

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
- 4 ‘Life of Mine’ (LOM) composites: Years 2/3, Years 4/5, Years 6/10, and “Remaining Life” (RL)
- 38 ‘variability’ samples were taken from individual drill core intervals selected to represent the orebody

The origins of the first two sets of composites are included in AMEC's metallurgical report (AMEC, Mar 2012). The 38 variability composites are listed in Table 13-3 and plotted on a map in Figure 13-2.

Note: For both maps, the SAGC holes identified as SAM-01, SAG-01, SAG-02, and SAG-03 are the same as holes Met1, R-12, R-03, and R-02, respectively.

**Table 13-2: Details of Individual Samples for Comminution Variability Samples**

| Sample ID | Hole No. | From (m) | To (m) | Ore Type (Geo) | Ore Type (Met) |
|---|---|---|---|---|---|
| 65007-C-01 | R-12 | 24.08 | 40.50 | Oxide | Oxide |
| 65007-C-02 | R-12 | 73.34 | 89.76 | Oxide | Oxide |
| 65007-C-03 | R-12 | 122.60 | 139.02 | Oxide | Oxide |
| 65007-C-04 | R-12 | 171.86 | 188.28 | Oxide | Oxide |
| 65007-C-05 | EB-124 | 75.00 | 90.00 | Oxide | Oxide |
| 65007-C-06 | EB-112 | 105.00 | 120.00 | Oxide | Oxide |
| 65007-C-07 | EB-28 | 103.00 | 118.00 | Oxide | Sulfide |
| 65007-C-08 | EB-41 | 25.00 | 40.00 | Oxide | Oxide |
| 65007-C-09 | BN-19 | 30.00 | 45.00 | Mixed | Oxide |
| 65007-C-10 | EB-11 | 15.00 | 30.00 | Mixed | Oxide |
| 65007-C-11 | EB-7 | 142.00 | 157.00 | Mixed | Sulfide |
| 65007-C-12 | EB-99 | 180.00 | 195.00 | Mixed | Sulfide |
| 65007-C-13 | EB-99 | 195.00 | 210.00 | Mixed | Sulfide |
| 65007-C-14 | EB-12 | 70.00 | 85.00 | Sulfide | Sulfide |
| 65007-C-15 | EB-18 | 120.00 | 135.00 | Sulfide | Sulfide |
| 65007-C-16 | EB-7 | 180.00 | 195.00 | Sulfide | Sulfide |
| 65007-C-17 | EB-71 | 135.00 | 150.00 | Sulfide | Sulfide |
| 65007-C-19 | R-02 | 170.73 | 186.07 | Sulfide | Sulfide |
| 65007-C-20 | R-02 | 400.76 | 416.09 | Sulfide | Sulfide |
| 65007-C-21 | R-03 | 184.03 | 199.89 | Sulfide | Sulfide |
| 65007-C-22 | R-03 | 326.80 | 342.67 | Sulfide | Sulfide |
| 65007-C-23 | R-12 | 221.12 | 237.54 | Sulfide | Sulfide |
| 65007-C-24 | R-12 | 270.38 | 286.79 | Sulfide | Sulfide |
| 65007-C-25 | R-12 | 319.63 | 336.05 | Sulfide | Sulfide |
| 65007-C-26 | R-12 | 368.89 | 385.31 | Sulfide | Sulfide |
| 65007-C-27 | Met 1 | 100.00 | 115.00 | Sulfide | Oxide |

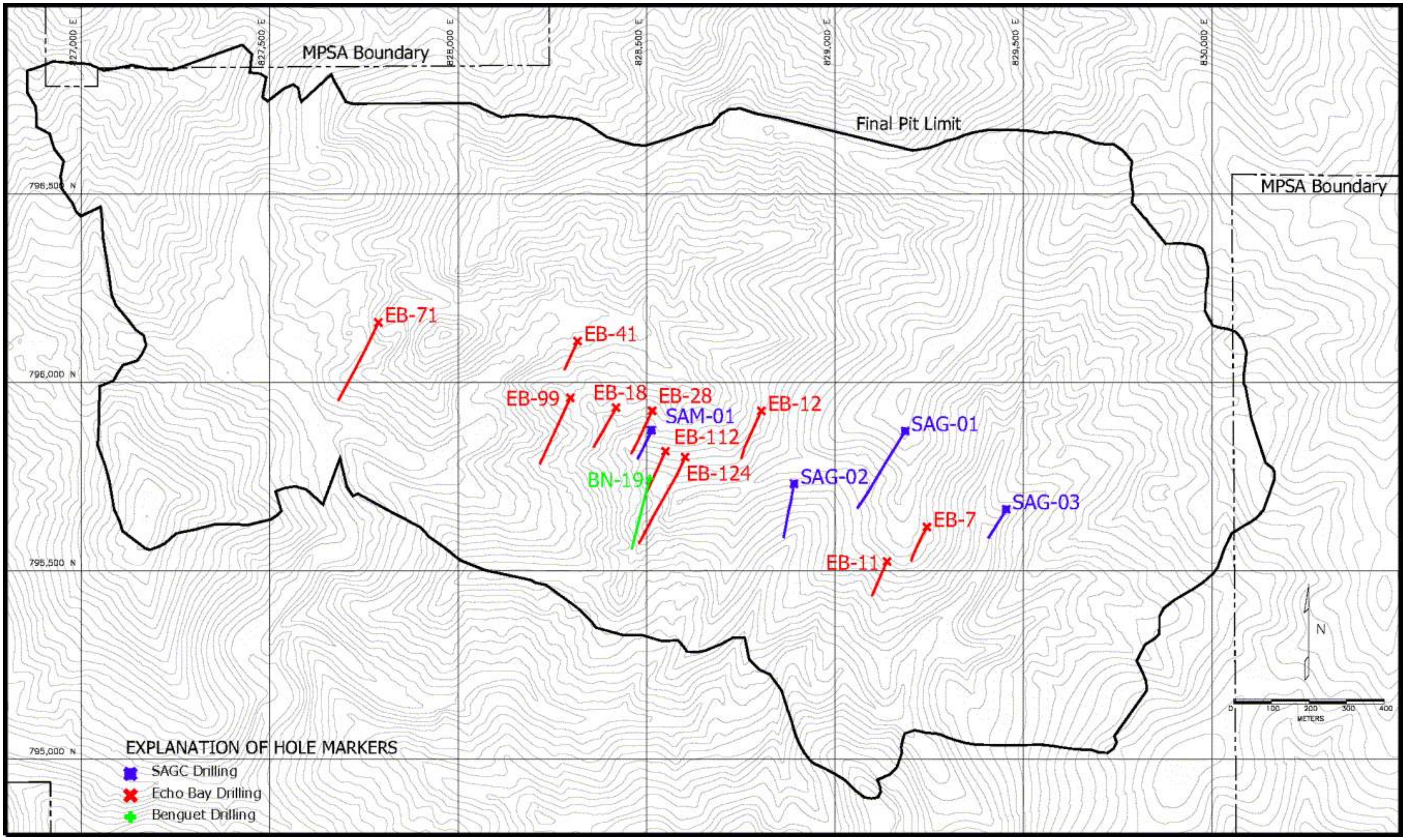



**Figure 13-1: Location of Comminution Variability Samples in the Orebody (IMC, 2024)**

**Table 13-3: Details of Individual Samples Used for Flotation Variability Samples**

| AMEC No. | Hole No. | From (m) | To (m) | Form | Ore Type |
|---|---|---|---|---|---|
| F-01 | BN-21 | 102 | 105 | -2mm | Oxide |
| F-02 | BN-25 | 54 | 57 | -2mm | Oxide |
| F-03 | EB-116 | 252 | 255 | -2mm | Oxide |
| F-04 | EB-116 | 102 | 105 | -2mm | Oxide |
| F-05 | EB-116 | 39 | 42 | -2mm | Oxide |
| F-06 | EB-27 | 129 | 132 | -2mm | Oxide |
| F-07 | EB-27 | 168 | 171 | -2mm | Oxide |
| F-08 | BN-25 | 117 | 120 | -2mm | Oxide |
| F-09 | EB-63 | 159 | 162 | -2mm | Oxide |
| F-10 | R-03 | 9.52 | 25.38 | | Oxide |
| F-11 | R-03 | 57.11 | 72.98 | | Oxide |
| F-12 | R-12 | 24.08 | 40.50 | | Oxide |
| F-13 | R-12 | 73.34 | 89.76 | | Oxide |
| F-14 | R-12 | 122.60 | 139.02 | | Oxide |
| F-15 | R-12 | 171.86 | 188.28 | | Oxide |
| F-16 | R-02 | 17.38 | 32.71 | | Oxide |
| F-17 | EB-103 | 186 | 189 | -2mm | Mixed (Oxide) |
| F-18 | EB-103 | 231 | 234 | -2mm | Mixed (Oxide) |
| F-19 | BN-8 | 129 | 132 | -2mm | Mixed (Oxide) |
| F-20 | EB-22 | 141 | 144 | -2mm | Mixed (Oxide) |
| F-21 | EB-78 | 255 | 258 | -2mm | Mixed (Oxide) |
| F-22 | BC-1 | 339 | 342 | -2mm | Mixed (Oxide) |
| F-23 | R-03 | 88.84 | 104.70 | | Mixed (Oxide) |
| F-24 | EB-103 | 198 | 201 | -2mm | Sulfide |
| F-25 | BC-01 | 102 | 105 | -2mm | Oxide |
| F-26 | EB-42 | 147 | 150 | -2mm | Sulfide |
| F-27 | EB-51 | 144 | 147 | -2mm | Sulfide |
| F-28 | EB-100 | 165 | 168 | -2mm | Sulfide |
| F-29 | BN-31 | 144 | 147 | -2mm | Sulfide |
| F-30 | R-02 | 63.39 | 78.72 | | Sulfide |
| F-31 | R-03 | 184.03 | 199.89 | | Sulfide |
| F-32 | R-12 | 221.12 | 237.54 | | Sulfide |
| F-33 | R-12 | 270.38 | 286.79 | | Sulfide |
| F-34 | R-12 | 319.63 | 336.05 | | Sulfide |
| F-35 | R-12 | 368.89 | 385.31 | | Sulfide |
| F-36 | Met 1 | 20.00 | 35.00 | | Oxide |
| F-37 | Met 1 | 80.00 | 95.00 | | Oxide |
| F-38 | Met 1 | 160.00 | 175.00 | | Sulfide |

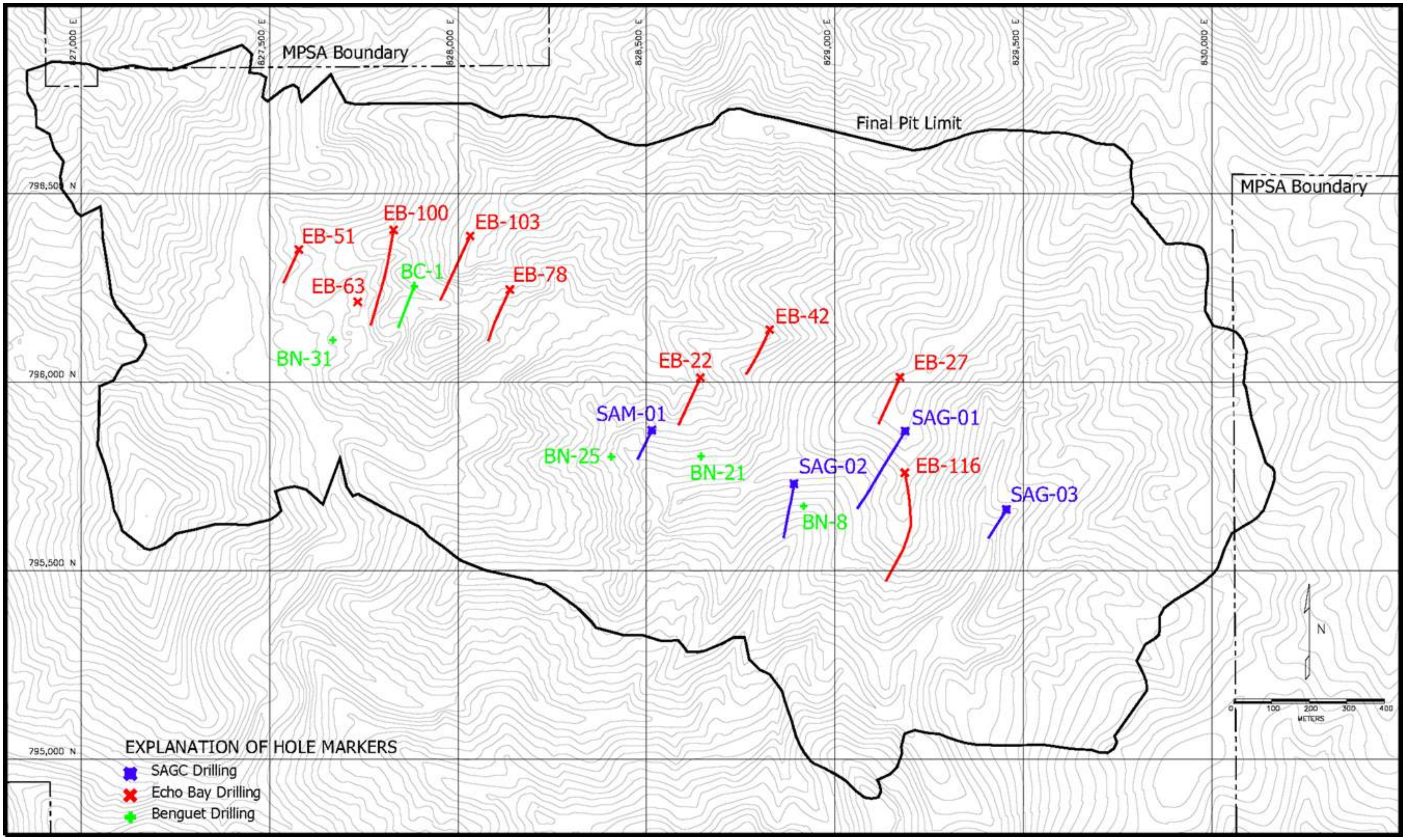



**Figure 13-2: Location of Flotation Variability Composites in the Orebody (IMC, 2024)**

The head samples were assayed for Cu, Au, S, and Fe. Copper was determined sequentially with the following techniques: (1) Deionized 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 analyzed by fire assay. 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.

The two ore type composites (Oxide and Sulfide) and the four life-of-mine (LOM) composites were subjected to 6-cycle locked cycle testing. The Oxide and Years 2/3 (LOM) were floated using the oxide float scheme (LCA) employing PAX as the collector, and the Sulfide, Years 4/5, Years 6/10, and RL were floated using a sulfide float scheme employing SIBX as the collector. The Oxide and Years 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. Only the sulfide locked-cycle flotation results are included in this Technical Report. The rest of them can be found in the AMEC Metallurgical report.

### 13.2.2 Heap Leaching of Oxide Ore Samples

A group of 25 samples, listed in Table 13-4 below, 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.

**Table 13-4: List of Samples Used for Column Leach Testing**

| Column No. | DDH I.D. | Sample No. | From (m) | To (m) | Length of Sample (m) |
|---|---|---|---|---|---|
| KK-1 | BC-20 | 3H-1 | 20.23 | 36.19 | 15.96 |
| KK-2 | BC-22 | 1L-1 | 20.35 | 41.99 | 21.64 |
| KK-3 | EB-17 | 3M-1 | 24.07 | 40.03 | 15.96 |
| KK-4 | EB-88 | 1H-1 | 65.49 | 81.45 | 15.96 |
| KK-5 | EB-88 | 0H-3 | 1.64 | 22.00 | 20.36 |
| KK-6 | EB-69 | 6M-1 | 7.19 | 35.70 | 28.51 |
| KK-7 | EB-119 | 2H-1 | 38.44 | 54.40 | 15.96 |
| KK-8 | EB-105 | 1M-2 | 13.89 | 29.85 | 15.96 |
| KK-9 | EB-124 | 2H-2 | 87.31 | 107.49 | 20.18 |
| KK-10 | EB-64 | 0L-1 | 35.08 | 51.04 | 15.96 |
| KK-11 | EB-92 | 5L-1 | 147.48 | 163.34 | 15.86 |
| KK-12 | EB-59 | 0H-2 | 32.31 | 48.80 | 16.49 |
| KK-13 | EB-59 | 0M-2 | 0.38 | 27.50 | 27.12 |
| KK-14 | EB-125 | 4H-2 | 125.22 | 152.55 | 27.33 |
| KK-15 | EB-105 | 2L-1 | 45.81 | 68.95 | 23.14 |
| KK-16 | EB-47 | 5H-2 | 63.50 | 88.94 | 25.44 |
| KK-17 | BN-16 | 1M-1 | 29.27 | 44.27 | 15.00 |
| KK-18 | EB-15 | 0H-1 | 80.35 | 96.31 | 15.96 |
| KK-19 | EB-40 | 0M-1 | 63.00 | 88.35 | 25.35 |
| KK-20 | EB-32 | 5H-1 | 125.74 | 152.30 | 26.56 |
| KK-21 | BN-21 | 2M-1 | 19.00 | 38.47 | 19.47 |
| KK-22 | BC-20 | 2H-3 | 36.19 | 52.80 | 16.61 |
| KK-23 | EB-64 | 0L-2 | 61.70 | 82.96 | 21.26 |
| KK-24 | EB-40 | 4M-1 | 152.20 | 185.40 | 33.20 |
| KK-25 | EB-36 | 4L-1 | 63.00 | 90.30 | 27.30 |

## 13.3 COMMINUTION TESTS

The results of these comminution tests indicate that the Kingking rock mineralization exhibits variable rock competency and ball mill grindability. Variation in material competency between the two major ore domains, oxide and sulfide, was also 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 of each parameter obtained from the Kingking test work are summarized in Table 13-5.

**Table 13-5: Comminution Test Work Results for Oxide and Sulfide Samples**

| Sample ID | Hole No. | From, m | To, m | Prod Year | Ai, g | Axb | $t_a$ | BWi, kWh/t 150m* | RWi, kWh/t 1180m* |
|---|---|---|---|---|---|---|---|---|---|
| ***Oxide*** | | | | | | | | | |
| 65007-C-01 | R-12 | 24 | 41 | 1 | 0.04 | 75.6 | 0.8 | 11.00 | 10.90 |
| 65007-C-02 | R-12 | 73 | 90 | 2 | 0.03 | 124.2 | 1.3 | 8.70 | 8.50 |
| 65007-C-03 | R-12 | 123 | 139 | 2 | 0.03 | 76.1 | 0.8 | 11.40 | 10.80 |
| 65007-C-04 | R-12 | 172 | 188 | 3 | 0.07 | 38.3 | 0.4 | 14.60 | 14.30 |
| 65007-C-05 | EB-124 | 75 | 90 | 3 | 0.10 | 36.9 | 0.4 | 12.80 | 14.30 |
| 65007-C-06 | EB-112 | 105 | 120 | 3 | 0.03 | 67.9 | 0.7 | 9.00 | 9.70 |
| 65007-C-08 | EB-41 | 25 | 40 | 5 | 0.07 | 85.1 | 0.9 | 9.10 | 9.20 |
| 65007-C-09 | BN-19 | 30 | 45 | 2 | 0.07 | 49.5 | 0.5 | 13.00 | 14.00 |
| 65007-C-10 | EB-11 | 15 | 30 | 2 | 0.14 | 78.8 | 0.8 | 6.70 | 7.80 |
| 65007-C-27 | Met 1 | 100 | 115 | 5 | 0.16 | 37.8 | 0.4 | 13.10 | 15.00 |
| ***Sulfide*** | | | | | | | | | |
| 65007-C-07 | EB-28 | 103 | 118 | 4 | 0.13 | 47.8 | 0.5 | 9.90 | 11.50 |
| 65007-C-11 | EB-7 | 142 | 157 | 3 | 0.31 | 46.2 | 0.5 | 10.10 | 10.30 |
| 65007-C-12 | EB-99 | 180 | 195 | 6 | 0.15 | 34.2 | 0.4 | 14.20 | 15.90 |
| 65007-C-13 | EB-99 | 195 | 210 | 12 | 0.13 | 31.9 | 0.3 | 15.60 | 16.80 |
| 65007-C-14 | EB-12 | 70 | 85 | 3 | 0.08 | 41.7 | 0.5 | 13.70 | 13.40 |
| 65007-C-15 | EB-18 | 120 | 135 | 5 | 0.23 | 41.0 | 0.4 | 12.20 | 12.20 |
| 65007-C-16 | EB-7 | 180 | 195 | 4 | 0.26 | 36.2 | 0.4 | 12.30 | 13.00 |
| 65007-C-17 | EB-71 | 135 | 150 | 7 | 0.17 | 47.5 | 0.5 | 13.90 | 12.90 |
| 65007-C-19 | R-02 | 171 | 186 | 4 | 0.20 | 28.4 | 0.3 | 15.00 | 16.50 |
| 65007-C-20 | R-02 | 401 | 416 | 16 | 0.14 | 30.5 | 0.3 | 15.60 | 16.30 |
| 65007-C-21 | R-03 | 184 | 200 | 6 | 0.21 | 25.0 | 0.3 | 17.60 | 17.40 |
| 65007-C-22 | R-03 | 327 | 343 | 10 | 0.30 | 25.0 | 0.3 | 20.70 | 21.90 |
| 65007-C-23 | R-12 | 221 | 238 | 7 | 0.10 | 56.7 | 0.6 | 10.90 | 11.30 |
| 65007-C-24 | R-12 | 270 | 287 | 9 | 0.03 | 47.2 | 0.5 | 13.70 | 14.20 |
| 65007-C-25 | R-12 | 320 | 336 | 12 | 0.07 | 33.0 | 0.3 | 15.20 | 15.10 |
| 65007-C-26 | R-12 | 369 | 385 | 14 | 0.06 | 35.8 | 0.4 | 16.10 | 15.90 |

*Closing the Screen Size

The 80$^{th}$ percentiles, median, and average hardness for the oxide and sulfide samples are summarized in Table 13-6 below.

**Table 13-6: Distribution Values for Comminution Parameters**

| Type | Parameter | 80th Percentile | Median | Average |
|---|---|---|---|---|
| Oxide | Axb | 38.2 | 71.8 | |
| | BWi, kWh/t | 12.8 | 11.0 | 10.8 |
| | RWi, kWh/t | 14.3 | 10.9 | 11.5 |
| | Ai, g | 0.108 | 0.071 | 0.075 |
| Sulfide | Axb | 30.5 | 36.0 | |
| | BWi, kWh/t | 15.6 | 14.0 | 13.8 |
| | RWi, kWh/t | 16.5 | 14.7 | 14.7 |
| | Ai, g | 0.233 | 0.144 | 0.162 |

Based on the median or average values, oxide materials are considered soft, and sulfide materials are considered hard for SAG milling. Rod mill and ball mill median or average values indicate medium hardness, with oxides being softer by about 3 kWh/t. Note that for Axb, the 80$^{th}$ percentile hardness was taken from the 20$^{th}$ percentile value of Axb because of the reverse relationship between hardness and Axb.

## 13.4 FLOTATION TESTS

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

Sequential flotation of sulfide and oxide copper was conducted 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 hydroxamates, to float oxides. This scheme is identified in this Technical 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 (with low concentrate grades). 'Phase 2 Oxide' or 'LCB' schemes were developed, 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 flotation test work programme 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.

### 13.4.1 Oxide Composite Flotation

Details of the oxide composite flotation tests are not 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).

### 13.4.2 Sulfide Composite Flotation

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

#### 13.4.2.1 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 Aero 3302 (xanthate allyl ester) and MIBC (methyl isobutyl carbinol) were used as collector and frother respectively. The results are shown in Figure 13-3 below.

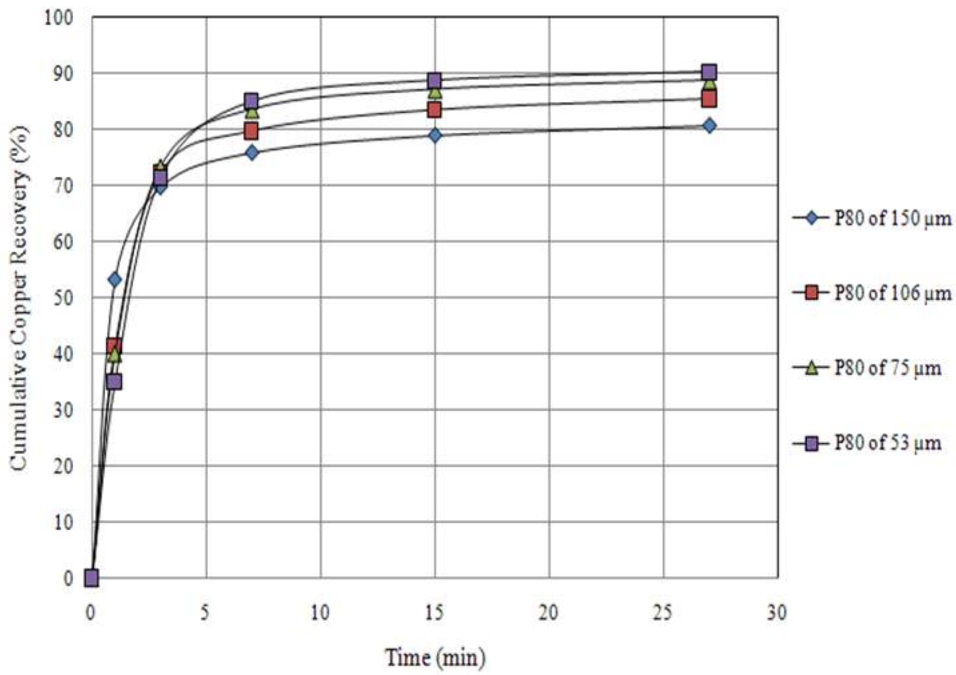



**Figure 13-3: Effect of Grind Size on Recovery of Total Copper from Sulfide Composite (AMEC, Mar 2012).**

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.

#### 13.4.2.2 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.

| | |
|---|---|
| Potassium amyl xanthate | PAX, |
| Sodium isobutyl xanthate | SIBX, |
| Sodium mercaptobenzothiazole | Aero 404 or A404, and |
| Xanthate allyl ester | Aero 3302 or A3302. |

The results are shown in Figure 13-4 and Figure 13-5 below. They demonstrate that the PAX and SIBX collectors yielded the highest total copper and mass recoveries. Not shown are gold recoveries, which were also highest with PAX and SIBX.

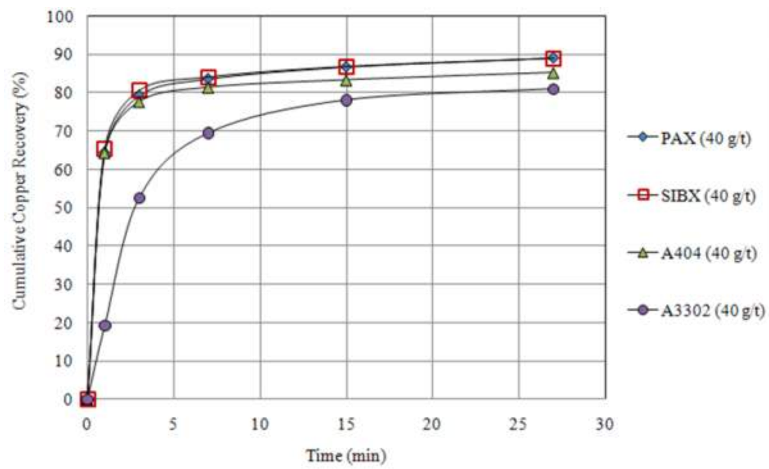



**Figure 13-4: Effect of Collector Type of Cu Recovery (AMEC, Mar 2012)**

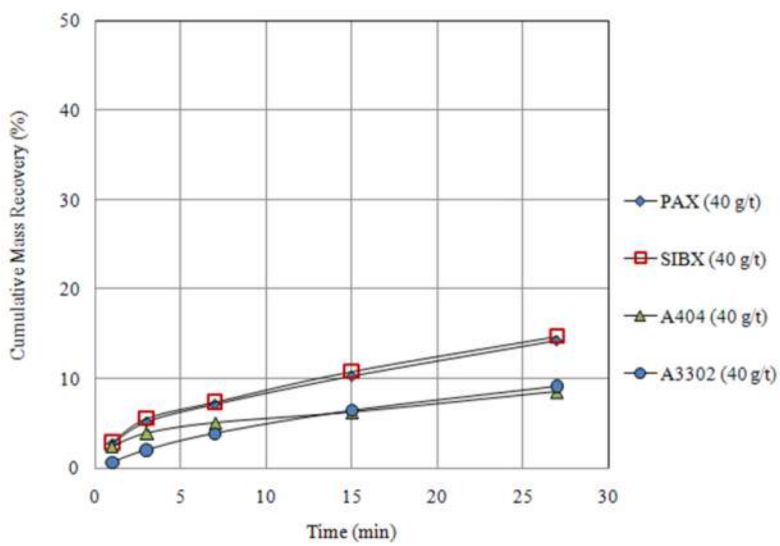



**Figure 13-5: Effect of Collector Type on Mass Recovery (AMEC, Mar 2012)**

***QP Comment:***

*PAX and SIBX are strong collectors that may not be selective against pyrite. The kinetics and recovery with Aero 404 are not too different from PAX and SIBX but had a lower mass pull (therefore, higher concentrate grade). Operations should consider more selective reagents such as dithiophosphates (for example, Aero 3477), coupled with Aero 3418 if it enhances gold recovery.*

#### 13.4.2.3 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, namely: 10, 20, 30, and 40 g/t. The results of the tests are shown in Figure 13-6 below.

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 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.

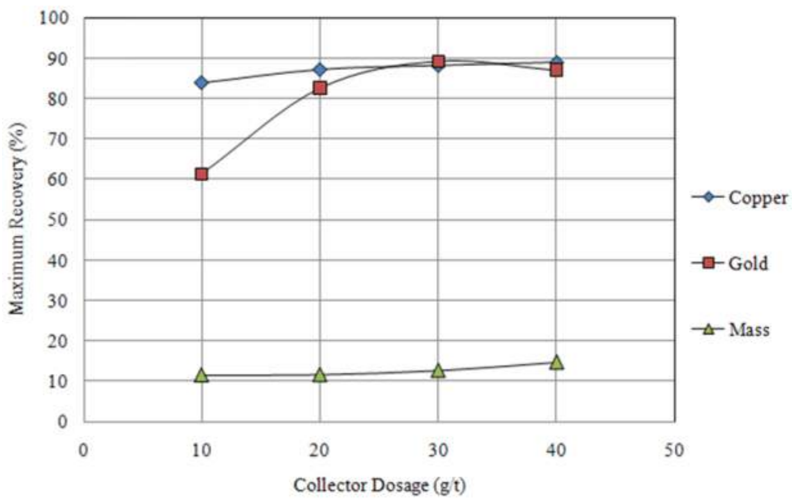



**Figure 13-6: Effect of Collector Dosage on Cu, Au, and Mass Recovery (AMEC, Mar 2012)**

***QP Comment:***
*From the plot, the optimum SIBX dosage is 30 g/t where gold recovery peaks, and with only a slight increase in Cu recovery at 40 g/t. In industrial scale, the reagent dosages end up being lower than the laboratory dosage due to recycling of reagents in process water reclaimed from the tailing thickener and the tailing storage facility.*

The results of flotation kinetics test at the four levels of SIBX addition are shown in Figure 13-7. They show that the kinetics and ultimate recoveries are similar, except for the g/t test, which had slightly slower kinetics and Cu recovery.

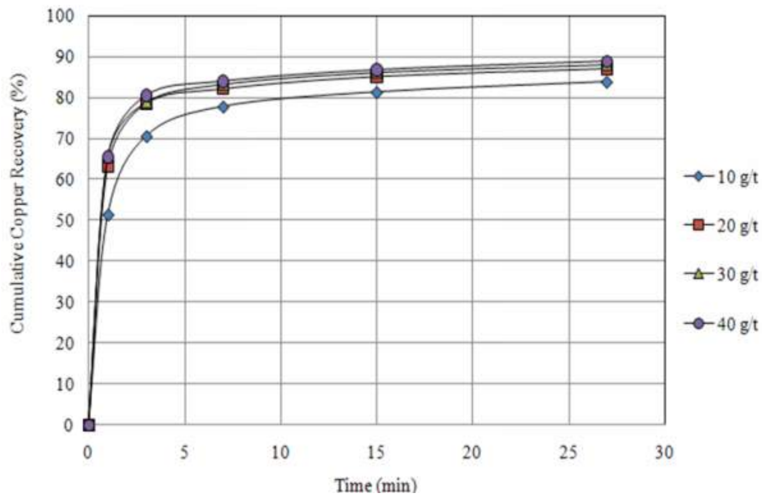



**Figure 13-7: Copper Recovery Kinetics at Different SIBX Additions (AMEC, Mar 2012)**

13.4.2.4 pH

Various pH values were tested to investigate the effect of pH on the flotation performance of the Kingking sulfide dominant composite. The pH values tested were natural (tap water), 9, 10, and 11 with 20 g/t SIBX addition.

Kinetics results and maximum recoveries for copper were very similar for pH 9, 10, and 11 as shown in Figure 13-8. Based on this, the AMEC Metallurgical Testwork Report recommended that a pH of 9 be used for rougher flotation. It pointed out 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 recoveries.

The recovery of gold, however, painted a different picture. Figure 13-9 clearly shows that gold recovery peaks at pH 10 and decreases at pH 11. The pH chosen should then have been 10 for optimum recovery.

In addition, a higher pH will reject pyrite better. Mineralogical studies on this ore report the presence of pyrite. One study (Pontifex, 2010) estimated 2 to 7 % $FeS_2$ compared to 1 to 2% $CuFeS_2$ for Year 1, Years 2-3, and Years 4-5 composites.

The decrease in gold recovery at pH 11 is consistent with the known depression effect of lime on gold flotation.

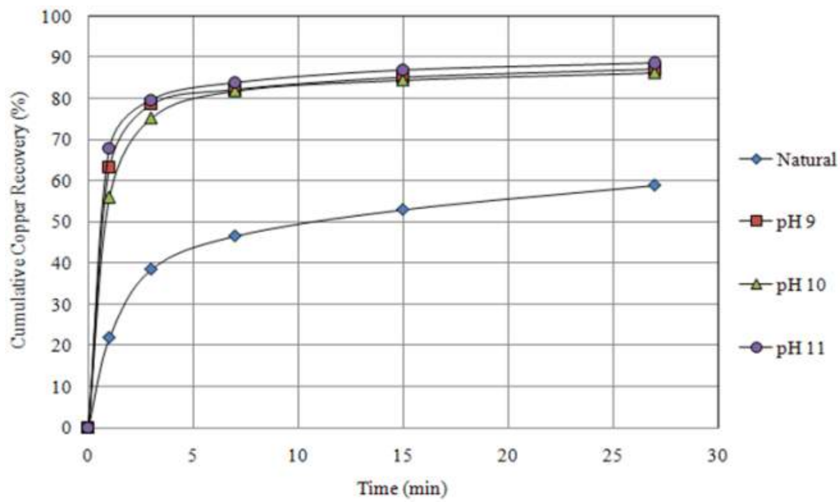



**Figure 13-8: Rougher Flotation Kinetics at Different pH Values (AMEC, Mar 2012)**

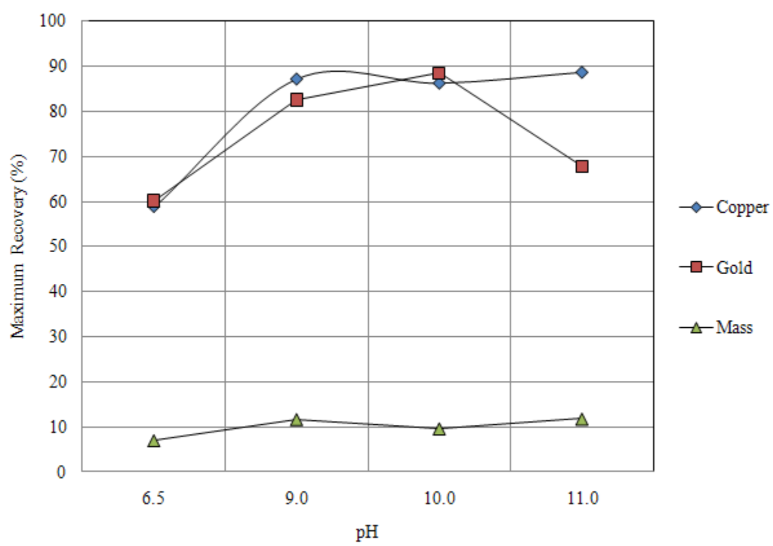



**Figure 13-9: Effect of pH on Copper Recovery (AMEC, Mar 2012)**

#### 13.4.2.5 Cleaner Stage Optimization

This set of tests to optimize cleaner flotation utilized the 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 (1) cleaner flotation pH, concentrate regrind particle size, collector addition, addition of a cleaner scavenger stage, and addition of a re-cleaner stage.

Cleaner pH: 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 and recovery. The results are presented in Table 13-7 and Figure 13-10. No major improvement in the grade of the final copper concentrate was observed by changing the pH. However, the enrichment ratio for copper is highest at pH 10.

**Table 13-7: Effect of pH on Cleaner Flotation**

| pH in Cleaning Stage | Recovery in Cleaning Stage (%) | | | Grade in Cleaning Stage | |
|---|---|---|---|---|---|
| | Mass | Copper | Gold | Copper (%) | Gold (ppm) |
| 9 | 3.8 | 91.4 | 90.4 | 6.4 | 8.33 |
| 10 | 4.4 | 91.8 | 89.8 | 7.57 | 11.4 |
| 11 | 4.8 | 92.7 | 89.4 | 6.52 | 7.38 |

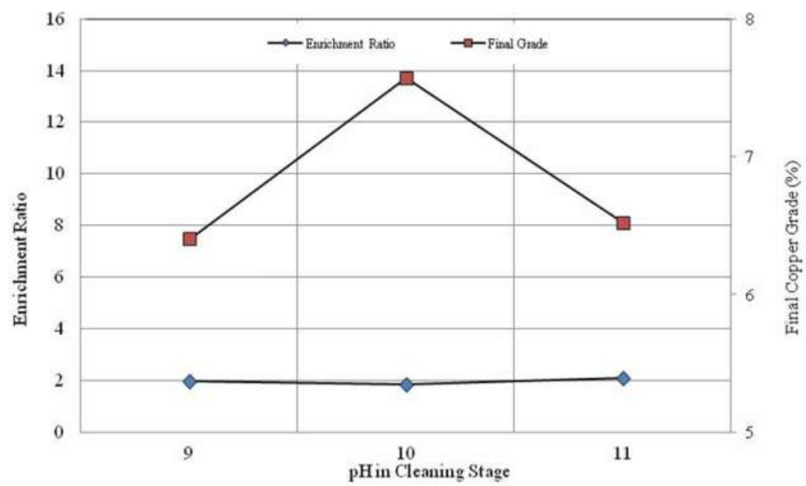



**Figure 13-10: Enrichment Ratios at Different Cleaner pH Values (AMEC, Mar 2012)**

Recovery v concentrate grade curves also indicate the pH 10 is the best pH for cleaner flotation, under the conditions used in the tests. This is illustrated by the plots in Figure 13-11.

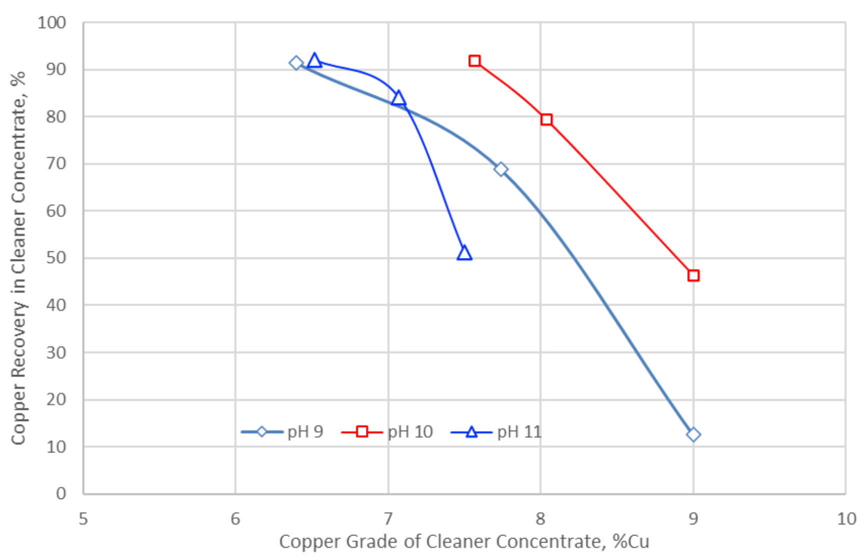



**Figure 13-11: Recovery v Concentrate Grade at Different Cleaner Flotation pHs (AMEC, Mar 2012)**

Concentrate Regrind: Two tests were performed to investigate the effect of two re-grind sizes, namely $P_{80}$ of 20 µm and $P_{100}$ of 20 µm on the cleaner flotation performance on a sulfide dominant composite. SIBX was added at 5 g/t with a total flotation time of 22 minutes. The results are shown in Table 13-8 and Figure 13-12.

The results indicate that a finer re-grind improves the cleaner concentrate copper grade but at a lower copper recovery.

**Table 13-8: Effect of Grind Size on Cleaner Flotation, Sulfide Dominant Composite**

| Re-grind Size | Recovery in Cleaning Stage (%) | | | Grade in Cleaning Stage | |
|---|---|---|---|---|---|
| | Mass | Copper | Gold | Copper (%) | Gold (ppm) |
| $P_{80}$ of 20 µm | 28.8 | 89.0 | 89.5 | 8.8 | 13.0 |
| $P_{100}$ of 20 µm | 9.9 | 47.8 | 41.4 | 15.1 | 20.6 |

Figure 13-12 shows that cleaner flotation at $P_{100}$ = 20 µm is much slower than at $P_{80}$ =20 µm. Unfortunately, the side distribution of the finer ground material was not provided, so there is no way to determine how fine the grind really was (that is, $P_{100}$ = 10 µm is the same as $P_{100}$ = 20 µm).

Ultimately, when the plant is built and operated, the fineness of regrind will have to be determined then because (a) plant grinding is very different from laboratory grinding because of the use of hydrocyclones in the plant, and (b) the hydrodynamics in a plant-scale flotation cell is different from the laboratory bench top cell, which have much higher agitation intensity. It is possible that regrind will not be necessary at all even if it is indicated in the laboratory.

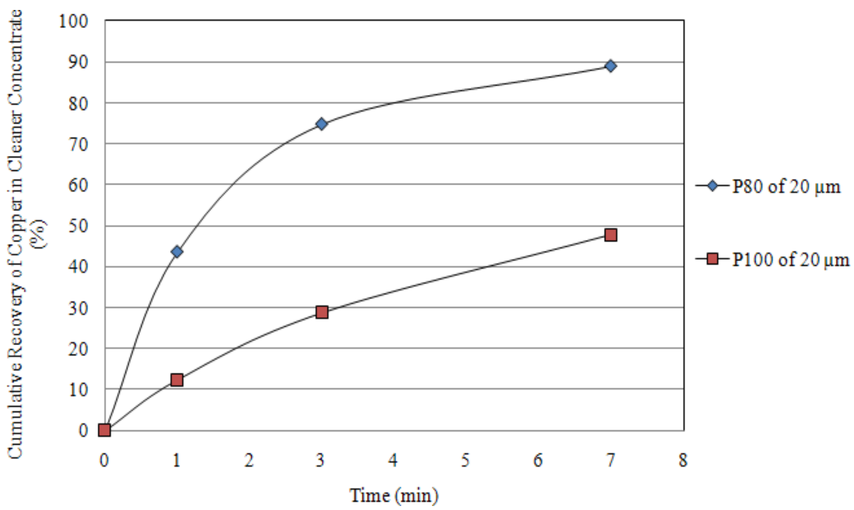



**Figure 13-12: Cleaner Flotation Kinetics at Two Regrind Sizes, Sulfide Dominant Composite (AMEC, Mar 2012)**

Collector Dosage to Cleaner: An additional cleaner test was performed to investigate the effect of collector addition in the cleaner stage on flotation performance of the Kingking 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 the addition of collector to the conditioner has negligible effect on the grade of the concentrate.

**Table 13-9: Effect of SIBX Addition Cleaner Flotation**

| SIBX in Cleaning Stage, g/t | Recovery, % | | | Concentrate Grade | |
|---|---|---|---|---|---|
| | Mass | Copper | Gold | Copper, % | Gold, g/t |
| 0 | 12.9 | 52.9 | 40.6 | 15.5 | 20.3 |
| 5 | 10.1 | 47.8 | 41.4 | 15.1 | 20.6 |

<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 Kingking 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 Cu and Au grade of the scavenger concentrate were low but comparable to the feed grades to the first cleaner stage. Concentrate from the cleaner scavenger flotation would then be recycled to first cleaner feed, while the tailing can report to final mill tails.

<u>The Addition of a Recleaning Stage</u>: 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. With two stages of recleaning, concentrate grades of 26 % Cu and 40 g/t Au were obtained.

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

***QP Comment:***
*Unfortunately, the grades and recoveries reported in the AMEC metallurgical report for recleaning tests could not be verified by the AMDEL flotation-results tables in the appendix.*

### 13.4.3 Locked Cycle Flotation Tests

Locked cycle flotation tests were conducted on the following sulfide dominant composites:

- Sulfide Dominant
- Projected Years 4 & 5 in the Life of Mine
- Projected Years 6 through 10 in the Life of Mine

Oxide dominant LOM and Years 2 & 3 composites were also tested but the results are not included here because the oxide-scheme flotation will not be implemented. However, the results can be found in the AMEC metallurgical report.

Each locked cycle test was conducted under the optimal conditions determined from the previous flotation test work. The test scheme, represented in Figure 13-13, 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 returned to the regrind stage. Tailing from the 2nd and 3rd Cleaner stages were returned to their corresponding previous stages. The locked cycles tests were run for six cycles.

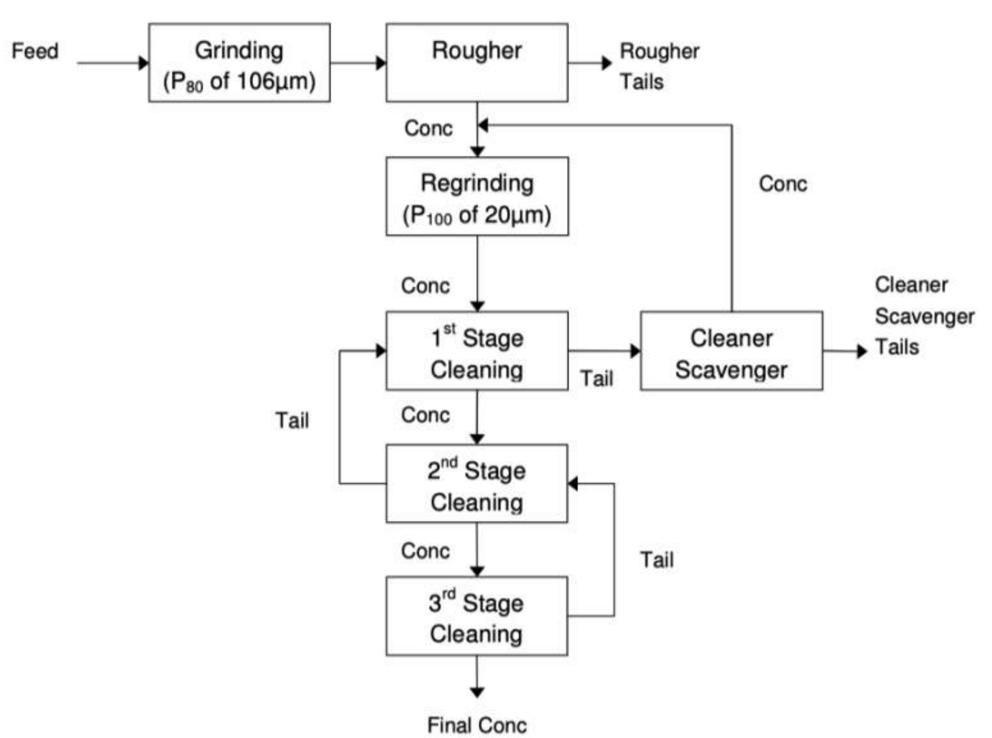



**Figure 13-13: Locked-Cycle Test Schematic**

The locked cycle tests were conducted flotation conditions listed in Table 13-10 below.

**Table 13-10: Flotation Conditions Used for Locked-Cycles Tests**

| Stage | Time, min | SIBX Dosage, g/t | Grind Size, µm | pH |
|---|---|---|---|---|
| Grinding | | | $P_{80}$ of 106 | 9 |
| Conditioning | 2 | | | |
| Rougher Stage | 12 | 25 | | 9 |
| Re-Grinding | | | $P_{100}$ of 20 | |
| Cleaning Stage | 10.5 | 5 | | 9 |
| Cleaning Scavenger | 10.5 | 1 | | |
| Re-Cleaning 1 | 10.5 | 5 | | |
| Re-Cleaning 2 | 10.5 | 2 | | |

The results of the locked cycle test are summarized in Table 13-11, showing Cycle 6 results.

**Table 13-11: Steady State Results of Locked Cycle Test Work**

| Composite | Head Grade | | 3rd Stage Conc Grade | | Overall Recovery, % | |
|---|---|---|---|---|---|---|
| | % Cu | g/t Au | % Cu | ppm Au | Cu | Au |
| Sulfide Dominant | 0.42 | 0.62 | 24.5 | 39.4 | 79.8 | 47.2 |
| Year 4/5 | 0.36 | 1.2 | 20.5 | 87 | 72.7 | 66.0 |
| Year 6/10 | 0.52 | 1.1 | 30.5 | 45 | 78.6 | 75.6 |

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 aluminum and fluorine in the Years 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 Years 4/5 LOM and Years 6/10 LOM composites.

Gold represented a payable by-product in all concentrates achieving grades of about 40 g/t or higher.

***QP Comments:***
*The overall recovery for all three composites were less than 80% for Cu and very low for gold, despite the use of more than 30 g/t SIBX. Figure 13-6 shows that copper and gold recoveries close to 90% can be attained. It appears that the final concentrate grades in the locked-cycle tests were attained at the expense of recovery. The takeaway from these results are as follows: (a) the 20 g/t SIBX dosage is too low, (b) SIBX is not the best reagent for this ore, and (c) the flotation should have been conducted at pH 10 instead of pH 9.*

### 13.4.4 Flotation Variability Tests

Flotation roughing tests were conducted to estimate the impact of the variability of Kingking samples on 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.

#### 13.4.4.1 Variation in Rougher Lime Consumption

Figure 13-11 shows the individual rougher lime consumption to achieve the targeted pH for sulfide dominant variability samples. The mean consumption at pH 10 was 1.7 kg/t and lower at pH 9 at 1 kg/t.

**Table 13-12: Flotation Variability Test – Variation in Rougher Lime Consumption**

| Sample ID | Ore Type | Target pH | Lime, kg/t |
|---|---|---|---|
| F3 | Sulfide | 10 | 1.47 |
| F5 | Sulfide | 10 | 2.38 |
| F9 | Sulfide | 10 | 1.06 |
| F18 | Sulfide | 10 | 1.93 |
| F25 | Sulfide | 10 | 1.66 |
| **pH 10 Mean Consumption** | | | **1.70** |
| **pH 10 Consumption Std Deviation** | | | **0.49** |
| F10 | Sulfide | 9 | 1.61 |
| F11 | Sulfide | 9 | 1.75 |
| F16 | Sulfide | 9 | 0.70 |
| F19 | Sulfide | 9 | 0.70 |
| F20 | Sulfide | 9 | 0.76 |
| F21 | Sulfide | 9 | 0.71 |
| F22 | Sulfide | 9 | 0.71 |
| F23 | Sulfide | 9 | 0.70 |
| F24 | Sulfide | 9 | 1.81 |
| F26 | Sulfide | 9 | 11.90* |
| F27 | Sulfide | 9 | 1.49 |
| F28 | Sulfide | 9 | 3.51 |
| F29 | Sulfide | 9 | 0.51 |
| F30 | Sulfide | 9 | 0.61 |
| F31 | Sulfide | 9 | 0.54 |
| F32 | Sulfide | 9 | 0.62 |
| F33 | Sulfide | 9 | 0.70 |
| F34 | Sulfide | 9 | 0.61 |
| F35 | Sulfide | 9 | 0.74 |
| F37 | Sulfide | 9 | 0.70 |
| F38 | Sulfide | 9 | 0.54 |
| **pH 9 Mean Consumption** | | | **1.00*** |
| **pH 9 Consumption Std Deviation** | | | **0.72*** |

*Outlier not included in calculations

The results of the variability flotation tests are summarized in Table 13-13. Cu recoveries for tests F12 through F15 and F36 were low due to the high copper oxide contents of the samples (18.5% or higher, in red italic font). F10 and F25 are outliers with poor recoveries primarily due to the low head grade, which was 0.05% Cu for both samples.

The average Cu recovery was 87.5% if the two outliers and the high-oxide samples are excluded. The average Au recovery was 63.5%, with no exclusions, and 72.3% with the same exclusions as with Cu.

**Table 13-13: Results of Variability Flotation Tests**

| Sample ID | Head Gr. | WAS Cu | Rougher Concentrate Grade | | Rougher Recovery, % | |
|---|---|---|---|---|---|---|
| | % Cu | % | % Cu | g/t Au | Cu | Au |
| | **Rougher Flotation at pH 10** | | | | | |
| F3 | 1.85 | 3.4% | 5.03 | 8.87 | 92.7 | 93.2 |
| F5 | 0.42 | 14.4% | 0.47 | 0.13 | 66.1 | 75.6 |
| F9 | 0.54 | 8.7% | 1.73 | 12.21 | 96.6 | 95.2 |
| F18 | 0.33 | 10.0% | 0.58 | 1.64 | 96.6 | 97.3 |
| F25 | 0.05 | 11.0% | 0.10 | 0.17 | 68.0 | 65.9 |
| | **Rougher Flotation at pH 9** | | | | | |
| F10 | 0.05 | 8.0% | 0.11 | 0.45 | 20.4 | 34.0 |
| F11 | 0.52 | 8.5% | 2.84 | 1.10 | 79.2 | 70.6 |
| F12 | 0.27 | *18.5%* | 0.38 | 0.58 | *10.8* | 41.9 |
| F13 | 0.32 | *22.0%* | 0.41 | 0.48 | *7.9* | 25.7 |
| F14 | 0.50 | *28.0%* | 0.75 | 2.86 | *11.3* | 50.1 |
| F15 | 1.6 | *25.9%* | 8.30 | 17.83 | *40.0* | 69.5 |
| F16 | 0.12 | 6.3% | 1.06 | 0.44 | 78.3 | 56.2 |
| F17 | 0.54 | 1.3% | 4.63 | 19.32 | 82.2 | 88.4 |
| F19 | 0.18 | 1.9% | 1.74 | 0.82 | 87.5 | 73.4 |
| F20 | 0.48 | 7.4% | 9.61 | 5.39 | 88.4 | 85.3 |
| F21 | 0.21 | 6.4% | 1.76 | 2.52 | 82.3 | 29.5 |
| F22 | 0.31 | 12.4% | 3.02 | 10.19 | 87.4 | 24.6 |
| F23 | 0.16 | 1.3% | 1.11 | 0.95 | 83.1 | 67.8 |
| F36 | 0.45 | *22.4%* | 0.74 | 1.59 | *7.8* | 40.0 |
| F37 | 1.70 | 4.2% | 10.01 | 5.86 | 95.1 | 85.8 |

### 13.4.5 Recommended Sulfide Flotation Conditions

The sulfide composite flotation tests collectively produced the recommended flotation conditions at the start of operations, and these are summarized in Table 13-14. The conditions need to be optimized in the actual plant to adjust for (a) reagents recycled in the reclaim water and (b) the finer grind of copper-bearing minerals (i.e., rougher concentrate) resulting from the sizing cyclones.

**Table 13-14: Composite Optimum Conditions**

| Stage | *Collector Dosage, g/t | Frother Dosage, g/t | Grind Size, µm | pH |
|---|---|---|---|---|
| Grinding | | | $P_{80}$ of 106 | 10 |
| Conditioning | | | | |
| Rougher Stage | 25 to 30 | 15 to 30 | | 10 |
| Re-Grinding | | | $P_{80}$ or $P_{100}$ of 20 | |
| Cleaning Stages | TBD* | | | 10 |

*To be determined at the start of operations.

While the collector used in these tests was SIBX, this QP recommends operations to perform laboratory tests or plant trials using more selective dithiphosphates, for example Aero 3477, and possibly Aero 3418 to enhance gold flotation.

With a more selective collector, it may be possible to attain concentrate grade with only two stages of cleaning instead of three, particularly if the final flotation cleaning stage is implemented in a flotation column(s).

## 13.5 TAILING LEACH OPTION STUDY

The flotation tailing leach option was tested at AMDEL under the supervision of AMEC and reported in "*Kingking Copper-Gold Project Tails Leach Option Study Test Work Results"*, Document No. 65007-00000-21-002-003, AMEC, February 2012.

***QP Comment:***
*The following discussion is the QP's independent analysis of the test results presented in the AMEC report. A large percentage of the calculation results could not be reproduced because of the lack of documentation of the analytical and test procedures. The discussions presented here were therefore limited to results that could be verified by this QP.*

At the beginning of Section 13.4, 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 tailing leach study was carried out to investigate this concept.

A total of 12 variability flotation tailing samples were tested and were either sulfide tails produced from the second set of flotation test work or oxide concentrate and rougher tails combined from the previous sequential flotation test work. Five variability samples (F-10, F-12, F-13, F-14, and F-15) were obtained directly as sulfide tails from the second stages of sulfide flotation test work. Insufficient tails samples were generated from the sulfide flotation, so the remaining 7 sulfide tails variability samples (F-1, F-2, F-4, F-5, F-6, F-7, and F-18) were produced by combining oxide concentrate and rougher tails samples from the sequential flotation test work. Table 13-15 below summarizes the composition of the samples used for leach tests.

**Table 13-15: Composition of Flotation Tailing Leach Samples**

| Test No. | Variability Sample | Classification | Oxide Flotation Concentrate | Rougher Tails | Leach Feed, % Cu | Acid + CN soluble Cu, % |
|---|---|---|---|---|---|---|
| L.1 | F-01 | Oxide | 7.6% | 92.4% | 0.89% | 92.58% |
| L.2 | F-02 | Oxide | 8.5% | 91.5% | 0.47% | 86.81% |
| L.3 | F-04 | Oxide | 39.0% | 61.0% | 0.53% | 90.38% |
| L.4 | F-05 | Oxide | 51.3% | 48.7% | 0.39% | 46.15% |
| L.5 | F-06 | Oxide | 22.1% | 77.9% | 0.39% | 75.51% |
| L.6 | F-07 | Oxide | 64.4% | 35.6% | 0.20% | 70.00% |
| L.7 | F-10 | Oxide | 0.0% | 100% | 0.05% | 24.00% |
| L.8 | F-12 | Oxide | 0.0% | 100% | 0.24% | 35.00% |
| L.9 | F-13 | Oxide | 0.0% | 100% | 0.30% | 60.67% |
| L.10 | F-14 | Oxide | 0.0% | 100% | 0.49% | 86.02% |
| L.11 | F-15 | Oxide | 0.0% | 100% | 1.01% | 95.05% |
| L.12 | F-18 | Mixed Oxide | 1.9% | 98.1% | 0.04% | 65.00% |

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.

The results of the tests are shown in Table 13-16 below. The leach kinetics plot indicates that after 6 hours, most samples had reached their peak dissolution. The results also show variability in the maximum recovery attained in the leach.

Tests L.7 and L.12 are outliers, both of which have low head grade of 0.05 and 0.04%, respectively, with L.7 containing only 24% acid-soluble copper.

**Table 13-16: Summary of Leach Test Results**

| Test No. | Variability Sample | Head Grade, % Cu | % Acid + CN soluble Cu, % | Leach Recovery, % |
|---|---|---|---|---|
| L.1 | F-01 | 0.89% | 92.58% | 94.38% |
| L.2 | F-02 | 0.47% | 86.81% | 89.36% |
| L.3 | F-04 | 0.53% | 90.38% | 92.45% |
| L.4 | F-05 | 0.39% | 46.15% | 74.36% |
| L.5 | F-06 | 0.39% | 75.51% | 84.62% |
| L.6 | F-07 | 0.20% | 70.00% | 80.00% |
| L.7 | F-10 | 0.05% | 24.00% | 20.00% |
| L.8 | F-12 | 0.24% | 35.00% | 70.83% |
| L.9 | F-13 | 0.30% | 60.67% | 73.33% |
| L.10 | F-14 | 0.49% | 86.02% | 89.80% |
| L.11 | F-15 | 1.01% | 95.05% | 94.06% |
| L.12 | F-18 | 0.04% | 65.00% | 50.00% |

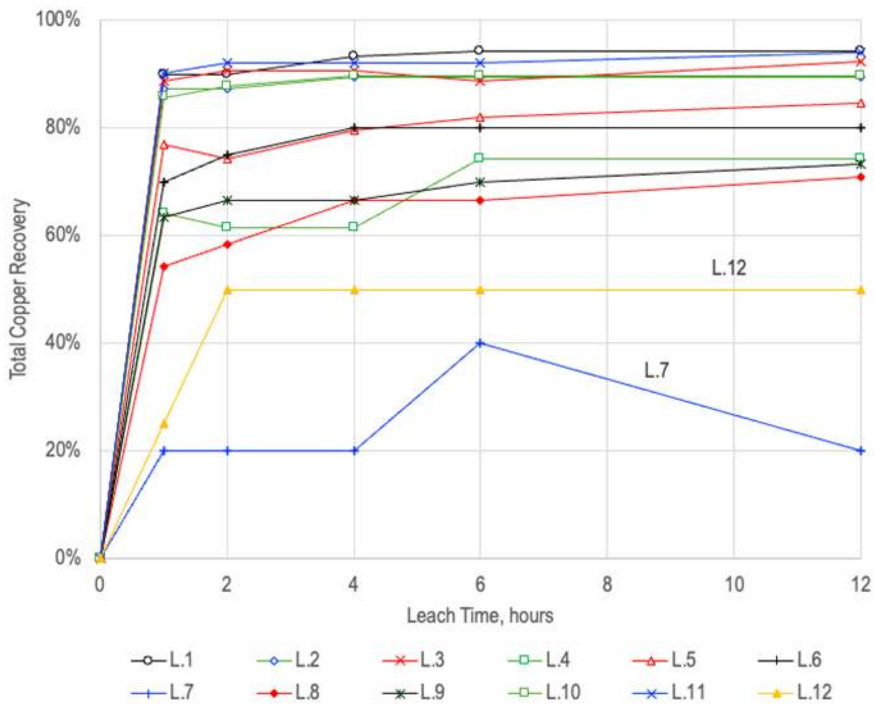



**Figure 13-14: Total Copper Dissolution v Time (AMEC, Feb 2012)**

The varying degrees of leach recoveries are related to the soluble Cu assay as determined by weak-acid soluble and CN-soluble copper determinations. Figure 13-15 below shows a linear relationship between copper recovery and soluble Cu assays, excluding the outliers L.7 and L.12.

The maximum recoveries obtained was 94% from samples that contained almost completely acid-soluble copper. At the lower soluble copper contents, the assays underestimate the recovery (e.g., about 70% recovery at 40% soluble copper assay). This suggests that the assay procedures to estimate acid-soluble and cyanide soluble copper are deficient – and this was observed during the column leach tests.

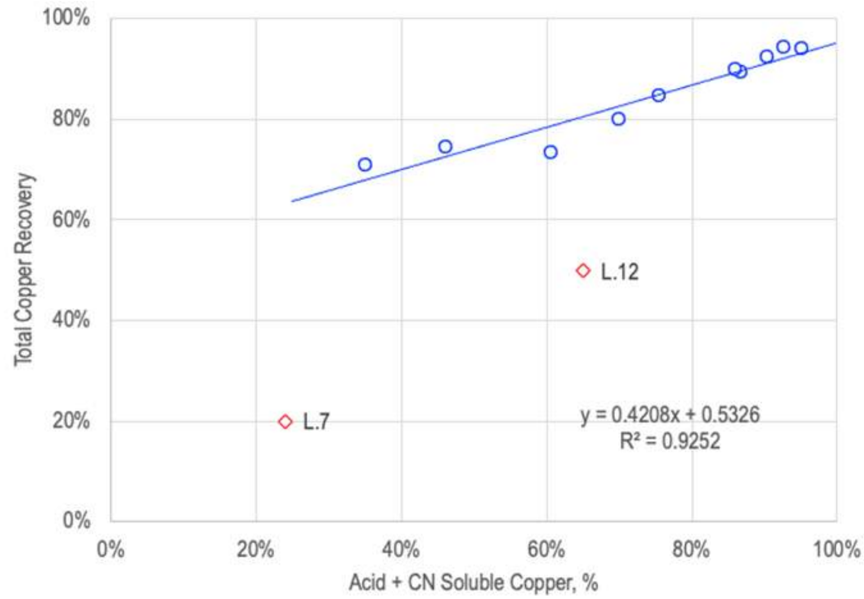



**Figure 13-15: Copper Leach Recovery v Soluble Cu Assay (FLC, 2024, based on data in AMEC, Feb 2012)**

Subsequent examination of acid consumption levels for Kingking 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 Kingking samples was developed after a review of test results from a previous investigation by Metcon (Kingking 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 Kingking oxide ore representing oxide ore types to be mined over the first thirteen years of the life of mine and in Years 15, 18, 19, 24, and 25. The tests were conducted in six-inch diameter six-foot high columns using acid solution based on actual 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 were 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 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 prepared for assay. The screen fractions were +3/4 in., 3/4 x 5/8 in., 5/8 x 1 in., 1/2 x 3/8 in., 3/8 x 1/4 in., and minus 1/4 in. Each of the screen fractions was crushed to -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 more than 100 percent suggesting that 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 13-17 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 13-17: 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 13-17 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

Fifty-kilogram portions of the feed were split from the drill core samples. Each sample was placed on a large sheet of plastic and the 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 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 fines 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 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 tonne of ore and per pound of copper recovered were also calculated for each PLS sample and cumulatively for the test.

### 13.6.3 Column 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 13-16 to Figure 13-21 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.

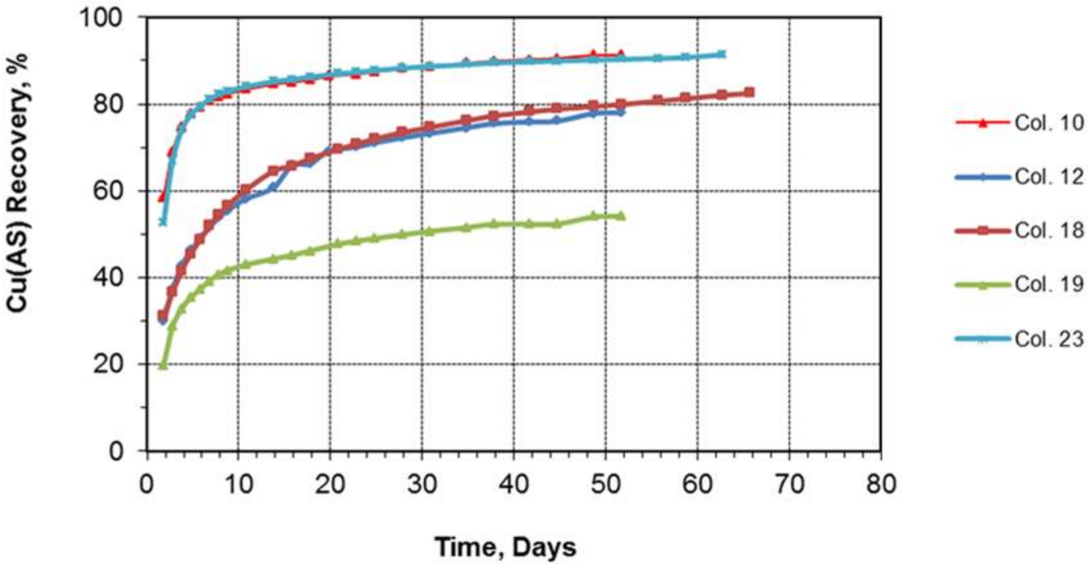



**Figure 13-16: Recovery/Time Plot, Year 0 (Leach, Inc., 2012)**

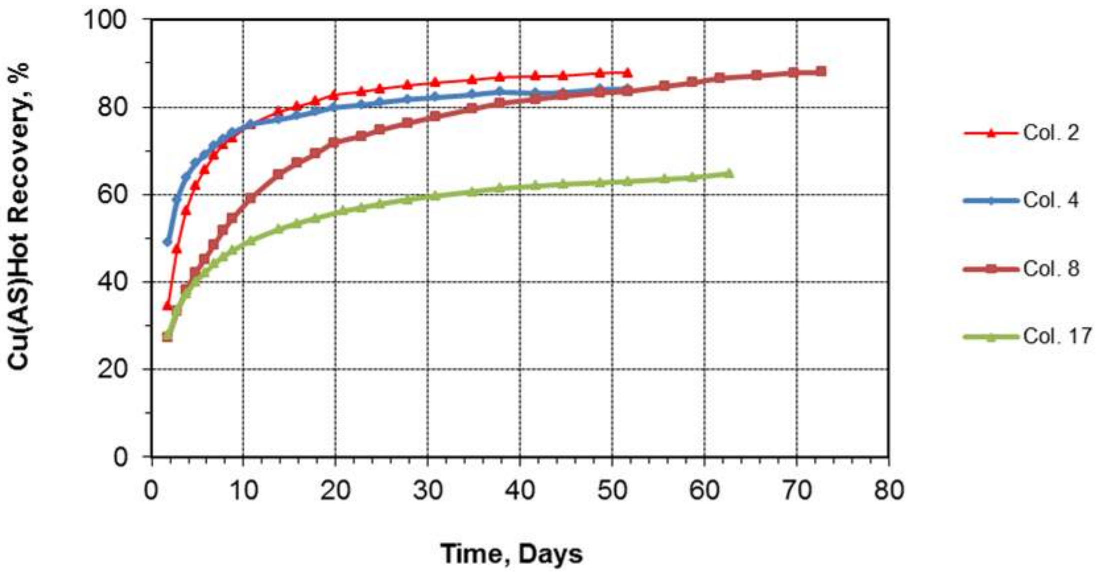



**Figure 13-17: Recovery/Time Plot, Year 1 (Leach, Inc., 2012)**

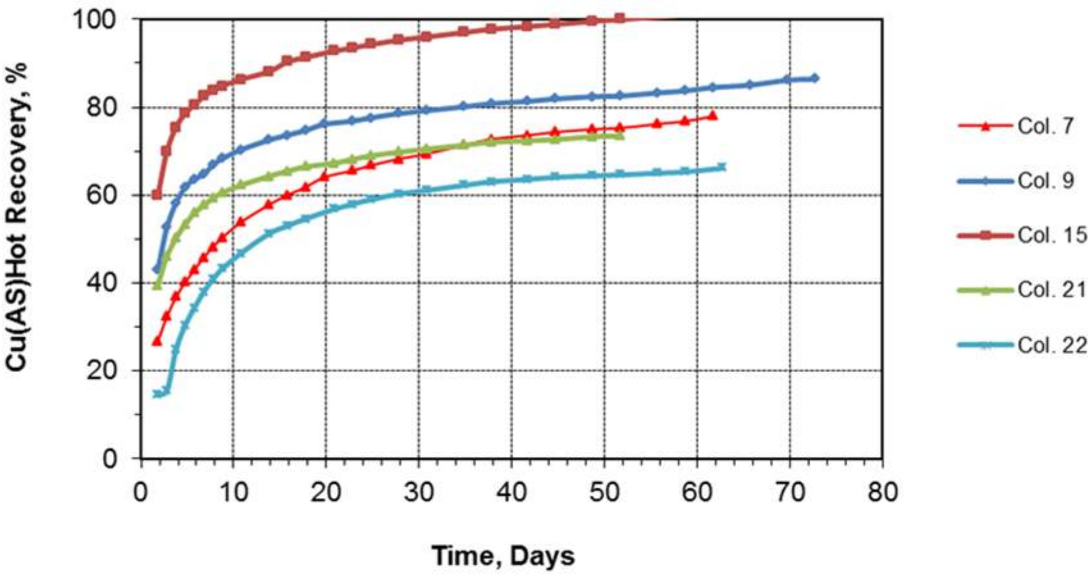



**Figure 13-18: Recovery/Time Plot, Year 2 (Leach, Inc., 2012)**

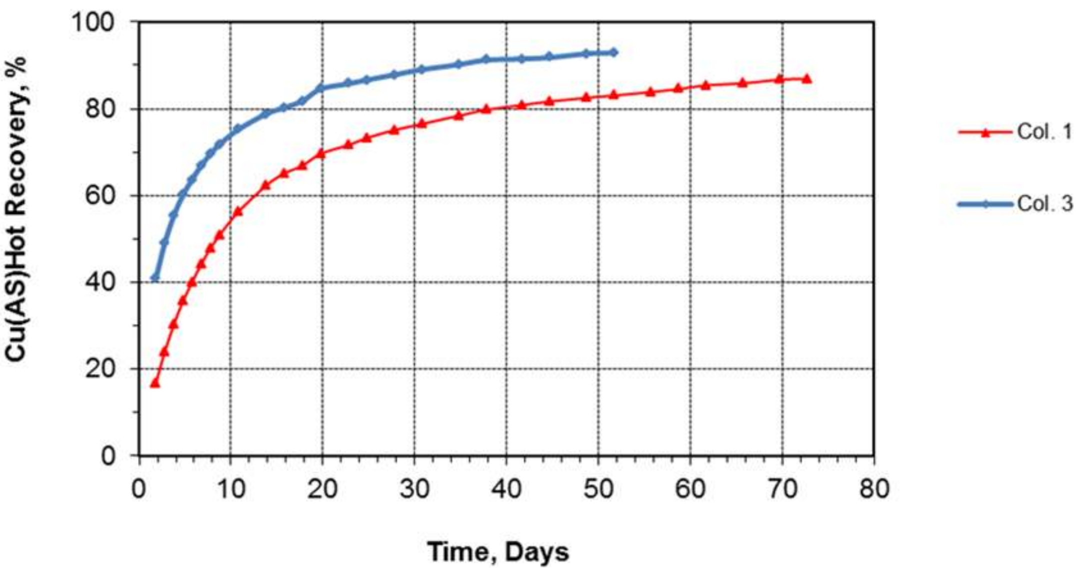



**Figure 13-19: Recovery/Time Plot, Year 3 (Leach, Inc., 2012)**

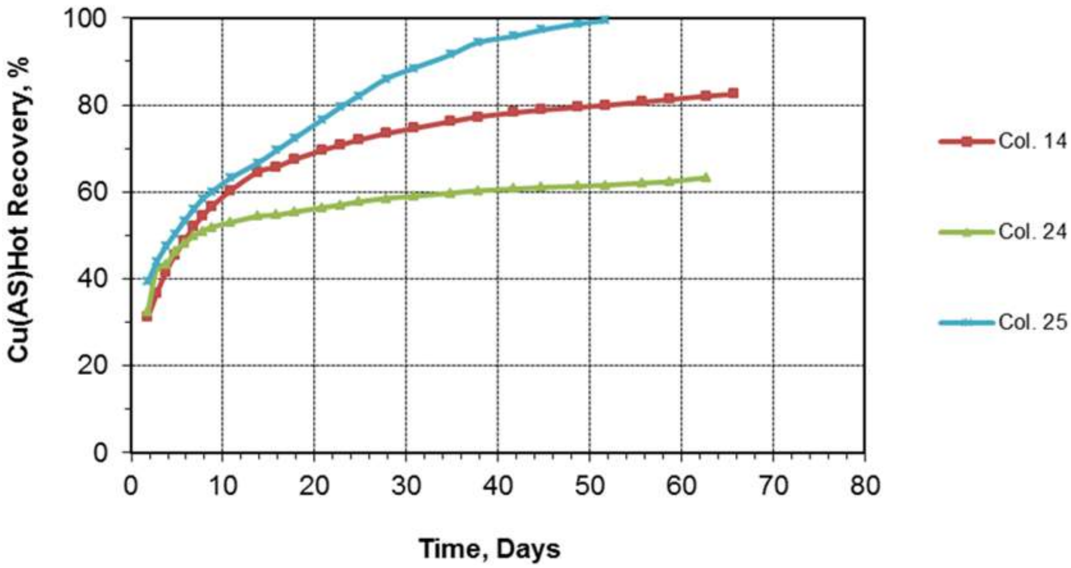



**Figure 13-20: Recovery/Time Plot, Year 4 (Leach, Inc., 2012)**

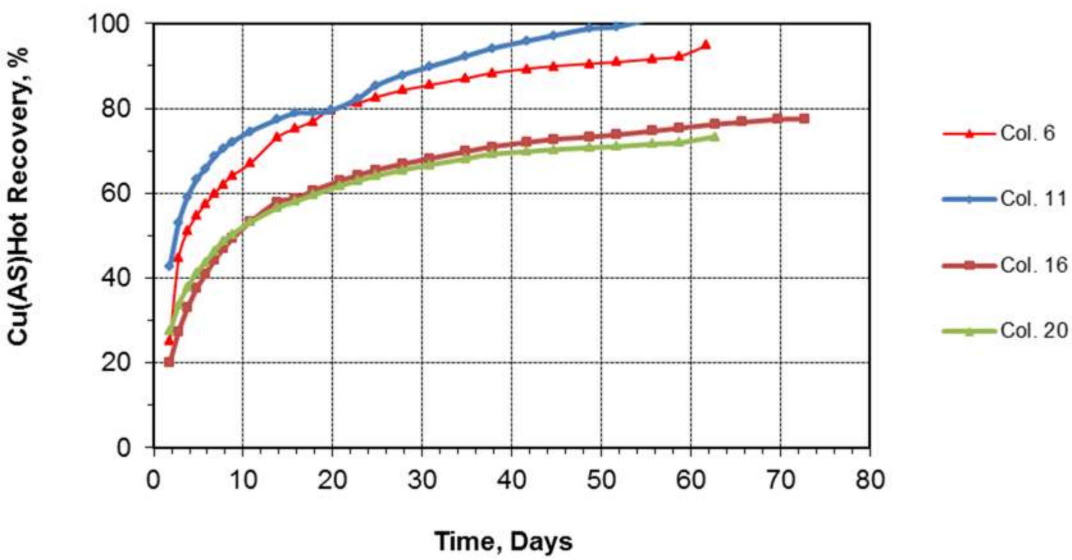



**Figure 13-21: Recovery/Time Plot, Year 5 & 6 (Leach, Inc., 2012)**

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 13-18 lists the recoveries from each size fraction for the 23-column tests.

The recovery by particle size suggests that all 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 13-18: Cu(total) Recovery Versus 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 13-19 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 13-19: 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 |

### 13.6.4 Column Leach Tests on ¾" Material

A subsequent set of column tests were performed on 22 of the 25 samples listed in Section 13.2.2 above (excluding KK-5, KK-13, and KK-25), comprising Phase 2 of the column leach testing program. The objectives were to determine the effects of a finer crush size, curing acid concentration and column height. Four column tests were performed on an oxide composite consisting of approximately equal portions of the 22 samples.

The results of the tests on the oxide composite are shown in Table 13-20 and Figure 13-22 below. The results were consistent with the results obtained in the Phase 1 testing program.

The ore for the current tests was crushed to minus ¾ inch instead of the minus 1 inch as is Phase 1. This resulted in better copper recoveries. On average, the recovery of acid soluble copper was greater than 90 percent, compared to an average of 79.4% in Phase 1. Acid consumptions of about 60 lbs/tonne can be expected.

Comparisons of the response of the ore to different cure acid dosages suggest that a minor improvement in recovery and rate of leaching can be achieved by varying the cure acid dosage.

**Table 13-20: Summary of Column Leach Test Results**

| Column No. | Leach Time, days | Height, m | Cure Dosage, lbs/t | Recovery, % | | Acid Consumption | |
|---|---|---|---|---|---|---|---|
| | | | | Cu(total) | Cu(AS) | lbs/lb Cu | Gangue, lbs/t |
| CL-01 (oxide ore) | 71 | 5.92 | 25 | 81.2 | 94.5 | 6.6 | 55.2 |
| CL-02 (oxide ore) | 57 | 1.84 | 25 | 85.0 | 98.0 | 6.6 | 64.1 |
| CL-03 (oxide ore) | 49 | 1.74 | 38 | 86.9 | 100.3 | 8.7 | 84.5 |
| CL-04 (oxide ore) | 61 | 1.74 | 17 | 84.3 | 97.2 | 11.2 | 107.7 |

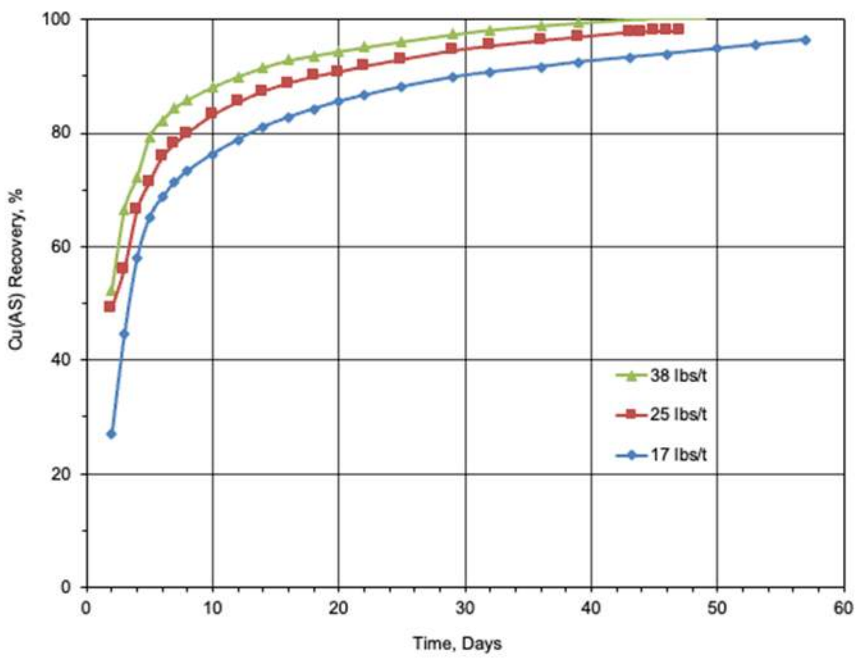



**Figure 13-22: Effect of Cure Dosage, Oxide Ore (Leach, Inc., 2013)**

Figure 13-23 shows that increasing the height of the leach column (lift height) from 6 feet to 20 feet in the laboratory column necessitates a doubling of the leach cycle time and results in about a 10 percent higher acid consumption per pound of copper recovery at a given copper recovery. Leach cycle time in the 20 ft column test of 70 days was sufficient but leach cycle times of 125 days or more should be expected in the commercial plant because of differences caused by the heap construction and the leach solution distribution by the irrigation system used.

For the ore sample tested, this results in a net acid consumption of about 6 lbs of acid per pound of copper recovered at an acid soluble copper recovery of 90 percent. Net acid consumption increased with increasing copper recovery.

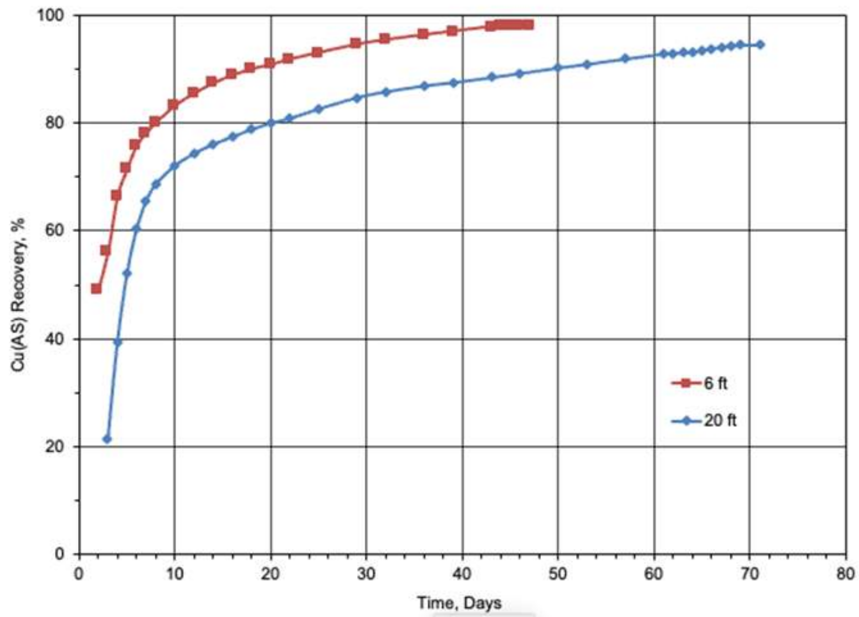



**Figure 13-23: Effect of Column Height, Oxide Ore (Leach, Inc., 2013)**

## 13.7 GOLD DEPORTMENT

A 2 kg charge of Kingking 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 to determine total liberated and locked gold.
- Mercury insoluble residues were separated into heavy liquid at SG = 3.32. The component with SG <3.32 was analyzed for gold as gold locked in silicates or carbonates.
- The component with SG >3.32 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 13-21.

**Table 13-21: 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.

## 13.8 ANCILLARY TESTS

### 13.8.1 Thickener Sizing Tests

Pocock Industrial (Salt Lake City) was commissioned to conduct solid-liquid separation (SLS) tests on Kingking 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 high-rate thickener design parameter recommendations shown in the following Table 13-22.

**Table 13-22: 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, to avoid phase disengagement problems in the downstream solvent extraction process. Flocculant consumption rates of 15 g/tonne (tailing) and 75 g/tonne (CCD circuit) are assumed based on direct operating experience with a copper beneficiation circuit similar to that planned for Kingking.

# 14 MINERAL RESOURCE ESTIMATES

## 14.1 MINERAL RESOURCES

Table 14-1 presents the Mineral Resources for the Kingking Project. The Mineral Resources include Mineral Resources amenable to milling and flotation concentration methods (mill material) and Mineral Resources amenable to heap leach recovery methods (leach material). The upper portion of the table presents the Mineral Resources for mill material. Measured and Indicated Mineral Resources amount to 1.03 billion tonnes at 0.24% total copper and 0.34 g/t gold for 5.55 billion pounds of contained copper and 11.1 million ounces contained gold. Inferred Mineral Resource is an additional 640.5 million tonnes at 0.20% total copper and 0.27 g/t gold for 2.85 billion pounds of contained copper and 5.46 million ounces of contained gold.

The middle portion of the table presents the Mineral Resource for heap leach material. Leach material is oxide/mixed dominant mineralization. Measured and Indicated Mineral Resources amount to 85.0 million tonnes at 0.23% total copper and 0.15% soluble copper and contained metal amounts to 430.0 million pounds of copper. Gold is not recovered in the heap leach process. Inferred Mineral Resource is an additional 34.7 million tonnes at 0.21% total copper and 0.12% soluble copper and contained metal amounts to 159.0 million pounds of copper.

The bottom portion of the table presents the Mineral Resource for combined mill and leach material for copper. Measured and Indicated Mineral Resources amount to 1.12 billion tonnes at 0.24% total copper for 5.98 billion pounds of contained copper. Inferred Mineral Resource is an additional 675.2 million tonnes at 0.20% total copper for 3.01 billion pounds of contained copper. The Mineral Resource for gold is as shown with mill resource since it will not be recovered for leach material.

The Mineral Resources are based on an updated block model developed by IMC during August 2023. The updated model incorporated about 9 additional holes that were drilled during 2011, but results were not available for incorporation into the June 2011 model.

The Measured, Indicated, and Inferred Mineral Resources reported herein are contained within a constraining pit shell to demonstrate "reasonable prospects for eventual economic extraction" to meet the definition of Mineral Resources in NI 43-101. Figure 14-1 shows the constraining pit shell that is based on Measured, Indicated, and Inferred Mineral Resource.

Also, the Mineral Resource is reported inclusive of the Mineral Reserve presented in Section 15.

**Table 14-1: Kingking Mineral Resource**

| Mineral Resource (Milling) | Tonnes Mt | NSR ($/t) | Tot Cu (%) | Sol Cu (%) | Gold (g/t) | Copper (Mlbs) | Gold (Koz) |
|---|---|---|---|---|---|---|---|
| **Measured Mineral Resource:** | **157.4** | **36.01** | **0.32** | **0.12** | **0.47** | **1,102** | **2,360** |
| Mixed Oxide/Sulfide | 56.3 | 51.34 | 0.46 | 0.26 | 0.62 | 570 | 1,115 |
| Sulfide | 101.1 | 27.47 | 0.24 | 0.03 | 0.38 | 533 | 1,245 |
| **Indicated Mineral Resource:** | **877.3** | **23.87** | **0.23** | **0.04** | **0.31** | **4,450** | **8,787** |
| Mixed Oxide/Sulfide | 67.7 | 38.29 | 0.33 | 0.18 | 0.51 | 488 | 1,109 |
| Sulfide | 809.5 | 22.66 | 0.22 | 0.03 | 0.30 | 3,962 | 7,678 |
| **Measured/Indicated Resource:** | **1,034.7** | **25.71** | **0.24** | **0.05** | **0.34** | **5,553** | **11,147** |
| Mixed Oxide/Sulfide | 124.1 | 44.21 | 0.39 | 0.22 | 0.56 | 1,058 | 2,224 |
| Sulfide | 910.6 | 23.19 | 0.22 | 0.03 | 0.30 | 4,495 | 8,923 |
| **Inferred Mineral Resource:** | **640.5** | **20.17** | **0.20** | **0.03** | **0.27** | **2,854** | **5,464** |
| Mixed Oxide/Sulfide | 19.6 | 30.82 | 0.24 | 0.12 | 0.50 | 103 | 313 |
| Sulfide | 620.9 | 19.83 | 0.20 | 0.02 | 0.26 | 2,751 | 5,151 |
| **Mineral Resource (Leaching)** | **Tonnes Mt** | **NSR ($/t)** | **Tot Cu (%)** | **Sol Cu (%)** | **Gold (g/t)** | **Copper (Mlbs)** | **Gold (Koz)** |
| Measured Mineral Resource | 39.3 | 16.28 | 0.25 | 0.18 | N.A. | 220 | N.A. |
| Indicated Mineral Resource | 45.7 | 12.74 | 0.21 | 0.13 | N.A | 210 | N.A. |
| **Measured/Indicated Resource** | **85.0** | **14.38** | **0.23** | **0.15** | **N.A.** | **430** | **N.A.** |
| Inferred Mineral Resource | 34.7 | 12.12 | 0.21 | 0.12 | N.A. | 159 | N.A. |
| **Copper Mineral Resource Milling and Leaching** | **Tonnes Mt** | **NSR ($/t)** | **Tot Cu (%)** | **Sol Cu (%)** | **Gold (g/t)** | **Copper (Mlbs)** | **Gold (Koz)** |
| Measured Mineral Resource | 196.7 | 32.07 | 0.30 | 0.13 | N.A. | 1,322 | N.A. |
| Indicated Mineral Resource | 923.0 | 23.32 | 0.23 | 0.04 | N.A. | 4,660 | N.A. |
| **Measured/Indicated Resource** | **1,119.7** | **24.85** | **0.24** | **0.06** | **N.A.** | **5,982** | **N.A.** |
| Inferred Mineral Resource | 675.2 | 19.75 | 0.20 | 0.03 | N.A. | 3,013 | N.A. |

Notes:

1. The Mineral Resources have an effective date of February 6, 2024, and the estimate was prepared using the definitions in CIM Definition Standards (10 May 2014).
2. All figures are rounded to reflect the relative accuracy of the estimate and therefore numbers may not appear to add precisely.
3. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
4. Mineral Resources are based on prices of $3.75/lb copper and $1,800/oz gold.
5. Mineral Resources for leach material are based on an NSR Cut-off of $3.26/t. For leach material, NSR($/t) = $74.85 x copper (%) x recovery where recovery is variable but averages 82.1%.
6. The NSR value due to mill copper is based on a flotation component (concentrates) and an agitation tail leach component (cathode). For flotation, NSR($/t) = $70.10 x copper (%) x recovery where recovery is variable but averages 36.5% for oxides and 67.9% for sulfides. For tails agitation, NSR($/t) = $74.85 x copper (%) x recovery where recovery averages 55.6% for oxide and 14.0% for sulfide.
7. The NSR value for mill gold is based on a flotation component (concentrates) and a gravity component. For flotation, NSR($/t) = $52.08 x gold (g/t) x recovery where recovery averages 38.6% for oxides and 55.1% for sulfides. For gravity, NSR($/t) = $54.31 x gold (g/t) x recovery where recovery averages 23.5% for oxides and 19.3% for sulfides.
8. Total NSR for milling is the sum of the copper and gold components. The NSR calculations account for smelter/refinery treatment charges and payables.
9. Mineral Resources for mill material are based on NSR Cut-offs of $11.20/t for oxide material and $11.43/t for sulfide material due to slight differences in crushing and grinding costs.
10. Table 14-2 accompanies this Mineral Resource and shows all relevant parameters.
11. Mineral Resources are reported in relation to a conceptual constraining pit shell in order to demonstrate reasonable prospects for eventual economic extraction, as required by the definition of Mineral Resource in NI 43-101; mineralization lying outside of the pit shell is excluded from the Mineral Resource.
12. The Mineral Resource is reported inclusive of the Mineral Reserve.
13. All the mineralization comprised in the Mineral Resource estimate is contained on mineral titles controlled by St. Augustine and NADECOR. The constraining pit shell that defines the Mineral Resource includes small amounts of waste material on mineral titles controlled by others. The extraction of the entire Mineral Resource will require agreements with other mineral title owners to mine waste on their mineral titles.

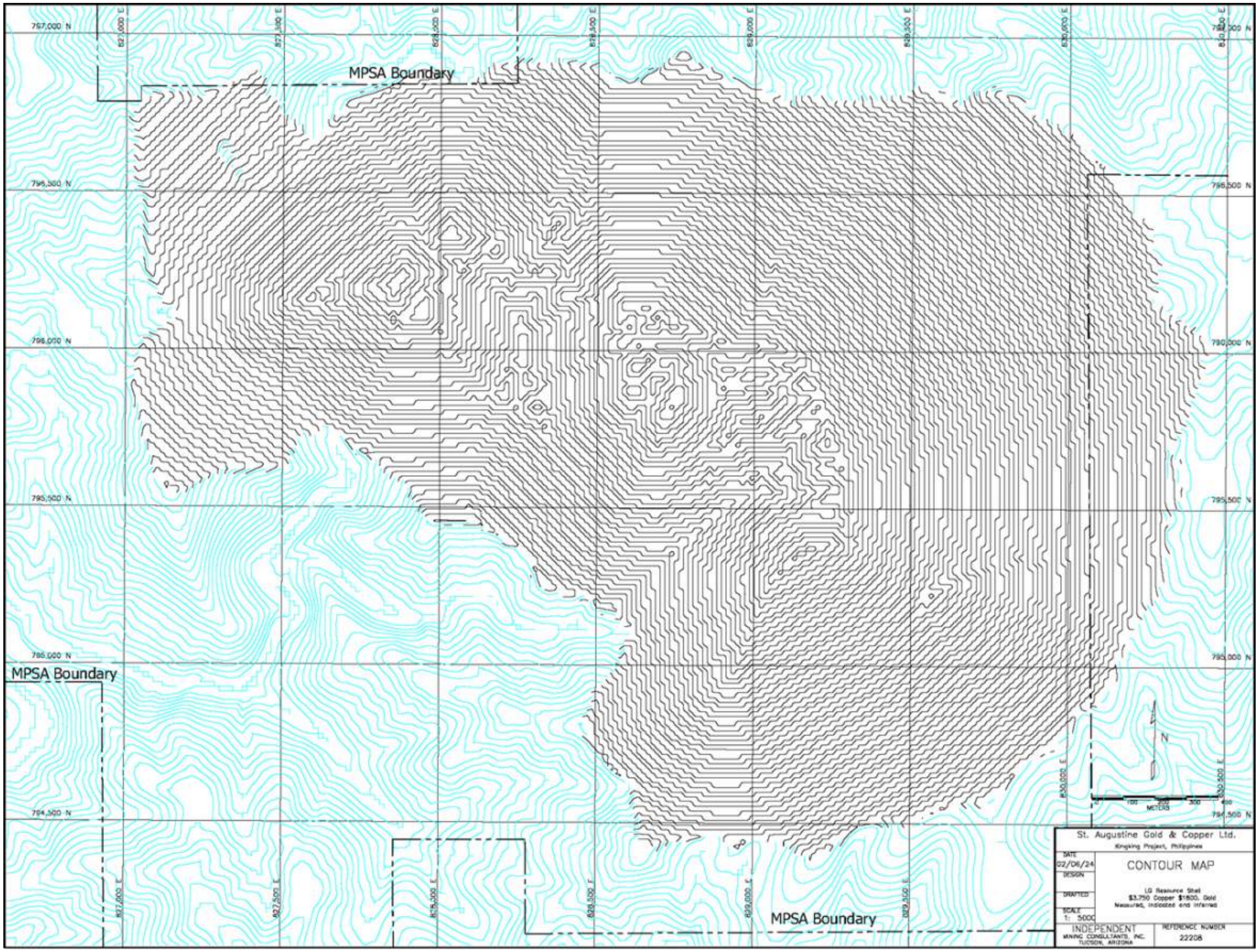



**Figure 14-1: Constraining Pit Shell for Mineral Resource Estimate (IMC, 2024)**

## 14.2 MINERAL RESOURCE PARAMETERS

### 14.2.1 Economic Parameters

Table 14-2 shows the economic parameters used for the Mineral Resource estimate. The base copper and gold prices are $3.75/lb copper and $1800/oz gold. The QP for this section believes these prices to be reasonable based on 1) current spot prices and historical 3-year trailing averages, 2) prices used by other companies for comparable projects, and 3) long range consensus price forecasts prepared by various bank economists.

The table shows parameters for heap leach resource, mixed oxide/sulfide material amenable to milling, and sulfide material amenable to milling. In the mill, copper and gold are recovered to a concentrate using conventional flotation, followed by agitated leach of the tails to recover additional copper. A portion of the gold is recovered in a gravity circuit. The heap leach process only recovers copper.

The heap leach resource consists of oxide and some mixed oxide/sulfide material. The decision to route the mixed oxide/sulfide resource to the heap leach versus the mill is based on highest value net of processing costs. This calculation was on a block-by-block basis in the model.

The mining costs on the table are estimated based on owner operation of mining equipment. The estimates are based on current costs for fuel, equipment parts, power, lubricant, blasting agents, and labor costs. The base mining cost per tonne is the same for all material types and the haulage cost varies by material type. In particular, the waste haulage cost is 25% higher than the cost for leach and mill resources. The estimate also includes an allowance for replacement capital equipment.

Estimated process costs were provided by Met Engineering, LLC, a client consultant, and confirmed by M3 and Art Ibrado of Ft. Lowell Consulting PLLC, the QP for mineral processing. It can be seen for mill resource the unit costs are broken out by flotation, agitation, and tailings costs. G&A unit costs are $1.508/t for mill resource and $0.322/t for heap leach resource.

Process recoveries are incorporated into the resource block model on a block-by-block basis. The equations are presented in Section 14.2.2. The recoveries shown on the table are averages for each material type based on the blocks that contribute to the Mineral Resource estimate.

Payable copper in concentrate is estimated at 96%, based on a copper concentrate grade of 25% and smelting, refining, and freight (SRF) cost is estimated at $0.263/lb. This estimate is based on a smelter treatment charge of $65 per metric tonne and refining charge of $0.065/lb. Refinery terms for gold in concentrate are estimated at 95.0% payable and $5.00/oz. Refinery terms for gold in gravity concentrates are estimated as 98.9% payable and $2.00/oz.

Copper recovered from the leach pad and agitated tails will be recovered in a solvent extraction and electrowinning (SX-EW) plant on the property. SX-EW and cathode freight and insurance are estimated at $0.176/lb copper.

The economic parameters also include an NSR royalty of 5%. The NSR factors and NSR cut-offs shown on the table are discussed below.

### Table 14-2: Economic Parameters for Mineral Resource Estimate

| Parameter | Units | Heap Leach Oxide/Mixed | Mill Mixed Ox/Slf | Mill Sulfide | Waste |
|---|---|---|---|---|---|
| Copper Price Per Lb | (US$) | 3.75 | 3.75 | 3.75 | |
| Gold Price Per Oz | (US$) | 1800 | 1800 | 1800 | |
| Mining Cost Per Tonne - Owner Operation | | | | | |
| Base Mining Cost Without Haulage | (US$) | 0.926 | 0.926 | 0.926 | 0.926 |
| Base Haulage Cost | (US$) | 0.579 | 0.618 | 0.618 | 1.029 |
| Mine Replacement Capital Per Tonne | (US$) | 0.130 | 0.130 | 0.130 | 0.130 |
| Mining Cost | (US$) | 1.635 | 1.674 | 1.674 | 2.085 |
| Process Cost Per Tonne | | | | | |
| Crushing, Grinding, Flotation | (US$) | N.A. | 4.728 | 4.958 | |
| Agitated Leach | (US$) | N.A. | 4.192 | 4.192 | |
| Tailings | (US$) | N.A. | 1.187 | 1.187 | |
| Heap Leach | (US$) | 3.385 | N.A. | N.A. | |
| Processing Cost Per Ore Tonne | (US$) | 3.385 | 10.107 | 10.337 | |
| G&A Cost Per Ore Tonne | (US$) | 0.322 | 1.508 | 1.508 | |
| Process Recoveries (Average Based on Recovery Equations) | | | | | |
| Copper (Concentrate + Agitation) | (%) | N.A. | 92.1% | 81.9% | |
| Copper to Concentrate | (%) | N.A. | 36.5% | 67.9% | |
| Copper to Agitation (or Heap Leach) | (%) | 82.1% | 55.6% | 14.0% | |
| Gold (Concentrate + Gravity) | (%) | N.A. | 62.1% | 74.4% | |
| Gold to Concentrate | (%) | N.A. | 38.6% | 55.1% | |
| Gold to Gravity Concentrate | (%) | N.A. | 23.5% | 19.3% | |
| Conventional Smelting/Refining | | | | | |
| Smelting/Refining Payable for Copper | (%) | N.A. | 96.0% | 96.0% | |
| Smelting/Refining Payable for Gold | (%) | N.A. | 95.0% | 95.0% | |
| SRF Cost Per Pound Copper | (US$) | N.A. | 0.263 | 0.263 | |
| Gold Refining Per Ounce | (US$) | N.A. | 5.00 | 5.00 | |
| Gold to Gravity Concentrate | | | | | |
| Refinery Payable (with Transport/Insurance) | (%) | N.A. | 98.9% | 98.9% | |
| Gold Refining Per Ounce | | N.A. | 2.00 | 2.00 | |
| Site Solvent Extraction/Electrowinning | | | | | |
| Payable Copper | (%) | 100% | 100% | 100% | |
| SX-EW Per Pound Copper | (US$) | 0.164 | 0.164 | 0.164 | |
| Cathode Freight/Insurance | (US$) | 0.012 | 0.012 | 0.012 | |
| Gross Royalty | (%) | 5.0% | 5.0% | 5.0% | |
| NSR Factors (Applied to Recovered Grade) | | | | | |
| Copper in Concentrate | (US$/t) | N.A. | 70.100 | 70.100 | |
| Copper in Agitation Leach | (US$/t) | N.A. | 74.853 | 74.853 | |
| Gold in Concentrate | (US$/t) | N.A. | 52.085 | 52.085 | |
| Gold in Gravity Concentrate | (US$/t) | N.A. | 54.313 | 54.313 | |
| Copper in Heap Leach | (US$/t) | 74.853 | N.A. | N.A. | |
| NSR Cut-offs | | | | | |
| Breakeven | (US$/t) | 5.34 | 13.29 | 13.52 | |
| Internal | (US$/t) | 3.26 | 11.20 | 11.43 | |

### 14.2.2 Metal Recoveries

The process recovery equations are complex and were provided by Met Engineering, LLC, a client consultant, and confirmed by Art Ibrado of Ft. Lowell Consulting PLLC, the QP for mineral processing. For heap leach resource, the recovery equation is:

$$Rec(\%) = 90\%(0.0151934 + 0.0526427 \times totcu + 1.1501882 \times solcu + 0.03629 \times ratio)/totcu$$

Where totcu is total copper, solcu is soluble copper (both in %) and ratio is solcu/totcu. The recovery is capped at 92%.

The recovery of total copper to concentrate is:

$$Rec(\%) = 100\%(1.2038653 + 0.131888 \times \ln(solcu) + 0.0681887 \times gold^{0.6757} - 0.0589333 \times \ln(nsol) - 1.42294 \times ratio)$$

Where gold is gold in ppm, nsol is totcu – solcu, and ln is the logarithm, base e. The recovery is capped at 90%. This equation can result in values less than 0; these were set to 0.

The copper recoveries to concentrate and concentrate grade are then used to calculate the mass of tails and the total and soluble copper grades of the tails. The details of this mass balance are not shown here. Copper recovery for agitation leach of copper in the tails is:

$$Rec(\%) = 100\%(0.1052638 + 0.918662 \times (t\_solcu/t\_totcu))$$

Where t_solcu and t_totcu are the soluble and total copper grades of the tails. This was capped at a maximum of 95%; the minimum is 10.5%. Once this recovery of copper in tails was calculated it was also used to calculate agitation recovery in terms of percent of total copper in the original plant feed.

The recovery of gold to the gravity concentrate is estimated as 19% when the soluble copper to total copper ratio is less than 50% and 26% when the ratio is greater than or equal to 50%.

The gold head grade to the concentrator is adjusted to account for the gravity gold recovery and the recovery of gold to concentrate is:

$$Rec(\%) = 100\%(0.3483337 + 0.2169788 \times c\_gold^{0.266} + 0.0878149 \times \ln(nsol) - 0.0907290 \times \ln(solcu))$$

Where c_gold is the gold grade in g/t net of gravity gold. All other variables and operators are as previously defined. Total gold recovery, gravity plus concentrate, was capped at 85%.

### 14.2.3 NSR Values

Due to multiple products and variable recoveries for copper and gold, NSR values, in US$/t, were calculated for each model block to classify blocks into potential resource or waste.

For heap leach resource the calculation is:

$$\begin{aligned} NSR\_lch &= (\$3.75 - 0.176) \times 0.95 \times totcu(\%) \times recov \times 22.046 \\ &= \$74.853 \times totcu(\%) \times recov \end{aligned}$$

Note that 74.853 is the NSR factor shown on Table 14-2 for heap leach resource. Recov is a decimal between 0 and 1 instead of a percent. The 0.95 term accounts for the royalty.

For mill resource (oxide/mixed and sulfide) the calculations are:

NSR_cu_conc = ($3.75 – $0.263) x 0.96 x 0.95 x totcu(%) x recov x 22.046
= $70.10 x recovered copper to concentrate (%)

NSR_cu_agit = ($3.75 – $0.176) x 1.0 x 0.95 x totcu(%) x recov x 22.046
= $74.853 x recovered copper to agitation (%)

NSR_au_conc = ($1800 – $5.00) x 0.95 x 0.95 x gold(g/t) x recov / 31.103
= $52.085 x recovered gold to concentrate (g/t)

NSR_au_grav = ($1800 - $2.00) x 0.989 x 0.95 x gold(g/t) x recov / 31.103
= $54.313 x recovered gold to gravity (g/t)

NSR_mill = NSR_cu_conc + NSR_cu_agit + NSR_au_conc + NSR_au_grav

For oxide/mixed zone material, the routing for heap leach versus the mill was based on the highest value of NSR less process/G&A costs.

The bottom of Table 14-2 shows NSR cut-offs for each resource type. For sulfide mill resources, the breakeven cut-off is the mining + processing + G&A cost and amounts to $13.52/t. Internal cut-off applies to blocks that must be removed from the pit, so mining is considered a sunk cost. For sulfide mill resource, this is $10.337 + 1.508 – ($2.085 - $1.674) = $11.43. The incremental mining cost in the calculation reflects the lower haulage cost to the crusher versus the waste dumps. The breakeven and internal cut-offs for oxide mill resource are slightly lower due to the lower crushing, grinding and flotation cost.

## 14.3 ADDITIONAL INFORMATION

The Mineral Resources are classified in accordance with the May 2014 Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") "CIM Definition Standards – For Mineral Resources and Mineral Reserves" adopted by the CIM Council (as amended, the "CIM Definition Standards") in accordance with the requirements of NI 43-101. Mineral Reserve and Mineral Resource estimates reflect the reasonable expectation that all necessary permits and approvals will be obtained and maintained.

There is no guarantee that any of the Mineral Resources will be converted to Mineral Reserve. The Inferred Mineral Resources included in this Technical Report meet the current definition of Inferred Mineral Resources. The quantity and grade of Inferred Mineral Resources are uncertain in nature and there has been insufficient exploration to define these inferred Mineral Resources as an Indicated Mineral Resource. It is, however, expected that the majority of Inferred Mineral Resource could be upgraded to Indicated Mineral Resource with continued exploration.

The QP for this section does not believe that there are significant risks to the Mineral Resource estimate based on environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors. The Kingking Project is in a jurisdiction friendly to mining. The most significant risks to the Mineral Resource are related to economic parameters such as prices lower than forecast, recoveries lower than forecast, or costs higher than the current estimates.

All the mineralization comprised in the Mineral Resource estimate with respect to the Kingking Project is contained on mineral titles controlled by St. Augustine and NADECOR, its partner in the Kingking Project. The constraining pit shell that defines the Mineral Resource includes small amounts of waste material on mineral titles controlled by others. The Mineral Resource estimate has been prepared based on the Qualified Person's reasoned judgment, in accordance with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practices Guidelines and his professional standards of competence, that there is a reasonable expectation that all the necessary permits, agreements and approvals will be obtained and maintained, including the additional agreements with other parties to allow mining of waste material on their mineral titles. In particular, when determining the prospects for eventual economic extraction, consideration was given to industry practice, and a timeframe of 50 years or so for base metals.

## 14.4 DESCRIPTION OF THE BLOCK MODEL

### 14.4.1 General

The deposit was modeled at 15 m x 15 m x 15 m high blocks.

### 14.4.2 Cap Grades and Compositing

The drill hole database consisted of 285 holes which represented 93,745 m of drilling. Benguet gold assays were not used; however, Echo Bay re-assays of Benguet samples amounted to 1493 assays that were used. Gold assays were capped at 10 g/t, which affected nine assays with original values of 44.3, 24.3, 18.6, 17.3, 16.2, 14.3, 13.3, 12.0, and 10.7 g/t. Copper assays were not capped. The highest assay was 7.2%.

The assay database was composited to 15 m bench composites for block grade estimation. Based on the bench composites, the data available for resource estimation consisted of 92,354 m of sample with a total copper assay (5,910 composites with average length 15.6 m) and 61,048 m of sample with a gold assay (3,847 composites with an average length of 15.9 m). Samples with a retained gold assay represent about 66.1% of samples with a copper assay.

Figure 14-2 shows the Kingking drill holes, by drilling campaign, and cross section lines used for this Technical Report.

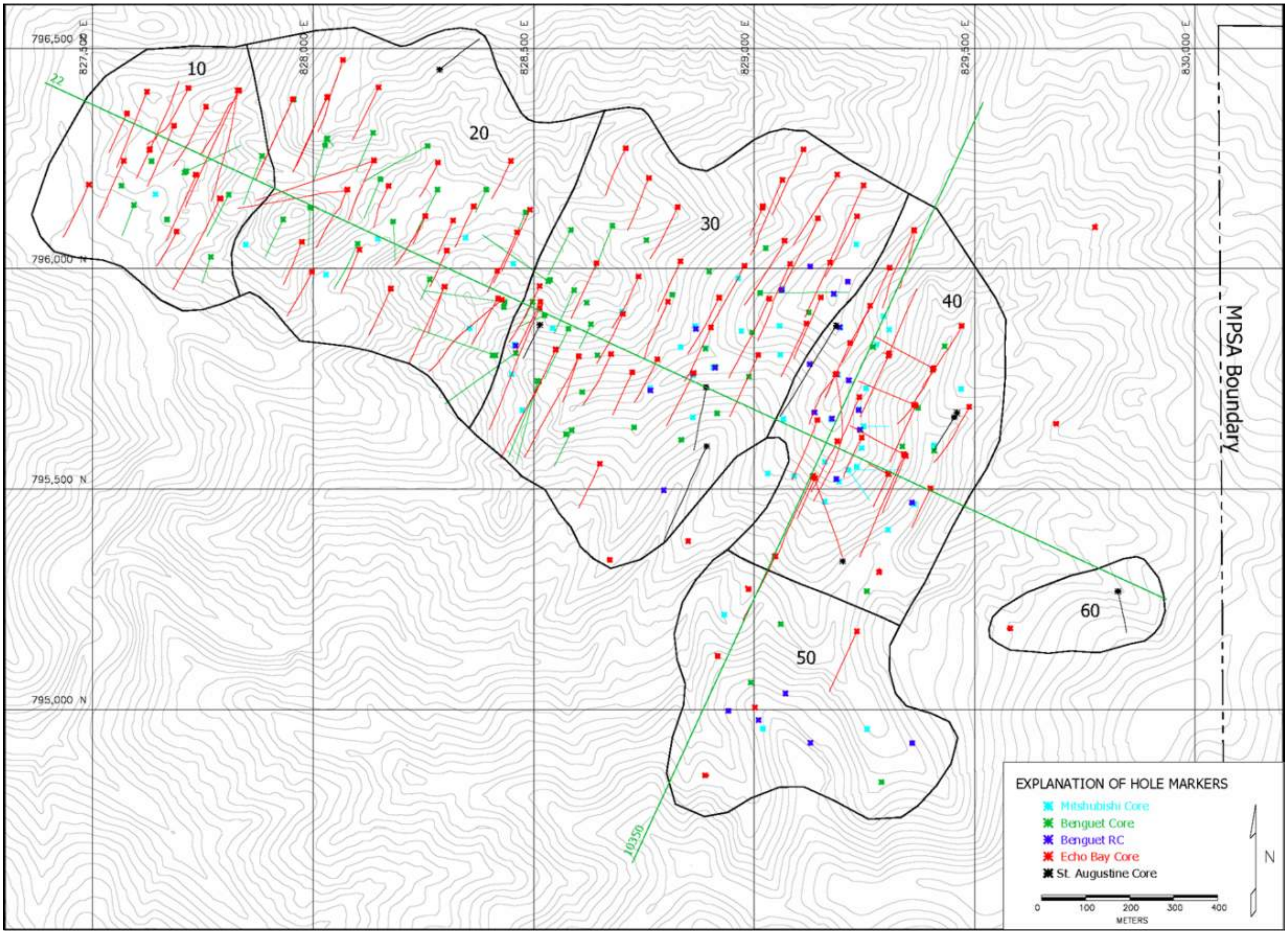



**Figure 14-2: Kingking Drill Holes by Campaign (IMC, 2024)**

### 14.4.3 Topography

Topography is in the world geodetic system 1998 (WSG84) coordinate system.

### 14.4.4 Lithology

Kingking lithology is complex. The original host rocks included sedimentary and volcanic flows that were intruded by multiple intrusive events. For this technical report, the rock types were categorized as shown in Table 14-3. The multiple intrusions were broadly categorized into pre-mineral/syn-mineral intrusions and post mineral intrusions.

**Table 14-3: Kingking 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 Kingking personnel for the 2011 model. The sectional data were used to develop the interpretation on bench level maps. This was then digitized, checked, and incorporated into the block model. There are only nine new holes since the previous model. It did not appear necessary to update the lithology model.

The Benguet core drilling generally recorded a depth of overburden. This was often only a few meters, up to approximately 15 m in some areas. This data was used to develop a surface to represent depth of overburden that was used to code overburden blocks in the model. Figure 14-3 and Figure 14-4 show cross sections of model lithology.

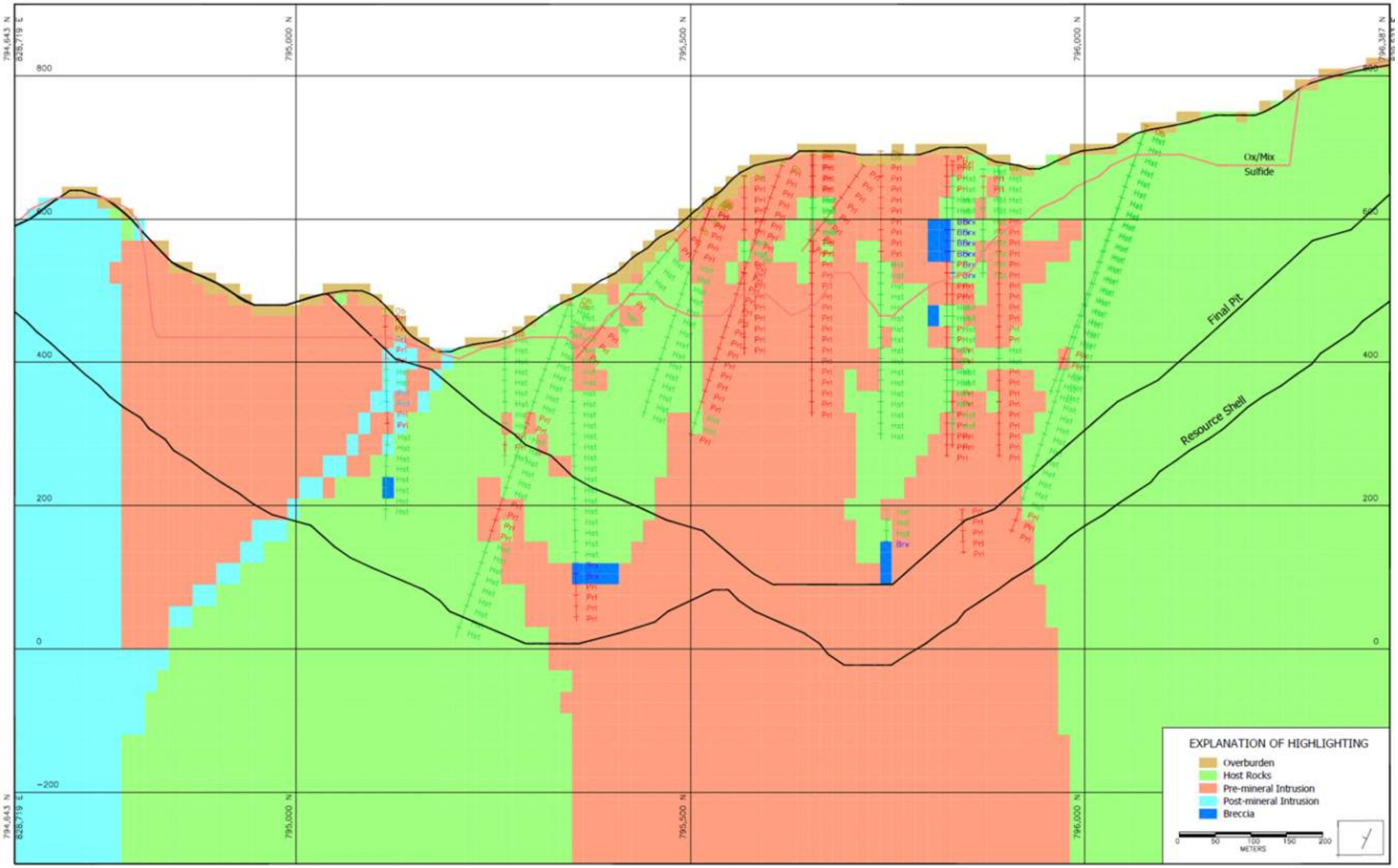



**Figure 14-3: Lithology on Cross Section 10,350 (IMC, 2024)**

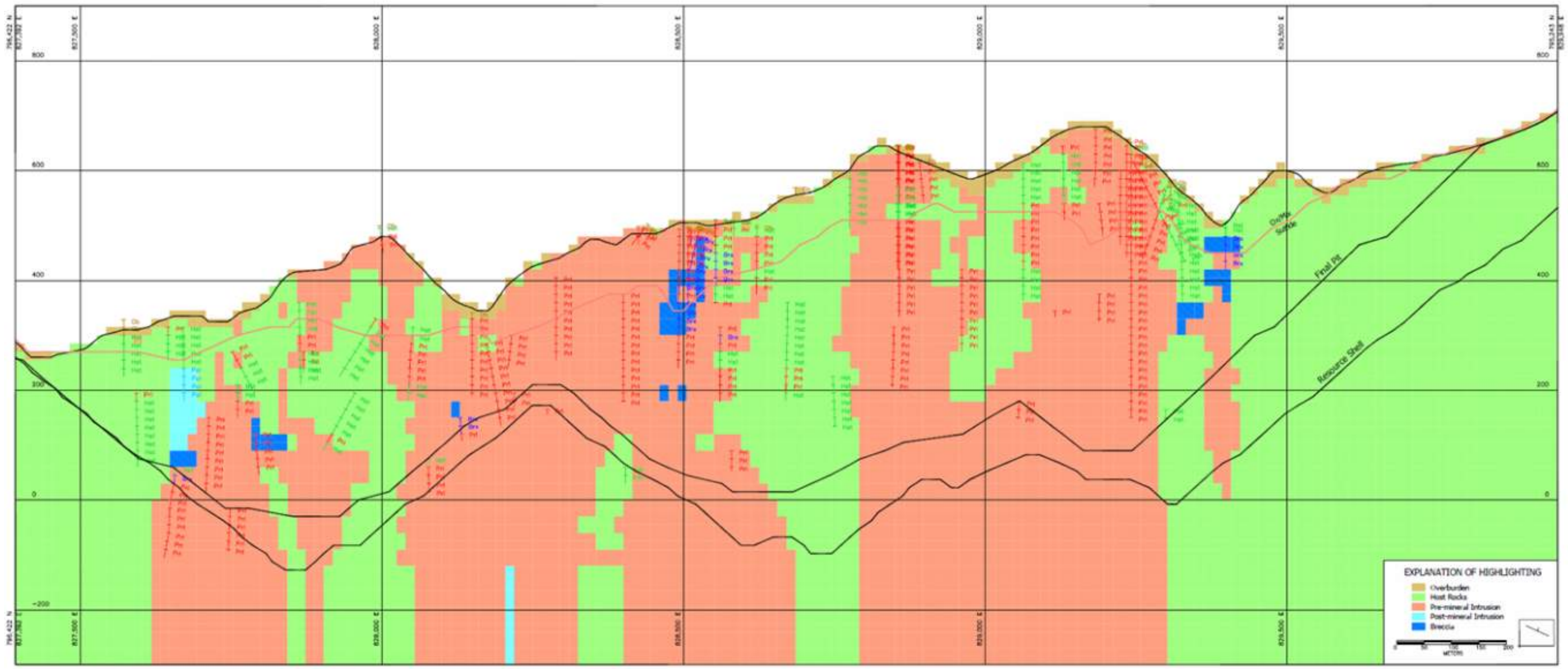



Figure 14-4: Lithology on Cross Section 22 (IMC, 2024)

### 14.4.5 Oxide/Sulfide Domains

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

**Table 14-4: General Ore Type Criteria**

| Domain | 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 15 m composites based on these criteria. The assignments were then reviewed on a hole-by-hole basis on data listings and cross sections to develop an interpretation of the top of primary mineralization in each hole.

The interpretation of the top of primary mineralization was represented as a triangulated surface and is shown on Figure 14-3 and Figure 14-4. Model blocks below the surface were coded as primary and blocks above the surface as oxide. Once the block grade estimates were completed (Section 14.4.9) the oxide zone was further segregated into oxide and mixed blocks based on the soluble copper to total copper ratio of the blocks. 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.

### 14.4.6 “Structural” Zones

IMC also developed six “structural” zones for the model. These were 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. Zones 30 and 40 have the highest copper grades. Zone 20 is slightly anomalous; it is characterized as relatively low in copper, but relatively high in gold compared to the other zones. Zone 10 is also relatively high in gold. The outer boundary of the zones represents an approximate 100 m boundary outside of the drilling. Block grades were not estimated outside the shown boundaries. Based on the new drilling information, the zones have been re-designed, and are slightly larger than the zones for the 2011 model. Zone 60 was added to define a small inferred Mineral Resource for the two mineralized holes in the east area.

These zones correspond to historical regional names that were used to describe the deposit as shown in Table 14-5.

**Table 14-5: Structural Zones**

| Zone | Regional Name |
|---|---|
| 10 | Tiogdan |
| 20 | Casagumayan |
| 30, 40 | Lumanggang |
| 50 | Bacada |
| 60 | Far East |

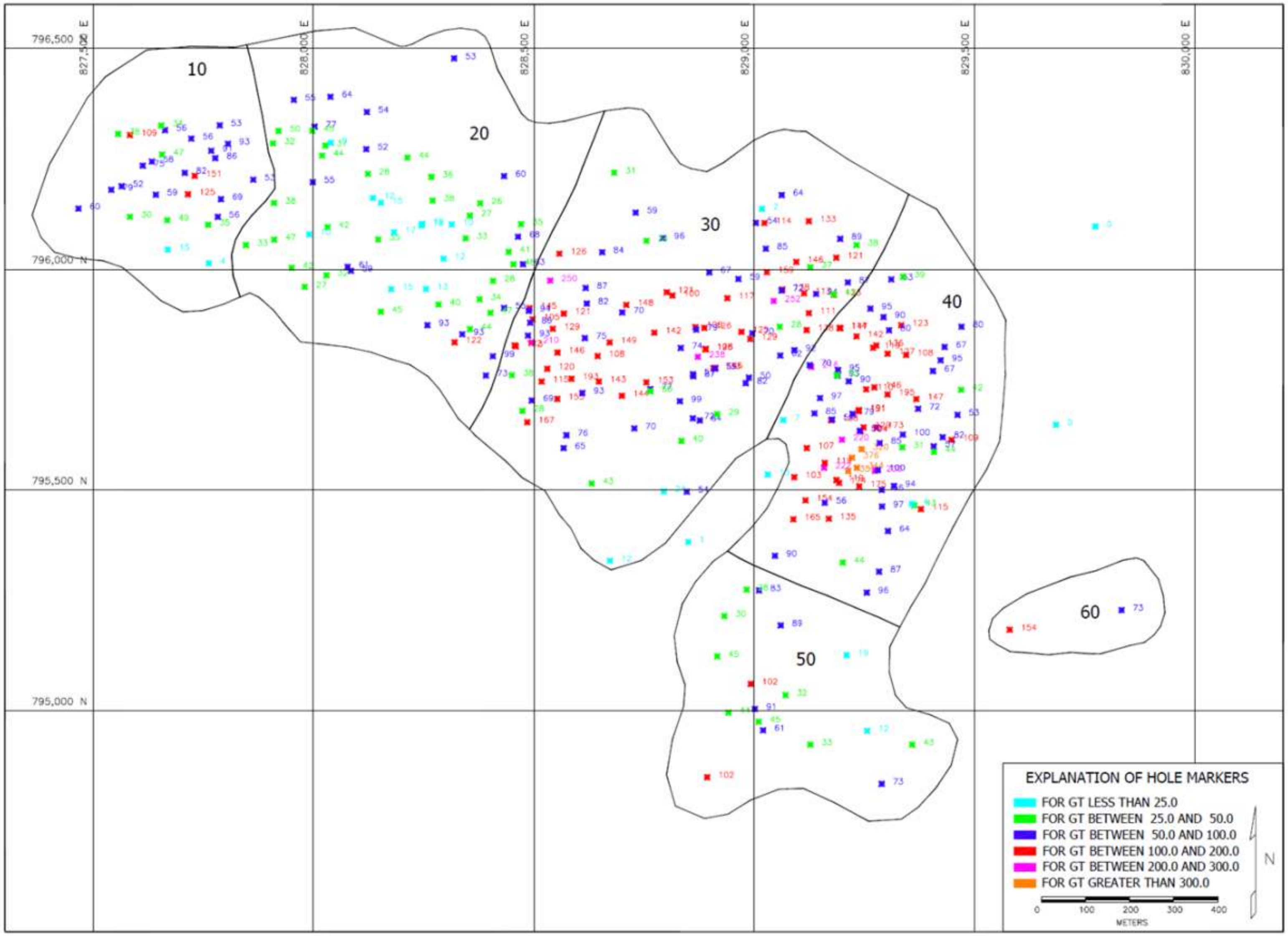



**Figure 14-5: Structural Zones with Copper Grade-Thickness (m-%) (IMC, 2024)**

### 14.4.7 Descriptive Statistics of Drill Hole Data

Table 14-6 shows descriptive statistics for total copper and gold for the assay intervals. The upper portion of the table shows values by rock type and the lower portion by the structural zones. The table represents un-capped assay values and includes only non-zero values for each population. It is interesting to note that the total copper and gold values in overburden are relatively high, though the number of samples is low. This could be due to surficial enrichment due to weathering or possible contamination due to artisanal mining activity.

By structural zone, zones 10 and 20 are relatively low in copper, particularly zone 20, but are significantly higher in gold than the other zones. The highest copper grades are in zones 30 and 40.

Table 14-7 shows results for 15 m bench composites.

**Table 14-6: Summary Statistics of Copper and Gold Assays**

| Rock Types | Copper | | | | Gold | | | |
|---|---|---|---|---|---|---|---|---|
| | No. of Samples | Mean (%) | Std Dev (%) | Max (%) | No. of Samples | Mean (g/t) | Std Dev (g/t) | Max (g/t) |
| All Samples | 34,149 | 0.287 | 0.357 | 7.23 | 20,652 | 0.314 | 0.686 | 44.31 |
| 10-Overburden | 816 | 0.430 | 0.581 | 4.16 | 459 | 0.594 | 1.622 | 24.34 |
| 20-Host Rock | 16,345 | 0.252 | 0.247 | 3.80 | 10,626 | 0.249 | 0.653 | 44.31 |
| 30-Pre-Mineral Intrusives | 15,243 | 0.321 | 0.434 | 7.23 | 8,246 | 0.388 | 0.660 | 18.58 |
| 40-Post-Mineral Intrusives | 395 | 0.163 | 0.184 | 1.37 | 275 | 0.238 | 0.311 | 2.28 |
| 50-Breccia | 1,350 | 0.288 | 0.312 | 6.17 | 1,046 | 0.284 | 0.486 | 6.72 |
| **Zone** | **Copper** | | | | **Gold** | | | |
| | **No. of Samples** | **Mean (%)** | **Std Dev (%)** | **Max (%)** | **No. of Samples** | **Mean (g/t)** | **Std Dev (g/t)** | **Max (g/t)** |
| All Samples | 34,149 | 0.287 | 0.357 | 7.23 | 20,652 | 0.314 | 0.686 | 44.31 |
| 10-Zone 10 | 3,179 | 0.205 | 0.241 | 4.95 | 2,887 | 0.404 | 0.923 | 24.34 |
| 20-Zone 20 | 5,722 | 0.162 | 0.168 | 2.71 | 4,218 | 0.416 | 0.718 | 16.21 |
| 30-Zone 30 | 12,671 | 0.311 | 0.290 | 4.07 | 6,920 | 0.268 | 0.501 | 18.58 |
| 40-Zone 40 | 9,903 | 0.378 | 0.518 | 7.23 | 5,378 | 0.282 | 0.768 | 44.31 |
| 50-Zone 50 | 2,338 | 0.202 | 0.144 | 1.54 | 912 | 0.159 | 0.278 | 5.23 |
| 60-Zone 60 | 336 | 0.239 | 0.203 | 1.08 | 337 | 0.130 | 0.103 | 0.51 |

**Table 14-7: Summary Statistics of Copper and Gold 15 m Composites**

| Rock Types | Copper | | | | Gold | | | |
|---|---|---|---|---|---|---|---|---|
| | No. of Samples | Mean (%) | Std Dev (%) | Max (%) | No. of Samples | Mean (g/t) | Std Dev (g/t) | Max (g/t) |
| All Samples | 5,807 | 0.280 | 0.306 | 4.44 | 3,754 | 0.309 | 0.460 | 10.24 |
| 10-Overburden | 123 | 0.413 | 0.511 | 2.61 | 80 | 0.555 | 0.926 | 4.58 |
| 20-Host Rock | 2,809 | 0.245 | 0.200 | 2.67 | 1,941 | 0.244 | 0.406 | 10.24 |
| 30-Pre-Mineral Intrusives | 2,560 | 0.314 | 0.382 | 4.44 | 1,494 | 0.384 | 0.488 | 5.92 |
| 40-Post-Mineral Intrusives | 65 | 0.174 | 0.157 | 0.87 | 47 | 0.230 | 0.211 | 0.93 |
| 50-Breccia | 250 | 0.290 | 0.237 | 1.55 | 192 | 0.300 | 0.384 | 2.18 |
| **Zone** | **Copper** | | | | **Gold** | | | |
| | **No. of Samples** | **Mean (%)** | **Std Dev (%)** | **Max (%)** | **No. of Samples** | **Mean (g/t)** | **Std Dev (g/t)** | **Max (g/t)** |
| All Samples | 5,807 | 0.280 | 0.306 | 4.44 | 3,754 | 0.309 | 0.460 | 10.24 |
| 10-Zone 10 | 589 | 0.206 | 0.193 | 2.58 | 543 | 0.414 | 0.640 | 5.92 |
| 20-Zone 20 | 1,040 | 0.162 | 0.132 | 1.00 | 777 | 0.412 | 0.487 | 3.95 |
| 30-Zone 30 | 2,122 | 0.309 | 0.243 | 2.16 | 1,241 | 0.264 | 0.346 | 3.38 |
| 40-Zone 40 | 1,646 | 0.363 | 0.452 | 4.44 | 945 | 0.266 | 0.472 | 10.24 |
| 50-Zone 50 | 343 | 0.195 | 0.117 | 0.99 | 181 | 0.162 | 0.161 | 1.16 |
| 60-Zone 60 | 67 | 0.238 | 0.180 | 0.70 | 67 | 0.130 | 0.086 | 0.36 |

### 14.4.8 Variograms

A variogram analysis of total copper was completed for host rocks and pre-mineral/syn-mineral intrusive rocks (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 14-6 shows variograms for total copper for host rocks. The variograms represent the primary and secondary directions of mineralization as interpreted by the QP for this section. The primary direction has an azimuth of 300° and an upward plunge of 65°, or alternatively an azimuth of 120° 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 659 m and 513 m, respectively

Figure 14-7 shows variograms for total copper for intrusive rocks. The variograms represent the primary and secondary directions of mineralization as interpreted by the QP for this section. 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 variograms are 410 m and 346 m, 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.

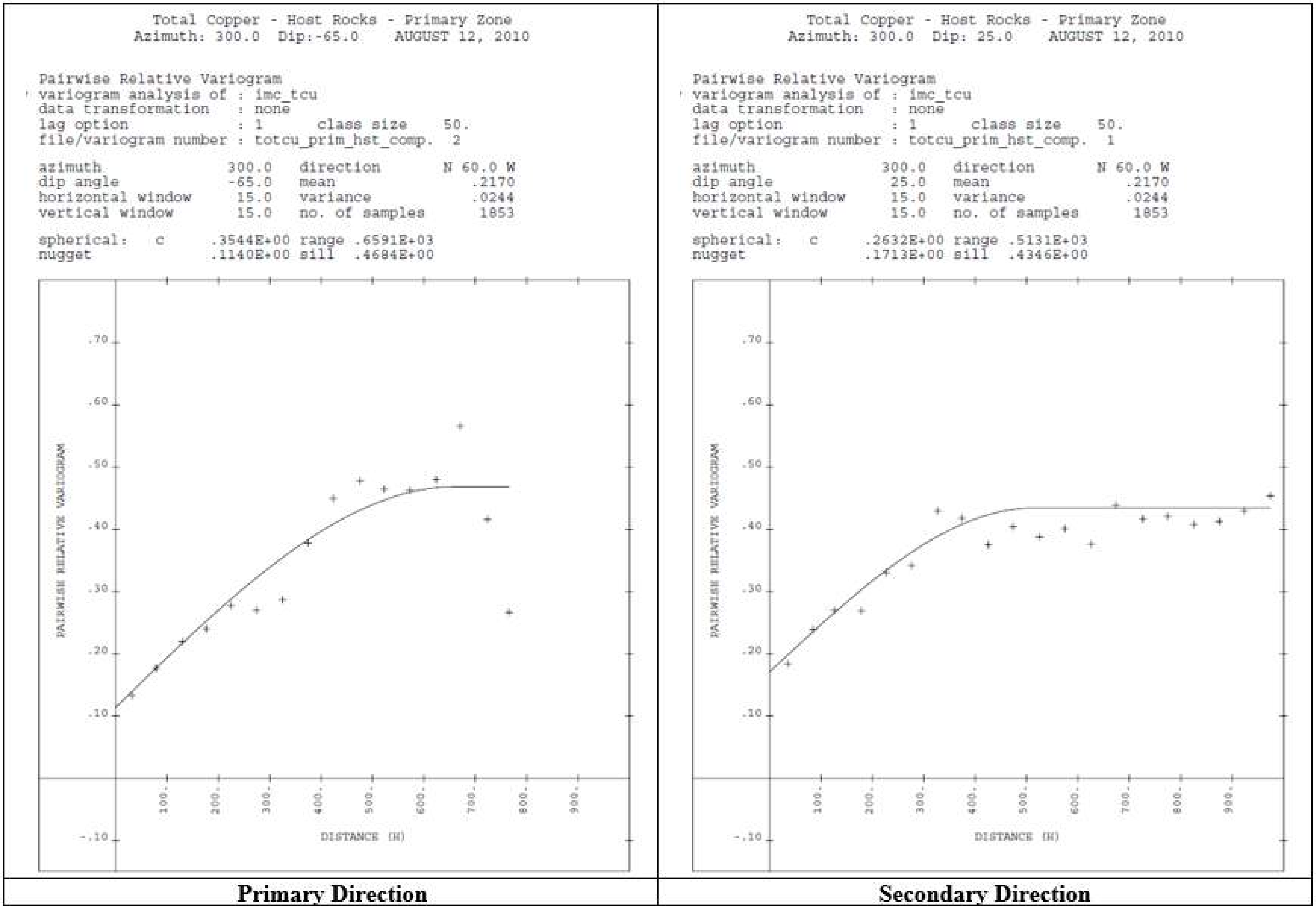



**Figure 14-6: Total Copper Variograms. Host Rocks in Sulfide Zone (IMC, 2010)**

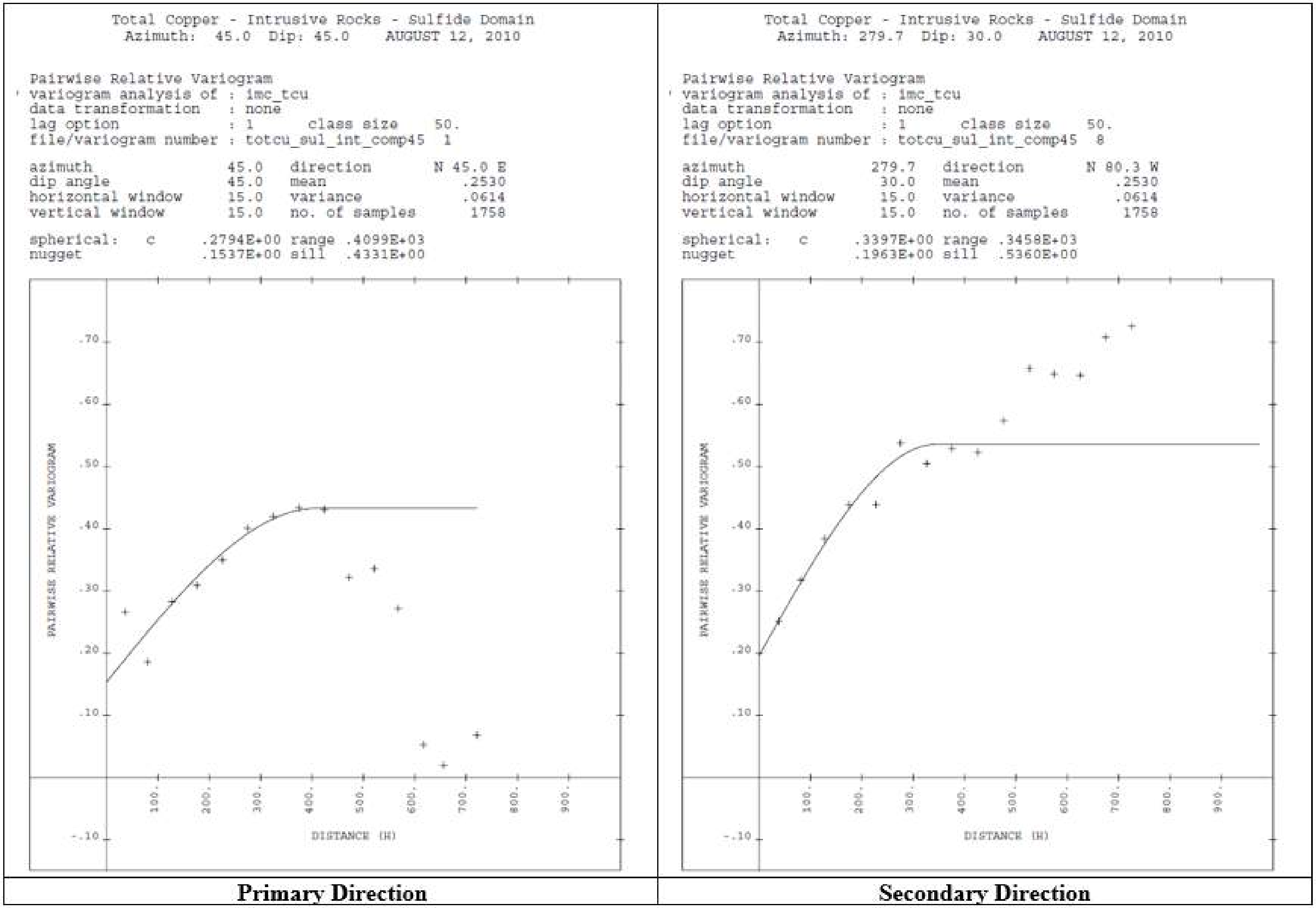



**Figure 14-7: Total Copper Variograms. Intrusive Rocks in Sulfide Zone (IMC, 2010)**

### 14.4.9 Block Grade Estimation

#### 14.4.9.1 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 200 m in the primary and secondary axes directions and 50 m 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 mineral intrusive composites were used for estimating them. 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 six zones were not estimated and composites outside the zones were not used. The boundaries between respective zones were not used as hard boundaries. Composites in Zone 20 could be used for Zone 10 blocks, etc. Note that the ID3 estimation will tend to honor the data regardless of boundaries.

#### 14.4.9.2 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. The oxide/mixed block designations were completed after grade estimation, based on soluble copper to total copper block grades.

A flat, circular search of 200 m by 200 m by 50 m 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° (S60°E) and a plunge of 65° and the secondary axis is oriented with an azimuth of 300° (N60oW) with a plunge of 25°. The tertiary axis is oriented with azimuth of 30° (N30°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°, -65°, and 0o, 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°E with a plunge of 45°. The secondary axis is orientated about N80°W with a plunge of approximately 30°. The GSLIB convention angles are 45°, -45°, and 45°.

Due to the relatively small size and complex orientations of the post mineral intrusive rocks, the search radius was increased to 200 m by 200 m by 200 m 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 14-8 and Figure 14-9 show the block grade estimation for total copper on cross sections.

#### 14.4.9.3 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 14-10 and Figure 14-11 show the block grade estimations for gold on cross-sections.

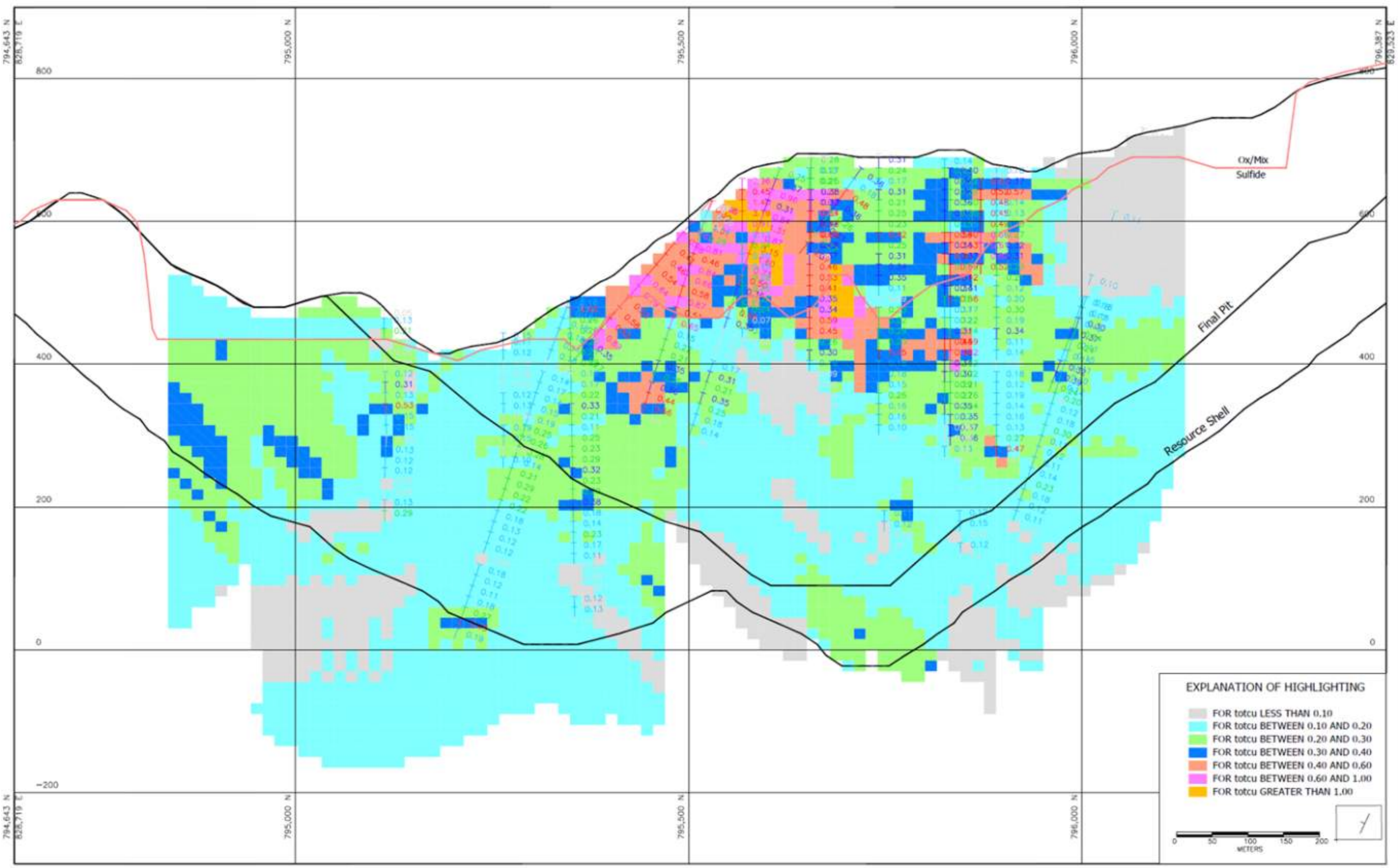



Figure 14-8: Copper Grades on Section 10,350 (IMC, 2024)

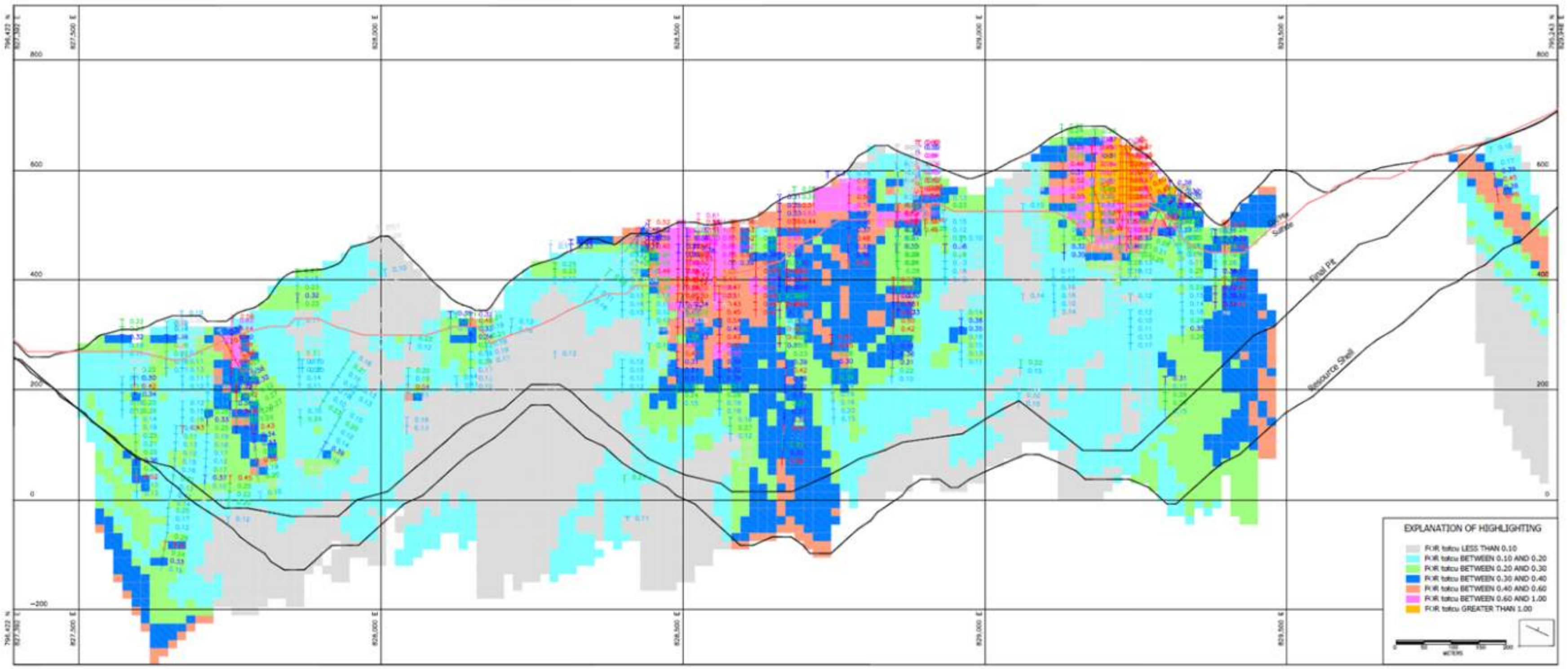



**Figure 14-9: Copper Grades on Section 22 (IMC, 2024)**

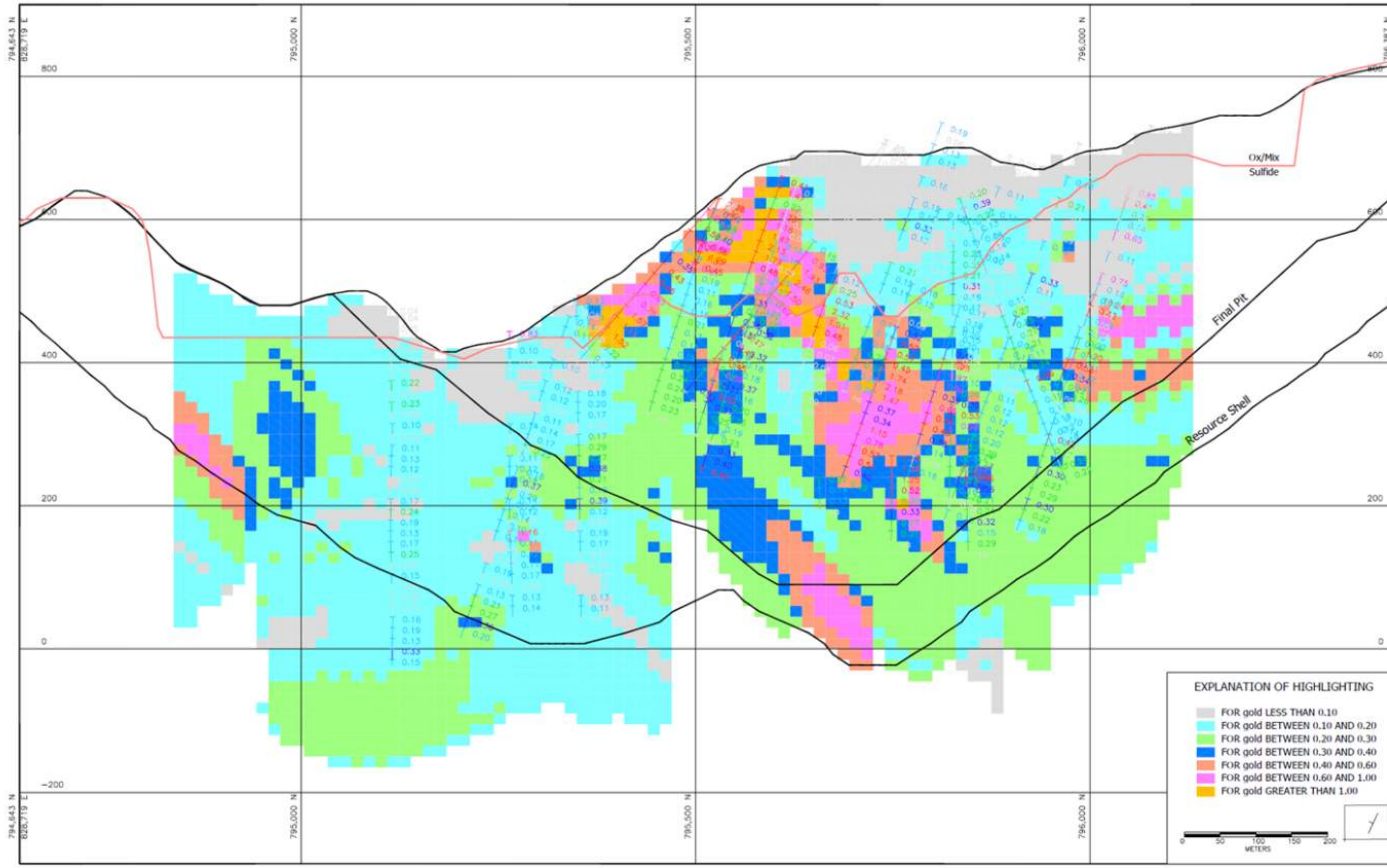



**Figure 14-10: Gold Grades on Section 10,350 (IMC, 2024)**

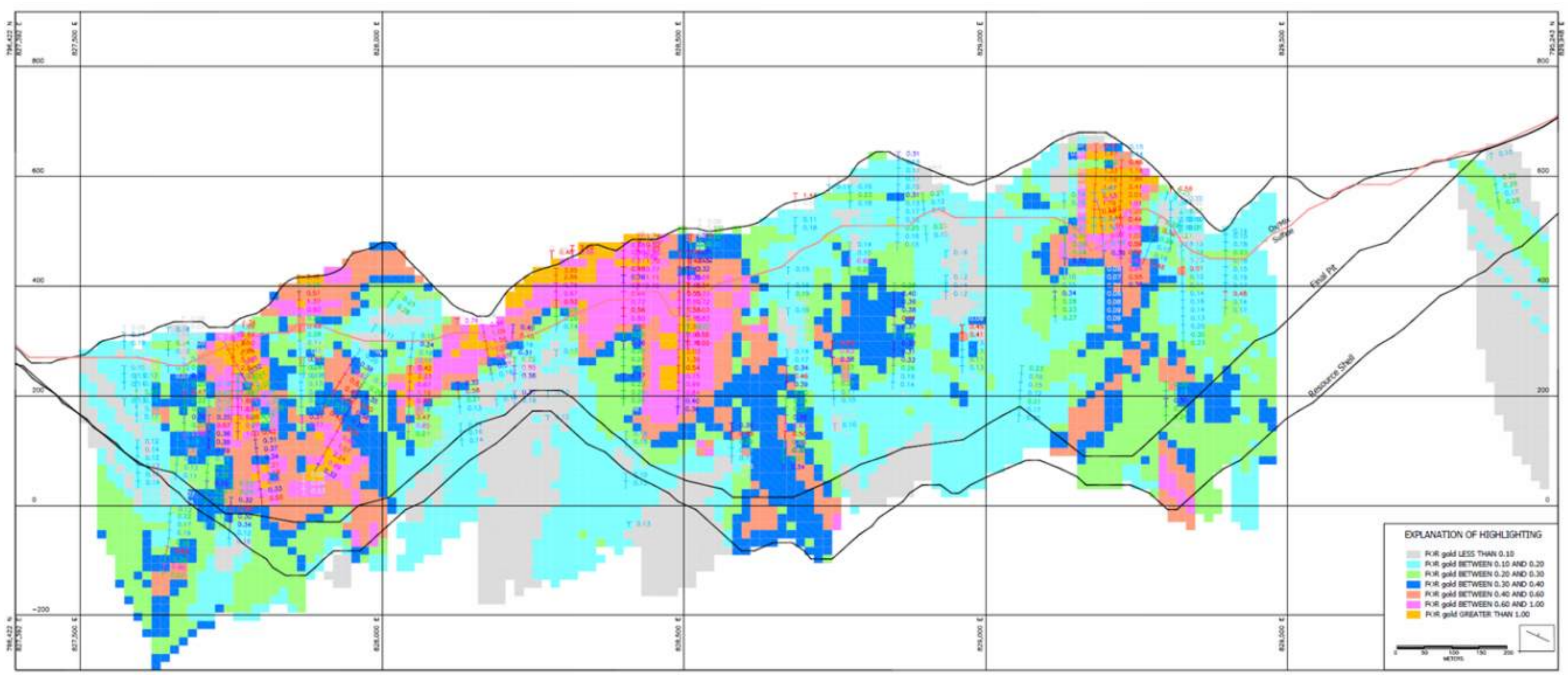



**Figure 14-11: Gold Grades on Section 22 (IMC, 2024)**

#### 14.4.9.4 Resource Classification

For classifying Measured and Indicated versus Inferred Mineral Resources, two additional block estimates were done. These were based on the same search orientations and search radii as the grade estimates. The first estimate was based on a maximum of three composites, a minimum of three, and a maximum of one composite per hole. The second estimate was based on a maximum of four composites, a minimum of four, and a maximum of one composite per hole. These estimates provide the average distance to the nearest three and four holes to each block and were put into the block model. The grade from these estimates were not used, only the average distance. The rock type and mineral domain populations for the estimations were the same as used for the grade estimates.

Blocks with the average distance to the nearest three holes less than (or equal to) 135 m are classified as Indicated Mineral Resources and blocks with the average distance to the nearest four holes less than (or equal to) 75 m are classified as Measured Mineral Resources. Blocks with grade estimates that did not meet either of these criteria were classified as Inferred Mineral Resources.

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 Mineral Resource.

Figure 14-12 and Figure 14-13 show probability plots of the average distance to the nearest three and four holes for copper in the oxide/mixed and sulfide domains respectively. For Figure 14-12, for the oxide/mixed domain, about 62% of the blocks meet the criteria Measured Mineral Resource (x-axis) with the average distance from the nearest four holes less than 75 m. About 30% of the blocks meet the criteria for Indicated Mineral Resource, and about 10% Inferred Mineral Resource. The percentages are based on all blocks in the model; not all of them are included in the shell used to determine Mineral Resources.

For Figure 14-13, for copper in the sulfide domain, about 14% of the blocks meet the criteria for Measured Mineral Resource, about 41% meet the criteria for Indicated Mineral Resource, and about 45% are Inferred Mineral Resource. Again, not all of these blocks are included in the shell used to determine Mineral Resources.

Figure 14-14 and Figure 14-15 show probability plots of the average distance to the nearest three and four holes for gold in the oxide/mixed and sulfide domains respectively. Based on the gold classification, significantly fewer blocks are classified as Measured and Indicated Mineral Resource than copper, due to the lower number of holes assayed for gold.

After calculating the average distances to the blocks and assigning an initial classification, a filtering algorithm was applied to the classification. The purpose of the filtering was to remove small clusters of blocks surrounded by blocks of different classification and to smooth out the boundaries of the various categories. The filtering identified blocks that were contacted on three or four edges by blocks of a different resource category. A filtering pass consisted of the following steps:

- Measured blocks with 3 or 4 adjacent indicated blocks were set to indicated.
- Measured blocks with 3 or 4 adjacent inferred blocks were set to inferred (rare case).
- Indicated blocks with 3 or 4 adjacent measured blocks were set to measured.
- Indicated blocks with 3 or 4 adjacent inferred blocks were set to inferred.
- Inferred blocks with 3 or 4 adjacent measured blocks were set to measured (rare case).
- Inferred blocks with 3 or 4 adjacent indicated blocks were set to indicated.
- Measured blocks with 3 or 4 adjacent indicated or inferred blocks were set to indicated.
- Indicated blocks with 3 or 4 adjacent measured or inferred blocks were set to inferred.
- Inferred blocks with 3 or 4 adjacent measured or indicated blocks were set to indicated.

Five of these filtering passes were done. Figure 14-16 and Figure 14-17 show the resource categories on cross sections.

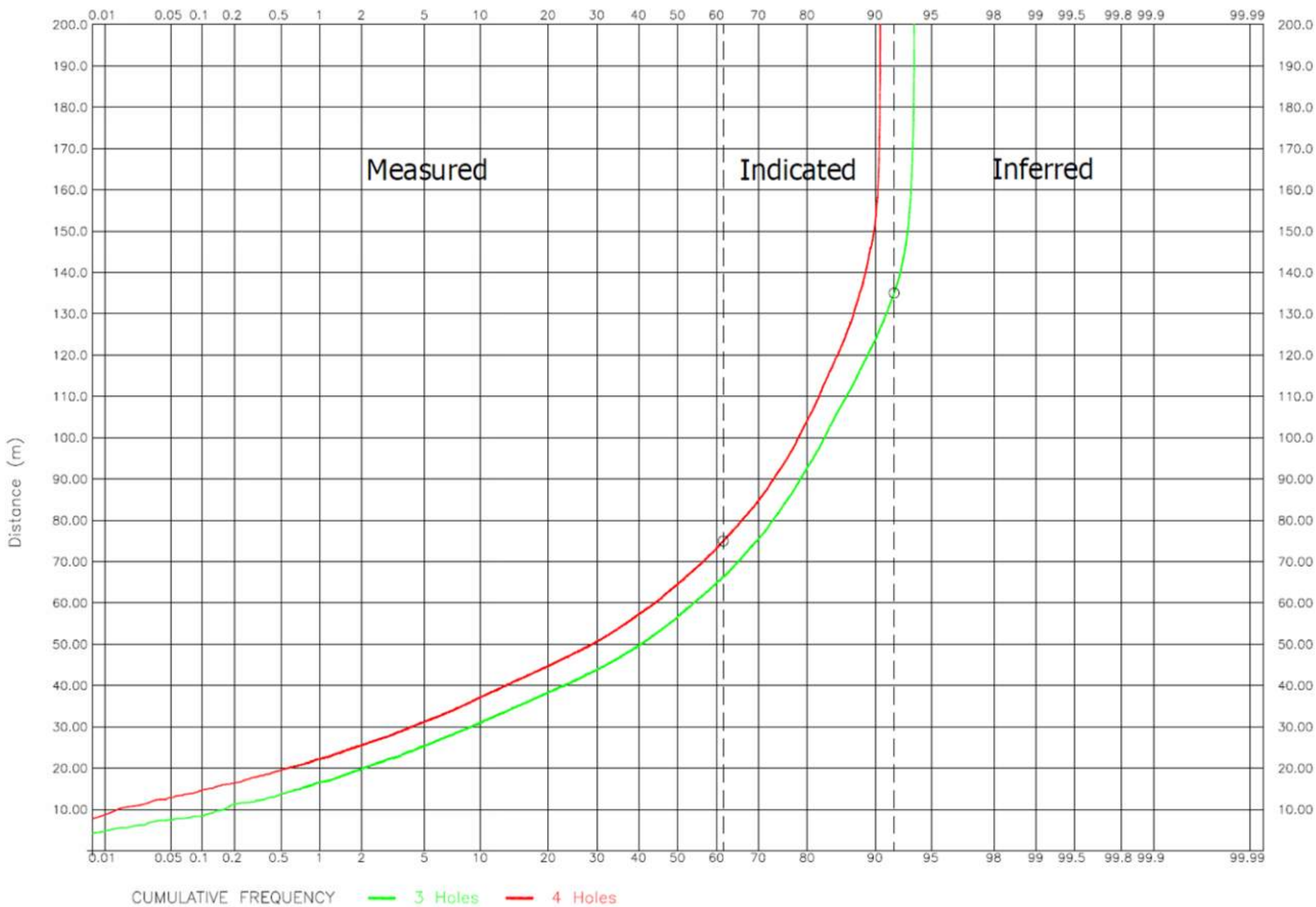



**Figure 14-12: Probability Plot of Average Distance to Nearest 3 and 4 Holes. Total Copper in the Oxide Zone (IMC, 2024)**

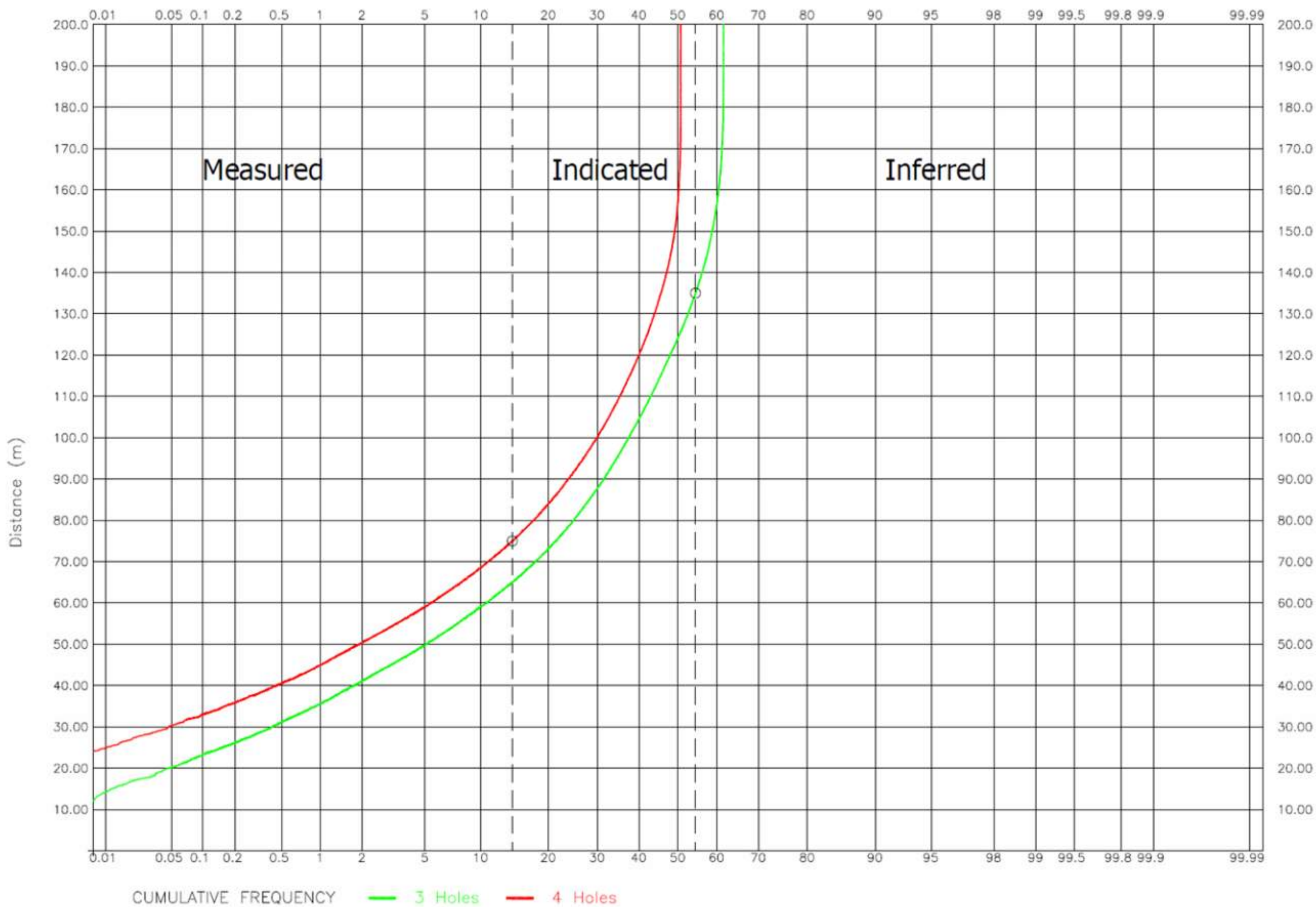



**Figure 14-13: Probability Plot of Average Distance to Nearest 3 and 4 Holes. Total Copper in the Sulfide Zone (IMC, 2024)**

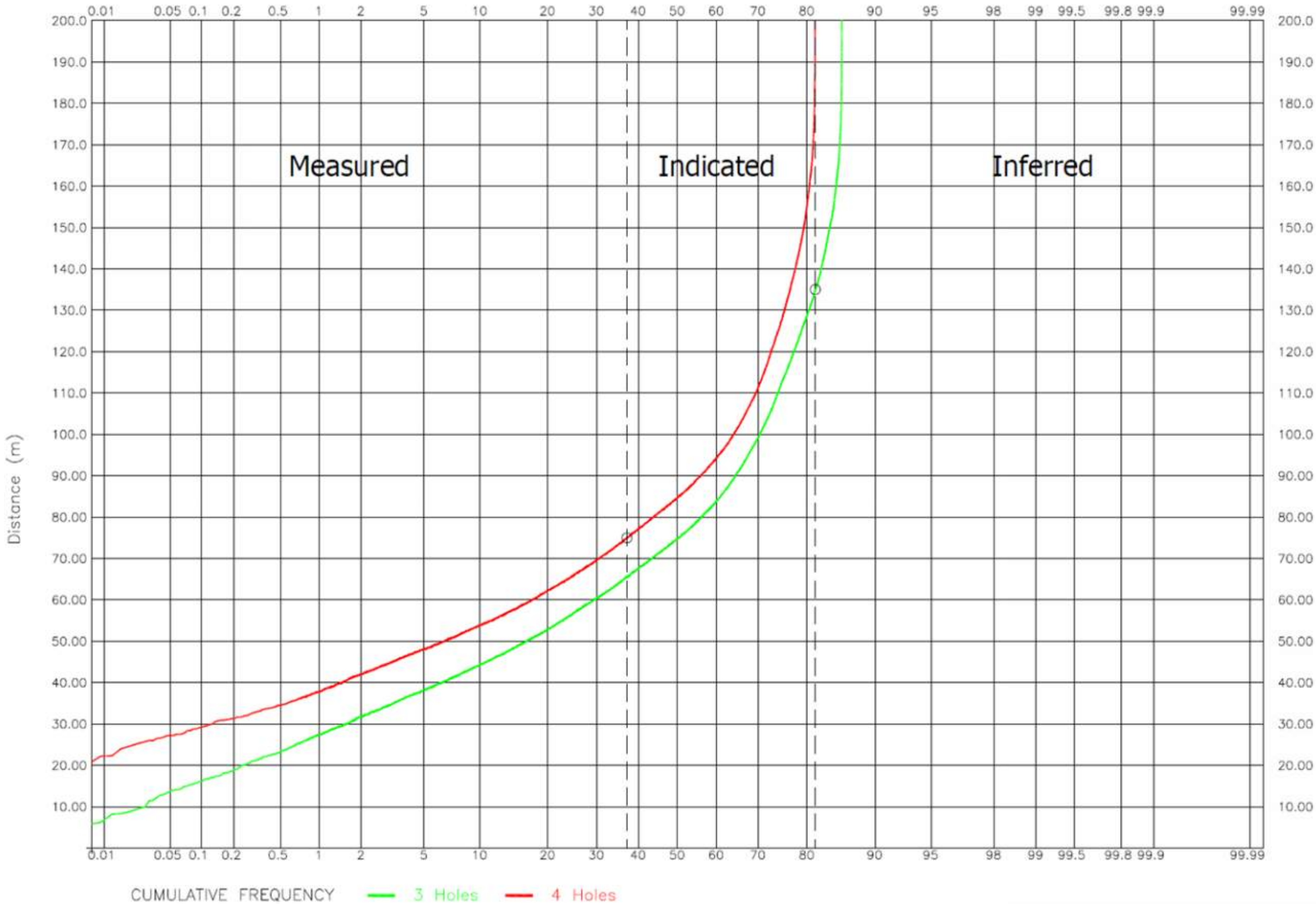



Figure 14-14: Probability Plot of Average Distance to Nearest 3 and 4 Holes. Gold in the Oxide Zone (IMC, 2024)

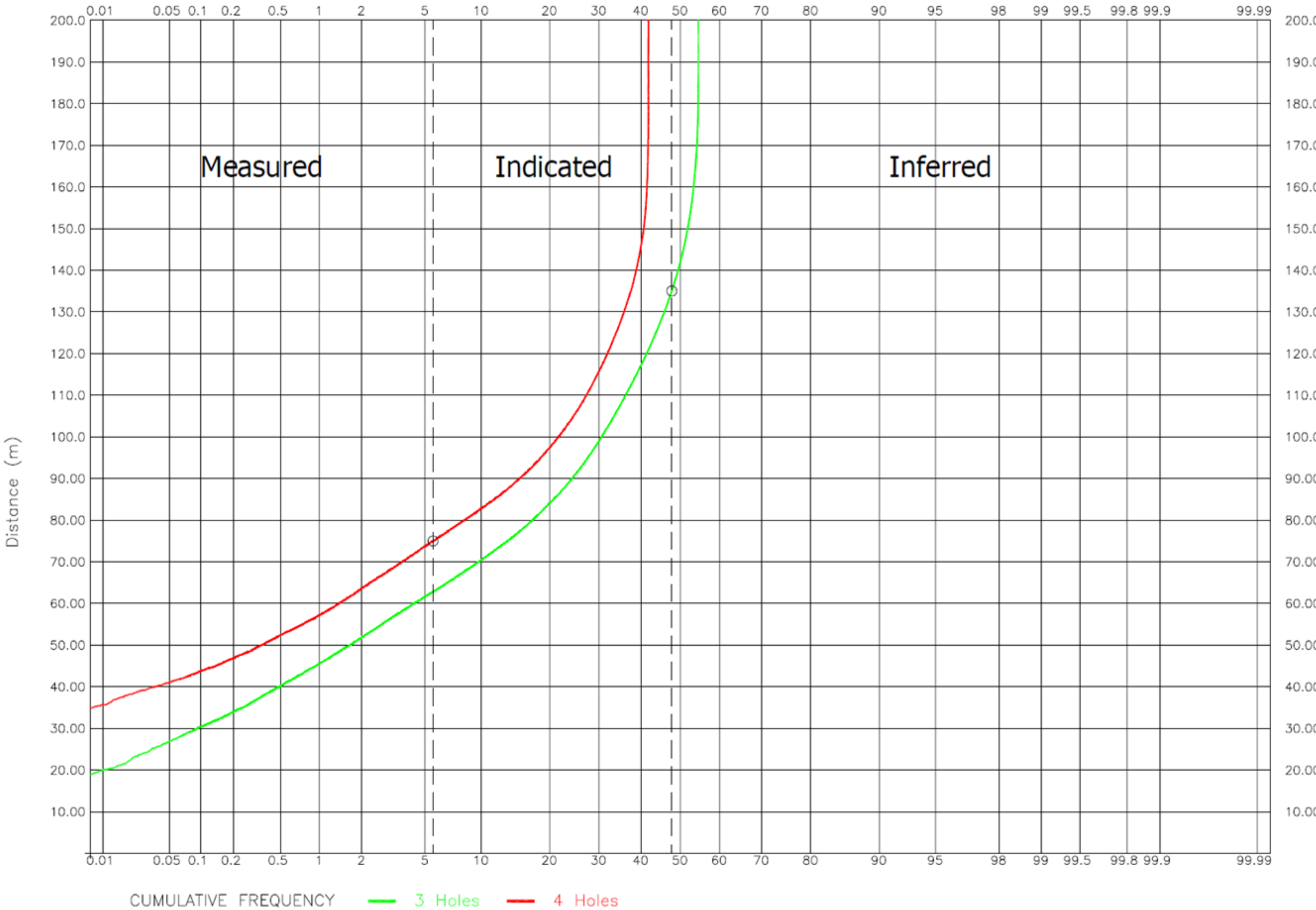



**Figure 14-15: Probability Plot of Average Distance to Nearest 3 and 4 Holes. Gold in the Sulfide Zone (IMC, 2024)**

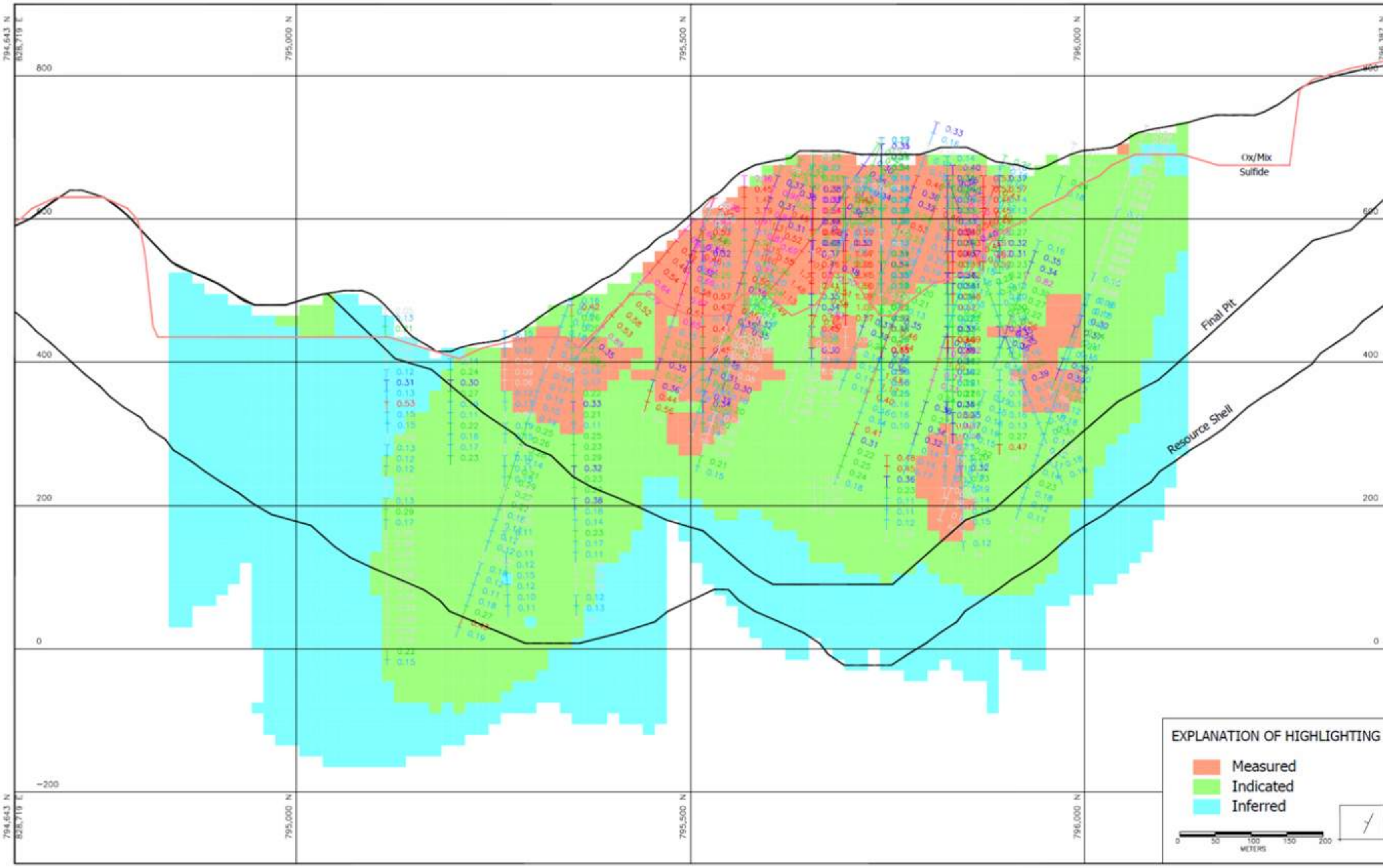



**Figure 14-16: Resource Classification on Section 10,350 (IMC, 2024)**

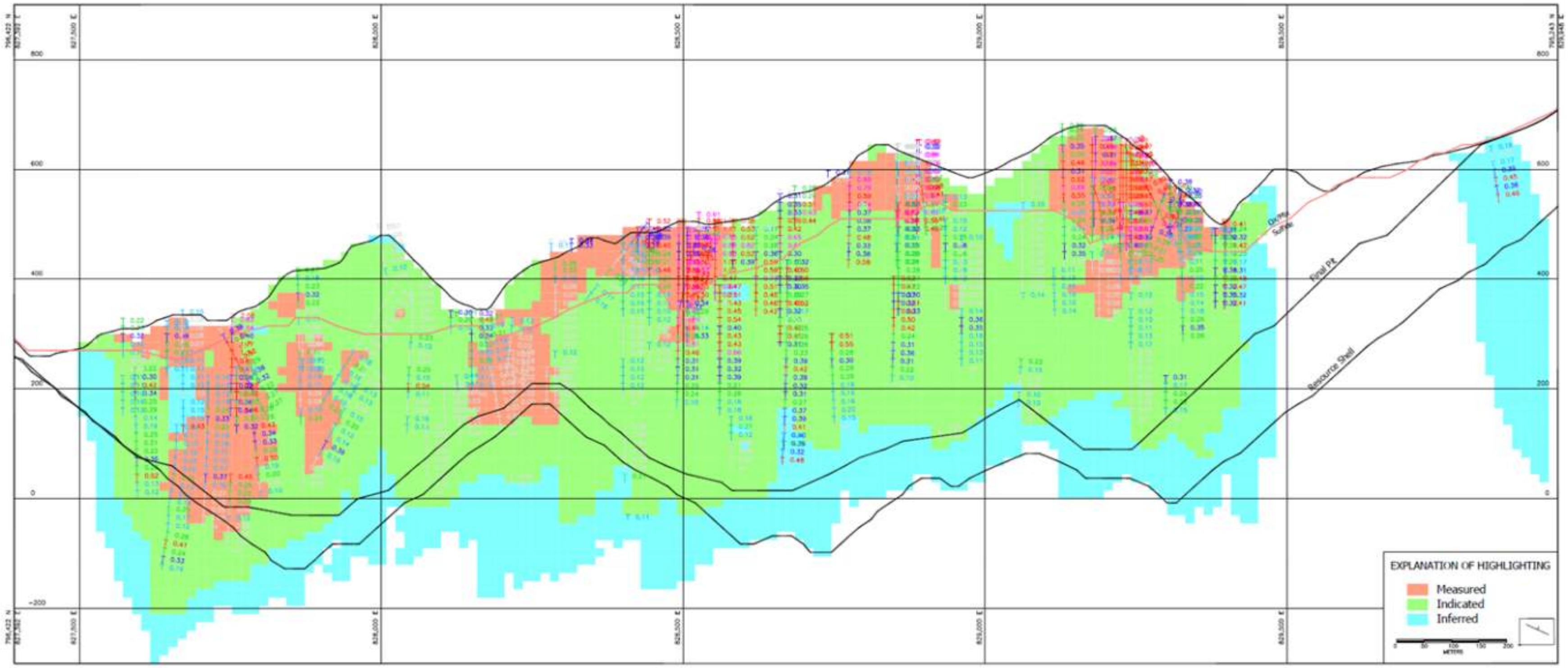



**Figure 14-17: Resource Classification on Section 22 (IMC, 2024)**

### 14.4.10 Bulk Density

Specific gravity measurements, by the water immersion method, were performed on 100 core samples selected for the check assay program described in Section 12.3. Figure 14-18 shows an xy plot of the specific gravity measurement versus soluble copper to total copper ratio by the various rock types. 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 14-8 shows basic statistics of the data by rock type and by higher versus lower soluble copper to total copper ratio.

The values shown on Table 14-8 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$^3$.

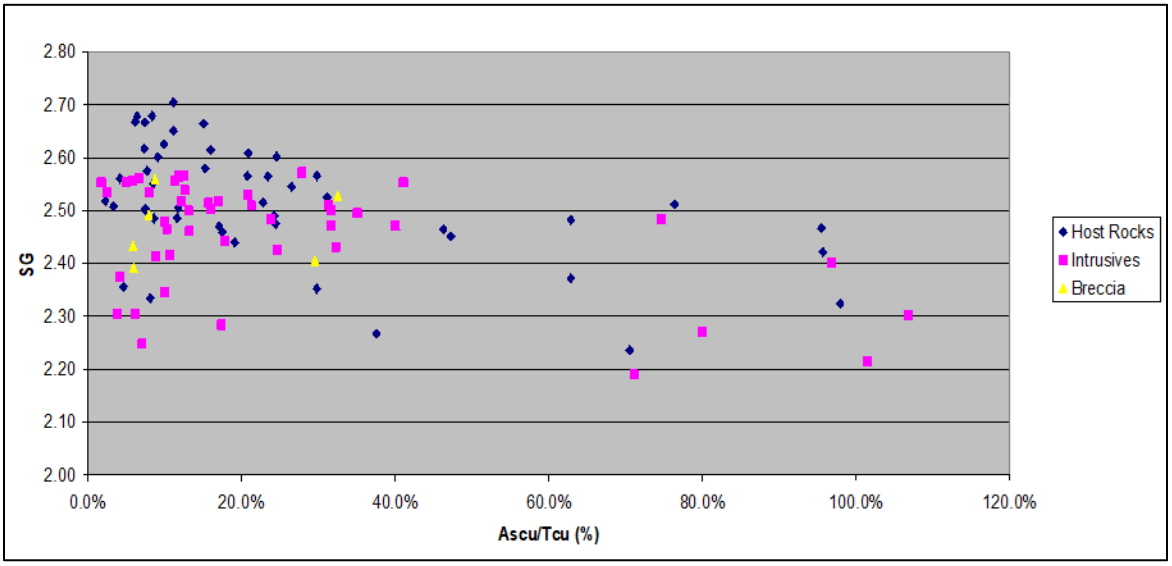



**Figure 14-18: Specific Gravity vs Ascu/Tcu Ratio by Rock Type (IMC, 2011)**

**Table 14-8: Specific Gravity Measurements 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 RECONCILIATION OF 2024 AND 2011 MINERAL RESOURCES

Table 14-9 presents a reconciliation of the August 2011 Mineral Resource estimate with the updated February 2024 estimate. Case 1 shows the 2011 estimate. Measured and Indicated Mineral Resources amounted to 962.3 million tonnes at 0.254% total copper and 0.334 g/t gold for 5.38 billion pounds of contained copper and 10.3 million ounces of contained gold. Inferred Mineral Resources was an additional 188.8 million tonnes at 0.215% total copper and 0.265 g/t gold for 895.2 million contained copper pounds and 1.61 million contained gold ounces. The 2011 Mineral Resource was based on commodity prices of $2.50/lb copper and $1100/oz gold. The floating cone shell for the 2011 Mineral Resource only allowed Measured and Indicated Mineral Resources to contribute to the economics; Inferred Mineral Resource was considered waste for generating the shell, though the Inferred Mineral Resource in the shell was tabulated as part of the Mineral Resource. This is considered as a conservative estimate of Mineral Resource.

Case 3 on the table shows what the 2011 Mineral Resource would have been if Inferred Mineral Resource was allowed to contribute to the economics to develop the constraining pit shell. For this case, Measured and Indicated Mineral Resources amount to 1.07 billion tonnes at 0.244% total copper and 0.323 g/t gold for 5.74 billion pounds of contained copper and 11.1 million ounces of contained gold. Inferred Mineral Resource is an additional 544.5 million tonnes at 0.203% total copper and 0.246 g/t gold for 2.44 billion pounds of contained copper and 4.31 million contained gold ounces. Case 2 on the table shows the differences between the cases. Measured and Indicated Mineral Resources increased by 105.7 million tonnes and Inferred Mineral Resources increased by 355.7 million tonnes.

Case 5 shows the result for a constraining pit shell ran with the August 2023 resource model and the 2011 economic and recovery parameters. Measured and Indicated Mineral Resources amount to 1.11 billion tonnes at 0.242% total copper and 0.321 g/t gold for 5.96 billion pounds of contained copper and 11.5 million ounces of contained gold. Inferred Mineral Resource is an additional 742.4 million tonnes at 0.204% total copper and 0.250 g/t gold for 3.34 billion pounds of contained copper and 5.97 million contained gold ounces. Case 4 on the table shows the differences between the cases. The change in Measured and Indicated Mineral Resources is small and Inferred Mineral Resources increased by 198.0 million tonnes.

Case 7 shows the February 2024 Mineral Resource estimate based on the August 2023 resource model and updated prices, costs, and recovery parameters. Measured and Indicated Mineral Resources amount to 1.12 billion tonnes at 0.242% total copper and 0.318 g/t gold for 5.98 billion pounds of contained copper and 11.4 million ounces of contained gold. Inferred Mineral Resource is an additional 675.2 million tonnes at 0.202% total copper and 0.256 g/t gold for 3.01 billion pounds of contained copper and 5.56 million contained gold ounces. Case 6 on the table shows the differences between the cases. The change in Measured and Indicated Mineral Resources is minimal and Inferred Mineral Resources decreased by 67.2 million tonnes. Increases in unit costs for mining, processing, and G&A offset the increases in commodity prices.

The biggest factor in the increased Mineral Resource estimate since 2011 is due to allowing Inferred Mineral Resources to contribute to economics to develop the constraining pit shell. Changes due to the updated resource model and economic parameters are relatively minimal.

Also, note that for comparison purposes, Case 7 includes the gold in the heap leach resource. This is not included in the official Mineral Resource estimate in Table 14-1.

**Table 14-9: Reconciliation of 2024 and 2011 Mineral Resource Estimates**

| Case | Ktonnes | Tot Cu (%) | Sol Cu (%) | Gold (g/t) | Copper (mlbs) | Gold (koz) |
|---|---|---|---|---|---|---|
| **1. August 2011 Mineral Resource** | | | | | | |
| Measured Mineral Resource | 120,342 | 0.315 | 0.112 | 0.462 | 835.2 | 1,789.3 |
| Indicated Mineral Resource | 841,910 | 0.245 | 0.054 | 0.316 | 4,549.5 | 8,558.8 |
| **Measured/Indicated Resource** | **962,252** | **0.254** | **0.062** | **0.334** | **5,384.7** | **10,348.1** |
| Inferred Mineral Resource | 188,816 | 0.215 | 0.048 | 0.265 | 895.2 | 1,605.8 |
| **2. Due to Allowing Inferred to Contribute to Economics** | | | | | | |
| Measured Mineral Resource | 102 | N.A. | N.A. | N.A. | N.A. | N.A. |
| Indicated Mineral Resource | 105,551 | 0.154 | 0.016 | 0.216 | 359.1 | 732.2 |
| Measured/Indicated Resource | 105,653 | 0.152 | 0.012 | 0.218 | 355.0 | 739.6 |
| Inferred Mineral Resource | 355,653 | 0.197 | 0.030 | 0.236 | 1,541.5 | 2,700.5 |
| **3. 2011 Model with Inferred Allowed to Contribute to Economics** | | | | | | |
| Measured Mineral Resource | 120,444 | 0.313 | 0.109 | 0.464 | 831.1 | 1,796.8 |
| Indicated Mineral Resource | 947,461 | 0.235 | 0.050 | 0.305 | 4,908.6 | 9,290.9 |
| Measured/Indicated Resource | 1,067,905 | 0.244 | 0.057 | 0.323 | 5,739.7 | 11,087.7 |
| Inferred Mineral Resource | 544,469 | 0.203 | 0.036 | 0.246 | 2,436.7 | 4,306.3 |
| **4. Due to 2023 Model** | | | | | | |
| Measured Mineral Resource | 75,048 | 0.305 | 0.143 | 0.331 | 504.9 | 799.0 |
| Indicated Mineral Resource | -28,014 | 0.465 | 0.280 | 0.436 | -287.0 | -393.0 |
| Measured/Indicated Resource | 47,034 | 0.210 | 0.061 | 0.269 | 217.9 | 406.1 |
| Inferred Mineral Resource | 197,960 | 0.207 | 0.010 | 0.261 | 902.3 | 1,661.2 |
| **5. 2023 Model with 2011 Economics** | | | | | | |
| Measured Mineral Resource | 195,492 | 0.310 | 0.122 | 0.413 | 1,336.0 | 2,595.8 |
| Indicated Mineral Resource | 919,447 | 0.228 | 0.043 | 0.301 | 4,621.6 | 8,898.0 |
| Measured/Indicated Resource | 1,114,939 | 0.242 | 0.057 | 0.321 | 5,957.6 | 11,493.8 |
| Inferred Mineral Resource | 742,429 | 0.204 | 0.029 | 0.250 | 3,339.0 | 5,967.5 |
| **6. Due to Updated Prices, Costs, Recoveries** | | | | | | |
| Measured Mineral Resource | 1,176 | -0.526 | 0.958 | -2.430 | -13.6 | -91.9 |
| Indicated Mineral Resource | 3,544 | 0.488 | 0.303 | 0.301 | 38.2 | 34.3 |
| Measured/Indicated Resource | 4,720 | 0.236 | 0.467 | -0.379 | 24.5 | -57.6 |
| Inferred Mineral Resource | -67,184 | 0.224 | 0.009 | 0.190 | -331.9 | -409.8 |
| **7. February 2024 Mineral Resource** | | | | | | |
| Measured Mineral Resource | 196,668 | 0.305 | 0.127 | 0.396 | 1,322.4 | 2,504.0 |
| Indicated Mineral Resource | 922,991 | 0.229 | 0.044 | 0.301 | 4,659.8 | 8,932.3 |
| **Measured/Indicated Resource** | **1,119,659** | **0.242** | **0.059** | **0.318** | **5,982.2** | **11,436.2** |
| Inferred Mineral Resource | 675,245 | 0.202 | 0.031 | 0.256 | 3,007.1 | 5,557.8 |

Includes gold in leach resource for comparison

# 15 MINERAL RESERVE ESTIMATES

## 15.1 MINERAL RESERVE

Table 15-1 presents the Mineral Reserve estimate for the Kingking Project. There are Mineral Reserves amenable to milling and Mineral Reserves amenable to heap leaching. The Proven and Probable Mineral Reserves amenable to milling amount to 848.9 million tonnes at 0.26% total copper and 0.36 g/t gold for 4.84 billion pounds of contained copper and 9.77 million ounces of contained gold. The Proven and Probable Mineral Reserve amenable to heap leaching amounts to 110.5 million tonnes at 0.23% copper for 555 million pounds of contained copper. The effective date of this Mineral Reserve estimate is April 15, 2024. The low-grade stockpile portion of the Mineral Reserve is economic material, but lower grade, that will be stockpiled and processed at the end of open-pit operations. The bottom portion of the table shows the total copper Mineral Reserve for milling and leaching. The gold Mineral Reserve is as shown for Mineral Reserves amenable to milling.

The Mineral Reserve estimate is based on an open-pit mine plan and mine production schedule developed by IMC. Figure 15-1 shows the final pit that is the basis for the Mineral Reserve estimate. The Mineral Reserve estimate is based on commodity prices of $3.75/lb copper and $1,800/oz gold. Measured Mineral Resource in the mine production schedule was converted to Proven Mineral Reserve, and Indicated Mineral Resource in the schedule was converted to Probable Mineral Reserve.

The Mineral Reserves are classified in accordance with the "CIM Definition Standards – For Mineral Resources and Mineral Reserves" adopted May 10, 2014, by the CIM Council (as amended, the "CIM Definition Standards") in accordance with the requirements of NI 43-101. Mineral Reserve estimates reflect the reasonable expectation that all necessary permits and approvals will be obtained and maintained.

The QP for this section does not believe that there are significant risks to the Mineral Reserve estimate based on metallurgical or infrastructure factors or environmental, permitting, legal, title, taxation, socio-economic, marketing, or political factors. There has been a significant amount of metallurgical testing, however recoveries lower than forecast would result in loss of revenue for the Kingking Project. Other risks to the Mineral Reserve estimate are related to economic parameters such as prices lower than forecast or costs higher than the current estimates. The impact of these is modeled in the sensitivity study with the economic analysis in Section 22.

All the mineralization comprised in the Mineral Reserve estimate with respect to the Kingking Project is contained on mineral titles controlled by St. Augustine and NADECOR, its partner in the Kingking Project. The current pit design includes small amounts of waste material are located outside of the mineral tenement title. The Mineral Reserve estimate has been prepared based on the Qualified Person's reasoned judgment, in accordance with Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Best Practices Guidelines and his professional standards of competence, that there is a reasonable expectation that all the necessary permits, agreements and approvals will be obtained and maintained, including the additional agreements with other parties to allow mining of waste material on their mineral titles.

The Mineral Reserve includes allowances for mining dilution and ore loss. The QP for this section believes that reasonable amounts of dilution and loss were incorporated into the block model used for this Technical Report. Compositing assays into composites and estimating blocks with multiple composites introduces some smoothing of model grades that are analogous to dilution and ore loss effects.

**Table 15-1: Mineral Reserve**

| Mineral Reserve (Milling): | Tonnes Mt | NSR ($/t) | Tot Cu (%) | Sol Cu (%) | Gold (g/t) | Copper (Mlbs) | Gold (Koz) |
|---|---|---|---|---|---|---|---|
| **Proven Mineral Reserve:** | **142.3** | **37.83** | **0.33** | **0.12** | **0.49** | **1,037** | **2,253** |
| Oxide Mill Ore | 45.4 | 58.56 | 0.52 | 0.30 | 0.71 | 516 | 1,033 |
| Sulfide Mill Ore | 76.5 | 31.90 | 0.26 | 0.04 | 0.45 | 444 | 1,117 |
| Low Grade Stockpile | 20.4 | 13.93 | 0.17 | 0.02 | 0.16 | 78 | 103 |
| **Probable Mineral Reserve:** | **706.5** | **25.51** | **0.24** | **0.04** | **0.33** | **3,803** | **7,519** |
| Oxide Mill Ore | 52.0 | 43.46 | 0.36 | 0.21 | 0.59 | 412 | 986 |
| Sulfide Mill Ore | 499.7 | 27.08 | 0.25 | 0.03 | 0.36 | 2,776 | 5,751 |
| Low Grade Stockpile | 154.9 | 14.43 | 0.18 | 0.02 | 0.16 | 615 | 782 |
| **Proven/Probable Reserve:** | **848.9** | **27.57** | **0.26** | **0.06** | **0.36** | **4,840** | **9,771** |
| Oxide Mill Ore | 97.4 | 50.50 | 0.43 | 0.25 | 0.64 | 928 | 2,018 |
| Sulfide Mill Ore | 576.2 | 27.72 | 0.25 | 0.03 | 0.37 | 3,220 | 6,868 |
| Low Grade Stockpile | 175.3 | 14.37 | 0.18 | 0.02 | 0.16 | 692 | 885 |
| **Mineral Reserve (Leaching):** | **Tonnes Mt** | **NSR ($/t)** | **Tot Cu (%)** | **Sol Cu (%)** | **Gold (g/t)** | **Copper (Mlbs)** | **Gold (Koz)** |
| Proven Mineral Reserve | 50.2 | 14.97 | 0.25 | 0.16 | N.A. | 275 | N.A. |
| Probable Mineral Reserve | 60.4 | 12.22 | 0.21 | 0.12 | N.A. | 280 | N.A. |
| **Prov/Prob Leach Reserve** | **110.5** | **13.47** | **0.23** | **0.14** | **N.A.** | **555** | **N.A.** |
| **Copper Mineral Reserve Milling and Leaching** | **Tonnes Mt** | **NSR ($/t)** | **Tot Cu (%)** | **Sol Cu (%)** | **Gold (g/t)** | **Copper (Mlbs)** | **Gold (Koz)** |
| **Proven/Probable Reserve** | **959.4** | **25.95** | **0.26** | **0.06** | **N.A.** | **5,396** | **N.A.** |
| Proven Mineral Reserve | 192.5 | 31.87 | 0.31 | 0.13 | N.A. | 1,312 | N.A. |
| Probable Mineral Reserve | 766.9 | 24.46 | 0.24 | 0.05 | N.A. | 4,083 | N.A. |

Notes:

1. The Mineral Reserve estimate has an effective date of April 15, 2024 and was prepared using the CIM Definition Standards (May 10, 2014).
2. Columns may not sum exactly due to rounding.
3. Mineral Reserves are based on commodity prices of $3.75/lb copper and $1,800/oz gold.
4. Mineral Reserves amenable to milling are based on net of processing (NOP) cut-offs that vary by time period to balance mine and plant production capacities (see Section 16). They range from a high of $21/t to a low of $0.01/t (internal cut-off grade). NOP is calculated as the NSR value minus processing and G&A costs.
5. The NSR value due to mill copper is based on a flotation component (concentrates) and an agitation tail leach component (cathode). For flotation, NSR($/t) = $70.10 x copper (%) x recovery where recovery is variable but averages 35.8% for oxides and 68.3% for sulfides. For tails agitation, NSR($/t) = $74.85 x copper (%) x recovery where recovery averages 57.4% for oxide and 14.6% for sulfide.
6. The NSR value for mill gold is based on a flotation component (concentrates) and a gravity component. For flotation, NSR($/t) = $52.08 x gold (g/t) x recovery where recovery averages 38.6% for oxides and 55.8% for sulfides. For gravity, NSR($/t) = $54.31 x gold (g/t) x recovery where recovery averages 23.7% for oxides and 19.0% for sulfides.
7. Total NSR for milling is the sum of the copper and gold components. The NSR calculations account for smelter/refinery treatment charges and payables.
8. Mineral Reserves amenable to heap leaching are based on an NSR cut-off of $3.26/t. For leach material, NSR($/t) = $74.85 x copper (%) x recovery where recovery is variable but averages 83.8%.
9. Table 15-2 accompanies this Mineral Reserve estimate and shows all relevant parameters.

## 15.2 ECONOMICS PARAMETERS

Table 15-2 shows the economic parameters used for mine design and production scheduling. The base copper and gold prices are $3.75/lb copper and $1,800/oz gold. The QP for this section believes these prices to be reasonable based on 1) current spot prices and historical 3-year trailing averages, 2) prices used by other companies for comparable projects, and 3) long range consensus price forecasts prepared by various bank economies.

The table shows parameters for five different mineral reserve types:

- Heap leach reserve; only copper is recovered.

- Mixed oxide/sulfide mill reserve with conventional copper flotation and agitation leach of the tails; copper and gold are recovered to concentrate and additional copper is recovered to cathode. A portion of the gold is recovered by gravity.
- Mixed oxide/sulfide mill reserve with conventional copper flotation only; copper and gold are recovered to concentrate. A portion of the gold is recovered by gravity.
- Sulfide mill reserve with conventional copper flotation and agitation leach of the tails; copper and gold are recovered to concentrate and additional copper is recovered to cathode. A portion of the gold is recovered by gravity.
- Sulfide mill reserve with conventional copper flotation only; copper and gold are recovered to concentrate. A portion of the gold is recovered by gravity.

The decision to route mixed oxide/sulfide reserve to the heap leach versus the mill is a routing decision based on highest value net of processing costs. This calculation was on a block-by-block basis in the model.

On a block-by-block basis the blocks for which the agitation leach process contributed additional profit, versus the blocks that were more profitable without agitation, were determined. This was important for the design of the final pit and production scheduling. In particular, the agitation process is not economic for the deep sulfide reserve that drives the final pit design.

The mining costs on the table are estimated based on owner operation of mining equipment. The estimates are based on current costs for fuel, equipment parts, power, lubricant, blasting agents, and labor costs. The base mining cost per tonne is the same for all material types and the haulage cost varies by material type. In particular, the waste haulage cost is 25% higher than the cost for leach and mill reserves. The estimate also includes an allowance for replacement capital equipment.

Estimated process costs were provided by Met Engineering, LLC, a client consultant, and confirmed by M3 and Art Ibrado of Ft. Lowell Consulting PLLC, the QP for mineral processing. It can be seen for mill reserve the unit costs are broken out by flotation, optional agitation, and tailings costs. G&A unit costs are $1.508/t for mill reserve and $0.322/t for heap leach reserve.

Process recoveries are incorporated into the resource block model on a block-by-block basis. The equations are presented below. The recoveries shown on the table are averages for each reserve type based on the mine production schedule.

**Table 15-2: Economic Parameters for Kingking Mine Design**

| Parameter | Units | Mixed Ox/Slf Heap Leach 50 | Mixed Ox/Slf Float/Agit 10 | Mixed Ox/Slf Float Only 20 | Sulfide Float/Agit 30 | Sulfide Float Only 40 | Waste |
|---|---|---|---|---|---|---|---|
| Copper Price Per Lb | (US$) | 3.75 | 3.75 | 3.75 | 3.75 | 3.75 | |
| Gold Price Per Oz | (US$) | 1800 | 1800 | 1800 | 1800 | 1800 | |
| **Mining Cost Per Tonne - Owner Operation** | | | | | | | |
| Base Mining Cost Without Haulage | (US$) | 0.926 | 0.926 | 0.926 | 0.926 | 0.926 | 0.926 |
| Base Haulage Cost | (US$) | 0.579 | 0.618 | 0.618 | 0.618 | 0.618 | 1.029 |
| Mine Replacement Capital Per Tonne | (US$) | 0.130 | 0.130 | 0.130 | 0.130 | 0.130 | 0.130 |
| Mining Cost | (US$) | 1.635 | 1.674 | 1.674 | 1.674 | 1.674 | 2.085 |
| **Process Cost Per Tonne** | | | | | | | |
| Crushing, Grinding, Flotation | (US$) | N.A. | 4.728 | 4.728 | 4.958 | 4.958 | |
| Agitated Leach | (US$) | N.A. | 4.192 | 0.000 | 4.192 | 0.000 | |
| Tailings | (US$) | N.A. | 1.187 | 1.187 | 1.187 | 1.187 | |
| Heap Leach | (US$) | 3.385 | N.A. | N.A. | N.A. | N.A. | |
| Processing Cost Per Ore Tonne | (US$) | 3.385 | 10.107 | 5.915 | 10.337 | 6.145 | |
| G&A Cost Per Ore Tonne | (US$) | 0.322 | 1.508 | 1.508 | 1.508 | 1.508 | |
| **Process Recoveries (Average Based on Recovery Equations)** | | | | | | | |
| Copper (Concentrate + Agitation) | (%) | N.A. | 93.2% | 35.8% | 82.9% | 68.3% | |
| Copper to Concentrate | (%) | N.A. | 35.8% | 35.8% | 68.3% | 68.3% | |
| Copper to Agitation (or Heap Leach) | (%) | 83.8% | 57.4% | 0.0% | 14.6% | 0.0% | |
| Gold (Concentrate + Gravity) | (%) | N.A. | 62.3% | 62.3% | 74.8% | 74.8% | |
| Gold to Concentrate | (%) | N.A. | 38.6% | 38.6% | 55.8% | 55.8% | |
| Gold to Gravity Concentrate | (%) | N.A. | 23.7% | 23.7% | 19.0% | 19.0% | |
| **Conventional Smelting/Refining** | | | | | | | |
| Smelting/Refining Payable for Copper | (%) | N.A. | 96.0% | 96.0% | 96.0% | 96.0% | |
| Smelting/Refining Payable for Gold | (%) | N.A. | 95.0% | 95.0% | 95.0% | 95.0% | |
| SRF Cost Per Pound Copper | (US$) | N.A. | 0.263 | 0.263 | 0.263 | 0.263 | |
| Gold Refining Per Ounce | (US$) | N.A. | 5.00 | 5.00 | 5.00 | 5.00 | |
| **Gold to Gravity Concentrate** | | | | | | | |
| Refinery Payable (with Transport/Insurance) | (%) | N.A. | 98.9% | 98.9% | 98.9% | 98.9% | |
| Gold Refining Per Ounce | | N.A. | 2.00 | 2.00 | 2.00 | 2.00 | |
| **Site Solvent Extraction/Electrowinning** | | | | | | | |
| Payable Copper | (%) | 100% | 100% | 0% | 100% | 0% | |
| SXEW Per Pound Copper | (US$) | 0.164 | 0.164 | 0.000 | 0.164 | 0.000 | |
| Cathode Freight/Insurance | (US$) | 0.012 | 0.012 | 0.000 | 0.012 | 0.000 | |
| Gross Royalty | (%) | 5.0% | 5.0% | 5.0% | 5.0% | 5.0% | |
| **NSR Factors (Applied to Recovered Grade)** | | | | | | | |
| Copper in Concentrate | (US$/t) | N.A. | 70.100 | 70.100 | 70.100 | 70.100 | |
| Copper in Agitation Leach | (US$/t) | N.A. | 74.853 | 0.000 | 74.853 | 0.000 | |
| Gold in Concentrate | (US$/t) | N.A. | 52.085 | 52.085 | 52.085 | 52.085 | |
| Gold in Gravity Concentrate | (US$/t) | N.A. | 54.313 | 54.313 | 54.313 | 54.313 | |
| Copper in Heap Leach | (US$/t) | 74.853 | N.A. | N.A. | N.A. | N.A. | |
| **NSR Cutoffs** | | | | | | | |
| Breakeven | (US$/t) | 5.34 | 13.29 | 9.10 | 13.52 | 9.33 | |
| Internal | (US$/t) | 3.26 | 11.20 | 7.01 | 11.43 | 7.24 | |
| With Stockpile Rehandle@ $1.10 | (US$/t) | 4.36 | 12.30 | 8.11 | 12.53 | 8.34 | |

Table 15-3 shows the basis for the payable copper in concentrate and smelting, refining, and freight (SRF) cost per pound. The estimates are based on a smelter treatment charge of $65 per metric tonne and refining charge of $0.065/lb. Refinery terms for gold in concentrate are estimated at 95.0% payable and $5.00/oz. Refinery terms for gold in gravity concentrates are estimated as 98.9% payable and $2.00/oz.

**Table 15-3: Copper Smelting, Refining, and Freight**

| Parameter | Units | Oxide | Sulfide |
|---|---|---|---|
| Concentrate Copper Grade | (%) | 25.0 | 25.0 |
| Payable Percentage of Copper | (%) | 96.0% | 96.0% |
| Payable Pounds per tonne of concentrate | (lbs) | 529.1 | 529.1 |
| Treatment Per Dry Tonne | (US$) | 65.00 | 65.00 |
| Refining Per Pound | (US$) | 0.065 | 0.065 |
| Transport Per Wet Tonne | (US$) | 36.00 | 36.00 |
| Moisture Content | (%) | 10.0% | 10.0% |
| Treatment Per Pound | (US$) | 0.123 | 0.123 |
| Refining Per Pound | (US$) | 0.065 | 0.065 |
| Transport Per Pound | (US$) | 0.076 | 0.076 |
| **Total SRF Per Pound Copper** | **(US$)** | **0.263** | **0.263** |

Copper recovered from the leach pad and agitated tails will be recovered in an SX-EW plant on the property. SX-EW and cathode freight and insurance are estimated at $0.176/lb copper.

The economic parameters also include an NSR royalty of 5%. The NSR factors and NSR cut-offs shown on Table 15-2 are discussed below.

## 15.3 METAL RECOVERIES

The process recovery equations are complex and were provided by Met Engineering, LLC, a client consultant, and confirmed by Art Ibrado of Ft. Lowell Consulting PLLC, the QP for mineral processing. For heap leach reserve, the recovery equation is:

$$Rec(\%) = 90\%(0.0151934 + 0.0526427*totcu + 1.1501882*solcu + 0.03629*ratio)/totcu$$

Where totcu is total copper, solcu is soluble copper (both in %) and ratio is solcu/totcu. The recovery is capped at 92%.

The recovery of total copper to concentrate is:

$$Rec(\%) = 100\%(1.2038653 + 0.131888*\ln(solcu) + 0.0681887*gold^{0.6757} - 0.0589333*\ln(nsol) - 1.42294*ratio)$$

Where gold is gold in ppm, nsol is totcu – solcu, and ln is the logarithm, base e. The recovery is capped at 90%. This equation can result in values less than 0; these were set to 0.

The copper recoveries to concentrate and concentrate grade are then used to calculate the mass of tails and the total and soluble copper grades of the tails. The details of this mass balance are not shown here. Copper recovery for agitation leach of copper in the tails is:

$$Rec(\%) = 100\%(0.1052638 + 0.918662*(t\_solcu/t\_totcu))$$

Where t_solcu and t_totcu are the soluble and total copper grades of the tails. This was capped at a maximum of 95%; the minimum is 10.5%. Once this recovery of copper in tails was calculated it was also used to calculate agitation recovery in terms of percent of total copper in the original plant feed.

The recovery of gold to the gravity concentrate is estimated as 19% when the soluble copper to total copper ratio is less than 50% and 26% when the ratio is greater than or equal to 50%.

The gold head grade to the concentrator is adjusted to account for the gravity gold recovery and the recovery of gold to concentrate is:

$$\text{Rec}(\%) = 100\%(0.3483337 + 0.2169788*\text{c\_gold}^{0.266} + 0.0878149*\ln(\text{nsol}) - 0.0907290*\ln(\text{solcu}))$$

Where c_gold is the gold grade in g/t net of gravity gold. All other variables and operators are as previously defined. Total gold recovery, gravity plus concentrate, was capped at 85%.

## 15.4 NSR VALUES

Due to multiple products and variable recoveries for copper and gold, NSR values, in US$/t, were calculated for each model block to classify blocks into potential reserve or waste.

For leach reserve, the calculation is:

$$\begin{aligned}\text{NSR\_lch} &= (\$3.75 - 0.176) \times 0.95 \times \text{totcu}(\%) \times \text{recov} \times 22.046 \\ &= \$74.853 \times \text{totcu}(\%) \times \text{recov}\end{aligned}$$

Note that 74.853 is the NSR factor shown on Table 15-2 for heap leach resource. Recov is a decimal between 0 and 1 instead of a percent. The 0.95 term accounts for the royalty.

For mill reserve (mixed oxide/sulfide), the calculations are:

$$\begin{aligned}\text{NSR\_cu\_conc} &= (\$3.75 - \$0.263) \times 0.96 \times 0.95 \times \text{totcu}(\%) \times \text{recov} \times 22.046 \\ &= \$70.10 \times \text{recovered copper to concentrate } (\%)\end{aligned}$$

$$\begin{aligned}\text{NSR\_cu\_agit} &= (\$3.75 - \$0.176) \times 1.0 \times 0.95 \times \text{totcu}(\%) \times \text{recov} \times 22.046 \\ &= \$74.853 \times \text{recovered copper to agitation } (\%)\end{aligned}$$

$$\begin{aligned}\text{NSR\_au\_conc} &= (\$1800 - \$5.00) \times 0.95 \times 0.95 \times \text{gold(g/t)} \times \text{recov} / 31.103 \\ &= \$52.085 \times \text{recovered gold to concentrate (g/t)}\end{aligned}$$

$$\begin{aligned}\text{NSR\_au\_grav} &= (\$1800 - \$2.00) \times 0.989 \times 0.95 \times \text{gold(g/t)} \times \text{recov} / 31.103 \\ &= \$54.313 \times \text{recovered gold to gravity (g/t)}\end{aligned}$$

$$\text{NSR\_mill} = \text{NSR\_cu\_conc} + \text{NSR\_cu\_agit} + \text{NSR\_au\_conc} + \text{NSR\_au\_grav}$$

Note that for tails to be profitable in the agitation leach circuit, the NSR_cu_agit value needs to exceed the incremental agitation cost of 4.192/t. This is a minimum recoverable copper grade of 0.056% in the equation for NSR_cu_agit above.

For mixed oxide/sulfide zone material, the routing for heap leach versus the mill was based on the highest value of NSR less process/G&A costs. The additional gravity gold recovery component tends to route more material to the mill versus heap leach than would occur without the gravity gold circuit.

The bottom of Table 15-2 shows NSR cut-offs for each reserve type. For sulfide mill reserve with agitation, the breakeven cut-off is the mining + processing + G&A cost and amounts to $13.52/t. Internal cut-off applies to blocks that must be removed from the pit, so mining is considered a sunk cost. For sulfide mill reserve (with agitation), this is $10.337 + 1.508 – ($2.085 - $1.674) = $11.43. The incremental mining cost in the calculation reflects the lower haulage cost to the crusher versus the waste dumps. The table also shows a minimum NSR cut-off for material that is stockpiled and rehandled. The rehandle cost is estimated at $1.10/t. The breakeven and internal cut-offs for mixed oxide/sulfide mill reserve are slightly lower due to the lower crushing, grinding and flotation cost.

Since the NSR cut-offs vary by reserve type, it is convenient to add a Net of Process (NOP) parameter for production scheduling. NOP is defined as the NSR minus processing and G&A costs. Internal NOP cut-off is any value greater than 0, i.e., $0.01/t and is the same for all reserve types.

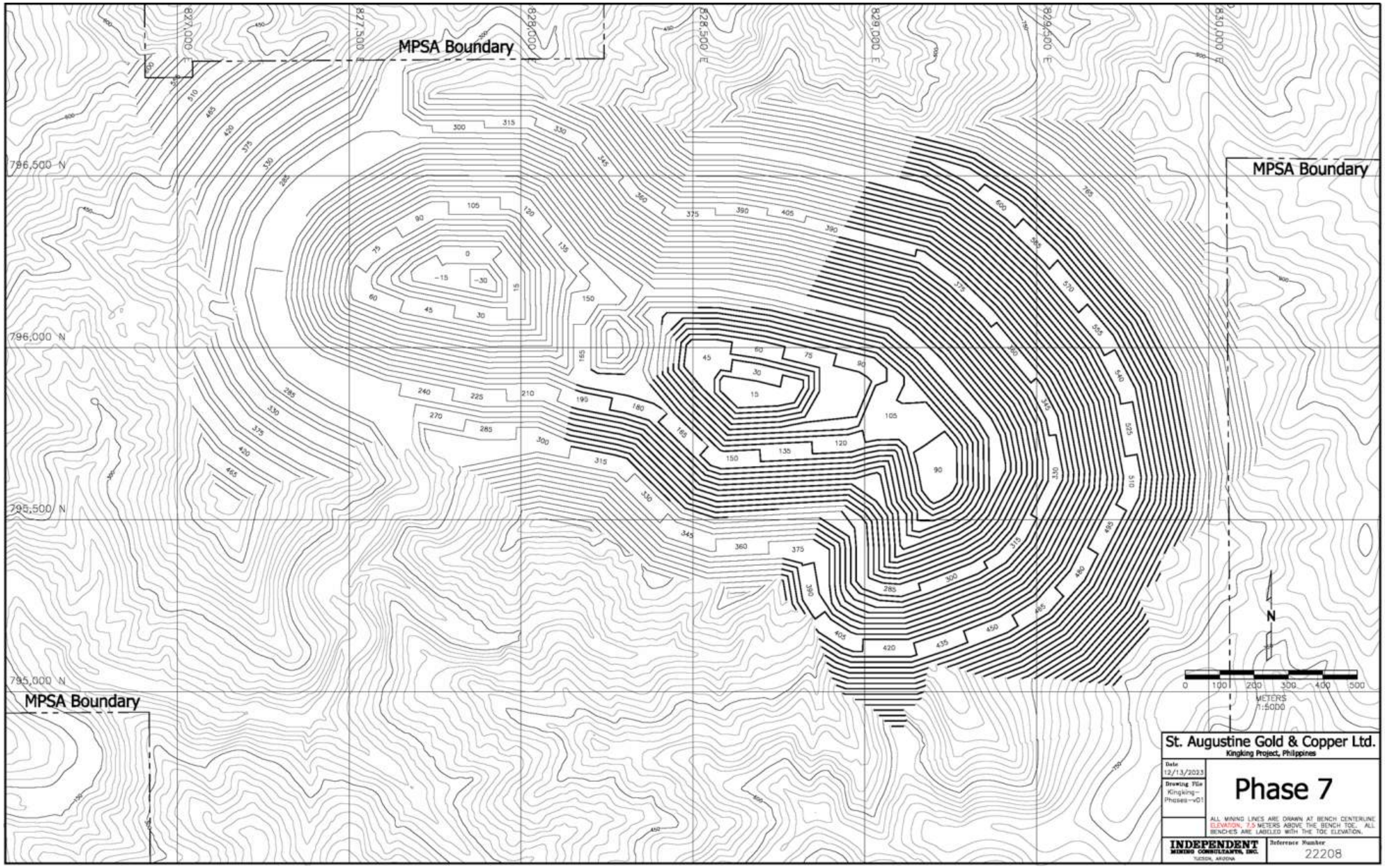



Figure 15-1: Kingking 2023 Final Pit (IMC, 2023)

# 16 MINING METHODS

## 16.1 OPERATING PARAMETERS AND CRITERIA

The Kingking mine will be a conventional open pit mine. Mine operations will consist of drilling holes with large diameter (27.3 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, large hydraulic 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 areas, stockpiles, and the roads.

The mine plan was developed to deliver mill ore at a nominal rate of 60,000 tpd or 21.9 million tonnes per year, though the production rate varies depending on the relative amounts of mixed oxide/sulfide versus sulfide material. Heap leach ore is processed at a maximum rate of 40,000 tpd or 14.6 million tonnes per year. The mining rate (ore plus waste) will be approximately 178,000 tpd or 65 million tonnes per year. The heap leach process is expected to start 18 months before the mill starts and finishes approximately 19 years into the Kingking 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 Kingking Project, which are commensurate with a scoping-level study. The slope study used information from drill hole data collected from five oriented core drill holes and three geohydrology drill holes 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 required to a feasibility level study for the Kingking Project. 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 Kingking Project is still under development. The incorporation of such structures in the geotechnical pit design is recommended for a feasibility level study report. Therefore, the overall slope angles provided herein will be adjusted as needed upon completion of additional slope studies.

The geotechnical design presented herein is based on analyses of the geotechnical drilling program completed in 2011. 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 Dioritie 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 Kingking 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 Kingking Project should target drilling in these sectors to validate the assumptions put forward in this section.

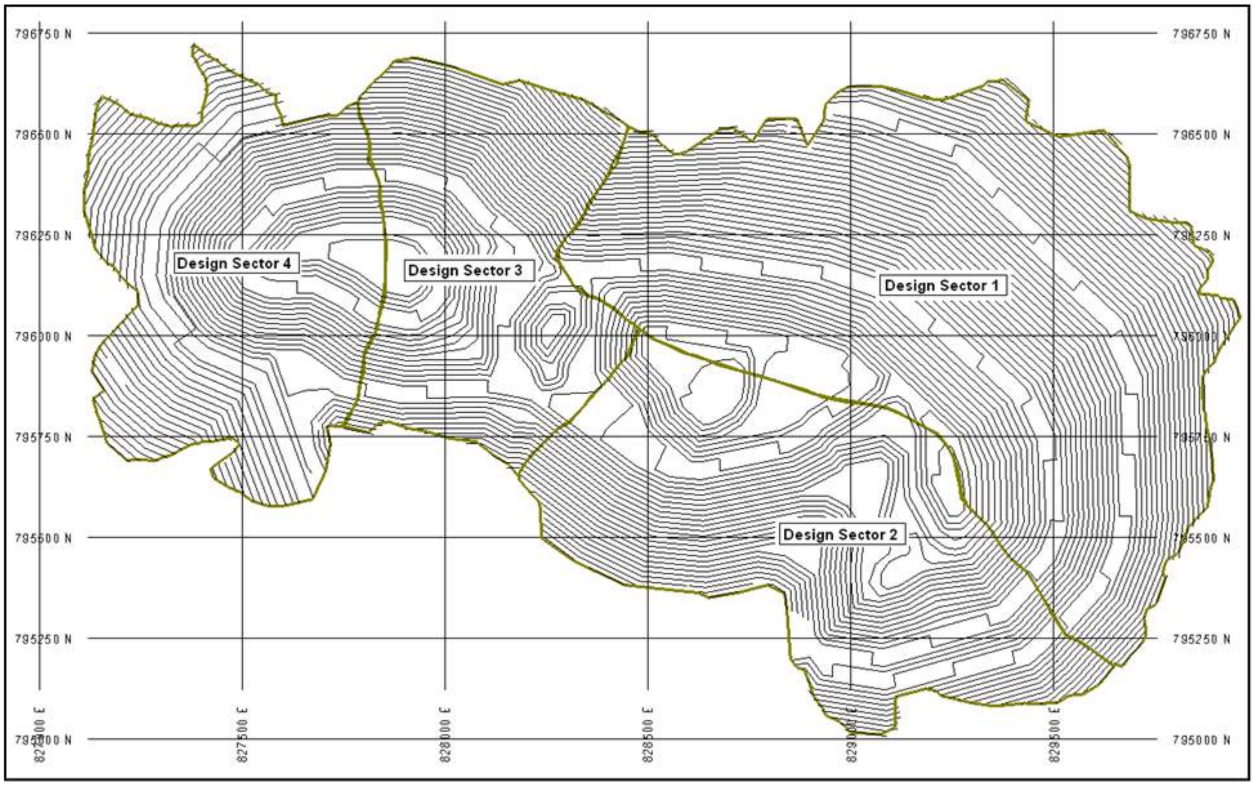
The proposed preliminary pit slope configuration and design sectors for which they apply are presented respectively in Table 16-1 and Figure 16-1.

**Table 16-1: Proposed Preliminary Pit Slope Configuration**

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

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%

It is the opinion of the QP for this section that the work done to date is adequate for this current Technical Report. As discussed above, significant additional work is required prior to detailed design and the commencement of mining operations.




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 (AMEC, 2011)**

## 16.3 PIT DIVERSION DESIGN

A significant aspect of the Kingking 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 12, 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 12 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 channel, safety berm(s), and access road(s). The layout of the channel should fit inside the pit bench as illustrated in Figure 16-2.

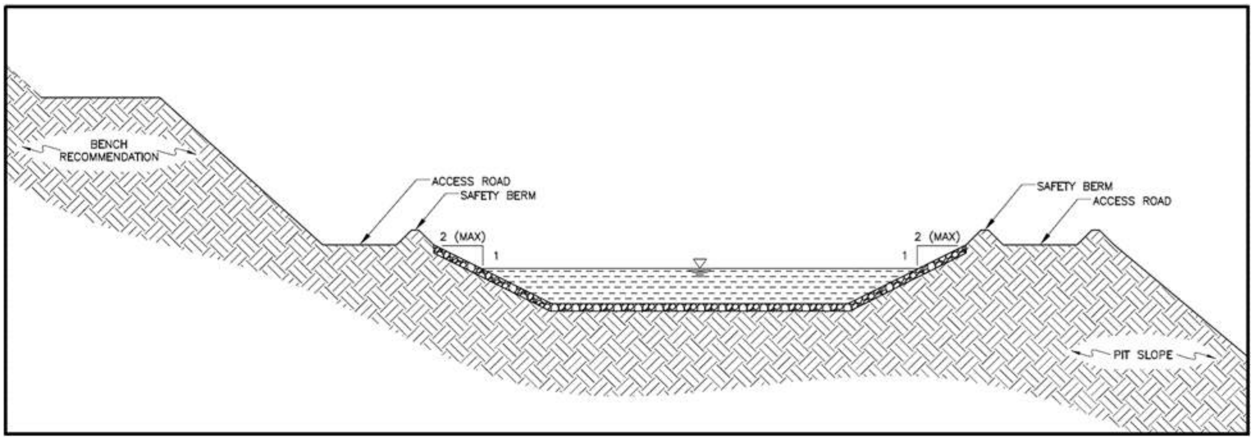



**Figure 16-2: Pit Diversion Schematic**

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 help minimize erosion; however on-going geotechnical investigations suggest that rock in this area may not be competent. Poor quality rock would be susceptible to erosion and a likely conduit for significant seepage loss from the channel into the pit, both of which are undesirable. If poor rock quality is encountered, the channel will require a low permeability armor layer. Required armoring may include spot treatment of higher permeable areas (if a majority of the rock is found to be competent), an impermeable liner overlain by standard riprap, or grouted riprap.

## 16.4 MINING PHASES

The mine phase designs were guided by a series of Lerchs-Grossman (LG) pit optimization runs.

The slope angles are based on the study provided by AMEC as discussed in Section 16.2. The recommended inter-ramp slope angle was 44°. For the 15 m benches, this was designed with a face angle of 70° and a 10 m catch bench. Overall slope angles are about 40° on most walls and are flattened to 32° above the Kingking River diversion.

The phase designs include haul roads and adequate working room for large mining equipment. The in-pit roads are 35 m wide at a maximum grade of 10%. The width will accommodate trucks of the 230 to 250 tonne class such as Caterpillar 793 or Komatsu 860E trucks.

Mining phase 1 (Figure 16-3, west) is a relatively high-grade, low waste ratio area defined by the central portion of the LG run at $1.50/lb copper. Mining phase 2 (Figure 16-3, east and highlighted) is a small, but high-grade, predominantly oxide zone, southeast of phase 1. It is also based on the LG run at $1.50/lb copper.

Mining phase 3 (Figure 16-4, east) pushes the pit to the east and deepens it in the phase 1 and phase 2 area. It is based on the LG run at about $2.00/lb in the east. Mining phase 4 (Figure 16-4, west and highlighted) pushes the pit to the west. It is based on the LG run at about $2.00/lb copper in the west.

Mining phase 5 (Figure 16-5, west) pushes the pit to final pit limits on the west side. The LG run at $3.25/lb copper was targeted for final pit design. The Kingking River diversion was constructed during phase 5. Mining phase 6 (Figure 16-5, east and highlighted) is a push to the east side. It is based on the LG run at $2.50/lb copper. The upper east side of the phase is at final pit limits.

Mining phase 7 (Figure 16-6) pushes the pit to final pit limits in the east. This targeted the $3.25/lb LG shell.

The LG shell target for final pit design was the $3.25/lb copper shell, which is a lower price than the official design price of $3.75/lb. The results of the optimization study indicated significant amounts of incremental material movement, about 250 million tonnes, between the two cases with no significant increase to free cashflow.

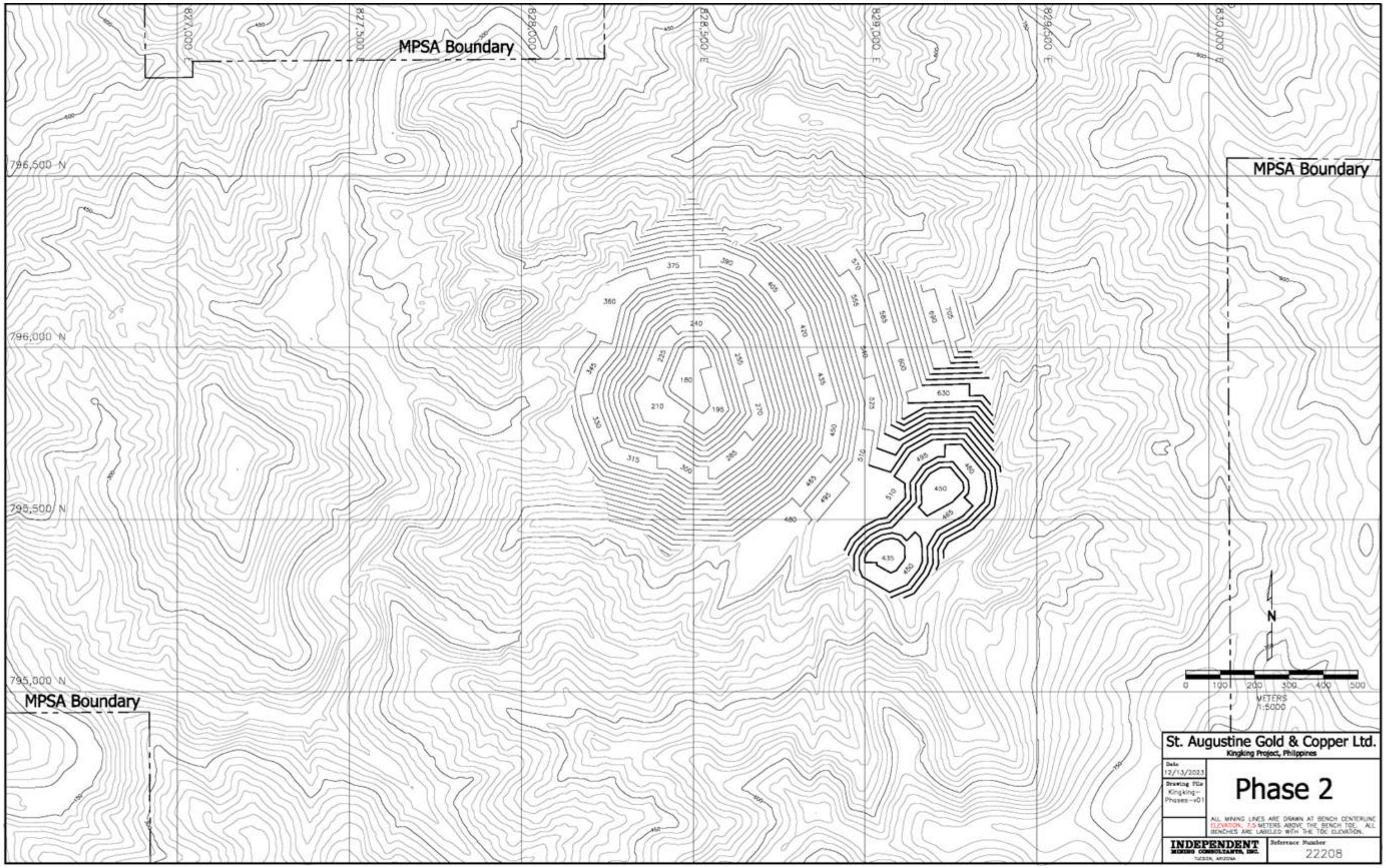



**Figure 16-3: Mining Phases 1 and 2 (IMC, 2023)**

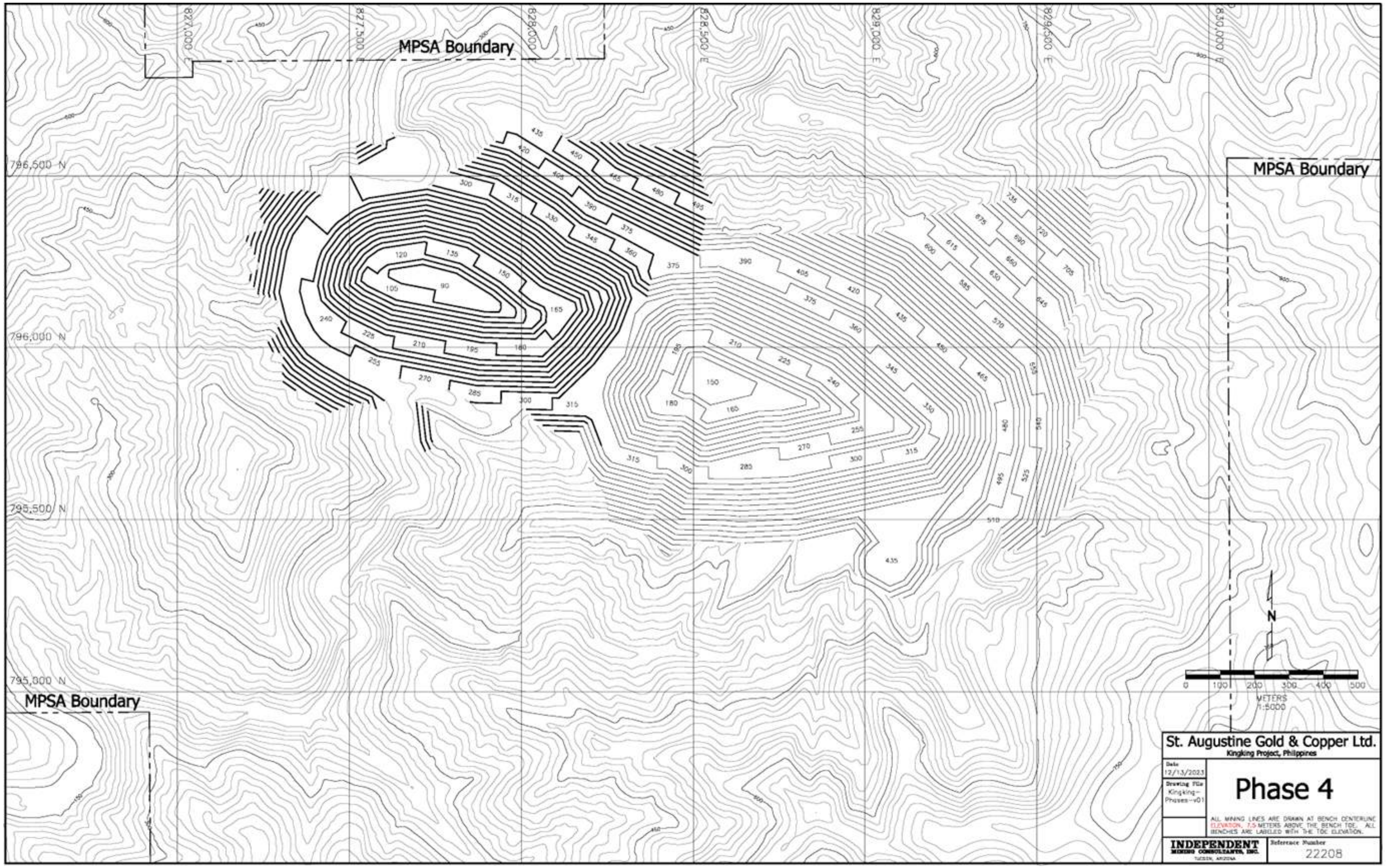



**Figure 16-4: Mining Phases 3 and 4 (IMC, 2023)**

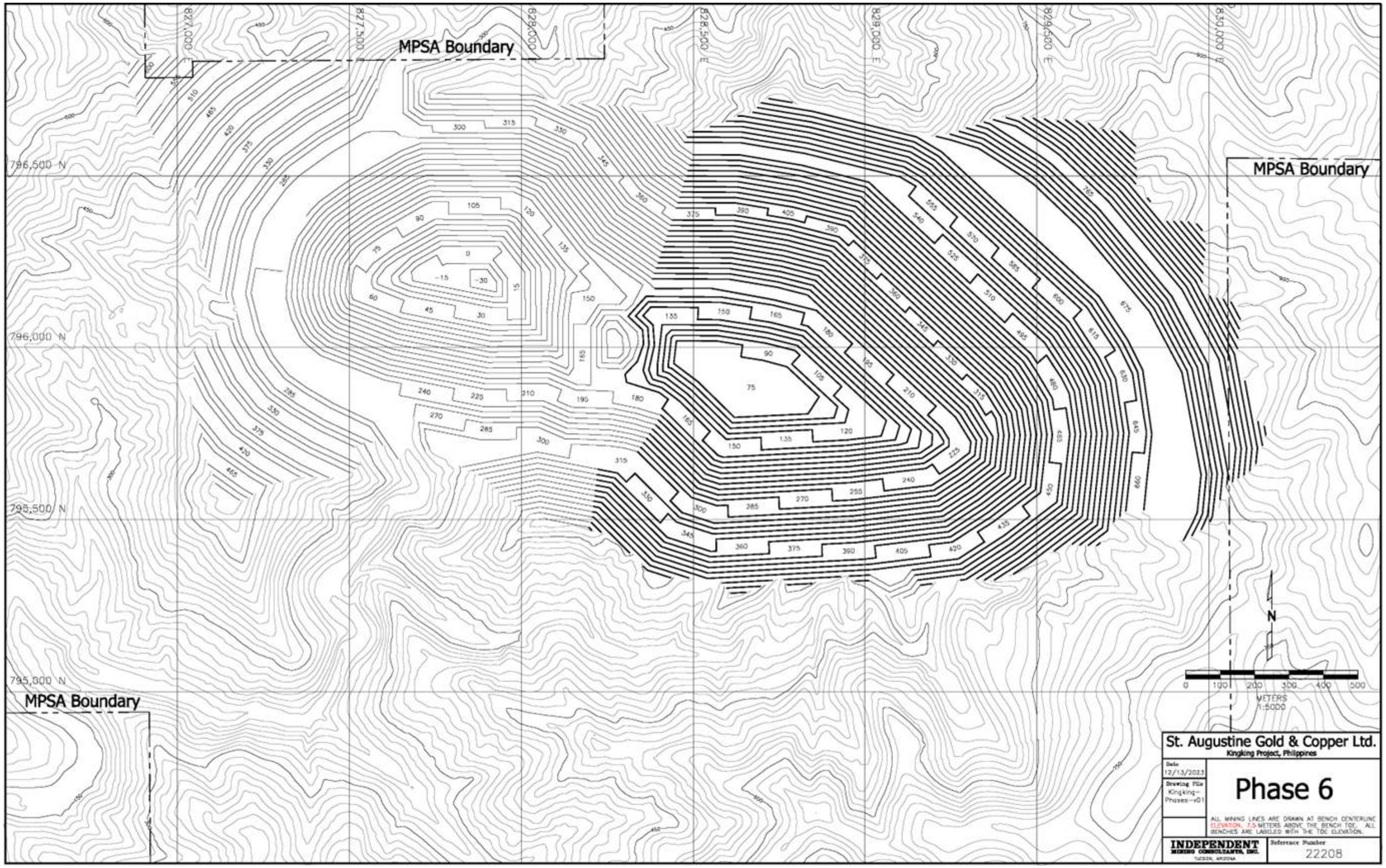



**Figure 16-5: Mining Phases 5 and 6 (IMC, 2023)**

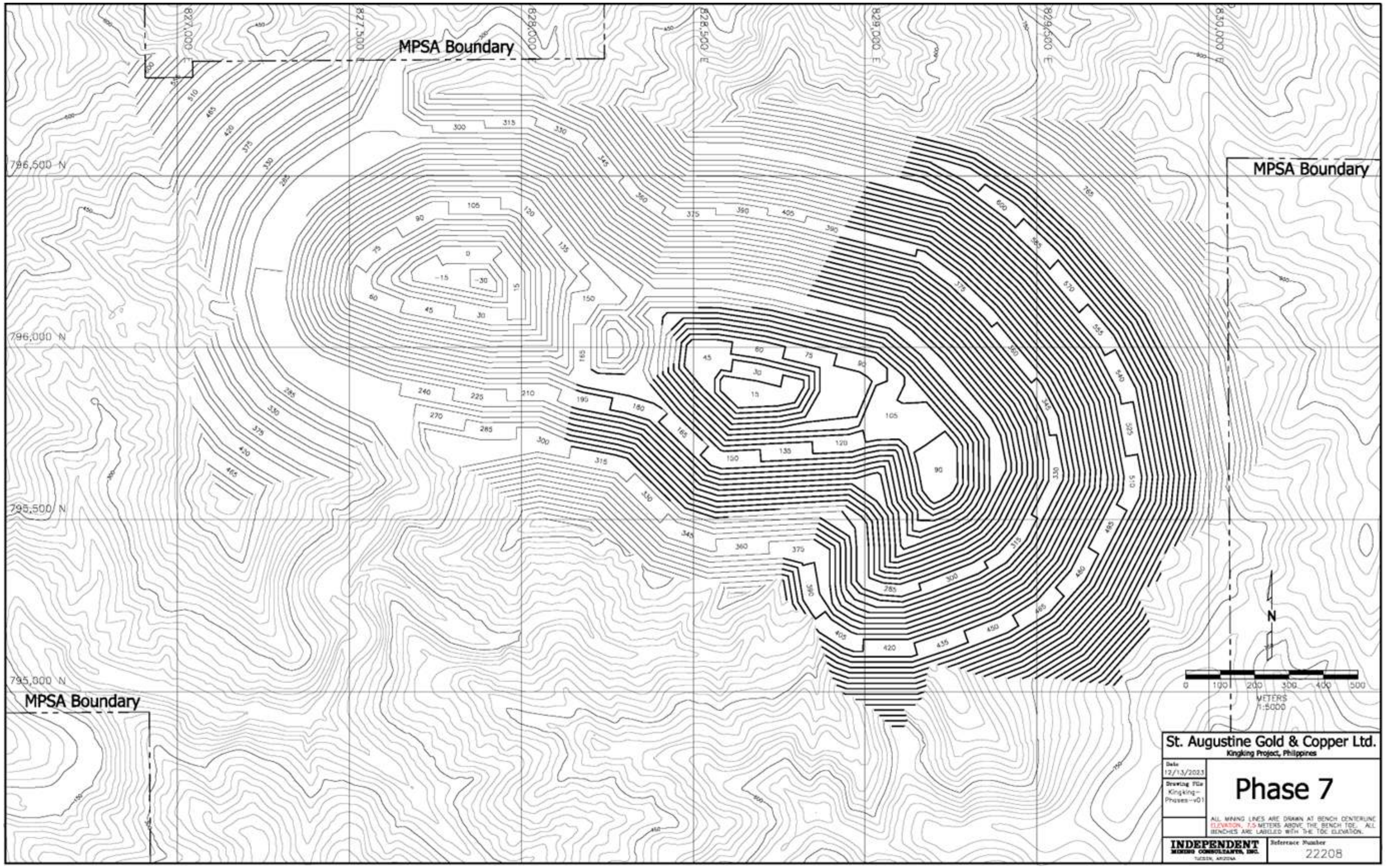



Figure 16-6: Mining Phase 7 – Final Pit (IMC, 2023)

## 16.5 PLANT THROUGHPUT

Figure 16-7 shows an equation that expresses annual mill throughput as a percentage of the plant feed that is mixed oxide/sulfide versus sulfide. The mixed oxide/sulfide material is softer and has a higher throughput rate. This was developed based on available metallurgical testing results.

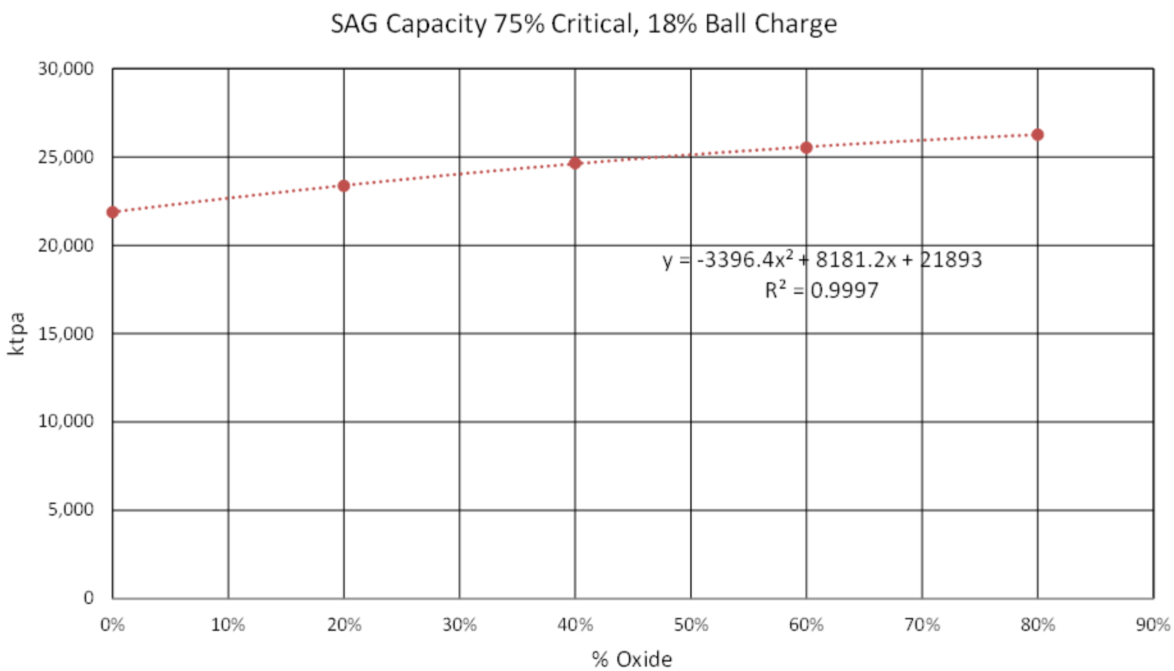



**Figure 16-7: Annual Plant Throughput versus Percent of Mixed Oxide/Sulfide versus Sulfide Ore (Ibrado, 2024)**

## 16.6 MINE PRODUCTION SCHEDULE

Mine and plant production schedules were developed for the Kingking Project based on the seven mining phases. The economic parameters, metal recoveries, and calculations of NSR values used to develop the schedules are described in Sections 15.2, 15.3, and 15.4 respectively.

As discussed in Sections 15.3 and 15.4, 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$/t and is revenue less smelting, refining, freight, and royalties for mill ore. Also, since the NSR cut-offs vary by ore type, it is convenient to add a Net of Process (NOP) parameter for production scheduling. NOP is defined as the NSR minus processing and G&A costs.

Table 16-2 show the mine production schedule. The upper section of the table shows the direct feed ore. This is ore that will be processed the same year it is mined. This amounts to 673.6 million tonnes at 0.279% total copper and 0.410 g/t gold. The open pit life is just over 30 years of commercial production after a 2-year preproduction period. The plant feed tonnes vary by year based on the percent of mixed oxide/sulfide material in the feed, as discussed in the previous section. Percent mixed oxide/sulfide and theoretical throughput are also shown as a check. 4.80 million tonnes of direct feed material are mined during preproduction and stockpiled to be part of the Year 1 plant feed. Total Year 1 feed is about 75% of the capacity indicated by the throughput equation.

For the mine production schedule, the NOP cut-off varies by year to balance mine and plant capacities. It ranges from a high of $21/t for Year 3 to a low of $0.01/t (internal cut-off) for Years 25, 30, and 31.

The second section of the tables shows low grade produced by year. This is material with an NOP cut-off between $1.10/t, the mine rehandle cost, and the operating cut-off for the year. This amounts to 175.3 million tonnes at 0.179% total copper and 0.157 g/t gold. This is only the sulfide portion of the potential low grade. The potential mixed oxide/sulfide mill material routed to low-grade was re-routed to the leach pad to generate earlier revenue. This is 26.4 million tonnes.

The third section of the tables shows the schedule of heap leach ore. This material is at an NSR cut-off of $3.26/t, internal NSR cut-off. This amounts to 110.5 million tonnes at 0.228% total copper, 0.140% soluble copper, and 0.144 g/t gold (though gold will not be recovered for the leach ore). This amount includes the low-grade mixed oxide/sulfide mill ore discussed above. Over half of this material is mined during Years -2 through 2. The current leach pad infrastructure is designed to accommodate a maximum of 14.6 million tonnes per year, so some of the Year -1 and 1 material will be stockpiled and processed during Years 2 through 4.

The bottom of the tables summarizes tonnages. Total material from the pit is 1.80 billion tonnes. Preproduction is 45.1 million tonnes staged over two years. This is the approximate amount required to prepare mining phase 1 for commercial production. Year 1 total material is scheduled at 50.4 million tonnes, after which the peak material movement of about 65 million tonnes per year is maintained through Year 18. Total waste is 839.0 million tonnes so the waste to ore ratio is about 0.87 if mill and leach ore are both counted as ore.

Table 16-3 show the proposed plant production schedule for concentrates. Total mill ore is 848.9 million tonnes at 0.259% copper and 0.358 g/t gold. The average NSR value of this ore is $27.57/t. The un-highlighted values on the tables were extracted from the resource block model. The values in blue are calculated. Copper and gold recovery to concentrate are estimated at 62.1% and 52.3% respectively. In addition, gold recovered in the gravity circuit amounts to about 20.0%. The copper recovery does not include the copper recovered in the agitation leach process. Copper recovered to concentrate amounts to 3.00 billion pounds and gold recovered in concentrate and the gravity circuit amounts to 7.06 million ounces. Based on this schedule, the commercial life of the Kingking Project is 38 years, after a two-year preproduction period. The low-grade material is processed during Years 30 through 38.

Table 16-4 shows a proposed schedule for the agitation tails leach process. The tonnes shown are based on all the mill feed going to the agitation leach process. This material amounts to 848.9 million tonnes at 0.259% copper and 0.055% soluble copper. The NSR values shown on the tables represent NSR due to the agitation process only. The NOP values are NSR - $4.192, the incremental processing cost. The un-highlighted values on the table were extracted from the resource block model. The values in blue are calculated. The table shows the tonnes of tails per Ktonne of original plant feed, and the total and soluble copper grades of the tails and the recovered copper grade from the tails. For all material, the average recovery of copper in the tails is 60.3% which amounts to about 22.8% of the copper in the original plant feed.

The tables also show the material by mixed oxide/sulfide and sulfide. The mixed oxide/sulfide amounts to 97.4 million tonnes at 0.432% total copper and 0.251% soluble copper and an NSR value of $18.55/t. Most of the value of the agitated leach process is in the oxides. The sulfides amount to 751.5 million tonnes at 0.236% total copper and 0.030% soluble copper and an NSR value of $2.58/t. If only the sulfides are processed, they do not result in a profit for any full year after the first two years of operation with the parameters used for this analysis. Under various copper price and unit cost assumptions this schedule allows the determination of the years the agitation leach process should be operated.

Table 16-5 shows a proposed leach pad stacking schedule. The upper section of the tables show the schedule. This amounts to 110.5 million tonnes at 0.228% total copper and 0.140% soluble copper. The schedule is based on the ability to crush and stack 14.6 million tonnes per year. Year -2 leach material stacked on the pad is 3.60 million tonnes based on ramping up production during the second half of the year. Recoverable copper is estimated at 438.1 million pounds at a 78.7% recovery.

The second section of the tables shows heap leach ore as produced from the mine. As previously discussed, over half of the material is produced during Years -2 through 2. The third and fourth sections show up to 14.6 million tonnes of mined ore as direct crusher feed and the excess going to a stockpile. Both components are shown at average grades for the year. The bottom tables show the stockpile balance at the end of each year and the stockpile reclaim schedule. The stockpile reclaim is on a last-in-first-out basis (LIFO). The peak size of the stockpile is 15.0 million tonnes at the end of Year 1.

Only measured and indicated mineral resource is included in the mine production schedule and converted to mineral reserves.

The mine production schedule includes allowances for mining dilution and ore loss. The QP for this section believes that reasonable amounts of dilution and loss were incorporated into the block model used for this Technical Report. Compositing assays into composites and estimating blocks with multiple composites introduces some smoothing of model grades that are analogous to dilution and mineralized material loss effects.

## Table 16-2: Mine Production Schedule

| | Units | -2 | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Direct Mill Feed: | | | | | | | | | | | | | | | | | | |
| NOP Cut-off | | 20.49 | 20.49 | 20.49 | 16.91 | 20.99 | 20.32 | 16.84 | 14.45 | 12.64 | 11.70 | 10.09 | 10.96 | 8.44 | 6.35 | 6.24 | 3.87 | 3.29 |
| Ktonnes | (kt) | 216 | 4,584 | 14,951 | 26,250 | 25,750 | 24,000 | 22,375 | 22,450 | 22,330 | 22,150 | 22,550 | 24,250 | 24,825 | 23,275 | 22,210 | 22,075 | 21,900 |
| NOP | ($/t) | 26.66 | 36.10 | 49.28 | 50.53 | 44.27 | 39.11 | 34.41 | 29.35 | 27.07 | 24.81 | 22.58 | 23.07 | 22.94 | 24.17 | 22.37 | 22.52 | 14.97 |
| NSR | ($/t) | 37.86 | 46.98 | 60.22 | 61.26 | 54.52 | 48.67 | 43.47 | 38.52 | 35.80 | 32.69 | 30.29 | 32.06 | 31.65 | 32.34 | 30.40 | 30.59 | 22.39 |
| Total Copper | (%) | 0.520 | 0.588 | 0.702 | 0.597 | 0.394 | 0.314 | 0.332 | 0.384 | 0.407 | 0.349 | 0.315 | 0.268 | 0.201 | 0.223 | 0.221 | 0.208 | 0.153 |
| Soluble Copper | (%) | 0.180 | 0.282 | 0.409 | 0.312 | 0.176 | 0.074 | 0.060 | 0.058 | 0.048 | 0.037 | 0.041 | 0.074 | 0.050 | 0.041 | 0.037 | 0.039 | 0.025 |
| Gold | (g/t) | 0.111 | 0.249 | 0.407 | 0.641 | 0.833 | 0.788 | 0.608 | 0.422 | 0.322 | 0.358 | 0.357 | 0.475 | 0.548 | 0.518 | 0.481 | 0.515 | 0.394 |
| Recovered Copper Grade | (%) | 0.476 | 0.542 | 0.653 | 0.555 | 0.360 | 0.279 | 0.293 | 0.334 | 0.349 | 0.291 | 0.262 | 0.230 | 0.173 | 0.191 | 0.188 | 0.179 | 0.127 |
| Recovered Gold Grade | (g/t) | 0.068 | 0.149 | 0.242 | 0.403 | 0.551 | 0.566 | 0.457 | 0.313 | 0.245 | 0.273 | 0.268 | 0.325 | 0.392 | 0.388 | 0.358 | 0.378 | 0.290 |
| Percent Oxide Resource | (%) | 100.00% | 88.59% | 84.54% | 79.30% | 64.73% | 29.54% | 6.04% | 7.03% | 5.38% | 3.26% | 8.42% | 33.65% | 43.66% | 18.22% | 3.91% | 2.32% | 0.04% |
| Plant Throughput Equation | (kt) | | | 26,382 | 26,245 | 25,766 | 24,013 | 22,375 | 22,452 | 22,324 | 22,156 | 22,558 | 24,261 | 24,818 | 23,271 | 22,208 | 22,081 | 21,896 |
| Difference | (kt) | | | | 5 | -16 | -13 | 0 | -2 | 6 | -6 | -8 | -11 | 7 | 4 | 2 | -6 | 4 |
| Low Grade Stockpile (Sulfide): | | | | | | | | | | | | | | | | | | |
| NOP Cut-off | ($/t) | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 |
| Ktonnes | (kt) | | 1,117 | 5,277 | 8,043 | 12,304 | 9,920 | 10,799 | 19,494 | 20,991 | 16,603 | 4,441 | 6,121 | 7,456 | 6,048 | 4,835 | 735 | 2,457 |
| NOP | ($/t) | | 10.85 | 9.57 | 9.90 | 13.23 | 12.84 | 8.59 | 8.01 | 7.34 | 6.48 | 6.31 | 6.65 | 5.00 | 3.81 | 3.89 | 2.75 | 2.17 |
| NSR | ($/t) | | 18.27 | 16.92 | 17.22 | 20.89 | 20.68 | 16.00 | 15.34 | 14.65 | 13.72 | 13.55 | 13.89 | 12.24 | 11.06 | 11.30 | 10.07 | 9.41 |
| Total Copper | (%) | | 0.282 | 0.247 | 0.247 | 0.265 | 0.235 | 0.167 | 0.211 | 0.202 | 0.165 | 0.158 | 0.154 | 0.118 | 0.115 | 0.117 | 0.107 | 0.087 |
| Soluble Copper | (%) | | 0.027 | 0.026 | 0.023 | 0.029 | 0.032 | 0.020 | 0.024 | 0.019 | 0.013 | 0.013 | 0.019 | 0.018 | 0.020 | 0.023 | 0.018 | 0.015 |
| Gold | (g/t) | | 0.118 | 0.132 | 0.138 | 0.196 | 0.229 | 0.210 | 0.144 | 0.134 | 0.156 | 0.160 | 0.183 | 0.192 | 0.169 | 0.173 | 0.150 | 0.161 |
| Recovered Copper Grade | (%) | | 0.226 | 0.199 | 0.195 | 0.215 | 0.194 | 0.133 | 0.168 | 0.158 | 0.124 | 0.120 | 0.121 | 0.093 | 0.091 | 0.093 | 0.084 | 0.068 |
| Recovered Gold Grade | (g/t) | | 0.087 | 0.096 | 0.103 | 0.147 | 0.169 | 0.156 | 0.104 | 0.100 | 0.119 | 0.122 | 0.133 | 0.137 | 0.119 | 0.121 | 0.106 | 0.112 |
| Heap Leach: | | | | | | | | | | | | | | | | | | |
| NSR Cut-off | ($/t) | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 |
| Ktonnes | (kt) | 7,717 | 21,646 | 18,476 | 13,395 | 4,026 | 4,040 | 7,360 | 5,289 | 633 | 4,355 | 1,642 | 8,092 | 9,951 | 1,213 | 6 | | 102 |
| NSR | ($/t) | 10.82 | 14.17 | 19.05 | 19.06 | 13.30 | 8.22 | 15.64 | 13.04 | 7.21 | 8.41 | 7.57 | 7.25 | 7.02 | 6.80 | 10.29 | | 7.33 |
| Total Copper | (%) | 0.199 | 0.240 | 0.318 | 0.310 | 0.217 | 0.143 | 0.269 | 0.242 | 0.211 | 0.139 | 0.138 | 0.118 | 0.118 | 0.156 | 0.235 | | 0.141 |
| Soluble Copper | (%) | 0.104 | 0.145 | 0.211 | 0.213 | 0.146 | 0.074 | 0.162 | 0.130 | 0.061 | 0.077 | 0.064 | 0.066 | 0.061 | 0.056 | 0.096 | | 0.061 |
| Gold | (g/t) | 0.079 | 0.114 | 0.135 | 0.153 | 0.337 | 0.207 | 0.091 | 0.108 | 0.131 | 0.155 | 0.172 | 0.217 | 0.166 | 0.110 | 0.093 | | 0.076 |
| Recovered Copper Grade | (%) | 0.144 | 0.189 | 0.254 | 0.254 | 0.178 | 0.110 | 0.209 | 0.175 | 0.096 | 0.112 | 0.101 | 0.097 | 0.094 | 0.091 | 0.138 | | 0.098 |
| Total Material, Waste, W:O | | | | | | | | | | | | | | | | | | |
| Total Material | (kt) | 11,238 | 33,833 | 50,383 | 65,862 | 65,210 | 64,923 | 64,788 | 64,681 | 64,757 | 64,909 | 65,722 | 65,431 | 65,093 | 65,151 | 64,862 | 65,420 | 65,678 |
| Waste (Low Grade as Resource) | (kt) | 3,305 | 6,486 | 11,679 | 18,174 | 23,130 | 26,963 | 24,254 | 17,448 | 20,803 | 21,801 | 37,089 | 26,968 | 22,861 | 34,615 | 37,811 | 42,610 | 41,219 |
| Waste:Ore Ratio (LG as Resource) | (none) | 0.42 | 0.24 | 0.30 | 0.38 | 0.55 | 0.71 | 0.60 | 0.37 | 0.47 | 0.51 | 1.30 | 0.70 | 0.54 | 1.13 | 1.40 | 1.87 | 1.69 |

| | Units | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 | 25 | 26 | 27 | 28 | 29 | 30 | 31 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Direct Mill Feed: | | | | | | | | | | | | | | | | | | |
| NOP Cut-off | | 3.73 | 0.92 | 1.78 | 4.31 | 5.99 | 5.49 | 5.93 | 5.00 | 4.52 | 0.01 | 3.79 | 4.15 | 7.67 | 4.70 | 0.01 | 0.01 | |
| Ktonnes | (kt) | 21,900 | 21,900 | 22,360 | 22,692 | 21,900 | 21,900 | 21,900 | 21,900 | 21,950 | 21,992 | 21,900 | 21,900 | 21,900 | 21,900 | 17,011 | 2,438 | 673,584 |
| NOP | ($/t) | 12.22 | 13.07 | 14.80 | 11.21 | 16.02 | 17.93 | 18.16 | 17.50 | 17.19 | 14.84 | 10.32 | 11.73 | 16.48 | 17.79 | 13.66 | 16.90 | 22.70 |
| NSR | ($/t) | 19.77 | 20.96 | 22.84 | 19.00 | 23.79 | 25.56 | 25.65 | 24.90 | 24.80 | 22.56 | 18.04 | 19.48 | 24.12 | 25.18 | 21.13 | 25.09 | 31.01 |
| Total Copper | (%) | 0.150 | 0.163 | 0.174 | 0.251 | 0.264 | 0.247 | 0.236 | 0.227 | 0.222 | 0.224 | 0.242 | 0.244 | 0.255 | 0.213 | 0.166 | 0.257 | 0.279 |
| Soluble Copper | (%) | 0.025 | 0.033 | 0.036 | 0.032 | 0.031 | 0.027 | 0.027 | 0.027 | 0.029 | 0.032 | 0.032 | 0.032 | 0.032 | 0.025 | 0.022 | 0.040 | 0.064 |
| Gold | (g/t) | 0.332 | 0.345 | 0.372 | 0.168 | 0.260 | 0.329 | 0.352 | 0.350 | 0.349 | 0.295 | 0.159 | 0.192 | 0.290 | 0.371 | 0.333 | 0.307 | 0.410 |
| Recovered Copper Grade | (%) | 0.123 | 0.137 | 0.146 | 0.205 | 0.218 | 0.202 | 0.194 | 0.187 | 0.185 | 0.186 | 0.198 | 0.200 | 0.211 | 0.174 | 0.135 | 0.219 | 0.240 |
| Recovered Gold Grade | (g/t) | 0.242 | 0.246 | 0.268 | 0.124 | 0.195 | 0.250 | 0.266 | 0.262 | 0.261 | 0.217 | 0.117 | 0.140 | 0.217 | 0.282 | 0.250 | 0.221 | 0.296 |
| Percent Oxide Resource | (%) | 0.00% | 0.00% | 5.91% | 10.25% | 0.03% | 0.00% | 0.00% | 0.00% | 0.84% | 3.05% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 0.00% | 14.46% |
| Plant Throughput Equation | (kt) | 21,893 | 21,893 | 22,364 | 22,696 | 21,896 | 21,893 | 21,893 | 21,893 | 21,961 | 22,139 | 21,893 | 21,893 | 21,893 | 21,893 | 21,893 | 21,893 | |
| Difference | (kt) | 7 | 7 | -4 | -4 | 4 | 7 | 7 | 7 | -11 | -147 | 7 | 7 | 7 | 7 | | | |
| Low Grade Stockpile (Sulfide): | | | | | | | | | | | | | | | | | | |
| NOP Cut-off | ($/t) | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | 1.10 | | | |
| Ktonnes | (kt) | 5,207 | | 473 | 3,625 | 3,460 | 3,327 | 4,551 | 3,570 | 3,336 | | 4,010 | 1,818 | 3,030 | 2,248 | | | 175,296 |
| NOP | ($/t) | 2.42 | | 1.51 | 3.17 | 4.26 | 3.72 | 3.82 | 3.44 | 3.03 | | 2.63 | 2.86 | 5.77 | 3.35 | | | 7.02 |
| NSR | ($/t) | 9.67 | | 8.75 | 10.43 | 11.50 | 10.96 | 11.06 | 10.68 | 10.27 | | 9.87 | 10.13 | 13.10 | 10.59 | | | 14.37 |
| Total Copper | (%) | 0.101 | | 0.097 | 0.174 | 0.160 | 0.136 | 0.136 | 0.120 | 0.100 | | 0.159 | 0.163 | 0.168 | 0.096 | | | 0.179 |
| Soluble Copper | (%) | 0.018 | | 0.011 | 0.018 | 0.017 | 0.013 | 0.011 | 0.011 | 0.011 | | 0.020 | 0.021 | 0.022 | 0.010 | | | 0.020 |
| Gold | (g/t) | 0.150 | | 0.122 | 0.071 | 0.113 | 0.128 | 0.129 | 0.140 | 0.159 | | 0.076 | 0.078 | 0.144 | 0.168 | | | 0.157 |
| Recovered Copper Grade | (%) | 0.079 | | 0.071 | 0.133 | 0.124 | 0.101 | 0.100 | 0.089 | 0.074 | | 0.124 | 0.128 | 0.133 | 0.070 | | | 0.141 |
| Recovered Gold Grade | (g/t) | 0.106 | | 0.089 | 0.051 | 0.083 | 0.096 | 0.097 | 0.104 | 0.116 | | 0.054 | 0.055 | 0.105 | 0.125 | | | 0.115 |
| Heap Leach: | | | | | | | | | | | | | | | | | | |
| NSR Cut-off | ($/t) | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | 3.26 | | | | | | | |
| Ktonnes | (kt) | | | 157 | 739 | | | | | 762 | 921 | | | | | | | 110,522 |
| NSR | ($/t) | | | 19.61 | 12.29 | | | | | 8.29 | 11.01 | | | | | | | 13.47 |
| Total Copper | (%) | | | 0.306 | 0.221 | | | | | 0.133 | 0.174 | | | | | | | 0.228 |
| Soluble Copper | (%) | | | 0.216 | 0.124 | | | | | 0.076 | 0.105 | | | | | | | 0.140 |
| Gold | (g/t) | | | 0.122 | 0.120 | | | | | 0.082 | 0.108 | | | | | | | 0.144 |
| Recovered Copper Grade | (%) | | | 0.262 | 0.164 | | | | | 0.111 | 0.147 | | | | | | | 0.180 |
| Total Material, Waste, W:O | | | | | | | | | | | | | | | | | | |
| Total Material | (kt) | 65,411 | 65,160 | 65,097 | 59,858 | 59,877 | 59,915 | 59,953 | 59,972 | 59,972 | 60,010 | 50,072 | 30,043 | 30,043 | 30,043 | 30,043 | 5,007 | 1,798,417 |
| Waste (Low Grade as Resource) | (kt) | 38,304 | 43,260 | 42,107 | 32,802 | 34,517 | 34,688 | 33,502 | 34,502 | 33,924 | 37,097 | 24,162 | 6,325 | 5,113 | 5,895 | 13,032 | 2,569 | 839,015 |
| Waste:Ore Ratio (LG as Resource) | (none) | 1.41 | 1.98 | 1.83 | 1.21 | 1.36 | 1.38 | 1.27 | 1.35 | 1.30 | 1.62 | 0.93 | 0.27 | 0.21 | 0.24 | 0.77 | 1.05 | 0.87 |

## Table 16-3: Proposed Plant Production Schedule – Concentrate

| | Units | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Plant Schedule - All Mill Material** | | | | | | | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 19,750 | 26,250 | 25,750 | 24,000 | 22,375 | 22,450 | 22,330 | 22,150 | 22,550 | 24,250 | 24,825 | 23,275 | 22,210 | 22,075 | 21,900 | 21,900 | 21,900 | 22,360 | 22,692 | 21,900 |
| NOP | ($/t) | 45.97 | 50.53 | 44.27 | 39.11 | 34.41 | 29.35 | 27.07 | 24.81 | 22.58 | 23.07 | 22.94 | 24.17 | 22.37 | 22.52 | 14.97 | 12.22 | 13.07 | 14.80 | 11.21 | 16.02 |
| NSR | ($/t) | 56.90 | 61.26 | 54.52 | 48.67 | 43.47 | 38.52 | 35.80 | 32.69 | 30.29 | 32.06 | 31.65 | 32.34 | 30.40 | 30.59 | 22.39 | 19.77 | 20.96 | 22.84 | 19.00 | 23.79 |
| Total Copper | (%) | 0.673 | 0.597 | 0.394 | 0.314 | 0.332 | 0.384 | 0.407 | 0.349 | 0.315 | 0.268 | 0.201 | 0.223 | 0.221 | 0.208 | 0.153 | 0.150 | 0.163 | 0.174 | 0.251 | 0.264 |
| Soluble Copper | (%) | 0.377 | 0.312 | 0.176 | 0.074 | 0.060 | 0.058 | 0.048 | 0.037 | 0.041 | 0.074 | 0.050 | 0.041 | 0.037 | 0.039 | 0.025 | 0.025 | 0.033 | 0.036 | 0.032 | 0.031 |
| Gold | (g/t) | 0.367 | 0.641 | 0.833 | 0.788 | 0.608 | 0.422 | 0.322 | 0.358 | 0.357 | 0.475 | 0.548 | 0.518 | 0.481 | 0.515 | 0.394 | 0.332 | 0.345 | 0.372 | 0.168 | 0.260 |
| Recovered Copper Grade | (%) | 0.625 | 0.555 | 0.360 | 0.279 | 0.293 | 0.334 | 0.349 | 0.291 | 0.262 | 0.230 | 0.173 | 0.191 | 0.188 | 0.179 | 0.127 | 0.123 | 0.137 | 0.146 | 0.205 | 0.218 |
| Recovered Copper Grade-Conc | (%) | 0.256 | 0.249 | 0.182 | 0.201 | 0.230 | 0.270 | 0.294 | 0.247 | 0.216 | 0.152 | 0.120 | 0.147 | 0.148 | 0.136 | 0.099 | 0.094 | 0.101 | 0.107 | 0.167 | 0.182 |
| Recovered Copper Grade-Agit | (%) | 0.369 | 0.304 | 0.178 | 0.077 | 0.064 | 0.064 | 0.055 | 0.044 | 0.047 | 0.078 | 0.054 | 0.043 | 0.040 | 0.043 | 0.028 | 0.028 | 0.036 | 0.039 | 0.038 | 0.036 |
| Recovered Gold Grade | g/t | 0.218 | 0.403 | 0.551 | 0.566 | 0.457 | 0.313 | 0.245 | 0.273 | 0.268 | 0.325 | 0.392 | 0.388 | 0.358 | 0.378 | 0.290 | 0.242 | 0.246 | 0.268 | 0.124 | 0.195 |
| Gold Grade to Flotation | g/t | 0.278 | 0.486 | 0.642 | 0.624 | 0.491 | 0.341 | 0.261 | 0.289 | 0.287 | 0.371 | 0.439 | 0.419 | 0.389 | 0.418 | 0.319 | 0.269 | 0.279 | 0.302 | 0.136 | 0.210 |
| Recovered Gold Grade-Gravity | g/t | 0.088 | 0.155 | 0.190 | 0.164 | 0.117 | 0.080 | 0.062 | 0.068 | 0.070 | 0.104 | 0.109 | 0.099 | 0.092 | 0.098 | 0.075 | 0.063 | 0.065 | 0.071 | 0.032 | 0.049 |
| Recovered Gold Grade-Conc | g/t | 0.130 | 0.249 | 0.361 | 0.402 | 0.341 | 0.233 | 0.184 | 0.204 | 0.198 | 0.221 | 0.283 | 0.289 | 0.267 | 0.279 | 0.215 | 0.179 | 0.181 | 0.197 | 0.092 | 0.146 |
| Tonne Conc per Kt Feed | t/kt | 10.257 | 9.994 | 7.289 | 8.049 | 9.184 | 10.773 | 11.766 | 9.889 | 8.610 | 6.091 | 4.792 | 5.903 | 5.912 | 5.458 | 3.952 | 3.777 | 4.040 | 4.274 | 6.672 | 7.269 |
| Rec Cu Conc / Conc Grade | (%/%) | 0.01026 | 0.00999 | 0.00729 | 0.00805 | 0.00918 | 0.01077 | 0.01177 | 0.00989 | 0.00861 | 0.00609 | 0.00479 | 0.00591 | 0.00591 | 0.00546 | 0.00395 | 0.00378 | 0.00404 | 0.00428 | 0.00667 | 0.00727 |
| Tonnes Cu in Conc/Kt Feed | (t/kt) | 2.564 | 2.498 | 1.823 | 2.012 | 2.296 | 2.694 | 2.942 | 2.472 | 2.153 | 1.522 | 1.198 | 1.476 | 1.478 | 1.365 | 0.988 | 0.944 | 1.010 | 1.069 | 1.668 | 1.818 |
| Tonnes Sol Cu in Conc/Kt Feed | (t/kt) | 0.108 | 0.089 | 0.051 | 0.021 | 0.017 | 0.017 | 0.014 | 0.011 | 0.012 | 0.021 | 0.015 | 0.011 | 0.011 | 0.011 | 0.007 | 0.007 | 0.010 | 0.010 | 0.009 | 0.009 |
| Copper Recovery to Concentrate | (%) | 38.0% | 41.8% | 46.2% | 64.0% | 69.2% | 70.2% | 72.2% | 70.9% | 68.4% | 56.8% | 59.4% | 66.1% | 67.0% | 65.6% | 64.7% | 62.7% | 62.0% | 61.8% | 66.4% | 68.9% |
| Gold Recovery to Concentrate | (%) | 35.5% | 38.8% | 43.3% | 51.1% | 56.0% | 55.3% | 57.0% | 57.2% | 55.5% | 46.6% | 51.6% | 55.8% | 55.5% | 54.2% | 54.6% | 53.9% | 52.5% | 53.0% | 54.7% | 56.2% |
| Gold Recovery to Gravity | (%) | 23.9% | 24.1% | 22.9% | 20.8% | 19.2% | 19.0% | 19.1% | 19.2% | 19.6% | 21.8% | 19.9% | 19.0% | 19.1% | 19.1% | 19.0% | 19.0% | 18.8% | 19.1% | 19.0% | 18.8% |
| Concentrate | (t) | 202,566 | 262,346 | 187,700 | 193,167 | 205,488 | 241,863 | 262,736 | 219,032 | 194,159 | 147,696 | 118,966 | 137,403 | 131,313 | 120,496 | 86,538 | 82,716 | 88,476 | 95,571 | 151,402 | 159,202 |
| Concentrate Grade - Copper | (%) | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.1 | 25.0 | 25.0 | 24.9 | 25.0 | 25.0 | 25.1 | 24.9 | 25.0 | 25.1 | 25.0 | 25.0 |
| Concentrate Grade - Gold | (g/t) | 12.70 | 24.87 | 49.47 | 49.97 | 37.09 | 21.64 | 15.63 | 20.68 | 23.00 | 36.29 | 59.05 | 48.99 | 45.15 | 51.18 | 54.41 | 47.39 | 44.80 | 46.12 | 13.78 | 20.09 |
| Copper to Concentrate | (klbs) | 111,536 | 144,393 | 103,267 | 106,317 | 113,262 | 133,409 | 144,697 | 120,609 | 107,239 | 81,402 | 65,568 | 75,543 | 72,500 | 66,411 | 47,816 | 45,348 | 48,764 | 52,790 | 83,458 | 87,864 |
| Gold to Gravity | (koz) | 55.8 | 130.4 | 157.6 | 126.6 | 83.8 | 57.9 | 44.3 | 48.8 | 50.7 | 80.8 | 87.2 | 73.8 | 65.5 | 69.9 | 52.8 | 44.4 | 45.8 | 51.2 | 23.3 | 34.5 |
| Gold to Concentrate | (koz) | 82.7 | 209.8 | 298.5 | 310.4 | 245.0 | 168.3 | 132.0 | 145.6 | 143.6 | 172.3 | 225.8 | 216.4 | 190.6 | 198.3 | 151.4 | 126.0 | 127.4 | 141.7 | 67.1 | 102.8 |
| Total Gold Recovered | (koz) | 138.5 | 340.2 | 456.1 | 436.9 | 328.9 | 226.2 | 176.3 | 194.4 | 194.3 | 253.1 | 313.0 | 290.2 | 256.1 | 268.1 | 204.2 | 170.4 | 173.2 | 192.9 | 90.4 | 137.3 |

| | Units | 21 | 22 | 23 | 24 | 25 | 26 | 27 | 28 | 29 | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Plant Schedule - All Mill Material** | | | | | | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 21,900 | 21,900 | 21,900 | 21,950 | 22,139 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 19,398 | 848,879 |
| NOP | ($/t) | 17.93 | 18.16 | 17.50 | 17.19 | 14.76 | 10.32 | 11.73 | 16.48 | 17.79 | 11.65 | 4.83 | 3.19 | 5.24 | 6.57 | 7.45 | 8.17 | 12.07 | 10.72 | 19.46 |
| NSR | ($/t) | 25.56 | 25.65 | 24.90 | 24.80 | 22.48 | 18.04 | 19.48 | 24.12 | 25.18 | 19.08 | 12.18 | 10.47 | 12.48 | 13.82 | 14.77 | 15.53 | 19.76 | 18.14 | 27.57 |
| Total Copper | (%) | 0.247 | 0.236 | 0.227 | 0.222 | 0.223 | 0.242 | 0.244 | 0.255 | 0.213 | 0.159 | 0.149 | 0.126 | 0.131 | 0.169 | 0.204 | 0.199 | 0.231 | 0.254 | 0.259 |
| Soluble Copper | (%) | 0.027 | 0.027 | 0.027 | 0.029 | 0.032 | 0.032 | 0.032 | 0.032 | 0.025 | 0.021 | 0.017 | 0.018 | 0.018 | 0.014 | 0.020 | 0.023 | 0.028 | 0.026 | 0.055 |
| Gold | (g/t) | 0.329 | 0.352 | 0.350 | 0.349 | 0.294 | 0.159 | 0.192 | 0.290 | 0.371 | 0.293 | 0.141 | 0.135 | 0.180 | 0.154 | 0.136 | 0.163 | 0.214 | 0.150 | 0.358 |
| Recovered Copper Grade | (%) | 0.202 | 0.194 | 0.187 | 0.185 | 0.186 | 0.198 | 0.200 | 0.211 | 0.174 | 0.128 | 0.115 | 0.098 | 0.103 | 0.128 | 0.160 | 0.158 | 0.188 | 0.203 | 0.220 |
| Recovered Copper Grade-Conc | (%) | 0.170 | 0.162 | 0.155 | 0.151 | 0.149 | 0.161 | 0.163 | 0.174 | 0.145 | 0.104 | 0.094 | 0.077 | 0.081 | 0.109 | 0.134 | 0.131 | 0.155 | 0.172 | 0.161 |

| | Units | 21 | 22 | 23 | 24 | 25 | 26 | 27 | 28 | 29 | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Recovered Copper Grade-Agit | (%) | 0.033 | 0.032 | 0.032 | 0.033 | 0.036 | 0.037 | 0.037 | 0.037 | 0.029 | 0.025 | 0.021 | 0.021 | 0.022 | 0.019 | 0.026 | 0.028 | 0.033 | 0.031 | 0.059 |
| Recovered Gold Grade | g/t | 0.250 | 0.266 | 0.262 | 0.261 | 0.216 | 0.117 | 0.140 | 0.217 | 0.282 | 0.220 | 0.103 | 0.096 | 0.129 | 0.117 | 0.101 | 0.119 | 0.159 | 0.111 | 0.259 |
| Gold Grade to Flotation | g/t | 0.267 | 0.285 | 0.283 | 0.283 | 0.238 | 0.129 | 0.155 | 0.235 | 0.301 | 0.238 | 0.114 | 0.109 | 0.146 | 0.124 | 0.109 | 0.131 | 0.173 | 0.122 | 0.286 |
| Recovered Gold Grade-Gravity | g/t | 0.063 | 0.067 | 0.066 | 0.067 | 0.056 | 0.030 | 0.036 | 0.055 | 0.071 | 0.055 | 0.027 | 0.026 | 0.034 | 0.029 | 0.025 | 0.031 | 0.041 | 0.028 | 0.072 |
| Recovered Gold Grade-Conc | g/t | 0.188 | 0.199 | 0.196 | 0.194 | 0.161 | 0.086 | 0.104 | 0.162 | 0.211 | 0.163 | 0.076 | 0.070 | 0.095 | 0.088 | 0.074 | 0.087 | 0.118 | 0.083 | 0.187 |
| Tonne Conc per Kt Feed | t/kt | 6.786 | 6.477 | 6.215 | 6.050 | 5.958 | 6.450 | 6.528 | 6.980 | 5.810 | 4.158 | 3.765 | 3.067 | 3.236 | 4.370 | 5.367 | 5.234 | 6.207 | 6.873 | 6.424 |
| Rec Cu Conc / Conc Grade | (%/%) | 0.00679 | 0.00648 | 0.00621 | 0.00605 | 0.00596 | 0.00645 | 0.00653 | 0.00698 | 0.00581 | 0.00416 | 0.00377 | 0.00307 | 0.00324 | 0.00437 | 0.00536 | 0.00523 | 0.00621 | 0.00687 | 0.00642 |
| Tonnes Cu in Conc/Kt Feed | (t/kt) | 1.697 | 1.619 | 1.554 | 1.513 | 1.489 | 1.612 | 1.632 | 1.745 | 1.453 | 1.039 | 0.941 | 0.767 | 0.809 | 1.093 | 1.342 | 1.309 | 1.552 | 1.718 | 1.606 |
| Tonnes Sol Cu in Conc/Kt Feed | (t/kt) | 0.008 | 0.008 | 0.008 | 0.008 | 0.009 | 0.009 | 0.009 | 0.009 | 0.007 | 0.006 | 0.005 | 0.005 | 0.005 | 0.004 | 0.006 | 0.007 | 0.008 | 0.007 | 0.016 |
| Copper Recovery to Concentrate | (%) | 68.8% | 68.6% | 68.3% | 67.9% | 66.8% | 66.5% | 66.8% | 68.2% | 68.1% | 65.6% | 63.2% | 60.5% | 61.8% | 64.5% | 65.9% | 66.1% | 67.3% | 67.8% | 62.1% |
| Gold Recovery to Concentrate | (%) | 57.1% | 56.5% | 56.0% | 55.5% | 54.8% | 54.1% | 54.2% | 55.9% | 56.9% | 55.7% | 54.1% | 51.9% | 52.7% | 57.3% | 54.8% | 53.7% | 55.2% | 55.4% | 52.3% |
| Gold Recovery to Gravity | (%) | 19.1% | 19.0% | 18.9% | 19.1% | 19.1% | 18.9% | 18.8% | 19.0% | 19.1% | 18.9% | 19.0% | 19.0% | 18.9% | 19.1% | 18.7% | 18.9% | 19.1% | 18.9% | 20.0% |
| Concentrate | (t) | 148,613 | 141,846 | 136,109 | 132,799 | 131,893 | 141,255 | 142,963 | 152,862 | 127,239 | 91,057 | 82,459 | 67,173 | 70,878 | 95,713 | 117,531 | 114,634 | 135,927 | 133,317 | 5,453,102 |
| Concentrate Grade - Copper | (%) | 25.0 | 25.0 | 25.0 | 24.9 | 25.0 | 25.0 | 25.0 | 24.9 | 25.0 | 25.1 | 25.0 | 25.0 | 25.0 | 25.0 | 25.0 | 25.1 | 25.0 | 25.0 | 25.0 |
| Concentrate Grade - Gold | (g/t) | 27.70 | 30.72 | 31.54 | 32.05 | 27.01 | 13.33 | 15.93 | 23.21 | 36.32 | 39.28 | 20.31 | 22.85 | 29.31 | 20.14 | 13.85 | 16.67 | 19.02 | 12.09 | 29.15 |
| Copper to Concentrate | (klbs) | 82,029 | 78,179 | 74,895 | 72,953 | 72,812 | 77,732 | 78,674 | 84,008 | 70,007 | 50,319 | 45,361 | 36,957 | 39,009 | 52,674 | 64,819 | 63,388 | 74,968 | 73,541 | 3,004,518 |
| Gold to Gravity | (koz) | 44.4 | 47.2 | 46.5 | 47.1 | 40.0 | 21.1 | 25.3 | 38.7 | 50.0 | 39.1 | 18.9 | 18.1 | 24.0 | 20.7 | 17.8 | 21.6 | 28.7 | 17.7 | 1,951.6 |
| Gold to Concentrate | (koz) | 132.4 | 140.1 | 138.0 | 136.9 | 114.6 | 60.6 | 73.2 | 114.1 | 148.6 | 115.0 | 53.8 | 49.3 | 66.8 | 62.0 | 52.3 | 61.4 | 83.1 | 51.8 | 5,109.9 |
| Total Gold Recovered | (koz) | 176.7 | 187.3 | 184.5 | 183.9 | 154.6 | 81.7 | 98.6 | 152.8 | 198.6 | 154.0 | 72.7 | 67.4 | 90.8 | 82.7 | 70.2 | 83.0 | 111.9 | 69.5 | 7,061.6 |

## Table 16-4: Plant Production Schedule – Agitated Tails Leach – All Mill Feed Routed to Agitated Leach

| | Units | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Plant Schedule - Tails Leach Material** | | | | | | | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 19,750 | 26,250 | 25,750 | 24,000 | 22,375 | 22,450 | 22,330 | 22,150 | 22,550 | 24,250 | 24,825 | 23,275 | 22,210 | 22,075 | 21,900 | 21,900 | 21,900 | 22,360 | 22,692 | 21,900 |
| NOP Due To Agitation | ($/t) | 23.43 | 18.62 | 9.10 | 1.61 | 0.58 | 0.60 | -0.05 | -0.95 | -0.67 | 1.63 | -0.17 | -0.92 | -1.14 | -1.01 | -2.10 | -2.07 | -1.47 | -1.28 | -1.38 | -1.46 |
| NSR Due to Agitation | ($/t) | 27.62 | 22.81 | 13.29 | 5.81 | 4.77 | 4.79 | 4.15 | 3.24 | 3.52 | 5.82 | 4.02 | 3.27 | 3.05 | 3.18 | 2.09 | 2.12 | 2.72 | 2.91 | 2.82 | 2.73 |
| Total Copper | (%) | 0.673 | 0.597 | 0.394 | 0.314 | 0.332 | 0.384 | 0.407 | 0.349 | 0.315 | 0.268 | 0.201 | 0.223 | 0.221 | 0.208 | 0.153 | 0.150 | 0.163 | 0.174 | 0.251 | 0.264 |
| Soluble Copper | (%) | 0.377 | 0.312 | 0.176 | 0.074 | 0.060 | 0.058 | 0.048 | 0.037 | 0.041 | 0.074 | 0.050 | 0.041 | 0.037 | 0.039 | 0.025 | 0.025 | 0.033 | 0.036 | 0.032 | 0.031 |
| Gold | (g/t) | 0.367 | 0.641 | 0.833 | 0.788 | 0.608 | 0.422 | 0.322 | 0.358 | 0.357 | 0.475 | 0.548 | 0.518 | 0.481 | 0.515 | 0.394 | 0.332 | 0.345 | 0.372 | 0.168 | 0.260 |
| Recovered Copper Grade | (%) | 0.625 | 0.555 | 0.360 | 0.279 | 0.293 | 0.334 | 0.349 | 0.291 | 0.262 | 0.230 | 0.173 | 0.191 | 0.188 | 0.179 | 0.127 | 0.123 | 0.137 | 0.146 | 0.205 | 0.218 |
| Recovered Copper Grade-Conc | (%) | 0.256 | 0.249 | 0.182 | 0.201 | 0.230 | 0.270 | 0.294 | 0.247 | 0.216 | 0.152 | 0.120 | 0.147 | 0.148 | 0.136 | 0.099 | 0.094 | 0.101 | 0.107 | 0.167 | 0.182 |
| Recovered Copper Grade-Agit | (%) | 0.369 | 0.304 | 0.178 | 0.077 | 0.064 | 0.064 | 0.055 | 0.044 | 0.047 | 0.078 | 0.054 | 0.043 | 0.040 | 0.043 | 0.028 | 0.028 | 0.036 | 0.039 | 0.038 | 0.036 |
| Tonnes Tails/Kt Feed | (t/kt) | 989.74 | 989.68 | 987.70 | 991.95 | 990.82 | 989.23 | 988.23 | 990.11 | 991.39 | 993.25 | 992.64 | 994.10 | 994.09 | 994.54 | 996.05 | 996.22 | 995.96 | 995.73 | 993.33 | 992.73 |
| Copper Grade of Flotation Tail | (%) | 0.422 | 0.351 | 0.212 | 0.114 | 0.103 | 0.115 | 0.115 | 0.103 | 0.100 | 0.116 | 0.083 | 0.076 | 0.073 | 0.072 | 0.054 | 0.055 | 0.063 | 0.067 | 0.084 | 0.083 |
| Soluble Copper Grade of Tail | (%) | 0.366 | 0.301 | 0.171 | 0.072 | 0.059 | 0.057 | 0.048 | 0.036 | 0.040 | 0.072 | 0.050 | 0.039 | 0.036 | 0.038 | 0.024 | 0.025 | 0.033 | 0.035 | 0.032 | 0.031 |
| Recovered Copper Grade From Tail | (%) | 0.373 | 0.308 | 0.179 | 0.078 | 0.065 | 0.065 | 0.056 | 0.044 | 0.048 | 0.078 | 0.054 | 0.044 | 0.041 | 0.043 | 0.028 | 0.028 | 0.037 | 0.039 | 0.038 | 0.037 |
| Tails Processed | (kt) | 19,547 | 25,979 | 25,433 | 23,807 | 22,170 | 22,208 | 22,067 | 21,931 | 22,356 | 24,086 | 24,642 | 23,138 | 22,079 | 21,955 | 21,813 | 21,817 | 21,812 | 22,264 | 22,541 | 21,741 |
| Recovery of Copper in Tails | (%) | 88.5% | 87.7% | 84.2% | 68.8% | 63.1% | 56.5% | 48.8% | 42.7% | 47.7% | 67.3% | 64.7% | 58.4% | 56.7% | 59.7% | 51.9% | 50.9% | 58.7% | 57.8% | 44.8% | 44.6% |
| Recovered Copper | (klbs) | 160,887 | 176,297 | 100,266 | 41,032 | 31,630 | 31,894 | 27,393 | 21,156 | 23,579 | 41,409 | 29,107 | 22,596 | 20,115 | 20,733 | 13,475 | 13,468 | 17,792 | 19,095 | 18,793 | 17,752 |
| Recovery of Copper in Feed | (%) | 54.9% | 51.0% | 44.8% | 24.7% | 19.3% | 16.8% | 13.7% | 12.4% | 15.1% | 28.9% | 26.4% | 19.8% | 18.6% | 20.5% | 18.2% | 18.6% | 22.6% | 22.3% | 15.0% | 13.9% |
| Recovered Copper Grade - Agit Check | (%) | 0.370 | 0.305 | 0.177 | 0.078 | 0.064 | 0.064 | 0.056 | 0.043 | 0.047 | 0.077 | 0.053 | 0.044 | 0.041 | 0.043 | 0.028 | 0.028 | 0.037 | 0.039 | 0.038 | 0.037 |
| **Plant Schedule - Oxide** | **Units** | **1** | **2** | **3** | **4** | **5** | **6** | **7** | **8** | **9** | **10** | **11** | **12** | **13** | **14** | **15** | **16** | **17** | **18** | **19** | **20** |
| Ktonnes | (kt) | 16,916 | 20,815 | 16,669 | 7,090 | 1,352 | 1,579 | 1,202 | 721 | 1,899 | 8,159 | 10,839 | 4,241 | 869 | 513 | 9 | | | 1,321 | 2,326 | 7 |
| NOP Due To Agitation | ($/t) | 27.33 | 23.39 | 14.32 | 6.47 | 16.45 | 9.61 | 5.52 | 2.93 | 6.54 | 6.66 | 2.10 | 2.86 | 4.97 | 1.64 | 1.54 | | | 4.50 | 1.75 | 7.40 |
| NSR Due to Agitation | ($/t) | 31.52 | 27.58 | 18.51 | 10.66 | 20.64 | 13.80 | 9.71 | 7.12 | 10.73 | 10.85 | 6.29 | 7.05 | 9.16 | 5.83 | 5.73 | | | 8.69 | 5.94 | 11.59 |
| Total Copper | (%) | 0.681 | 0.588 | 0.402 | 0.267 | 0.563 | 0.447 | 0.516 | 0.220 | 0.250 | 0.222 | 0.206 | 0.298 | 0.343 | 0.215 | 0.299 | | | 0.309 | 0.278 | 0.566 |
| Soluble Copper | (%) | 0.433 | 0.380 | 0.248 | 0.140 | 0.276 | 0.180 | 0.125 | 0.092 | 0.141 | 0.143 | 0.082 | 0.093 | 0.120 | 0.075 | 0.073 | | | 0.112 | 0.076 | 0.151 |
| Gold | (g/t) | 0.389 | 0.704 | 0.934 | 0.997 | 0.437 | 0.208 | 0.294 | 0.552 | 0.462 | 0.644 | 0.595 | 0.652 | 0.968 | 0.610 | 0.437 | | | 0.132 | 0.161 | 0.393 |
| Recovered Copper Grade | (%) | 0.642 | 0.557 | 0.377 | 0.245 | 0.529 | 0.409 | 0.474 | 0.196 | 0.224 | 0.197 | 0.184 | 0.274 | 0.322 | 0.193 | 0.267 | | | 0.274 | 0.244 | 0.530 |
| Recovered Copper Grade-Conc | (%) | 0.221 | 0.188 | 0.129 | 0.103 | 0.253 | 0.224 | 0.345 | 0.101 | 0.081 | 0.052 | 0.100 | 0.180 | 0.200 | 0.115 | 0.191 | | | 0.158 | 0.165 | 0.375 |
| Recovered Copper Grade-Agit | (%) | 0.421 | 0.368 | 0.247 | 0.142 | 0.276 | 0.184 | 0.130 | 0.095 | 0.143 | 0.145 | 0.084 | 0.094 | 0.122 | 0.078 | 0.077 | | | 0.116 | 0.079 | 0.155 |
| Tonnes Tails/Kt Feed | (t/kt) | 991.15 | 992.05 | 987.09 | 995.89 | 989.89 | 991.03 | 986.21 | 995.98 | 996.76 | 995.95 | 990.12 | 992.81 | 992.02 | 995.40 | 992.37 | | | 993.68 | 993.42 | 984.99 |
| Copper Grade of Flotation Tail | (%) | 0.465 | 0.403 | 0.273 | 0.165 | 0.313 | 0.225 | 0.174 | 0.120 | 0.170 | 0.169 | 0.107 | 0.120 | 0.145 | 0.101 | 0.109 | | | 0.152 | 0.114 | 0.194 |
| Soluble Copper Grade of Tail | (%) | 0.420 | 0.366 | 0.241 | 0.137 | 0.269 | 0.177 | 0.123 | 0.090 | 0.137 | 0.139 | 0.080 | 0.090 | 0.118 | 0.074 | 0.072 | | | 0.110 | 0.074 | 0.149 |
| Recovered Copper Grade From Tail | (%) | 0.426 | 0.372 | 0.249 | 0.143 | 0.278 | 0.186 | 0.132 | 0.096 | 0.144 | 0.145 | 0.084 | 0.095 | 0.123 | 0.078 | 0.077 | | | 0.117 | 0.080 | 0.157 |
| Tails Processed | (kt) | 16,766 | 20,650 | 16,454 | 7,061 | 1,338 | 1,565 | 1,185 | 718 | 1,893 | 8,126 | 10,732 | 4,211 | 862 | 511 | 9 | 0 | 0 | 1,313 | 2,311 | 7 |
| Recovery of Copper in Tails | (%) | 91.6% | 92.3% | 91.2% | 86.7% | 88.8% | 82.7% | 75.9% | 80.0% | 84.7% | 85.8% | 78.5% | 79.2% | 84.8% | 77.2% | 70.6% | 0.0% | 0.0% | 77.0% | 70.2% | 80.9% |
| Recovered Copper | (klbs) | 157,462 | 169,349 | 90,322 | 22,260 | 8,202 | 6,417 | 3,450 | 1,520 | 6,009 | 25,976 | 19,874 | 8,818 | 2,338 | 878 | 15 | 0 | 0 | 3,386 | 4,075 | 24 |
| Recovery of Copper in Feed | (%) | 62.0% | 62.8% | 61.1% | 53.3% | 48.9% | 41.2% | 25.2% | 43.5% | 57.4% | 65.1% | 40.4% | 31.7% | 35.6% | 36.1% | 25.6% | 0.0% | 0.0% | 37.6% | 28.6% | 27.3% |
| Recovered Copper Grade - Agit Check | (%) | 0.422 | 0.369 | 0.246 | 0.142 | 0.275 | 0.184 | 0.130 | 0.096 | 0.144 | 0.144 | 0.083 | 0.094 | 0.122 | 0.078 | 0.076 | 0.000 | 0.000 | 0.116 | 0.079 | 0.155 |

| Plant Schedule - Sulfide | Units | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ktonnes | (kt) | 2,834 | 5,435 | 9,081 | 16,910 | 21,023 | 20,871 | 21,128 | 21,429 | 20,651 | 16,091 | 13,986 | 19,034 | 21,341 | 21,562 | 21,891 | 21,900 | 21,900 | 21,039 | 20,366 | 21,893 |
| NOP Due To Agitation | ($/t) | 0.16 | 0.36 | -0.47 | -0.42 | -0.44 | -0.08 | -0.36 | -1.08 | -1.33 | -0.92 | -1.93 | -1.76 | -1.39 | -1.07 | -2.10 | -2.07 | -1.47 | -1.64 | -1.73 | -1.46 |
| NSR Due to Agitation | ($/t) | 4.35 | 4.55 | 3.72 | 3.77 | 3.75 | 4.11 | 3.83 | 3.11 | 2.86 | 3.27 | 2.26 | 2.43 | 2.80 | 3.12 | 2.09 | 2.12 | 2.72 | 2.55 | 2.46 | 2.73 |
| Total Copper | (%) | 0.628 | 0.633 | 0.379 | 0.334 | 0.317 | 0.379 | 0.401 | 0.353 | 0.321 | 0.291 | 0.198 | 0.206 | 0.216 | 0.208 | 0.153 | 0.150 | 0.163 | 0.165 | 0.248 | 0.264 |
| Soluble Copper | (%) | 0.046 | 0.051 | 0.044 | 0.046 | 0.046 | 0.049 | 0.044 | 0.035 | 0.032 | 0.039 | 0.026 | 0.029 | 0.034 | 0.038 | 0.025 | 0.025 | 0.033 | 0.031 | 0.027 | 0.031 |
| Gold | (g/t) | 0.235 | 0.398 | 0.647 | 0.700 | 0.619 | 0.438 | 0.324 | 0.351 | 0.347 | 0.389 | 0.512 | 0.488 | 0.461 | 0.513 | 0.394 | 0.332 | 0.345 | 0.387 | 0.169 | 0.260 |
| Recovered Copper Grade | (%) | 0.524 | 0.546 | 0.329 | 0.293 | 0.278 | 0.328 | 0.342 | 0.294 | 0.266 | 0.247 | 0.165 | 0.173 | 0.183 | 0.179 | 0.127 | 0.123 | 0.137 | 0.138 | 0.200 | 0.218 |
| Recovered Copper Grade-Conc | (%) | 0.466 | 0.485 | 0.279 | 0.242 | 0.228 | 0.273 | 0.291 | 0.252 | 0.228 | 0.203 | 0.135 | 0.140 | 0.146 | 0.137 | 0.099 | 0.094 | 0.101 | 0.104 | 0.167 | 0.182 |
| Recovered Copper Grade-Agit | (%) | 0.058 | 0.061 | 0.050 | 0.050 | 0.050 | 0.055 | 0.051 | 0.042 | 0.038 | 0.044 | 0.030 | 0.032 | 0.037 | 0.042 | 0.028 | 0.028 | 0.036 | 0.034 | 0.033 | 0.036 |
| Tonnes Tails/Kt Feed | (t/kt) | 981.35 | 980.61 | 988.84 | 990.30 | 990.88 | 989.09 | 988.35 | 989.91 | 990.90 | 991.89 | 994.60 | 994.38 | 994.17 | 994.52 | 996.05 | 996.22 | 995.96 | 995.85 | 993.32 | 992.73 |
| Copper Grade of Flotation Tail | (%) | 0.165 | 0.152 | 0.101 | 0.092 | 0.089 | 0.107 | 0.112 | 0.102 | 0.094 | 0.089 | 0.064 | 0.066 | 0.070 | 0.071 | 0.054 | 0.055 | 0.063 | 0.062 | 0.081 | 0.083 |
| Soluble Copper Grade of Tail | (%) | 0.045 | 0.050 | 0.043 | 0.045 | 0.045 | 0.048 | 0.044 | 0.034 | 0.031 | 0.038 | 0.026 | 0.028 | 0.033 | 0.037 | 0.024 | 0.025 | 0.033 | 0.030 | 0.027 | 0.031 |
| Recovered Copper Grade From Tail | (%) | 0.059 | 0.062 | 0.050 | 0.051 | 0.051 | 0.056 | 0.052 | 0.042 | 0.039 | 0.044 | 0.030 | 0.033 | 0.038 | 0.042 | 0.028 | 0.028 | 0.037 | 0.034 | 0.033 | 0.037 |
| Tails Processed | (kt) | 2,781 | 5,330 | 8,980 | 16,746 | 20,831 | 20,643 | 20,882 | 21,213 | 20,463 | 15,960 | 13,910 | 18,927 | 21,217 | 21,444 | 21,805 | 21,817 | 21,812 | 20,952 | 20,230 | 21,734 |
| Recovery of Copper in Tails | (%) | 35.8% | 40.8% | 49.5% | 55.4% | 57.3% | 52.3% | 46.4% | 41.2% | 41.5% | 49.4% | 46.9% | 50.0% | 54.3% | 59.2% | 51.9% | 50.9% | 58.7% | 54.8% | 40.7% | 44.6% |
| Recovered Copper | (klbs) | 3,617 | 7,285 | 9,898 | 18,828 | 23,421 | 25,486 | 23,939 | 19,642 | 17,594 | 15,482 | 9,200 | 13,770 | 17,774 | 19,856 | 13,460 | 13,468 | 17,792 | 15,705 | 14,718 | 17,728 |
| Recovery of Copper in Feed | (%) | 9.2% | 9.6% | 13.0% | 15.1% | 15.9% | 14.6% | 12.8% | 11.8% | 12.0% | 15.0% | 15.1% | 15.9% | 17.5% | 20.1% | 18.2% | 18.6% | 22.6% | 20.5% | 13.2% | 13.9% |
| Recovered Copper Grade - Agit Check | (%) | 0.058 | 0.061 | 0.049 | 0.051 | 0.051 | 0.055 | 0.051 | 0.042 | 0.039 | 0.044 | 0.030 | 0.033 | 0.038 | 0.042 | 0.028 | 0.028 | 0.037 | 0.034 | 0.033 | 0.037 |

| Plant Schedule - Tails Leach Material | Units | 21 | 22 | 23 | 24 | 25 | 26 | 27 | 28 | 29 | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ktonnes | (kt) | 21,900 | 21,900 | 21,900 | 21,950 | 22,139 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 19,398 | 848,879 |
| NOP Due To Agitation | ($/t) | -1.75 | -1.77 | -1.79 | -1.68 | -1.46 | -1.44 | -1.42 | -1.44 | -2.01 | -2.37 | -2.62 | -2.60 | -2.57 | -2.81 | -2.30 | -2.13 | -1.71 | -1.85 | 0.22 |
| NSR Due to Agitation | ($/t) | 2.44 | 2.42 | 2.40 | 2.51 | 2.73 | 2.75 | 2.77 | 2.75 | 2.18 | 1.82 | 1.57 | 1.59 | 1.63 | 1.38 | 1.89 | 2.06 | 2.48 | 2.34 | 4.41 |
| Total Copper | (%) | 0.247 | 0.236 | 0.227 | 0.222 | 0.223 | 0.242 | 0.244 | 0.255 | 0.213 | 0.159 | 0.149 | 0.126 | 0.131 | 0.169 | 0.204 | 0.199 | 0.231 | 0.254 | 0.259 |
| Soluble Copper | (%) | 0.027 | 0.027 | 0.027 | 0.029 | 0.032 | 0.032 | 0.032 | 0.032 | 0.025 | 0.021 | 0.017 | 0.018 | 0.018 | 0.014 | 0.020 | 0.023 | 0.028 | 0.026 | 0.055 |
| Gold | (g/t) | 0.329 | 0.352 | 0.350 | 0.349 | 0.294 | 0.159 | 0.192 | 0.290 | 0.371 | 0.293 | 0.141 | 0.135 | 0.180 | 0.154 | 0.136 | 0.163 | 0.214 | 0.150 | 0.358 |
| Recovered Copper Grade | (%) | 0.202 | 0.194 | 0.187 | 0.185 | 0.186 | 0.198 | 0.200 | 0.211 | 0.174 | 0.128 | 0.115 | 0.098 | 0.103 | 0.128 | 0.160 | 0.158 | 0.188 | 0.203 | 0.220 |
| Recovered Copper Grade-Conc | (%) | 0.170 | 0.162 | 0.155 | 0.151 | 0.149 | 0.161 | 0.163 | 0.174 | 0.145 | 0.104 | 0.094 | 0.077 | 0.081 | 0.109 | 0.134 | 0.131 | 0.155 | 0.172 | 0.161 |
| Recovered Copper Grade-Agit | (%) | 0.033 | 0.032 | 0.032 | 0.033 | 0.036 | 0.037 | 0.037 | 0.037 | 0.029 | 0.025 | 0.021 | 0.021 | 0.022 | 0.019 | 0.026 | 0.028 | 0.033 | 0.031 | 0.059 |
| Tonnes Tails/Kt Feed | (t/kt) | 993.21 | 993.52 | 993.79 | 993.95 | 994.04 | 993.55 | 993.47 | 993.02 | 994.19 | 995.84 | 996.23 | 996.93 | 996.76 | 995.63 | 994.63 | 994.77 | 993.79 | 993.13 | 993.32 |
| Copper Grade of Flotation Tail | (%) | 0.078 | 0.075 | 0.072 | 0.071 | 0.074 | 0.081 | 0.081 | 0.081 | 0.068 | 0.056 | 0.055 | 0.050 | 0.050 | 0.060 | 0.070 | 0.068 | 0.076 | 0.082 | 0.099 |
| Soluble Copper Grade of Tail | (%) | 0.027 | 0.027 | 0.027 | 0.028 | 0.031 | 0.031 | 0.031 | 0.031 | 0.024 | 0.020 | 0.017 | 0.018 | 0.018 | 0.013 | 0.020 | 0.022 | 0.028 | 0.025 | 0.054 |
| Recovered Copper Grade From Tail | (%) | 0.033 | 0.033 | 0.032 | 0.033 | 0.036 | 0.037 | 0.037 | 0.037 | 0.029 | 0.025 | 0.021 | 0.021 | 0.022 | 0.019 | 0.026 | 0.028 | 0.034 | 0.032 | 0.059 |
| Tails Processed | (kt) | 21,751 | 21,758 | 21,764 | 21,817 | 22,007 | 21,759 | 21,757 | 21,747 | 21,773 | 21,809 | 21,818 | 21,833 | 21,829 | 21,804 | 21,783 | 21,785 | 21,764 | 19,265 | 843,209 |
| Recovery of Copper in Tails | (%) | 42.3% | 44.0% | 44.4% | 46.8% | 49.1% | 45.7% | 45.7% | 45.7% | 42.6% | 44.6% | 38.5% | 42.8% | 43.1% | 31.4% | 36.5% | 40.5% | 44.4% | 38.5% | 60.3% |
| Recovered Copper | (klbs) | 15,825 | 15,829 | 15,354 | 16,074 | 17,618 | 17,749 | 17,747 | 17,739 | 13,920 | 11,917 | 10,131 | 10,272 | 10,400 | 9,090 | 12,335 | 13,252 | 16,110 | 13,437 | 1,105,157 |
| Recovery of Copper in Feed | (%) | 13.3% | 13.9% | 14.0% | 15.0% | 16.2% | 15.2% | 15.1% | 14.4% | 13.5% | 15.5% | 14.1% | 16.8% | 16.5% | 11.1% | 12.6% | 13.8% | 14.5% | 12.4% | 22.8% |
| Recovered Copper Grade - Agit Check | (%) | 0.033 | 0.033 | 0.032 | 0.033 | 0.036 | 0.037 | 0.037 | 0.037 | 0.029 | 0.025 | 0.021 | 0.021 | 0.022 | 0.019 | 0.026 | 0.027 | 0.033 | 0.031 | 0.059 |

| Plant Schedule - Oxide | Units | 21 | 22 | 23 | 24 | 25 | 26 | 27 | 28 | 29 | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| Ktonnes | (kt) | | | | 184 | 670 | | | | | | | 0 | | | | | | | 97,381 |
| NOP Due To Agitation | ($/t) | | | | 1.98 | 2.03 | | | | | | | 0.00 | | | | | | | 14.36 |
| NSR Due to Agitation | ($/t) | | | | 6.17 | 6.22 | | | | | | | 0.00 | | | | | | | 18.55 |
| Total Copper | (%) | | | | 0.238 | 0.225 | | | | | | | 0.000 | | | | | | | 0.432 |
| Soluble Copper | (%) | | | | 0.079 | 0.080 | | | | | | | 0.000 | | | | | | | 0.251 |
| Gold | (g/t) | | | | 0.150 | 0.189 | | | | | | | 0.000 | | | | | | | 0.644 |
| Recovered Copper Grade | (%) | | | | 0.207 | 0.195 | | | | | | | 0.000 | | | | | | | 0.403 |
| Recovered Copper Grade-Conc | (%) | | | | 0.125 | 0.112 | | | | | | | 0.000 | | | | | | | 0.155 |
| Recovered Copper Grade-Agit | (%) | | | | 0.082 | 0.083 | | | | | | | 0.000 | | | | | | | 0.248 |
| Tonnes Tails/Kt Feed | (t/kt) | | | | 995.01 | 995.53 | | | | | | | 0.00 | | | | | | | 991.57 |
| Copper Grade of Flotation Tail | (%) | | | | 0.114 | 0.114 | | | | | | | 0.000 | | | | | | | 0.279 |
| Soluble Copper Grade of Tail | (%) | | | | 0.077 | 0.078 | | | | | | | 0.000 | | | | | | | 0.244 |
| Recovered Copper Grade From Tail | (%) | | | | 0.083 | 0.083 | | | | | | | 0.000 | | | | | | | 0.250 |
| Tails Processed | (kt) | 0 | 0 | 0 | 183 | 667 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 96,560 |
| Recovery of Copper in Tails | (%) | 0.0% | 0.0% | 0.0% | 72.8% | 72.8% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 89.6% |
| Recovered Copper | (klbs) | 0 | 0 | 0 | 335 | 1,221 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 532,107 |
| Recovery of Copper in Feed | (%) | 0.0% | 0.0% | 0.0% | 34.7% | 36.7% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 0.0% | 57.4% |
| Recovered Copper Grade - Agit Check | (%) | 0.000 | 0.000 | 0.000 | 0.083 | 0.083 | 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 | 0.248 |
| **Plant Schedule - Sulfide** | **Units** | **21** | **22** | **23** | **24** | **25** | **26** | **27** | **28** | **29** | **30** | **31** | **32** | **33** | **34** | **35** | **36** | **37** | **38** | **TOTAL** |
| Ktonnes | (kt) | 21,900 | 21,900 | 21,900 | 21,766 | 21,469 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 19,398 | 751,498 |
| NOP Due To Agitation | ($/t) | -1.75 | -1.77 | -1.79 | -1.71 | -1.57 | -1.44 | -1.42 | -1.44 | -2.01 | -2.37 | -2.62 | -2.60 | -2.57 | -2.81 | -2.30 | -2.13 | -1.71 | -1.85 | -1.61 |
| NSR Due to Agitation | ($/t) | 2.44 | 2.42 | 2.40 | 2.48 | 2.62 | 2.75 | 2.77 | 2.75 | 2.18 | 1.82 | 1.57 | 1.59 | 1.63 | 1.38 | 1.89 | 2.06 | 2.48 | 2.34 | 2.58 |
| Total Copper | (%) | 0.247 | 0.236 | 0.227 | 0.222 | 0.223 | 0.242 | 0.244 | 0.255 | 0.213 | 0.159 | 0.149 | 0.126 | 0.131 | 0.169 | 0.204 | 0.199 | 0.231 | 0.254 | 0.236 |
| Soluble Copper | (%) | 0.027 | 0.027 | 0.027 | 0.029 | 0.031 | 0.032 | 0.032 | 0.032 | 0.025 | 0.021 | 0.017 | 0.018 | 0.018 | 0.014 | 0.020 | 0.023 | 0.028 | 0.026 | 0.030 |
| Gold | (g/t) | 0.329 | 0.352 | 0.350 | 0.351 | 0.297 | 0.159 | 0.192 | 0.290 | 0.371 | 0.293 | 0.141 | 0.135 | 0.180 | 0.154 | 0.136 | 0.163 | 0.214 | 0.150 | 0.321 |
| Recovered Copper Grade | (%) | 0.202 | 0.194 | 0.187 | 0.185 | 0.185 | 0.198 | 0.200 | 0.211 | 0.174 | 0.128 | 0.115 | 0.098 | 0.103 | 0.128 | 0.160 | 0.158 | 0.188 | 0.203 | 0.196 |
| Recovered Copper Grade-Conc | (%) | 0.170 | 0.162 | 0.155 | 0.151 | 0.150 | 0.161 | 0.163 | 0.174 | 0.145 | 0.104 | 0.094 | 0.077 | 0.081 | 0.109 | 0.134 | 0.131 | 0.155 | 0.172 | 0.161 |
| Recovered Copper Grade-Agit | (%) | 0.033 | 0.032 | 0.032 | 0.033 | 0.035 | 0.037 | 0.037 | 0.037 | 0.029 | 0.025 | 0.021 | 0.021 | 0.022 | 0.019 | 0.026 | 0.028 | 0.033 | 0.031 | 0.034 |
| Tonnes Tails/Kt Feed | (t/kt) | 993.21 | 993.52 | 993.79 | 993.94 | 994.00 | 993.55 | 993.47 | 993.02 | 994.19 | 995.84 | 996.23 | 996.93 | 996.76 | 995.63 | 994.63 | 994.77 | 993.79 | 993.13 | 993.55 |
| Copper Grade of Flotation Tail | (%) | 0.078 | 0.075 | 0.072 | 0.071 | 0.073 | 0.081 | 0.081 | 0.081 | 0.068 | 0.056 | 0.055 | 0.050 | 0.050 | 0.060 | 0.070 | 0.068 | 0.076 | 0.082 | 0.075 |
| Soluble Copper Grade of Tail | (%) | 0.027 | 0.027 | 0.027 | 0.028 | 0.030 | 0.031 | 0.031 | 0.031 | 0.024 | 0.020 | 0.017 | 0.018 | 0.018 | 0.013 | 0.020 | 0.022 | 0.028 | 0.025 | 0.029 |
| Recovered Copper Grade From Tail | (%) | 0.033 | 0.033 | 0.032 | 0.033 | 0.035 | 0.037 | 0.037 | 0.037 | 0.029 | 0.025 | 0.021 | 0.021 | 0.022 | 0.019 | 0.026 | 0.028 | 0.034 | 0.032 | 0.035 |
| Tails Processed | (kt) | 21,751 | 21,758 | 21,764 | 21,634 | 21,340 | 21,759 | 21,757 | 21,747 | 21,773 | 21,809 | 21,818 | 21,833 | 21,829 | 21,804 | 21,783 | 21,785 | 21,764 | 19,265 | 746,649 |
| Recovery of Copper in Tails | (%) | 42.3% | 44.0% | 44.4% | 46.5% | 47.9% | 45.7% | 45.7% | 45.7% | 42.6% | 44.6% | 38.5% | 42.8% | 43.1% | 31.4% | 36.5% | 40.5% | 44.4% | 38.5% | 46.2% |
| Recovered Copper | (klbs) | 15,825 | 15,829 | 15,354 | 15,739 | 16,399 | 17,749 | 17,747 | 17,739 | 13,920 | 11,917 | 10,131 | 10,272 | 10,400 | 9,090 | 12,335 | 13,252 | 16,110 | 13,437 | 572,242 |
| Recovery of Copper in Feed | (%) | 13.3% | 13.9% | 14.0% | 14.8% | 15.5% | 15.2% | 15.1% | 14.4% | 13.5% | 15.5% | 14.1% | 16.8% | 16.5% | 11.1% | 12.6% | 13.8% | 14.5% | 12.4% | 14.6% |
| Recovered Copper Grade - Agit Check | (%) | 0.033 | 0.033 | 0.032 | 0.033 | 0.035 | 0.037 | 0.037 | 0.037 | 0.029 | 0.025 | 0.021 | 0.021 | 0.022 | 0.019 | 0.026 | 0.027 | 0.033 | 0.031 | 0.035 |

## Table 16-5: Leach Pad Stacking Schedule

| | Units | -2 | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Leach Pad Stacking Schedule:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 3,600 | 14,600 | 14,600 | 14,600 | 14,600 | 7,300 | 7,360 | 5,289 | 633 | 4,355 | 1,642 | 8,092 | 9,951 | 1,213 |
| NSR | ($/t) | 10.82 | 14.17 | 19.05 | 19.06 | 14.63 | 9.38 | 15.64 | 13.04 | 7.21 | 8.41 | 7.57 | 7.25 | 7.02 | 6.80 |
| Total Copper | (%) | 0.199 | 0.240 | 0.318 | 0.311 | 0.245 | 0.168 | 0.269 | 0.242 | 0.211 | 0.139 | 0.138 | 0.118 | 0.118 | 0.156 |
| Soluble Copper | (%) | 0.104 | 0.145 | 0.211 | 0.213 | 0.155 | 0.087 | 0.162 | 0.130 | 0.061 | 0.077 | 0.064 | 0.066 | 0.061 | 0.056 |
| Gold | (g/t) | 0.079 | 0.114 | 0.135 | 0.152 | 0.177 | 0.150 | 0.091 | 0.108 | 0.131 | 0.155 | 0.172 | 0.217 | 0.166 | 0.110 |
| Recovered Copper Grade | (%) | 0.144 | 0.189 | 0.254 | 0.254 | 0.195 | 0.125 | 0.209 | 0.175 | 0.096 | 0.112 | 0.101 | 0.097 | 0.094 | 0.091 |
| Contained Copper | (klbs) | 15,795 | 77,164 | 102,234 | 99,969 | 78,975 | 27,008 | 43,630 | 28,247 | 2,951 | 13,350 | 4,988 | 21,095 | 25,924 | 4,181 |
| Recoverable Copper | (klbs) | 11,465 | 60,847 | 81,853 | 81,829 | 62,838 | 20,166 | 33,900 | 20,348 | 1,338 | 10,769 | 3,650 | 17,349 | 20,600 | 2,441 |
| Copper Recovery | (%) | 72.6% | 78.9% | 80.1% | 81.9% | 79.6% | 74.7% | 77.7% | 72.0% | 45.3% | 80.7% | 73.2% | 82.2% | 79.5% | 58.4% |
| **As Produced From Mine:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 7,717 | 21,646 | 18,476 | 13,395 | 4,026 | 4,040 | 7,360 | 5,289 | 633 | 4,355 | 1,642 | 8,092 | 9,951 | 1,213 |
| NSR | ($/t) | 10.82 | 14.17 | 19.05 | 19.06 | 13.30 | 8.22 | 15.64 | 13.04 | 7.21 | 8.41 | 7.57 | 7.25 | 7.02 | 6.80 |
| Total Copper | (%) | 0.199 | 0.240 | 0.318 | 0.310 | 0.217 | 0.143 | 0.269 | 0.242 | 0.211 | 0.139 | 0.138 | 0.118 | 0.118 | 0.156 |
| Soluble Copper | (%) | 0.104 | 0.145 | 0.211 | 0.213 | 0.146 | 0.074 | 0.162 | 0.130 | 0.061 | 0.077 | 0.064 | 0.066 | 0.061 | 0.056 |
| Gold | (g/t) | 0.079 | 0.114 | 0.135 | 0.153 | 0.337 | 0.207 | 0.091 | 0.108 | 0.131 | 0.155 | 0.172 | 0.217 | 0.166 | 0.110 |
| Recovered Copper Grade | (%) | 0.144 | 0.189 | 0.254 | 0.254 | 0.178 | 0.110 | 0.209 | 0.175 | 0.096 | 0.112 | 0.101 | 0.097 | 0.094 | 0.091 |
| **Direct to Crusher:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 3,600 | 14,600 | 14,600 | 13,395 | 4,026 | 4,040 | 7,360 | 5,289 | 633 | 4,355 | 1,642 | 8,092 | 9,951 | 1,213 |
| NSR | ($/t) | 10.82 | 14.17 | 19.05 | 19.06 | 13.30 | 8.22 | 15.64 | 13.04 | 7.21 | 8.41 | 7.57 | 7.25 | 7.02 | 6.80 |
| Total Copper | (%) | 0.199 | 0.240 | 0.318 | 0.310 | 0.217 | 0.143 | 0.269 | 0.242 | 0.211 | 0.139 | 0.138 | 0.118 | 0.118 | 0.156 |
| Soluble Copper | (%) | 0.104 | 0.145 | 0.211 | 0.213 | 0.146 | 0.074 | 0.162 | 0.130 | 0.061 | 0.077 | 0.064 | 0.066 | 0.061 | 0.056 |
| Gold | (g/t) | 0.079 | 0.114 | 0.135 | 0.153 | 0.337 | 0.207 | 0.091 | 0.108 | 0.131 | 0.155 | 0.172 | 0.217 | 0.166 | 0.110 |
| Recovered Copper Grade | (%) | 0.144 | 0.189 | 0.254 | 0.254 | 0.178 | 0.110 | 0.209 | 0.175 | 0.096 | 0.112 | 0.101 | 0.097 | 0.094 | 0.091 |
| **To Leach Stockpile:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 4117 | 7,046 | 3,876 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| NSR | ($/t) | 10.82 | 14.17 | 19.05 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 | 0.00 |
| Total Copper | (%) | 0.199 | 0.240 | 0.318 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 |
| Soluble Copper | (%) | 0.104 | 0.145 | 0.211 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 |
| Gold | (g/t) | 0.079 | 0.114 | 0.135 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 |
| Recovered Copper Grade | (%) | 0.144 | 0.189 | 0.254 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 | 0.000 |
| **Stockpile Balance:** | | | | | | | | | | | | | | | |
| Beginning Balance | (kt) | 0 | 4,117 | 11,163 | 15,039 | 13,834 | 3,260 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Added During Period | (kt) | 4,117 | 7,046 | 3,876 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Removed During Period | (kt) | 0 | 0 | 0 | 1,205 | 10,574 | 3,260 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| Ending Balance | (kt) | 4,117 | 11,163 | 15,039 | 13,834 | 3,260 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 |
| **Stockpile Reclaim:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | | | | 1,205 | 10,574 | 3,260 | | | | | | | | |
| NSR | ($/t) | | | | 19.05 | 15.13 | 10.82 | | | | | | | | |
| Total Copper | (%) | | | | 0.318 | 0.256 | 0.199 | | | | | | | | |
| Soluble Copper | (%) | | | | 0.211 | 0.158 | 0.104 | | | | | | | | |
| Gold | (g/t) | | | | 0.135 | 0.117 | 0.079 | | | | | | | | |
| Recovered Copper Grade | (%) | | | | 0.254 | 0.202 | 0.144 | | | | | | | | |

| | Units | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 | 25 | TOTAL |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Leach Pad Stacking Schedule:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 6 | 0 | 102 | 0 | 0 | 157 | 739 | 0 | 0 | 0 | 0 | 762 | 921 | 110,522 |
| NSR | ($/t) | 10.29 | 0.00 | 7.33 | 0.00 | 0.00 | 19.61 | 12.29 | 0.00 | 0.00 | 0.00 | 0.00 | 8.29 | 11.01 | 13.47 |
| Total Copper | (%) | 0.235 | 0.000 | 0.141 | 0.000 | 0.000 | 0.306 | 0.221 | 0.000 | 0.000 | 0.000 | 0.000 | 0.133 | 0.174 | 0.228 |
| Soluble Copper | (%) | 0.096 | 0.000 | 0.061 | 0.000 | 0.000 | 0.216 | 0.124 | 0.000 | 0.000 | 0.000 | 0.000 | 0.076 | 0.105 | 0.140 |
| Gold | (g/t) | 0.093 | 0.000 | 0.076 | 0.000 | 0.000 | 0.122 | 0.120 | 0.000 | 0.000 | 0.000 | 0.000 | 0.082 | 0.108 | 0.144 |
| Recovered Copper Grade | (%) | 0.138 | 0.000 | 0.098 | 0.000 | 0.000 | 0.262 | 0.164 | 0.000 | 0.000 | 0.000 | 0.000 | 0.111 | 0.147 | 0.180 |
| Contained Copper | (klbs) | 31 | 0 | 317 | 0 | 0 | 1,059 | 3,603 | 0 | 0 | 0 | 0 | 2,234 | 3,533 | 556,290 |
| Recoverable Copper | (klbs) | 18 | 0 | 220 | 0 | 0 | 907 | 2,674 | 0 | 0 | 0 | 0 | 1,865 | 2,985 | 438,062 |
| Copper Recovery | (%) | 58.7% | 0.0% | 69.5% | 0.0% | 0.0% | 85.6% | 74.2% | 0.0% | 0.0% | 0.0% | 0.0% | 83.5% | 84.5% | 78.7% |
| **As Produced From Mine:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 6 | 0 | 102 | 0 | 0 | 157 | 739 | 0 | 0 | 0 | 0 | 762 | 921 | 110,522 |
| NSR | ($/t) | 10.29 | 0.00 | 7.33 | 0.00 | 0.00 | 19.61 | 12.29 | 0.00 | 0.00 | 0.00 | 0.00 | 8.29 | 11.01 | 13.47 |
| Total Copper | (%) | 0.235 | 0.000 | 0.141 | 0.000 | 0.000 | 0.306 | 0.221 | 0.000 | 0.000 | 0.000 | 0.000 | 0.133 | 0.174 | 0.228 |
| Soluble Copper | (%) | 0.096 | 0.000 | 0.061 | 0.000 | 0.000 | 0.216 | 0.124 | 0.000 | 0.000 | 0.000 | 0.000 | 0.076 | 0.105 | 0.140 |
| Gold | (g/t) | 0.093 | 0.000 | 0.076 | 0.000 | 0.000 | 0.122 | 0.120 | 0.000 | 0.000 | 0.000 | 0.000 | 0.082 | 0.108 | 0.144 |
| Recovered Copper Grade | (%) | 0.138 | 0.000 | 0.098 | 0.000 | 0.000 | 0.262 | 0.164 | 0.000 | 0.000 | 0.000 | 0.000 | 0.111 | 0.147 | 0.180 |
| **Direct to Crusher:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 6 | 0 | 102 | 0 | 0 | 157 | 739 | 0 | 0 | 0 | 0 | 762 | 921 | 95,483 |
| NSR | ($/t) | 10.29 | 0.00 | 7.33 | 0.00 | 0.00 | 19.61 | 12.29 | 0.00 | 0.00 | 0.00 | 0.00 | 8.29 | 11.01 | 13.30 |
| Total Copper | (%) | 0.235 | 0.000 | 0.141 | 0.000 | 0.000 | 0.306 | 0.221 | 0.000 | 0.000 | 0.000 | 0.000 | 0.133 | 0.174 | 0.225 |
| Soluble Copper | (%) | 0.096 | 0.000 | 0.061 | 0.000 | 0.000 | 0.216 | 0.124 | 0.000 | 0.000 | 0.000 | 0.000 | 0.076 | 0.105 | 0.138 |
| Gold | (g/t) | 0.093 | 0.000 | 0.076 | 0.000 | 0.000 | 0.122 | 0.120 | 0.000 | 0.000 | 0.000 | 0.000 | 0.082 | 0.108 | 0.149 |
| Recovered Copper Grade | (%) | 0.138 | 0.000 | 0.098 | 0.000 | 0.000 | 0.262 | 0.164 | 0.000 | 0.000 | 0.000 | 0.000 | 0.111 | 0.147 | 0.178 |
| **To Leach Stockpile:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 15,039 |
| NSR | ($/t) | 0.00 | 0.00 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 14.51 |
| Total Copper | (%) | 0.000 | 0.000 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0.249 |
| Soluble Copper | (%) | 0.000 | 0.000 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0.150 |
| Gold | (g/t) | 0.000 | 0.000 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0.110 |
| Recovered Copper Grade | (%) | 0.000 | 0.000 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0.194 |
| **Stockpile Balance:** | | | | | | | | | | | | | | | |
| Beginning Balance | (kt) | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| Added During Period | (kt) | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 15,039 |
| Removed During Period | (kt) | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 15,039 |
| Ending Balance | (kt) | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | 0 | |
| **Stockpile Reclaim:** | | | | | | | | | | | | | | | |
| Ktonnes | (kt) | | | | | | | | | | | | | | 15,039 |
| NSR | ($/t) | | | | | | | | | | | | | | 14.51 |
| Total Copper | (%) | | | | | | | | | | | | | | 0.249 |
| Soluble Copper | (%) | | | | | | | | | | | | | | 0.150 |
| Gold | (g/t) | | | | | | | | | | | | | | 0.110 |
| Recovered Copper Grade | (%) | | | | | | | | | | | | | | 0.194 |

## 16.7 VALUELESS ROCK MANAGEMENT

Figure 16-8 shows the final pit and the various valueless rock management areas (VRMA's) and stockpiles. Total waste in the mine plan is 839.0 million tonnes that consists of 748.1 million tonnes of rock and 90.9 million tonnes of overburden. The facilities to contain this material are:

- The southwest VRMA contains 332.1 million tonnes.
- The west VRMA contains 377.8 million tonnes.
- The pit backfill area in the west pit area contains 129.1 million tonnes.

Low-grade mill ore that is stockpiled during open pit operations and processed after open pit mining is complete is 175.3 million tonnes. This is stored in two facilities:

- The north stockpile (Stage 1) contains 114.4 million tonnes.
- The south stockpile (Stage 2) contains 60.9 million tonnes.

During preproduction and the first four years of commercial production the north stockpile will be used to stockpile heap leach ore (only). This amounts to 15 million tonnes which will all be processed by the end of Year 4.

The VRMA's and stockpiles are all constructed in lifts from the bottom up. They are developed in 30 m lifts at angle of repose (37°). There is a 50 m setback between lifts so the overall slope angle is 3H:1V, about 18.5°. It is anticipated that this is flat enough to make closure easier.

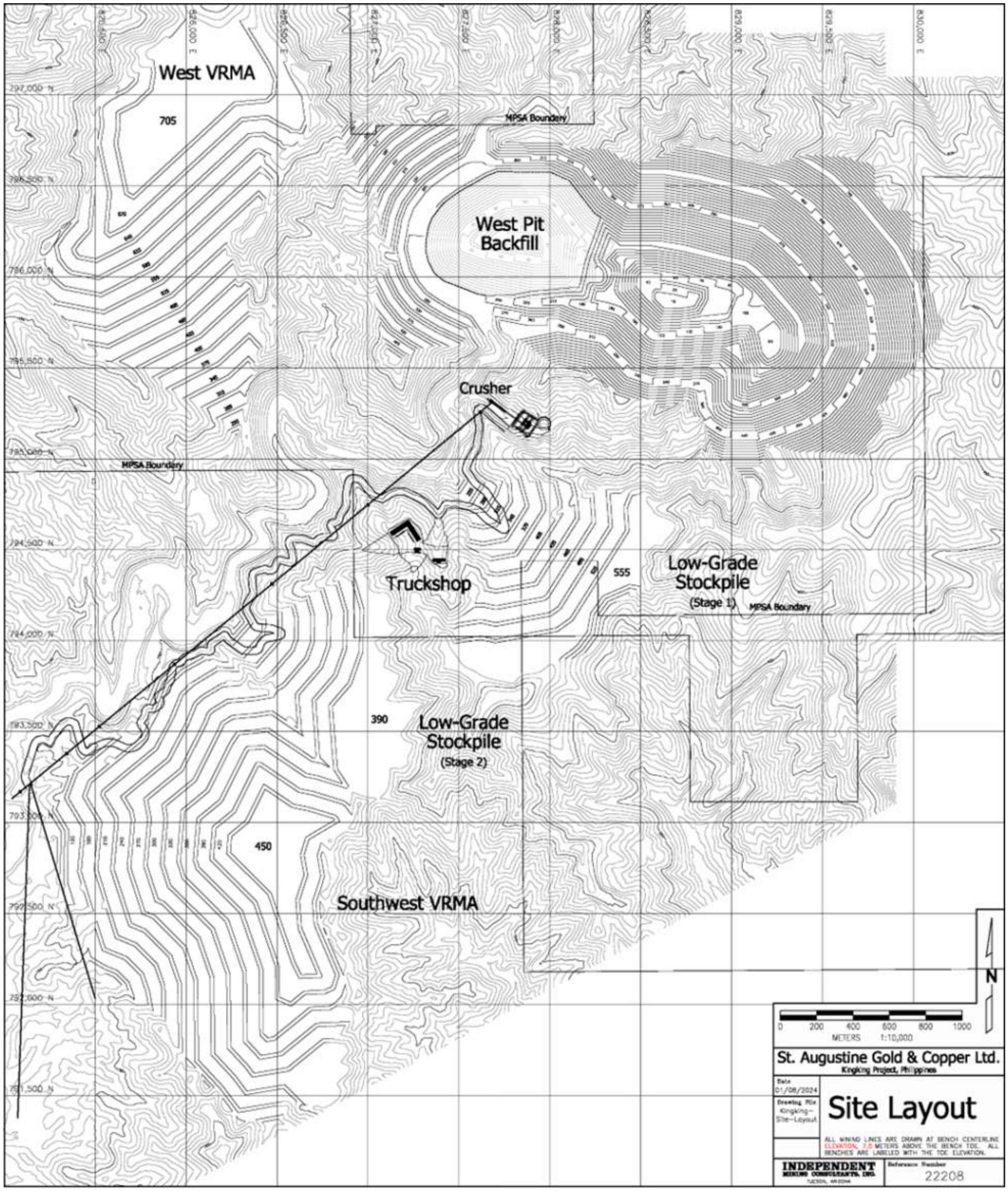



Figure 16-8: Site Layout (IMC, 2024)

## 16.8 MINING EQUIPMENT

This Technical Report assumes that the mining will be performed by a contractor throughout the life of the mine (contract mining). Mine major equipment requirements were sized and estimated on a first principles basis, based on the mine production schedule, the mine work schedule, and estimated equipment productivity rates. The size and type of mining equipment is consistent with the size of the Kingking Project, i.e. peak material movements of approximately 65 million tonnes per year.

Table 16-6 summarizes the requirements. It shows ramp up of the fleet for the two years of preproduction and the first two years of commercial production and the peak fleet requirements. Year -2 requirements are one small drill, one large loader, and 10 Caterpillar 785 class trucks. For Year -1 one large drill, one hydraulic excavator and 9 Caterpillar 793 class trucks are added. During Year 1, the first cable shovel is put into production, and an additional large drill and small drill are added to the fleet, along with additional large trucks. The second cable shovel is added to the fleet during Year 2.

**Table 16-6: Mining Major Equipment Fleet Requirements**

| Equipment Type | Capacity/ Power | Year -2 | Year -1 | Year 1 | Year 2 | Peak Year 16 |
|---|---|---|---|---|---|---|
| Caterpillar MD6380 Drill | (273 mm) | 0 | 1 | 2 | 2 | 2 |
| Epiroc SmartROC D65 | (152 mm) | 1 | 1 | 2 | 2 | 2 |
| Caterpillar 7395 Cable Shovel | (39 cu m) | 0 | 0 | 1 | 2 | 2 |
| Caterpillar 6060FS Hyd Shovel | (34 cu m) | 0 | 1 | 1 | 1 | 1 |
| Caterpillar 995 Wheel Loader | (26 cu m) | 1 | 1 | 1 | 1 | 1 |
| Caterpillar 793F Truck | (226 mt) | 0 | 9 | 16 | 21 | 30 |
| Caterpillar 785G Truck | (133 mt) | 10 | 10 | 8 | 5 | 10 |
| Caterpillar D10T2 Track Dozer | (447 kw) | 0 | 1 | 2 | 3 | 3 |
| Caterpillar D9T Track Dozer | (325 kw) | 1 | 2 | 3 | 3 | 3 |
| Caterpillar 844K Wheel Dozer | (521 kw) | 0 | 1 | 2 | 3 | 3 |
| Caterpillar 16M3 Motor Grader | (216 kw) | 1 | 2 | 3 | 3 | 3 |
| Water Truck - 30,000 gal | (113,550 l) | 1 | 2 | 3 | 3 | 3 |
| Caterpillar 350 Excavator | (2.7 cu m) | 1 | 1 | 2 | 2 | 2 |
| **Total** | | **16** | **32** | **46** | **51** | **65** |

In addition to this equipment, the mine capital cost estimate will include an allowance for small equipment such as fuel and lubricant trucks, mechanic and welding trucks, tire handlers, lowboy trailers and tractors to transport track equipment, cranes, surveying and engineering equipment. The mine operating cost estimate will include an allowance for mine services which includes the cost of running the small equipment.

The specifications for mine infrastructure such as the shop and warehouse, fuel and lubricant facilities, explosive storage facilities, and offices is included in the scope of the general infrastructure contractor.

# 17 RECOVERY METHODS

## 17.1 PROCESS DESCRIPTION

The Kingking processing facility will recover copper by conventional flotation, agitated leaching of the flotation tails, and 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 Minproc (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 to design a process plant for the Kingking 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 Kingking Project is based on a 365-day calendar year. The yearly ore tonnage is nominally 21.9 million tonnes, 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 Kingking process using MetSim software. The process simulation assumed overall recoveries shown on Table 17-2 for gold, sulfide copper and oxide copper. These recoveries do not represent average recoveries but rather estimate scenarios of maximum expected flowrates to adequately size equipment. These figures need to be revisited in the next phase of the study in conjunction with an updated mining schedule.

These recoveries are based on the recovery equations supplied by AMEC Minproc 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.

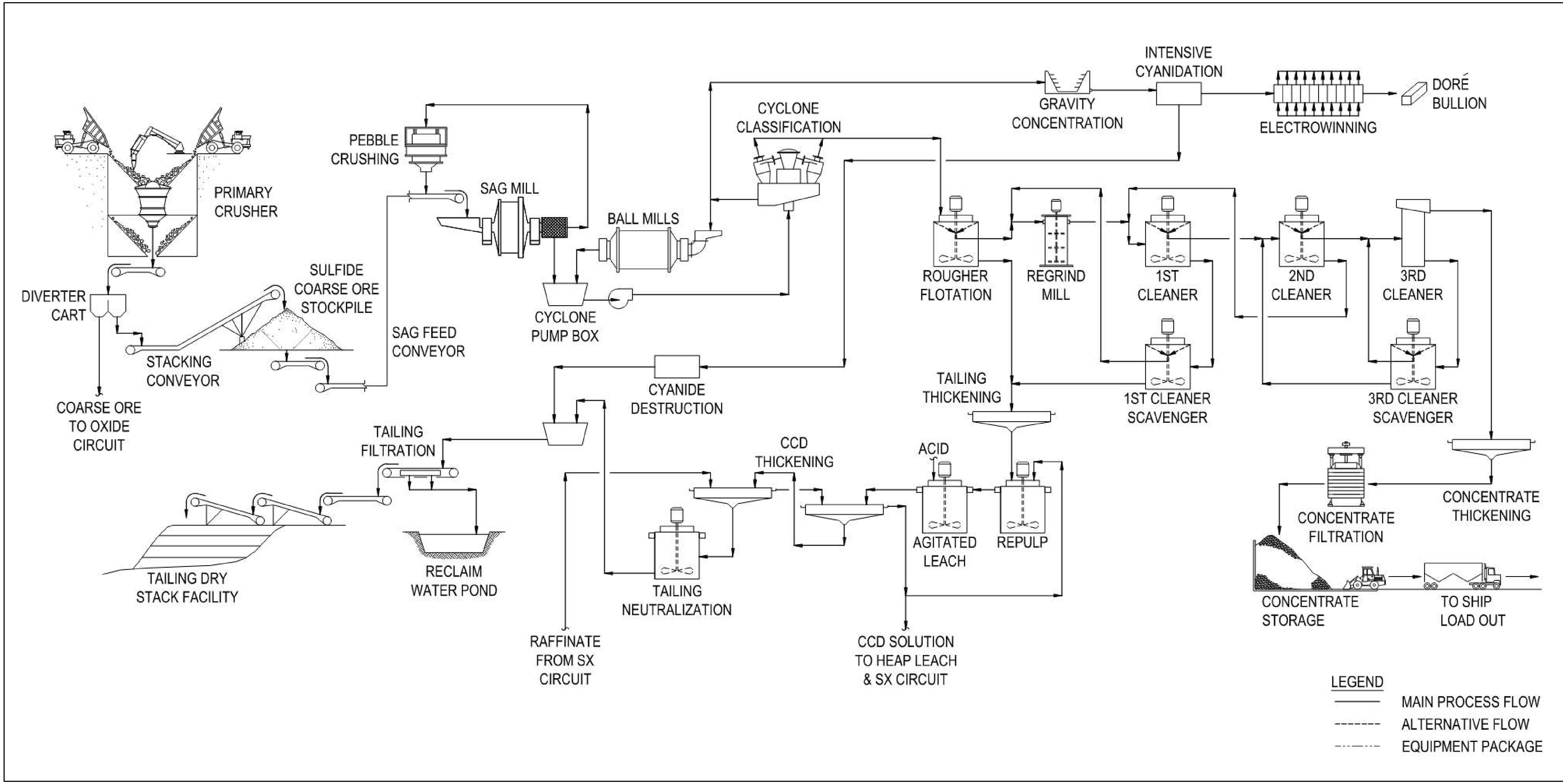



**Figure 17-1: Simplified Process Flow Diagram for the Kingking Sulfide Plant (M3, 2013)**

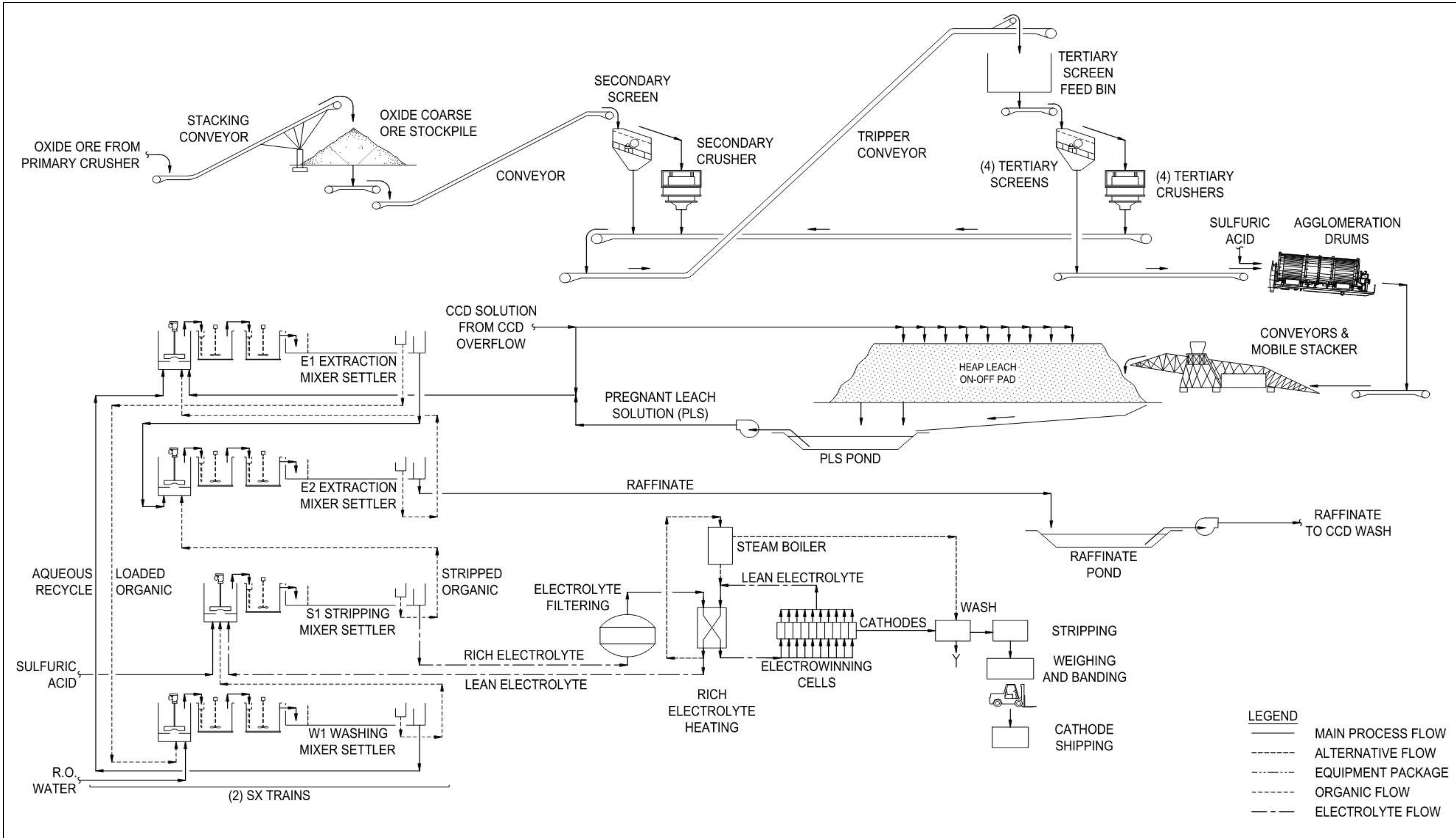



**Figure 17-2: Simplified Process Flow Diagram for the Kingking Heap Leach Operation (M3, 2013)**

**Table 17-1: Process Design Criteria**

| Description | Sulfides | Oxides |
|---|---|---|
| **Primary Crusher** | | |
| Feed F80, mm | 1,200 | 400 |
| Product P80, mm | 120 | 120 |
| Crushing Work Index, kWh/t | 11 | 11 |
| **SAG Mill Grinding** | | |
| Feed F80, mm | 120 | 120 |
| Product P80, mm | 11.1 | 11.1 |
| SAG Mill Axb Parameter, 80th Percentile | 30.5 | 38.2 |
| SAG Mill Axb Parameter, Median | 36.0 | 71.8 |
| **Ball Mill Grinding** | | |
| Feed F80, microns | 3,105 | 3,105 |
| Product P80, microns | 106 | 106 |
| Ball Mill Work Index, kWh/t, 80th Percentile | 15.6 | 12.8 |
| Ball Mill Work Index, kWh/t, Average | 13.8 | 11.0 |
| **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 | |
| pH (same for all flotation stages) | 10-10.5 | 10-10.5 |
| **Flotation Tailing, Agitated Leach** | | |
| Leach Time, h | 4 | 4 |
| % Solids | 35 - 40 | 35 - 40 |
| Acid Consumption, kg/t | 25 | 25 |
| Leach Temperature, $^{o}C$ | 50 | 50 |

**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 to Concentrate | 86 |
| Au | 0.39 – 0.61 g/t | Au to Concentrate | 60 |
| | | Au to 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 each a have total capacity of 200,000 tonnes and live capacity of 60,000 tonnes. The live capacities are nominally equivalent to 24 hours of SAG mill feed and 36 hours of secondary/tertiary crushing plant feed at the peak of heap leach ore production. 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 Kingking 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 Kingking 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 80 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 being reground.

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, 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.

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

### 17.7.1 Cyanide Destruction and Cyanide Code Certification

The leached solids and 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 destroy free cyanide and reduce weak-acid dissociable (WAD) cyanide down to <50 ppm, in accordance with the International Cyanide Management Code (ICMC or "Cyanide Code"). The detoxified slurry will be disposed into tailing handling facility.

Reagents used for cyanide detoxification include milk of lime, sodium metabisulfite or ammonium bisulfite, and oxygen supplied by air injected into the reactors. Having come from a copper deposit, the cyanide leach tailing slurry is expected to contain enough dissolved copper, which eliminates the need for copper sulfate addition.

The operations are designed to follow ICMC guidelines. The Company plans to become a signatory to the Cyanide Code when construction starts, and to apply for initial certification audit as soon as the operations ramp up is completed, well within the designated time allotted for by the Cyanide Code.

## 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 tonne of ore for four hours at 50°C. Pregnant solution will be recovered in a counter-current decantation (CCD) circuit using raffinate from the SX-EW plant as the wash solution.

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 CCD using two trains of high-rate thickeners (55-m diameter) with six thickeners per train. The clear solution overflow from the CCD trains is 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.

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. To attain the 50°C leach temperature target, the PLS for leach dilution will be preheated to 63°C by steam through shell-and-tube heat exchangers. Steam will be generated by a diesel-fired steam generator.

## 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 6 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 Kingking 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 Kingking 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 tph and reclaiming equipment capacity of 2500 tph. Ore will be stacked on the heap leach pads in 6 m lift heights.

#### 17.9.2 Water Balance and Solution Management

The Kingking 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 by covering the top of the heap with removable plastic liners to divert meteoric water away from the ore.

### 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 Kingking Project using MetSim modeling software. The Kingking process plant and heap leaching are projected to require 673.6 $m^3$/h of freshwater 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 freshwater requirement will then be 855.6 $m^3$/h. If the consumption by heap leaching is excluded, the equivalent freshwater consumption is 0.28 $m^3$/tonne. This is lower than the water consumption from typical operations (approximately 0.4 to 0.5 $m^3$/tonne) because of better water recycling from tailing dewatering.

**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 Suppressor, 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/tonne | 0.345 |
| Grinding Balls, 75 mm kg/tonne | 0.241 |

## 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/tonne or $1.48/tonne of ore processed.

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

| Area | Total kWh/Year | Total Cost US$/year* |
|---|---|---|
| **Concentrator** | | |
| Primary Crushing/Coarse Ore Storage & Reclaim | 32,367,128 | $ 3,301,447 |
| Grinding | 526,455,372 | $ 53,698,448 |
| Flotation | 55,512,732 | $ 5,662,299 |
| Concentrate Thickening and Filtration | 7,426,064 | $ 757,459 |
| Agitated Leach | 95,656,233 | $ 9,756,936 |
| Tailing Disposal | 41,910,719 | $ 4,274,893 |
| Water Systems | 1,693,378 | $ 172,725 |
| Ancillary Facilities | 3,163,778 | $ 322,705 |
| Gravity Gold Circuit and Intensive Leach | 825,929 | $ 84,245 |
| Gold Refinery | 1,136,414 | $ 115,914 |
| Subtotal | 766,147,748 | $ 78,147,070 |
| **Heap Leach** | | |
| Crusher | 53,540,828 | $ 5,461,164 |
| Heap Leach | 55,578,492 | $ 5,669,006 |
| Solvent Extraction | 2,879,625 | $ 293,722 |
| Tank Farm | 3,483,919 | $ 355,360 |
| Electrowinning | 201,331,983 | $ 20,535,862 |
| Subtotal | 316,814,983 | $ 32,315,114 |
| **Grand Total** | **1,082,962,595** | **$ 110,462,185** |

*Power Cost: US$ 0.102/kWh

### 17.15.1 Control System

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 | Quantity | 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 |
| Forklifts | 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-tonne 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 by 60-meter 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 17-5.

## Table 17-7: Metal Production

| Project Year | CONCENTRATOR | | | | | | HEAP LEACH | |
|---|---|---|---|---|---|---|---|---|
| | Production, kt | Flotation | | | Agitated Leach | Gravity Gold | | |
| | | Concentrate, kt | Recovered Copper, klbs | Recovered Gold, kOzs | Copper Cathode, klbs | Gold Bullion, kOzs | Production, kt | Copper Cathode, klbs |
| -2 | | | | | | | 3,600 | |
| -1 | | | | | | | 14,600 | 72,313 |
| 1 | 19,750 | 202.56 | 111,533 | 82.73 | 160,889 | 55.8 | 14,600 | 81,854 |
| 2 | 26,250 | 262.33 | 144,385 | 209.77 | 176,298 | 130.4 | 14,600 | 81,829 |
| 3 | 25,750 | 187.69 | 103,262 | 298.52 | 100,266 | 157.6 | 14,600 | 62,839 |
| 4 | 24,000 | 193.17 | 106,318 | 310.37 | 41,032 | 126.6 | 7,300 | 20,166 |
| 5 | 22,375 | 205.40 | 113,214 | 245.04 | 31,630 | 83.8 | 7,360 | 33,901 |
| 6 | 22,450 | 241.87 | 133,412 | 168.30 | 31,895 | 57.9 | 5,289 | 20,348 |
| 7 | 22,330 | 262.72 | 144,688 | 131.99 | 27,393 | 44.3 | 633 | 1,338 |
| 8 | 22,150 | 219.12 | 120,657 | 145.63 | 21,157 | 48.8 | 4,355 | 10,769 |
| 9 | 22,550 | 194.08 | 107,194 | 143.60 | 23,580 | 50.7 | 1,642 | 3,650 |
| 10 | 24,250 | 147.63 | 81,367 | 172.33 | 41,409 | 80.8 | 8,092 | 17,349 |
| 11 | 24,825 | 118.88 | 65,522 | 225.84 | 29,107 | 87.2 | 9,951 | 20,600 |
| 12 | 23,275 | 137.46 | 75,577 | 216.41 | 22,596 | 73.8 | 1,213 | 2,441 |
| 13 | 22,210 | 131.35 | 72,523 | 190.60 | 20,115 | 65.5 | 6 | 18 |
| 14 | 22,075 | 120.52 | 66,425 | 198.28 | 20,733 | 69.9 | | |
| 15 | 21,900 | 86.54 | 47,817 | 151.38 | 13,475 | 52.8 | 102 | 220 |
| 16 | 21,900 | 82.78 | 45,384 | 126.03 | 13,468 | 44.4 | | |
| 17 | 21,900 | 88.48 | 48,764 | 127.44 | 17,792 | 45.8 | | |
| 18 | 22,360 | 95.66 | 52,840 | 141.72 | 19,096 | 51.2 | 157 | 907 |
| 19 | 22,692 | 151.37 | 83,443 | 67.08 | 18,793 | 23.3 | 739 | 2,674 |
| 20 | 21,900 | 159.27 | 87,902 | 102.81 | 17,753 | 34.5 | | |
| 21 | 21,900 | 148.70 | 82,078 | 132.37 | 15,825 | 44.4 | | |
| 22 | 21,900 | 141.91 | 78,216 | 140.12 | 15,830 | 47.2 | | |
| 23 | 21,900 | 136.00 | 74,836 | 138.00 | 15,354 | 46.5 | | |
| 24 | 21,950 | 132.82 | 72,966 | 136.85 | 16,074 | 47.1 | 762 | 1,865 |
| 25 | 22,139 | 131.92 | 72,826 | 114.55 | 17,618 | 40.0 | 921 | 2,985 |
| 26 | 21,900 | 141.26 | 77,733 | 60.55 | 17,749 | 21.1 | | |
| 27 | 21,900 | 143.01 | 78,698 | 73.23 | 17,747 | 25.3 | | |
| 28 | 21,900 | 152.86 | 84,009 | 114.06 | 17,739 | 38.7 | | |
| 29 | 21,900 | 127.24 | 70,008 | 148.57 | 13,920 | 50.0 | | |
| 30 | 21,900 | 91.13 | 50,357 | 114.99 | 11,917 | 39.1 | | |
| 31 | 21,900 | 82.49 | 45,377 | 53.84 | 10,131 | 18.9 | | |
| 32 | 21,900 | 67.18 | 36,964 | 49.34 | 10,272 | 18.1 | | |
| 33 | 21,900 | 70.91 | 39,029 | 66.80 | 10,400 | 24.0 | | |
| 34 | 21,900 | 95.68 | 52,657 | 61.98 | 9,090 | 20.7 | | |
| 35 | 21,900 | 117.49 | 64,795 | 52.35 | 12,335 | 17.8 | | |
| 36 | 21,900 | 114.62 | 63,381 | 61.44 | 13,252 | 21.6 | | |
| 37 | 21,900 | 135.98 | 74,998 | 83.13 | 16,111 | 28.7 | | |
| 38 | 19,398 | 133.36 | 73,565 | 51.80 | 13,437 | 17.7 | | |
| **Total** | **848,879** | **5,453.42** | **3,004,718** | **5,109.85** | **1,103,278** | **1,951.6** | **110,522** | **438,066** |

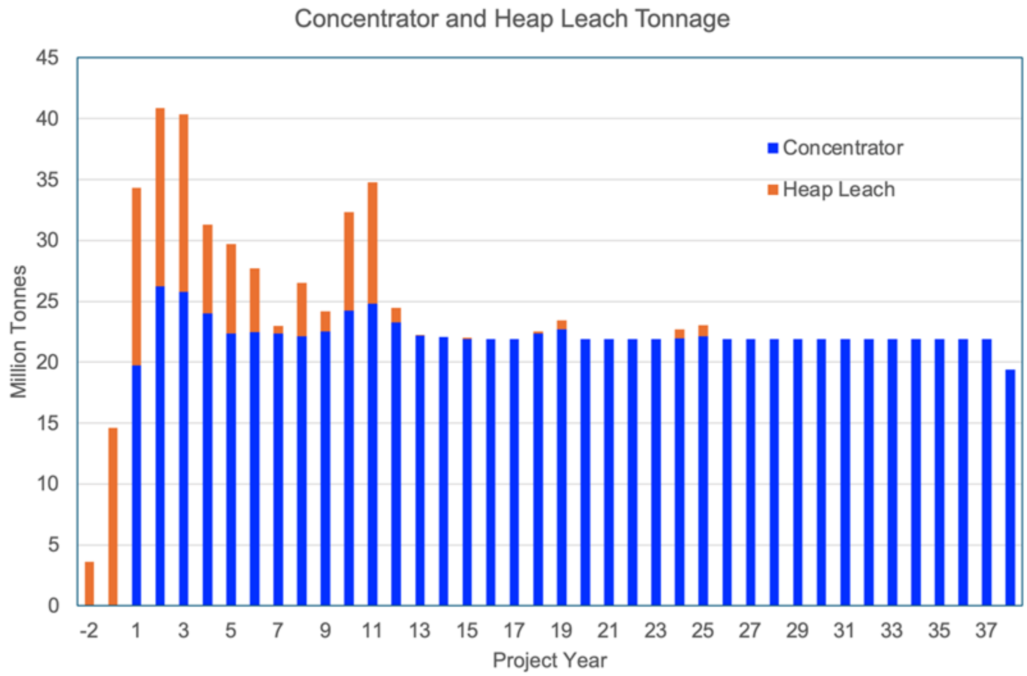



**Figure 17-3: Concentrator and Heap Leach Production by Project Year**

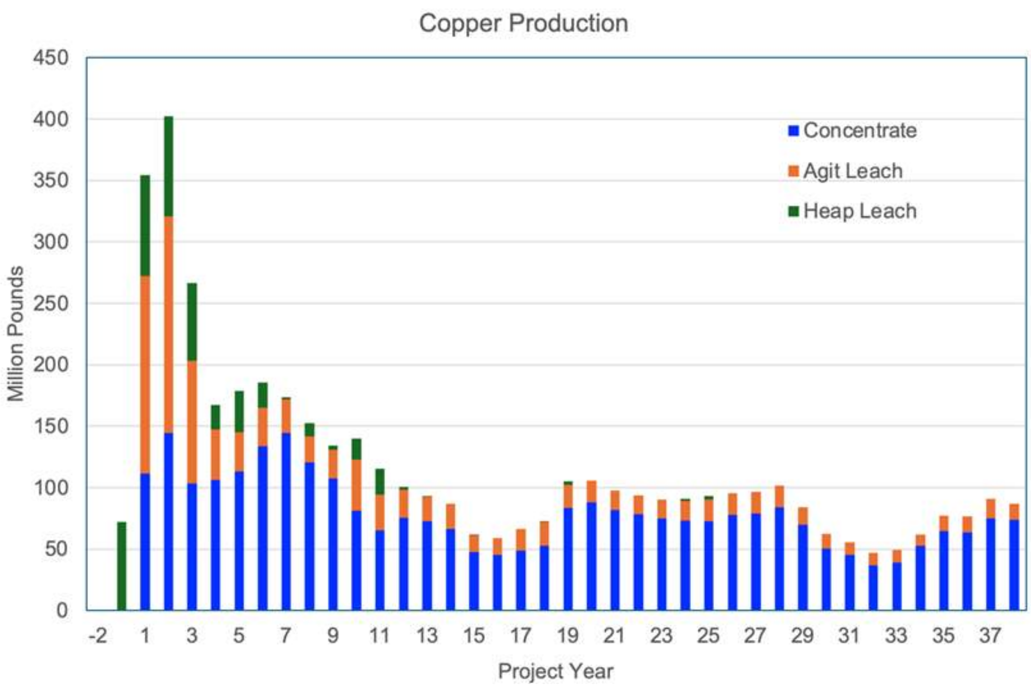



**Figure 17-4: Copper Production by Project Year**

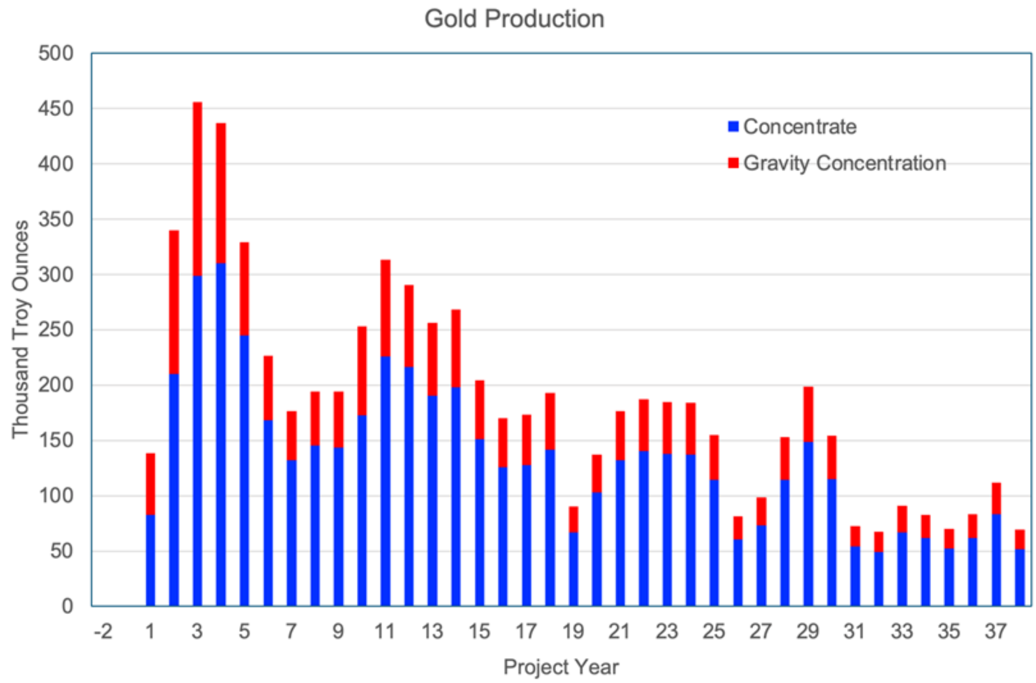



**Figure 17-5: Gold Production by Project Year**

# 18 PROJECT INFRASTRUCTURE

## 18.1 ACCESS ROAD

### 18.1.1 Main Access Road

The 13.5 km long main access road will be utilized to transport materials and people on an unpaved two-lane road, with 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 berm 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 two 710-mm HDPE tailing lines in an HDPE lined trench. The road will be used for access to the dewatering facility and inspection of the tailing lines. The road route will have to cross the Kingking River over a low-level culvert structure.

## 18.2 POWER SUPPLY

SAGC conducted studies between April and November 2024 to identify the nearest possible source of power for the Kingking Project. The studies included routing, areas affected, permitting requirements, transmission tower type and costing. Several meetings were held with National Grid Corporation of the Philippines (NGCP), who identified the Maco substation located in Maco, Davao de Oro for project tie in. A 32 km 138 kV suspended power transmission line with a 60 m right of way between the Maco Substation and the Mill Site Substation is proposed in Figure 18-1 below.

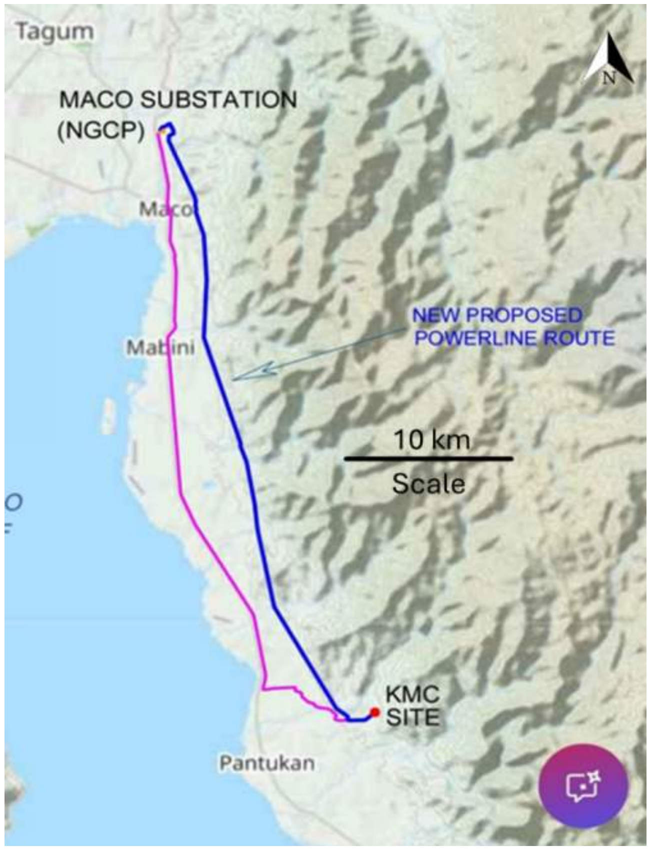



**Figure 18-1: Power Transmission Line Route (PROJECTFEAT, 2024)**

## 18.3 POWER DISTRIBUTION

At the plant site the 138 kV 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 2MW 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 so one transformer is capable of supplying power to the facility in the event of a failure of one unit. 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 to support the Kingking 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 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 conveyor system, unloading limestone from Panamax and Handymax vessels, the unloading of sulfuric acid, fuel and heavy cargo, and a dock for service/utility vessels.

The dock will comprise three (3) 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/areas and apron where the vessels will dock and where the unloading and loading of cargos 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 the cargos coming from the apron and from the 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 cargos are brought out of the port to be delivered to the consignees and where the incoming cargos 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 wash area and refueling station, equipment shed, stevedore's lounge, control room, power station and amenities.

Expected cargo traffic is outlined in Table 18-1 below.

**Table 18-1: Expected Cargo Traffic**

| Commodities | Volume per Year (tonnes) | Packaging |
|---|---|---|
| a) INCOMING | | |
| 1. Limestone | 88,000 | Dry bulk |
| 2. Sulfuric Acid | 940,000 | Liquid bulk |
| 3. Diesel | 47,600 | Liquid bulk |
| 4. Diluent | 2,332 | Liquid bulk |
| 5. HFO/LFO | 8,905 | Liquid bulk |
| 6. Lime | 52,000 | Containerize |
| 7. Concentrator Reagents | 2,695 | Containerize |
| 8. Concentrator Spare Parts | varies | Containerize |
| 9. Equipment Spare Parts | varies | Containerize |
| 10. SX-EW Reagents | 76 | Containerize |
| 11. SX-EW Spare Parts | varies | Containerize |
| 12. Explosives | 21,500 | Containerize |
| 13. Grinding Balls | 28,000 | Containerize |
| 14. Extractant | 210 | Containerize |
| b) OUTGOING | | |
| 1. Gold/Copper Concentrate | 250,000 | Dry bulk |
| 2. Cathodes | 120,000 | Containerize |

## 18.5 TAILING STORAGE FACILITY

The design methodology for the tailing storage facility, as originally described in subsections 18.5.1 through 18.5.5 of the February 25, 2013 PFS Technical Report (St. Augustine, 2013), remains valid and has been faithfully reproduced in this Study. However, the current (2025) technical economic analysis indicates that an increase in processed mill ore will generate approximately 849 Mt of tailings during the 38-year mine life. While the general design approach continues to be appropriate, the facility will require an expansion later in the mine life to accommodate the additional tonnage. Detailed modifications to the design will be incorporated in future studies.

### 18.5.1 Background

The Kingking 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 dry-stack tailing option is being proposed as the preferred alternative for the Kingking 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 percent solids (by weight) to a dewatering plant located near the dry-stack 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 and equipment was sized and quoted by the supplier (vacuum horizontal belt 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 dry-stack is 523 Mt.

The dry-stack will be constructed with dewatered tailing being placed as either structural shell tailing or general placement tailing with the prior being placed when conditions are the most favorable. Table 18-2 presents the representative volumes for each zone. The ratio of general to shell tailing is approximately 50:50 for the present configuration.

**Table 18-2: Dry-stack 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 |

There are times when placement for the shell is not possible because of the weather conditions; however general placement should still be able to be performed. The geotechnical nature of the design is based on the strength in 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 into articulated haul trucks and transported from the dewatering plant to the dry-stack. Dozers will further spread the tailing and a compactor will be utilized to compact the tailing in lifts. Test work indicates an approximate 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 dry-stack) 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 dry-stack is shown in Figure 18-2 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).

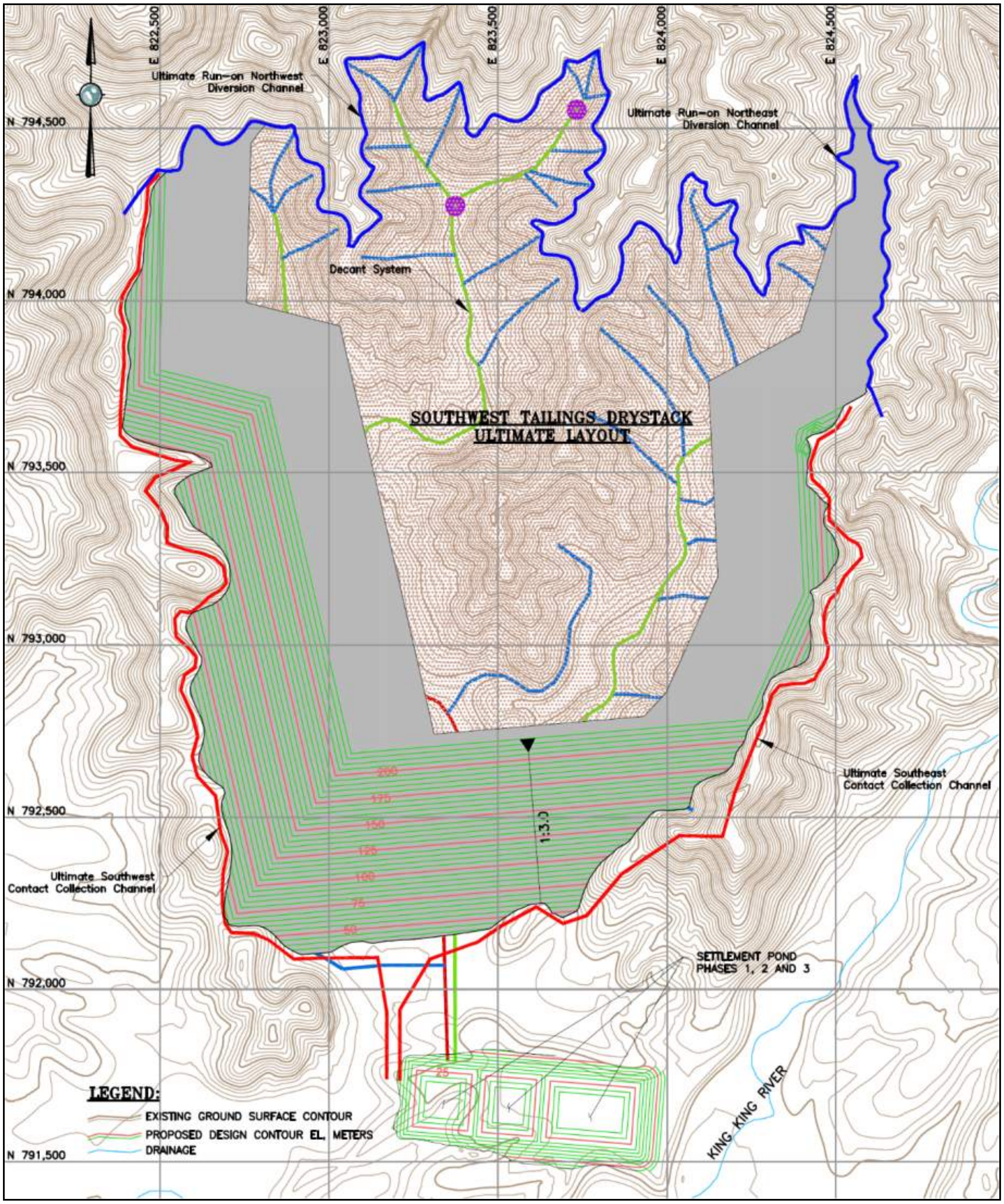



**Figure 18-2: Southwest Tailing Dry-stack Ultimate Layout (AMEC, 2013)**

The dry-stack 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 dry-stack 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 dry-stack to improve trafficability and to facilitate progressive reclamation of the exterior slopes of the dry-stack that have been constructed to their final configuration.

### 18.5.4 Surface Water Management

One of the most significant issues of concern with a dry-stack is surface erosion from unmanaged runoff. Surface runoff on the face of the dry-stack 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 Dry-stack Facility: (i) run-on channels; and (ii) contact water collection channels. Run-on channels will be used to divert non-contact stormwater around the Southwest Tailing Dry-stack 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 stormwater ponds. Refer to Figure 18-2 for 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 dry-stack 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 dry-stack will be routed to a stormwater 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, stormwater will be routed to two designated collection locations where it will enter the decant system.

The proposed stormwater collection pond will be located south of the dry-stack facility. Water from the contact collection channels and runoff from the surface of the dry-stack 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 (refer to Figure 18-3) considered the maximum cross-section through the downstream shell of the Southwest Tailing Dry-stack Facility. Section B-B (refer to Figure 18-4) 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) = 0.6 g (AMEC Geomatrix, 2011); and
- Maximum Design Earthquake (MDE) PGA = 1.01 g (AMEC Geomatrix, 2011).

#### 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 dry-stack is assumed to be comprised of two main material types:

- Foundation – Foundation materials are the original ground the dry-stack will rest on after it has been stripped and prepared for material placement. It is assumed that bedrock is relatively shallow in the proposed dry-stack area and that the structural shell of the dry-stack 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 18-3.

**Table 18-3: Material Parameters used for Stability Analysis**

| Zone | Bulk Density | Strength | |
|---|---|---|---|
| | | Static | Psuedo-static |
| Foundation Bedrock | 19.5 kN/m$^3$ | $\phi'$ = 40°,<br>c' = 25 kN/m$^2$ | $\phi'$ = 40°,<br>c' = 25 kN/m$^2$ |
| Shell (Compacted Tailing) | 16.5 kN/m$^3$ | $\phi'$ = 32°<br>c' = 0 kN/m$^2$ | $\phi'$ = 32°<br>c' = 0 kN/m$^2$ |
| General Placement (Tailing) | 15.7 kN/m$^3$ | $\phi'$ = 31°<br>c' = 0 kN/m$^2$ | $\phi'$ = 26.6°*<br>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 18-3 and Figure 18-4 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.

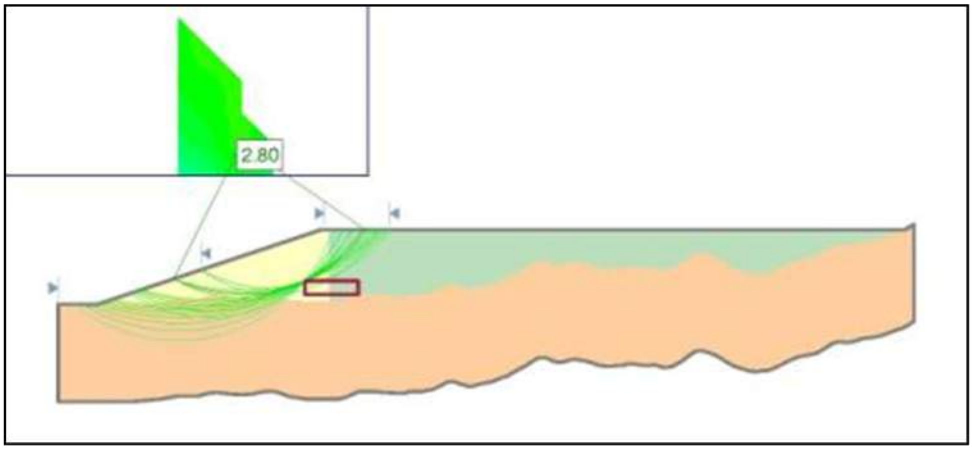



Figure 18-3: Cross Section A-A - Stability Evaluation

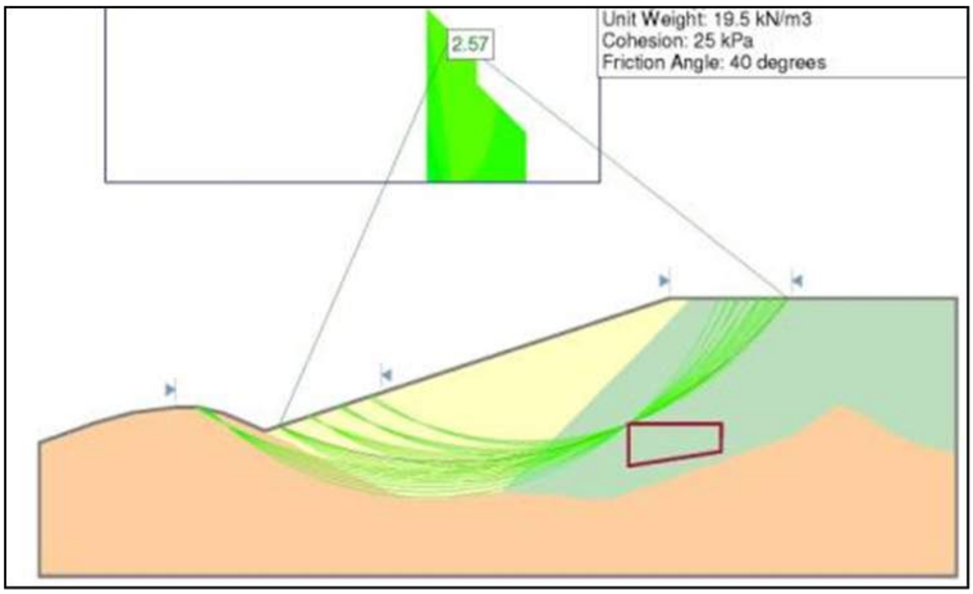



Figure 18-4: Cross Section B-B - Stability Evaluation

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 dry-stack. The results of the stability evaluation are presented in Table 18-4.

Table 18-4: Stability Analysis 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 dry-stack.
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 dry-stack 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

The original design approach for the valueless rock storage facility, as detailed in subsections 18.6.1 through 18.6.3 of the February 25, 2013 PFS Technical Report (St. Augustine, 2013), remains applicable and has been faithfully reproduced in this Technical Report. In the current (2025) technical economic analysis, the mine plan now requires storage for approximately 839 Mt of valueless rock. Although the existing design is fundamentally sound, an expansion of storage capacity will be necessary later in the mine life to manage the increased tonnage. Future studies will revisit subsections 18.6.1 through 18.6.3 to incorporate the required design enhancements.

### 18.6.1 VRMA

Based on the current mine production schedule, the valueless rock production for the Kingking 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 tonnes 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% (Knight Piésold, 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 includes 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 sufficient 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 18-5 with associated diversion channels and ponds.

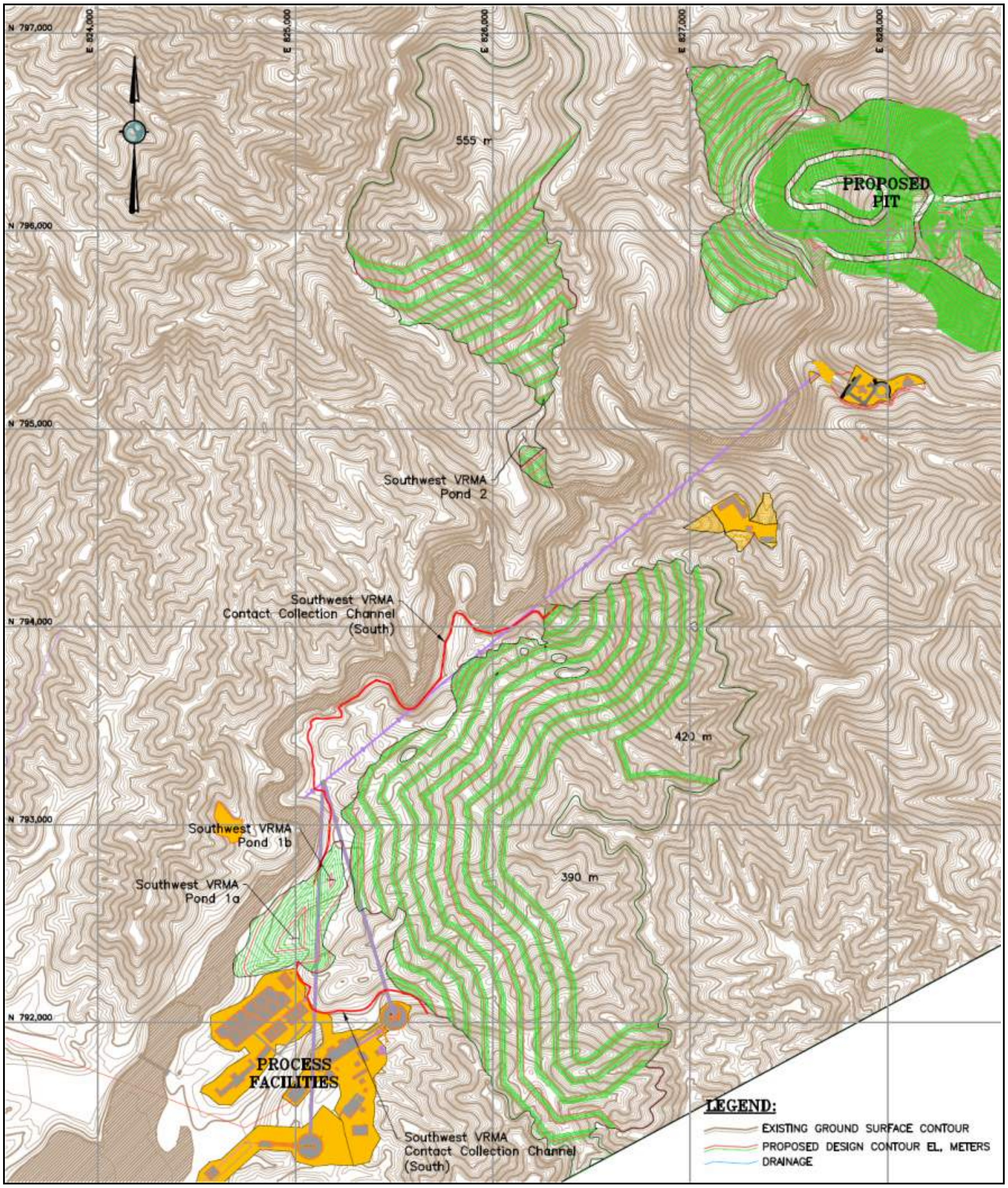



**Figure 18-5: Proposed VRMA Ultimate Layout (M3, 2013)**

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 Kingking 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) = 0.60 g; and
- Maximum Design Earthquake (MCE) PGA = 1.01 g.

#### 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 comprised 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. We anticipate 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 18-6 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).

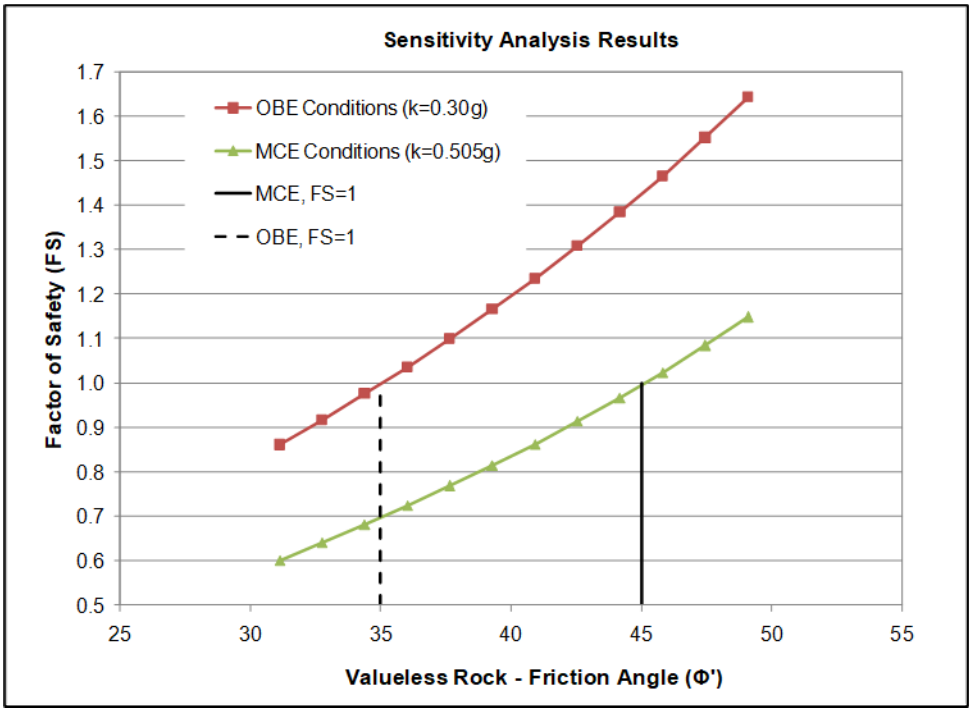



**Figure 18-6: Sensitivity of Valueless Rock Material Friction Angle on Factor of Safety**

Figure 18-7 illustrates the evaluated cross-section and the circular failure surface with the minimum factor of safety evaluated for the Kingking Project. Based on the current configuration with 3H:1V downstream slopes, veneer failure surfaces through the downstream shell appear to control stability.

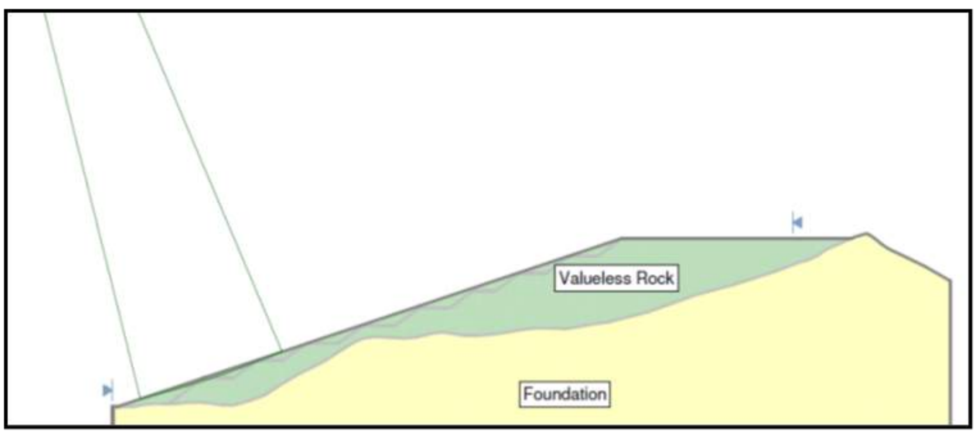



**Figure 18-7: Assumed Cross Section for Stability Evaluation**

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 Kingking Project. The process water demand is estimated to be approximately 600,000 m$^3$/month, or 20,000 m$^3$/d.

The wellfield 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 Kingking 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 Kingking 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, following by the drilling of exploration boreholes (completed as piezometers) and then 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 for dilution purposes of 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 Year 4-5, 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 Water/ Pit 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

Runoff and collected water from the heap leach area(s) will require treatment. A separate treatment package/plant will be built to serve the heap leach area only and will discharge treated water to the Kingking River. Initial assumption is that the discharge pipe from Heap Leach to Kingking River will be 1.9 km maximum.

### 18.7.5 Potable Water

Water from the pit depressurization system will be used for potable water at the mine and process facilities. 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.

## 18.8 WATER TREATMENT

The preliminary level evaluation of water treatment requirements for the Kingking Project has been developed 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 and water quality data was provided by AATA for the six sources including the Dry Stack 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 dated February 13, 2012 provided the basis of ten percent oxide ore and ninety percent sulfide ore in the dry stack, and associated TSF runoff water quality (AATA, 2012a).
- 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, 2012b).
- 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, 2012a). 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, 2012b).

The overall water treatment/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 percent permeate recovery can be achieved. The secondary waste or brine stream is 20 percent 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 and/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.

Pit groundwater is the likely source of potable water. 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. Excess potable water treatment capacity could be designed to supplement the water supply of the nearby community off-site.

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 4,000 people.

# 19 MARKET STUDIES AND CONTRACTS

No market studies were performed, and no sales contracts are in place. The commodities involved in this project are commonly traded on the open market.

# 20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This section summarizes the current understanding of the Kingking Project's environmental and social context and the corporate, regulatory, and international framework guiding its development. SAGC has prepared environmental and social reports and programs to comply with municipal, provincial and national regulations requirements as well as international standards. Permitting sequences, as well as regular governmental inspections have occurred over the past number of years since the Kingking Project was initiated. The Kingking Project has engaged local communities and stakeholders and has achieved required permits. The following provides details on the permitting, physical, chemical, biological, and social aspects of the Kingking Project.

## 20.1 PERMITS AND APPROVALS

The Department of Environment and Natural Resources (DENR) is the primary government agency responsible for the conservation, management, development, and proper use of the country's environment and natural resources, including mineral resources, as well as the licensing and regulation of all natural resources. The Kingking Project's current status on permitting for environmental and social considerations is based on several key documents and activities.

The Kingking Project in Mindanao, Philippines, developed by St. Augustine Gold & Copper, Ltd. (SAGC) has developed key environmental and social permit support documents, and has secured key permits and approvals essential for its development:

1. **Mineral Production Sharing Agreement (MPSA):** The Kingking Project operates under MPSA No. 009-92-XI Amended II, initially granted in 1992 and renewed for an additional 25 years in 2016, authorizing the extraction and sale of copper, gold, and associated minerals within a 2,976-hectare area.
2. **Environmental Compliance Certificate (ECC):** Issued in February 2015 by the Environmental Management Bureau (EMB) of the Philippines, the ECC confirms that the project's Environmental Impact Statement (EIS) complies with national environmental regulations, allowing the Kingking Project to proceed with development. Some minor updates to the EIS are anticipated given changes in project layout, configuration, and change of power source (Nationwide grid power line versus onsite coal-fired power plant).
3. **Declaration of Mining Project Feasibility (DMPF):** Approved by the Mines and Geosciences Bureau (MGB) on May 16, 2016, the DMPF is the final major permit required, enabling the Kingking Project to commence development, construction, and operation phases.
4. **Certificate of Precondition - Free, Prior, & Informed Consent (FPIC):** SAGC obtained its Certification Precondition from the Philippine National Commission on Indigenous Peoples (NCIP) in compliance with the Indigenous Peoples Rights Act of 1997 in February 2016 (FPIC guidelines) based upon a Memorandum of Understanding with the Mansaka Indigenous Cultural Communities/Indigenous Peoples.
5. **Social Consultation and Community Outreach:** Social Impact Assessment (SIA): SAGC has completed a detailed independent Social Impact Assessment (SIA-Asian Sustainability and Quality Bureau, 2025) that provides fine details of the socio-economic and cultural characteristics throughout the project area. The company has engaged the broad range of communities, organizations, indigenous representatives, and other stakeholders who have been identified to be potentially affected by the Kingking Project and its operations.
6. **ISO 14001:2015: Certification of Environmental Management System:** The Kingking Project secured ISO 14001: 2015 Certification for Environmental Management System (EMS) since 2018 and has been certified for 2023-2026 under Preliminary Support Activities.
7. **Environmental Protection and Management Plan (EPMP):** The Environmental Protection and Management Plan (EPMP) of the Kingking Project is composed of the Environmental Protection and Enhancement Program (EPEP) and the Final Mine Rehabilitation and Decommissioning Plan (FMR/DP) which follow the commitments of the Environmental Management Plan (EMP) of the Environmental Impact Statement (EIS) and the conditions as stated in the approved Environmental Compliance Certificate (ECC).

8. **Mineral Processing Permit:** As the Processing Plant is located outside the MPSA area, there is a need to secure a Mineral Processing Permit (MPP) under the Philippine Mining Act. In 2016, the MGB issued Mineral Processing Permit (MPP) No. 16-2016-XI in favor of NADECOR. However, the imposition of the open-pit mining ban that same year, coupled with the impact of the COVID-19 pandemic from 2020 to 2022, constrained the operational and economic viability of the Kingking Project and hindering compliance with MPP requirements. With the lifting of the open-pit ban in late 2021, the easing of pandemic restrictions, and improved metal prices, NADECOR is now in a position to move forward and is currently completing the requirements to renew the MPP and advance the Kingking Project.
9. **Permit and Inspection Tracking System:** SAGC operates a detailed permit inspection and tracking system as part of its normal management program which keeps tabs on all permits and inspections, agency responsibilities and schedules, background documentation, and active dates for key compliance and response activities.

## 20.2 PHYSICAL ENVIRONMENT

The Kingking Project area features steep and rugged topography, with elevations ranging from 250 to 950 meters above mean sea level (amsl). To the west of the proposed mine site, the landscape transitions from the mountains to rolling hills, eventually leveling into coastal plains at the Davao Gulf. The region experiences a tropical climate without distinct wet or dry seasons, and daytime temperatures range from 18ºC to 35ºC. Annual precipitation varies from 2,000 to 3,200 mm in the mountains and 1,800 to 2,000 mm per along the coastal plain (SAGC, 2012).

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 m long with an average slope of 58 m per km. 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. Background baseline surface water hydrology of the Kingking River was monitored using advanced digital flow monitoring systems for extended periods at strategic locations.

The Philippine Fault, a major structural feature extending the length of the archipelago, influences the structural layout, contributing to the emplacement of intrusive rocks and mineralization in the Kingking deposit (Kilborn International Inc., 1997). Copper and gold mineralization primarily occurs at or near the apex of the diorite intrusive complex and extends into the surrounding wall rocks (IMC, 2010).

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. A comprehensive noise management program shall be a component of the Environmental and Social Management and Monitoring Program.

## 20.3 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. Strategic locations were sampled periodically using modern sampling and laboratory methodologies using modern advanced laboratory techniques.

The soils are slightly acidic (pH 5.4 - 6.1) and most samples showed high aluminum and iron concentrations (comparable to bedrock compositions). Water samples from Kingking River showed relatively high total suspended solids (TSS), copper, mercury, cyanide and total coliform concentrations, above applicable water quality criteria. Ground water quality is generally good with the mountains, while unsanitary sewage disposal has directly impacted portions of the alluvial aquifer in the low land areas.

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.

Sediment chemistry data indicated elevated concentrations of metals such as aluminum, chromium, copper, iron, and manganese. High mercury concentrations were documented at the mouth of the Kingking River, and out into the bay within the mixing zone. Mercury is utilized for amalgamation of gold in small processing centers set up and operated by small-scale mining operators. Implementation and operation of the Kingking Project shall reduce mercury contamination significantly by eliminating the small miner gold processing activities on and near the property.

Surface water quality is of high interest to the Kingking Project. Baseline monitoring indicated high levels of coliform bacteria in exceedance of Philippine health standards. Secondary wastewater quality treatment and improved watershed management are anticipated to have a positive effect on reducing coliforms and sediments.

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). Surface water analysis revealed that samples from the Kingking River had the highest copper and mercury levels, with some sites exceeding Philippine and U.S. EPA standards.

## 20.4 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.

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.

Six general types of vegetation were recognized: open-canopy mid-mountain forest, brush land, wooded grass land, 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.

Mitigation measures are being developed to protect environmentally sensitive species based upon the results of the Environmental Impact Statement and will be implemented prior to construction and operation.

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 Kingking 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.

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, as well as species of commercial value.

### 20.5 SOCIAL ENVIRONMENT

Davao de Oro 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 2020 Census, Pantukan has a population of 90,786 in 21,663 households. For comparison, during the 2000 Census, Pantukan had a population of 61,801 people in 13,311 households, while during 2007 census, the population was 69,656 people. Pantukan is divided into 13 barangays. Barangays Bongbong, Kingking, 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 Davao de Oro 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.

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.

According to the National Statistics Office of the Philippines, the 2020 populations of the five barangays directly impacted by the Kingking Project were:

- Barangay Bongbong: 4,579 people
- Barangay Kingking: 29,317 people;
- Barangay Magnaga: 10,766 people;
- Barangay Napnapan: 12,529 people; and
- Barangay Tagdangua: 4,765 people.

Of the barangays surveyed, about three-quarters of the population are of Visayan origin. Indigenous people account for seven (7) to thirty-two (32) 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. A new census may be conducted in 2025, and, if so, data will be updated.

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, six percent 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 Kingking reported wages above PhP 50,000. (NOTE: the current exchange rate is 1 US$ to 58.24 Philippine Pesos.)

Electric lighting is used by more than two-thirds of households. The remainder of the households generally use gas lamps for lighting. Wood and charcoal are used as cooking fuel by more than three-quarters of households. Most households use streams, springs, or wells for their water supply. Some households use personal or communal faucets for their water supply. There are reported brownouts that occur in the region with some regularity. These events are expected to be reduced or eliminated due to the new power line being advanced by the Philippine National Grid into the project area. As mentioned previously, the coal fired power plant which was planned for the Kingking Project was eliminated due to the advancement of a new electrical power line as part of the National Grid. Detailed engineering and environmental impact assessment were provided by the project development contractor.

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. There is considerable sea transport to and from Davao City across the Davao Bay.

A comprehensive Social Impact Assessment (SIA) was developed under contract to SAGC by the Asian Sustainability and Quality Bureau, Davao City, Mindanao, Philippines.

## 20.6 INTERNATIONAL STANDARDS

SAGC has committed to following various applicable international standards and guidelines including the Performance Standards of the International Finance Corporation, World Bank Group (IFC, Washington, D.C., USA) and World Bank Group Environmental, Health, and Safety (EHS) guidelines (WBG, Washington, D.C., USA). The Kingking Project also has acknowledged including consideration of the highly complementary Equator Principles (www.Equator-Principles.com). The Mining Principles of the International Council of Mining and Metals (ICMM, www.icmm.com) are recognized by SAGC for ensuring best practices of environmental and social sustainability throughout all mining phases. SAGC also commits to compliance and certification with the International Cyanide Management Code (https://cyanidecode.org).

These standards are widely respected worldwide and help ensure that mining operations maintain high levels of environmental protection, social responsibility, and safety. This commitment enhances trust with government agencies, the public, and other stakeholders and aligns with global best practices in sustainability.

# 21 CAPITAL AND OPERATING COSTS

M3 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 and Technology Corporation (M3)* – Tucson, AZ – Process plant and overall project site layout infrastructure.
- *Independent Mining Consultants (IMC)* – Tucson, AZ – Mine and contract mining.
- *AMEC Earth & Environmental, Inc. (AMEC)*** – Denver, CO – Dry-stack tailing facility, VRMA and pit dewatering.
- *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 ($) unless noted otherwise.

** Indicates that cost structure from 2013 PFS Technical Report for Kingking were adjusted to reflect revised tonnage and Q4-2024 pricing. Design guidelines were maintained per the 2013 PFS Technical Report.

## 21.1 CAPITAL COST SUMMARY

The detailed initial capital cost estimate is summarized by area in Table 21-1 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 and civil 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 Q4 2024 US dollars. No escalation has been assumed.

**Table 21-1: Summary of Capital Costs**

| Area | Description | ($ Millions) |
|---|---|---|
| Process Plant and General Infrastructure | General Site, primary crushing, aerial conveyors, heap leach, grinding, flotation, SX-EW, Agitated leach, tailing dewatering, water systems, power transmission line, on site power distribution, mine support infrastructure, ancillary facilities, EPCM, freight, import duties. | $1,638.25 |
| Mine | Contract mining development costs prior to the start of production. | $131.92 |
| Port Facility | Dock Facility, Concentrate loading, Coastal Complex | $50.00 |
| Owners Costs | Land Acquisition, Construction/ Operating Camps, Environmental Permits, Initial Fills, Owner's Project Management, Security, Early Staffing, Community relations. | $163.48 |
| Contingency | Contingency on all parts of the Kingking Project | $390.13 |
| Escalation | Not included in this estimate | $0 |
| Total Before VAT | | $2,373.78 |
| Value Added Tax (VAT) | | $189.62 |

### 21.1.1 Mine Capital Basis

The mine capital cost estimate assumes 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 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 21-2 shows summary of Mine Capital for the life of mine for contract mining and includes the following:

- Mobilization – Major equipment buildup is during Years -2, -1, 1, and 2. The purchase price for this equipment is estimated at $193.0 million for the owner operation case. Contractor mobilization was estimated at 8% of this amount, or $15.44 million over these four years. This is $7.13 for initial capital and $8.31 million for Years 1 and 2 sustaining capital. The 8% mobilization fee is an allowance of about 5% of the equipment new price for equipment transportation and an additional 3% for logistics, hiring of personnel, procuring supplies/equipment, etc.
- Demobilization – Due to the 40-year life of the Kingking Project, the demobilization cost at the end of the project is not material. It is likely the owner will purchase the equipment prior to that time.
- Owner Equipment - An allowance for owner equipment is estimated at 5% of the mine major equipment purchases for the owner operation case for Years -2, -1, 1, and 2. This amounts to $9.65 million for these years.
- Mine Development – $120.3 million is the estimated operating cost to mine 45.1 million tonnes of material during the preproduction period.

**Table 21-2: Summary of Mine Capital – Contract Mining**

| Mine Capital Costs - Contract Mining | | Units | Initial Capital | | Initial Capital | Sustaining Capital | Total Capital |
|---|---|---|---|---|---|---|---|
| | | | Year -2 | Year -1 | | | |
| Contractor Mobilization (% of Major) | 8.0% | ($x1000) | 2,802 | 4,327 | 7,128 | 8,312 | 15,440 |
| Owner Equipment (% of Major) | 5.0% | ($x1000) | 1,751 | 2,704 | 4,455 | 5,195 | 9,650 |
| Mine Development | | ($x1000) | 36,066 | 84,269 | 120,335 | 0 | 120,335 |
| **Total** | | **($x1000)** | **40,619** | **91,299** | **131,919** | **13,507** | **145,426** |

21.1.1.1 Mine Capital – Owner Operation

An owner operation case was used to develop the contract mining estimates presented in the previous section. Most of the major equipment prices are based on quotations from Monark Caterpillar in the Philippines. The quotes for large shovels were provided by Empire Caterpillar in Arizona. Table 21-3 summarizes the mine capital costs by category for initial and sustaining capital periods.

**Table 21-3: Mining Capital – Mine Equipment and Mine Development (US$1000)**

| Category | | Initial Capital | | Initial Capital | Sustaining Capital | Total Capital |
|---|---|---|---|---|---|---|
| | | Year -2 | Year -1 | | | |
| Major Equipment | | 35,021 | 54,082 | 89,103 | 287,183 | 376,286 |
| Support Equipment @ | 15.00% | 5,253 | 8,112 | 13,365 | 43,077 | 56,443 |
| **Equipment Subtotal** | | **40,274** | **62,194** | **102,468** | **330,261** | **432,729** |
| Equipment Contingency @ | 10.0% | 4,027 | 6,219 | 10,247 | 0 | 10,247 |
| Mine Development | | 30,877 | 72,012 | 102,889 | 0 | 102,889 |
| **Total** | | **75,178** | **140,426** | **215,604** | **330,261** | **545,864** |
| Exclusions: Mine shop and warehouse, fuel and lubricant storage, explosives storage, and offices. | | | | | | |

Note that the mine development cost is higher for the contract mining case due to contractor profit and overhead and equipment depreciation charges.

The mine capital costs include the following items:

- Mine major equipment
- Mine support equipment and initial spare parts
- Mine preproduction development expense

The estimated cost of the following facilities is included in the infrastructure capital estimate:

- The mine shop and warehouse
- Fuel and lubricant storage facilities
- Storage facilities for blasting agents and high explosives
- Office facilities

Support equipment and initial spare parts is estimated at 15% of major equipment by year. Support equipment includes fuel and lubricant trucks, mechanic and welding trucks, tire handlers, low boys and tractors to transport track equipment, cranes, surveying and engineering equipment, etc. This list is not exhaustive.

The cost estimate also includes a 10% contingency for the initial capital equipment purchases.

### 21.1.2 Infrastructure and Process Plant

Estimates were based on these general arrangement civil and architectural drawings, a detailed equipment list, process design criteria and flowsheets. Approximately half of the major equipment was quoted.

### 21.1.3 Power Transmission Line

Local Philippine electrical contractors were contacted for estimation of the power transmission line, as well as right-of-way cost estimation. These costs are included in the capital cost estimate.

### 21.1.4 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.

### 21.1.5 Tailing and VRMA Facilities

The cost estimate includes site preparation, underdrains, decants, diversion channels and ponds. Tailing stacking will be performed by an earthworks contractor as an operating cost and thus mobile equipment is not included in this cost estimate.

### 21.1.6 Pit Diversions

The proposed system consists primarily of run-on channels used to divert non-contact stormwater 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

Owner's costs include land acquisition, permitting, first fills, early staffing, legal costs, construction/operations camp, owner's team, and site security.

## 21.2 OPERATING COST SUMMARY

Life of mine (38 years) consolidated net cash costs per pound of payable copper equivalent are $2.46. Life of mine payable copper equivalent is 7,863.1 million pounds. The consolidated net cash costs for the first 10 years of full production are $1.73 per pound of payable copper equivalent. Payable copper equivalent in the first 10 years of full production is 3,517.8 million pounds. Life of mine consolidated net cash costs per tonne of ore processed (heap leach and mill ores combined) are $20.18. Life of mine processed ore is 959.4 million tonnes. Consolidated net cash costs per tonne of ore processed in the first 10 years of full production are $18.49. Ore processed in the first 10 years of full production is 328.5 million tonnes.

**Table 21-4: Production Cost**

| Cost Component | Units | Time Period | | |
|---|---|---|---|---|
| | | Years 1-5 | Years 1-10 | LOM |
| Payable Pounds of Copper Equivalent | millions | 2,250.2 | 3,517.8 | 7,863.1 |
| Mining | $/lb Cu | $0.30 | $0.38 | $0.53 |
| Processing | $/lb Cu | $0.68 | $0.82 | $1.27 |
| **Operating Costs** | **$/lb Cu** | **$0.98** | **$1.20** | **$1.80** |
| G&A | $/lb Cu | $0.11 | $0.11 | $0.14 |
| Reclamation & Closure | $/lb Cu | $0.00 | $0.00 | $0.01 |
| **Cash Costs at Mine** | **$/lb Cu** | **$1.09** | **$1.31** | **$1.95** |
| Government Fees | $/lb Cu | $0.30 | $0.31 | $0.39 |
| **Total Cash Costs at Mine** | **$/lb Cu** | **$1.38** | **$1.62** | **$2.34** |
| Shipping, Smelting & Refining | $/lb Cu | $0.09 | $0.11 | $0.12 |
| **Total Consolidated Net Cash Costs** | **$/lb Cu** | **$1.47** | **$1.73** | **$2.46** |

## Table 21-5: Summary of Operating Cost by Year ($000)

| Year | Mining (Incl pit dewatering) | Concentrator | Gravity Gold Circuit | Agitated Tailings Leach | SXEW - Tailings Leach | Tailings Filtration and Stacking | Heap Leach & SXEW | General Administration | Laboratory | Port | Custom Duties | Total |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| -2 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $10,738 | $0 | $0 | $0 | **$10,738** |
| -1 | $0 | $0 | $0 | $0 | $0 | $0 | $51,342 | $23,865 | $469 | $2,036 | $1,469 | **$79,181** |
| 1 | $114,783 | $97,592 | $507 | $49,252 | $20,917 | $70,987 | $42,454 | $36,836 | $760 | $2,071 | $2,937 | **$439,097** |
| 2 | $133,756 | $111,854 | $514 | $60,228 | $22,741 | $92,030 | $42,368 | $38,978 | $760 | $2,071 | $2,937 | **$508,237** |
| 3 | $150,581 | $110,662 | $513 | $59,113 | $14,251 | $90,443 | $40,744 | $34,721 | $760 | $2,071 | $2,937 | **$506,799** |
| 4 | $135,640 | $109,372 | $511 | $56,986 | $8,012 | $84,891 | $22,166 | $31,761 | $760 | $2,071 | $2,937 | **$455,108** |
| 5 | $137,285 | $108,352 | $509 | $54,601 | $5,986 | $79,735 | $24,755 | $30,240 | $760 | $2,071 | $2,937 | **$447,232** |
| 6 | $139,135 | $109,281 | $509 | $54,632 | $6,699 | $79,973 | $18,760 | $27,304 | $760 | $2,071 | $2,937 | **$442,063** |
| 7 | $132,736 | $108,993 | $509 | $54,518 | $8,083 | $79,593 | $6,216 | $25,970 | $760 | $2,071 | $2,937 | **$422,388** |
| 8 | $133,279 | $108,844 | $509 | $53,363 | $5,782 | $79,022 | $15,690 | $25,022 | $760 | $2,071 | $2,937 | **$427,280** |
| 9 | $132,040 | $110,491 | $510 | $55,158 | $7,211 | $80,291 | $8,815 | $23,902 | $760 | $2,071 | $2,937 | **$424,186** |
| 10 | $129,557 | $112,487 | $511 | $56,714 | $8,208 | $85,684 | $23,141 | $24,008 | $760 | $2,071 | $2,937 | **$446,079** |
| 11 | $119,120 | $113,109 | $512 | $57,561 | $6,239 | $87,509 | $27,558 | $23,407 | $760 | $2,071 | $2,937 | **$440,784** |
| 12 | $141,778 | $110,930 | $510 | $56,542 | $7,325 | $82,591 | $7,673 | $22,323 | $760 | $2,071 | $2,937 | **$435,442** |
| 13 | $160,122 | $109,632 | $509 | $54,960 | $7,607 | $79,212 | $4,640 | $21,091 | $677 | $2,071 | $2,937 | **$443,458** |
| 14 | $149,088 | $109,394 | $509 | $53,447 | $7,550 | $78,784 | $0 | $20,563 | $677 | $2,071 | $2,937 | **$425,021** |
| 15 | $150,100 | $109,355 | $509 | $53,259 | $6,684 | $78,228 | $4,884 | $19,023 | $677 | $2,071 | $2,937 | **$427,727** |
| 16 | $166,540 | $109,388 | $509 | $53,268 | $6,771 | $78,228 | $0 | $18,571 | $677 | $2,071 | $2,937 | **$438,961** |
| 17 | $161,437 | $109,703 | $509 | $54,733 | $7,386 | $78,228 | $0 | $18,712 | $677 | $2,071 | $2,937 | **$436,395** |
| 18 | $158,146 | $110,191 | $509 | $54,031 | $7,142 | $79,688 | $5,195 | $19,060 | $677 | $2,071 | $2,937 | **$439,647** |
| 19 | $131,090 | $109,825 | $510 | $54,476 | $6,677 | $80,741 | $6,663 | $18,770 | $677 | $2,071 | $2,937 | **$414,437** |
| 20 | $121,417 | $109,538 | $509 | $54,555 | $7,364 | $78,228 | $0 | $19,427 | $677 | $2,071 | $2,937 | **$396,724** |
| 21 | $109,539 | $108,996 | $509 | $53,107 | $7,010 | $89,178 | $0 | $19,946 | $677 | $2,071 | $2,937 | **$393,972** |
| 22 | $114,997 | $108,996 | $509 | $54,784 | $7,178 | $89,178 | $0 | $20,008 | $677 | $2,071 | $2,937 | **$401,336** |
| 23 | $149,480 | $112,180 | $509 | $53,138 | $6,961 | $83,976 | $0 | $19,412 | $677 | $2,072 | $2,937 | **$431,343** |
| 24 | $154,891 | $107,938 | $509 | $53,146 | $5,587 | $89,005 | $6,395 | $19,657 | $677 | $2,075 | $2,937 | **$442,817** |
| 25 | $155,926 | $110,074 | $509 | $53,512 | $5,576 | $89,735 | $6,925 | $19,590 | $677 | $2,071 | $2,937 | **$447,534** |
| 26 | $117,395 | $109,088 | $509 | $53,012 | $1,925 | $88,811 | $0 | $19,168 | $677 | $2,071 | $2,937 | **$395,594** |
| 27 | $66,479 | $109,088 | $509 | $53,008 | $1,925 | $88,811 | $0 | $19,640 | $677 | $2,071 | $2,937 | **$345,146** |
| 28 | $70,270 | $109,088 | $509 | $52,983 | $1,924 | $88,811 | $0 | $20,331 | $677 | $2,071 | $2,937 | **$349,602** |
| 29 | $68,303 | $109,088 | $509 | $53,046 | $1,510 | $88,811 | $0 | $20,324 | $677 | $2,071 | $2,937 | **$347,277** |
| 30 | $82,707 | $109,088 | $509 | $53,394 | $1,293 | $88,811 | $0 | $19,204 | $677 | $2,071 | $2,937 | **$360,691** |
| 31 | $44,623 | $109,089 | $509 | $53,483 | $1,099 | $88,812 | $0 | $18,648 | $677 | $2,071 | $2,937 | **$321,949** |
| 32 | $32,674 | $109,089 | $509 | $53,193 | $1,114 | $88,812 | $0 | $18,608 | $677 | $2,071 | $2,937 | **$309,685** |
| 33 | $31,440 | $109,087 | $509 | $53,183 | $1,128 | $88,810 | $0 | $18,563 | $677 | $2,071 | $2,937 | **$308,406** |
| 34 | $30,558 | $109,089 | $509 | $53,123 | $986 | $88,812 | $0 | $18,651 | $677 | $2,071 | $2,937 | **$307,413** |
| 35 | $29,380 | $109,090 | $509 | $53,071 | $1,338 | $88,813 | $0 | $18,791 | $677 | $2,071 | $2,937 | **$306,678** |
| 36 | $28,247 | $109,087 | $509 | $53,076 | $1,437 | $88,810 | $0 | $18,874 | $677 | $2,071 | $2,937 | **$305,726** |
| 37 | $26,859 | $109,088 | $509 | $54,299 | $1,747 | $88,811 | $0 | $19,408 | $677 | $2,030 | $2,937 | **$306,366** |
| 38 | $25,692 | $83,529 | $506 | $40,535 | $1,457 | $75,616 | $0 | $19,084 | $280 | $1,711 | $2,937 | **$251,347** |
| **Total** | **$4,137,091** | **$4,129,768** | **$19,356** | **$2,048,488** | **$229,830** | **$3,206,514** | **$366,382** | **$882,198** | **$26,794** | **$80,355** | **$113,092** | **$15,239,868** |

### 21.2.1 Mine Operating Costs

Table 21-6 summarizes the mining costs for the commercial production period by cost center. The total cost for the contract mining case is $4.11 billion which amounts to $2.11 per total tonne, $4.84 per mill tonne, and $4.37 per mill plus leach tonne. The costs for the owner operation case are also shown.

**Table 21-6: Mine Operating Cost by Cost Center – Commercial Production**

| Cost Center | Total Cost (US$ x 1000) | Cost Per Total Tonne (US$/t) | Cost Per Mill Tonne (US$/t) | Cost Per Mill/ Leach Tonne (US$/t) | % of Total |
|---|---|---|---|---|---|
| Drilling | 173,234 | 0.089 | 0.204 | 0.184 | 4.2% |
| Blasting | 662,174 | 0.340 | 0.780 | 0.704 | 16.1% |
| Loading | 341,069 | 0.175 | 0.402 | 0.362 | 8.3% |
| Hauling | 1,777,271 | 0.912 | 2.094 | 1.888 | 43.2% |
| Roads and Dumps | 324,401 | 0.166 | 0.382 | 0.345 | 7.9% |
| Mine Services | 165,823 | 0.085 | 0.195 | 0.176 | 4.0% |
| Mine Administration | 47,359 | 0.024 | 0.056 | 0.050 | 1.2% |
| **Sub-total Owner Operation Cost** | **3,491,331** | **1.792** | **4.113** | **3.709** | **84.9%** |
| Contractor Depreciation | 400,259 | 0.205 | 0.472 | 0.425 | 9.7% |
| Contractor Overhead/Profit | 218,527 | 0.112 | 0.257 | 0.232 | 5.3% |
| **Total Cost - Contract Mining** | **4,110,116** | **2.109** | **4.842** | **4.367** | **100.0%** |
| Total, Mill, and Mill/Leach Ktonnes | | 1,948,481 | 848,880 | 941,202 | |

Contractor overhead and profit is 15% of the contractor's direct cost, where the direct cost is as follows:

| | |
|---|---|
| Owner Operation Cost | $3,491.3 million |
| Less Fuel (Provided by Owner) | $1,355.4 million |
| Less Blasting (Separate Contract) | $661.4 million |
| Less Technical Services Personnel/Supplies | $17.7 million |
| Contractor Direct Cost | $1,456.8 million |
| Overhead and Profit @ 15% | $218.5 million |

The contract mining cost will include significant charges for equipment depreciation. It is not certain how a specific contractor will calculate this cost. Table 21-7 shows IMC's estimate of hourly and shift depreciation costs for each equipment type. The equipment replacement cost is the new, delivered price used for the owner operation case. The column labeled IMC Life shows the life of the equipment, in metered hours, that IMC used for equipment replacement calculations. IMC deducted 10,000 hours from each piece of equipment to obtain the adjusted equipment life. This can be thought of as accelerated depreciation or a risk premium for the contractor. Assuming a 10% salvage value for equipment, and straight-line depreciation, the equipment depreciation per metered hour and per shift are shown. This is based on 10.75 metered hours per shift.

A 10% allowance has also been added to account for small equipment. Equipment depreciation during the commercial production period is estimated at $400.3 million. This is comparable to the $432.7 million life of mine equipment capital costs for the owner operation case. Equipment depreciation is 11.5% of the direct operating cost estimate of $3.49 billion.

The costs are in Q4 2024 US dollars. The estimate is based on assumed prices for commodities such as fuel, explosives, parts, tires, etc. that are subject to wide variations depending on market conditions. The estimate is based on the following prices for key commodities:

- Diesel fuel delivered to the site for $0.964/liter,
- Blasting agents produced on site by the vendor for $1.26/kg.

There are some specific risks related to contract mining. There is a risk that the contractor may need financial assistance from the owner either in terms of cash, or loan guarantees, to procure some equipment, increasing the capital cost.

**Table 21-7: Contractor Equipment Depreciation Per Shift**

| Equipment Type | Replac. Cost (US$) | IMC Equip Life (hrs) | Contractor Life (hrs) | Depreciation (Notes 1, 2) | |
|---|---|---|---|---|---|
| | | | | Per Hour ($/hr) | Per Shift ($/shift) |
| Caterpillar MD6380 Drill | 4,395,000 | 70,000 | 60,000 | 65.93 | 703.20 |
| Epiroc SmartROC D65 | 1,100,000 | 55,000 | 45,000 | 22.00 | 234.67 |
| Caterpillar 7395 Cable Shovel | 25,000,000 | 120,000 | 110,000 | 204.55 | 2,181.82 |
| Caterpillar 6060FS Hyd Shovel | 14,000,000 | 70,000 | 60,000 | 210.00 | 2,240.00 |
| Caterpillar 995 Wheel Loader | 5,698,000 | 60,000 | 50,000 | 102.56 | 1,094.02 |
| Caterpillar 793F Truck | 3,130,000 | 100,000 | 90,000 | 31.30 | 333.87 |
| Caterpillar 785G Truck | 2,347,500 | 100,000 | 90,000 | 23.48 | 250.40 |
| Caterpillar D10T2 Track Dozer | 1,237,000 | 50,000 | 40,000 | 27.83 | 296.88 |
| Caterpillar D9T Track Dozer | 931,000 | 50,000 | 40,000 | 20.95 | 223.44 |
| Caterpillar 844K Wheel Dozer | 1,812,000 | 50,000 | 40,000 | 40.77 | 434.88 |
| Caterpillar 16M3 Motor Grader | 1,103,000 | 50,000 | 40,000 | 24.82 | 264.72 |
| Water Truck - 30,000 gal | 2,433,820 | 90,000 | 80,000 | 27.38 | 292.06 |
| Caterpillar 350 Excavator | 279,964 | 40,000 | 30,000 | 8.40 | 89.59 |

Note 1: Depreciation assumes 10% salvage value, i.e. hourly depreciation = 0.9 x cost/life.
Note 2: Assumes 10.67 metered hours per shift.

Table 21-8 summarizes the mine operating cost by various time periods for contract mining.

**Table 21-8: Summary of Total and Unit Mining Costs by Various Time Periods – Contract Mining**

| Time Period | Total Material (kt) | Mill (kt) | Leach (kt) | Total Cost (US$x1000) | Cost Per Total Tonne (US$/t) | Cost Per Mill Tonne (US$/t) | Cost Per Mill/ Leach Tonne (US$/t) |
|---|---|---|---|---|---|---|---|
| Mine Development (Years -2, -1) | 45,071 | 0 | 18,200 | 120,335 | 2.670 | 0.000 | 6.612 |
| **Commercial Production (Years 1 to 38)** | **1,948,481** | **848,880** | **92,322** | **4,110,116** | **2.109** | **4.842** | **4.367** |
| All Time Periods | 1,993,552 | 848,880 | 110,522 | 4,230,451 | 2.122 | 4.984 | 4.409 |
| Commercial Production Years 1 – 5 | 331,005 | 118,126 | 58,460 | 657,632 | 1.987 | 5.567 | 3.724 |
| Commercial Production Years 6 - 10 | 325,500 | 113,730 | 20,011 | 664,951 | 2.043 | 5.847 | 4.972 |
| Commercial Production Years 11 - 20 | 641,607 | 225,037 | 12,168 | 1,452,547 | 2.264 | 6.455 | 6.124 |
| Commercial Production Years 21 - 30 | 475,102 | 219,289 | 1,683 | 1,085,513 | 2.285 | 4.950 | 4.912 |
| Commercial Production Year 31 - Final (LG) | 175,267 | 172,698 | 0 | 249,472 | 1.423 | 1.445 | 1.445 |

### 21.2.2 Process Plant Operating Costs

The process plant includes the following three processes:

- 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 tool.

#### 21.2.2.1 Heap Leach Operations

The heap leach is showing an operating cost of $5.62 per tonne of heap leach ore (110.5 million tonnes) during its 25-year operation. Table 21-9 below shows the operating cost by area.

**Table 21-9: Heap Leach Life of Mine Operating Cost**

| | Total Cost | $/tonne Processed |
|---|---:|---:|
| Heap Leach & Crushing & Conveying | $323,462,814 | $2.93 |
| Solvent Extraction | $54,632,775 | $0.49 |
| Tank Farm | $16,706,900 | $0.15 |
| Electrowinning | $211,036,764 | $1.91 |
| Ancillary Services | $16,020,332 | $0.14 |
| **Total Heap Leach & SX-EW Plant** | **$621,859,585** | **$5.62** |

#### 21.2.2.2 Concentrator Operations

The concentrator is showing an operating cost of $5.61 per tonne of mill ore (848.9 million tonnes) during its 38-year operation. Table 21-10 below displays the operating cost by area.

**Table 21-10: Concentrator Life of Mine Operating Cost**

| Concentrator Operations | Total Cost | $/tonne Mill Ore |
|---|---:|---:|
| Crushing & Conveying | $320,058,617 | $0.38 |
| Grinding & Classification | $2,954,051,587 | $3.48 |
| Flotation & Regrind | $536,653,414 | $0.63 |
| Concentrate Thickening, Filtration & Dewatering | $167,680,953 | $0.20 |
| Tailing Filtration* | $608,192,942 | $0.72 |
| Gravity Gold Circuit | $9,093,009 | $0.01 |
| Gold Refinery | $10,250,773 | $0.01 |
| Ancillary Services | $151,323,188 | $0.18 |
| **Total Concentrator Operations** | **$4,757,304,483** | **$5.61** |

* Does not include tailing stacking costs.

#### 21.2.2.3 Agitated Leach Operations

The agitated leach process is showing an operating cost of $2.68 per tonne of mill ore (848.9 million tonnes) during its 38-year operation. Table 21-11 shows the operating cost by area.

**Table 21-11: Total Operating Costs by Area**

| | Total Cost | $/tonne Mill Ore Leached |
|---|---:|---:|
| Agitated Tailing Leach | $2,048,488,137 | $2.41 |
| SX-EW Tailing Leach | $229,829,741 | $0.27 |
| **Total Agitated Leach Operations** | **$2,278,317,878** | **$2.68** |

#### 21.2.2.4 Filtered Tailing Placement Costs

The cost for placing the filtered tailing material is assumed to be a rate of 60,000 tonnes/day. Stacking will be performed by an earthworks contractor and will include loading, hauling, placement, and compaction. The cost for years 1 through 20 are $3.56 per tonne. The costs for years 21 through 38 are $4.04 per tonne. These costs are not included in the concentrator operating costs.

#### 21.2.2.5 Power Operating Costs

Operating costs for power assume that line power is available and will be provided by GNPower Kauswagan Ltd. Discussions with GNPower show that there is sufficient dependable capacity for the project power needs. The average annual power costs are summarized in Table 21-12 below.

**Table 21-12: Average Annual Power Costs**

| | Average Annual Cost ($ Millions) | Cost per kWh |
|---|---|---|
| Concentrator Power | $78.4 | $0.102 |
| SXEW-Leach Power | $3.20 | $0.102 |
| Gravity Gold Power | $0.20 | $0.102 |
| Ancillary Power | $0.04 | $0.102 |
| **Total Power** | **$81.8** | |

### 21.2.3 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, administrative services (e.g., accounting, human resources, security), 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 Technical Report 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 Kingking Project. The sales revenue was based on the production of copper concentrate containing gold, gold Dore bullion and copper cathode. The estimates of capital expenditures and site production costs have been developed specifically for this project and have been presented in Section 21 of this Technical 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 Technical Report.

The life of mine ore quantities and ore grades are presented in Table 22-1 below.

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

| | Tonnes (kt) | Copper (%) | Gold (g/t) |
|---|---|---|---|
| Heap Leach Ore | 110,522 | 0.228% | 0.169 |
| Mill Sulfide Ore | 751,498 | 0.236% | 0.321 |
| Mill Oxide Ore | 97,381 | 0.432% | 0.644 |
| Oxide Stockpile | 15,039 | 0.225% | 0.244 |
| Sulfide Stockpile | 175,296 | 0.179% | 0.157 |
| Waste | 839,015 | | |
| **Total Material Mined** | **1,988,751** | | |

## 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 cathode. The heap leach ore will be processed using a SX-EW Process. The metal recoveries over LOM are projected as follows in Table 22-2.

**Table 22-2: Metal Recoveries**

| Recovery | Copper Concentrate | Copper Cathode | Gold Bullion |
|---|---|---|---|
| Mill Ore Copper | 62.1% | 22.8% | |
| Mill Ore Gold | 52.3% | | 20.0% |
| Heap Leach Ore Copper | | 78.8% | |

**Table 22-3: Metal Production**

| Metal | Total |
|---|---|
| Copper Concentrate (kt) | 5,453 |
| Copper (klbs) | 3,004,718 |
| Gold (kozs) | 5,110 |
| Gold Dore (kozs) | 1,952 |
| Mill Ore Copper Cathode (klbs) | 1,103,278 |
| Heap Leach Ore Copper Cathode (klbs) | 438,066 |

## 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.

The Asian market smelter charges calculated in the financial evaluation researched by Simon Hunt Strategic Services are presented in Table 22-4 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) | $70.00 |
| Refining charge – Cu ($/lb.) | $0.07 |
| Refining charge – Au ($/payable oz.) | $5.00 |
| Refining charge – Ag ($/payable oz) | $0.50 |
| Gold Insurance (% of gross revenue) | 0.4% |
| Copper Concentrate Transportation ($/wmt) | $40.00 |
| Moisture | 10% |
| **Cathode Terms** | |
| Payable Copper (%) | 100.0% |
| Transportation ($/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 assuming 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 $2,373.8 million expended over a 4-year period. The initial capital includes all required cost categories including Owner's costs and contingency. The majority (96%) of the initial capital cash flow is estimated to be expended in the 3 years prior to the first production year with the remainder (4%) expended in the first production year.

Presented below in Table 22-5 is the initial capital summary.

**Table 22-5: Initial Capital**

| Area | $ Millions |
|---|---|
| Process Plant and General Infrastructure | $1,638.25 |
| Mine | $131.92 |
| Port Facility | $50.00 |
| Owners Costs | $163.48 |
| Contingency | $390.13 |
| Total Before VAT | $2,373.78 |
| Value Added Tax (VAT) | $189.62 |

*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 $798.4 million as shown in Table 22-6. This capital will be expended during a 38-year period. Sustaining capital includes mine equipment, pit diversions, tailings foundations, VRMA, heap leach, site general, and equipment replacement.

**Table 22-6: Sustaining Capital ($ 000s)**

| Item | Year 1 | Year 2 | Year 3 | Year 4 | Year 5 | Year 6 | Year 7 | Year 8 | Year 9 | Year 10 |
|---|---|---|---|---|---|---|---|---|---|---|
| Sustaining Capital | $47,049 | $16,189 | $7,424 | $98,646 | $11,285 | $4,683 | $8,810 | $8,160 | $31,655 | $23,757 |

| Item | Year 11 | Year 12 | Year 13 | Year 14 | Year 15 | Year 16 | Year 17 | Year 18 | Year 19 | Year 20 |
|---|---|---|---|---|---|---|---|---|---|---|
| Sustaining Capital | $962 | $163 | $678 | $32,954 | $20,518 | $1,959 | $1,240 | $805 | $27,458 | $9,441 |

| Item | Year 21 | Year 22 | Year 23 | Year 24 | Year 25 | Year 26 | Year 27 | Year 28 | Year 29 | Year 30 |
|---|---|---|---|---|---|---|---|---|---|---|
| Sustaining Capital | $79,577 | $75,000 | $75,000 | $75,000 | $75,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 |

| Item | Year 31 | Year 32 | Year 33 | Year 34 | Year 35 | Year 36 | Year 37 | Year 38 | Total |
|---|---|---|---|---|---|---|---|---|---|
| Sustaining Capital | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | **$798,412** |

### 22.5.3 Working Capital

A 45-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 are based on the gross value of the payable metals sold before refining and transportation charges.

Copper $4.30 per pound

Gold $2,150.00 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. Table 22-7 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 |
| Payable Pounds of Copper Equivalent | millions | 2,250.2 | 3,517.8 | 7,863.1 |
| Mining | $/lb Cu | $0.30 | $0.38 | $0.53 |
| Processing | $/lb Cu | $0.68 | $0.82 | $1.27 |
| **Operating Costs** | **$/lb Cu** | **$0.98** | **$1.20** | **$1.80** |
| G&A | $/lb Cu | $0.11 | $0.11 | $0.14 |
| Reclamation & Closure | $/lb Cu | $0.00 | $0.00 | $0.01 |
| **Cash Costs at Mine** | **$/lb Cu** | **$1.09** | **$1.31** | **$1.95** |
| Government Fees | $/lb Cu | $0.30 | $0.31 | $0.39 |
| **Total Cash Costs at Mine** | **$/lb Cu** | **$1.38** | **$1.62** | **$2.34** |
| Shipping, Smelting & Refining | $/lb Cu | $0.09 | $0.11 | $0.12 |
| **Total Consolidated Net Cash Costs** | **$/lb Cu** | **$1.47** | **$1.73** | **$2.46** |

## 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 the projects' feasibility level study.

## 22.7 GOVERNMENT FEES

The following government payments were estimated in the cash flow analysis:

- Excise Tax – 4% of NSR revenue, starting from production start; estimated life of mine cost is $1,314.0 million.

- Local Business Tax – 2% of NSR revenue, starting after Income Tax Holiday; estimated life of mine cost is $362.1 million.
- Royalty – ICC - 1% of NSR revenue (w/ credit for Community Development); estimated life of mine cost is $328.5 million.
- Development Mining Technology (ComDev) - Min of 1.5% of total operating cost (incl. depreciation, excise tax); estimated life of mine cost is $320.5 million.
- Environmental Protection and Enhancement Program (EPEP) - 4.0% of total operating cost; estimated life of mine cost is $600.8 million.
- Monitoring Trust Fund (MTF) - PhP150,000/QTR as determined by MRF Committee; estimated life of mine cost is $0.4 million.
- Property Taxes (Land & Equipment) – PhP8,000/hectare and 1.6% of equipment net book value; estimate life of mine cost is $157.6 million.
- Mine Waste and Tailing (MWT) Fees – Waste - PhP0.11 per MT of waste; estimated life of mine cost is $1.5 million.
- Mine Waste and Tailing (MWT) Fees – Tailing - PhP0.17 per MT of waste; estimated life of mine cost is $2.6 million.
- Occupational Fees - PhP75/he/yr post MPSA renewal; PhP50/he/yr prior to MPSA renewal; estimated life of mine cost is $0.13 million.

#### 22.7.1 Income Tax

The Philippines enacted the Create More Tax Act (Republic Act No. 12066) on July 22, 2024, which revised tax incentives in the National Internal Revenue Code. There are four tax incentives, available to projects with capital purchases greater than $1.0 billion, that are applicable to this project. The applicability of these tax incentives has been confirmed by the Philippine's Board of Investments (BOI) regulatory body. The four tax incentives are listed below and have been included in the economic analysis.

1. Extend an Income Tax Holiday (ITH) to 10 years
2. Net Operating Loss (NOL) carry forward allowed after the ITH period
3. Special Corporate Income Tax (SCIT) of 5% in year 11 through year 20
4. VAT Exemption on equipment purchases in both initial and sustaining capital

The income tax rate, beginning in Year 21, will be paid at a rate of 25% based on operating profits. Total income taxes paid during the life of the mine is estimated to be $662.5 million.

### 22.8 NET INCOME AFTER-TAX

Net income after-tax amounts to $10,613.9 million.

### 22.9 PROJECT FINANCING

It is assumed that the Kingking Project will be 100% equity financed.

### 22.10 NET PRESENT VALUE, INTERNAL RATE OF RETURN, PAYBACK

The project's economic results are summarized below in Table 22-8. The Benefit Cost Ratio (BCR) represents the estimated total cashflow benefit of the Kingking Project relative to the estimated total capital cost for the Kingking Project and provides a summary of the overall relationship between the relative costs and benefits of the Kingking Project.

**Table 22-8: Key Economic Results**

| | After-Tax |
|---|---|
| NPV @ 7% (billions) | $4.2 |
| IRR | 34.2% |
| Payback (Years) | 1.9 |
| Benefit Cost Ratio (BCR) | 1.8 |

## 22.11 SENSITIVITY ANALYSIS

Table 22-9 and Figure 22-1 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, operating cost, and initial capital. This table indicates that NPV is most sensitive to the metal prices, then operating cost and much less to initial capital. As stated above, the base case of the Kingking Project was estimated at a conservative gold price and current copper price, which has been relatively steady and projected to increase.

**Table 22-9: After-Tax Sensitivity Analysis**

| | NPV 7% ($000) | IRR | Payback |
|---|---|---|---|
| Base Case Economic Results | $4,181,602 | 34.2% | 1.9 |
| **Metal Prices** | | | |
| +20% | $6,518,987 | 44.5% | 1.5 |
| +10% | $5,350,806 | 39.5% | 1.7 |
| -10% | $3,010,885 | 28.5% | 2.1 |
| -20% | $1,835,129 | 22.0% | 2.6 |
| **Initial Capital** | | | |
| +20% | $3,759,713 | 28.1% | 2.2 |
| +10% | $3,970,658 | 30.9% | 2.0 |
| -10% | $4,392,546 | 38.1% | 1.7 |
| -20% | $4,603,491 | 42.7% | 1.6 |
| **Operating Cost** | | | |
| +20% | $3,236,567 | 30.7% | 2.0 |
| +10% | $3,710,583 | 32.5% | 1.9 |
| -10% | $4,651,818 | 35.9% | 1.8 |
| -20% | $5,120,945 | 37.4% | 1.8 |

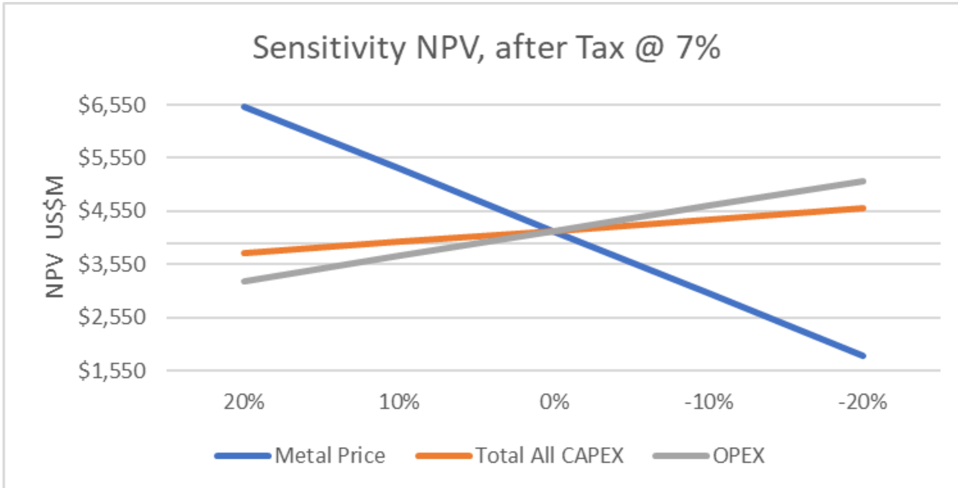



**Figure 22-1: After-Tax NPV (7%) Sensitivities**

## 22.12 DETAILED FINANCIAL MODEL

The detailed bottom-up financial model, shown in Table 22-10 below, was developed in compliance with Preliminary Feasibility Study (PFS) requirements per the Association for the Advancement of Cost Engineering International (AACEI) and National Instruments 43-101 (NI43-101). 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 Kingking Project as well as for sensitivity analysis.

## Table 22-10: Detailed Financial Model

| | Contract<br>Total | 0<br>2025<br>-4 | 1<br>2026<br>-3 | 2<br>2027<br>-2 | 3<br>2028<br>-1 | 4<br>2029<br>1 | 5<br>2030<br>2 | 6<br>2031<br>3 | 7<br>2032<br>4 | 8<br>2033<br>5 | 9<br>2034<br>6 | 10<br>2035<br>7 | 11<br>2036<br>8 | 12<br>2037<br>9 | 13<br>2038<br>10 | 14<br>2039<br>11 | 15<br>2040<br>12 | 16<br>2041<br>13 | 17<br>2042<br>14 | 18<br>2043<br>15 | 19<br>2044<br>16 | 20<br>2045<br>17 | 21<br>2046<br>18 | 22<br>2047<br>19 | 23<br>2048<br>20 | 24<br>2049<br>21 | 25<br>2050<br>22 | 26<br>2051<br>23 | 27<br>2052<br>24 | 28<br>2053<br>25 | 29<br>2054<br>26 | 30<br>2055<br>27 | 31<br>2056<br>28 | 32<br>2057<br>29 | 33<br>2058<br>30 | 34<br>2059<br>31 | 35<br>2060<br>32 | 36<br>2061<br>33 | 37<br>2062<br>34 | 38<br>2063<br>35 | 39<br>2064<br>36 | 40<br>2065<br>37 | 41<br>2066<br>38 | 42<br>2067<br>39 | 43<br>2068<br>40 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| **Mining Operations** |
| **Heap Leach Ore** |
| Beginning Inventory (kt) | 110,522 | 110,522 | 110,522 | 110,522 | 106,922 | 92,322 | 77,722 | 63,122 | 48,522 | 41,222 | 33,862 | 28,573 | 27,940 | 23,585 | 21,943 | 13,851 | 3,900 | 2,687 | 2,681 | 2,681 | 2,579 | 2,579 | 2,579 | 2,422 | 1,683 | 1,683 | 1,683 | 1,683 | 1,683 | 921 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Mined (kt) | 110,522 | - | - | 3,600 | 14,600 | 14,600 | 14,600 | 14,600 | 7,300 | 7,360 | 5,289 | 633 | 4,355 | 1,642 | 8,092 | 9,951 | 1,213 | 6 | - | 102 | - | - | 157 | 739 | - | - | - | - | 762 | 921 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Ending Inventory (kt) | - | 110,522 | 110,522 | 106,922 | 92,322 | 77,722 | 63,122 | 48,522 | 41,222 | 33,862 | 28,573 | 27,940 | 23,585 | 21,943 | 13,851 | 3,900 | 2,687 | 2,681 | 2,681 | 2,579 | 2,579 | 2,579 | 2,422 | 1,683 | 1,683 | 1,683 | 1,683 | 1,683 | 921 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Gold Grade (g/t) | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Copper Grade (%) | 0.228% | 0% | 0% | 0.199% | 0.240% | 0.318% | 0.311% | 0.245% | 0.168% | 0.269% | 0.242% | 0.211% | 0.139% | 0.138% | 0.118% | 0.118% | 0.156% | 0.235% | 0.000% | 0.141% | 0.000% | 0.000% | 0.306% | 0.221% | 0.000% | 0.000% | 0.000% | 0.000% | 0.133% | 0.174% | 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) | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Contained Copper (klbs) | 556,295 | - | - | 15,795 | 77,165 | 102,235 | 99,970 | 78,975 | 27,008 | 43,630 | 28,247 | 2,951 | 13,350 | 4,988 | 21,095 | 25,924 | 4,181 | 31 | - | 317 | - | - | 1,059 | 3,604 | - | - | - | - | 2,234 | 3,533 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Mill Ore** |
| Beginning Inventory (kt) | 848,879 | 848,879 | 848,879 | 848,879 | 848,879 | 848,879 | 829,129 | 802,879 | 777,129 | 753,129 | 730,754 | 708,304 | 685,974 | 663,824 | 641,274 | 617,024 | 592,199 | 568,924 | 546,714 | 524,639 | 502,739 | 480,839 | 458,939 | 436,579 | 413,887 | 391,987 | 370,087 | 348,187 | 326,287 | 304,337 | 282,198 | 260,298 | 238,398 | 216,498 | 194,598 | 172,698 | 150,798 | 128,898 | 106,998 | 85,098 | 63,198 | 41,298 | 19,398 | 0 | 0 |
| Mined (kt) | 848,879 | - | - | - | - | 19,750 | 26,250 | 25,750 | 24,000 | 22,375 | 22,450 | 22,330 | 22,150 | 22,550 | 24,250 | 24,825 | 23,275 | 22,210 | 22,075 | 21,900 | 21,900 | 21,900 | 22,360 | 22,692 | 21,900 | 21,900 | 21,900 | 21,900 | 21,950 | 22,139 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 19,398 | - | - |
| Ending Inventory (kt) | - | 848,879 | 848,879 | 848,879 | 848,879 | 829,129 | 802,879 | 777,129 | 753,129 | 730,754 | 708,304 | 685,974 | 663,824 | 641,274 | 617,024 | 592,199 | 568,924 | 546,714 | 524,639 | 502,739 | 480,839 | 458,939 | 436,579 | 413,887 | 391,987 | 370,087 | 348,187 | 326,287 | 304,337 | 282,198 | 260,298 | 238,398 | 216,498 | 194,598 | 172,698 | 150,798 | 128,898 | 106,998 | 85,098 | 63,198 | 41,298 | 19,398 | 0 | 0 | 0 |
| Gold Grade (g/t) | 0.358 | - | - | - | - | 0.367 | 0.641 | 0.833 | 0.788 | 0.608 | 0.422 | 0.322 | 0.358 | 0.357 | 0.475 | 0.548 | 0.518 | 0.481 | 0.515 | 0.394 | 0.332 | 0.345 | 0.372 | 0.168 | 0.260 | 0.329 | 0.352 | 0.350 | 0.349 | 0.294 | 0.159 | 0.192 | 0.290 | 0.371 | 0.293 | 0.141 | 0.135 | 0.180 | 0.154 | 0.136 | 0.163 | 0.214 | 0.150 | - | - |
| Copper Grade (%) | 0.259% | 0.000% | 0.000% | 0.000% | 0.000% | 0.673% | 0.597% | 0.394% | 0.314% | 0.332% | 0.384% | 0.407% | 0.349% | 0.315% | 0.268% | 0.201% | 0.223% | 0.221% | 0.208% | 0.153% | 0.150% | 0.163% | 0.174% | 0.251% | 0.264% | 0.247% | 0.236% | 0.227% | 0.222% | 0.223% | 0.242% | 0.244% | 0.255% | 0.213% | 0.159% | 0.149% | 0.126% | 0.131% | 0.169% | 0.204% | 0.199% | 0.231% | 0.254% | 0.000% | 0.000% |
| Contained Gold (kozs) | 9,773 | - | - | - | - | 233 | 541 | 689 | 608 | 437 | 304 | 231 | 255 | 259 | 370 | 438 | 388 | 343 | 366 | 277 | 234 | 243 | 267 | 123 | 183 | 232 | 248 | 246 | 247 | 209 | 112 | 135 | 204 | 261 | 206 | 99 | 95 | 127 | 108 | 96 | 114 | 151 | 94 | - | - |
| Contained Copper (klbs) | 4,838,885 | - | - | - | - | 293,204 | 345,675 | 223,607 | 166,250 | 163,703 | 189,948 | 200,456 | 170,264 | 156,610 | 143,163 | 110,276 | 114,306 | 108,197 | 101,306 | 73,899 | 72,422 | 78,698 | 85,531 | 125,606 | 127,509 | 119,255 | 113,944 | 109,598 | 107,494 | 108,943 | 116,840 | 117,806 | 123,117 | 102,839 | 76,794 | 71,766 | 61,076 | 63,178 | 81,609 | 98,275 | 95,894 | 111,366 | 108,461 | - | - |
| **All Ore** |
| Beginning Inventory (kt) | 1,149,736 | 1,149,736 | 1,149,736 | 1,149,736 | 1,142,019 | 1,119,256 | 1,075,753 | 1,026,860 | 974,206 | 932,986 | 892,452 | 845,219 | 801,265 | 758,157 | 729,524 | 691,061 | 648,829 | 618,293 | 591,242 | 568,432 | 543,973 | 516,866 | 494,966 | 471,976 | 444,920 | 419,560 | 394,333 | 367,882 | 342,412 | 316,364 | 293,304 | 267,394 | 243,676 | 218,746 | 194,598 | 172,698 | 150,798 | 128,898 | 106,998 | 85,098 | 63,198 | 41,298 | 19,398 | 0 | 0 |
| Mined (kt) | 1,149,736 | - | - | 7,717 | 22,763 | 43,503 | 48,893 | 52,654 | 41,220 | 40,534 | 47,233 | 43,954 | 43,108 | 28,633 | 38,463 | 42,232 | 30,536 | 27,051 | 22,810 | 24,459 | 27,107 | 21,900 | 22,990 | 27,056 | 25,360 | 25,227 | 26,451 | 25,470 | 26,048 | 23,060 | 25,910 | 23,718 | 24,930 | 24,148 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 19,398 | - | - |
| Ending Inventory (kt) | - | 1,149,736 | 1,149,736 | 1,142,019 | 1,119,256 | 1,075,753 | 1,026,860 | 974,206 | 932,986 | 892,452 | 845,219 | 801,265 | 758,157 | 729,524 | 691,061 | 648,829 | 618,293 | 591,242 | 568,432 | 543,973 | 516,866 | 494,966 | 471,976 | 444,920 | 419,560 | 394,333 | 367,882 | 342,412 | 316,364 | 293,304 | 267,394 | 243,676 | 218,746 | 194,598 | 172,698 | 150,798 | 128,898 | 106,998 | 85,098 | 63,198 | 41,298 | 19,398 | 0 | 0 | 0 |
| Gold Grade (g/t) | 0.288 | - | - | - | 0.006 | 0.183 | 0.367 | 0.453 | 0.514 | 0.392 | 0.260 | 0.228 | 0.244 | 0.306 | 0.328 | 0.356 | 0.428 | 0.426 | 0.503 | 0.369 | 0.297 | 0.345 | 0.364 | 0.151 | 0.240 | 0.302 | 0.314 | 0.321 | 0.315 | 0.282 | 0.146 | 0.183 | 0.272 | 0.352 | 0.293 | 0.141 | 0.135 | 0.180 | 0.154 | 0.136 | 0.163 | 0.214 | 0.150 | - | - |
| Copper Grade (%) | 0.240% | 0.000% | 0.000% | 0.093% | 0.168% | 0.442% | 0.454% | 0.323% | 0.269% | 0.277% | 0.297% | 0.306% | 0.257% | 0.281% | 0.218% | 0.167% | 0.199% | 0.202% | 0.205% | 0.146% | 0.141% | 0.163% | 0.173% | 0.240% | 0.250% | 0.232% | 0.219% | 0.212% | 0.204% | 0.221% | 0.229% | 0.238% | 0.244% | 0.202% | 0.159% | 0.149% | 0.126% | 0.131% | 0.169% | 0.204% | 0.199% | 0.231% | 0.254% | 0.000% | 0.000% |
| Contained Gold (kozs) | 10,658 | - | - | - | 4 | 255 | 576 | 767 | 681 | 510 | 395 | 322 | 338 | 281 | 406 | 484 | 420 | 370 | 369 | 290 | 259 | 243 | 269 | 131 | 196 | 245 | 267 | 263 | 264 | 209 | 122 | 140 | 218 | 273 | 206 | 99 | 95 | 127 | 108 | 96 | 114 | 151 | 94 | - | - |
| Contained Copper (klbs) | 6,087,854 | - | - | 15,795 | 84,109 | 424,175 | 489,442 | 374,465 | 244,652 | 247,093 | 308,877 | 296,887 | 244,010 | 177,068 | 185,040 | 155,597 | 133,820 | 120,699 | 103,040 | 78,929 | 84,016 | 78,698 | 87,602 | 143,115 | 139,714 | 129,230 | 127,589 | 119,043 | 117,083 | 112,476 | 130,897 | 124,339 | 134,339 | 107,597 | 76,794 | 71,766 | 61,076 | 63,178 | 81,609 | 98,275 | 95,894 | 111,366 | 108,461 | - | - |
| **Waste** |
| Beginning Inventory(kt) | 839,015 | 839,015 | 839,015 | 839,015 | 835,710 | 829,224 | 817,545 | 799,371 | 776,241 | 749,278 | 725,024 | 707,576 | 686,773 | 664,972 | 627,883 | 600,915 | 578,054 | 543,439 | 505,628 | 463,018 | 421,799 | 383,495 | 340,235 | 298,128 | 265,326 | 230,809 | 196,121 | 162,619 | 128,117 | 94,193 | 57,096 | 32,934 | 26,609 | 21,496 | 15,601 | 2,569 | - | - | - | - | - | - | - | - | - |
| Mined (kt) | 839,015 | - | - | 3,305 | 6,486 | 11,679 | 18,174 | 23,130 | 26,963 | 24,254 | 17,448 | 20,803 | 21,801 | 37,089 | 26,968 | 22,861 | 34,615 | 37,811 | 42,610 | 41,219 | 38,304 | 43,260 | 42,107 | 32,802 | 34,517 | 34,688 | 33,502 | 34,502 | 33,924 | 37,097 | 24,162 | 6,325 | 5,113 | 5,895 | 13,032 | 2,569 | - | - | - | - | - | - | - | - | - |
| Ending Inventory (kt) | - | 839,015 | 839,015 | 835,710 | 829,224 | 817,545 | 799,371 | 776,241 | 749,278 | 725,024 | 707,576 | 686,773 | 664,972 | 627,883 | 600,915 | 578,054 | 543,439 | 505,628 | 463,018 | 421,799 | 383,495 | 340,235 | 298,128 | 265,326 | 230,809 | 196,121 | 162,619 | 128,117 | 94,193 | 57,096 | 32,934 | 26,609 | 21,496 | 15,601 | 2,569 | - | - | - | - | - | - | - | - | - | - |
| Total Material Mined (kt) | 1,988,751 | - | - | 11,022 | 29,249 | 55,182 | 67,067 | 75,784 | 68,183 | 64,788 | 64,681 | 64,757 | 64,909 | 65,722 | 65,431 | 65,093 | 65,151 | 64,862 | 65,420 | 65,678 | 65,411 | 65,160 | 65,097 | 59,858 | 59,877 | 59,915 | 59,953 | 59,972 | 59,972 | 60,157 | 50,072 | 30,043 | 30,043 | 30,043 | 34,932 | 24,469 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 19,398 | - | - |
| **Process Plant Operations** |
| **Concentrator** |
| Milled Ore - Processed (kt) | 848,879 | - | - | - | - | 19,750 | 26,250 | 25,750 | 24,000 | 22,375 | 22,450 | 22,330 | 22,150 | 22,550 | 24,250 | 24,825 | 23,275 | 22,210 | 22,075 | 21,900 | 21,900 | 21,900 | 22,360 | 22,692 | 21,900 | 21,900 | 21,900 | 21,900 | 21,950 | 22,139 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 21,900 | 19,398 | - | - |
| Gold Grade (g/t) | 0.358 | - | - | - | - | 0.367 | 0.641 | 0.833 | 0.788 | 0.608 | 0.422 | 0.322 | 0.358 | 0.357 | 0.475 | 0.548 | 0.518 | 0.481 | 0.515 | 0.394 | 0.332 | 0.345 | 0.372 | 0.168 | 0.260 | 0.329 | 0.352 | 0.350 | 0.349 | 0.294 | 0.159 | 0.192 | 0.290 | 0.371 | 0.293 | 0.141 | 0.135 | 0.180 | 0.154 | 0.136 | 0.163 | 0.214 | 0.150 | - | - |
| Copper Grade (%) | 0.259% | 0.000% | 0.000% | 0.000% | 0.000% | 0.673% | 0.597% | 0.394% | 0.314% | 0.332% | 0.384% | 0.407% | 0.349% | 0.315% | 0.268% | 0.201% | 0.223% | 0.221% | 0.208% | 0.153% | 0.150% | 0.163% | 0.174% | 0.251% | 0.264% | 0.247% | 0.236% | 0.227% | 0.222% | 0.223% | 0.242% | 0.244% | 0.255% | 0.213% | 0.159% | 0.149% | 0.126% | 0.131% | 0.169% | 0.204% | 0.199% | 0.231% | 0.254% | 0.000% | 0.000% |
| Contained Gold (kozs) | 9,773 | - | - | - | - | 232.974 | 541 | 689 | 608 | 437 | 304 | 231 | 255 | 259 | 370 | 438 | 388 | 343 | 366 | 277 | 234 | 243 | 267 | 123 | 183 | 232 | 248 | 246 | 247 | 209 | 112 | 135 | 204 | 261 | 206 | 99 | 95 | 127 | 108 | 96 | 114 | 151 | 94 | - | - |
| Contained Copper (klbs) | 4,838,885 | - | - | - | - | 293,204 | 345,675 | 223,607 | 166,250 | 163,703 | 189,948 | 200,456 | 170,264 | 156,610 | 143,163 | 110,276 | 114,306 | 108,197 | 101,306 | 73,899 | 72,422 | 78,698 | 85,531 | 125,606 | 127,509 | 119,255 | 113,944 | 109,598 | 107,494 | 108,943 | 116,840 | 117,806 | 123,117 | 102,839 | 76,794 | 71,766 | 61,076 | 63,178 | 81,609 | 98,275 | 95,894 | 111,366 | 108,461 | - | - |
| Recovery Gold to Conc. (%) | 52.28% | 0.00% | 0.00% | 0.00% | 0.00% | 35.51% | 38.80% | 43.30% | 51.06% | 56.03% | 55.28% | 57.03% | 57.20% | 55.53% | 46.55% | 51.61% | 55.84% | 55.51% | 54.22% | 54.57% | 53.92% | 52.46% | 53.00% | 54.67% | 56.15% | 57.14% | 56.53% | 56.00% | 55.51% | 54.78% | 54.09% | 54.17% | 55.86% | 56.87% | 55.69% | 54.15% | 51.88% | 52.72% | 57.32% | 54.78% | 53.67% | 55.19% | 55.35% | 0.00% | 0.00% |
| Recovery Gold to Bullion (%) | 19.97% | 0.00% | 0.00% | 0.00% | 0.00% | 23.94% | 24.12% | 22.85% | 20.82% | 19.17% | 19.03% | 19.15% | 19.16% | 19.59% | 21.82% | 19.92% | 19.05% | 19.08% | 19.11% | 19.04% | 18.98% | 18.84% | 19.13% | 18.97% | 18.85% | 19.15% | 19.03% | 18.86% | 19.09% | 19.15% | 18.87% | 18.75% | 18.97% | 19.14% | 18.92% | 19.00% | 19.02% | 18.91% | 19.11% | 18.67% | 18.86% | 19.08% | 18.87% | 0.00% | 0.00% |
| Recovery Copper to Conc. (%) | 62.10% | 0.00% | 0.00% | 0.00% | 0.00% | 38.04% | 41.77% | 46.18% | 63.95% | 69.16% | 70.24% | 72.18% | 70.86% | 68.45% | 56.84% | 59.42% | 66.12% | 67.03% | 65.57% | 64.71% | 62.67% | 61.96% | 61.78% | 66.43% | 68.94% | 68.83% | 68.64% | 68.28% | 67.88% | 66.85% | 66.53% | 66.80% | 68.24% | 68.08% | 65.57% | 63.23% | 60.52% | 61.78% | 64.52% | 65.93% | 66.09% | 67.34% | 67.83% | 0.00% | 0.00% |
| Recovery Copper to Cathode (%) | 22.80% | 0.00% | 0.00% | 0.00% | 0.00% | 54.87% | 51.00% | 44.84% | 24.68% | 19.32% | 16.79% | 13.67% | 12.43% | 15.06% | 28.92% | 26.39% | 19.77% | 18.59% | 20.47% | 18.23% | 18.60% | 22.61% | 22.33% | 14.96% | 13.92% | 13.27% | 13.89% | 14.01% | 14.95% | 16.17% | 15.19% | 15.06% | 14.41% | 13.54% | 15.52% | 14.12% | 16.82% | 16.46% | 11.14% | 12.55% | 13.82% | 14.47% | 12.39% | 0.00% | 0.00% |
| **Copper Concentrate (kt)** | 5,453 | - | - | - | - | 203 | 262 | 188 | 193 | 205 | 242 | 263 | 219 | 194 | 148 | 119 | 137 | 131 | 121 | 87 | 83 | 88 | 96 | 151 | 159 | 149 | 142 | 136 | 133 | 132 | 141 | 143 | 153 | 127 | 91 | 82 | 67 | 71 | 96 | 117 | 115 | 136 | 133 | - | - |
| Copper Concentrate Grade (%) | 25% | 0% | 0% | 0% | 0% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 25% | 0% | 0% |
| Recovered Gold (kozs) | 5,110 | - | - | - | - | 83 | 210 | 299 | 310 | 245 | 168 | 132 | 146 | 144 | 172 | 226 | 216 | 191 | 198 | 151 | 126 | 127 | 142 | 67 | 103 | 132 | 140 | 138 | 137 | 115 | 61 | 73 | 114 | 149 | 115 | 54 | 49 | 67 | 62 | 52 | 61 | 83 | 52 | - | - |
| Recovered Copper (klbs) | 3,004,718 | - | - | - | - | 111,533 | 144,385 | 103,262 | 106,318 | 113,214 | 133,412 | 144,688 | 120,657 | 107,194 | 81,367 | 65,522 | 75,577 | 72,523 | 66,425 | 47,817 | 45,384 | 48,764 | 52,840 | 83,443 | 87,902 | 82,078 | 78,216 | 74,836 | 72,966 | 72,826 | 77,733 | 78,698 | 84,009 | 70,008 | 50,357 | 45,377 | 36,964 | 39,029 | 52,657 | 64,795 | 63,381 | 74,998 | 73,565 | - | - |
| **Cathode Copper (klbs)** | 1,103,278 | - | - | - | - | 160,889 | 176,298 | 100,266 | 41,032 | 31,630 | 31,895 | 27,393 | 21,157 | 23,580 | 41,409 | 29,107 | 22,596 | 20,115 | 20,733 | 13,475 | 13,468 | 17,792 | 19,096 | 18,793 | 17,753 | 15,825 | 15,830 | 15,354 | 16,074 | 17,618 | 17,749 | 17,747 | 17,739 | 13,920 | 11,917 | 10,131 | 10,272 | 10,400 | 9,090 | 12,335 | 13,252 | 16,111 | 13,437 | - | - |
| **Gold Bullion (kozs)** | 1,952 | - | - | - | - | 56 | 130 | 158 | 127 | 84 | 58 | 44 | 49 | 51 | 81 | 87 | 74 | 66 | 70 | 53 | 44 | 46 | 51 | 23 | 35 | 44 | 47 | 46 | 47 | 40 | 21 | 25 | 39 | 50 | 39 | 19 | 18 | 24 | 21 | 18 | 22 | 29 | 18 | - | - |
| **Heap Leach** |
| Heap Leach Ore - Processed (kt) | 110,522 | - | - | 3,600 | 14,600 | 14,600 | 14,600 | 14,600 | 7,300 | 7,360 | 5,289 | 633 | 4,355 | 1,642 | 8,092 | 9,951 | 1,213 | 6 | - | 102 | - | - | 157 | 739 | - | - | - | - | 762 | 921 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Gold Grade (g/t) | 0.000 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Copper Grade (%) | 0.228% | 0.000% | 0.000% | 0.199% | 0.240% | 0.318% | 0.311% | 0.245% | 0.168% | 0.269% | 0.242% | 0.211% | 0.139% | 0.138% | 0.118% | 0.118% | 0.156% | 0.235% | 0.000% | 0.141% | 0.000% | 0.000% | 0.306% | 0.221% | 0.000% | 0.000% | 0.000% | 0.000% | 0.133% | 0.174% | 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) | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Contained Copper (klbs) | 556,295 | - | - | 15,795 | 77,165 | 102,235 | 99,970 | 78,975 | 27,008 | 43,630 | 28,247 | 2,951 | 13,350 | 4,988 | 21,095 | 25,924 | 4,181 | 31 | - | 317 | - | - | 1,059 | 3,604 | - | - | - | - | 2,234 | 3,533 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Recovery Copper to Cathode (%) | 78.75% | 0.00% | 0.00% | 72.58% | 78.85% | 80.06% | 81.85% | 79.57% | 74.67% | 77.70% | 72.04% | 45.33% | 80.67% | 73.16% | 82.24% | 79.46% | 58.40% | 58.72% | 0.00% | 69.50% | 0.00% | 0.00% | 85.62% | 74.21% | 0.00% | 0.00% | 0.00% | 0.00% | 83.46% | 84.48% | 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)** | 438,066 | - | - | | 72,313 | 81,854 | 81,829 | 62,839 | 20,166 | 33,901 | 20,348 | 1,338 | 10,769 | 3,650 | 17,349 | 20,600 | 2,441 | 18 | - | 220 | - | - | 907 | 2,674 | - | - | - | - | 1,865 | 2,985 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| **Payable Metals** |
| **Copper Concentrate** |
| Payable Gold (kozs) | 4,925 | - | - | - | - | 76 | 201 | 291 | 303 | 238 | 161 | 124 | 139 | 137 | 168 | 220 | 211 | 186 | 193 | 148 | 123 | 124 | 138 | 62 | 98 | 128 | 136 | 134 | 133 | 110 | 56 | 69 | 109 | 144 | 112 | 51 | 47 | 65 | 59 | 49 | 58 | 79 | 48 | - | - |
| Payable Copper (klbs) | 2,884,491 | - | - | - | - | 107,068 | 138,601 | 99,124 | 102,059 | 108,686 | 128,080 | 138,896 | 115,827 | 102,915 | 78,112 | 62,901 | 72,547 | 69,627 | 63,768 | 45,909 | 43,559 | 46,813 | 50,731 | 80,106 | 84,390 | 78,800 | 75,087 | 71,838 | 70,037 | 69,917 | 74,619 | 75,546 | 80,639 | 67,203 | 48,348 | 43,558 | 35,483 | 37,466 | 50,548 | 62,205 | 60,854 | 72,000 | 70,625 | - | - |
| **Cathode Copper** |
| Heap Leach - Payable Copper (klbs) | 438,066 | - | - | - | 72,313 | 81,854 | 81,829 | 62,839 | 20,166 | 33,901 | 20,348 | 1,338 | 10,769 | 3,650 | 17,349 | 20,600 | 2,441 | 18 | - | 220 | - | - | 907 | 2,674 | - | - | - | - | 1,865 | 2,985 | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |
| Tailings Leach - Payable Copper (klbs) | 1,103,278 | - | - | - | - | 160,889 | 176,298 | 100,266 | 41,032 | 31,630 | 31,895 | 27,393 | 21,157 | 23,580 | 41,409 | 29,107 | 22,596 | 20,115 | 20,733 | 13,475 | 13,468 | 17,792 | 19,096 | 18,793 | 17,753 | 15,825 | 15,830 | 15,354 | 16,074 | 17,618 | 17,749 | 17,747 | 17,739 | 13,920 | 11,917 | 10,131 | 10,272 | 10,400 | 9,090 | 12,335 | 13,252 | 16,111 | 13,437 | - | - |
| **Gold Bullion** |
| Payable Gold (kozs) | 1,950 | - | - | - | - | 56 | 130 | 157 | 126 | 84 | 58 | 44 | 49 | 51 | 81 | 87 | 74 | 65 | 70 | 53 | 44 | 46 | 51 | 23 | 34 | 44 | 47 | 46 | 47 | 40 | 21 | 25 | 39 | 50 | 39 | 19 | 18 | 24 | 21 | 18 | 22 | 29 | 18 | - | - |
| **Income Statement ($000)** |
| **Metal Prices** |
| Gold ($/oz) | $2,150 | | | $0 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 | $2,150 |
| Copper ($/lb) | $4.30 | | | $0 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 |

| | Contract | 0 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 |
| | Total | -4 | -3 | -2 | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 |
| **Revenues** | | | | | | | | | | | | | |
| **Copper Concentrate** | | | | | | | | | | | | | |
| Gold Revenue ($ 000) | $10,588,319 | $0 | $0 | $0 | $0 | $163,872 | $432,864 | $625,767 | $650,605 | $512,644 | $345,121 | $265,615 | $297,959 |
| Copper Revenue ($ 000) | $12,403,310 | $0 | $0 | $0 | $0 | $460,392 | $595,986 | $426,235 | $438,854 | $467,348 | $550,744 | $597,253 | $498,054 |
| Less: Treatment & Refining Charges | | | | | | | | | | | | | |
| Copper Concentrate | | | | | | | | | | | | | |
| Treatment Charges | -$381,739 | | | $0 | $0 | -$14,179 | -$18,363 | -$13,138 | -$13,522 | -$14,378 | -$16,931 | -$18,390 | -$15,338 |
| Copper Refining Charges | -$210,330 | | | $0 | $0 | -$7,807 | -$10,107 | -$7,228 | -$7,442 | -$7,925 | -$9,339 | -$10,128 | -$8,446 |
| Gold Refining Charges | -$24,624 | | | $0 | $0 | -$381 | -$1,007 | -$1,455 | -$1,513 | -$1,192 | -$803 | -$618 | -$693 |
| Silver Refining Charges | $0 | | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Gold Insurance Cost | -$42,353 | | | $0 | $0 | -$655 | -$1,731 | -$2,503 | -$2,602 | -$2,051 | -$1,380 | -$1,062 | -$1,192 |
| Transportation | -$239,950 | | | $0 | $0 | -$8,913 | -$11,543 | -$8,258 | -$8,499 | -$9,038 | -$10,642 | -$11,560 | -$9,641 |
| **Net Smelter Return - Copper Concentrate** | **$22,092,632** | | | **$0** | **$0** | **$592,328** | **$986,100** | **$1,019,419** | **$1,055,879** | **$945,409** | **$856,769** | **$821,110** | **$760,704** |
| **Copper Cathode Revenue ($ 000)** | $0 | $0 | | | | | | | | | | | |
| Heap Leach | $1,883,685 | $0 | $0 | $0 | $310,945 | $351,971 | $351,866 | $270,207 | $86,716 | $145,772 | $87,498 | $5,752 | $46,308 |
| Tailings Leach | $4,744,096 | $0 | $0 | $0 | $0 | $691,821 | $758,083 | $431,146 | $176,437 | $136,011 | $137,147 | $117,790 | $90,974 |
| Less: Treatment & Refining Charges | | | | | | | | | | | | | |
| Transportation | -$15,413 | | | $0 | -$723 | -$2,427 | -$2,581 | -$1,631 | -$612 | -$655 | -$522 | -$287 | -$319 |
| **Gold Bullion** | $4,191,777 | $0 | $0 | $0 | $0 | $119,779 | $280,065 | $338,426 | $271,831 | $180,082 | $124,421 | $95,189 | $104,771 |
| Less: Treatment & Refining Charges | | | | | | | | | | | | | |
| Gold Refining Charges | -$3,899 | | | $0 | $0 | -$111 | -$261 | -$315 | -$253 | -$168 | -$116 | -$89 | -$97 |
| Gold Transportation/Insurance Cost | -$41,918 | | | $0 | $0 | -$1,198 | -$2,801 | -$3,384 | -$2,718 | -$1,801 | -$1,244 | -$952 | -$1,048 |
| Total Revenues (NSR) | $32,850,960 | $0 | $0 | $0 | $310,221 | $1,752,163 | $2,370,471 | $2,053,868 | $1,587,281 | $1,404,650 | $1,203,953 | $1,038,513 | $1,001,292 |
| **Operating Cost** | | | | | | | | | | | | | |
| Mining (Incl pit dewatering) | $4,137,091 | | | $0 | $0 | $114,783 | $133,756 | $150,581 | $135,640 | $137,285 | $139,135 | $132,736 | $133,279 |
| Concentrator | $4,129,768 | | | $0 | $0 | $97,592 | $111,854 | $110,662 | $109,372 | $108,352 | $109,281 | $108,993 | $108,844 |
| Gravity Gold Circuit | $19,356 | | | $0 | $0 | $507 | $514 | $513 | $511 | $509 | $509 | $509 | $509 |
| Agitated Tailings Leach | $2,048,488 | | | $0 | $0 | $49,252 | $60,228 | $59,113 | $56,986 | $54,601 | $54,632 | $54,518 | $53,363 |
| SXEW - Tailings Leach | $229,830 | | | $0 | $0 | $20,917 | $22,741 | $14,251 | $8,012 | $5,986 | $6,699 | $8,083 | $5,782 |
| Tailings Disposal | $3,206,514 | | | $0 | $0 | $70,987 | $92,030 | $90,443 | $84,891 | $79,735 | $79,973 | $79,593 | $79,022 |
| Heap Leach & SXEW | $366,382 | | | $0 | $51,342 | $42,454 | $42,368 | $40,744 | $22,166 | $24,755 | $18,760 | $6,216 | $15,690 |
| General Administration | $882,198 | | | $10,738 | $23,865 | $36,836 | $38,978 | $34,721 | $31,761 | $30,240 | $27,304 | $25,970 | $25,022 |
| Laboratory | $26,794 | | | $0 | $469 | $760 | $760 | $760 | $760 | $760 | $760 | $760 | $760 |
| Port | $80,355 | | | $0 | $2,036 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 |
| Custom Duties | $113,092 | | | | $1,469 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 |
| Total Operating Cost | $15,239,868 | | | $10,738 | $79,181 | $439,097 | $508,237 | $506,799 | $455,108 | $447,232 | $442,063 | $422,388 | $427,280 |
| | $18 | | | | | | | | | | | | |
| Salvage Value | $0 | | | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Reclamation & Closure | $74,458 | | | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Total Production Cost | $15,314,326 | | $0 | $10,738 | $79,181 | $439,097 | $508,237 | $506,799 | $455,108 | $447,232 | $442,063 | $422,388 | $427,280 |
| Operating Income | $17,536,635 | | $0 | -$10,738 | $231,041 | $1,313,066 | $1,862,234 | $1,547,069 | $1,132,173 | $957,418 | $761,890 | $616,125 | $574,012 |
| Depreciation | $3,172,196 | | | | $90,180 | $161,389 | $162,468 | $162,963 | $169,539 | $170,292 | $170,604 | $171,191 | $171,735 |
| Total Depreciation | $3,172,196 | | $0 | $0 | $90,180 | $161,389 | $162,468 | $162,963 | $169,539 | $170,292 | $170,604 | $171,191 | $171,735 |
| Net Income After Depreciation | $14,364,439 | | $0 | -$10,738 | $140,861 | $1,151,677 | $1,699,766 | $1,384,106 | $962,633 | $787,127 | $591,286 | $444,934 | $402,276 |
| Government Fees | | | | | | | | | | | | | |
| Excise Tax | $1,314,038 | | $0 | $0 | $12,409 | $70,087 | $94,819 | $82,155 | $63,491 | $56,186 | $48,158 | $41,541 | $40,052 |
| Local Business Tax | $362,054 | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Royalty - ICC | $328,510 | | $0 | $0 | $3,102 | $17,522 | $23,705 | $20,539 | $15,873 | $14,047 | $12,040 | $10,385 | $10,013 |
| Development Mining Technology (ComDev) | $320,456 | | $0 | $168 | $2,812 | $10,866 | $12,479 | $12,159 | $11,066 | $10,798 | $10,572 | $10,142 | $10,158 |
| Annual EPEP | $600,785 | | $0 | $430 | $3,008 | $17,333 | $20,099 | $20,041 | $17,974 | $17,659 | $17,452 | $16,665 | $16,860 |
| Monitoring Trust Fund (MTF) | $342 | | $0 | $0 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 |
| Rehabilitation Cash Fund (RCF) | $0 | | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Property Taxes (Land & Equipment) | $157,579 | $0 | $0 | $0 | $288 | $9,267 | $8,460 | $7,599 | $6,692 | $5,776 | $4,911 | $3,994 | $3,078 |
| MWT - Waste | $1,492 | | | $0 | $12 | $21 | $32 | $41 | $48 | $43 | $31 | $37 | $39 |
| MWT - Tailings | $2,560 | | | $0 | $39 | $92 | $109 | $108 | $84 | $80 | $74 | $62 | $71 |
| Occupational Fees | $125 | | | $0 | $3 | $3 | $4 | $4 | $4 | $4 | $4 | $4 | $4 |
| Net Income After Government Expenses | $11,276,497 | $0 | $0 | -$11,335 | $119,178 | $1,026,477 | $1,540,049 | $1,241,450 | $847,391 | $682,524 | $498,034 | $362,094 | $321,991 |
| Income Taxes | $662,527 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| **Net Income After Taxes** | $10,613,969 | | $0 | -$11,335 | $119,178 | $1,026,477 | $1,540,049 | $1,241,450 | $847,391 | $682,524 | $498,034 | $362,094 | $321,991 |

| | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 | 25 | 26 | 27 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 2037 | 2038 | 2039 | 2040 | 2041 | 2042 | 2043 | 2044 | 2045 | 2046 | 2047 | 2048 | 2049 | 2050 | 2051 | 2052 |
| | 9 | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 |
| **Revenues** | | | | | | | | | | | | | | | | |
| **Copper Concentrate** | | | | | | | | | | | | | | | | |
| Gold Revenue ($ 000) | $295,315 | $360,311 | $473,418 | $453,657 | $399,544 | $415,640 | $317,336 | $264,199 | $267,151 | $297,082 | $133,766 | $210,030 | $274,319 | $291,440 | $287,307 | $285,047 |
| Copper Revenue ($ 000) | $442,536 | $335,882 | $270,473 | $311,952 | $299,396 | $274,202 | $197,408 | $187,305 | $201,298 | $218,142 | $344,455 | $362,878 | $338,839 | $322,874 | $308,902 | $301,161 |
| Less: Treatment & Refining Charges | | | | | | | | | | | | | | | | |
| Copper Concentrate | | | | | | | | | | | | | | | | |
| Treatment Charges | -$13,585 | -$10,334 | -$8,322 | -$9,622 | -$9,195 | -$8,436 | -$6,058 | -$5,795 | -$6,193 | -$6,696 | -$10,596 | -$11,149 | -$10,409 | -$9,934 | -$9,520 | -$9,297 |
| Copper Refining Charges | -$7,504 | -$5,696 | -$4,587 | -$5,290 | -$5,077 | -$4,650 | -$3,347 | -$3,177 | -$3,413 | -$3,699 | -$5,841 | -$6,153 | -$5,745 | -$5,475 | -$5,239 | -$5,108 |
| Gold Refining Charges | -$687 | -$838 | -$1,101 | -$1,055 | -$929 | -$967 | -$738 | -$614 | -$621 | -$691 | -$311 | -$488 | -$638 | -$678 | -$668 | -$663 |
| Silver Refining Charges | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Gold Insurance Cost | -$1,181 | -$1,441 | -$1,894 | -$1,815 | -$1,598 | -$1,663 | -$1,269 | -$1,057 | -$1,069 | -$1,188 | -$535 | -$840 | -$1,097 | -$1,166 | -$1,149 | -$1,140 |
| Transportation | -$8,539 | -$6,496 | -$5,231 | -$6,048 | -$5,780 | -$5,303 | -$3,808 | -$3,642 | -$3,893 | -$4,209 | -$6,660 | -$7,008 | -$6,543 | -$6,244 | -$5,984 | -$5,844 |
| **Net Smelter Return - Copper Concentrate** | **$706,355** | **$671,388** | **$722,758** | **$741,778** | **$676,363** | **$668,824** | **$499,524** | **$437,219** | **$453,259** | **$498,741** | **$454,277** | **$547,271** | **$588,725** | **$590,817** | **$573,649** | **$564,155** |
| **Copper Cathode Revenue ($ 000)** | | | | | | | | | | | | | | | | |
| Heap Leach | $15,694 | $74,600 | $88,581 | $10,498 | $78 | $0 | $948 | $0 | $0 | $3,899 | $11,499 | $0 | $0 | $0 | $0 | $8,018 |
| Tailings Leach | $101,392 | $178,060 | $125,160 | $97,162 | $86,496 | $89,154 | $57,942 | $57,911 | $76,505 | $82,111 | $80,809 | $76,336 | $68,046 | $68,067 | $66,022 | $69,119 |
| Less: Treatment & Refining Charges | | | | | | | | | | | | | | | | |
| Transportation | -$272 | -$588 | -$497 | -$250 | -$201 | -$207 | -$137 | -$135 | -$178 | -$200 | -$215 | -$178 | -$158 | -$158 | -$154 | -$179 |
| **Gold Bullion** | $108,807 | $173,500 | $187,244 | $158,553 | $140,727 | $150,063 | $113,428 | $95,275 | $98,300 | $109,882 | $49,983 | $74,115 | $95,275 | $101,324 | $99,812 | $101,060 |
| Less: Treatment & Refining Charges | | | | | | | | | | | | | | | | |
| Gold Refining Charges | -$101 | -$161 | -$174 | -$147 | -$131 | -$140 | -$106 | -$89 | -$91 | -$102 | -$46 | -$69 | -$89 | -$94 | -$93 | -$94 |
| Gold Transportation/Insurance Cost | -$1,088 | -$1,735 | -$1,872 | -$1,586 | -$1,407 | -$1,501 | -$1,134 | -$953 | -$983 | -$1,099 | -$500 | -$741 | -$953 | -$1,013 | -$998 | -$1,011 |
| Total Revenues (NSR) | $930,786 | $1,095,064 | $1,121,198 | $1,006,008 | $901,925 | $906,194 | $670,465 | $589,229 | $626,812 | $693,234 | $595,807 | $696,734 | $750,847 | $758,943 | $738,239 | $741,068 |
| **Operating Cost** | | | | | | | | | | | | | | | | |
| Mining (Incl pit dewatering) | $132,040 | $129,557 | $119,120 | $141,778 | $160,122 | $149,088 | $150,100 | $166,540 | $161,437 | $158,146 | $131,090 | $121,417 | $109,539 | $114,997 | $149,480 | $154,891 |
| Concentrator | $110,491 | $112,487 | $113,109 | $110,930 | $109,632 | $109,394 | $109,355 | $109,388 | $109,703 | $110,191 | $109,825 | $109,538 | $108,996 | $108,996 | $112,180 | $107,938 |
| Gravity Gold Circuit | $510 | $511 | $512 | $510 | $509 | $509 | $509 | $509 | $509 | $509 | $510 | $509 | $509 | $509 | $509 | $509 |
| Agitated Tailings Leach | $55,158 | $56,714 | $57,561 | $56,542 | $54,960 | $53,447 | $53,259 | $53,268 | $54,733 | $54,031 | $54,476 | $54,555 | $53,107 | $54,784 | $53,138 | $53,146 |
| SXEW - Tailings Leach | $7,211 | $8,208 | $6,239 | $7,325 | $7,607 | $7,550 | $6,684 | $6,771 | $7,386 | $7,142 | $6,677 | $7,364 | $7,010 | $7,178 | $6,961 | $5,587 |
| Tailings Disposal | $80,291 | $85,684 | $87,509 | $82,591 | $79,212 | $78,784 | $78,228 | $78,228 | $78,228 | $79,688 | $80,741 | $78,228 | $89,178 | $89,178 | $83,976 | $89,005 |
| Heap Leach & SXEW | $8,815 | $23,141 | $27,558 | $7,673 | $4,640 | $0 | $4,884 | $0 | $0 | $5,195 | $6,663 | $0 | $0 | $0 | $0 | $6,395 |
| General Administration | $23,902 | $24,008 | $23,407 | $22,323 | $21,091 | $20,563 | $19,023 | $18,571 | $18,712 | $19,060 | $18,770 | $19,427 | $19,946 | $20,008 | $19,412 | $19,657 |
| Laboratory | $760 | $760 | $760 | $760 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 |
| Port | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,072 | $2,075 |
| Custom Duties | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 |
| Total Operating Cost | $424,186 | $446,079 | $440,784 | $435,442 | $443,458 | $425,021 | $427,727 | $438,961 | $436,395 | $439,647 | $414,437 | $396,724 | $393,972 | $401,336 | $431,343 | $442,817 |
| | | | | | | | | | | | | | | | | |
| Salvage Value | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Reclamation & Closure | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Total Production Cost | $424,186 | $446,079 | $440,784 | $435,442 | $443,458 | $425,021 | $427,727 | $438,961 | $436,395 | $439,647 | $414,437 | $396,724 | $393,972 | $401,336 | $431,343 | $442,817 |
| Operating Income | $506,600 | $648,985 | $680,413 | $570,567 | $458,467 | $481,173 | $242,738 | $150,268 | $190,416 | $253,587 | $181,370 | $300,010 | $356,874 | $357,607 | $306,896 | $298,252 |
| Depreciation | $173,846 | $175,430 | $175,494 | $175,504 | $175,550 | $177,747 | $88,935 | $17,856 | $16,860 | $16,418 | $11,672 | $11,549 | $16,542 | $20,955 | $25,411 | $28,301 |
| Total Depreciation | $173,846 | $175,430 | $175,494 | $175,504 | $175,550 | $177,747 | $88,935 | $17,856 | $16,860 | $16,418 | $11,672 | $11,549 | $16,542 | $20,955 | $25,411 | $28,301 |
| Net Income After Depreciation | $332,754 | $473,556 | $504,920 | $395,062 | $282,917 | $303,426 | $153,803 | $132,411 | $173,557 | $237,168 | $169,697 | $288,461 | $340,332 | $336,652 | $281,485 | $269,951 |
| Government Fees | | | | | | | | | | | | | | | | |
| Excise Tax | $37,231 | $43,803 | $44,848 | $40,240 | $36,077 | $36,248 | $26,819 | $23,569 | $25,072 | $27,729 | $23,832 | $27,869 | $30,034 | $30,358 | $29,530 | $29,643 |
| Local Business Tax | $0 | $0 | $22,424 | $20,120 | $18,039 | $18,124 | $13,409 | $11,785 | $12,536 | $13,865 | $11,916 | $13,935 | $15,017 | $15,179 | $14,765 | $14,821 |
| Royalty - ICC | $9,308 | $10,951 | $11,212 | $10,060 | $9,019 | $9,062 | $6,705 | $5,892 | $6,268 | $6,932 | $5,958 | $6,967 | $7,508 | $7,589 | $7,382 | $7,411 |
| Development Mining Technology (ComDev) | $10,059 | $10,531 | $10,787 | $10,587 | $10,596 | $10,340 | $8,774 | $7,792 | $7,781 | $7,902 | $7,363 | $7,199 | $7,302 | $7,504 | $8,027 | $8,264 |
| Annual EPEP | $16,737 | $17,612 | $17,401 | $17,187 | $17,511 | $16,773 | $16,882 | $17,331 | $17,228 | $17,358 | $16,350 | $15,642 | $15,531 | $15,826 | $17,026 | $17,485 |
| Monitoring Trust Fund (MTF) | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $11 |
| Rehabilitation Cash Fund (RCF) | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Property Taxes (Land & Equipment) | $2,208 | $2,185 | $2,171 | $2,162 | $2,153 | $2,143 | $2,139 | $2,134 | $2,130 | $2,130 | $2,130 | $2,130 | $3,330 | $4,410 | $5,370 | $6,210 |
| MWT - Waste | $66 | $48 | $41 | $62 | $68 | $76 | $74 | $68 | $77 | $75 | $59 | $62 | $62 | $60 | $62 | $61 |
| MWT - Tailings | $65 | $87 | $93 | $66 | $60 | $59 | $59 | $59 | $59 | $60 | $63 | $59 | $59 | $59 | $59 | $61 |
| Occupational Fees | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $4 |
| Net Income After Government Expenses | $257,065 | $388,324 | $395,928 | $294,564 | $189,381 | $210,586 | $78,930 | $63,766 | $102,390 | $161,102 | $102,012 | $214,584 | $261,475 | $255,653 | $199,250 | $185,982 |
| Income Taxes | $0 | $0 | $19,230 | $14,728 | $9,469 | $10,529 | $3,946 | $3,188 | $5,120 | $8,055 | $5,101 | $10,729 | $65,369 | $63,913 | $49,813 | $46,495 |
| **Net Income After Taxes** | $257,065 | $388,324 | $376,699 | $279,835 | $179,912 | $200,057 | $74,983 | $60,578 | $97,271 | $153,047 | $96,912 | $203,855 | $196,106 | $191,740 | $149,438 | $139,486 |

| | 28 | 29 | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | 39 | 40 | 41 | 42 | 43 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 2053 | 2054 | 2055 | 2056 | 2057 | 2058 | 2059 | 2060 | 2061 | 2062 | 2063 | 2064 | 2065 | 2066 | 2067 | 2068 |
| | 25 | 26 | 27 | 28 | 29 | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | 39 | 40 |
| **Revenues** | | | | | | | | | | | | | | | | |
| **Copper Concentrate** | | | | | | | | | | | | | | | | |
| Gold Revenue ($ 000) | $237,173 | $120,424 | $147,552 | $234,672 | $310,620 | $240,921 | $110,060 | $101,444 | $138,723 | $126,651 | $104,423 | $124,163 | $169,328 | $102,153 | $0 | $0 |
| Copper Revenue ($ 000) | $300,645 | $320,860 | $324,846 | $346,749 | $288,971 | $207,898 | $187,300 | $152,577 | $161,102 | $217,354 | $267,481 | $261,673 | $309,602 | $303,686 | $0 | $0 |
| Less: Treatment & Refining Charges | | | | | | | | | | | | | | | | |
| Copper Concentrate | | | | | | | | | | | | | | | | |
| Treatment Charges | -$9,234 | -$9,888 | -$10,010 | -$10,700 | -$8,907 | -$6,379 | -$5,774 | -$4,703 | -$4,964 | -$6,698 | -$8,224 | -$8,023 | -$9,519 | -$9,335 | $0 | $0 |
| Copper Refining Charges | -$5,098 | -$5,441 | -$5,509 | -$5,881 | -$4,901 | -$3,525 | -$3,176 | -$2,588 | -$2,732 | -$3,686 | -$4,536 | -$4,437 | -$5,250 | -$5,150 | $0 | $0 |
| Gold Refining Charges | -$552 | -$280 | -$343 | -$546 | -$722 | -$560 | -$256 | -$236 | -$323 | -$295 | -$243 | -$289 | -$394 | -$238 | $0 | $0 |
| Silver Refining Charges | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Gold Insurance Cost | -$949 | -$482 | -$590 | -$939 | -$1,242 | -$964 | -$440 | -$406 | -$555 | -$507 | -$418 | -$497 | -$677 | -$409 | $0 | $0 |
| Transportation | -$5,804 | -$6,215 | -$6,292 | -$6,726 | -$5,599 | -$4,010 | -$3,629 | -$2,956 | -$3,120 | -$4,210 | -$5,169 | -$5,043 | -$5,983 | -$5,868 | $0 | $0 |
| **Net Smelter Return - Copper Concentrate** | **$516,181** | **$418,978** | **$449,653** | **$556,629** | **$578,221** | **$433,382** | **$284,083** | **$243,133** | **$288,131** | **$328,611** | **$353,315** | **$367,547** | **$457,107** | **$384,840** | **$0** | **$0** |
| **Copper Cathode Revenue ($ 000)** | | | | | | | | | | | | | | | | |
| Heap Leach | $12,835 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Tailings Leach | $75,758 | $76,320 | $76,314 | $76,279 | $59,857 | $51,244 | $43,562 | $44,169 | $44,720 | $39,087 | $53,042 | $56,985 | $69,276 | $57,780 | $0 | $0 |
| Less: Treatment & Refining Charges | | | | | | | | | | | | | | | | |
| Transportation | -$206 | -$177 | -$177 | -$177 | -$139 | -$119 | -$101 | -$103 | -$104 | -$91 | -$123 | -$133 | -$161 | -$134 | $0 | $0 |
| **Gold Bullion** | $85,988 | $45,369 | $54,443 | $83,177 | $107,374 | $83,897 | $40,577 | $38,860 | $51,465 | $44,386 | $38,327 | $46,363 | $61,711 | $37,923 | $0 | $0 |
| Less: Treatment & Refining Charges | | | | | | | | | | | | | | | | |
| Gold Refining Charges | -$80 | -$42 | -$51 | -$77 | -$100 | -$78 | -$38 | -$36 | -$48 | -$41 | -$36 | -$43 | -$57 | -$35 | $0 | $0 |
| Gold Transportation/Insurance Cost | -$860 | -$454 | -$544 | -$832 | -$1,074 | -$839 | -$406 | -$389 | -$515 | -$444 | -$383 | -$464 | -$617 | -$379 | $0 | $0 |
| Total Revenues (NSR) | $689,616 | $539,994 | $579,637 | $714,999 | $744,138 | $567,487 | $367,678 | $325,636 | $383,651 | $411,508 | $444,141 | $470,256 | $587,258 | $479,994 | $0 | $0 |
| **Operating Cost** | | | | | | | | | | | | | | | | |
| Mining (Incl pit dewatering) | $155,926 | $117,395 | $66,479 | $70,270 | $68,303 | $82,707 | $44,623 | $32,674 | $31,440 | $30,558 | $29,380 | $28,247 | $26,859 | $25,692 | $0 | $0 |
| Concentrator | $110,074 | $109,088 | $109,088 | $109,088 | $109,088 | $109,088 | $109,089 | $109,089 | $109,087 | $109,089 | $109,090 | $109,087 | $109,088 | $83,529 | $0 | $0 |
| Gravity Gold Circuit | $509 | $509 | $509 | $509 | $509 | $509 | $509 | $509 | $509 | $509 | $509 | $509 | $509 | $506 | $0 | $0 |
| Agitated Tailings Leach | $53,512 | $53,012 | $53,008 | $52,983 | $53,046 | $53,394 | $53,483 | $53,193 | $53,183 | $53,123 | $53,071 | $53,076 | $54,299 | $40,535 | $0 | $0 |
| SXEW - Tailings Leach | $5,576 | $1,925 | $1,925 | $1,924 | $1,510 | $1,293 | $1,099 | $1,114 | $1,128 | $986 | $1,338 | $1,437 | $1,747 | $1,457 | $0 | $0 |
| Tailings Disposal | $89,735 | $88,811 | $88,811 | $88,811 | $88,811 | $88,811 | $88,812 | $88,812 | $88,810 | $88,812 | $88,813 | $88,810 | $88,811 | $75,616 | $0 | $0 |
| Heap Leach & SXEW | $6,925 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| General Administration | $19,590 | $19,168 | $19,640 | $20,331 | $20,324 | $19,204 | $18,648 | $18,608 | $18,563 | $18,651 | $18,791 | $18,874 | $19,408 | $19,084 | $0 | $0 |
| Laboratory | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $677 | $280 | $0 | $0 |
| Port | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,071 | $2,030 | $1,711 | $0 | $0 |
| Custom Duties | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $2,937 | $0 | $0 |
| Total Operating Cost | $447,534 | $395,594 | $345,146 | $349,602 | $347,277 | $360,691 | $321,949 | $309,685 | $308,406 | $307,413 | $306,678 | $305,726 | $306,366 | $251,347 | $0 | $0 |
| | | | | | | | | | | | | | | | | |
| Salvage Value | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Reclamation & Closure | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $18,614 | $18,614 | $18,614 | $18,614 | $0 | $0 |
| Total Production Cost | $447,534 | $395,594 | $345,146 | $349,602 | $347,277 | $360,691 | $321,949 | $309,685 | $308,406 | $307,413 | $325,292 | $324,340 | $324,981 | $269,961 | $0 | $0 |
| Operating Income | $242,082 | $144,399 | $234,491 | $365,396 | $396,861 | $206,796 | $45,729 | $15,951 | $75,245 | $104,096 | $118,849 | $145,916 | $262,278 | $210,033 | $0 | $0 |
| Depreciation | $32,074 | $32,395 | $32,800 | $33,210 | $31,513 | $30,700 | $31,195 | $31,826 | $32,606 | $31,776 | $32,396 | $28,758 | $26,258 | $26,258 | $0 | $0 |
| Total Depreciation | $32,074 | $32,395 | $32,800 | $33,210 | $31,513 | $30,700 | $31,195 | $31,826 | $32,606 | $31,776 | $32,396 | $28,758 | $26,258 | $26,258 | $0 | $0 |
| Net Income After Depreciation | $210,008 | $112,005 | $201,691 | $332,187 | $365,348 | $176,095 | $14,534 | -$15,875 | $42,639 | $72,320 | $86,453 | $117,158 | $236,020 | $183,775 | $0 | $0 |
| Government Fees | | | | | | | | | | | | | | | | |
| Excise Tax | $27,585 | $21,600 | $23,185 | $28,600 | $29,766 | $22,699 | $14,707 | $13,025 | $15,346 | $16,460 | $17,766 | $18,810 | $23,490 | $19,200 | $0 | $0 |
| Local Business Tax | $13,792 | $10,800 | $11,593 | $14,300 | $14,883 | $11,350 | $7,354 | $6,513 | $7,673 | $8,230 | $8,883 | $9,405 | $11,745 | $9,600 | $0 | $0 |
| Royalty - ICC | $6,896 | $5,400 | $5,796 | $7,150 | $7,441 | $5,675 | $3,677 | $3,256 | $3,837 | $4,115 | $4,441 | $4,703 | $5,873 | $4,800 | $0 | $0 |
| Development Mining Technology (ComDev) | $8,348 | $7,377 | $6,632 | $6,853 | $6,800 | $6,776 | $5,951 | $5,710 | $5,765 | $5,781 | $5,827 | $5,785 | $5,898 | $4,923 | $0 | $0 |
| Annual EPEP | $17,674 | $15,596 | $13,578 | $13,757 | $13,664 | $14,200 | $12,651 | $12,160 | $12,109 | $12,069 | $12,040 | $12,002 | $12,029 | $9,857 | $0 | $0 |
| Monitoring Trust Fund (MTF) | $11 | $11 | $11 | $11 | $11 | $11 | $11 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Rehabilitation Cash Fund (RCF) | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Property Taxes (Land & Equipment) | $6,930 | $6,410 | $5,882 | $5,346 | $4,802 | $4,298 | $3,915 | $3,653 | $3,514 | $3,497 | $3,492 | $3,502 | $3,533 | $3,613 | $0 | $0 |
| MWT - Waste | $66 | $43 | $11 | $9 | $11 | $23 | $5 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| MWT - Tailings | $62 | $59 | $59 | $59 | $59 | $59 | $59 | $59 | $59 | $59 | $59 | $59 | $59 | $52 | $0 | $0 |
| Occupational Fees | $4 | $4 | $4 | $4 | $4 | $4 | $4 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Net Income After Government Expenses | $128,640 | $44,705 | $134,940 | $256,099 | $287,910 | $111,001 | -$33,798 | -$60,252 | -$5,663 | $22,109 | $33,945 | $62,893 | $173,394 | $131,732 | $0 | $0 |
| Income Taxes | $32,160 | $11,176 | $33,735 | $64,025 | $71,977 | $27,750 | $0 | $0 | $0 | $5,527 | $8,486 | $15,723 | $43,348 | $32,933 | $0 | $0 |
| **Net Income After Taxes** | $96,480 | $33,529 | $101,205 | $192,075 | $215,932 | $83,250 | -$33,798 | -$60,252 | -$5,663 | $16,581 | $25,459 | $47,170 | $130,045 | $98,799 | $0 | $0 |

| | Contract | 0 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 | 10 | 11 | 12 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | 2025 | 2026 | 2027 | 2028 | 2029 | 2030 | 2031 | 2032 | 2033 | 2034 | 2035 | 2036 | 2037 |
| | Total | -4 | -3 | -2 | -1 | 1 | 2 | 3 | 4 | 5 | 6 | 7 | 8 | 9 |
| **Cash Flow** | | | | | | | | | | | | | | |
| Operating Income & Government Expenses | $14,448,693 | $0 | $0 | -$11,335 | $209,357 | $1,187,865 | $1,702,517 | $1,404,413 | $1,016,930 | $852,816 | $668,638 | $533,285 | $493,726 | $430,911 |
| Working Capital | | | | | | | | | | | | | | |
| Account Receivable (60 days) | $0 | $0 | $0 | $0 | -$77,555 | -$138,465 | -$76,230 | $39,033 | $57,524 | $22,516 | $24,743 | $20,397 | $4,589 | $8,692 |
| Accounts Payable (30 days) | $0 | $0 | $0 | $883 | $12,314 | $22,893 | $5,683 | -$118 | -$4,249 | -$647 | -$425 | -$1,617 | $402 | -$254 |
| Inventory - Parts, Supplies | $0 | $0 | $0 | $0 | -$17,500 | -$17,500 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Total Working Capital | $0 | $0 | $0 | $883 | -$82,741 | -$133,071 | -$70,547 | $38,915 | $53,276 | $21,869 | $24,319 | $18,780 | $4,991 | $8,438 |
| Contingent Liability and Reclamation Fund | $0 | | | | | $15,413 | $13,402 | $11,169 | $9,680 | $7,446 | $5,957 | $4,691 | $3,723 | $1,489 |
| Capital & Opex VAT - Paid | -$1,202,399 | $0 | -$31,706 | -$88,691 | -$87,723 | -$42,003 | -$35,036 | -$32,574 | -$39,922 | -$28,779 | -$26,830 | -$26,087 | -$26,704 | -$28,671 |
| Capital & Opex VAT - Recovered (80%) | $961,919 | $0 | $0 | $0 | $25,365 | $70,953 | $70,179 | $33,602 | $28,029 | $26,059 | $31,938 | $23,023 | $21,464 | $20,869 |
| Capital Expenditures | | | | | | | | | | | | | | |
| Initial Capital | | | | | | | | | | | | | | |
| Mine Development | $151,707 | $0 | $0 | $46,726 | $104,981 | $0 | | | | | | | | |
| Process Plant | $1,914,019 | $0 | $191,402 | $861,309 | $765,608 | $95,701 | | | | | | | | |
| Power Plant | $51,877 | $0 | $15,563 | $20,751 | $15,563 | $0 | | | | | | | | |
| Port | $60,000 | $0 | $24,000 | $36,000 | $0 | $0 | | | | | | | | |
| Owner's Cost | $196,181 | $0 | $98,091 | $58,854 | $39,236 | $0 | | | | | | | | |
| Sustaining Capital | | | | | | | | | | | | | | |
| Mine Equipment | $13,507 | $0 | $0 | $0 | $0 | $7,826 | $5,681 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Pit Diversions | $33,931 | $0 | $0 | $0 | $0 | $1,067 | $1,562 | $3,861 | $872 | $3,244 | $1,626 | $0 | $1,769 | $778 |
| Dry Stack Tailings | $180,886 | $0 | $0 | $0 | $0 | $27,376 | $0 | $0 | $76,997 | $0 | $0 | $0 | $0 | $26,268 |
| Southwest VRMA | $37,468 | $0 | $0 | $0 | $0 | $0 | $8,946 | $529 | $19,670 | $60 | $0 | $0 | $0 | $0 |
| Tailings Stacking Conveyor | $8,681 | $0 | $0 | $0 | $0 | $0 | $0 | $2,894 | $0 | $0 | $2,894 | $0 | $0 | $2,894 |
| Process and Tailings Susex (Years 21 to 38) | $440,000 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Site General | $71,912 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $7,981 | $163 | $8,810 | $6,391 | $1,715 |
| Heap Leach | $12,027 | $0 | $0 | $0 | $0 | $10,780 | $0 | $140 | $1,107 | $0 | $0 | $0 | $0 | $0 |
| Total Capital Expenditures | $3,172,196 | $0 | $329,056 | $1,023,639 | $925,388 | $142,750 | $16,189 | $7,424 | $98,646 | $11,285 | $4,683 | $8,810 | $8,160 | $31,655 |
| Cash Flow before Taxes | $11,036,017 | $0 | -$360,762 | -$1,122,783 | -$861,130 | $925,581 | $1,637,521 | $1,425,764 | $949,987 | $853,233 | $687,425 | $535,500 | $481,594 | $398,403 |
| Cummulative Cash Flow before Taxes | | $0 | -$360,762 | -$1,483,545 | -$2,344,675 | -$1,419,094 | $218,427 | $1,644,191 | $2,594,178 | $3,447,411 | $4,134,836 | $4,670,336 | $5,151,930 | $5,550,333 |
| | | | | | | $1 | $1 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Taxes | | | | | | | | | | | | | | |
| Income Taxes | $662,527 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Cash Flow after Taxes | $10,373,490 | $0 | -$360,762 | -$1,122,783 | -$861,130 | $925,581 | $1,637,521 | $1,425,764 | $949,987 | $853,233 | $687,425 | $535,500 | $481,594 | $398,403 |
| Cummulative Cash Flow after Taxes | | $0 | -$360,762 | -$1,483,545 | -$2,344,675 | -$1,419,094 | $218,427 | $1,644,191 | $2,594,178 | $3,447,411 | $4,134,836 | $4,670,336 | $5,151,930 | $5,550,333 |
| | | | | | | 1.0 | 0.9 | - | - | - | - | - | - | - |

| | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 | 25 | 26 | 27 | 28 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 2038 | 2039 | 2040 | 2041 | 2042 | 2043 | 2044 | 2045 | 2046 | 2047 | 2048 | 2049 | 2050 | 2051 | 2052 | 2053 |
| | 10 | 11 | 12 | 13 | 14 | 15 | 16 | 17 | 18 | 19 | 20 | 21 | 22 | 23 | 24 | 25 |
| **Cash Flow** | | | | | | | | | | | | | | | | |
| Operating Income & Government Expenses | $563,754 | $571,422 | $470,068 | $364,931 | $388,332 | $167,864 | $81,623 | $119,250 | $177,520 | $113,685 | $226,134 | $278,017 | $276,608 | $224,661 | $214,282 | $160,714 |
| Working Capital | | | | | | | | | | | | | | | | |
| Account Receivable (60 days) | -$20,253 | -$3,222 | $14,201 | $12,832 | -$526 | $29,062 | $10,015 | -$4,633 | -$8,189 | $12,011 | -$12,443 | -$6,671 | -$998 | $2,553 | -$349 | $6,343 |
| Accounts Payable (30 days) | $1,799 | -$435 | -$439 | $659 | -$1,515 | $222 | $923 | -$211 | $267 | -$2,072 | -$1,456 | -$226 | $605 | $2,466 | $943 | $388 |
| Inventory - Parts, Supplies | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $35,000 | $0 | $0 | $0 |
| Total Working Capital | -$18,454 | -$3,657 | $13,762 | $13,491 | -$2,042 | $29,285 | $10,939 | -$4,844 | -$7,922 | $9,939 | -$13,899 | -$6,898 | $34,607 | $5,019 | $594 | $6,731 |
| Contingent Liability and Reclamation Fund | $745 | $745 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Capital & Opex VAT - Paid | -$30,613 | -$28,316 | -$25,153 | -$24,298 | -$28,058 | -$26,259 | -$23,957 | -$23,975 | -$24,202 | -$27,528 | -$25,019 | -$25,014 | -$24,466 | -$23,764 | -$24,466 | -$24,841 |
| Capital & Opex VAT - Recovered (80%) | $21,363 | $22,937 | $24,490 | $22,653 | $20,123 | $19,438 | $22,447 | $21,007 | $19,166 | $19,180 | $19,362 | $22,022 | $20,015 | $20,011 | $19,573 | $19,011 |
| Capital Expenditures | | | | | | | | | | | | | | | | |
| Initial Capital | | | | | | | | | | | | | | | | |
| Mine Development | | | | | | | | | | | | | | | | |
| Process Plant | | | | | | | | | | | | | | | | |
| Power Plant | | | | | | | | | | | | | | | | |
| Port | | | | | | | | | | | | | | | | |
| Owner's Cost | | | | | | | | | | | | | | | | |
| Sustaining Capital | | | | | | | | | | | | | | | | |
| Mine Equipment | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Pit Diversions | $6,983 | $833 | $0 | $549 | $420 | $6,111 | $277 | $711 | $642 | $1,394 | $1,230 | $0 | $0 | $0 | $0 | $0 |
| Dry Stack Tailings | $0 | $0 | $0 | $0 | $24,311 | $0 | $0 | $0 | $0 | $25,935 | $0 | $0 | $0 | $0 | $0 | $0 |
| Southwest VRMA | $8,263 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Tailings Stacking Conveyor | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Process and Tailings Susex (Years 21 to 38) | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $75,000 | $75,000 | $75,000 | $75,000 | $75,000 |
| Site General | $8,511 | $129 | $163 | $129 | $8,223 | $14,407 | $1,681 | $529 | $163 | $129 | $8,211 | $4,577 | $0 | $0 | $0 | $0 |
| Heap Leach | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Total Capital Expenditures | $23,757 | $962 | $163 | $678 | $32,954 | $20,518 | $1,959 | $1,240 | $805 | $27,458 | $9,441 | $79,577 | $75,000 | $75,000 | $75,000 | $75,000 |
| Cash Flow before Taxes | $511,549 | $560,679 | $483,004 | $376,098 | $345,401 | $169,811 | $89,093 | $110,198 | $163,757 | $87,818 | $197,137 | $188,552 | $231,765 | $150,927 | $134,983 | $86,615 |
| Cummulative Cash Flow before Taxes | $6,061,882 | $6,622,561 | $7,105,565 | $7,481,664 | $7,827,065 | $7,996,876 | $8,085,968 | $8,196,166 | $8,359,923 | $8,447,741 | $8,644,878 | $8,833,430 | $9,065,194 | $9,216,121 | $9,351,104 | $9,437,720 |
| | $0 | $0 | $0 | | | | | | | | | | | | | |
| Taxes | | | | | | | | | | | | | | | | |
| Income Taxes | $0 | $19,230 | $14,728 | $9,469 | $10,529 | $3,946 | $3,188 | $5,120 | $8,055 | $5,101 | $10,729 | $65,369 | $63,913 | $49,813 | $46,495 | $32,160 |
| Cash Flow after Taxes | $511,549 | $541,450 | $468,276 | $366,629 | $334,872 | $165,864 | $85,904 | $105,078 | $155,702 | $82,717 | $186,408 | $123,183 | $167,851 | $101,115 | $88,488 | $54,455 |
| Cummulative Cash Flow after Taxes | $6,061,882 | $6,603,332 | $7,071,608 | $7,438,237 | $7,773,109 | $7,938,973 | $8,024,877 | $8,129,956 | $8,285,657 | $8,368,375 | $8,554,783 | $8,677,965 | $8,845,817 | $8,946,932 | $9,035,419 | $9,089,874 |
| | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |

| | 29 | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | 39 | 40 | 41 | 42 | 43 |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | 2054 | 2055 | 2056 | 2057 | 2058 | 2059 | 2060 | 2061 | 2062 | 2063 | 2064 | 2065 | 2066 | 2067 | 2068 |
| | 26 | 27 | 28 | 29 | 30 | 31 | 32 | 33 | 34 | 35 | 36 | 37 | 38 | 39 | 40 |
| **Cash Flow** | | | | | | | | | | | | | | | |
| Operating Income & Government Expenses | $77,100 | $167,740 | $289,309 | $319,423 | $141,701 | -$2,603 | -$28,426 | $26,943 | $53,884 | $66,342 | $91,651 | $199,652 | $157,989 | $0 | $0 |
| Working Capital | | | | | | | | | | | | | | | |
| Account Receivable (60 days) | $18,447 | -$4,888 | -$16,688 | -$3,593 | $21,779 | $24,634 | $5,183 | -$7,153 | -$3,435 | -$4,023 | -$3,220 | -$14,425 | $13,224 | $59,177 | $0 |
| Accounts Payable (30 days) | -$4,269 | -$4,146 | $366 | -$191 | $1,103 | -$3,184 | -$1,008 | -$105 | -$82 | -$60 | -$78 | $53 | -$4,522 | -$20,659 | $0 |
| Inventory - Parts, Supplies | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Total Working Capital | $14,178 | -$9,034 | -$16,322 | -$3,784 | $22,881 | $21,450 | $4,175 | -$7,258 | -$3,516 | -$4,084 | -$3,298 | -$14,372 | $8,702 | $38,519 | $0 |
| Contingent Liability and Reclamation Fund | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | -$18,614 | -$18,614 | -$18,614 | -$18,614 | $0 | $0 |
| Capital & Opex VAT - Paid | -$23,356 | -$23,412 | -$23,495 | -$23,445 | -$23,284 | -$23,195 | -$23,192 | -$23,188 | -$23,182 | -$23,241 | -$23,262 | -$23,362 | -$18,129 | $0 | $0 |
| Capital & Opex VAT - Recovered (80%) | $19,573 | $19,873 | $18,685 | $18,730 | $18,796 | $18,756 | $18,627 | $18,556 | $18,554 | $18,551 | $18,546 | $18,593 | $18,610 | $18,690 | $14,503 |
| Capital Expenditures | | | | | | | | | | | | | | | |
| Initial Capital | | | | | | | | | | | | | | | |
| Mine Development | | | | | | | | | | | | | | | |
| Process Plant | | | | | | | | | | | | | | | |
| Power Plant | | | | | | | | | | | | | | | |
| Port | | | | | | | | | | | | | | | |
| Owner's Cost | | | | | | | | | | | | | | | |
| Sustaining Capital | | | | | | | | | | | | | | | |
| Mine Equipment | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Pit Diversions | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Dry Stack Tailings | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Southwest VRMA | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Tailings Stacking Conveyor | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Process and Tailings Susex (Years 21 to 38) | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $0 | $0 |
| Site General | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Heap Leach | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 | $0 |
| Total Capital Expenditures | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $5,000 | $0 | $0 |
| Cash Flow before Taxes | $82,495 | $150,166 | $263,176 | $305,924 | $155,095 | $9,408 | -$33,815 | $10,053 | $40,739 | $71,182 | $97,251 | $194,125 | $180,787 | $57,208 | $14,503 |
| Cummulative Cash Flow before Taxes | $9,520,214 | $9,670,381 | $9,933,557 | $10,239,481 | $10,394,575 | $10,403,983 | $10,370,168 | $10,380,221 | $10,420,961 | $10,492,142 | $10,589,393 | $10,783,518 | $10,964,305 | $11,021,513 | $11,036,017 |
| Taxes | | | | | | | | | | | | | | | |
| Income Taxes | $11,176 | $33,735 | $64,025 | $71,977 | $27,750 | $0 | $0 | $0 | $5,527 | $8,486 | $15,723 | $43,348 | $32,933 | $0 | $0 |
| Cash Flow after Taxes | $71,318 | $116,431 | $199,151 | $233,946 | $127,344 | $9,408 | -$33,815 | $10,053 | $35,212 | $62,695 | $81,528 | $150,776 | $147,854 | $57,208 | $14,503 |
| Cummulative Cash Flow after Taxes | $9,161,193 | $9,277,624 | $9,476,775 | $9,710,722 | $9,838,066 | $9,847,474 | $9,813,659 | $9,823,712 | $9,858,924 | $9,921,620 | $10,003,148 | $10,153,924 | $10,301,778 | $10,358,986 | $10,373,490 |
| | - | - | - | - | - | - | - | - | - | - | - | - | - | - | - |

| Economic Indicators | | Before-Taxes | After-Taxes |
|---|---|---|---|
| NPV @ 0% | 0% | $ 11,036,017 | $ 10,373,490 |
| NPV @ 5% | 5% | $ 5,467,160 | $ 5,294,186 |
| NPV @ 7% | 7% | $ 4,287,508 | $ 4,181,607 |
| NPV @ 8% | 8% | $ 3,815,112 | $ 3,731,476 |
| NPV @ 10% | 10% | $ 3,040,566 | $ 2,987,467 |
| IRR | % | 34.2% | 34.2% |
| Payback | Years | 1.9 | 1.9 |

# 23 ADJACENT PROPERTIES

The Kingking Project will be primarily located at the west-central section of the Municipality of Pantukan in the Province of Davao de Oro. There are currently no active mineral projects that are adjacent to the Kingking Deposit.

There is significant artisanal mining for gold in the Kingking Mineral Property Area and in adjacent mining tenements surrounding the Kingking claims. These areas are north and northeast of the Kingking Deposit. To the best of the author'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 have 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 $2.82 per pound of copper over LOM and $1.23 per pound of copper in the initial 5 years
- Low-cost heap leach production starting earlier than concentrator and generating early cash flow
- Ten-year income tax holiday

Table 25-1 below outlines the base case key economic results at the following metal prices:

- Gold = $2,150 per troy ounce
- Copper = $4.30 per pound

**Table 25-1: Economic Indicators After-Tax**

| Economic Indicator | After-Tax |
|---|---:|
| NPV @ 0% ($000) | $10,373,427 |
| NPV @ 5% ($000) | $5,294,176 |
| NPV @ 7% ($000) | $4,181,602 |
| NPV @ 10% ($000) | $2,987,465 |
| IRR % | 34.2% |
| Payback (Years) | 1.9 |

The results further indicate that if the prices in the future drop by 20% from the base case assumptions shown above ($1,720/oz gold and $3.44/lb copper), the Kingking Project would still produce positive economics ($1.8 billion NPV@7% and 22% 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 $2.4 billion by 20% to $2.8 billion, the Kingking Project would still remain robust at $3.8 billion NPV@7% and 28.1% IRR.

## 25.2 EXPLORATION AND GEOLOGY

The interpretation of the exploration work performed to date is that the Kingking deposit is a significant copper-gold porphyry system with the potential to become an economic project. The drilling performed through 2012 has been used to develop an NI 43-101 compliant mineral resource for the deposit, as presented in Section 14.

Fourteen (14) drill holes were completed in 2011 with a total depth of 5,987 meters. Nine of these holes are in the deposit area and were incorporated into the updated resource model. The other five holes were for geotechnical and hydrological study outside the deposit. New information from these drill holes confirmed results from the historical Benguet and Echo Bay drilling.

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 Kingking Project, based on a review of data recently recovered from historical 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 drill hole (a 237-meter-deep core hole that intercepted 81 continuous meters averaging 0.44% total copper and 0.34 g/t gold) confirm wide intervals of porphyry Au-Cu mineralization that were intersected in the historic initial drill tests of two areas located 1 km and 4 km northeast of the Kingking deposit.

## 25.3 MINING

The results of this Technical Report indicate that the Kingking Project has the potential to become an economic producer of copper and gold.

This study has developed a proven and probable mineral reserve amenable to milling of 848.9 million ore tonnes at 0.26% total copper and 0.36 g/t gold. This amounts to 4.84 billion pounds of contained copper and 9.77 million ounces of contained gold. A proven and probable mineral reserve amenable to heap leaching amounts to 110.5 million tonnes at 0.23% copper for 555 million pounds of contained copper.

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

The mining methods proposed for the Kingking Project are conventional open pit methods for bulk mining. There are no significant technical challenges to mining at Kingking.

One of the goals of this study was to define an NI 43-101 compliant mineral reserve for the Kingking Project. The study has met this goal.

## 25.4 TAILING AND GEOTECHNICAL

The following subsections, 25.4.1 through 25.4.2, are maintained in their entirety from the PFS Technical Report for the Kingking Project with an effective date of February 25, 2013. All findings and recommendations remain in effect.

### 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 $CaCO_3$/tonne ANP and 1.7 kg $CaCO_3$/tonne AGP. By comparison, sulfide tailing have 12 kg $CaCO_3$/tonne ANP and 4.2 kg $CaCO_3$/tonne 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 are anticipated to be neutralized prior to placement in the TSF. Further, the dewatered tailing are 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) onedimensional 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 percent. 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 Dry-stack 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 Kingking 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 some of the 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. Additional copper recovery from heap leaching could be achieved by adding a chloride cure, with NaCl, to the conventional acid cure originally planned. A significant number of South American copper mines have proven the chloride cure is effective at the commercial stage in the past 12 years.

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 VRMA

The following subsections, 25.6.1 through 25.6.2, are maintained in their entirety from the PFS Technical Report for the Kingking Project with an effective date of February 25, 2013. All findings and recommendations remain in effect.

### 25.6.1 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 tonne (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), while for 37% the ratio is greater than 3 and not anticipated to produce acidic drainage. The remaining approximately 20% are uncertain. The relative proportions of site rock types anticipated to be excavated produced, as estimated by a block model, suggest that approximately 35% of valueless rock overall 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).

Field barrels, which tested only material collected within 10 meters of ground surface, samples with a Net Neutralization Potential (NNP; ANP-AGP) less than zero and an NPR <1 ( 3 of 7 barrels) produced acidic leachate (pH 2.4 - 3.8) with relatively high TDS and metals. Remaining barrels with NPR > 1 and NNP > 0 produced mildly acidic leachate (pH 5.6 – 6.8) with limited TDS.

### 25.6.2 VRMA Design Conclusions

Based on the stability analyses performed on the maximum cross-section (refer to 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.

# 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 are recommended in the next phase in collaboration with the industry experts in the US, Europe, and Asia.

The engineering cost for geoscience and mining studies, feasibility study (FS) through to 15% of engineering, is estimated at $21.3 million, as shown in Table 26-1.

**Table 26-1: Summary of Cost to Complete a Feasibility Study and Basic Engineering**

| Cost Category | Estimate ($ millions) |
|---|---:|
| Exploration and Geology | 2.5 |
| Mining | 5.0 |
| TSF, VRMA, Pit Diversions & Dewatering | 1.5 |
| Metallurgical Testing | 0.2 |
| FS and Basic Engineering | 11.6 |
| Due Diligence | 0.5 |
| **Total** | **21.3** |

## 26.2 EXPLORATION AND GEOLOGY

The geologic interpretation of the Kingking 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). The QP for this section 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 the re-logging of select drill core intervals.
- The lithologic model has to be reviewed and updated and 3-D solids developed to update the resource model.
- As part of the upcoming feasibility study, the resulting structural model should be incorporated in the mineral resource and mineral reserve model updates.

It is recommended that additional drilling be completed for the following:

- Increase confidence in the current indicated resource estimate in areas where current drill hole 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 effort contributes to the potential upgrade of a portion of the 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) drill holes are budgeted for additional drill hole information requested above. This should amount to approximately 4,700 meters of drilling.

Third party consultant oversight and the drilling program has an estimated cost of $2.5 million.

## 26.3 MINING

It is recommended that the pit slope angle study be optimized and 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.

The higher level of detail from these updates and programs may eliminate some of the conservative assumptions made thus far and allow the slope angles for the pit walls to increase from those used in the preliminary feasibility study. This would improve project economics by decreasing the waste to ore ratio for the mine, thereby reducing waste mining costs and potentially increasing the mine reserve.

It is also recommended to perform some early mining and open up some faces and examine them in regard to slope angles versus stability.

The mining models and studies have an estimated cost of $5 million.

## 26.4 TSF, VRMA, PIT DIVERSIONS AND DEWATERING

The following subsections, 26.4.1 through 26.4.6, are maintained in their entirety from the PFS Technical Report for the Kingking Project with an effective date of February 25, 2013. All findings and recommendations remain in effect.

### 26.4.1 Tailing Test work

The test work 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 test work, bench-scale filtration testing, extended moisture density work, Soil Water Characteristic Curve (SWCC) testing, variable moisture test work, and possible field compaction trial.

Additional tailing materials are required to facilitate the tailing geotechnical test work to support the feasibility level designs. The tailing samples will need to span the range of anticipated ore lithologies.

### 26.4.2 TSF Geotechnical Investigation Program

The first phase geotechnical field investigation for the proposed Southwest Tailing Dry Stack facility is anticipated to include drilling of eight (8) geotechnical drill holes, 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 Drill Holes

AMEC has selected drill hole locations within the footprint of the proposed Southwest Tailing Dry Stack facility to support the feasibility-level design. The primary objectives of the geotechnical drill holes are to provide empirical strength data for the overburden materials, provide depths to bedrock at the drill hole 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 dry stack facility, and supplement previous fieldwork and laboratory testing information. By achieving these objectives, the drill holes will help to provide subsurface data for design decisions within the proposed Southwest Tailing Dry Stack facility footprint.

A total of eight (8) drill holes 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 drill hole.

#### 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 dry stack 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 (7) geotechnical drill holes and seven (7) test pits excavations will be required to investigate the foundation of the VRMA area.

It is anticipated that drill holes 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 drill hole, 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 stormwater 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 hydrogeologic 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 drilling and testing programs have an estimated cost of $1.5 million:

- Drilling of approximately 2,000 linear meters at various sites
- Cone Penetration Testing (CPT) at the Port, Tailing and VRMA facilities
- Performing subsurface geophysics at the Tailing and VRMA facilities
- Completion of test pits at various sites
- Laboratory testing of collected samples
- Completion of pumping tests at the site of the pit

### 26.5 METALLURGY

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 may be possible from the sale of magnetite concentrate.

Actual test work, using a laboratory-scale Knelson or Falcon concentrator, is necessary to confirm the gravity concentration test results conducted in 2013.

Mineralogical studies of the Kingking deposit indicate a large presence of pyrite. One study (Pontifex, 2010) estimates the $FeS_2$ content to be 2 to 7% when the $CuFeS_2$ content is at 1 to 2% in the first five years of operation. Despite this, the flotation test program settled on pH 9, even though the optimization tests showed better recoveries for both copper and gold, and better pyrite rejection at pH 10. In addition, the testing selected SIBX as the collector, which may not be selective enough against pyrite. Using pH 9 and SIBX, locked-cycle testing yielded recoveries that were less than 80% for Cu and very low for gold. The results suggest poor pyrite rejection such that target concentrate grades were attained only at the expense of recovery.

Operations should consider more selective reagents such as dithiophosphates (for example, Aero 3477), coupled with Aero 3418 if it enhances gold recovery and a rougher pH of 10 to 10.5.

Additional locked-cycle flotation tests are recommended to optimize copper and gold recovery with improved reagent suites. These would be performed on three composites. One composite of ore expected to be mined in Years 2 and 3 of the production schedule. A second composite of ore expected to be mined in Years 4-5 of the production schedule, and a third composite of sulfide dominant ore.

Once the locked-cycle flotation work is optimized, bulk flotation will be conducted to produce enough mill tailing material for downstream process tests. The downstream process tests are as follows:

- Vendor equipment testing to optimize thickening and filtration for the dry-stack tailing storage facility.
- Magnetite recovery from flotation tailing to decrease acid consumption in the weak-acid tailing leaching of copper and to produce iron concentrate for sale to the iron and steel markets.
- Flotation tailing leach.
- Geotechnical testing and studies for the dry stacking of tailing from the three composites. 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.

The metallurgical testing has an estimated cost of $0.2 million.

## 26.6 FEASIBILITY STUDY AND BASIC ENGINEERING

The following process trade-off studies will be completed as part of the feasibility study effort:

- Chloride Leach: Determination if the presence chloride during heap leaching would increase leach kinetics and significantly accelerate the revenue stream from the leach pad.
- High-Pressure Grinding Rolls (HPGR) versus SAG mill: Comparison between high-pressure grinding rolls and semi-autogenous grinding mill, including operating and capital cost estimates.
- Gravity Test: Determination if gravity separation is economically viable for the recovery of coarse gold and silver in the grinding circuit.
- Tailing filtration: Evaluation to determine the most efficient technology for tailing dewatering to meet the requirements of dry-stack tailing deposition.
- Magnetic Separation: Determination if removing magnetite from flotation tailing would economically reduce acid consumption when flotation tailing is leached. The production of a saleable magnetite product will also be explored.

The Feasibility Study and Basic Engineering project scope is as follows:

- Process trade-off studies
- NI 43-101 Feasibility Study
- Basic Engineering to 15% project completion
- Capital and operating cost estimates
- Financial model
- Expediting and Logistics Study
- Project Execution Plan

The metallurgical processing trade-off studies, feasibility and basic engineering is estimated to cost $11.6 million.

## 26.7 INFRASTRUCTURE

Continue coordination with National Grid Corporation of the Philippines (NGCP) regarding power supply infrastructure and right of way.

## 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.

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# APPENDIX A: PRELIMINARY FEASIBILITY STUDY CONTRIBUTORS AND PROFESSIONAL QUALIFICATIONS

**CERTIFICATE OF QUALIFIED PERSON**

I, Daniel Roth, PE, P.Eng. do hereby certify that:

1. I am currently employed as a project manager and civil engineer at M3 Engineering & Technology Corp. located at 2051 West Sunset Rd, Suite 101, Tucson, AZ 85704.
2. I graduated with a Bachelor of Science degree in Civil Engineering from The University of Manitoba in 1990.
3. I am a registered professional engineer in good standing in the following jurisdictions:
   - British Columbia, Canada (No. 38037)
   - Alberta, Canada (No. 62310)
   - Ontario, Canada (No. 100156213)
   - Yukon, Canada (No. 1998)
   - New Mexico, USA (No. 17342)
   - Arizona, USA (No. 37319)
   - Alaska, USA (No. 102317)
   - Minnesota, USA (No. 54138)
4. I have worked continuously as a design engineer, engineering and project manager since 1990, a period of 35 years. I have worked in the minerals industry as a project manager for M3 Engineering & Technology Corporation since 2003, with extensive experience in hard rock mine process plant and infrastructure design and construction, environmental permitting review, as well as development of capital cost estimates, operating cost estimates, financial analyses, preliminary economic assessments, pre-feasibility and feasibility studies.
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 contributing author for the preparation of the technical report titled "Kingking Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study" (the "Technical Report"), dated effective July 10, 2025 and prepared for St. Augustine Gold & Copper, Ltd.
7. I am responsible for the preparation of sections 1.1, 1.2, 1.3, 1.10, 1.15.1, 1.15.5, 1.15.7, 1.15.8, 2, 3, 4, 5, 6, 18, 19, 23, 24, 25.4, 25.6, 26.4, 26.6, 26.7, 26.8, and 27.
8. I visited the Kingking property on July 10, 2025.
9. I have no prior involvement with the project or property that is the subject of the Technical Report.
10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am independent of St. Augustine Gold & Copper, Ltd. as independence is described in Section 1.5 of NI 43-101.
12. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
13. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 16th day of July 2025.

(Signed and Sealed) "*Daniel Roth*"
Signature of Qualified Person

Daniel Roth
Print Name of Qualified Person

**CERTIFICATE OF QUALIFIED PERSON**

**Benjamin Bermudez**

I, Benjamin Bermudez, PE, do hereby certify that:

1. I am currently employed as a Chemical/Process Engineer at M3 Engineering & Technology Corporation, 2051 W Sunset Rd, Suite 101, Tucson, AZ 85704, USA.
2. I am a graduate of Arizona State University and received a Bachelor of Science degree in Chemical Engineering in 2009.
3. I am a Registered Professional Engineer in good standing in the State of Arizona in the area of Chemical Engineering (No. 54919).
4. I have worked as a chemical engineer, project manager, and process department for a total of 16 years. My experience includes mineral process plant engineering, support of new and on-going process plant operations, capital and operating cost development, and financial modeling of mineral properties. Additionally, my experience includes working on studies through development for projects containing precious metals (gold and silver), base metals (copper, nickel, lead, zinc), and battery metals (cobalt and molybdenum). I was part of the commissioning and start-up teams for Peñasquito (Mexico), Buenavista del Cobre Concentradora No. 2 (Mexico), Moss (USA), Gold Bar (USA), Cerro Negro (Argentina), Idaho Cobalt (USA), Bagdad Copper Cleaner (USA), and Buritica (Colombia). I have also facilitated HAZOP workshops for numerous projects, including projects in the USA, Mexico, Chile, Peru, and Colombia.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 *Standards of Disclosure for Mineral Projects* ("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 contributing author for the preparation of the technical report titled "Kingking Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study" (the "Technical Report"), dated effective July 10, 2025 and prepared for St. Augustine Gold & Copper, Ltd.
7. I am responsible for the preparation of sections 1.12, 1.13, 1.14, 1.15.2, 21 (except 21.1.1 and 21.2.1), 22, 25.1, and 26.1, and have not visited the project site.
8. I have no prior involvement with the project or property that is the subject of the Technical Report.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of NI 43-101.
11. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 16th day of July 2025.

(Signed and Sealed) "*Benjamin Bermudez*"
Signature of Qualified Person

Benjamin Bermudez, PE
Print Name of Qualified Person

**CERTIFICATE OF QUALIFIED PERSON**

I, Art S. Ibrado, PhD, PE, do hereby certify that:

1. I am an independent metallurgical consultant with Fort Lowell Consulting PLLC, 1700 E River Rd #64833, Tucson, AZ 85728, USA.
2. I hold the following degrees:

   Bachelor of Science in Metallurgical Engineering*, cum laude*, University of the Philippines, 1980
   Master of Science (Metallurgy), University of California, Berkeley, 1986
   Doctor of Philosophy (Metallurgy), University of California, Berkeley, 1993

3. I am a registered professional engineer in the State of Arizona (No. 58140), the State of Idaho (No. 22492), and the State of Nevada (No. 031704).
4. I have worked as a metallurgist in the academic and research settings for fifteen years, including research on the mechanism of adsorption of gold cyanide on activated carbon (graduate research) and the oxidation of refractory gold ores (AJ Parker Centre for Hydrometallurgy, Perth, Australia). My industrial experience includes copper flotation for 7 years at Philex Mining (Philippines) and 1.5 years at the Phoenix Mine (Battle Mountain, NV); carbon-in-pulp (CIP) and carbon-in-leach (CIL) processes for gold recovery for a combined 9 years at Philex Mining, Barrick Gold Strike and Newmont's Twin Creeks and Phoenix operations; pressure oxidation (POX) of refractory gold ores at Barrick Goldstrike and Newmont's Twin Creeks operations; carbon elution using the Zadra and modified AARL processes; and gold smelting. I was part of the owner's team for the design and engineering of the Mount Hope molybdenum project (Eureka, NV) for 1.5 years, before joining M3 Engineering as a metallurgical engineer from May 2009 to July 2021 (12 years). At M3, I was project manager or lead process engineer for several studies involving the processing of Cu, Au, Pb, Zn minerals, was part of the commissioning team for the Peñasquito and Cananea process plants, and conducted HAZOPS workshops for the Toquepala expansion project. As an independent consultant, I have worked on the commissioning of the old Sutter Creek mine process plant, commissioning of the Pumpkin Hollow plant (Nevada), supported the restart of the adsorption, desorption and regeneration (ADR) plant at Çöpler Mine's heap leach operation in Türkiye, and provided metallurgical support for a few studies involving gold and copper processing plants.
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, professional engineer registration, 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 1.6, 1.7, 1.15.6, 13, 17, 25.5, and 26.5 of the Technical Report titled "Kingking Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study" (the "Technical Report"), dated effective July 10, 2025 and prepared for St. Augustine Gold & Copper, Ltd.
7. I visited the Kingking property on January 25, 2011.
8. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of St. Augustine Gold & Copper, Ltd. as independence is described in Section 1.5 of NI 43-101. I do not own any St. Augustine Gold & Copper, Ltd. stocks or shares.
10. My prior involvement with the Kinking Project includes the preparation of "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study," issued on October 28, 2013.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 16th day of July 2025.

(Signed and Sealed) "*Art S. Ibrado*"
Art S. Ibrado, PhD, PE

**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 E. Gas Road, Tucson, Arizona, 84714, USA.
2. I graduated with a Bachelor of Science degree in Mining Engineering from the University of Arizona in 1979 and a Master of Science degree in Mining Engineering from the University of Arizona in 1982.
3. I am a Fellow of the Australasian 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").
4. I have worked in the minerals industry as an engineer continuously since 1979, a period of 46 years. I am a founding partner, Vice President, and Principal Mining Engineer for 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 have also been a member of the Resources and Reserves Committee of the Society of Mining, Metallurgy, and Exploration since March 2012. During my career, I have had extensive experience developing mineral resource models, developing open pit mine plans, estimating equipment requirements for open pit mining operations, developing mine capital and operating cost estimates, performing economic analysis of mining operations and managing various preliminary economic assessments, pre-feasibility, and feasibility studies.
5. I have read the definition of "qualified person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
6. I am responsible for Sections 1.8, 1.9, 1.15.4, 10, 11, 12, 14, 15, 16, 21.1.1, 21.2.1, 25.3, and 26.3 of the technical report titled "Kingking Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study" (the "Technical Report"), dated effective July 10, 2025 and prepared for St. Augustine Gold & Copper, Ltd.
7. I have prior involvement with the property that is the subject of the Technical Report. I was a QP for the technical report titled "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study," issued on October 28, 2013, and a QP for the technical report titled "King-king Copper-Gold Project, Mindanao, Philippines" issued on October 12, 2010 for Russell Mining and Minerals, Inc.
8. As of the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
9. I am independent of St. Augustine Gold & Copper, Ltd. and its subsidiaries as defined by Section 1.5 of NI 43-101.
10. I have read NI 43-101 and Form 43-101F1. The sections of the Technical Report that I am responsible for have been prepared in compliance with that instrument and form.
11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their website accessible by the public, of the Technical Report.

Dated this 16th day of July 2025.

(Signed) "*Michael G. Hester*"
Signature of Qualified Person

Michael G. Hester, FAusIMM
Print Name of Qualified Person

**CERTIFICATE OF QUALIFIED PERSON**

**Donald Earnest**

I, Donald Earnest, P. Geo., do hereby certify that:

1. I am the President of:

   Resource Evaluation Inc.
   550 E. Cambridge Dr. Tucson, AZ 85704

2. I am a graduate of the Ohio State University with a degree of Bachelor of Science, Geology, 1973.
3. I am a Registered Professional Geologist in the State of Arizona (Registration No.), and in the State of Idaho (Registration No.746). I am also a Registered Member (No. 883600RM) of the Society for Mining, Metallurgy, and Exploration, Inc. (SME) of Mining Engineers, and with this registration I am considered a "Qualified Person" (QP) under the definitions of Canada National Instrument NI 43-101.
4. I have worked for 51 years as a Mine Geologist and Senior Mine Geologist at underground and open pit mining operations, a Resident Manager of an underground mining operation, a corporate-level Manager of Exploration and Vice President, Geology, and the President of an international mining consulting company.
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 a contributing author for the preparation of the technical report titled "Kingking Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study" (the "Technical Report") dated effective July 10, 2025 and prepared for St. Augustine Gold & Copper, Ltd.
7. I am responsible for Sections 1.4, 1.5, 1.15.3, 7, 8, 9, 25.2, and 26.2. I visited the Kingking Project site on March 19 - 23, 2011.
8. I have prior involvement with the property that is the subject of this Technical Report, limited to serving as the QP and contributing author for the Technical Report titled "King-king Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study," issued on October 28, 2013.
9. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
10. I am independent of the issuer applying all of the tests in Section 1.5 of National Instrument 43-101.
11. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
12. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 16th day of July 2025.

(Signed) "*Donald Earnest*"
Signature of Qualified Person

Donald Earnest
Print Name of Qualified Person

## CERTIFICATE OF QUALIFIED PERSON

**John G. Aronson**

I, John G. Aronson, SME, CEP, do hereby certify that:

1. I am President/Principal Advisor of:

   ESG Resiliency Plus, LLC
   122 Kitty Hawk Drive
   Windsor, Colorado, USA. 80550

2. I graduated with a BS in Environmental Sciences from Nebraska Wesleyan University, Lincoln, NE; an M.S. in Zoology/Limnology with minor in Civil Engineering, University of Nebraska, Lincoln, NE and completed post graduate training in Statistics, Botany, Chemistry, Geology, and Electron Microscopy at Colorado State University, Fort Collins, Colorado.

3. I am a member in good standing of many professional scientific organizations, including the Society of Mining, Metallurgy, and Exploration (Member No. 05230901). I am a Certified Environmental Professional, (Academy of Board Certified Environmental Professionals, #14031076; Certified Senior Ecologist (Ecological Society of America, #254 Certified Lake Manager, (North American Lake Management Society, NALMS, #44615399) and Certified Fisheries Scientist, (American Fisheries Society #1590).

4. I have worked on environmental and social impact and risk assessment for international mining projects for over 51 years. My experience includes Principal in Charge and Lead Environmental Scientist for hundreds of mining projects throughout the USA and around the world.

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 a contributing author for the preparation of the technical report titled "Kingking Copper-Gold Project, NI 43-101 Technical Report, Preliminary Feasibility Study" (the "Technical Report"), dated effective July 10, 2025, prepared for St. Augustine Gold and Copper, Ltd. (SAGC); and am responsible for Sections 1.11 and 20. I have relied on the work and statements of other qualified persons and experts, as detailed in the Technical Report, for specific sections or aspects of the report. This reliance is identified and justified in the relevant sections. As of the effective date (July 10, 2025) of the Technical Report, the information and conclusions provided are true and accurate to the best of my knowledge, and I have exercised all reasonable diligence in preparing the report.

7. I served as ESIA Principal in Charge for the Kingking Copper-Gold Project and conducted three major site visits on October 25-28, 2010; November 30-December 9, 2010; and January 13-20, 2011 to the Kingking Project to examine relevant project areas, conduct baseline monitoring of air, meteorology, aquatic, terrestrial, and marine ecology, hydrology, soils, and sociology and verify information included in the Technical Report. I personally conducted and managed onsite environmental investigations of aquatic, terrestrial, and marine environments at the site. I was responsible for organizing, conducting, and reporting comprehensive baseline data and environmental analysis reports for the entire project, including the ESIA.

8. I have significant prior involvement with the property that is the subject of the Technical Report. I served as the Environmental Program Manager for the EIS and conducted multiple onsite investigations and inspections. I served as the QP for the original NI 43-101 report.

9. I have continued to be engaged as a senior consultant on the Kingking Copper-Gold Project.

10. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.
11. I am independent of St. Augustine Gold and Copper, Ltd. (SAGC) as defined by Section 1.5 of National Instrument 43-101 (NI 43-101).
12. 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 to the best of my knowledge and belief.
13. I consent to the public filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Signed and dated this 16th day of July 2025.

(Signed) "*John G. Aronson*"
Signature of Qualified Person

John G. Aronson
Print Name of Qualified Person