BERUANG KANAN MAIN COPPER PROJECT

CENTRAL KALIMANTAN, INDONESIA

NI 43-101 INDEPENDENT TECHNICAL REPORT - Preliminary Economic Assessment

Effective Date: 19 May, 2016

Qualified Persons

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Refer to Appendix A for relevant "Statement of Qualified Person". These Certificates are considered valid as of the effective date and are in accordance with National Instrument 43-101

IMPORTANT NOTE

This report was prepared as a National Instrument 43-101 Technical Report for PT Kalimantan Surya Kencana (KSK) by Orelogy Consulting Pty. Ltd. (Orelogy). The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in Orelogy's services, based on:

i) information available at the time of preparation
ii) data supplied by outside sources, and
iii) the assumptions, conditions, and qualifications set forth in this report.

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Glossary of Acronyms/Abbreviations

Type	Abbreviation	Description
Distances	cm	centimeter
	km	kilometre
	m	meter
	mm	millimeter
Areas	m^2	square meter
	ha	hectare
	km^2	square kilometer
Weights / Density	dmt	dry metric tonne (i.e. no moisture content)
	g	gram
	g/cc	grams per cubic centimeter
	kg	kilogram(s)
	oz	ounce
	t	metric tonne (1000 kg)
	t / m^3	Tonne / cubic meter
	wmt	wet metric tonne (i.e. inclusive of moisture content)
Time	min	minute
	hr	hour
	op hr	operating hour
	eng hr	engine hour
	d	day
	yr	year
Volume / Flow	m^3	cubic meter
	BCM	Bank Cubic Meter (i.e. insitu)
	m^3/hr	cubic meters per hour
	LCM	Loose Cubic Meter (i.e. excavated)
	PCM	Placed Cubic Meters (i.e. after placement and compaction)
	tpa	metric tonne per annum
Assay / Grade	g/t	g/t – grams per tonne
	ppm	parts per million
	ppb	parts per billion
Other	%	Percent
	%Difference	Percentage difference (duplicate - original)/original
	%RSD	Percentage Relative Standard Deviation, StdDev/Average *100
	A.B.N.	Australian Business Number
	AAS	Atomic Absorption Spectroscopy - method for measuring element concentrations in solution (assays)
	AE	Advanced Electrolyte
	Ag	silver
	AIM	the Alternative Investment Market - a sub-market of the London Stock Exchange

Type	Abbreviation	Description
	AMDAL	Analisis Mnegenai Dampak Lingkungan (Environmental Impact Analysis)
	ARD/ML	Acid Rock Drainage/Metal Leaching
	ASL	Above Sea Level
	Au	gold
	AuEq	gold equivalent
	B.App.Sc.	Bachelor of Applied Science
	BAT	Best Available Technology
	BCM	Bank Cubic Meters (i.e. In-situ volume)
	BE-COG	Break-Even Cut-Off Grade (i.e. grade above which mineralization is reported as LFI)
	BK	Beruang Kanan
	BKM	Beruang Kanan Main Zone Prospect/mineralization
	BSc.(Hons)	Bachelor of Science (with Honors)
	COG	Cut-Off Grade
	CoW	Contract of Work
	CRM	Certified Reference Material
	CSR	Corporate Social Responsibility
	csv	comma separated values
	Cu	copper
	DCF	Discounted Cashflow
	E-COG	Elevated cut-off grade (i.e. cut-off grade above break-even applied over time)
	E	East
	ENJ	PT Eksplorasi Nusa Jaya (a PT Freeport Indonesia subsidiary)
	ENJ-KSK	PT Eksplorasi Nusa Jaya and PT Kalimantan Surya Kencana Joint Venture period of work
	et al.	and others
	ESHIA	Environmental, Social, and Health Impact Assessment
	GA	PT GeoAssay (laboratory)
	GIIP	Good International Industry Practice
	Grade	Quantity of metal per unit weight of host rock.
	GT	Grade Tonnage
	H&A	Hackman and Associates Pty Ltd
	ICP-MS	Inductively coupled plasma mass spectrometry - method for measuring element concentrations in solution (assays)
	ICP-OES	Inductively coupled plasma optical emission spectrometry - method for measuring element concentrations in solution (assays)
	ICP-OES AAS	methods for measuring element concentrations in solution (assays)
	IP	Induced Polarization - involves transmitting a current into the ground using two electrodes and measuring the voltage between another pair of electrodes.
	IUP	Mining Business License (Izin Usaha Pertambangan).
	JV	Joint Venture
	KGCL	Kalimantan Gold Corporation Limited
	KNA	Kriging Neighborhood Analysis
	KP	Mining Authorization (Kuasa Pertambangan) - now defunct.
	kWh	Kilowatt hour
	LFI	Leach Feed Inventory (mineralized material economically viable for processing)
	LO	Loaded Organic
	MAIG	Member Australian Institute of Geoscientists

Type	Abbreviation	Description
	max	maximum
	mesh	grid mesh (measurement of aperture)
	MIC	Maximum Instantaneous Charge
	MIK	Multiple Indicator Kriging
	Mo	Molybdenum
	MPD	Mean Paired Difference (expressed as a percent)
	MPRD	Mean Paired Relative Difference (expressed as a percent)
	MSc.	Master of Science
	MW	Megawatt
	N	North
	NAF	non-acid forming (with respect to rock properties)
	NB	Please note
	NI 43-101	"Canadian National Instrument 43-101 - Standards of Disclosure for Mineral Projects" defines and regulates public disclosure in Canada for mineral projects and it relies on resource and reserve classification as defined by CIM.
	NPV	Net Present Value
	Ordinary Kriging	3D interpolation method.
	OSA	Overall Slope Angle - Angle from the upper crest to the toe of the slope at the pit bottom
	OX	Oxiana Limited
	OX-KSK	Oxiana Limited and PT Kalimantan Surya Kencana Joint Venture period of work
	PAF	Potentially Acid Forming (with respect to rock properties)
	Pb	Lead
	pH	measure of the acidity or basicity of an aqueous solution
	PMA	Indonesian limited liability company established with foreign capital
	PQ HQ NQ BQ	Diamond Drill Hole Core sizes
	PT	Perseroan Terbatas ("Limited Liability")
	Py	Pyrite
	QA	Quality Assurance
	QC	Quality Control
	Q-Q	Quartile - Quartile (plot)
	Rd	Road
	RE	Reference to
	RF	Revenue Factor
	RL	Reduced Level
	S	South
	Sb	Antimony
	SEDAR	System for Electronic Document Analysis and Retrieval (Canadian - www.sedar.com)
	SFK	PT Sucofindo (Persero) Laboratory
	SG	Specific Gravity (mass/volume)
	Si	Silica
	SOP	Standard Operating Procedure
	SPPL	Surat pernyataan pengelolaan lingkungan (Statement of Ability to Manage and Monitor the environment)
	SR	Strip Ratio (i.e. waste/ mineralized material)
	StdDev	Standard Deviation

Type	Abbreviation	Description
	TIN	Triangulated Irregular Network (3D computer solid model)
	TM	Trade Mark
	tpa	tonnes per annum
	tpd	tonnes per day
	UCF	Undiscounted Cashflow
	UKL-UPL	Upaya Pengelolaan Lingkungan Hidup dan Upaya Pemantauan Lingkungan Hidup (Environmental Management Effort and Environmental Monitoring Effort)
	USD	United States dollar
	UTM	Universal Transvers Mercator (Cartesian coordinate grid system)
	vol%	Percentage of total volume
	W	West
	WA	Western Australia
	WGS84, UTM Zone 49S	Spheroid projection and grid datum for the geographical location of data at Beruang Kanan
	WRD	Waste Rock Dump

1 EXECUTIVE SUMMARY

1.1 INTRODUCTION

Orelogy Consulting Pty. Ltd. (Orelogy) was commissioned by PT Kalimantan Surya Kencana (KSK) through Asiamet Resources Ltd (ARS) to undertake a Preliminary Economic Assessment (PEA) of the Beruang Kanan Main (BKM) Copper Project in Kalimantan, Indonesia. The purpose of this assessment is to determine the economic viability of BKM in accordance with the guidelines of Canadian National Instrument (CNI) 43-101.

This independent NI 43-101 Technical Report (the Report) has been generated to support the results of the PEA.

The Report is in support of Asiamet's press releases entitled "*Asiamet BKM deposit PEA delivers US$204m NPV10 and 39% IRR*", dated 5 April, 2016.

1.2 KEY OUTCOMES

The BKM deposit is a covellite, chalcocite and chalcopyrite vein style copper mineralized system hosted in sheared sediments and volcanics within an interpreted thrust fault-coupling or ramping zone. It is highly amenable to heap leach processing and therefore the BKM Copper Project is based on conventional open-pit mining and a heap leach - SX/EW process path. The main purpose of the PEA was to determine whether there was sufficient justification to progress the BKM to the next level of feasibility assessment. The project currently has an active mine life of just over seven (7) years, with 1 year of heap leach processing post-mining for a total operating life of just over eight (8) years. A nominal production start-up year of 2019 has been proposed.

Mining will occur sequentially over a number of interim stages out to the final mine design in order to minimize upfront waste movement and maintain an acceptable ore supply. The proposed SX-EW production rate is a maximum of 25,000 tonnes per year of LME "A" Grade copper cathode. This target is reached in five (5) of the 8+ years of copper production. 2019, 2026 and 2027 are below name plate due to the ramp up and ramp down in the in the leaching response curve. Metal production also drops to 21.8 ktpa in 2022 due to a drop in Cu head grade.

The key outcomes of the Preliminary Economic Assessment of the Beruang Kanan Main (BKM) Copper Project are outlined in Table 1-1.

Table 1-1 Key Outcomes of the BKM Copper Project Preliminary Economic Assessment

Economic Summary		Unit	Base Case
Life of Mine (LOM)		Years	8
Copper Cathode Sold		Million lbs.	391.0
Copper Price (LOM Average)		$US/lb.	3.25
Gross Revenue		$US	1,270.6 M
LOM C1 Operating Costs		$US	499.5 M
LOM C1 Operating Cost (recovered copper)		$US/lb.	1.28
Royalties		$US	63.5 M
Off-site transport		$US	19.8 M
LOM All In Operating Cost		$US	582.8 M
		$US/lb.	1.49
Initial Capital Cost (including a 15% Contingency)		$US	163.8 M
Taxes		$US	136.6 M
NPV and IRR (Base Case)			
Discount Rate		Percent (%)	10
Pre-Tax	**Net Free Cash Flow(including royalties)**	$US	524.0 M
	NPV	$US	290.7 M
	IRR	%	47.5
	Payback Period	Years	2.1
After Tax	**Net Free Cash Flow (incl. royalties)**	$US	387.5 M
	NPV	$US	204.3 M
	IRR	%	38.7
	Payback Period	Years	2.4

Note: All figures reported are in 2016 Q1 US dollars. NPV calculations assume that cash flows occur at the beginning of each year. The PEA is preliminary in nature and includes the use of Inferred Mineral Resources which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves and there is no certainty that PEA results will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.3 CAUTIONARY NOTES

1.3.1 Cautionary Note Regarding Use of Inferred in Resources in Financial Analysis

This Preliminary Economic Assessment is, by definition, preliminary in nature. As such it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary economic assessment will be realized.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

1.3.2 Cautionary Note Regarding Forward-Looking Information

Certain information and statements contained in this section and in the Report are “forward looking” in nature. Forward-looking statements are described in more detail in Section 22 of this report.

1.3.3 Cautionary Note Regarding Production Dates

The production schedules and the annualized cashflow table of the financial analysis are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. They were also used as the basis for the forward-looking copper prices used.

1.4 PROJECT DESCRIPTION AND LOCATION

1.4.1 Location

BKM is located within the KSK Contract of Work area (KSK CoW) in Central Kalimantan, just south of the equator (Figure 2-1). It is about 190 kilometers north and slightly west of Palangkaraya, the capital city of Central Kalimantan (Figure 4-1). The BKM (centered on Long. 113 25 00 E, Lat. 00 37 00 S) is in mountainous jungle terrain at the headwaters of the south flowing Kahayan and Samba rivers in a remote area where no permanent villages exist. The location is isolated and access both to and around the prospect is difficult and imposes certain restrictions on field operations.

1.4.2 Ownership

PT Kalimantan Surya Kencana (KSK, incorporated in Indonesia) is the 100% owner of the 6th generation Contract of Work (KSK CoW) within which BKM is located. KSK in turn is owned 75% by Indokal Limited (incorporated in Hong Kong) and 25% by PT Pancaran Cahaya Kahayan (incorporated in Indonesia). Indokal Limited owns 99% of PT Pancaran Cahaya Kahayan with the remaining 1% owned by Mr. Mansur Geiger (held in trust for Asiamet Resources Limited). The parent company to the corporate structure is a Bermuda company, Asiamet Resources Limited (ARS), formally Kalimantan Gold Corporation Limited, which is a publically listed company on the TSX-V (Canada) and AIM (London) stock exchanges. ARS owns 100% of the shares in Indokal Limited.

1.4.3 Mineral Tenure and Surface Rights

On August 16, 1999, by decree of the Government of the Republic of Indonesia, the KSK CoW and nearby PPK CoW were amalgamated into the one holding (KSK CoW, effective date of April 28, 1999). The KSK CoW is now in the fourth year of its Exploration Period.

On August 24 2004, 5,100ha was added to the KSK CoW making the maximum holding of the KSK CoW 129,290ha. KSK has since relinquished ~50% of the tenement area in two stages so that the current holding now stands at 61,003ha. The next relinquishment is scheduled to coincide with the completion of the feasibility stage of the tenement.

KSK has signed a non-binding Memorandum of Understanding (MOU) with the Government of the Republic of Indonesia (GOI) covering amendments to its KSK Contract of Work. The CoW system provides security of tenure for a minimum of 38 years of exploration, development and operations and KSK continues discussions with the GOI regarding possible amendments to some of the KSK CoW terms in order to achieve closer alignment with the current Law No. 4/2009.

1.4.4 Environmental Permitting

Indonesian environmental laws require the preparation of an environmental study for projects requiring an Exploitation Permit. This is generally undertaken as part of the Feasibility Study. As BKM and the CoW are not at the feasibility stage there is no requirement at this point in time for KSK to undertake an environmental study

1.5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The BKM site is approximately 190 kilometers (6 hours drive or 50 minutes flying time by helicopter) northwest of Palangkaraya, the capital city of Central Kalimantan. Daily air flights connect Jakarta with Palangkaraya.

The site is approximately eight to ten hours travel by vehicle (approx. 400km) from Banjarmasin, the nearest port and main port of South Kalimantan.

Logging roads provide direct access into this previously remote area. Access to the project is currently via foot from the field camp, which is located on its eastern side of the project at the base of the main ridge.

Indonesia has a typically tropical climate with two seasons, wet and dry. In most of the country the dry season occurs from May to October with the wet season occurring in the rest of the year. Humidity is on average a minimum of 60% with an average temperature of 25 to 27 degrees Celsius all year round.

The Central Kalimantan climate is a wet weather equatorial zone with rainfall or precipitation of 2,800 - 3,400 mm per annum with an average of 145 rainy days annually.

There are no villages or significant habitation in the BKM area. The BKM area has been logged and a lot of the forest now present is regrowth following this activity. Local artesian miners are scattered throughout the region. None are active within the BKM area. Apart from logging roads, the area has little infrastructure.

1.6 HISTORY

Recorded exploration on the KSK CoW started in 1981 when PT. Pancaran Cahaya Mulia (PCM, *later PT. Pancaran Bahagia (PCB)*) and Sinar Enterprises International B.V. intended to explore the area. Reconnaissance surveys were conducted from 1982 until 1985 into the upper Kahayan area. This period of exploration was undertaken primarily for placer gold.

In 1985, the exploration emphasis changed to hard rock epithermal gold and on October 7, 1985 a joint venture was signed between PCB and PT Pancaran Paringa Kalimantan (PPK, owned by Australian mining company Molopo). The agreement (20 percent PCB, 80% Paringa Mining and Exploration Company PLC, subsidiary of Molopo) resulted in a fourth generation CoW signed between PPK and the Republic of Indonesia on December 2, 1986. The original CoW covered 613,700 hectares but over time was reduced in size to 33,170 hectares and added as Block A to the current KSK CoW. All of the KP's were relinquished.

During the joint venture exploration phase, several areas were recognized as having potential for porphyry copper style mineralization, specifically the Beruang Kanan and Tumbang Huoi prospects. In 1990 the Molopo/PPK joint venture was dissolved.

In 1992, Kalimantan Investment Corporation (KIC) took over field operations from Molopo and PPK. During the period 1992 to 95 the Tumbang Huoi and Mansur prospects were evaluated by IP and diamond drilling.

Kalimantan Gold Corporation Limited (KGCL) was formed during May 1996 and listed on the then Vancouver Stock Exchange. KGCL made application through KSK to the Department of Mines for a 121,900 hectares sixth generation CoW subsequently officially granted on April 28, 1997. The details and history of this current CoW are summarized in Section 1.4.3. On July 24, 2015, KGCL changed its name to Asiamet Resources Limited (ARS) which is listed on both the TSX Venture Exchange in Canada and the AIM in England.

Exploration and evaluation of the KSK CoW has centered on four main areas (Baroi, Beruang Tengah, Beruang Kanan and Mansur, refer to Figure 4-1) where KSK and two consecutive Joint Venture partners Oxiana Limited and Eksplorasi Nusa Jaya (ENJ) focused on identifying porphyry mineralization at the prospects. The subsequent ENJ-KSK and KSK drilling data constitutes the majority of the data utilized in preparing the 2015 Resource Estimate used in this study.

1.7 GEOLOGICAL SETTING AND MINERALIZATION

The KSK CoW is situated within a mid-Tertiary age magmatic arc (Carlile JC, and Mitchell AHG, 1994) that hosts a number of epithermal gold deposits (e.g., Kelian, Indon, Muro) and significant prospects such as Muyup, Masupa Ria, Gunung Mas and Mirah (Figure 7-1).

Porphyry style copper-gold mineralization in the KSK CoW is associated with a number of intrusions that have been emplaced at shallow crustal levels at the junction between Mesozoic metamorphic rocks to the south and accreted Lower Tertiary sediments to the north. These intrusions are interpreted to be part of the Oligocene Central Kalimantan arc of (Carlile JC, and Mitchell AHG, 1994). Older intrusions, and associated volcanic and volcaniclastic rocks, of probably Cretaceous age also outcrop along this contact (Carlile JC, and Mitchell AHG, 1994).

Structures in the region are dominated by a northeast striking set of faults that are interpreted to be features of the Kalimantan Suture (van Leeuwen et al., 1990) and are probably arc parallel, or accretionary, faults. The major gold prospects and deposits in Kalimantan are also localized in a similar structural setting (Corbett GJ, and Leach TM, 1998).

Large circular features (as evident on satellite imagery etc.) are interpreted to be volcanic collapse features and they host many of the porphyry copper-gold prospects within the KSK CoW.

The mineralization has been delineated as twenty-five stacked and adjacent domains covering a strike length of 1300m (towards 000°), across a total width of 900m and a vertical extent of 450m. Mineralization is centered on three areas whose lateral and vertical extents are well defined. Structural interpretation indicates potential for repeat settings to exist at depth and in laterally detached locations to Beruang Kanan.

Covellite, chalcocite and chalcopyrite vein style copper mineralization is hosted in sheared and blocky sediments and volcanics of Cretaceous to Tertiary age. The mineralization is located within and adjacent to an interpreted thrust fault-coupling or ramping zone. Extensive and intense phyllic-style alteration persists throughout the mineralized zone.

1.8 EXPLORATION

Detail of the surface exploration activities and results are not included in this report as they have been documented in the report "*Competent Person's Report on the Mineral Exploration Assets of Kalimantan Gold Corporation Limited*" (Munroe S, and Clayton M, 2006). Apart for the mapping of silica ledges, the surface work has had no input into the BKM resource estimate or BKM PEA.

1.9 DRILLING

145 diamond drillholes have been drilled within and around the Beruang Kanan mineralization. 71 of these holes were drilled by KSK between May and September 2015, and the additional drilling and data from these holes form the basis of the 2014 to 2015 resource estimate update for the deposit. The mineralization is delineated by 93 of the 145 holes, totalling 17,538m of which 4,798m have intercepted the domained mineralization. Drilling of the deposit was undertaken in five programs by three separate companies; PT Kalimantan Surya Kencana (KSK), Oxiana Limited (OX) and PT Eksplorasi Nusa Jaya (ENJ). The latter two companies undertook their work in Joint Venture with KSK. Hole attitudes are mostly angled between 60° and 70° towards 270°. There are no twin holes drilled at Beruang Kanan Main Zone.

1.10 SAMPLING AND ANALYSIS

Historical sampling of mineralization is at a nominal 3m length. Drilling of mineralization undertaken by KSK in 2015 is sampled at nominal 1m lengths while non-mineralized core is sampled at nominal 2m lengths. Copper, gold, silver, antimony, lead, zinc, arsenic and molybdenum assays from 8,211 half-PQ, half-HQ, half-NQ and half-BQ diamond core samples populate the resource dataset, with the ENJ samples including an additional 14 elements and the 2015 KSK dataset including an additional 27 elements. 3,198 of the drill intervals are modeled within the mineralized domains at Beruang Kanan. Copper is the only element with potentially economic grades and is accompanied by 0.5ppm to 1.0ppm silver.

Copper grades of samples from NQ/BQ core average 26% lower than those from PQ/HQ core samples. This difference is due to a base shift or systematic relative bias between the two datasets and may be related to the fundamental sampling error or to variations in grade throughout the mineralization (PQ and HQ drilling samples shallower depths of mineralization than NQ and BQ drilling). It is unknown if the laboratory sample reduction methods are appropriate, where historically samples were reduced to 1kg in size at -4mm crush size and in 2015 samples were reduced to 1kg in size at -2mm crush size. Early analysis of QC data in 2015 showed that -2mm particle size returned acceptable levels of precision, however later QC results showed that 28% of crusher duplicates possessed percent mean paired differences of >5%. Investigations into the issue is ongoing at the time of report preparation, however the comparatively uniform grade profile in the dataset suggests that any introduced variance at this stage of sub-sampling will not materially affect confidence in the global resource estimate. Samples were digested by mixed 3 acid-digest methods and determined by both ICP-OES and AAS instruments. Assay quality control samples included with the ENJ and 2015 KSK drill samples show that confidence can be placed in assays from this dataset. Comparison of data population distributions between the ENJ copper assays, the 2015 KSK assays and the historic assays indicate that the earlier assays are also of acceptable reliability for estimating global resources. The assay data is considered of acceptable quality to underpin Indicated and Inferred Resources (NI 43-101) at Beruang Kanan.

1.11 DATA VERIFICATION

Data verification was undertaken for inputs into the resource estimate separately for each of the five drilling programs. Where available, data was validated by assessing QA and QC documentation and data. Where QA/QC data was not available, these data subsets (drill campaigns) were compared against those data subsets whose validity was understood (through evaluation of their associated QA/QC data).

Of note:

- Sample location: Drill collar locations are well known. Thirty holes have no downhole survey data however holes with data show little deviation at depths within the resource estimate volume and therefor it is considered that sample locations are known with reliable accuracy and are fit for use in the 2015 resource estimate.
- Topographic surface: The consistency between the drill collars, surface mapping (including contours) and the LIDAR topographic surface confirms that the BKM project has internal spatial integrity.
- Primary Sample size: NQ-BQ drill core samples show a 26% low bias in copper assays compared with HQ-PQ core samples. NQ-BQ samples are located in deeper sections of the mineralization and the grade tenor shift may reflect grade variability at BKM. Alternatively the bias may be due to the fundamental error in dealing with inherent heterogeneity of the mineralization. In general the fundamental sampling

issue diminishes with increase in the primary sample size; therefor it is likely that the grade of the copper for the global resource estimate is negatively biased, if this issue exists.

- Sample Preparation and Assay: The sample comminution and reduction process employed are not theoretically ideal (according to Gy's generalized sampling nomogram) however the relatively narrow band of copper assays within the mineralization suggests that any issues may not be of significance when the risk is assessed at the global scale. Three acid digest was undertaken on ½ core samples. There is only one reference to copper silicates at BKM adding confidence that the analysis of total copper is suitable for use in resource estimates. Issues were encountered in laboratory procedures for sequential copper assays, where grade accounting between the H_2SO_4 and CN soluble copper resulted in low bias for these portions of the assay. The total leachable copper determined in the 2015 resource estimate utilizes the total copper grade and the residual copper grade (non-leachable) to determine the leachable copper percentage.
- Assay data quality: The KSK 2015 assay QC program and QC work undertaken by ENJ-KSK contains sufficient quality control samples to assess reliability of the copper assays which show that data from these programs are suitable for use in estimating and classifying global resources. Earlier work by OX-KSK contained limited quality control samples and there were no quality control samples submitted with assays for the early work undertaken by KSK (pre 2002). The copper assay data populations from these earlier drill campaigns are comparable with the assay population from the KSK 2015 and ENJ-KSK work. This favorable comparison adds confidence in that the early copper assays are suitable for inclusion in the BKM 2015 Resource Estimate and acceptable for classifying global resources.
- Tonnage factors: Dry bulk density (DBD) measurements were undertaken on core from the KSK 2015 drilling program. QC data and evaluation of the DBD data show that this dataset is suitable for estimating tonnages at BKM.
- Core recovery: Verification of the core recovery data was undertaken by assessing the logged data against observations in core photos. A small number of significant discrepancies between the logging and core photo observations however these are unlikely to impact on the evaluation or risk assessment to the BKM 2015 Resource Estimate. There is no observed relationship between length core recovery and copper grade.
- Geological, Mineralization, Alteration logging: Validation of drill core logging through cross checking against core photos and copper assay results has built confidence in this information. Further photography review was undertaken to identify, group and model gangue material types for input into metallurgical studies. These datasets are suitable for use in estimating and evaluating mineralization at BKM.

1.12 MINERAL PROCESSING AND METALLURGICAL TESTING

Miller Metallurgical Services (MMS) has undertaken an analysis of the preliminary metallurgical characterization tests on the BKM Project. The testing has been undertaken at two laboratories; PT Intertek Utama Services (Intertek) in Indonesia and Core Metallurgy Pty Ltd in Australia.

The geological interpretation of mineralized material types has been extended to include the sequential assay results of the copper mineralized material and characterization of leachable copper content. The copper mineralogy is dominated by chalcocite in most areas, with mixtures of chalcocite, covellite and lesser bornite. Chalcopyrite is present in most mineralized zones to some extent. Pyrite is the dominant sulphide with high Fe:Cu ratios. There is little meteoric weathering except for a shallow 'soil' layer. The geo-mechanical characteristics of unleached and leached rocks may predominate over the geological typing and the presence of clays is noted in the logs.

The finer crush size leached slower than the coarser crush size in the two column tests. It is likely that crushing in the 12 mm to 19 mm range may be needed.

The hydrology and geotechnical characteristics of the mineralized materials are yet to be fully determined.

Higher leach recoveries than expected have been achieved without overt signs of biological assistance. It is proposed that the copper minerals are being galvanically leached by the high pyrite to copper ratio in the mineralization. A maximum of 85% recovery of soluble copper has been set to provide some further margin for less than ideal operational control. The level of recovery is consistent with operations heap leaching similar secondary sulphide minerals. A base line leaching response (recovery vs time) has been developed based on the proposed leach chemistry and operational experience

The acid consumption is likely to be low or negative and therefore there will be no economic limit to leaching and leach times can potentially be extended to achieve higher soluble copper recovery. The leachate chemistry shows no issues for the production of high grade copper metal using the solvent extraction – electrowinning process.

Heap leaching has been selected as the best processing option for the BKM Project based on:

- Poor flotation response
- Reasonably high total 'leachable' copper
- Geological assessment of rock competency
- Preliminary results from test work

1.13 MINERAL RESOURCE ESTIMATES

Copper grade is estimated by ordinary kriging interpolation methodology. Interpolation is guided and constrained by solid TIN (triangulated) boundaries. 1,672 three metre composites inform the grade interpolation within domains. Parent cell estimates (25m E x 25m N x 10m RL) were written to a sub-blocked model. High grade values (>3%Cu) were restricted from informing block grades at greater than 50m (E and N) and 25m (RL) distance from sample locations. 30 copper composites were affected by this treatment. Tonnage factors (based on 1,389 dry bulk density measurements) of 2.33g/cc, 2.70g/cc, 2.77g/cc and 2.85g/cc were stamped on the model according to mineralization and weathering characteristics.

The estimate is assigned Indicated and Inferred Mineral Resource classifications under the guidelines outlined in the Canadian National Instrument 43-101. Risk associated with drilling density and orientation suitability, primary sampling reliability, certainty in geological and grade continuity, sample reduction strategy suitability and the unknown reliability of historic assay data are the key inputs in determining the resource classification.

Only a proportion of the total copper present in the mineralized zone can actually be qualified as “leachable” for the purposes of recovery through a heap leach process. This proportion is required to effectively evaluate the economic viability of the various mineralized material types. Therefore, as part of the PEA process, the proportion of total copper that can be considered leachable was modelled as a new resource model item (Cu Leach).

Issues were uncovered in the total leachable copper (H_2SO_4 + CN), and consequently in the total Sequential (SEQ) soluble copper assays, due to poor laboratory procedures. These assays are unsuitable for determining the percentage leachable copper within the resource/reserve model. The copper assaying of the residual material from the SEQ soluble copper assay procedure was shown to be reliable (through QC evaluation) and therefore the percent leachable copper can be determined by use of the original three acid digest copper result (pre November 2015) and the three acid digest copper assay result from the residual material in the SEQ soluble copper assay analyses (December 2015).

The percent leachable copper is determined by the following formulae:

percent leachable copper = (2015_Cu_assay - Seq_Sol_Cu_residual_assay) / 2015_Cu_assay

The percent leachable copper variable was interpolated into the resource model utilizing an Inverse Distance Squared estimation technique, a 3m composite length, the same domain hard boundaries as the 2015 resource estimate and a two pass interpolation process.

1.14 MINERAL RESOURCE STATEMENT

The Mineral Resource estimate for BKM assumes open pit mining methods and are reported using the 2014 CIM Definition Standards. Indicated and Inferred classifications only have been estimated; no Measured Mineral Resources were classified.

The Mineral Resource estimates were prepared by Mr. Duncan Hackman (B.App.Sc., MSc., MAIG) with an effective date of 30 November, 2015.

Table 1-2 Tabulated Copper Resources - Beruang Kanan [Indicated and Inferred Classified Resources reported separately].

Indicated Mineral Resources				
Reporting cut (Cu %)	Tonnes ('000)	Cu Grade (Cu %)	Contained Cu ('000 tonnes)	Contained Cu ('000,000 lbs.)
0.2	15,000	0.7	105	231
0.5	12,600	0.7	88	194
0.7	5,600	0.9	50	110

Inferred Mineral Resources				
Reporting cut (Cu %)	Tonnes ('000)	Cu Grade (Cu %)	Contained Cu ('000 tonnes)	Contained Cu ('000,000 lbs.)
0.2	49,700	0.6	298	657
0.5	25,300	0.7	177	390
0.7	9,800	0.9	88	194

Notes: *Mineral Resources for the Beruang Kanan mineralization have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. In the opinion of Duncan Hackman, the block model Resource estimate and Resource classification reported herein are a reasonable representation of the copper Mineral Resources found in the defined area of the Beruang Kanan Main mineralization. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserve. Computational discrepancies in the table and the body of the Release are the result of rounding*.

The base case resource is reported at a 0.2% copper cut. A 0.2% copper cut (following rounding to reflect confidence) approximates the calculated break-even (BE) cut-off grade value of 0.21% copper derived from an initial economic evaluation of BKM. This economic evaluation was designed to provide a preliminary assessment of the viability of the project prior to the PEA proper. Details of the calculated of BE cut-off are detailed in Table 14-5.

Subsequently the detailed PEA work has utilized a variable grade strategy optimized to maximize project value that equates to a copper reporting range of approximately 0.16% Cu Total to 0.20% Cu Total. The use of 0.2% Cu Total in reporting global resources is appropriate. There are a total of 50kt of resource material within the PEA optimized pit shell with grades below 0.20% Cu Total.

1.15 MINE METHODS

As there is no Mineral Reserve associated with this study, the term "ore" has not been used to avoid any implication of levels of certainty. The that has been material determined as economically and metallurgically suitable for processing, and has been included in the mine plan developed for this study, is referred to as "leach feed inventory". This has been abbreviated to LFI for the remainder of this document.

The PEA is based on a conventional truck-and-shovel, open-pit mine design at a single pit. The mining schedule was developed based on providing a maximum of 25,000 tonnes of recoverable copper to the heap leach pad per year. The BKM Project's active mine life of just over seven (7) years, with 1 year of heap leach processing post-mining for a total operating life of just over eight (8) years.

Over the 8 year life, the pit produces 48.7 Mt of LFI and 59.8 Mt of waste rock. The Life of Mine (LOM) stripping ratio is 1.23.

The pit design incorporates a bench height of 5 m and a 52.5° inter-ramp angle. After adding the ramps and stack berms as required, the overall slope angle averages 45°. The pit uses nine (9) pushbacks and a minimum mining width of 50 m.

The schedule has been generated using the Maptek Evolution scheduling tool, which develops and elevated cut-off grade strategy over time to maximize the project value. Consequently the cut-off grade utilized to classify LFI in the schedule varies over the life of the mine between 0.09% Cu Leach and 0.11% Cu Leach.

The LOM schedule is presented in Table 1-3.

Table 1-3 LOM Mining Schedule

Item	Unit	Totals	2019	2020	2021	2022	2023	2024	2025	2026	2027
Mined Tonnes	Mt	108.6	8.7	17.1	17.3	18.0	13.3	12.2	17.1	4.9	
Mill-Feed Tonnes	Mt	48.7	5.2	6.6	7.4	7.1	6.9	6.0	7.1	2.5	
Waste Tonnes	Mt	59.8	3.5	10.5	9.9	10.9	6.4	6.2	10.0	2.5	
Strip Ratio	(Waste t : Ore t)	1.23	0.67	1.59	1.34	1.54	0.93	1.04	1.40	0.99	
Cu Total	%	0.58	0.79	0.58	0.55	0.52	0.56	0.61	0.52	0.50	
Cu Leach	%	0.50	0.66	0.52	0.47	0.39	0.50	0.58	0.49	0.41	
Cu Leach Recovery	%	85	85	85	85	85	85	85	85	85	
Cu Leach COG	%		0.09	0.09	0.11	0.11	0.09	0.09	0.10	0.09	
Insitu Metal	kt	208.6	29.4	29.4	29.4	23.5	29.4	29.4	29.4	8.6	
Recovered Metal Delivered	kt	177.3	25.0	25.0	25.0	20.0	25.0	25.0	25.0	7.0	
Copper Cathode Produced	kt	177.3	15.8	25.0	25.0	21.8	23.2	25.0	25.0	13.8	2.7
	lb. M	391.0	34.9	55.1	55.1	48.1	51.1	55.1	55.1	30.5	5.9

1.16 RECOVERY METHODS

The overall processing scheme consists of the following assumptions:

- Crushing – Primary crushing to reduce the ROM LFI to a P80 of between 19 mm to 8 mm in a three-stage closed circuit crushing plant.

- Agglomeration – binding crushed LFI fines with larger fragments to form more uniform particles, which will assist stabilization of fines in the heap and also allow the mixing of sulphuric acid with the raffinate.
- Stacking – placing of agglomerated crushed LFI on the heap leach pad incorporating transportation to a radial stacker.
- Heap Leaching – whereby copper is leached from a heap of crushed LFI by leaching solutions percolating down through the heap and then collected from a sloping, impermeable liner below.
- Solvent Extraction (SX) – whereby PLS and electrolyte from the Electrowinning (EW) Plant are contacted with an organic extractant that selectively transfers copper from the PLS to electrolyte.
- Electrowinning (EW) – whereby a direct electrical current is applied to the circulating electrolyte in cells containing multiple lead anodes and stainless steel cathode mother plates. Oxygen is formed at the anode and copper metal is plated on the stainless steel mother-plates. This copper deposit will be stripped from the cathode mother plates and bundled and strapped for sale.

Other allowances within the process plant include:

- reagent storage, preparation and distribution,
- Plant Services (i.e. compressed air system, water reticulation, fire services, hot water heating system).

1.17 PROJECT INFRASTRUCTURE

The following components of project infrastructure have been assessed as part of the PEA.

1.17.1 Bulk Earthworks

Designs and an associated Material Take-Off (MTO) have been developed for the following bulk earthworks:

- Haul Roads
- Diversion Channels
- Heap Leach Pad
- ROM Pad/Secondary Crusher
- SX-EW pad
- Raffinate pond
- Power Station Pad
- Lime Pad
- Pump Station Pad
- Environmental Dam
- Accommodation Camp Pad
- Stormwater HDPE Lined Dam

1.17.2 Access Roads

Due to limited information regarding the condition of the existing local road network, a high level desktop review was conducted of the proposed access route from Palangkaraya to site and recommendations made for the sections that may require upgrading.

While the logging roads to site are currently in use, a detailed route survey will need to be conducted during subsequent phases of the project to ascertain the condition of the road and accurately assess any upgrades necessary.

1.17.3 Site Infrastructure and Accommodation

Designs and associated layouts have been developed for the following site infrastructure:

- Plant Motor Control Centers (MCC's),
- Plant control room,
- Plant laboratory & equipment,
- Plant gate house (pre-fabricated or timber construction),
- Plant workshop & equipment,
- Plant store & equipment,
- Plant offices & furniture (pre-fabricated or timber construction),
- Plant ablutions (pre-fabricated or timber construction).
- Accommodation camp for KSK personnel. The mining contractor will be responsible for their own personnel.

Allowance has also been made for potable water, sewage treatment, communication, weighbridge and site fencing.

1.17.4 Water Management

All stormwater which falls onto the mining or process areas including stockpiles will be contained and re-used wherever possible, in line with a zero discharge philosophy. Stormwater control ponds will be constructed to collect contaminated stormwater for return to the process circuit. Stormwater cut-off berms will be constructed around the mining pit, plant areas and other infrastructure facilities to prevent stormwater run-off from higher laying areas becoming contaminated by being exposed to mining or process related activities or infrastructure. The heap leach will be provided with plastic covers (rain coats) to minimize infiltration of rain water. The clean run off from the raincoats will be collected separately and discharged from the site.

A neutralization plant is used during operation for both the reduction in solution acid concentration and precipitation of metals from water to be discharged from the site.

1.17.5 Power

The BKM Copper Project will require approximately 20 MW of peak load power for 25,000 tonne-per-annum operation demand. A key infrastructure component is the supply of electrical power as this constitutes 25% of the total operating cost. An established Indonesian power supply company has provided initial cost estimates to KSK for a power rental and supply arrangement based on an appropriately sized liquefied natural gas fired power plant.

1.17.6 Transport and Logistics

Bulk materials movement is proposed to be via a combination of trucking and low-cost barge transportation using the current all-weather unsealed road route extending from the BKM site to the Kasongan River Port (120 km) and then barging from the Kasongan port to the Java Sea and onto Banjarmasin Sea Port (510km). The assumption in this PEA study is for KSK to negotiate an ongoing access and maintenance agreement with the company that constructed and maintains the all-weather logging road.

The Banjarmasin Sea Port is a deep water port located 25km from the Java Sea and caters to international vessels. It has a large container handling and storage service together with a customs clearance/bonded area for international freight.

This solution will be used for transporting construction and operating items to site and copper cathode off-site. All transport, logistics and product security will be outsourced to specialist contractors.

Assessment of bulk materials transport and logistics for the project, and the associated capital and operating costs, has been completed by PT. Resindo Resources & Energy (RRE), a highly experienced provider of similar infrastructure solutions throughout Indonesia.

1.18 MARKETING

The SX/EW process to be employed at BKM is anticipated to produce copper cathode that attracts an LME (London Metal Exchange) “A” Grade Copper Price and assumes that it will met the required quality and physical properties to command the associated price premium.

KSK, through Asiamet Resources Ltd, sourced an independent copper price forecast from Wood Mackenzie (Woodmac). Woodmac are a recognized international consulting and research group providing market analysis across a broad spectrum of commodities.

In line with KSK’s view on copper price, the average of the Woodmac forecast LME copper price (Constant 2016 $US), over the seven years 2019 to 2025, of $3.25/lb. was applied for the financial analysis of this study.

1.19 ENVIRONMENTAL, PERMITTING AND SOCIAL LICENSE

The senior management team at KSK has extensive experience with mining projects in Indonesia and fully understands the importance of conducting all aspects of mining activities in an environmentally and socially responsible manner to ensure the success of the project. Specifically KSK is committed to:

- Complying with all applicable Indonesian environmental and social laws and regulations pertaining to mining operations;
- Adopting an inclusive and transparent approach with all stakeholders, with a focus on local communities and Indigenous Peoples;
- Appling Best Available Technology (BAT) and Good International Industry Practice (GIIP) to the design and operation of mining activities at the BKM Project; and,
- Leaving a positive legacy subsequent to cessation of mining activities.

Preliminary environmental baseline studies and monitoring programs have been conducted at the KSK CoW area since 2012. No significant issues have currently been identified that may prevent the permitting requirements for the BKM Project moving forward in a timely manner.

A comprehensive assessment of the potential environmental and social impacts (both negative and positive) from the project will be determined subsequent to further project definition, collection of requisite baseline data and compilation of the Environmental, Social, and Health Impact Assessment (ESHIA)

1.20 CAPITAL COST ESTIMATE

The capital cost estimate covers all major capital cost items required to commence and maintain operations at BKM. All costs are provided in US dollars (USD) and an exchange rate for Indonesian Rupiah of 13,400 IDR per 1 USD.

The accuracy range for the project estimate is ±35%. The estimate makes no provision for future escalation and currency exchange rate fluctuations beyond the basis date. Any other estimate qualifications, assumptions and exclusions are summarized in the relevant sections below.

All mining related costs are based on the preferred submission to a Request for Quotation for mining contract services (refer to Section 16.2).

Heap leach pad earthworks and construction is based on a similar request for budget quotation pricing.

The crushing / agglomeration / stacking and SX/EW plant capital cost estimate have been developed by MillerMet and are based on their database of costs and previous experience with similar plants and projects.

Project infrastructure capital is a combination of budget quotation pricing sought from suitable local construction / engineering groups or first principal estimates developed by DRA, the study infrastructure engineering consultant.

No allowance has been made for import duties, taxes, insurances etc. However a global contingency of 15% has been allowed on all capital items.

The total initial and sustaining capital costs developed for the PEA are summarized in Table 1-4 below.

Table 1-4 Summary of Capital Cost Estimate

Item		USD M
Initial	Mining	$1.7
	Primary Crusher + Agglomerator	$24.6
	Leach Pads	$31.3
	SX/EW (Incl. Neutralization)	$82.7
	Infrastructure	$2.1
	Subtotal	$142.4
	Contingency @ 15%	$21.4
	Total	**$163.8**
Sustaining and Closure	Mining	$5.0
	Leach Pads	$1.0
	SX/EW	$1.6
	Subtotal	$7.6
	Contingency @ 15%	$1.1
	Total	**$8.7**
Grand Total		**$172.5**

Capital cost exclusions:

- Scope or schedule changes
- Force majeure occurrences or cost exclusions
- Hazardous waste remediation
- Financing cost (interest during construction)
- Indirect allowances for sustaining capital estimate, excluding closure costs
- Future currency exchange rates beyond first quarter of 2016
- Any costs related to lost production
- Sunk costs such as previous studies and previously constructed facilities
- Future escalation beyond first quarter of 2016
- Value added or sales taxes
- Surplus material or equipment salvage value credits

1.21 Operating Cost Estimate

The operating cost estimate covers all operating cost centers required to commence and maintain operations at BKM.

The exchange rate for Indonesian Rupiah is 13,400 IDR per $1 US.

The accuracy range for the project estimate is ±35%. The estimate makes no provision for future escalation and currency exchange rate fluctuations beyond the basis date. Any other estimate qualifications, assumptions and exclusions are summarized in the relevant sections below.

All mining related costs are based on the preferred submission to a Request for Quotation for mining contract services (refer to Section 16.2).

The crushing / agglomeration / stacking and SX/EW plant capital cost estimate have been developed by MillerMet and id based on a first principal cost model.

Project infrastructure capital is a combination of budget quotation pricing sought from suitable local construction / engineering groups or first principal estimates developed by DRA, the study infrastructure engineering consultant. For this study it is assumed the C1 cost includes sustaining costs (i.e. Mine Development, Rehabilitation and Closure) but excludes off–site transport (i.e. transport of cathode). Details of the project C1 Cash Operating Cost are provided in Table 1-5.

Table 1-5 C1 Cash Operating Cost

Item	$M	$ / tonne total	$ / tonne ore	$/ rec. lb.
Mining	$233.0	$2.15	$4.78	$0.60
Crushing / Stacking	$66.0	$0.61	$1.35	$0.17
SX/EW Processing	$47.2	$0.44	$0.97	$0.12
Power	$131.2	$1.21	$2.69	$0.34
G&A and Support	$13.5	$0.12	$0.28	$0.03
Sustaining	$8.7	$0.08	$0.18	$0.02
C1 Cash Cost	$499.5	$4.60	$10.25	$1.28

Operating cost exclusions:

- Operating and sustaining costs associated with maintenance of off-site infrastructure.
    - Transportation costs beyond Banjarmasin Port.
- All Owners' costs, including but not limited to:
    - GST, import duties and/or any other statutory taxation, levies and/or national and local institutions.
    - Contributions to health and social welfare programs.
    - Head office costs, administration charges, insurance premiums, interest payments, payroll, etc.
    - EIA, EMPR, contributions to rehabilitation funds, environmental monitoring and conformance to environmental requirements.
    - Working capital, bridging finance and costs associated with finance raisings.

No allowance has been made for import duties, taxes, insurances etc. for consumables or parts.

1.22 ECONOMIC ANALYSIS

The results of the Base Case economic analysis and the key underlying assumptions are provided in Table 1-6.

Table 1-6 BKM Project Economic Analysis Results

Economic Summary		Unit	Base Case
Life of Mine (LOM)		Years	8
Copper Cathode Sold		Million lbs	391.0
Copper Price (LOM Average)		$US/lb	3.25
Gross Revenue		$US	1,270.6 M
LOM C1 Operating Costs		$US	499.5 M
LOM C1 Operating Cost (recovered copper)		$US/lb	1.28
Royalties		$US	63.5 M
Off-site transport		$US	19.8 M
LOM All In Operating Cost		$US	582.8 M
		$US/lb	1.49
Initial Capital Cost (including a 15% Contingency)		$US	163.8 M
Taxes		$US	136.6 M
NPV and IRR (Base Case)			
Discount Rate		Percent (%)	10
Pre-Tax	**Net Free Cash Flow(including royalties)**	$US	524.0 M
	NPV	$US	290.7 M
	IRR	%	47.5
	Payback Period	Years	2.1
After Tax	**Net Free Cash Flow (incl. royalties)**	$US	387.5 M
	NPV	$US	204.3 M
	IRR	%	38.7
	Payback Period	Years	2.4

1.23 SENSITIVITY ANALYSIS

The primary sensitivity scenario run was for copper price, which effectively replicates a variation in grade and recovery as it is a variation in revenue. The results of this sensitivity are presented in Table 1-7. The sensitivity to price and costs are presented in Table 1-8 and graphically in Figure 1-1.

Table 1-7 Sensitivity Analysis - Price

Copper Base Price		$2.75	$3.00	$3.25	$3.50	$3.75
		-15.5%	-7.5%	(Base Case)	7.5%	15.5%
Net Price after Royalty		$2.61	$2.85	$3.09	$3.33	$3.56
Pre-Tax	NPV @ 10%	$161.0 M	$225.9 M	$290.7 M	$355.6 M	$420.4 M
	I.R.R.	32.3%	40.1%	47.5%	54.7%	61.7%
	Pay-back (Undisc.)	2.8	2.4	2.1	1.8	1.7
Post-Tax	NPV @ 10%	$74.6 M	$139.4 M	$204.3 M	$269.1 M	$334.0 M
	I.R.R.	21.6%	30.5%	38.7%	46.4%	53.8%
	Pay-back (Undisc.)	3.7	2.8	2.4	2.1	1.8

Figure 1-1 NPV Sensitivity by Variation in Price and Costs

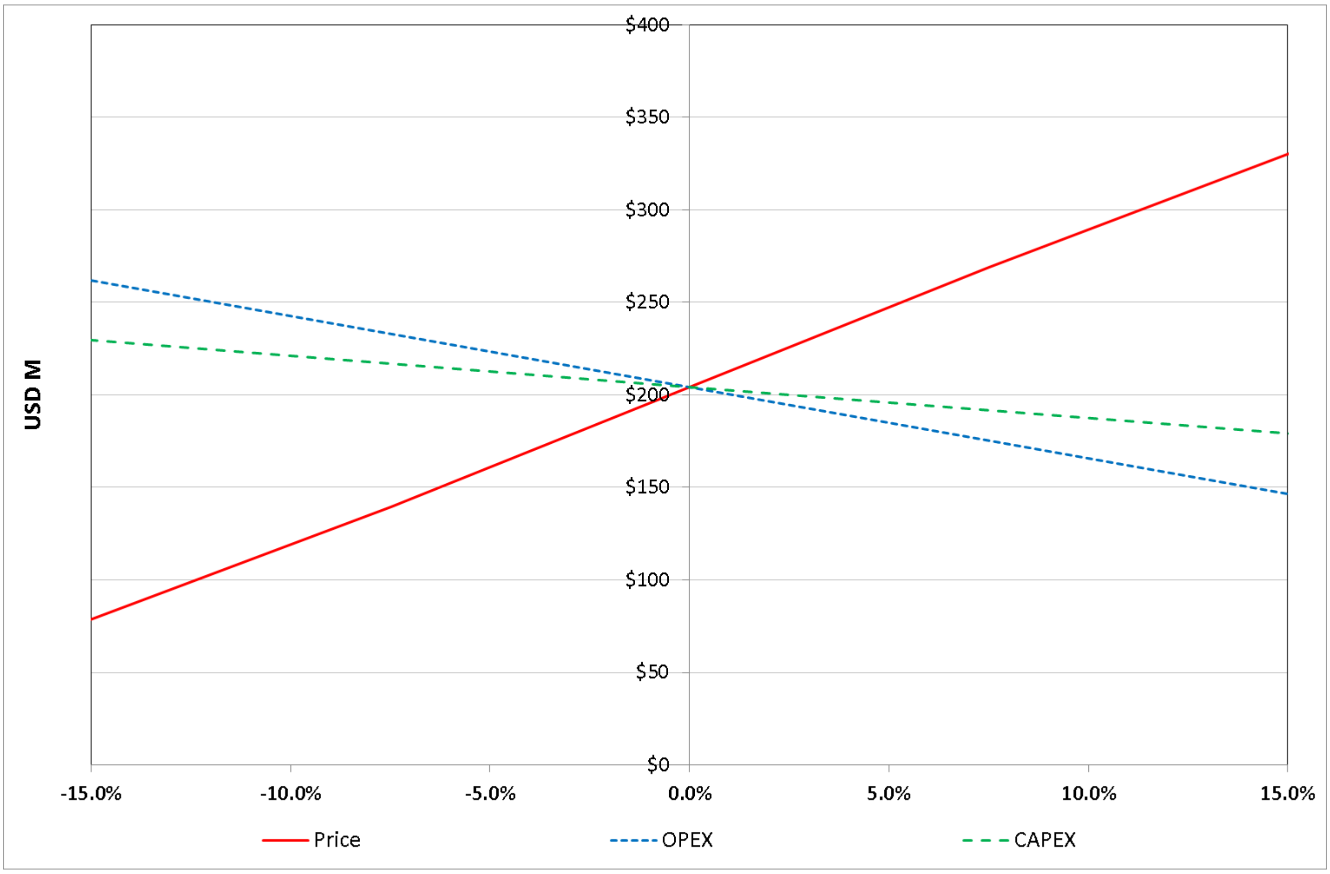

Table 1-8 Sensitivity Analysis – % Variation in NPV10 by % Variation in Key Driver

Key Driver		% Variation in Key Driver		
		-15.5%	Base Case	+15.5%
NPV10	Price	\$74.6 M	\$204.3 M	\$334.0 M
	OPEX	\$263.8 M		\$144.7 M
	CAPEX	\$230.3 M		\$178.2 M
IRR	Price	21.6%	38.7%	53.8%
	OPEX	45.9%		31.1%
	CAPEX	47.0%		32.4%

1.24 OPPORTUNITIES AND RISKS

The BKM Project is subject to the same inherent risks and opportunities as any extractive industry project. This includes, but is not limited to, positive and negative variations in:

- Estimated Resources
- Copper price
- Geotechnical assessment of open pits, waste rock dumps and heap leach
- Capital and operating cost
- Inflation
- Fiscal / royalty / taxation regimes
- Land tenure and environmental permitting
- Impacts on terrestrial and aquatic ecology
- Continuity and effectiveness of community relations programs

Specific opportunities identified include, but are not limited to:

- Extensions of the orebody along strike or to depth, or inclusion of adjacent deposits within the KSK CoW
- Increased recovery of leachable copper above the 85% assumed
- Potential to reduce power costs further by utilizing hydro generation
- Improved community relations through increased opportunities for employment and provision of services/good by local residents

Specific risks that future works will need to focus on are:

- Uncertainties in long term management of acid rock drainage and metal leaching from mine workings, mine waste and the exhausted heap leach
- Site water balance and associated water management plan is not yet defined or quantified due to the current lack of data. This may result in:
    - Increased sediment loads in area streams during construction
    - Overtopping of stormwater retention pond due to extreme rain event

- Deterioration in community relations associated with in migration and perceived unfair hiring practices

1.25 Conclusions

Orelogy has prepared a PEA for the BKM Project and a supporting technical report prepared in accordance with NI 43-101 and Form 43-101F1. The PEA describes the potential technical and economic viability of establishing a conventional open pit mine, a heap leach / SX / EW operation and related infrastructure to process porphyry style copper mineralization from the BKM Deposit. Based on the work carried out in this PEA and the resultant economic evaluation, this study should be followed by further technical and economic studies leading to a prefeasibility study.

1.26 Recommendations

The Qualified Person's that have undertaken this PEA collectively recommend the actions and activities detailed in the following sections to support the advancement of the BKM Project.

This assumes KSK decide to proceed to the next level of project assessment.

The following additional works are recommended to progress the BKM Project:

- Geotechnical assessment
- Metallurgical evaluation
- Mining and process engineering studies
- Waste-rock characterization
- Additional baseline studies and environmental permitting activities
- Marketing studies
- Various trade-off studies.

As part of the recommended work program, additional exploration and geotechnical drilling will be required to support some of the further works. Additional areas for work are also likely to be identified as activities progress.

2 INTRODUCTION

2.1 TERMS OF REFERENCE

Orelogy Consulting Pty. Ltd. (ORELOGY) was commissioned by PT Kalimantan Surya Kencana (KSK), through Asiamet Resources Limited (ARS), to provide an independent Qualified Person's Technical Report for the BKM Copper Project (BKM) located in Central Kalimantan, Indonesia.

This technical report has been prepared in accordance with National Instrument NI 43-101 (Standards of Disclosure for Mineral Projects) and Form 43-101F1. It is based on the results of a Preliminary Economic Assessment (PEA) carried out in late 2015 and early 2016, which assessed the potential technical and economic viability of the BKM based on conventional open-pit mining and a heap leach - SX/EW process path. The main purpose of the PEA was to determine whether there was sufficient justification to progress the BKM to the next level of feasibility assessment.

The PEA has been prepared on a 100% ownership basis and all amounts are stated in US dollars (USD) unless otherwise noted.

2.2 UNITS OF MEASUREMENT

All units of measurement in this technical report and resource estimate are metric unless otherwise stated.

2.3 QUALIFIED PERSONS

The Qualified Persons (QP's), as defined in NI 43-101 and in compliance with Form 43-101F1 (the Technical Report), responsible for the preparation of the technical report are detailed in Table 2-1.

Table 2-1 Qualified Persons and Relevant Responsibilities

Report Section		Company	QP
1	Summary	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
2	Introduction		
3	Reliance on Other Experts		
4	Property Description and Location	Hackman & Assoc. Pty. Ltd.	Duncan Hackman B.App.Sc., MSc., MAIG
5	Accessibility, Climate, Local Resources, Infrastructure and Physiography		
6	History		
7	Geological Setting and Mineralization		
8	Deposit Types		
9	Exploration		
10	Drilling		
11	Sample Preparation, Analyses and Security		
12	Data Verification		

Report Section		Company	QP
13	Mineral Processing and Metallurgical Testing	MillerMet Pty. Ltd.	Graeme Miller BE(Chem), FAusIMM, CP-AusIMM
14	Mineral Resource Estimates	Hackman & Assoc. Pty. Ltd.	Duncan Hackman B.App.Sc., MSc., MAIG
15	Mineral Reserve Estimates	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
16	Mining Methods		
17	Recovery Methods	MillerMet Pty. Ltd.	Graeme Miller BE (Mech), FAusIMM, CP-AusIMM
18	Project Infrastructure	DRA Pacific Ltd.	Johan Du Preez BSc (Eng.), P.Eng
18.1	Site Development	MillerMet Pty. Ltd.	Graeme Miller BE(Chem), FAusIMM, CP-AusIMM
18.2	Roads	DRA Pacific Ltd.	Johan Du Preez BSc (Eng.), P.Eng
18.3	Power Supply	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
18.4	Water Supply	DRA Pacific Ltd.	Johan Du Preez BSc (Eng.), P.Eng
18.5	Mine Site Infrastructure	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
18.6	Process Infrastructure	DRA Pacific Ltd.	Johan Du Preez BSc (Eng.), P.Eng
18.7	Accommodation Camp		
18.8	Site Utilities & Services		
19	Market Studies and Contracts	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
20	Environmental Studies, Permitting and Social or Community Impact	PT Lorax Indonesia	Ali Sahami, Ph.D.
21	Capital and Operating Costs		
21.1	Capital Cost Estimate		
21.1.1	Project Capital Cost Assumptions	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
21.1.2	Mine Capital Cost Estimate		
21.1.3	Crushing / Agglomerating / Stacking Capital Cost Estimate	MillerMet Pty. Ltd.	Graeme Miller BE(Chem), FAusIMM, CP-AusIMM
21.1.4	Heap Leach Capital Cost Estimate		
21.1.5	SX/EW Plant Capital Cost Estimate		
21.1.6	Infrastructure Capital Cost Estimate	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
21.1.7	Power Supply Capital Cost Estimate		
21.1.8	Mine Development, Rehabilitation and Closure Capital Cost Estimate		
21.1.9	Capital Cost Estimate Contingency		
21.1.10	Capital Cost Estimate Exclusions and Qualifications		
21.2	Operating Cost Estimate		
21.2.1	Project Operating Cost Assumptions	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
21.2.2	Mining Operating Cost Estimate		
21.2.3	Process Operating Cost Estimate	MillerMet Pty. Ltd.	Graeme Miller BE(Chem), FAusIMM, CP-AusIMM

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21.2.4	Power Supply Operating Cost Estimate	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
21.2.5	General and Administrative Operating Cost Estimate		
21.2.6	C1 Cash Operating Cost		
21.2.7	Transport and Logistics Operating Cost Estimate		
21.2.8	Operating Cost Estimate Exclusions		
21.3	Cost Estimate Summary		
22	Economic Analysis	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
23	Adjacent Properties	Hackman & Assoc. Pty. Ltd.	Duncan Hackman B.App.Sc., MSc., MAIG
24	Other Relevant Data and Information	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
25	Interpretation and Conclusions		
25.1	General	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
25.2	Property Description and Location	Hackman & Assoc. Pty. Ltd.	Duncan Hackman B.App.Sc., MSc., MAIG
25.3	Geology		
25.4	Mineral Resource Estimation		
25.5	Metallurgical Test Work and Process Design	MillerMet Pty. Ltd.	Graeme Miller BE(Chem), FAusIMM, CP-AusIMM
25.6	Mining Methods	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
25.7	Project Infrastructure	DRA Pacific Ltd.	Johan Du Preez BSc (Eng.), P.Eng
25.8	Water Management	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
25.9	Environmental Studies, Permitting, and Social or Community Impact	PT Lorax Indonesia	Ali Sahami, Ph.D.
25.10	Economic Analysis	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
26	Recommendations		
26.1	General	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
26.2	Geology	Hackman & Assoc. Pty. Ltd.	Duncan Hackman B.App.Sc., MSc., MAIG
26.3	Mineral Processing and Metallurgical Testing	MillerMet Pty. Ltd.	Graeme Miller BE(Chem), FAusIMM, CP-AusIMM
26.4	Mining Methods	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
26.5	Process Plant Design	MillerMet Pty. Ltd.	Graeme Miller BE(Chem), FAusIMM, CP-AusIMM
26.6	Project Infrastructure	DRA Pacific Ltd.	Johan Du Preez BSc (Eng.), P.Eng
26.7	Environmental Studies, Permitting, and Social Or Community Impact	PT Lorax Indonesia	Ali Sahami, Ph.D.
26.8	Capital and Operating Costs	Orelogy Consulting Pty. Ltd.	Ross Cheyne BE (Mining), FAusIMM
26.9	Economic Analysis		

Report Section		Company	QP
27	Bibliography	All	Sign-off by Section

The independence of all contributing QP's is ensured by the fact that none hold any equity in Asiamet Resources Limited, and the ownership of BKM rests solely with its principals.

All organizations providing QP's for this PEA have demonstrated track records in undertaking independent assessments of Mineral Resources, Ore Reserves, project reviews and audits, competent person's reports and independent feasibility evaluations on behalf of exploration and mining companies and financial institutions world-wide. Mr. Ross Cheyne (Orelogy), Mr. Duncan Hackman (H&A) and Mr. Graeme Miller (MillerMet) have the expertise and experience required to be considered a Qualified Person under the guidelines outlined in the Canadian National Instrument 43-101 for undertaking a PEA level assessment of the BKM.

All contributors to this PEA have relied on various datasets, reports and documentation provided by KSK to support the findings and conclusions of this study. The data provided was deemed to be sufficient in quantity and, to the best of their knowledge, reliable and in good stead. The contributors are not aware of any critical data that has been omitted so as to be detrimental to the objectives of this report.

2.4 Effective Date

Two effective dates are relevant to this report, as shown below;

- Effective Date of the Mineral Resources – 30 November 2015
- Effective Date of the Preliminary Economic Assessment – 19 May 2016

2.5 Project Background

BKM is located inside the tenement held PT Kalimantan Surya Kencana (KSK) under a Generation 6, Contract of Work (KSK CoW). The KSK CoW was granted April 28, 1997 between the Republic of Indonesia and KSK. The KSK CoW is centered on Long. 113 25 00 E, Lat. 00 37 00 S and sits within the Gunung Mas Regency of Central Kalimantan and lies just south of the equator (approximately 100km) and is about 190 kilometers north and slightly west of Palangkaraya, the capital city of Central Kalimantan (refer to Figure 2-1).

KSK is 75% owned by Indokal Limited (a 100% owned subsidiary of Asiamet Resources Limited *(formerly Kalimantan Gold Corporation Limited)* and 25% by PT Pancaran Cahaya Kahayan (Kahayan). Kahayan is a 99% owned subsidiary of Indokal Limited with the remaining 1% owned by Mr. Mansur Geiger (held in trust for Asiamet Resources Limited).

The KSK CoW is officially in its 5th year of exploration stage following a number of previously granted suspensions (the KSK CoW has been granted for a minimum of 38 years). The KSK CoW is in good standing with regard to meeting expenditure, social and environmental commitments and KSK possess current permits to operate within 7,422 hectares of production forest covered by the CoW.

BKM is situated in mountainous jungle terrain at the headwaters of the south flowing Kahayan River in a remote area where no permanent villages exist. Although the location is relatively isolated, BKM is accessible via a sealed road from Palangakraya to Tumbang Manggu (2.5 – 3.5 hours driving time) and an unsealed all weather road from Tumbang Manggu to BKM (2.5 – 3.5 hours driving time). BKM lies within an area classified as production forest which has already been logged. KSK was granted permission to work within the forestry area over BKM on the 23 April 2015 (permit 29/1/IPPKH/PMDN/2015. Details of this permission can be found in Appendix 10 of the BKM 2015 resource report titled "*Beruang Kanan Main Zone, Kalimantan, Indonesia; 2015 Resource Estimate Report*" (Hackman, 2015), referred to hereafter as "the resource report".

KSK, through Asiamet Resources Limited, publically reported the Beruang Kanan Main Zone 2015 Copper Resource Estimate on the 21st October 2015. The 2015 Estimate is an update of the 2014 Estimate of mineralization at Beruang Kanan Main Zone (BKM) and is based on the drill hole logging, sample assay databases and geological / structural interpretation as at 15th October. This resource forms the basis of the PEA that is reported in this document.

Figure 2-1 Location of KSK CoW (containing the BKM)

Note: ARS tenement holding in Kalimantan, showing KSK CoW and Jelai IUP

The Indonesian Government is in the process of addressing historic Contracts of Work's to ensure that they are aligned with the current mining law. KSK and the Indonesian Government are negotiating details of a non-binding Memorandum of Understanding to update terms of the KSK CoW that addresses details of 1) royalties 2) size of CoW in Exploration vs. Production 3) domestic processing 4) divestment obligations 5) State revenues and 6) prioritizing the use of local manpower and local products. KSK states that continued progress is being made and they are encouraged by their discussions with the Indonesian Government. The amendments will not alter KSK's holding in the CoW.

2.6 SITE VISITS

Duncan Hackman of Hackman and Associates (H&A) has undertaken two site inspections of the Beruang Kanan Main Zone and the KSK Tangkiling core processing facilities where historic drill core was processed and where remaining core is currently stored. The primary reason for visiting the prospect and core shed was to locate and confirm evidence of historic exploration activities reported by KSK and their historic JV partners, to observe and confirm copper mineralization in core and outcrop and to observe and review drilling and sampling protocols employed by KSK in their 2015 drilling campaign. Duncan Hackman also visited the PT Intertek Utama Services laboratory in Jakarta (ITS) to review the sampling reduction and preparation procedures employed.

Key observations and comments from the site visits are included in Appendix 8 of the resource report (Hackman, 2015).

2.7 SCOPE OF WORK

KSK, through Asiamet Resources Limited, commissioned Orelogy Consulting Pty. Ltd. (Orelogy) to evaluate the BKM Copper Project to Preliminary Economic Assessment (PEA) standards. The PEA is led by Orelogy and incorporates work from other groups including major contributors Hackman and Associates Pty. Ltd. (H&A) and DRA Pacific Pty. Ltd. (DRA).

The PEA presents details on the following, including associated background information:

- Current resource estimate
- Proposed mining methodology
- Project design criteria (mining, processing and infrastructure)
- Preliminary site plans (incl. roads, buildings, facilities, access etc.)
- Preliminary process design drawings including flow sheets and GA's
- Preliminary estimate of overall site power requirements
- List of major equipment
- Cost estimate (Capital and Operating) to within a 30% - 35% cost range subject to further study to resolve unknowns identified in the PEA
- Information from others relevant to the project

Quotes have been obtained from suppliers for mining, infrastructure items and other major capital such as earthworks. The cost estimate for the SX/EW component has been based on previous experience with similar projects. The PEA is intended to provide a view of potential project economics and to give guidance for future metallurgical testing, project design and feasibility requirements.

2.8 SOURCES OF INFORMATION

The reports and documents listed in Section 27 were used to support the preparation of the PEA. Additional information was sought from KSK / ARS personnel where required.

2.9 CAUTIONARY NOTES

2.9.1 Cautionary Note Regarding Use of Inferred Resources in Financial Analysis

This Preliminary Economic Assessment is, by definition, preliminary in nature. As such it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves, and there is no certainty that the preliminary economic assessment will be realized.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

2.9.2 Cautionary Note Regarding Forward-Looking Information

Certain information and statements contained in this section and in the Report are “forward looking” in nature. Forward-looking statements are described in more detail in Section 22 of this report.

2.9.3 Cautionary Note Regarding Production Dates

The production schedules and the annualized cashflow table of the financial analysis are presented with conceptual years shown. Years shown in these tables are for illustrative purposes only. These dates were also used as the basis for the forward-looking copper prices used.

3 RELIANCE ON OTHER EXPERTS AND SUPPLIED INFORMATION

This report was prepared by Orelogy Consulting Pty. Ltd. (Orelogy) as a Preliminary Economic Assessment (PEA) for PT Kalimantan Surya Kencana (KSK). The information, conclusions, and estimates contained in this report are based on the information available at the time of preparation, the data supplied by outside sources, and the assumptions, conditions, and qualifications set forth in this report.

The following list outlines the various contributors to this report, and their respective responsibilities. The list also includes other groups that provided studies, assessments or budgetary cost estimates that are referenced within this report.

Table 3-1 Other Experts and Contributing Parties

Company	Contribution
PT. Resindo Resources & Energy	Roads and Logistics Assessment
PT. Prastiwahyu Trimitra Engineering	Alternative Power Supply Assessment

This report is intended for the exclusive use of KSK. Any other uses of this report by any third party are at that party's sole risk.

Orelogy has taken all reasonable care in producing the information contained in this report. No inferences or conclusions should be drawn from reading any one section or part of this report in isolation.

The information in this report is not a substitute for independent professional advice before making any investment decisions.

Furthermore, any information contained in this report may not be modified without permission from Orelogy Consulting Pty. Ltd. General information and data in this report was derived from many sources including qualified person/consultants and non- "QP" sources including the KSK staff and the authors of and contributors to previous reports prepared on their behalf.

KSK advises that there is no knowledge of any environmental liabilities associated with the project. Comment and conclusions regarding tenement status, legal right to mine and explore, environmental liability etc. are based on data provided from KSK and have been accepted in good faith, and are outside the expertize of Orelogy Consulting Pty Ltd or other Independent Experts involved in this PEA.

4 PROPERTY DESCRIPTION AND LOCATION

BKM is located within the KSK Contract of Work area (KSK CoW) in Central Kalimantan, just south of the equator (Figure 2-1). It is about 190 kilometers north and slightly west of Palangkaraya, the capital city of Central Kalimantan (Figure 4-1). The BKM (centered on Long. 113 25 00 E, Lat. 00 37 00 S) is in mountainous jungle terrain at the headwaters of the south flowing Kahayan and Samba rivers in a remote area where no permanent villages exist. The location is isolated and access both to and around the prospect is difficult and imposes certain restrictions on field operations.

Details of the KSK CoW area, tenure, obligations to the Republic of Indonesia Government, environmental permitting and other details relating to the evaluation of mineralization within the KSK CoW area are described in the following sub-sections.

4.1 LAND USE

The BKM Project is located within an area that has a forestry classification of "Production Forest "and has already been logged. A forestry permit was granted to KSK on the 23rd April 2015 (permit 29/1/IPPKH/PMDN/2015, refer to Appendix 10 of the resource report (Hackman, 2015)) for an area totaling 7,422 hectares of the KSK CoW. The permit gives the rights to the holder to carry out full mineral exploration activities within the permitted concession area for two years. Both open pit and underground mining are permitted within production forest areas.

There are no other commercial undertakings covering the BKM area. BKM is located on Government land and there are no known local land owners within the project area. KSK has established a cooperative relationship with the people living in the region which has played an integral part in the facilitation of work undertaken to date.

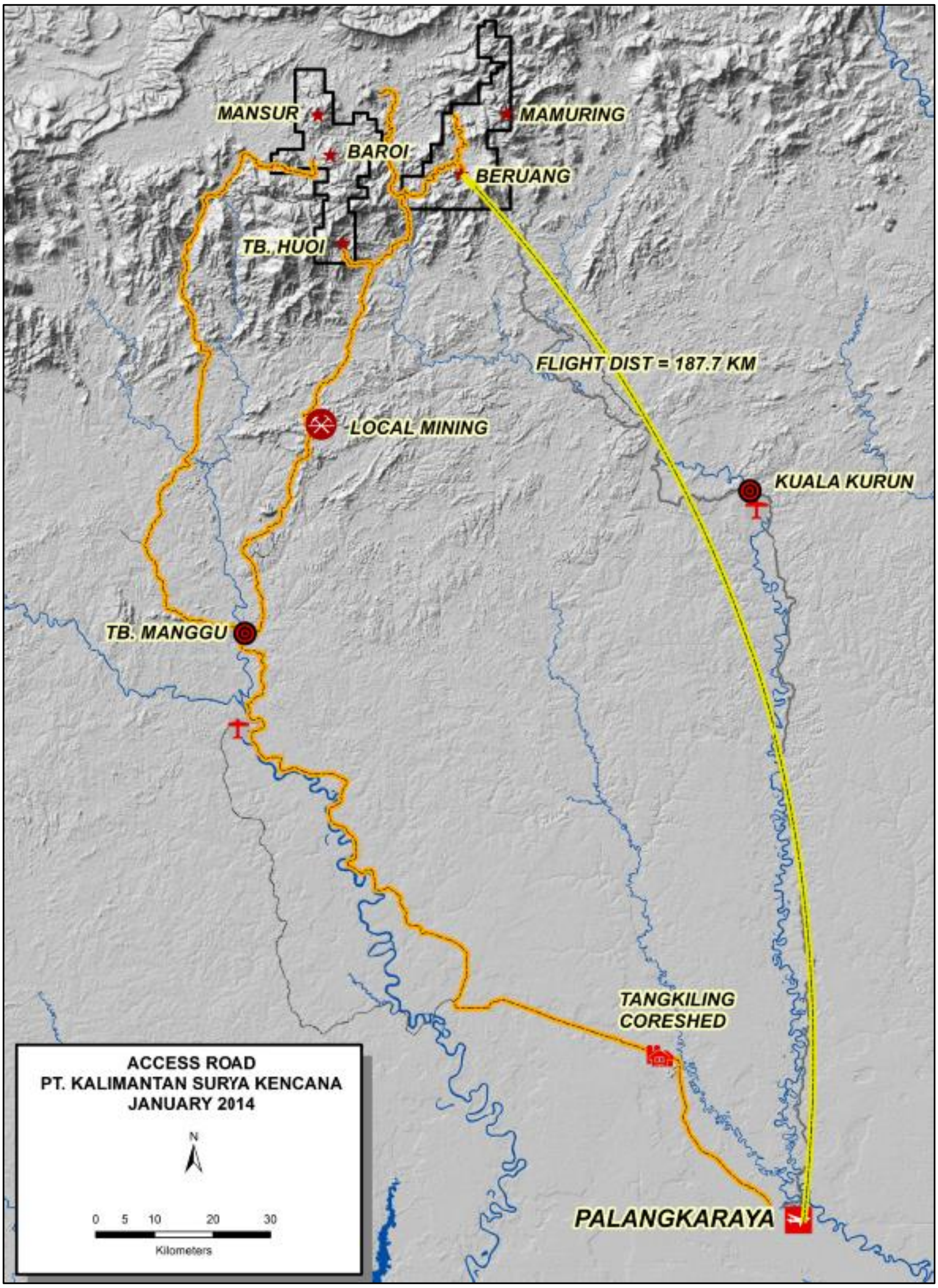
4.2 CORPORATE STRUCTURE, TENURE AND PERMITTING

The following outlines the details of the PT Kalimantan Surya Kencana Contract of Work (KSK CoW), the history and current status of the tenement and other permits required for exploring the KSK CoW.

4.2.1 Corporate Structure and Ownership of Mining Rights

PT Kalimantan Surya Kencana (KSK, incorporated in Indonesia) is the 100% owner of the 6th generation Contract of Work (KSK CoW) within which BKM is located. KSK in turn is owned 75% by Indokal Limited (incorporated in Hong Kong) and 25% by PT Pancaran Cahaya Kahayan (incorporated in Indonesia). Indokal Limited owns 99% of PT Pancaran Cahaya Kahayan with the remaining 1% owned by Mr. Mansur Geiger (held in trust for Asiamet Resources Limited). The parent company to the corporate structure is a Bermuda company, Asiamet Resources Limited (ARS), formally Kalimantan Gold Corporation Limited, which is a publically listed company on the TSX-V (Canada) and AIM (London) stock exchanges. ARS owns 100% of the shares in Indokal Limited.

Figure 4-1 Access to the BKM from Palangkaraya, capital of Central Kalimantan Province



Note: refer Figure 2-1 for location of Palangkaraya

The KSK CoW is the subject of an agreement between KSK and the Government of the Republic of Indonesia whereby in the preamble it is stated that the parties:

"Witnesseth that:

A. All Mineral resources contained in the territories of the Republic of Indonesia, including the offshore areas, are the national wealth of the Indonesian Nation;

B. The Government desires to encourage and promote the exploration and development of the Mineral resources of Indonesia. The Government is also desirous of facilitating the development of ore deposits if commercial quantities are found to exist and the operation of Mining enterprises in connection therewith;

C. The Government, through the operation of Mining enterprises, is desirous of creating growth centers for regional development, creating more employment opportunities, encouraging and developing local business and ensuring that skills, know-how and technology are transferred to Indonesian nationals, acquiring basic data regarding and related to the country's Mineral resources and preserving, and rehabilitating the natural Environment for further development of Indonesia;

D. The Company through Indokal Limited, a Company incorporated in Hong Kong has and has access to the information, knowledge, experience and proven technical and financial capability and other resources to undertake a program of General Survey, Exploration, Feasibility Study, Development, Construction, Mining, Processing and Marketing with respect to the Contract Area, and is ready and willing to proceed thereto under the terms and subject to the conditions set forth in this Agreement;

E. The Government and the Company are willing to cooperate in developing the Mineral resources hereinafter described on the basic provisions hereof and of the laws and regulations of the Republic of Indonesia, specifically Law No. 11 of 1967 on the Basic Provisions of Mining (Undang-Undang Pokok Pertambangan) and Law No. 1 of 1967 on Foreign Capital Investment (Undang- Undang Penanaman Modal Asing) and its amendment Law No. 11 of 1970 and the relevant laws and regulations pertaining thereto.

NOW, THEREFORE, in consideration of the mutual promises, covenants and conditions hereinafter set out to be performed and kept by the Parties hereto, and intending to be legally bound hereby, it is stipulated and agreed between the Parties hereto as follows :"

There are 25 Articles and 8 Annexures covering terms of the agreement that follow this preamble, and the headers of these are listed in Table 4-1 below for completeness.

Table 4-1 Headers of Articles and Annexures detailing the terms of the KSK CoW

Headers of Articles covered in the KSK CoW							
Definitions	Appointment and Responsibility of the Company	Modus Operandi	Contract Area	General Survey Period	Exploration Period	Report and Security Deposit	Feasibility Study Period
Construction Period	Operating Period	Marketing	Import and Re-Export Facilities	Taxes and Other Financial Obligations of the Company	Records, Inspection and Work Program	Currency Exchange	Special Rights of the Government
Employment and Training of Indonesian Nationals	Enabling Provisions	force Majeure	Default	Settlement of Disputes	Termination	Cooperation of the Parties	Promotion of National Interest
Regional Cooperation In Regard To Additional Infrastructure	Environmental Management and Protection	Local Business Development	Misc. Provisions	Assignment	Financing	Term	Governing Law
Headers of Annexures covered in the KSK CoW							
Contract Area	Map of Contract Area	List of Out Standing Mining Authorizations	Deadrent for Various Stages of Activities	Feasibility Study Report	Royalty On Mineral Production	the Implementing of Royalties	Rules for Computation of Income Tax

The original KSK CoW was signed on the 28th April 1997 for a minimum period of 38 years. It has since become part of an amalgamated title and its history is outlined in Section 4.2.2.

4.2.2 Tenure History and Status

The following outlines the tenure history of the KSK CoW.

KSK held an 80% interest in the now terminated company PT Pancaran Paringa Kalimantan who was the holder of the 4th generation PPK CoW. On August 16, 1999, by decree of the Government of the Republic of Indonesia, the KSK CoW and nearby PPK CoW were amalgamated into the one holding (KSK CoW, effective date of April 28, 1999). As a result of the amalgamation the KSK CoW comprises two blocks, A and B.

On August 24 2004, 5,100ha was added to the KSK CoW making the maximum holding of the KSK CoW 129,290ha. According to the conditions of the CoW, KSK has since relinquished ~50% of the tenement area in two stages so that the current holding now stands at 61,003ha. The next relinquishment is scheduled to coincide with the completion of the feasibility stage of the tenement. KSK has requested confirmation from the Government of Indonesia (GOI) that the KSK CoW is currently in the fifth year of its recognized exploration stage as stated in a letter from the government dated 28th October 2015 (refer to Appendix 10 of the resource report (Hackman, 2015). The exploration stage is to be followed by a minimum 1 year feasibility stage (believed to commence 29th October 2016), an approved construction stage and 30 years of production stage (both the exploration and feasibility stages can be extended by request to the Republic of Indonesia Government).

KSK has signed a non-binding Memorandum of Understanding (MOU) with the Government of the Republic of Indonesia (GOI) covering amendments to its KSK Contract of Work. The CoW system provides security of tenure for a minimum of 38 years of exploration, development and operations and KSK continues discussions with the GOI regarding possible amendments to some of the KSK CoW terms in order to achieve closer alignment with the current Law No. 4/2009.

Following the completion of negotiations, items contained within the MOU will be incorporated as an amendment to the CoW. Pursuant to the MOU, and subject to final negotiation, agreement has been reached in principle on the following six points:

1. The size of the CoW shall remain unchanged at 61,003 hectares.
2. The MOU contemplates that after 30 years of Operating under the CoW, the Company may apply to continue operations in the form of a Special Mining Business License for a further 2 x 10 year periods.
3. Under the agreed MOU terms the corporate income tax rate will continue to be as prescribed in the CoW, that being "*thirty percent (30%) or lower rate as set forth by the Government regulations*", but royalties will now follow the provisions of the prevailing law. Gold and copper royalties under the prevailing laws are 3.75% and 4%, respectively.
4. The CoW currently has a provision that requires the Company to work towards, and assist, the Government in supporting the policy of establishing metals processing facilities in Indonesia in relation to smelting and refining. The Company is now under obligation to process and refine the mineral ores domestically in line with the current provisions of the rules of law in Indonesia.
5. The Company's Indonesian subsidiary that holds the CoW is a Foreign Investment Company ("PMA"). Current law mandates that Indonesian Nationals or Companies be offered the opportunity to invest in a PMA Company, the level and timing of divestment being dependent on the type of mining and processing. As an example, the current regulation for a PMA company holding an IUP (mining license) that is conducting open pit mining and undertaking its own processing and/or refining activities is divestiture of 20% at year 6, 10% in year 10 and a further 10% in year 15, for a total of 40% divestiture over 15 years. The divestiture of shares is to be at fair value and subject to pre-emptive rights allowing holders to maintain relative percentage ownership. Pursuant to the MOU, shares of a PMA, listed on the Indonesia Stock Exchange may be recognized as a 20% Indonesian shareholding.
6. The CoW currently contemplates the priority use of local labor, products and registered mining service companies and the MOU reinforces this requirement.

In 2014 H&A noted that the online Directorate General of Minerals and Coals WEB GIS tenement map showed that the northwest region of the KSK CoW had a conflicting and overlapping boundary with a later granted IUP issued to PT. Persada Makmur Sejahtera. KSK requested and received confirmation from the Ministry that there is no overlapping of the KSK CoW and the PT. Persada Makmur Sejahtera IUP (refer to

Appendix 10 of the resource report (Hackman, 2015)) and that the WEB GIS was wrong and would be corrected. The correction of the WEB GIS has not yet been undertaken.

4.2.3 Environmental Permitting

Indonesian environmental laws require the preparation of an environmental study for projects requiring an Exploitation Permit. This is generally undertaken as part of the Feasibility Study. As BKM and the CoW are not at the feasibility stage there is no requirement at this point in time for KSK to undertake an environmental study. Further detail on the current status and future requirements for environmental permitting is provided in Section 20.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 ACCESSIBILITY

The BKM site is approximately 190 kilometers northwest of Palangkaraya, the capital city of Central Kalimantan. It is approximately six hours drive by sealed and unsealed road, or 50 minutes flying time by helicopter from Palangkaraya. Daily air flights connect Jakarta with Palangkaraya.

The site is approximately eight to ten hours travel by vehicle (approx. 400km) from Banjarmasin, the nearest port and main port of South Kalimantan.

Logging roads provide direct access into this previously remote area. Access to the project is currently via foot from the field camp, which is located on its eastern side of the project at the base of the main ridge.

The BKM Project (centered on Long. 113° 25′ 00″ E; Lat. 00° 37′ 00″ S) is in hilly to mountainous terrain at the headwaters of the south flowing Kahayan River in a remote area where no permanent villages exist. However KSK has established a cooperative relationship with the people living in the district which has played an integral part in the facilitation of work undertaken to date.

5.2 PHYSIOGRAPHY

The topography in the Beruang area is mountainous with the mountains being moderate to locally very steep. Drainages in the area are dendritic. The area is thickly vegetated and the rugged topography ranges in elevations from 300 meters to about 1,000 meters Above Sea Level (ASL).

Mine site topography can be described as rolling to rugged. BKM topographical conditions can be divided into two parts, north and south. The northern part is a hilly area with an elevation between 360 m ASL - 580 m ASL with slopes varying between 20° – 30°. The southern part is a hilly area with an altitude between 330m ASL – 480m ASL, and surface gradients ranging from 10° – 20°.

The low plains are cut by seasonal water courses, where some have formed deep gullies with eroded banks. Large drainage systems to the west and east of the BKM area present as potential sites for locating mining infrastructure.

5.3 CLIMATE

Indonesia has a typically tropical climate with two seasons, wet and dry. In most of the country the dry season occurs from May to October with the wet season occurring in the rest of the year. Humidity is on average a minimum of 60% with an average temperature of 25 to 27 degrees Celsius all year round.

The Central Kalimantan climate is a wet weather equatorial zone with rainfall or precipitation of 2,800 - 3,400 mm per annum with an average of 145 rainy days annually. Rainfall at BKM would generally sit at the higher end of this scale; with frequent and rapid changes to river levels (rises of 2 to 3 meters in river levels are not uncommon). Erosion is rapid, resulting in steep-sided river valleys where landslides are common. The nature of the overlying topography will have an impact upon access road design and ongoing mining operations.

5.4 LOCAL RESOURCES

There are no villages or significant habitation in the BKM area.

The BKM area has been logged and a lot of the forest now present is regrowth following this activity. Local artesian miners are scattered throughout the region. None are active within the BKM area. Apart from logging roads, the area has little infrastructure.

6 HISTORY

The history of the KSK CoW tenure is detailed in Section 4.2.2. The following briefly outlines the operational and exploration history of the KSK CoW and incorporated Beruang Kanan area.

6.1 OPERATIONAL HISTORY

Recorded exploration on the KSK CoW essentially started in 1981 when PT. Pancaran Cahaya Mulia (PCM) and Sinar Enterprises International B.V. intended to explore the area. PCM changed its name to PT. Pancaran Bahagia (PCB) and in the same year hired two expatriate geologists Mansur Geiger and Mathew Mayberry. Mr. Geiger is currently the President Director of KSK and has dedicated his career to the area.

Mr. Geiger and Mr. Mayberry conducted reconnaissance surveys from 1982 until 1985 into the upper Kahayan area. Access to the area in these early days was only by small boat and was very difficult. Mr. Geiger reports it took two weeks to get from Palangkaraya to the Beruang Kanan camp. This period of exploration was undertaken primarily for placer gold.

In 1985, the exploration emphasis changed to looking for hard rock epithermal gold. To finance the exploration, a joint venture was signed with Molopo, an Australian mining company. The vehicle used by Molopo for this joint venture was PT Pancaran Paringa Kalimantan (PPK). The joint venture agreement between PCB and PPK was signed on October 7, 1985. The agreement provided PCB with a 20 percent interest and Paringa Mining and Exploration Company PLC, subsidiary of Molopo with an 80 percent interest in a fourth generation CoW. This CoW was signed between PPK and the Republic of Indonesia on December 2, 1986. The original CoW covered 613,700 hectares but it did not include mining permits (KP's) also held by the joint venture. The original CoW over time was reduced in size to 33,170 hectares and added as Block A to the current KSK CoW. All of the KP's were relinquished.

During the joint venture exploration phase, several areas were recognized as having potential for porphyry copper style mineralization, specifically the Beruang Kanan and Tumbang Huoi prospects. In 1990 the Molopo/PPK joint venture was dissolved.

In 1992, Kalimantan Investment Corporation (KIC) took over field operations from Molopo and PPK. The new company consisted of essentially the same people that formed PCB. During this period (1992-95) the Tumbang Huoi and Mansur prospects were evaluated by IP and diamond drilling.

PT. Cyprus Indonesia signed an option for the Mansur Prospect, which gave Cyprus the option to earn a 67.5 percent interest in the prospect. In December 1996 Cyprus terminated their option.

Kalimantan Gold Corporation Limited (KGCL) was formed during May 1996 and listed on the then Vancouver Stock Exchange. KGCL made application through KSK to the Department of Mines for a 121,900 hectares sixth generation CoW subsequently officially granted on April 28, 1997. The details and history of this current CoW are outlined in Section 4.2.2. On January 14, 2015, KGCL through a private share placement acquired a 40% interest in the Beutong copper-gold project in Sumatra, Indonesia and completed changes to management. On July 24, 2015, KGCL changed its name to Asiamet Resources Limited (ARS) which is listed on both the TSX Venture Exchange in Canada and the AIM in England.

Exploration and evaluation of the KSK CoW has centered on four main areas (Baroi, Beruang Tengah, Beruang Kanan and Mansur, refer to Figure 4-1) where KSK and two consecutive Joint Venture partners Oxiana Limited and Eksplorasi Nusa Jaya (ENJ) focused on identifying porphyry mineralization at the prospects. During their involvement, ENJ also undertook delineation drilling of the near surface mineralization at BKM. In 2015 KSK continued the delineation drilling of BKM and the ENJ-KSK and KSK drilling data constitutes the majority of the data utilized in preparing the 2015 Resource Estimate for the project.

6.2 EXPLORATION HISTORY – BERUANG KANAN MAIN ZONE

Detail of the surface exploration activities and results are not included in this report as they have been documented in a report by (Munroe S, and Clayton M, 2006). The details of the drilling and evaluation activities pertinent to the estimation of resources at BKM are included at Sections 9, 0 and 0.

The following exploration and evaluation of the BKM area has been conducted over the last 18 years:

- 1997 to 2004 - KSK:
  - Field mapping and rock chip sampling
  - Outcrop channel sampling
  - 200m by 50m soil sampling
  - Dipole dipole IP
  - Drilling 17 holes totaling 3,631m
- 2005 to 2007 - Oxiana Limited and KSK JV:
  - Reprocessing of IP data
  - Drilling 5 holes totaling 2,450m
- 2012 to 2013 - ENJ and KSK JV:
  - Aerial magnetic, gravity and LIDAR survey
  - Field mapping and rock chip sampling
  - Drilling 32 holes totaling 11,851m
- 2014:
  - Maiden resource estimate undertaken
- 2015 – KSK
  - Drilling of 71 holes totaling 6,178m
  - Resource estimate update

KSK continue to advance the BKM Project and this Preliminary Economic Assessment (PEA) represents the next step in this process.

7 GEOLOGICAL SETTING AND MINERALIZATION

The following details the current understanding of the regional geology and the geological setting and mineralization styles of the Beruang Kanan Main Zone.

7.1 REGIONAL SETTING

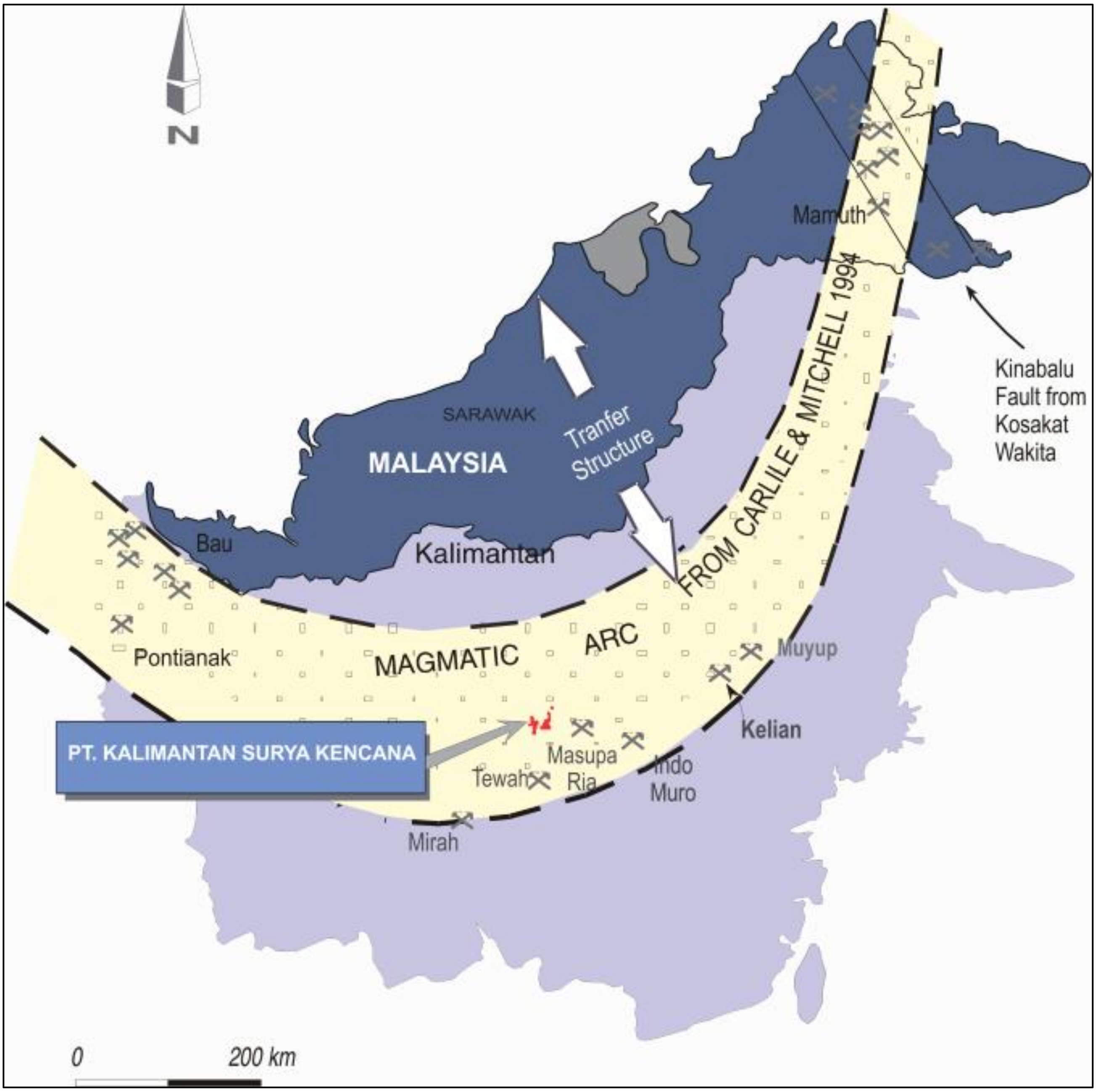
The KSK CoW is situated within a mid-Tertiary age magmatic arc (Carlile JC, and Mitchell AHG, 1994) that hosts a number of epithermal gold deposits (e.g., Kelian, Indon, Muro) and significant prospects such as Muyup, Masupa Ria, Gunung Mas and Mirah (Figure 7-1).

Porphyry style copper-gold mineralization in the KSK CoW is associated with a number of intrusions that have been emplaced at shallow crustal levels at the junction between Mesozoic metamorphic rocks to the south and accreted Lower Tertiary sediments to the north. These intrusions are interpreted to be part of the Oligocene Central Kalimantan arc of (Carlile JC, and Mitchell AHG, 1994) . Older intrusions, and associated volcanic and volcaniclastic rocks, of probably Cretaceous age also outcrop along this contact (Carlile JC, and Mitchell AHG, 1994).

Structures in the region are dominated by a northeast striking set of faults that are interpreted to be features of the Kalimantan Suture (van Leeuwen et al., 1990) and are probably arc parallel, or accretionary, faults. Subsidiary northwest trending arc normal, or transfer faults cross-cut the northeast structures. The mid-Tertiary intrusions have commonly been emplaced within dilational settings at the intersection of these major structural features. The major gold prospects and deposits in Kalimantan are also localized in a similar structural setting (Corbett GJ, and Leach TM, 1998). The shallow level intrusions are apparent as major anomalies on aero magnetic survey data.

Large circular features, that are evident on satellite, Landsat, radar, and aerial photo images commonly coincide with the mid-Tertiary intrusions and associated magnetic high anomalies. These circular structures are interpreted to be volcanic collapse features and they host many of the porphyry copper-gold prospects within the KSK CoW. To date, more than 38 porphyry and porphyry-related copper and/or gold prospects have been defined in the KSK CoW, and only a few of these, namely the Baroi, Mansur and Beruang prospects have undergone any detailed exploration.

Figure 7-1 Location of significant mineralization within the Central Kalimantan Island Arc



Note: after (Carlile JC, and Mitchell AHG, 1994)

7.2 BERUANG KANAN GEOLOGY AND MINERALIZATION

The following description is taken from a KSK internal report "*EXPLORATION SUMMARY REPORT, 1997 through 2007*" (author not stated). The report lists a comprehensive reference list containing reports by consultants and KSK personnel who have worked and reported on the BKM prospect.

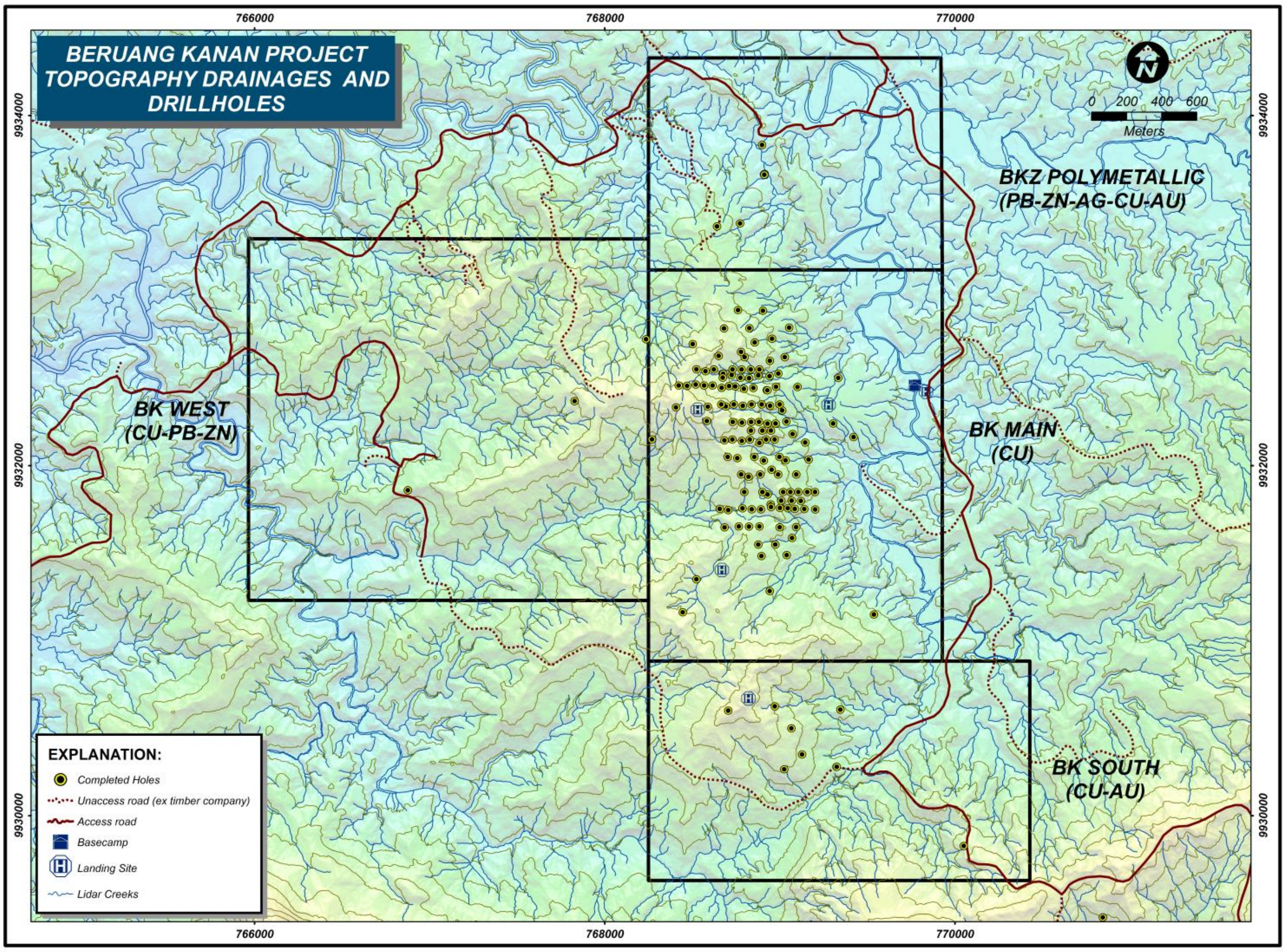
The Beruang Kanan prospect is defined by a 16km^2 zone of propylitic, local phyllic, and rare advanced argillic altered sequence of dacitic tuffs and sediments returning greater than 200ppm Cu in soils. It is situated in the Central Eastern portion of the KSK CoW (refer to Figure 23-1).

The Beruang Kanan mineralization is hosted in a sequence of dacite tuff of probable Oligocene age that overlies lower Tertiary volcaniclastic siltstone and sandstones in the eastern prospect area. Pre-mineral Sintang dacite porphyry intrusions of probable Oligocene age, and post-mineral (may be Miocene) andesite, dacite to basalt-gabbro dykes are intruded into the tuffs and sediments.

Geological, geochemical (soil, ridge and spur, auger and rock chip) and geophysical (IP and ground magnetics) surveys delineate three centers of possible porphyry-style alteration and mineralization; the Main, South and West Zones (Figure 7-2). These are:

- The South Zone; consisting of a 1km northeast striking zone of quartz-sulphide stockwork veining with anomalous gold, molybdenum and copper soil geochemistry.
- The Central Zone; consisting of locally intense sericite-quartz-pyrite alteration, stockwork quartz + sulphide veins, weak anomalous Au-Mo in soils and a deep IP anomaly. A broad outer halo of anomalous copper-zinc-arsenic-antimony in soils is hosted in intensely chlorite-pyrite altered tuff. Sporadic quartz-pyrite-chalcopyrite (with minute native gold inclusions) veins cut the chloritized tuff.
- The West Zone; defined by scattered zones of phyllic alteration and anomalous copper and base metal geochemistry which suggest that in the West Zone the system is less well defined.

Figure 7-2 Prospects within the Beruang Kanan area



Note: This utilizes the resource estimate for the estimated mineralization within the BK Main zone only as shown above.

Exploration activity to date at Beruang Kanan has been focused on the Main zone (the subject of this resource estimate), that is defined by a north-south elongate, 1.0km by 1.5km area of anomalous copper (>0.1% in rock chips) ± gold-molybdenum geochemistry, high chargeability, and by intense phyllic alteration. The phyllic alteration is capped at high elevations on the western margin of the Main zone by advanced argillic alteration. The alteration and mineralization are hosted almost entirely in dacite tuff and are cut by post-mineral dacite dykes. These dykes defined by ground magnetics, are up to 100-200m wide, and radiate northwest and west through the prospect area. Copper-gold-molybdenum in soils are aligned northeast within the Main Zone.

Zinc, and to a lesser extent lead, form broad anomalous geochemical halos around the Main Zone, and massive northeast trending zone of polymetallic mineralization outcrop to the north and south of the Main zone. Limited drilling into the zones to the north intersected intervals of up to 16m @ 2.8%Pb, 5.8%Zn, 58 g/t Ag, 0.65 g/t Au and 0.17% Cu (BKZ-1) associated with quartz-chlorite-illite alteration. Base metal sulphide (sphalerite, galena, tennantite, and chalcopyrite-bornite) mineralization occurs as wallrock disseminated grains, in shear zones and as sheeted veins.

In the Main Zone, drilling has intersected a north-northwest trending zone of intensely sheared and silicified, highly pyritic, zoned phyllic to advanced argillic altered rock. This zone was found to host copper grades of up to 167m @ 0.59% Cu.

Early quartz veins, that are commonly re-crystallized and strained, contain rare anhydrite and apatite inclusions and are locally cut by thin anhydrite veinlets. Subsequent shearing of the quartz veins was accompanied by wallrock alteration and vein deposition of mineral assemblages that are zoned temporally from early advanced argillic, through intermediate phyllic to late stage argillic and sub-propylitic.

The alteration assemblages and associated mineralization are also spatially zoned. From deeper levels in drillholes to the south-east to shallow levels in drillholes to the northwest the zoning is: chlorite + pyrite + chalcopyrite ^ sericite + pyrite + chalcopyrite ± bornite ± sphalerite ± galena ^ dickite/kaolinite + pyrite + bornite + chalcopyrite + tennantite ^ alunite + pyrite ± barite ± enargite. This zonation indicates progressive oxidation and decrease in fluid pH as the hydrothermal fluids migrated to cooler and shallow levels northeastward from a source inferred to lie at depth to the south-east. Supergene chalcocite and covellite overgrow and replace many of the primary sulphides and account for much of the copper mineralization at shallow levels.

8 DEPOSIT TYPES

The deposit type and structural controls on mineralization and discussion on deposit type can be found at Sections 12.1.3 and 12.2.1 and follows direct investigations of copper grade and core photography re-logging undertaken as part of the work to model and estimate the resources at the BKM prospect.

9 EXPLORATION

Detail of the surface exploration activities and results are not included in this report as they have been documented in a report by (Munroe S, and Clayton M, 2006). A brief outline of these activities is included in Section 6.2.

The details of the drilling and evaluation activities pertinent to the estimation of resources at BKM are included at Sections 10, 0 and 0.

10 DRILLING

The BKM prospect has been a focus of copper exploration in the KSK CoW for 18 years, being the subject of drilling for KSK and joint venture partners in seven distinct programs (refer to Table 10-1). Prior to 2015, KSK and Oxiana Limited (in Joint Venture with KGCL) undertook shallow to moderate depth exploration drilling (max ~600m) and identified that a near surface body of mineralization could exist at BKM. ENJ (in Joint Venture with KSK) undertook definition (delineation) drilling of this mineralization and drilled three deep holes (>1000m) into BKM. Moderate and deep holes drilled by Oxiana and ENJ were targeted to test for porphyry style mineralization at BKM. In 2015 KSK drilled 71 holes into and peripheral to the mineralized zones to better define and understand the copper mineralization at BKM. The 2015 KSK and historic drilling underpin the 2015 resource estimate. The quality assurance and quality control practices attached to the 2015 drilling has enabled the evaluation of and improved understanding of risks to the project (as identified in 2014) which in turn has improved confidence in and an upgrade of areas within the resource. For a list of significant intercepts refer to Appendix 7 of the resource report (Hackman, 2015).

All holes were drilled utilizing diamond drill rigs. The historic holes typically started at PQ or HQ core sizes, reducing to NQ and BQ when required due to drilling conditions and rig capabilities. The 2015 holes were drilled with HQ triple tube running gear and 1.5m core barrels. A list of core diameters through the significant mineralized intercepts is presented at Table 10-2. Approximately 73% of the mineralization has been sampled with HQ core. Shallower mineralization has been intercepted with PQ and HQ core while deeper mineralization intercepted with NQ and BQ core. The differences in average Cu grades reported in Table 10-2 could be attributed to grade tenor changes with depth or to the effect of the fundamental sampling error. No work has been undertaken to identify the reason for the grade differential between drill core sizes.

Table 10-1 Diamond Drilling within Beruang Kanan Prospect Area

Program	AREA	Drill Corp	Rig	Start Date	End Date	Number of Holes	Average Depth	Total Meters
KSK Phase I	**Main Zone**	**R&B**	**R&B Rig**	**13-Jan-98**	**29-Apr-98**	**10**	**192**	**1921**
KSK North Poly	**North Polymetallic**	**R&B**	**R&B Rig**	**6-May-99**	**2-Jul-99**	**6**	**145**	**871**
KSK Phase II	Main Zone	R&B	R&B Rig	14-Apr-01	4-Aug-01	7	244	1710
	South Zone	R&B	R&B Rig	13-Mar-01	29-Mar-01	2	129	258
	KSK Phase II Total			**13-Mar-01**	**4-Aug-01**	**9**	**219**	**1967**
JV Oxiana	Main Zone	ANTERO	AD1000/ AD500	25-Apr-07	18-Jul-07	5	490	2450
	South Zone	ANTERO	AD500	26-Jun-07	19-Jul-07	2	349	698
		KSK RIG	Rig34	21-Apr-07	14-Aug-07	2	227	455
	West Zone	ANTERO	AD500	22-Jul-07	8-Apr-07	1	279	279
	JV Oxiana Total			**21-Apr-07**	**14-Aug-07**	**10**	**388**	**3882**
Definition - KSK-ENJ	Main Zone	KSK RIG	Jackro-MUT240/ Rig34	3-Jul-12	15-Jul-13	29	300	8696
	South Zone	KSK RIG	Jackro MUT240	14-Apr-13	13-May-13	2	300	600
Definition Drilling - KSK	Main Zone	DBM RIG	DBM240	18-Jul-15	19-Sep-15	16	106	1695
		KSK RIG	Jackro MUT240	12-May-15	18-Sep-15	24	86	2070
	South Zone	DBM RIG	DBM240	7-Jul-15	17-Jul-15	4	82	326
		KSK RIG	Rig34	21-May-15	4-Sep-15	27	78	2096
	Definition Total			**3-Jul-12**	**19-Sep-15**	**102**	**152**	**15483**
Deep - KSK-ENJ	Low Zone	KSK RIG	AID350	3-Jun-13	10-Jul-13	2	601	1203
	Main Zone	PONTIL	Duralite#2	7-Jun-12	27-Jan-13	3	1052	3155
	South Zone	PONTIL	Duralite#2/ LF130#2	12-Aug-12	7-May-13	3	826	2478
	Deep Total			**7-Jun-12**	**10-Jul-13**	**8**	**854**	**6836**
Grand Total				**13-Jan-98**	**19-Sep-15**	**145**	**214**	**30960**

Note: includes re-drills and 20 holes located at adjacent prospects - The BKM 2015 Resource Estimate is centered on the Main Zone mineralization).

Table 10-2 Core sizes through mineralized intercepts

Core Size	Number of Intervals	Length (m)	Av. Depth (m)	Av Cu Grade (ppm)
PQ	61	117	13	8934
HQ	2210	3245	47	8080
NQ	279	815	119	5279
BQ	34	95	202	6980
Historic/Unknown	66	179	129	6147
Total	**2650**	**4451**	**65**	**7743**

Sample lengths are listed in Table 10-3 below.

Table 10-3 Sample Lengths Used

Domain	Sample Length (m)	Count
Mineralized	<1.0	13
	1.0	1380
	1 to 1.5	91
	1.5 to 1.99	30
	2	466
	2.01 to 2.5	56
	2.5 to 2.99	49
	3	563
	3.01 to 3.5	93
	>3.5	3
Mineralized Total		**2744**
Non-Mineralized	<1.0	17
	1.0	826
	1 to 1.5	319
	1.5 to 1.99	177
	2	2234
	2.01 to 2.5	618
	2.5 to 2.99	492
	3	3797
	3.01 to 3.5	873
	>3.5	34
Non-Mineralised Total		**9387**
Total BK Samples		**12131**

Note: These intervals have been employed regardless of drill core size. The most common sample length in mineralization is 1m (52% of dataset) with 2m and 3m forming 24% and 24% respectively.

These lengths have been employed by all workers and for all core sizes. Approximately 46% of the samples within the mineralized domains are ≥2.0m (with 24% being ≥3.0m). Half core sampling was employed, generating the following nominal sample sizes for these long intervals:

- PQ: 22kg
- HQ: 12kg
- NQ: 7kg
- BQ: 4kg

There is no experimental data available to assess the suitability of sample sizes with respect to the fundamental sampling error (considers the in situ heterogeneity of the mineralization) or if the sample reduction strategy employed is appropriate. The generalized sample mass nomogram proposed by Gy shows that crushing to 1.5mm is the maximum particle size recommended for reduction to a 1kg sub-sample mass to ensure that sample comminution strategies remain in the safe zone (for unknown material). Historic workers at BKM have reduced to 1kg at -4mm crush size (Gy's standard nomogram recommends reducing to

5kg at this crush size) and the KSK 2015 program reduces to 1kg at -2mm crush size (Gy's standard nomogram recommends reducing to 1.5kg at this crush size).

The effect of this significant deviation from recommended comminution strategies cannot be determined from the current dataset. Two opposing observations can be made of the sub-sampling strategy:

- The first is that the copper mineralization, being vein hosted, can be considered to have properties not dissimilar to gold, i.e. nuggetty in nature. If this is the case then crush or grind sizes must be small to maximize sample homogeneity before sub-sampling is undertaken. [NB. The suspected inappropriately small primary sample size of NQ and BQ core may be the reason that the average copper grades of the small samples is lower than the average grade of the larger samples (HQ and PQ core).]
- The second is that the uniform copper grades within the dataset suggest that BKM mineralization is adequately homogeneous when crushed to -4mm for sub-sampling to 1kg sample mass. Twin holes, core and crusher duplicate samples are required to verify if this is the case. Coarse Crush and split duplicates from the KSK 2015 program on the whole support that -2mm comminution particle size is acceptable for the BKM mineralization, however reproducibility of copper assays between duplicate pairs deteriorated in the latter part of the program (see Section 12.2.3.2 where observations and possible reasons for this phenomena are discussed).

H&A cannot discern the effect on sample reliability or risk to the resource estimate related to the sampling and sub-sampling procedures employed for the BKM mineralization and this unknown is a consideration to be factored into the classification of the BKM 2015 Resource Estimate.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 PRE 2015

Detailed documentation on security, sampling and core yard procedures specific to the pre-2015 drilling at BKM is non-existent.

Descriptions of work by KSK have been reported by (Geiger M, and Prasetyo D, 2004). Although this report focusses on activities at the Baroi Project, it is reasonable to assume that similar procedures were undertaken at BKM where drilling was undertaken during the same time period. (Geiger M, and Prasetyo D, 2004) report:

"The Company's diamond drill programs are designed by and implemented under the supervision of Mansur Geiger, Vice President Exploration and Didik Prasetyo, Senior Project Geologist.

The drill core is lifted from the drill and placed into a core box, which is 1m in length and usually has 5 divisions across. Core boxes are marked with the drill hole number, box number, and the top and bottom of each box is clearly marked. A wooden marker is placed at the end of each drill run with the depth to the bottom of each drill run marked on the divider. Drill runs are laid into the core boxes in consecutive order running from top to bottom. Drill cores are recorded in a drill log detailing:

- *the hole number,*
- *depth from, to, and length of each drill run in meters,*
- *% recovery per drill run and relevant notes, and*
- *measurements are taken as the cores are laid into the core sample boxes.*

Completed core boxes are kept at the drill site until moved, usually two at a time, to the central drill core logging area.

Once at the central drill core logging area, the core boxes for each hole are laid out and drill log cover sheets are prepared for each hole under the supervision of the senior geologist. A check is made that the hole was drilled in accordance with the drill plan and that the driller's information is correct. The drill log cover sheet is then completed detailing:

- *hole number,*
- *location, coordinates, bearing, inclination, and total depth,*
- *date commenced and completed,*
- *% recovery,*
- *drilling supervisor,*
- *core size, and*
- *signature of person completing the log.*

The senior geologist carries out a detailed geological analysis of each drill run at various depths, usually where a geological change has occurred, and completes a detailed drill core log sheet. The detailed log is a structured, hand-written analysis of each hole covering lithology, alteration and mineralization, with a detailed description of drill cores at various depths.

The drill core is marked as to exactly where sections of the drill core are to be cut based on mineralogy and geological composition. Selected drill core sections are then dissected with a one-half section of the drill core placed/swept into a plastic bag. The other half is placed back into the core box. The hole number and depths from/to are written in indelible ink on both sides of the plastic bags. In addition, an aluminum tag with both the hole number and depths is placed inside each bag. The plastic bags are then sealed by wrapping masking tape completely around the bag with the hole number and the depths written again on the masking tape.

The cut drill core samples that have been placed back into the core boxes are re-analyzed for any further relevant geological information and the detailed log is updated.

The completed drill log cover sheets and detailed drill core log sheets are sent to the KSK office for entry into the drilling database. All details of the core sample from each hole are entered into the drilling database maintained at the KSK office in Palangkaraya. A drill hole summary report is produced for each hole from the drilling database for the Vice President Exploration.

Under the supervision of the senior project geologist, approximately four of the bound plastic core sample bags from the same hole are placed into a larger sample bag and tied with string at the top. The larger sample bag is then marked clearly with the word 'SAMPLE' on the front and back. A detailed list of the core samples in each bag is produced at the drill site, a copy of which is kept with each bag.

The large sample bags are then transported to the KSK office in Palangkaraya by boat/road or helicopter. Each sample bag received remains sealed and is checked against the accompanying detailed list prepared at the drill site and is inspected for damage under the supervision of the senior office geologist. The contents of any damaged bag is packed into a new bag and labeled. Smashed bags are rejected and not sent to the laboratory for assay.

The core sample bags are packed into a new sack and an assay sample dispatch sheet pre-printed by the assay company, PT Indoassay Laboratories in Balikpapan, is prepared. The assay sample dispatch sheet includes all sample numbers, with hole numbers and depths, sample type and elements to be recorded. The core sample bags are then sent by commercial courier to PT Indoassay Laboratories together with the original assay sample dispatch sheet, copies of which are kept in data and geological files.

When the core samples and accompanying dispatch sheets are delivered, PT Indoassay Laboratories confirms receipt to the KSK office. They produce pulp and residue from each core sample bag and 50 grams of pulp is used to conduct assay analysis. The results of the assay from each sample are sent to the KSK office by email *and then in hardcopy via courier, and are entered into the mining database. Results include the hole number, starting depth, ending depth, total length and percentages of various minerals in each sample.*

Residue for each sample is placed into a new plastic bag by the laboratory and labeled with reference numbers.

Pulp is placed in an envelope and labeled accordingly. The residue bags and pulp samples are then sent back to the KSK office by commercial courier. These are stored in a secure archival building in Palangkaraya, away from the KSK office."

There is no record of how Oxiana Limited core was handled during their involvement at BKM.

Details of core handling and sampling undertaken by ENJ during their involvement at BKM is recorded in an internal document "*Sample Preparation and Assay Quality Control report, 30 January 2014*" (refer to Appendix 3 of the resource report (Hackman, 2015)), which describes the following activities:

"This report address the assay quality collected from the diamond drill core and geochemical samples during the Exploration program starting May 2012 to December 2013 at KSK CoW area at Kalimantan Tengah, Indonesia.

Geoassay Laboratory was chose to give its services for sample preparation and assaying. The sample preparation established at Tangkiling at about 35 km from PT ENJ and PT KSK main office at Palangkaraya.

The total amount of 18,522 samples consisted of 10,852 drill core and 7,670 geochem samples were sent to laboratory for prep and analysis.

The preparation and assay procedures utilized by Geoassay follow the Standard Operating Procedures (SOP) developed by PT ENJ and Geoassay to suite the conditions and criteria required for KSK samples.

All the drill core and geochem samples from work site are transported from Marinyuoi to Tengkiling. The arrival of samples at Tengkiling is confirmed with paperwork transfer. Trips between the Marinyuoi core handling facility and sample preparation area generally occur about 2 times per week. Upon arrival at Tengkiling, containers are unpacked and checked against the shipping orders from Marinyuoi. A sequential KSK job number is assigned and written on a laboratory worksheet and the ENJ transmittal form.

The core is marked for sawing to split for assay and storage. The core is split longitudinally with a core saw. Conventional splitters are also available for small diameter core. Half of the core is returned to the core box after splitting and the other half is bagged and numbered for sample preparation processing by GeoAssay personnel in the building adjacent to PT KSK's core shed. The samples are then processed and finally placed into kraft paper bag and shipped to the GeoAssay Analytical Laboratory in Cikarang, Jakarta (GA) for assaying. Transmittal and assay instruction forms accompany the sample shipment to GA.

The sample preparation work effective started on July 3, 2012 according to the following procedures:

1. *The samples are weighed before drying in an oven for a maximum of 8 hours at 105°C. Samples weights are also taken after drying and recorded on the transmittal sheet.*
2. *The entire sample (half core) is placed into a jaw crusher; the output is crushed to between -8mm and -10mm. All the crushed material is then fed to the Boyd RSD Combination crusher and splitter with nominal output size of –4mm.*
3. *The rotary splitter opening is set to get about a 1 kilogram sample. This 1 kilogram sample is directly output from the Boyd Crusher to the LM2 pulverizer to pulverize. The rest of the material reports to coarse reject.*
4. *Additional reject splits are retained for future metallurgical work and for duplicate coarse reject analysis. Roughly 1 in 25 of the duplicate coarse reject (DR) samples are prepared and assayed as a check on the pulp preparation process. As well, 1 in 25 coarse reject samples are also screen analyzed to confirm the comminution size of 95% passing 4mm is achieved.*
5. *Approximately 1000g of the primary split is pulverized to produce a 95% passing 200 mesh pulp. One out of every 25 pulps is wet screened to monitor the comminution of -200 mesh pulp size. This sample also forms the Duplicate Assay sample (DA) which is separately assayed for QC purposes.*
6. *After pulverizing, the 1000g sample is mat rolled then split into 4 components using a spoon. The entire pulped sample is divided and placed into 4 kraft paper pulp bags.*

One of the pulp bags is sent out for analysis to Geoassay right away. The remaining three pulp bag are individually sealed then placed into zip lock plastic bags and submitted to ENJ. This will be used for assay check programs with the frequency 1 in 20. Check assay pulps are sent out for analysis to Intertek and Sucofinfo.

Following the Standard Operating Procedures (SOP) document, CRM Standards are inserted on a 1 in 20 basis and one blank is inserted per batch.

Assay instructions are supplied to GeoAssay electronically by PT ENJ personnel. GeoAssay labs use Inductively Coupled Plasma (ICP) Optical Emission Spectrometer (OES) methodology for determining the base metal content. Assay requests are complete ICP-OES packages (36 elements) with three acid digest from a 0.5g pulp sample (aliquot). If the result of that method reports greater than 10.0% copper, the assay is rerun as an "ore grade" sample where a 1.0g sample is digested with three acids and followed with flame AA."

The core handling, sampling and sample reduction protocols as described appear suitable for preparation of BKM material for assay. The procedure where ENJ mat-rolls and divides the pulverized material to generate four pulps from each sample by spooning into paper bags would appear to be poor practice, however the acceptable duplicate pulp and inter-laboratory check copper assay indicates that this practice has not adversely affected the sample reliability.

Evaluation and interpretation of the sampling suitability and assay reliability (quality control evaluation, assessment of copper grade against sampling interval lengths and recovery etc.) is included along with the assessment of all datasets utilized in the resource estimate in Section 0.

11.2 2015

KSK undertook the 2015 drilling following protocols set out in the following SOPs and the June 27, 2015 *revision:*

- KSK_SOP_002_2015.01.14_Chain_of_Custody_Doccumentation_FINAL.doc
- KSK_SOP_004_2015.01.14_Core Pickup, Handling and Processing - FINAL.doc
- SOP Revision By Duncan Hackman_20150627.docx

Chain of custody documentation is available for all holes drilled in 2015. They revolve around establishing responsible persons for sections of work and signing-off on the completion and verification of this work. They also record the transferal of the responsibility for core, samples and data through the employ of hand-over signatures. Chain of custody documentation is available for the following activities undertaken during the 2015 drilling program:

- Drill surveys
- Core pick-up at rig
- Core received at camp
- Core photos
- Core logging
- Core geotech-logging
- Core data collection
- Core sampling
- Core sample transport record
- Data entry checklist
- Core summary log
- Core processing finalization checklist

The standard operating procedures document is presented as a flow chart centered on photographs depicting the activities employed to process core and samples (refer to Appendix 12 of the resource report (Hackman, 2015)). Although not detailed in their description the procedures (including the H&A changes) are considered appropriate for the processing of the BKM core and samples. H&A observed that the protocols being followed at site were in line with those in the document "Kalimantan Surya Kencana – Core Handling Procedure: Drill Site – Core Shed - Processing". H&A implemented minor changes to protocols in undertaking the following activities (refer details in document "*SOP REVISION BY DUNCAN HACKMAN, 27 June 2015. KSK, BERUANG KANAN PROJECT*"):

- Specific Gravity (six points)
- Down-hole survey intervals (two points)

- QA/QC for down-hole survey camera tool (four points)
- Drilling and core logging of veining (three points)
- Sampling (three points)
- RQD (two points)
- Core Photo (two points)

These changes have added further quality assurance and quality control features to the protocols.

Then onsite processing workflow is as follows:

- Core is packed and carried by hand from drill sites to the core processing facility at camp (located to the east of the BKM mineralization).
- Core blocks and tray details are checked and hole depth details recorded on core.
- Core trays are weighed and photographed wet.
- Geotechnical and geological logging undertaken.
- Geologist selects segments of core for SG determination, which is undertaken by core yard technicians.
- Sample intervals are determined by geologists and core is split longitudinally by core saw. Clayey and incompetent core is wrapped in glad-wrap and packing tape prior to cutting.
- CRM Standards, coarse blanks (granite), pulp blanks and coarse crush duplicates are inserted into sample sequence (coarse crush duplicates are generated at ITS during sample preparation, empty, numbered bags are included within the sampling sequence in preparation for their creation).
- Core and QC samples are bagged and tagged for transport to ITS Jakarta.
- Dispatch paperwork is prepared for ITS which includes the list of coarse crush duplicates to be prepared and samples where SG segments require drying separately and recombined with the remaining material before crushing).
- Half core in trays is photographed both wet and dry.
- Core block details inscribed onto aluminum tags which are then attached back onto core blocks. Tray details are engraved onto trays before being packed and transported by light vehicle to the Tengkiling core shed for rack storing under cover.

KSK employs the use of numbered, tamper-proof zip ties to seal sample bags being transported off-site.

Details of sample preparation and analysis and QC insertion rates are included in Section 12.2.3.b.

12 DATA VERIFICATION

Table 12-1 lists the files and data supplied to H&A that underpins the BKM resource estimate. Table 12-2 lists the files generated by H&A in undertaking the resource estimate. Table 12-11 and Table 12-12 list intervals where the laboratory assay files were not supplied to H&A and Table 12-3 lists the photos not supplied. In addition there is no assay quality control data or reports for the early KSK drilling (pre 2002), no protocol documentation for the early KSK and OX-KSK drilling programs and no core photos for the early KSK drilling.

H&A is satisfied that the files/data and information supplied by KSK is sufficient and suitable for producing a resource estimate on the mineralization at BKM and for evaluating the risk inherent in the estimate and reporting findings following the guidelines set out under the Canadian NI 43-101. KSK has provided written assurance that the data supplied is current, complete, accurate and true and that they have disclosed all data and information material for the assessment of the resources at BKM (refer to Appendix 11 of the resource report (Hackman, 2015)).

Table 12-1 Files supplied to H&A by KSK and utilized in undertaking the 2015 BKM Resource Estimate and NI 43-101 report

File	Description	Use
Ass.csv; Coll.csv; Surv.csv	2014 resource data (74 Drillhole collar, survey and assay data. DHs within BK)	historic hole assay data for estimate (2015 hole data appended to these files)
Coll2015_final_check_20151013.xlsm Surveys_20151005.xlsm QC_Analysis_V2-11.xlsm	2015 Drillhole collar, survey and assay data. 71 holes. Compiled and cross-checked from individual excel files and sheets transferred from site	drill hole display, update 2014 domaining and grade interpolation. 2015 assay QC evaluation.
2015.10.13_KSK_DB.accdb	KSK compiled dataset	cross-check 2015 dataset against H&A compiled dataset
52 files: [*Creation Date*]_List Sample Dispatch_BKM00[03-12, 15-26, 28-57].xls	KSK sampling sheets for 71 holes drilled in 2015	generating resource dataset and QC evaluation data
526 files: [*Hole_Name*]_Data Entry_[*Geo Logging Sheets_New/MagSus/Photography and Weight_NEW/RQD/Sample Number and Recovery/SG Sample*].xls	KSK 2015 drillholes - Lithology, Alteration, Structure, vein and non-vein mineralization, SG, Core Diameter	drill hole display, domaining and mineralization setting interpretation, tonnage factor determination, investigation into fundamental sampling error effect
DTM-BK-Lidar_A.00t; DTM-BK-Lidar_B.00t; DTM-BK-Lidar_C.00t	Vulcan 3D TIN Topo Surface	drill hole collar location validation, resource domaining
119 x [*ITS Lab Job No*].xls	Intertek Utama Services Laboratory Assay Report files (assays, drying weights and Lab QC results (including ".01" and "D" reissue files))	construct 2015 Assay and QC dataset
072[581, 781, 838, 936].csv 073[520, 741].csv	DHs KBK-00[19-22, 24] Laboratory Assay Report files	Validation of Cu assay data - Lab Duplicate and Repeat Assay analysis
150269.xls 150424.xls CIK.MIN.06503.xls	ITS and GeoAssay Laboratory Assay QC Report files	Determination of Coarse Blank Assay Values and umpire lab assay results
20140604_bk_geo_interp_with_silica ledges_no_ddh.png	interpretation map	domaining
20140827_bk_rock_geo_alt.csv	Rock Chip Sample data	domaining
20140827_bk_so_ctr.DXF	Cu soils contours	domaining
KSK Sample Prep and Assay QC Final Report Jan2014.docx	ENJ-KSK drill sample assay QC report	Assay QC analysis
oxiana qaqc 2007.pdf	OX-KSK drill sample assay QC report	Assay QC analysis
Beruang LeachReportRecommendations.doc; All Beruang Summary Drill Holes.doc; Beruang Report - Dave Howard.pdf; Report_Central BK 17July-3Aug_2012_TP96.pdf; P_Pollard_KSK copper review.pdf; Bob_BurkeKGC REPORT.pdf; MG_Report_43-101_Appraisal_2004.docx; 43-101 Appraisal Report.pdf; Kalimantan Gold_Technical Report_150606.pdf	Reports on BK by Historic Authors	Project Familiarization and NI43-101 Report Compilation
KSK_CoW[1].pdf	KSK CoW agreement between KSK and Gov. RI	Reference - Terms and Conditions
1453 x core photo files (*.jpg)	2015 KSK drilling core photo files	mineralization setting investigation, resource domaining - assay and DBD validation and verification

Table 12-2 Files generated by H&A and used in undertaking the BKM2015 Resource Estimate

File	Description	Use
29_060_Solid_min_6.00t; 29_060_Solid_min_7.00t; 29_060_Solid_min_22.00t; 29_060_Solid_min_23.00t; 29_060_Solid_min_24.00t 36_095_Solid_min_8.00t; 36_095_Solid_min_9.00t; 36_095_Solid_min_10.00t; 40_025_Solid_min_16.00t; 40_025_Solid_min_17.00t; 40_025_Solid_min_18.00t; 22_017_Solid_min_11.00t; 22_017_Solid_min_12.00t; 22_017_Solid_min_13.00t; 22_017_Solid_min_14.00t; 22_017_Solid_min_15.00t; 22_030_Solid_min_25.00t; 22_030_Solid_min_19.00t; 22_030_Solid_min_20.00t; 22_030_Solid_min_21.00t; 29_060_Solid_min_1.00t; 29_060_Solid_min_2.00t; 29_060_Solid_min_3.00t; 29_060_Solid_min_4.00t; 29_060_Solid_min_5.00t	Grade interpolation domains (nominally >2000ppm Cu)	Cu resource estimate grade interpolation
base_2-70_DBD_2015.00t	topo surface transposed -17m	to assign Tonnage Factors
Indicated_combined.00t	increased drill density and mineralization thickness	to assign Indicated Classification
71 x [*hole name*].xlsm	core photo compilation files	verification of assay geology and DBD data
QC_Analysis_V2-11.xlsm	Compiled Assay and QC data	Resource and QC datasets
bkcu_3m.map	3m composited Cu assay data	Cu resource estimate grade interpolation
BM_Create.bdf; bkcuID2OK.bef	Vulcan Definition Files	Generate Cu resource estimate
BK_postestimte30000_Oct2015.bmf	BK 2015 RE block model	Cu resource estimate

Table 12-3 List of core photos missing from those supplied to H&A from ENJ-KSK JV drilling

Hole	No. Photos	From	To	Interval
KBK0019	1 photo	61.2	68.62	7.42
KBK0020	1 photo	190.9	198.25	7.35
KBK0021	5 photos	365.6	405.05	39.45
KBK0025	58 photos	76.05	393.2	317.15
KBK0026	1 photo	6.9	13.2	6.3
KBK0026	9 photos	55.65	103.75	48.1

12.1 VERIFICATION AND VALIDATION

There is missing metadata and QA/QC information for the pre 2012 drilling for assay, density and geological inputs used in the resource estimate. H&A has endeavored to verify and validate historical data firstly by assessing the validity and suitability of the ENJ-KSK and 2015 KSK data and then comparing this data with the earlier data.

The following sub-sections outline the verification and validation approach for data used in generating the 2015 resource estimate and highlights issues uncovered and considers the risks to the estimate associated with the issues.

12.1.1 Grid Reference and Sample Location

All work is undertaken and recorded in WGS84, UTM Zone 49S. The site visit confirmed that work was undertaken in this reference as drill hole collar markers are marked with coordinates matching those in the resource dataset and coordinates for four pre-2015 holes checked by GPS agree within acceptable limits to records in the KSK dataset. H&A has been informed that there are no translation issues to investigate and that the grid references for drill holes and the LIDAR topographic surface are congruent.

KSK employed geoindo Survey Services (geoindo) to locate the holes drilled in 2015 and to locate and survey historic holes. For the body of the geoindo report, including survey pickups, refer to Appendix 13 of the resource report (Hackman, 2015). H&A cross-checked the original geoindo pickups against the KSK collar file to verify collar locations for geoindo surveyed holes.

The collar locations for the thirty pre-2015 holes not located and surveyed by geoindo were verified by:

- Cross checking the supplied dataset with an historic listing of holes supplied as text files.
  Only coordinates for hole BK052 differ by more than 5m between the two sources. This hole is located 200m south of the BMK resource.
- Cross checking holes against the LIDAR topographic surface.
  All drill hole collar elevations concurred within acceptable accuracy to the topographic surface.

H&A is of the opinion that drill collars are known within sufficient accuracy for the BKM 2015 Resource Estimate to be considered for all Classifications under the NI 43-101.

There is no downhole survey data for the early KSK drill holes BK-[01-16,17A,17B,18] and later holes BK0[30-32,55-57], BKD02-01 and KBK-00[19,28]. Drill traces for these holes are defined by collar surveys. Of the remaining holes, traces are defined by:

- KBK-00[20-27]: 30m to 50m spaced Eastman Camera Surveys
- BK0[53,54,58]: 3m spaced Gyro Surveys

- BK0[29,33-44,44-02,45-52]: 3m spaced Maxibore Surveys

- 2015 holes (BKM series): 20m spaced single shot digital camera surveys

Surveyed holes show absolute azimuth deviations range between 3 and 15 degrees and absolute dip deviation ranges between 2 and 6 degrees for all survey measurement types. Holes in general deviate clockwise, with the majority steepening with depth. Final drill hole survey directions are comparable with collar hole survey directions, with the majority of holes showing less than 5 degrees difference in both azimuth and dip.

In general holes show minimal deviation and, when considered in relationship to the geometries of the interpreted mineralized domains, drill hole traces are considered to be well defined. The consistent minor deviations observed in holes where survey control is well established lends confidence to the trace path and therefor sample locations in those 30 holes whose trace is defined by a single collar survey.

H&A considers that sample locations known to an acceptable level of accuracy for the BKM 2015 Resource Estimate to be considered for all Classifications under the NI 43-101.

12.1.2 Topographic Surface

The drill collars and surface mapping (including contours) overlay with good correlation on the LIDAR topographic surface. The consistency between the datasets assures H&A that the BKM 2015 Resource Estimate has internal integrity.

12.1.3 Geology, Mineralization, Alteration, Structural and Oxidation Logging

Simplified coding of logged intervals in the digital dataset describes the geology, mineralization and alteration at BKM. There is no logging of the oxidation state in the resource dataset. H&A and Mr. Stephen Hughes of KSK reviewed core photos of the mineralized intervals at BKM to verify the logging, to generate an oxidation log and to identify and classify settings for the BKM mineralization. The grouping or classification of the mineralization settings was undertaken with the specific purpose of guiding resource domaining and grade interpolation as the KSK supplied logging is not suited or readily formatted for this purpose.

Mineralized intervals for 26 historic holes, BK0[29-31,33-36,38,44,44-02,45-52,54-5] and KBK00[21,23,24] were re-logged from the core photographs. For details of the 299 observations (287 mineralized intervals) refer to Appendix 5 of the resource report (Hackman, 2015). The key findings and styles of mineralization identified from this work and utilized in the resource estimate are:

- 12 styles of mineralization were identified (refer to Appendix 6 of the resource report (Hackman, 2015)). These can be further grouped into three main classifications; Sheared Veins, Veins in Shears and Veins in Breccia. The Veins in Shears type is the most dominant style, whereas the Cu grade is highest in Sheared Veins (Table 12-4).

Table 12-4 Styles of mineralization identified from logging of core photos

Photo Logging Classification					Summary		
Mineralization Setting	**Number**	**Meters**	**Proportion of Mineralization**	**Average Cu (%)**	**Setting**	**Proportion**	**Average Cu (%)**
Brecciated and oxidized	3	23	1%	0.00	**Other**	**2%**	**0.28**
Cc	2	11	1%	0.91			
Sheared Si-Py-Cp veins	2	11	1%	0.73	**Sheared Veins**	**6%**	**0.69**
Sheared Si-Py-Cv veins	6	27	2%	0.92			
Sheared Si-Py-Cv/Cc veins	11	63	4%	0.59			
Cp veins in shear	22	182	11%	0.51	**Veins in Shear**	**91%**	**0.62**
Si-Py-Cp veins in shear	57	385	23%	0.53			
Si-Py-Cc veins in shear	5	21	1%	0.35			
Si-Py-Cp-Cv veins in shear	10	60	4%	0.72			
Si-Py-Cv veins in shear	118	609	37%	0.63			
Si-Py-Cv/Cc veins in shear	47	241	15%	0.81			
Si-Py-Cv veins in breccia	4	20	1%	0.49	**Veins in Breccia**	**1%**	**0.49**
Total	**287**	**1654**		**0.63**			

- The digital structural logging shows that the ENJ-KSK recognized the structure within the mineralized intercepts (Table 12-5) however the significant observation made in the core photo logging is that the structural deformation is intense and shows a distinct shear fabric.

Table 12-5 Structural logging associated with mineralization styles identified from logging of core photos

Mineralization Setting (from Photo Logging)	**Structure Logging in RE Dataset (from KSK)**		
Veins in Shear	blocky	crushed-fractured	fractured-brecciated
	blocky-fractured	crushed-gouged	fractured-gouged
	blocky-veined	crushed-veined	fractured-veined
	crushed	fractured	gouged
	crushed-blocky	fractured-banded	gouged-veined
	crushed-brecciated	fractured-blocky	unconsolidated-veined
Sheared Veins	crushed-blocky	fractured-gouged	
	crushed-fractured	fractured-veined	
	fractured	gouged	
	fractured-banded	gouged-blocky	
Veins in Breccia	crushed-fractured		
	fractured		

- Copper mineralization is associated with veining (various Cu mineral species are hosted in veins and fractures, mostly with, but can be without silica and pyrite). A direct relationship between vein intensity/thickness (%veining) and copper grade was noted in the core photos. This relationship also becomes apparent when copper grade is assessed against the logged percentage of veining in the sample interval (Table 12-6).
- There is no apparent association between copper grade and total percent logged pyrite veins (Table 12-6). There is however an association between copper grade and intensity/frequency of pyrite veining and to a lesser extent with quartz-sulphide veining (Table 12-7). The core shows that copper minerals are located in thin crack-seal veinlets and not in a pervasive replacement (chalcocite disease) form. These observations support that copper mineralization is of hypogene origin, quite likely a subsequent and distinct event to the pyrite alteration.
- The ENJ-KSK Main Structure logging within the mineralized domains shows that geologists have recognized that veining (42% of intervals) and faulting (brecciation 4%, gouge 15%) are key features of the mineralized intervals. The recognition that 30% of the mineralized intervals are fractured is of significance; however it is not discernible from the logging if this fracturing is important with respect to mineralization and related to faulting or is insignificant and related to late stage shattering/jointing. The core photo logging has clearly identified that mineralized intervals contain structure-related faulting and shearing which appears to be important with respect to mineralization (enhancing ground preparation, forming fluid conduits and reworking and upgrading mineralization through reactivation events).

Table 12-6 ENJ-KSK Vein%, Pyrite% and Main Structure logging (within mineralized domains)

Logged Vein % in RE Domains			Logged Pyrite % in RE Domains			Logged Main Struct in RE Domains		
Percent (%)	Proportion of Mineralization	Av. of Cu%	Logged Py Perc (%)	Proportion of Mineralization	Av. of Cu%	Logged Structure	Proportion of Mineralization	Av. of Cu%
0	2%	0.5	0-2	3%	0.5	Blocky	9%	0.9
0-2	50%	0.6	2-4	37%	0.7	Brecciated	4%	0.7
2-4	14%	0.7	4-6	23%	0.6	Fractured	30%	0.9
4-6	24%	0.5	6-8	17%	0.5	Gouge	15%	0.7
6-8	3%	1.1	8-10	5%	1.5	Vein	42%	0.6
8-10	3%	1.3	>10	15%	0.8			
>10	4%	2.3						

The photos for the 71 KSK 2015 holes were logged for veins/mineralization styles in conjunction with the assay data. Table 12-7 and Table 12-8 present the findings from 1292 observations and show that:

- Significant copper grades (>0.5%) are associated with low vein frequencies.

- It appears that higher copper grades are more common with low frequencies of pyrite veins than they are with low frequencies of quartz veins

- 85% of the observations have ≤ 3 quartz veins and ≤ 5 pyrite veins per meter confirming observations that very little veining can host significant copper mineralization.

Table 12-7 Average copper grades for logged mineralized intervals, split by veining type

Copper Grades (%) split by veining frequency - Qtz (+/- sulphides) and Pyrite (+/- copper sulphides)																	
Pyrite vn freq (+/- Cu sulphides) - vns/m	Quartz veining frequency (+/- sulphides) - vns/m																
	0	1	2	3	4	5	6	7	8	10	12	15	20	25	30	80	all
0	0.6	0.9	0.5	0.5	0.4	1.2	0.3	0.4									0.7
1	0.2	0.3	0.3	0.4	0.4	0.6	0.4			1.0							0.3
2	0.4	0.3	0.5	0.6	0.5	0.7			0.0	0.6							0.4
3	0.6	0.4	0.7	0.5	0.4	0.6			1.6	4.5		0.7					0.6
4	0.4	0.5	0.8	0.5		0.7		0.0	0.8								0.6
5	0.7	1.0	1.2	0.7	0.9	0.9		0.6	0.7	1.5	1.3	0.9	2.6	0.7			1.0
6	0.8	0.7	0.8			1.3						1.0	5.0				1.1
7	1.2	1.2	1.2	0.9	0.6	0.5						1.1					1.1
8	1.2	1.2	1.3	1.1		3.1			2.0			0.5					1.3
9		1.2	0.9														1.1
10	2.6	2.0	2.2	1.6	1.2	1.7				0.2		1.5			2.1		2.0
12		1.2		1.5													1.4
15	3.7	1.8	4.4	2.5		2.1				1.2		2.7					2.5
18						5.3											5.3
20				4.6		3.2			7.3	2.0							3.9
25				2.6					1.4								1.8
30						0.2											0.2
35						2.3											2.3
40																12.5	12.5
all	**0.7**	**0.6**	**0.8**	**0.8**	**0.5**	**1.2**	**0.4**	**0.3**	**1.7**	**1.5**	**1.3**	**1.2**	**3.8**	**0.7**	**2.1**	**12.5**	**0.8**

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Table 12-8 Number of logged mineralized intervals, split by veining type

Number of Logged Intervals split by veining frequency - Qtz (+/- sulphides) and Pyrite (+/- copper sulphides)																	
Pyrite vn freq (+/- Cu sulphides) - vns/m	Quartz veining frequency (+/- sulphides) - vns/m																
	0	1	2	3	4	5	6	7	8	10	12	15	20	25	30	80	all
0	84	33	32	6	4	4	1	1									165
1	20	79	34	33	7	7	1			1							182
2	59	49	53	36	4	10			1	1							213
3	57	73	64	25	2	13			4	1		1					240
4	8	18	10	5		4		1	2								48
5	51	73	51	25	2	13		1	2	7	2	2	1	1			231
6	5	2	7			1						1	1				17
7	10	13	7	4	2	1						2					39
8	18	7	18	8		1			2			1					55
9		2	1														3
10	20	8	8	10	2	12				1		2			1		64
12		1		2													3
15	1	1	1	8		4				1		1					17
18						1											1
20				2		4			1	1							8
25				1					2								3
30						1											1
35						1											1
40																1	1
all	333	359	286	165	23	77	2	3	14	13	2	10	2	1	1	1	1292

An oxidation log was produced from both historic and 2015 drilling. Table 12-9 shows that the base of complete oxidation is encountered at shallow depths at BKM. H&A has not modeled the base of oxidation for the 2015 Resource Estimate as it is considered that the identification and reporting of this material is not of material interest at this stage of the project.

Table 12-9 Depth to base of complete oxidation, split by mineralized domain classification

Domain	Number of Intercepts	Average Depth to BOCO
Unmineralized	63	10.1
Mineralised	11	8.1
All	74	9.8

12.1.4 Specific Gravity

KSK collected 1807 bulk density and dry bulk density measurements from core during the 2015 drill program utilizing the Archimedes principle for determining volume and drying permeable samples at the ITS laboratory. Quality assurance procedures included confirming scale stability over time (weighing a standard steel bar) and ensuring the water depth for immersed weight measurements were at a constant level before each SG batch was processed. H&A improved the robustness of the workstation setup during the June 2015 site visit to ensure stability over time. Quality control data from weighing the steel standard confirms that the scale measurement was constant throughout the program and records show that water levels were checked and stable as intended.

KSK dispatched 130 competent pieces of core to ITS between May 30, and June 20, 2015 for check DBD measurements of competent/non-permeable core which confirmed that the scales were calibrated correctly (with respect to ITS scales) and that the BD measurements were being correctly undertaken at site (with the avg. %MPD being 0% and the avg. %|MPD| being 0.4% for pairs from both SG determinations). These samples contained and estimated moisture content of between 0 and 2% (average 0.8%). To further confirm that competent/non-permeable core could be processed for DBD at site, KSK undertook an oven drying test on 5 pieces of core and found that between 3g and 8g of moisture was driven off in the first hour of drying (1% relative wt) with only a further 0.5g to 1.5g being removed with continued drying.

With the significant amount and robust dataset created by KSK in 2015, H&A has eliminated the riskier 330 SG measurements taken by ENJ-KSK from the evaluation dataset. The key considerations in deciding to remove this data are:

- ENJ-KSK employed a process that produced a bulk density measurement, not a dry bulk density measurement as required for the resource estimate. There was no drying of any samples.
- Tonnage factors for milled shear zones containing silica fragments and crushed (clayey) matrix will be impacted by the inappropriate methodology employed. It is not easy to determine the clay content of the shears from core photos and flag SG readings from this material for consideration.
- The risk associated with this issue is not readily determinable as the extent of significant (clayey) shear zones is not easily determined from the current drill configuration/spacing at BKM.

The 1807 DBD measurement taken by KSK were validated and 54 records show spurious results (caused by missing data and/or data entry errors). These records were removed from the evaluation dataset, leaving 1753 records to determine tonnage factors to apply to the 2015 resource estimate.

12.1.5 Assays

12.1.5.a Pre-2015 Data

Handling and storage of the pre-2015 BKM data is poorly documented. To verify that the resource dataset has not been corrupted H&A rebuilt the dataset from source files. Not all laboratory report files were available for this process. Table 12-7 and Table 12-8 list the assay intervals still to be verified.

Key findings:

- 739 of the 1051 early KSK Cu assays (pre 2002, BK-01 to BK-18 series of holes) were cross checked with their laboratory report records and no issues were detected.
- 802 of the 1658 OX-KSK Cu assays from (KBK-0019 to KBK-0028 series of holes) were cross checked with their laboratory report records and no issues were detected.
- 3163 of the 4923 ENJ-KSK Cu assays (BK series of holes) match with their primary laboratory report records (GeoAssay Laboratory results).
- ENJ-KSK compliance to protocols regarding assay-result prioritization and the management of quality control data appears to be poorly observed by personnel as Intertek Services and Sucofindo umpire laboratory results have supplanted the primary GeoAssay results in the resource estimate dataset supplied to H&A (Table 12-12).
- H&A has not corrected the resource dataset as the entire dataset could not be corrected due to the missing or non-supplied assay results files.
- Mixed primary and umpire Cu results in the resource dataset will not affect the outcome or confidence in the BKM 2015 Resource Estimate as the comparison of data populations (Figure 12-4) and 'duplicate' results from both the source and umpire laboratories (Figure 12-5) show that the results can be interchanged with negligible local and global impact.

Table 12-10 List of intervals where Laboratory Report Files not supplied to H&A for assay data verification (KSK and OX-KSK drilling programs)

KSK Drilling				OX-KSK Drilling			
HOLEID	From	To	No. Assays	HOLEID	From	To	No. Assays
BK-11	3	132	44	KBK-0019	221	277	28
BK-12	1.1	107.6	26	KBK-0021	517.9	634.5	10
BK-13	18	220.65	68	KBK-0023	2	566	290
BK-14	3	240.55	68	KBK-0025	2	343	174
BK-16	11	257	42	KBK-0026	2	374	185
BK-17A	2	53	17	KBK-0027	2.4	188.9	94
BK-17B	2.65	277.2	34	KBK-0028	2.5	151.3	75
BK-18	215.8	267.8	13				
Total			**312**	**Total**			**856**

Table 12-11 List of intervals where Laboratory Report Files not supplied to H&A for assay data verification (ENJ-KSK drilling program)

<table>
<tr><th>Supplied Laboratory Report Files</th><th>Determination of assay source in ENJ-KSK Cu dataset</th><th>HOLEID</th><th>No. of Assays</th><th>From</th><th>To</th></tr>
<tr><td rowspan="10">No Lab Report files to H&A (20140820)</td><td rowspan="10">Source of RE dataset cannot be determined</td><td>BK037</td><td>55</td><td>5</td><td>181.2</td></tr>
<tr><td>BK038</td><td>50</td><td>18</td><td>183.3</td></tr>
<tr><td>BK040</td><td>54</td><td>2.9</td><td>187</td></tr>
<tr><td>BK041</td><td>34</td><td>188.4</td><td>300.1</td></tr>
<tr><td>BK042</td><td>54</td><td>6.4</td><td>181.9</td></tr>
<tr><td>BK043</td><td>54</td><td>10.5</td><td>189.6</td></tr>
<tr><td>BK044</td><td>54</td><td>3.3</td><td>180.1</td></tr>
<tr><td>BK045</td><td>54</td><td>1.4</td><td>171.3</td></tr>
<tr><td>BK046</td><td>54</td><td>6</td><td>181.5</td></tr>
<tr><td>BKD03-02</td><td>42</td><td>362</td><td>540.8</td></tr>
<tr><td>Have ITS assay results only</td><td>ITS assays do not match RE dataset - source of RE dataset not determinable</td><td>BKD03-02</td><td>12</td><td>391</td><td>537.8</td></tr>
<tr><td rowspan="10">Have SFK assay results only</td><td rowspan="10">Most likely SFK in RE dataset as these consistently match sequence in RE dataset</td><td>BK037</td><td>6</td><td>30.35</td><td>169.97</td></tr>
<tr><td>BK038</td><td>3</td><td>45.8</td><td>105.3</td></tr>
<tr><td>BK040</td><td>6</td><td>28.9</td><td>178</td></tr>
<tr><td>BK041</td><td>3</td><td>214.9</td><td>275.5</td></tr>
<tr><td>BK042</td><td>6</td><td>28.8</td><td>172.9</td></tr>
<tr><td>BK043</td><td>6</td><td>37.3</td><td>180.1</td></tr>
<tr><td>BK044</td><td>6</td><td>28.1</td><td>171</td></tr>
<tr><td>BK045</td><td>6</td><td>27.8</td><td>163</td></tr>
<tr><td>BK046</td><td>3</td><td>32.4</td><td>146.8</td></tr>
<tr><td>BKD03-02</td><td>5</td><td>415</td><td>531.8</td></tr>
<tr><td rowspan="2">Have GA results - however suspect other assays missing</td><td rowspan="2">DB records are not GA - most likely ITS as consistent with adjoining assays in RE dataset</td><td>BK029</td><td>54</td><td>135.5</td><td>300</td></tr>
<tr><td>BK030</td><td>44</td><td>169.1</td><td>301</td></tr>
</table>

Table 12-12 List of copper assay sources for pre 2015 data utilized in BKM 2015 Resource Estimate

<table>
<tr><th>Assay Results files</th><th>Comments regarding prioritization for inclusion in FPT RE dataset</th><th>Number of Samples</th></tr>
<tr><td>No Lab Report files to H&A (20140820)</td><td>Source of RE dataset cannot be determined</td><td>505</td></tr>
<tr><td>Have ITS assay results only</td><td>ITS assays do not match RE dataset - source of RE dataset not determinable</td><td>12</td></tr>
<tr><td rowspan="2">Have SFK assay results only</td><td>Most likely SFK in RE dataset as these consistently match sequence in RE dataset</td><td>50</td></tr>
<tr><td>SFK-GAM match RE dataset however have SFK OreGrade Assays not prioritized in RE dataset</td><td>5</td></tr>
<tr><td>GA results only</td><td>Most likely GA in RE dataset as these consistently match adjoining sequence</td><td>2982</td></tr>
<tr><td>Have GA results - however suspect other assays</td><td>DB records are not GA - most likely ITS as consistent with adjoining assays in RE dataset</td><td>98</td></tr>
<tr><td rowspan="2">Have GA and ITS assay results</td><td>GA prioritized in RE dataset</td><td>153</td></tr>
<tr><td>ITS prioritized in RE dataset</td><td>872</td></tr>
<tr><td>GA-ITS-SFK</td><td>SFK prioritized in RE dataset</td><td>14</td></tr>
<tr><td rowspan="3">GA-SFK</td><td>GA prioritized in RE dataset</td><td>28</td></tr>
<tr><td>SFK prioritized in RE dataset</td><td>197</td></tr>
<tr><td>SFK-GAM match RE dataset however have SFK OreGrade Assays not prioritized in RE dataset</td><td>7</td></tr>
<tr><td colspan="2">Total FPT drill assays in BK</td><td>4,923</td></tr>
</table>

12.1.5.b 2015 Data

H&A was engaged by KSK at the beginning of the 2015 drilling program to monitor copper assay quality assurance and quality control (QAQC). H&A reviewed and improved the KSK QAQC practices in two stages, the first being in early June and the second being during the late June site visit. The review included:

- Reviewing analytical method; resulting in increasing the elements reported by ITS,
- Reviewing standard type, grade ranges, insertion positions and rates; resulting in preferentially positioning coarse blanks and duplicates in mineralized intervals,
- Assessing sample dispatch sizes with respect to the standard inclusion rates and ITS laboratory batch/work flow sheet; resulting in an increase in batch sizes,
- Reviewing standards, duplicates and blanks performance for assays already received (batches BKM00[3-12, 15-24]; resulting in feedback to laboratory regarding copper assay drift and correction issues and the continuation of -2mm crush and split of primary sample to produce a ~1kg subsample for pulverizing,
- A visit to the ITS Jakarta laboratory to review sample preparation workstations and procedures; resulting in the following recommendations and requests:
    - To de-clutter the sample crushing and pulverizing area,
    - The Boyd Crusher to be used exclusively for reducing the samples to -2mm in size,
    - Barren wash to be processed between each sample processed through the crusher and pulverizer,

- Move the barren wash storage bins to more accessible places (with respect to workstations),
- Use a better shaped, square sided scoop for sampling of pulverized material,
- Use pulp package that is capable of holding >>250g (e.g. 500g) and ensure that the 250g pulp material is not tightly packed into this satchel (allowing analytical charge to be selected from any portion of in the satchel),
- Both the -2mm and -75micron comminution test results to be reported with assay results.

Section 12.2.3.b reports on the quality control assessment of samples included in the 2015 assay batches and on the findings from resubmitting selected samples to an umpire or check laboratory. In addition to reviewing the QAQC protocols and quality control assay data H&A constructed a parallel dataset from site DPO files and ITS Laboratory results files which was cross-checked with the KSK generated assay dataset before being used to generate the 2015 resource estimate. No issues were uncovered.

12.1.6 Core Diameter

The historic drill core diameter data was delivered to H&A with significant and numerous errors. It appeared that the dataset was corrupted at some point and H&A suspects that drag-drop or copy-down processes are responsible. H&A re-generated the drill core diameter data from logging files for some of the early KSK drill holes (pre 2002) and from core photographs for the OX-KSK and ENJ-KSK drilling. Core diameter logging for holes BK-[06-18], BK044 and BKD[01-01,02-01,02-02,03-01,03-02,04-01] could not be verified.

Complete and reliable data was delivered for the KSK 2015 drillholes.

The dataset used in evaluating the impact of the fundamental error on the BKM 2015 Resource Estimate is accurate, however is also incomplete as H&A does not have original logs and/or photos for mineralized sections for a number of holes. H&A is of the belief that the missing data will not impact on the outcomes of the evaluation but will impact on the confidence in assessing any risk to the BKM 2015 Resource Estimate related to drilling recovery issues.

12.1.7 Core Recovery

Core recovery data for holes BK[29-36,38,44-01,44-02,45-50,54,55,57,58], BKD03-[01,02] and the KSK 2015 drillholes was delivered to H&A and available to evaluate the association between recovery/loss and copper assays.

Verification of the logged core recovery data was undertaken by assessing the core photos for a number of intervals. H&A noted a small number of significant discrepancies between the logging and core photo observations however these are unlikely to impact on the evaluation or risk assessment to the BKM 2015 Resource Estimate. This is due to most core recoveries through mineralized zones being close to 100% (due to the presence of intense silica and pyrite alteration) leading to a dataset with low numbers of poor recovery samples (to assess any relationships) and the low impact of core recovery on confidence in the estimate.

12.2 ANALYSIS AND INVESTIGATION

The following analyses and investigations underpin the modeling, grade interpolation strategies and classification of the BKM 2015 Resource Estimate.

12.2.1 Geology, Mineralization and Structure

Historical workers' reports on the BKM geology have largely focused on aspects of the geology and mineralization for the targeting of world-class systems in the prospect area. Descriptions of the Beruang Kanan Main Zone style(s) of mineralization and settings are brief and mainly directed at how they relate to both porphyry and breccia-pipe systems. There are limited references to the geology, mineralization and structural setting of the copper mineralization that is the subject of the BKM 2015 Resource Estimate. Historical authors recognize the structural setting hosting the mineralization and the following references can be found in historic reports:

- (Geiger M, and Prasetyo D, 2004):
  - In the Main Zone, fourteen drill holes, to 280m measured depth, have intersected a north-northwest trending zone of intensely sheared and silicified, highly pyritic, zoned phyllic to advanced argillic altered wallrock. This zone was found to host copper grades of up to 167m @ 0.59% Cu.
  - The IP surveys clearly define the highly pyritic north northeast trending shear zone that hosts most of the copper mineralization in the Main Zone.
- (Munroe S, and Clayton M, 2006):
  - Drill core from Beruang Kanan which was observed during this review indicated a strongly altered (phyllic and advanced argillic) center which was strongly deformed in some areas. Quartz + pyrite + chalcopyrite in veins are associated with a strong cleavage in the altered rock, suggesting a strong structural control to the veins.
  - The drilling indicates a significant zone of narrow vein and disseminated mineralization which returns 0.7-0.9% Cu.
- (Johansen, G, 2007):
  - Alteration and copper mineralization are strongly structurally controlled. This is particularly obvious in the copper soil geochemistry data. Based on the drilling completed at Beruang Kanan by Oxiana Ltd (10 holes for 3,881.25m) alteration within or close to structures is dominated by sericite (phyllic) with peripheral alteration dominated by chlorite (propylitic). Copper mineralization is associated with zones of white, irregular, mesothermal quartz veins (1 to 5cm wide). The veining distribution is closely associated with faulting. The majority of the chalcocite, covellite, digenite and enargite at the main zone at Beruang Kanan are more likely to be supergene though there is still some evidence for the remnants of a high sulphidation system.
- (Pollard, P, 2006) reports of observations from drill holes BK[01,02,04,05,15]:

- Most of the drill core is composed of milled breccia with fragments commonly 1-4cm in size and a matrix component of 10-30 vol%. The fragments are commonly rounded and are composed of silica- and silica-sericite altered material ranging from dark black-brown (possibly chloritic), to pale yellow-brown, to dark and pale grey in color. These may reflect different original rock types and/or different alteration styles. The matrix probably consisted of rock flour material but is now completely altered to silica-sericite. Much of the core exhibits strong shearing (sic. Fig. 10) which appears to be post-mineral in timing, i.e. the alteration zone is sheared rather than being alteration of a shear zone.

KSK personnel (Mr. S. Hughes) and H&A verified the core logging and reviewed the geological setting of the BKM mineralization (Section 12.1.3, refer to Appendix 5 / 6 / 8 of the resource report (Hackman, 2015)) and their observations clearly support that the mineralization is vein related and the host rock is strongly sheared, milled and faulted. Key observations relevant to the modeling of the mineralization for the 2015 Resource Estimate are:

- There is strong indication that the advanced argillic alteration spans the deformation as the milled matrix is commonly silicified.
- Copper bearing cross-cutting veins and breccia veins-fragments were noted, suggesting that the mineralization veining event spans the deformation (though it could not be determined at what stage the covellite/chalcopyrite replacement of pyrite occurred).
- There is indication that the covellite/chalcopyrite replacement of pyrite is of hypogene origin and occurred during a single, or specific number of, event(s) as it occupies unique locations within veins (commonly along vein extremities) leaving untouched other apparently favorable pyrite rich bands.
- Later pyrite alteration overprints the covellite/chalcopyrite replacement. In hand specimen there is no evidence that this amorphous pyrite has been attacked by the copper bearing fluids.
- Copper grade tenor is loosely correlated to the veining intensity/thickness. Ground preparation (shearing, brecciation, silicification etc.) appears to be an important step in focusing and increasing veining.

H&A has adopted the shear/thrust related mineralization setting in modeling resource domains to guide copper grade interpolation at BKM. The majority of the mineralization at BKM is located along the main east-west trending ridge and spur defined loosely between 768400E and 769000E; 9932000N and 9932700N. A second, smaller center of mineralization is located along a lower spur to the south and defined loosely between 768700E and 769100E; 9931600N and 9931900N. The geomorphology over the mineralization is reflective of E-W striking north dipping thrust faulting.

Following detailed evaluation (Section 12.2.2) it can be interpreted that two thrust systems are present and strike at approximately 20 degrees to each other (Figure 12-2). It cannot be interpreted from the current data and information as to how the system may have manifested which could be by ramping, coupling or reactivation (following a change in the stress regime). Determining the nature of the structural model is not of major concern regarding risk to the resource modeling at this stage of the project, as favorable locations for hosting mineralization are coincident with each model and in-order with the location of mineralization at BKM. It will be however imperative to understand the trusting regime for designing further drilling of the mineralization as, although overall geometries of mineralization are similar for each model, the internal veining and alteration geometries/fabric will differ for each.

12.2.2 Resource Domaining

H&A undertook the activities in domaining the mineralization at BKM for resource estimation:

- Reviewed historic reports and drill core (at site and in supplied ENJ-KSK, OX-KSK and KSK 2015 photos with the assistance of KSK geologists)
- Generated a 3D working environment in Minesight™ presenting the drill hole copper assays, the drill hole structure, lithology and oxidation logging, surface soils and rockchip copper assays, the LIDAR topography (TIN surface) and the KSK interpreted surface geology mapping.
- Presented the LIDAR topography utilizing two Minesight™ routines that color/contour surfaces by associated features (in this case azimuth and dip).
- Identified and interpreted topographic surfaces (faces of the TIN) at 020 degree azimuth ranges between 040 and 180 degrees (i.e. 020-040, 040-060 etc., Figure 12-1) and interpreted key or main thrust surfaces
- Interpreting and visualizing the multi-element assay data (refer to Appendix 14 of the resource report (Hackman, 2015)) to identify volumes of favorable mineralization signatures.
- Generated surfaces projecting the interpreted major thrusts (from topographic surface) through the volume defined by the drilling assisted by features observed in the multi-element geochemistry volumes.
- Statistically reviewing the copper drill hole assay data to establish likely natural cutoff grades for modeling the mineralization
- Linking/domaining the >2000ppm Cu intercepts (Figure 12-3) utilizing the topographic interpreted thrusts and their projected depth surfaces as guides to identify related intercepts.

Visually and statistically validating the modelled domains to ensure that TIN surfaces snapped to drillhole traces and are consistent with adjoining modelled intervals (Table 12-13).

East-west cross-sections of the domains can be seen in Appendix 9 of the resource report (Hackman, 2015).

Figure 12-1 Geomorphology Interpretation and interpreted thrust-control on mineralization (idealized)

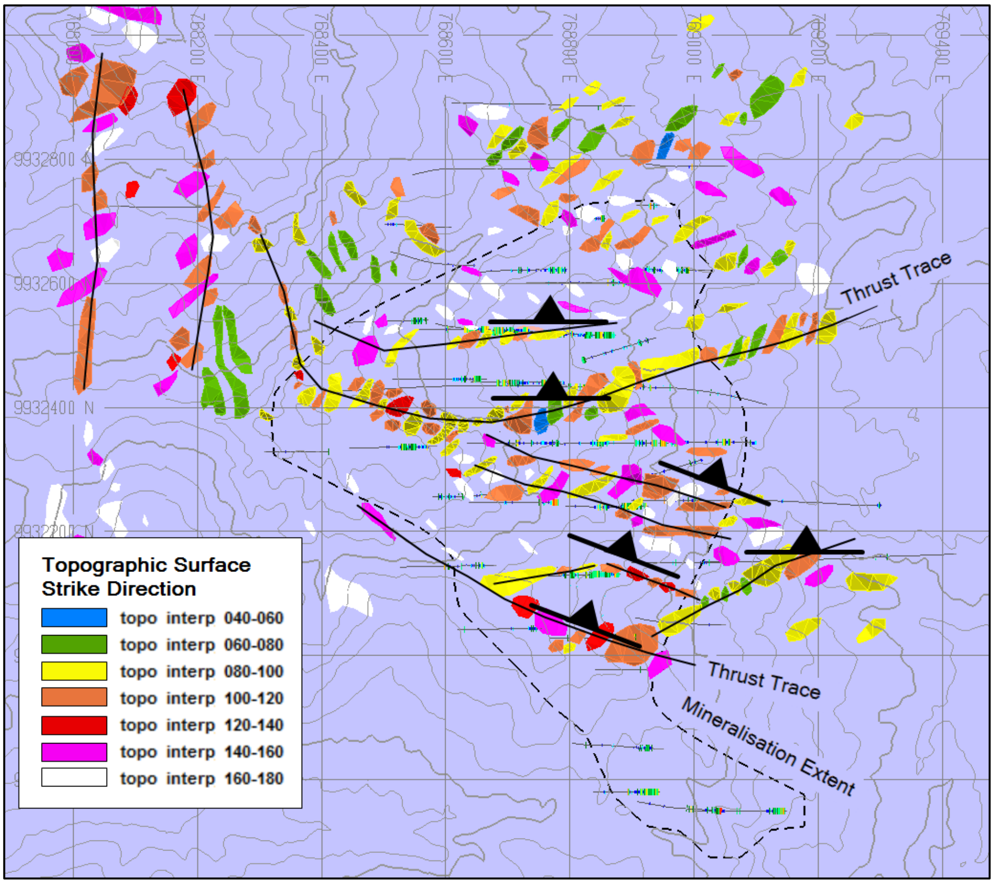


Figure 12-2 Mineralized domains (stacked total thickness) and thrust-control interpretation (idealized)

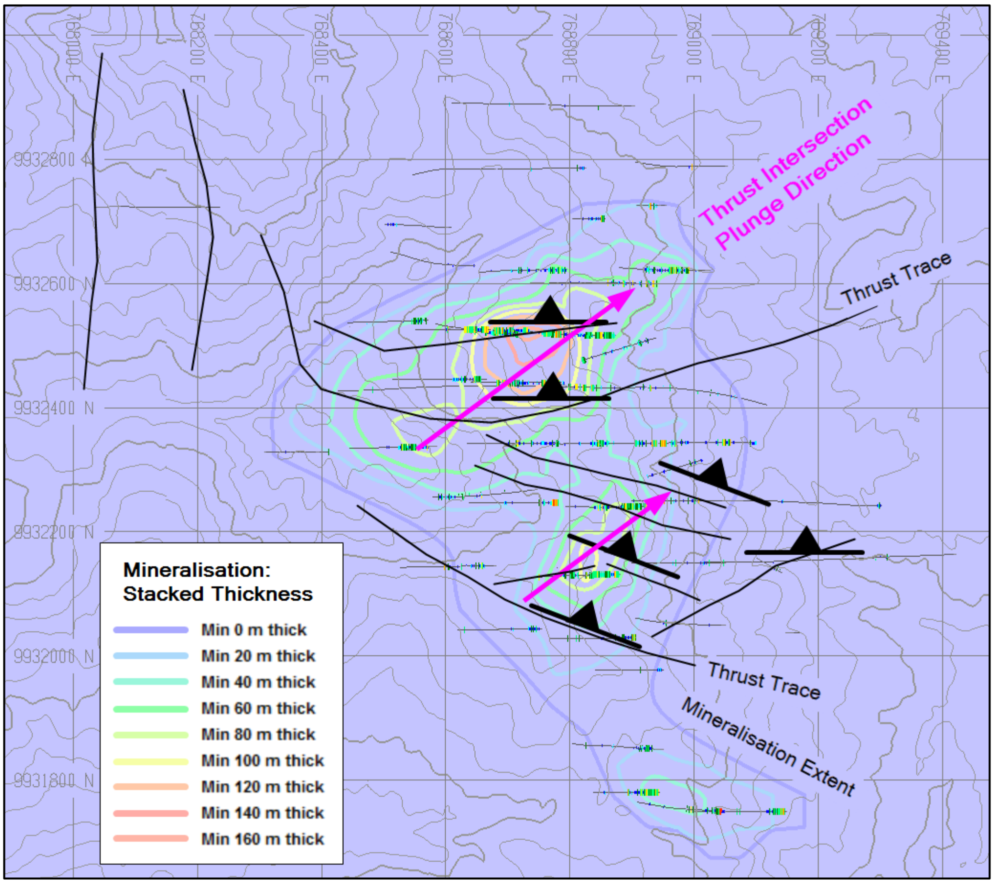
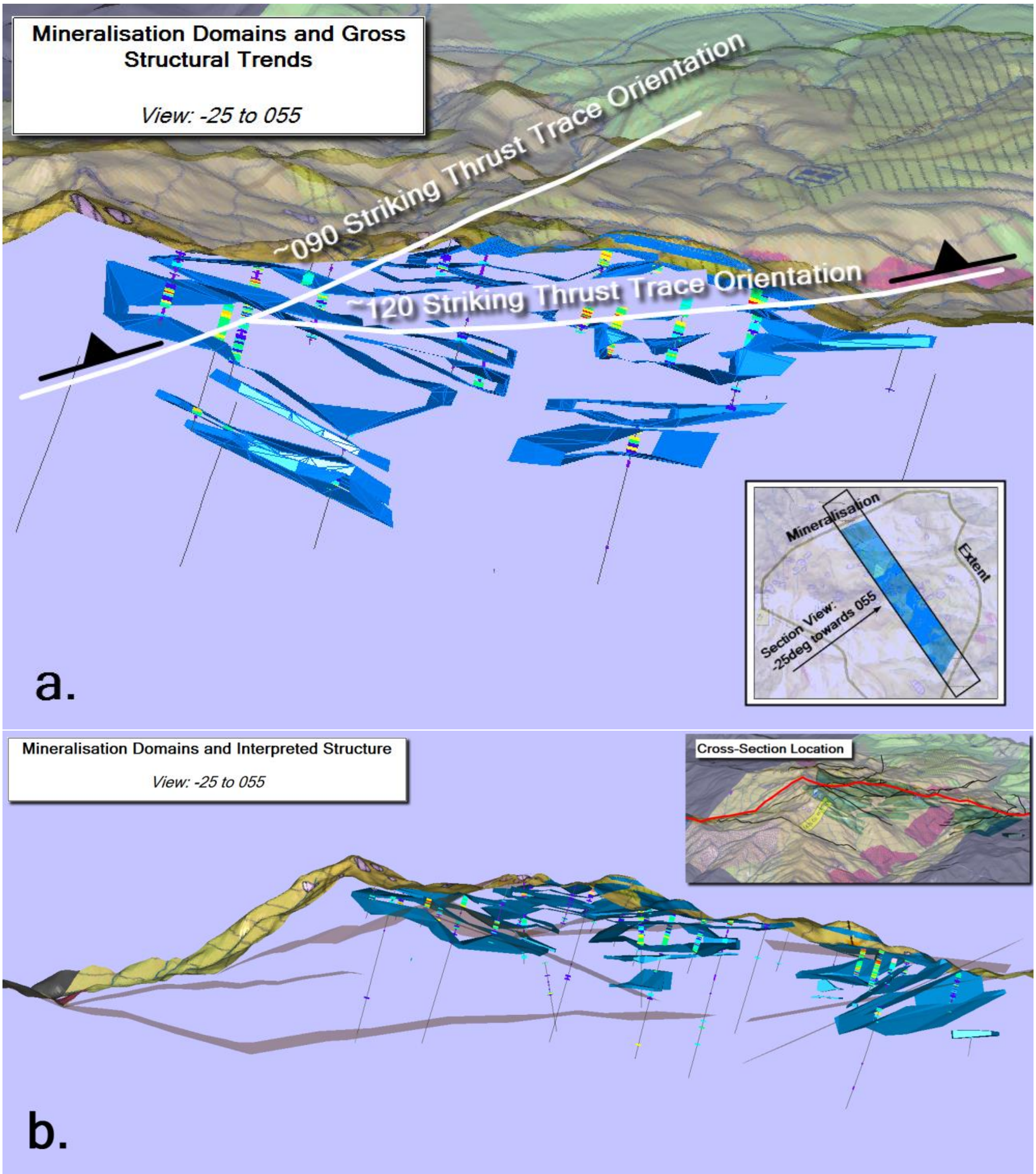


Figure 12-3 Mineralization Domains



Note: View down plunge extension of domains.

a. showing two main thrust directions

b. interpreted thrust and fault planes interpreted from geomorphology and multi-element domains associated with copper mineralization

Table 12-13 Average copper grade across domain contacts

Composites from Contact	Domained Mineralization					Contact	Not Mineralized				
	Av. Cu Grade (ppm)						Av. Cu Grade (ppm)				
	-5	-4	-3	-2	-1		1	2	3	4	5
Mineralization Upper Contact	6724	5912	7315	6609	6875		1126	943	882	652	693
Mineralization Lower Contact	8190	7112	7218	6926	5684		877	843	999	837	519

Note: Significant and sharp grade tenor change

12.2.3 Copper Assays – Quality Assessment

12.2.3.a Pre-2015 Copper Assays

Quality Control Assay samples were submitted with routine samples for the OX-KSK and ENJ-KSK drilling programs. There were no quality control samples inserted into the early KSK drill samples to check the reliability of copper results.

ENJ-KSK compiled a detailed assay quality control report (refer to Appendix 3 of the resource report (Hackman, 2015)). H&A has confirmed that the assay results for the QC samples are as reported from the laboratories and agrees with the ENJ-KSK findings, these being:

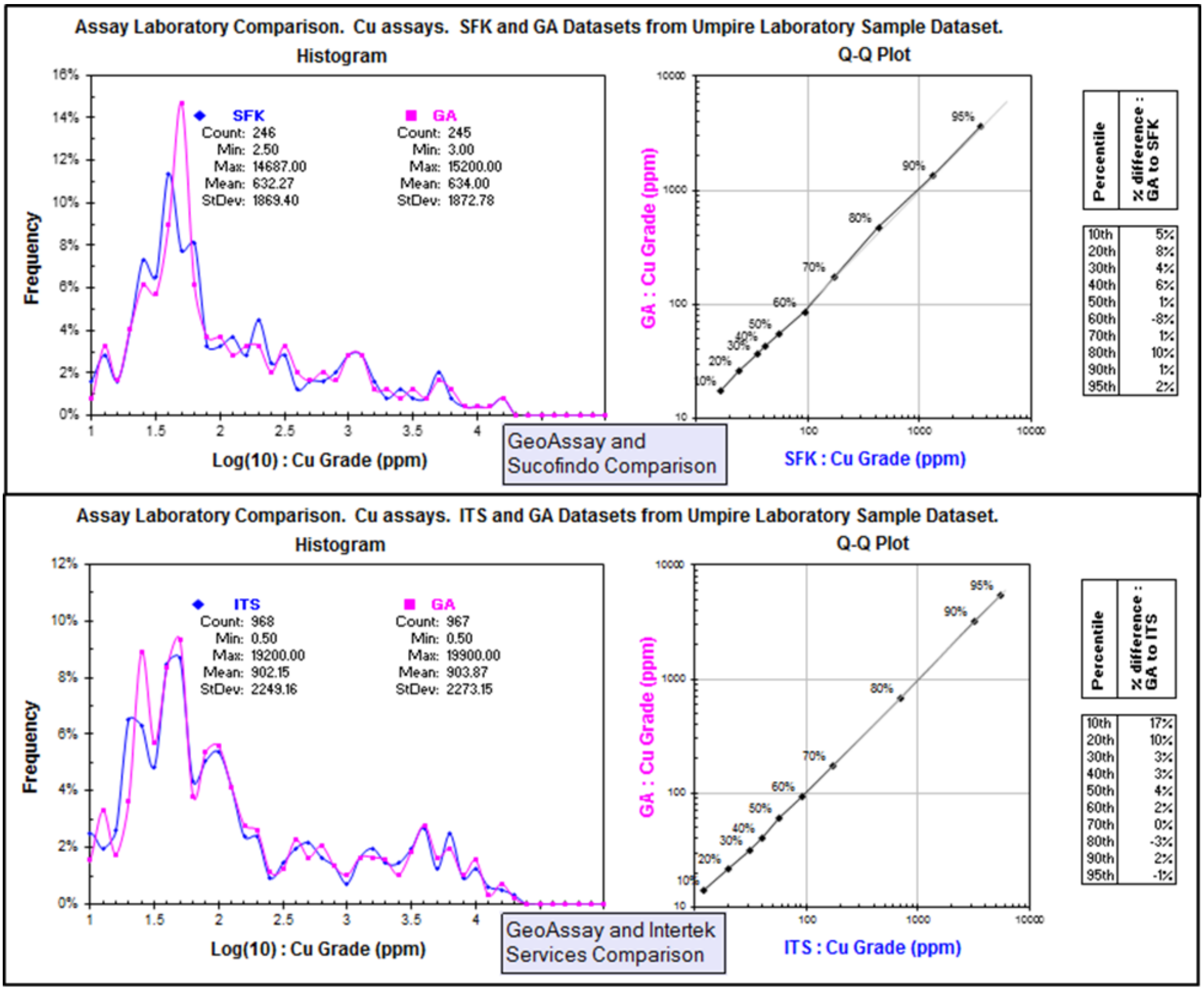
- There is no detectable cross contamination issues to be considered
- The CRM assays show that the laboratories (GeoAssay, Intertek Services and Sucofindo) return reliable copper assays for all batches
- Check assays to reference laboratories show good correlation with the primary laboratory copper assays.

H&A also notes that ENJ-KSK:

- Submitted both barren quartz and unconsolidated sand as their blank material at the rate of one per batch. The use of sand is not ideal as exposure to crusher contamination cannot be detected. The inclusion rate of blanks is low.
- Sourced four standards from those used by PT Freeport Indonesia and produced one matrix matched standard from the BKM prospect. Globally the matrix matched standard BKSH-01 performs poorly with respect to the other standards, H&A suspects that this is more likely due to features of the standard rather than issues with the laboratories and therefor has no reason to question the reliability of the routine assay at this stage of the project.
- Copper assays of the standards from ITS and SFK increase from ~+/-1% difference from their certified values pre May 2013 to +3-5% difference from these values post May 2013. The GA results are acceptable for all periods bar August 2012 where they are 4% greater than the certified values. ENJ-KSK offers no reason for the deviation in assay accuracy.
- The inter-laboratory check sample results analysis presented by ENJ-KSK show that assays generally differ by less than 4% (mean paired difference). The ENJ-KSK report does not show direct comparisons between the primary laboratory (GeoAssay) and the check laboratory sample results. H&A presents this comparison in Figure 12-4 and Figure 12-5. This analysis confirms the ENJ-KSK findings, being that the umpire laboratories' copper assays compare well with the primary laboratory assays.

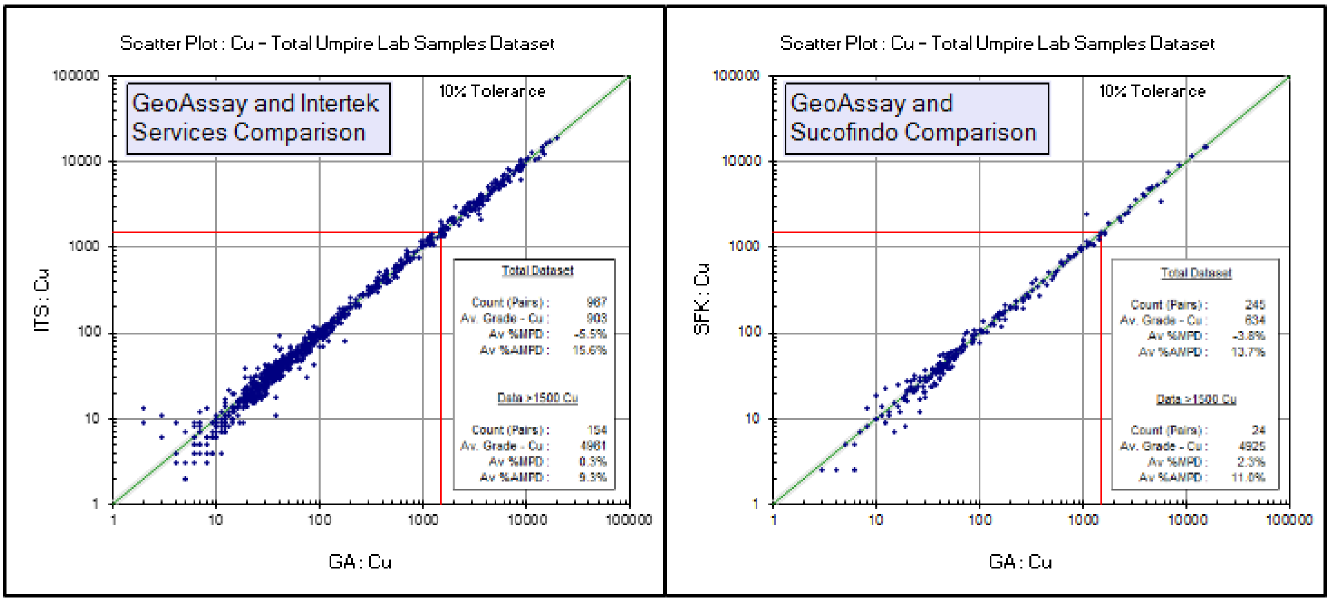
In addition, H&A reviewed all laboratory inserted standards, duplicate assays and repeat assays inserted by GeoAssay, Intertek Services and Sucofindo. No material issues were uncovered that would impact on assay confidence for generating and classifying the BKM 2015 Resource Estimate.

Figure 12-4 Umpire Laboratory copper assays (ITS and SFK) comparison



Note: Primary GeoAssay copper assays. Histogram and Q-Q Plot presentation.

Figure 12-5 Umpire Laboratory copper assays (ITS and SFK) comparison



Note: Primary GeoAssay copper assays. Scatter Plots and MPD presentation.

Blanks and standards were submitted into the routine sample stream for assaying by OX-KSK. There is no reference in the dataset supplied to H&A as to which assay results belong to the quality control samples, therefor H&A is not able to cross-check the graphs presented by OX-KSK on the assay quality control assay results (refer to Appendix 4 of the resource report (Hackman, 2015)).

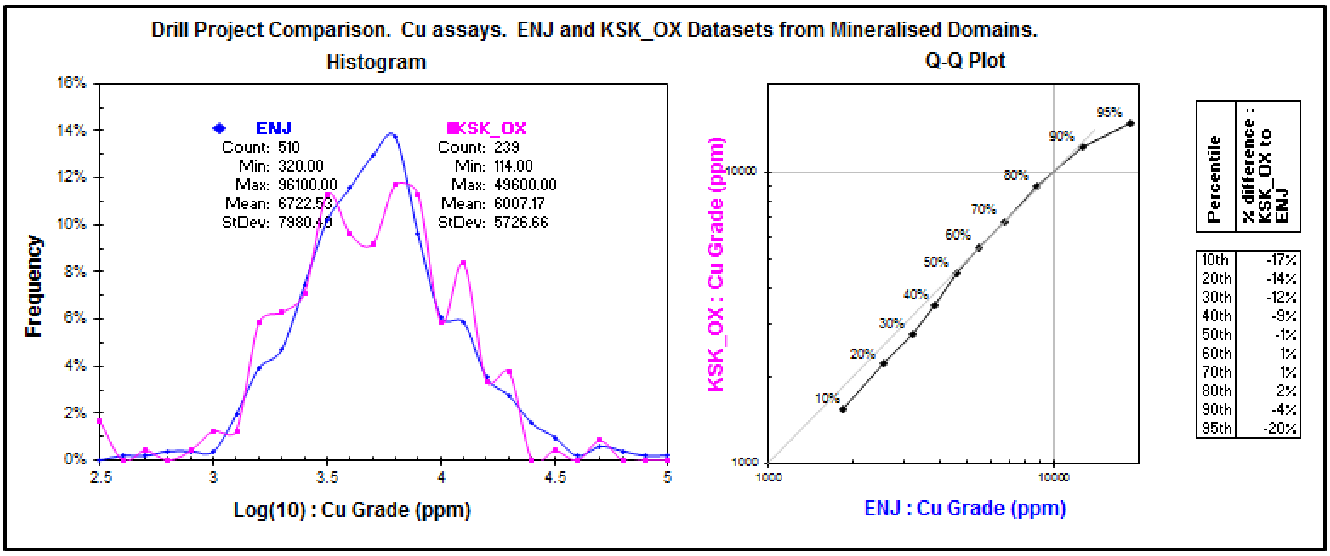
H&A notes from the OX-KSK graphs:

- The QC program undertaken is limited and not ideal for assessing the reliability of assaying of samples to be utilized in generating resource estimates.
- There is no concern regarding the degree of cross-sample contamination, however early batches (K30001 to K30010)show that the laboratory performance is questionable with the level of contamination in coarse blanks being up to double that of later batches.
- CRM standards show that laboratory performance for early batches (K30001 to K30009) is of concern, as:
  - All copper results for standard OREAS52pb (3338ppm Cu) are within the “warning” classification (>2StdDev from expected value as specified by the CRM documentation).
  - Copper results for the inserted standard OREAS50pb (7440ppm Cu) are more in alignment with their expected value, however the precision in batches K30001 to K30009 is poor compared with batches K30010 and above.

The reliability of copper results for batches K30001 to K30009 is yet to be confirmed. This casts doubt on the suitability of assays in mineralized intervals for hole KBK-0021 in underpinning resource estimates. Hole KBK-0021 is located in the eastern extent of the modeled mineralization. There is significant drilling in mineralization to the west of hole KBK-0021 and three holes are located to the east of KBK-0021. The weighting of samples from the surrounding holes will effectively restrict the influence of hole KBK-0021 to informing resources within the immediate vicinity of its drill trace. The impact of any confidence in the assays for hole KBK-0021 is expected to be minimal and most likely immaterial when considering the classification criteria for the BKM 2015 Resource Estimate at global scale.

H&A is of the opinion that the copper assays for the ENJ-KSK drill program are suitable for underpinning resource estimates being considered for Classification under the guidelines set out in the Canadian NI 43-101. H&A has compared the copper assay populations from ENJ-KSK with the combined KSK and OX-KSK programs and with the assays from pre-2015 with the 2015 KSK drilling and considers that, for the purpose of generating the BKM 2015 Resource Estimate, all populations are statistically the same (Figure 12-6 and Figure 12-12). H&A is of the opinion that, although the reliability of the pre ENJ-KSK drill assay data cannot be assessed directly, the similarity of the statistical-distributions adds confidence in this data and H&A proposes that the probability this data containing material issues affecting accuracy or confidence in the BKM 2015 Resource Estimate is low.

Figure 12-6 Copper grade comparison - recent ENJ-KSK assays vs combined historic KSK and OX-KSK assays



12.2.3.b 2015 Copper Assays

KSK submitted half core routine samples to PT Intertek Utama Services (ITS) Jakarta laboratory for sample preparation and analysis (for the laboratory KAN accreditation certificates refer to Appendix 15 of the resource report (Hackman, 2015). The sample preparation flowsheet is presented at Figure 12-7. All samples were assayed for copper by ITS method IC30 with four samples returning assays of >12%Cu being re-assayed by ITS method GA30. Details of the analytical methods are as follows:

- Sample assay charge: IC30 = 0.50g; GA30 = 0.25g
- Digest method: digested to incipient dryness with Nitric, Hydrochloric and Perchloric acids. The salts are re-dissolved in Hydrochloric Acid and made to final volume in a volumetric flask using distilled water.
- Analytical method: ICP-OES
- Lower limit of detection, Cu: IC30 = 2ppm; GA30 = 0.01%

- Upper limit of detection, Cu: IC30 = 10%; GA30 = unlimited. Reanalysis by GA30 is primarily due to the upper limit for IC30 however may also be conducted to confirm higher IC30 grade results for QC purposes.

KSK employed coarse and pulp blanks, standards and coarse crush and split duplicates with the routine samples to assess copper assay reliability. ITS included blanks, standards, second charges (same batch) and repeat assays (subsequent assay batch) in the analytical stream. Insertion rates and sizing test results are shown at Table 12-14.

Coarse blanks and coarse crush and split duplicates were preferentially inserted where mineralization was observed. KSK pulp blanks were inserted following standards. KSK utilized the commercially available Ore Research & Exploration Pty Ltd standards OREAS 50C (7420ppm Cu, performance gate of 1STDEV = 160ppm Cu) and OREAS 151A (1660ppm Cu, performance gate of 1STDEV = 50ppm Cu). Details of the standards employed by ITS are presented at Table 12-15.

Figure 12-7 KSK 2015 drillcore sample preparation flowsheet.

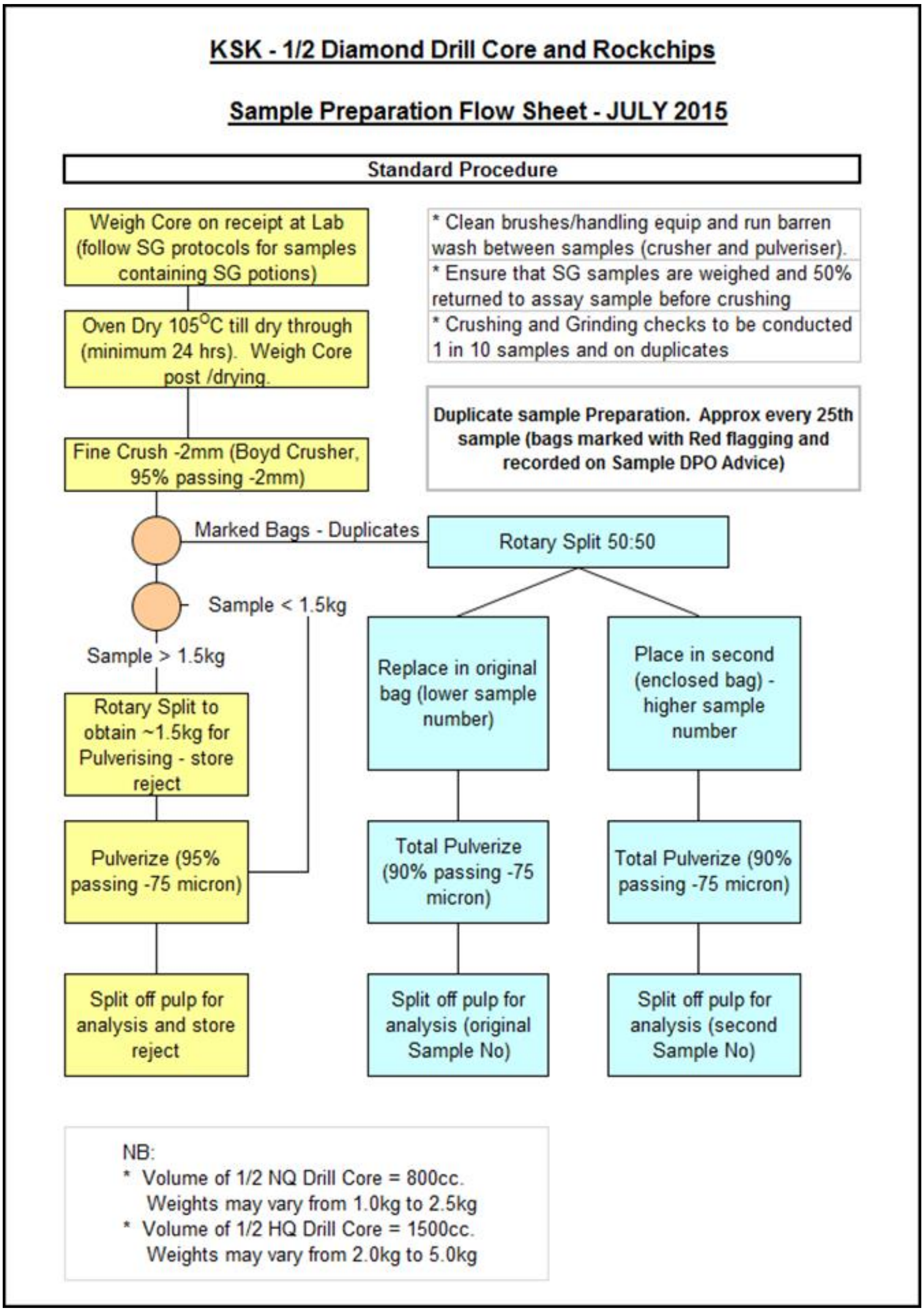


Table 12-14 Quality control sample insertion rates and sieve sizing analysis

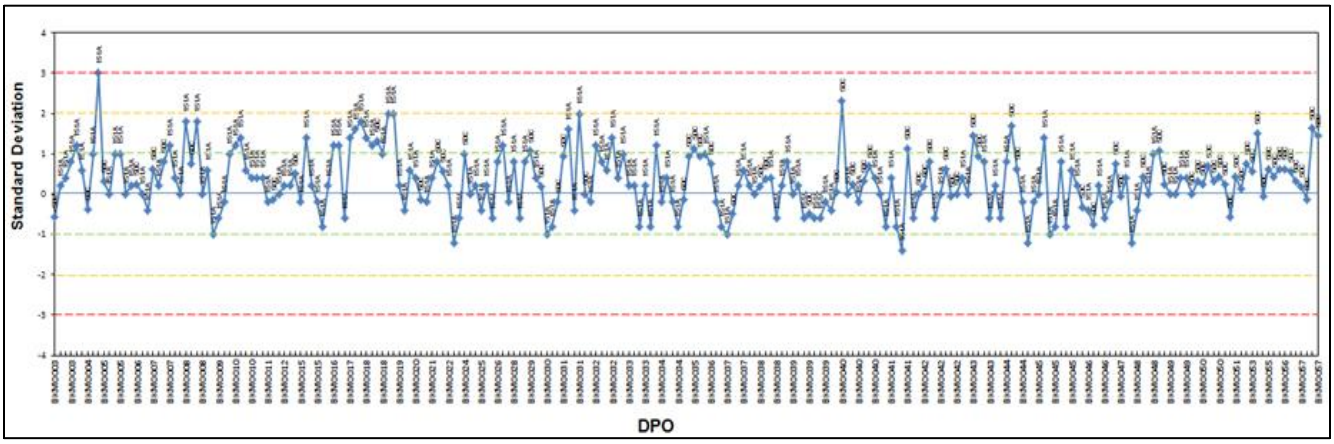
Batch	Routine Sample Count	Inclusion Sample Percentage of Batch								Sizing Distribution (number of tests in each material passing category)			
		KSK Quality Control Samples				ITS Lab Quality Control Samples				-2mm	-75micron		
		Standards	Coarse Blanks	Pulp Blanks	Coarse Crush Duplicates	Standards	Blanks	Second Charge	Repeat Assay	> 95% < 100%	> 85% < 90%	> 90% < 95%	> 95% 100%
BKM0003	74	6%	2%	2%	3%	10%	3%	6%	3%				5
BKM0004	54	6%	2%	2%	5%	8%	3%	6%	5%				4
BKM0005	66	5%	3%	1%	5%	8%	3%	6%	5%				3
BKM0006	60	6%	3%	1%	4%	11%	3%	4%	6%				5
BKM0007	84	7%	2%	3%	3%	9%	3%	6%	5%				5
BKM0008	71	7%	2%	2%	4%	10%	2%	6%	4%				4
BKM0009	45	6%	2%	2%	4%	12%	4%	6%	6%				3
BKM0010	47	7%	2%	4%	4%	11%	4%	5%	5%				3
BKM0011	72	6%	2%	1%	5%	8%	2%	5%	5%				5
BKM0012	14	12%	6%	0%	0%	12%	6%	6%	6%				1
BKM0015	56	8%	2%	3%	3%	12%	3%	6%	6%				4
BKM0016	57	6%	3%	3%	4%	12%	3%	6%	6%				4
BKM0017	61	6%	1%	1%	4%	9%	3%	4%	7%				4
BKM0018	59	6%	1%	1%	4%	9%	3%	4%	6%				4
BKM0019	55	5%	3%	2%	3%	6%	3%	6%	5%				3
BKM0020	59	7%	1%	1%	3%	7%	3%	6%	4%				5
BKM0021	26	3%	3%	3%	6%	6%	3%	6%	6%				2
BKM0022	52	5%	2%	2%	5%	7%	3%	7%	3%				3
BKM0023	22	4%	0%	0%	4%	8%	4%	4%	8%				2
BKM0024	37	7%	5%	2%	2%	14%	5%	5%	5%				3
BKM0025	39	7%	2%	2%	4%	13%	4%	7%	4%				3
BKM0026	36	5%	0%	2%	5%	10%	2%	2%	7%				3
BKM0028	41	8%	0%	4%	2%	10%	4%	2%	6%				3
BKM0029	45	4%	4%	2%	4%	10%	4%	6%	6%				3
BKM0030	63	4%	3%	3%	4%	8%	3%	5%	4%				4
BKM0031	60	9%	1%	1%	3%	10%	3%	6%	3%				4
BKM0032	118	6%	2%	1%	4%	7%	3%	5%	4%	13			13
BKM0033	60	6%	3%	3%	4%	6%	3%	6%	6%	8			4
BKM0034	106	5%	2%	2%	4%	5%	2%	7%	4%	10			7
BKM0035	78	4%	3%	1%	4%	6%	3%	7%	4%	10			5
BKM0036	68	5%	4%	3%	4%	9%	3%	6%	5%	9			4
BKM0037	70	6%	4%	1%	4%	6%	2%	6%	5%	9			9
BKM0038	101	6%	3%	3%	4%	8%	3%	5%	6%	10			7
BKM0039	112	6%	2%	2%	4%	6%	3%	5%	8%	4			6
BKM0040	143	5%	2%	2%	4%	7%	3%	7%	5%	17	1		10
BKM0041	100	5%	4%	4%	4%	7%	2%	7%	5%	10			6
BKM0042	190	4%	3%	3%	4%	7%	3%	6%	5%	23			8
BKM0043	162	4%	2%	2%	4%	8%	3%	7%	3%	19			13
BKM0044	99	4%	3%	3%	5%	8%	3%	6%	5%	12			12
BKM0045	116	6%	2%	2%	4%	8%	3%	7%	7%	14			7
BKM0046	101	6%	2%	4%	4%	6%	3%	5%	6%	13			7
BKM0047	68	5%	3%	3%	4%	8%	3%	6%	3%	8			4
BKM0048	59	9%	3%	0%	3%	4%	3%	6%	9%	7			5
BKM0049	84	5%	3%	0%	4%	7%	3%	6%	5%	10			6
BKM0050	156	6%	2%	2%	3%	6%	3%	6%	6%	18			11
BKM0051	88	7%	4%	2%	3%	9%	3%	5%	6%	11			7
BKM0052	57	5%	2%	0%	3%	11%	3%	6%	8%	7			7
BKM0053	71	6%	1%	2%	5%	7%	2%	5%	10%	9			5
BKM0054	73	7%	2%	2%	3%	8%	3%	6%	5%	9			5
BKM0055	98	4%	3%	2%	4%	6%	3%	6%	5%	12			6
BKM0056	85	6%	2%	2%	4%	7%	3%	6%	5%	10			9
BKM0057	100	6%	2%	3%	3%	8%	3%	6%	5%	12			6

Table 12-15 Laboratory standards performance criteria

Cu Standards used by ITS Laboratory (Cu Grade ppm)			
Lab Standard	**Expected Value**	**Performance Gate**	**Performance Gate Criteria**
OREAS 50C	7420	160	1STD
OREAS 151A	1660	50	1STD
OREAS 501B	2600	110	1STD
OREAS 502B	7730	200	1STD
OREAS 503B	5310	230	1STD
OREAS 504B	11100	420	1STD
BM 161	687	43	1STD
BM 49/197	3881	195	1STD
BM-16/214	15022	552	1STD
GBM399-5	29424	1446	1STD
LKSD-4	31	1.2	4%RSD
NI_LTRT13	10	0.4	4%RSD
STSD-1	36	1	4%RSD

No contamination or carry-over issues were detected in the coarse blanks or pulp blanks (both KSK and ITS). No material issues were detected in the KSK or ITS standards (KSK inserted standards shewhart control chart is presented at Figure 12-8).

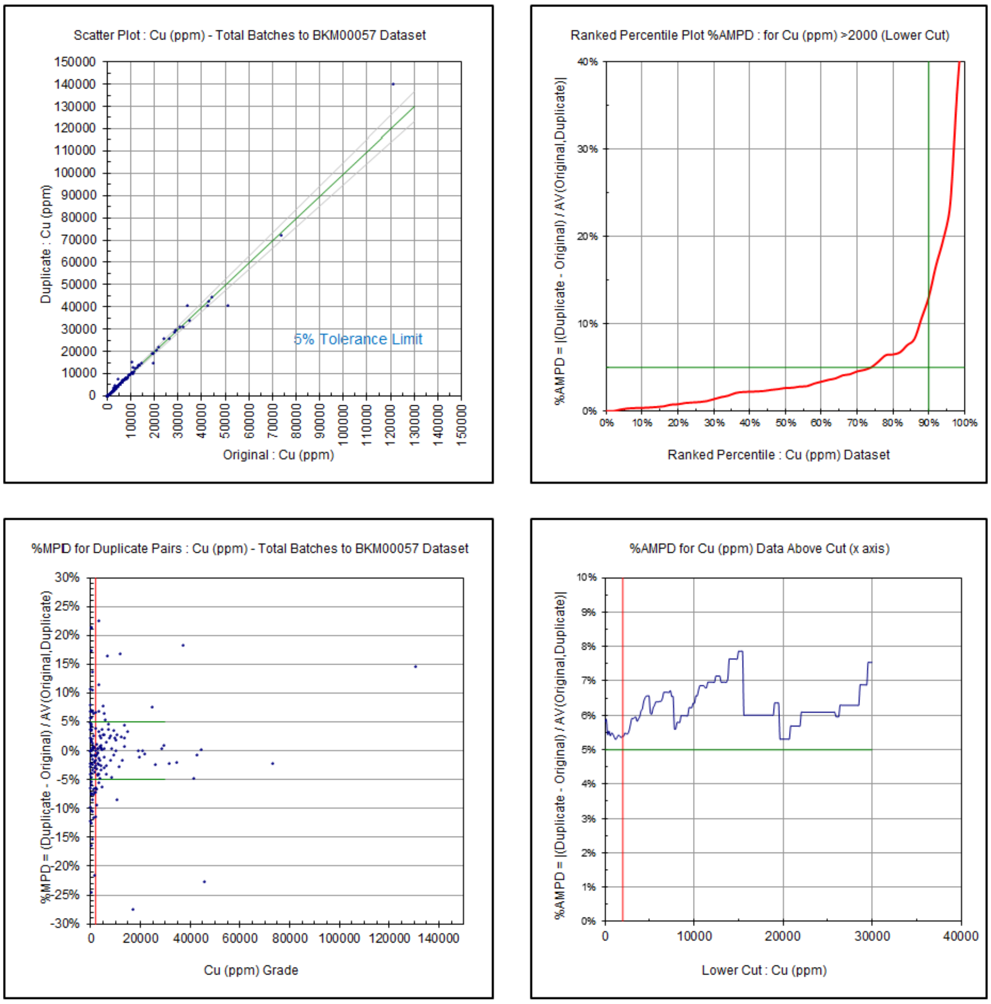
Figure 12-8 KSK standards for 2015 assays (Shewhart control chart)



The KSK Coarse crush and split duplicate copper assays (Figure 12-9) show acceptable repeatability for early batches (to BKM0038, 1956 samples), with only 7 of the 39 pairs with grades greater than 0.2% Cu showing %MPDs of greater than 5% (maximum 11% AMPD). Later batches (from BKM0039, 1962 samples) show a marked breakdown in the duplication of copper assays. For these batches, 17 of the 60 pairs with grades greater than 0.2% Cu show %MPDs of greater than 5% (maximum 47% AMPD).

Figure 12-9 Coarse Crush and Split Duplicate Analysis.

Cu (ppm) - BKM Duplicate Analysis - Split Following Coarse Crush - 2015 : Batches to BKM00057



Summary Statistics

Total Dataset - Batches to BKM00057

Count (Pairs) : 172
Av. Grade - Cu (ppm) : 6924.69
Av %MPD : 0.2%
Av %AMPD : 5.8%

Data >2000 Cu (ppm) - Batches to BKM00057

Count (Pairs) : 96
Av. Grade - Cu (ppm) : 11833.3
Av %MPD : 1.3%
Av %AMPD : 5.4%

Top 30 Pairs (%AMPD) : >2000 Cu (ppm)

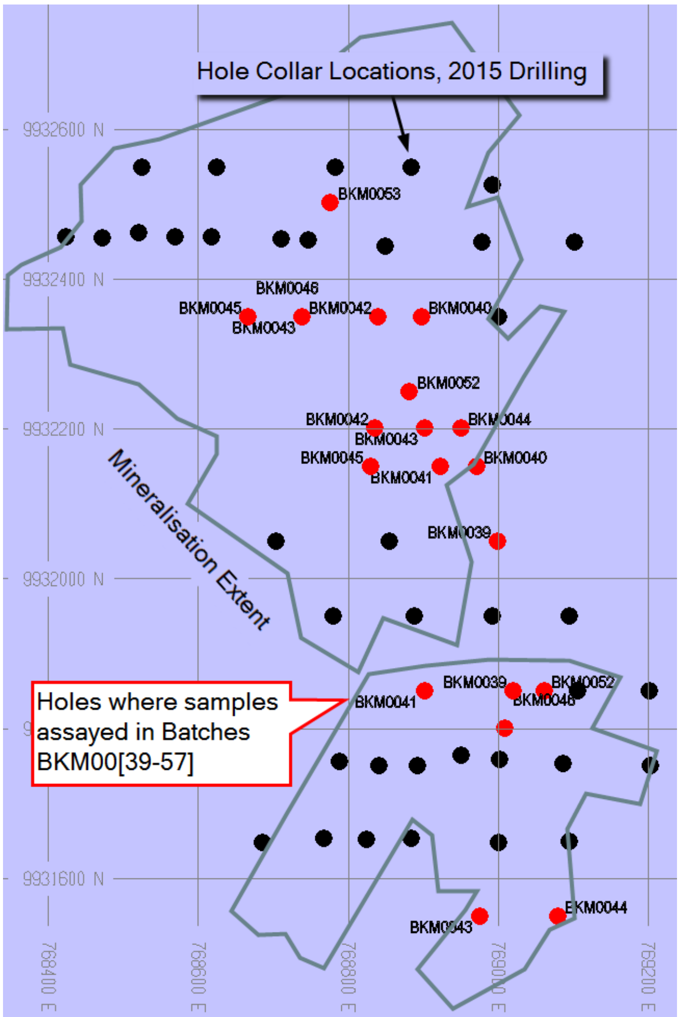
Original	Duplicate	%AMPD	Batch
4770	7670	47%	BKM0053
10200	15500	41%	BKM0041
3420	4950	37%	BKM0039
19500	14800	27%	BKM0042
50900	40500	23%	BKM0044
2840	3560	23%	BKM0045
33700	40500	18%	BKM0044
10900	12900	17%	BKM0052
6240	7360	16%	BKM0053
121000	140000	15%	BKM0040
2940	3300	12%	BKM0043
2210	1970	11%	BKM0018
2680	2440	9%	BKM0005
11100	10200	8%	BKM0003
4810	5200	8%	BKM0046
23900	25800	8%	BKM0053
2020	1880	7%	BKM0011
3290	3520	7%	BKM0025
2340	2190	7%	BKM0007
2070	1940	6%	BKM0050
5270	5620	6%	BKM0042
2260	2120	6%	BKM0055
4590	4310	6%	BKM0043
3530	3340	6%	BKM0031
5720	6040	5%	BKM0035
3800	3620	5%	BKM0038
42500	40500	5%	BKM0040
6940	7270	5%	BKM0045
8440	8060	5%	BKM0041
13100	13700	4%	BKM0040

H&A has undertaken the following evaluation of the later batches to establish reasons for the poor repeatability in coarse crush duplicate samples:

- Discussed issue with ITS who then:
    - Confirmed that no prep equipment, procedures or personnel changes have taken place at any time over the duration of the KSK work.
    - Re-assayed coarse crush duplicate pulps and “duplicated” results of original assays, confirming discrepancies in Cu grades between original and duplicate portions of samples.
    - Retrieved coarse rejects from adjacent intervals (to duplicated samples), generated coarse crush duplicates and established that grades of these duplicates portions are comparable.
    - Screened pulps of the poor duplicates and established that Cu grades of the fine and coarse fractions are comparable (with coarse fraction being marginally lower grade, reason unknown, but likely due to loading-bias of silicates in coarse material fraction).
    - Observed fine shiny filaments within the plus portion of the screened samples and shiny flecks within the minus portion of the screened samples.
    - Are now undertaking a program of isolating and analyzing the filaments and of photographing the filaments at higher magnification in an effort to establish their makeup.
- H&A further reviewed the QC data, multi-element assay data and core photos and established that:

- There is nothing unique in the duplicate samples' multi-element geochemistry from that of adjacent samples.
- Ag, As Fe, S and Sr show a similar relationship between the duplicate pairs as that observed for Cu.
- The laboratory second sample from pulps and laboratory repeat of pulps, where assayed, compare with the original assays of either duplicate (depending on which of the duplicate pair samples were selected for repeating by the laboratory).
- There is nothing noticeably different in the lithology, alteration or mineralization about the duplicate samples from other mineralized intervals in the deposit (observable from the core photos)
- The duplicate samples issue is not related to the weathering or oxidation profiles.
- The holes whose assays are questioned with regard to the poor repeatability of the coarse crush and split duplicates are located primarily within the central section of the mineralized area (Figure 12-10).

Figure 12-10 Location of holes affected by batches recording poor copper assay repeatability in coarse crush and split duplicates.



The investigation into the cause of the poor repeatability of copper assays in the coarse crush and split is continuing at the time of preparing this resource report. At this stage of the investigation it appears that the poor copper assay repeatability in the coarse crush and split duplicates is caused by the presence of a yet to be identified shiny material in the form of filaments and fine flecks. ITS suspect that this material is metallic (native copper) that has stretched and fragmented during pulverizing. H&A concurs that the presence of native copper is a likely reason for the poor repeatability (supported by the observations in the multi-element assays and the clustering of holes where the issue occurs). The absence of any visual logging of copper, any detected smearing or carry-over between samples (coarse blanks analysis) and the late appearance of and random nature of the poor repeatability in the duplicates indicates that there may be other reasons for the issue (such as external contamination). H&A is awaiting results of further analysis and identification of the material forming the filaments which should allow confident identification of the reason for poor repeatability of the duplicate assays.

There is a low to moderate risk to the 2015 resource estimate associated with the uncertainty in establishing a reason for the poor copper grade repeatability in coarse crush and split duplicate assays and determining the extent and impact of the issue with respect to assay reliability and resource copper grade. The key reasons for considering the risk at this level are:

- If the reason for issue is due to the presence of native copper:
  - The intensity and extent of this occurrence is likely to be restricted (relative to the amount and location of drilling) as native copper is yet to be identified in core.
  - Substituting particles of chalcocite (79.9%Cu) or covellite (66.5%Cu) with native Cu (100%Cu) within sulphide veins would not significantly alter the grade of samples as the relative difference in copper contents of these materials are small (wrt deposits such as gold where the impact of nuggets on an interval's grade is considerably higher and of material concern to establishing reliable grades).
  - At present 24% of duplicate samples (with grades greater than 0.2%Cu) show precisions greater than 5%MPD (average 15%) and 12% show precisions greater than 10%MPD, indicating that either native copper is not a common occurrence or that if it is then comminution and analysis of native copper is not commonly an issue.
- If the reason for the issue is laboratory or alternative:
  - Is most likely to be an accidental or hygiene issue as sample security and chain of custody protocols minimize the opportunity for deliberate tampering with samples.
  - As stated above, the intensity and extent of the issue indicates that this is not a common occurrence.
- Global comparison of assay datasets:
  - The good statistical comparison of assay datasets from the three main drilling regimes indicate that the issue does not materially impact on the reproducibility of a global dataset suitable for estimating global copper resources at BK.

12.2.3.c 2015 Umpire Laboratory Check Assays

A total of 45 mostly mineralized samples were selected from batches BKM00[3-24,26] whose QC analysis showed any issues that warranted checking at an umpire laboratory (N.B. QC for these batches showed no material issues with respect to undertaking and classifying the 2015 copper resource estimate). Four standards and three pulp blanks were included in the inter-laboratory check batch and dispatched to PT GeoAssay Laboratory, Jakarta (GA) where copper <1.0% was assayed by method GAI03 (0.5g charge, 3 acid digest, ICP-OES determination) and copper >1.0% assayed by method GOA03 (1.0g charge, 3 acid digest, AAS determination).

The following copper check assays were generated from the 45 samples:

- 54 coarse crush reject assays to compare ITS copper results with GA copper results and further assess the comminution at -2mm crush size. These duplicates were selected from those batches where the ITS coarse crush and split duplicates reported between 3% and 8% mean paired differences. 50 of the 54 comparisons were selected from mineralized samples (>0.2%Cu). There are three comparisons to be made from the 54 pairs, these are:
  - 34 direct comparisons through submitting total reject material to GA (GA pulverized and analyzed samples)
  - 12 50:50 riffle splits of coarse reject material (undertaken at ITS) and submitted "blind" to GA (GA pulverized and analyzed samples). Generating an internal GA coarse crush and split dataset for comparison with the ITS:GA dataset.
  - Pulps from 8 of the 48 above mentioned samples were also submitted, generating a further 8 comparisons of comminution at -2mm crush size.
- 39 inter-laboratory pulp repeat assays to compare ITS copper results with GA copper results to assess the robustness of the ITS analytical protocols and practices. These pulps were selected from batches where base-shifts, trends and abrupt corrections were noted in the standards QC analysis. 33 of the 39 pulps were selected from mineralized samples (>0.2%Cu). (NB. There was no consideration in preserving the original sample material integrity in storing rejects and pulps at ITS. Oxidation of sample may affect the repeatability of assay results.)

There are no discernible issues with respect to the GA copper assays detected from the standards and blanks inserted into the inter-laboratory check batch or from the 7 lab pulp repeat assays undertaken by GA. Internally it is considered that the GA copper assays are reliable.

Figure 12-11 and Figure 12-12 present the comparison between the ITS copper assays ("Original") and GA assays ("Duplicate"). Of note:

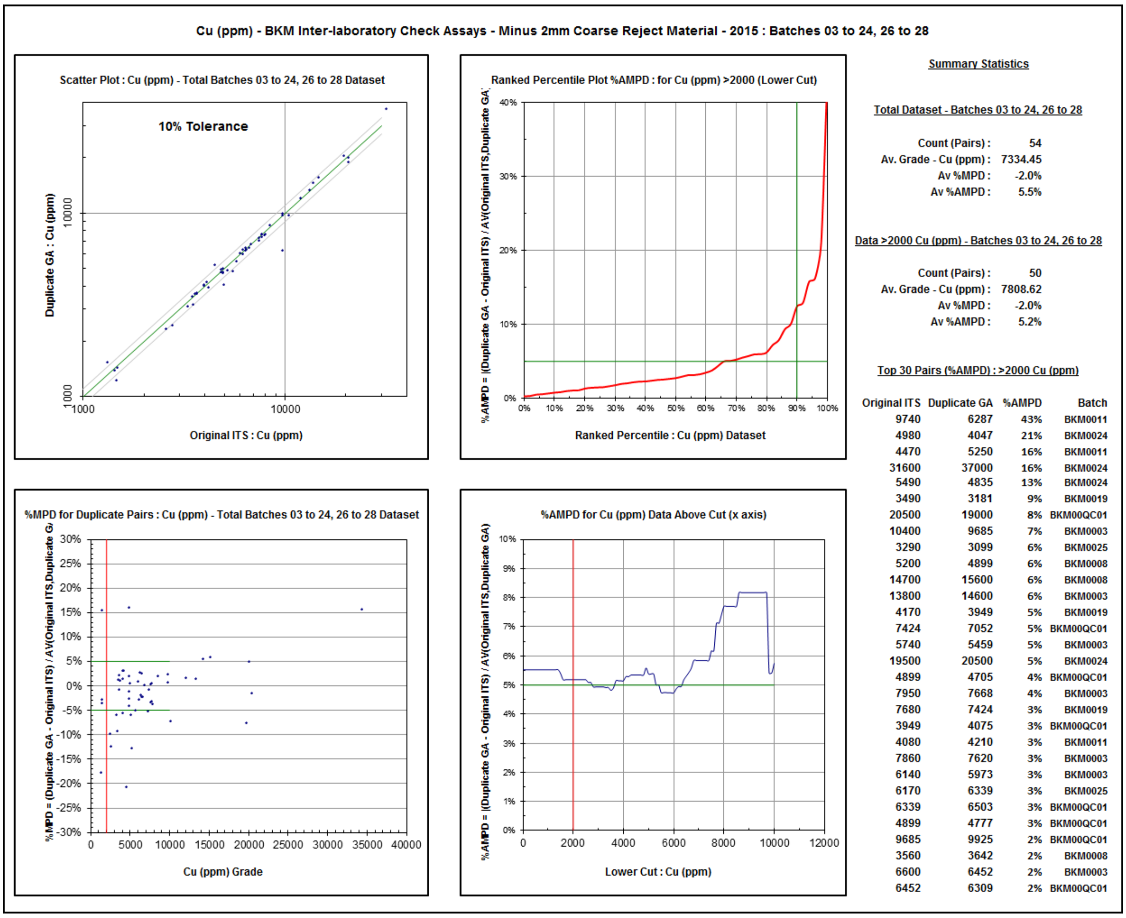
- 12 of the 54 coarse reject check assays show variance of >5%MPD with 5 of these showing >10%MPD. There is a weak negative relative bias in the GA assays for copper assays <10,000ppm.
- 16 of the 39 pulp check assays show variance of >5%MPD with 3 of these showing >10%MPD. There is a negative relative bias in the GA assays for samples assaying <10,000ppmCu.

Although the inter laboratory assay checks do not show excellent repeatability with the ITS assays, they support the robustness of the original ITS assays and further increase the belief that the ITS assays are robust and reliable for use in estimating copper mineralization at BKM. Of note, when assessing the correlation:

- GA report's copper assays <10,000ppm by method GAI03 and >10,000ppm by GOA03. ITS utilizes a threshold of 100,000ppm for re-assaying of samples by their ore grade method. The relative bias between assays from the two laboratories is only observed in the samples assaying <10,000ppmCu suggesting that GA is returning low values for these samples. The four standards submitted with the batch to GA do not show low assays, suggesting that even though GA is capable of returning reliable assays, they may not have been able to do so for the BKM samples at this time (NB. There is no detectable bias or issue in GA assays from the pre-2015 drilling as shown in Figure 12-4)

- It is likely that the coarse crush duplicate issue discussed in Section 12.2.3.2 is observable in the -2mm coarse reject comparison.
- Sampling by GA of the -2mm coarse reject material will produce a similar sample to the original split taken for preparation by ITS, however theoretically the two samples are different and this difference may account for some of the features observed in Figure 12-11.

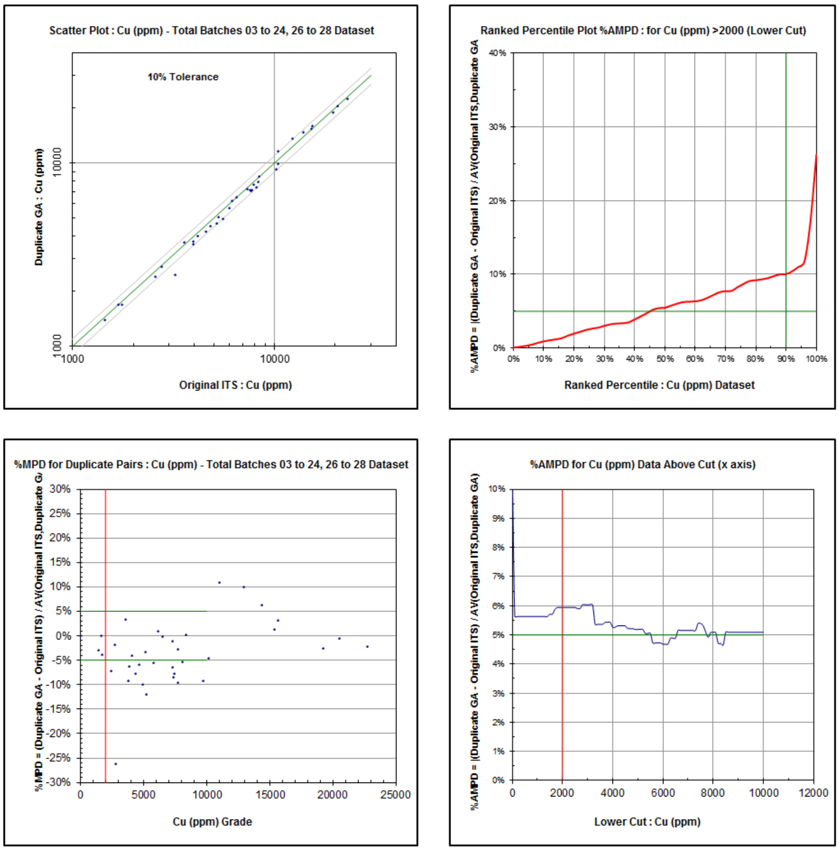
Figure 12-11 Inter laboratory check copper assay analysis; minus 2mm coarse reject material.



Original ITS	Duplicate GA	%AMPD	Batch
9740	6287	43%	BKM0011
4980	4047	21%	BKM0024
4470	5250	16%	BKM0011
31600	37000	16%	BKM0024
5490	4835	13%	BKM0024
3490	3181	9%	BKM0019
20500	19000	8%	BKM00QC01
10400	9685	7%	BKM0003
3290	3099	6%	BKM0025
5200	4899	6%	BKM0008
14700	15600	6%	BKM0008
13800	14600	6%	BKM0003
4170	3949	5%	BKM0019
7424	7052	5%	BKM00QC01
5740	5459	5%	BKM0003
19500	20500	5%	BKM0024
4899	4705	4%	BKM00QC01
7950	7668	4%	BKM0003
7680	7424	3%	BKM0019
3949	4075	3%	BKM00QC01
4080	4210	3%	BKM0011
7860	7620	3%	BKM0003
6140	5973	3%	BKM0003
6170	6339	3%	BKM0025
6339	6503	3%	BKM00QC01
4899	4777	3%	BKM00QC01
9685	9925	2%	BKM00QC01
3560	3642	2%	BKM0008
6600	6452	2%	BKM0003
6452	6309	2%	BKM00QC01

Figure 12-12 Inter laboratory check copper assay analysis; pulp material.

Cu (ppm) - BKM Inter-laboratory Check Assays - Pulp Material - 2015 : Batches 03 to 24, 26 to 28



Summary Statistics

Total Dataset - Batches 03 to 24, 26 to 28

Count (Pairs) :	39
Av. Grade - Cu (ppm) :	7295.74
Av %MPD :	2.2%
Av %AMPD :	10.8%

Data >2000 Cu (ppm) - Batches 03 to 24, 26 to 28

Count (Pairs) :	33
Av. Grade - Cu (ppm) :	8474.58
Av %MPD :	-3.8%
Av %AMPD :	5.9%

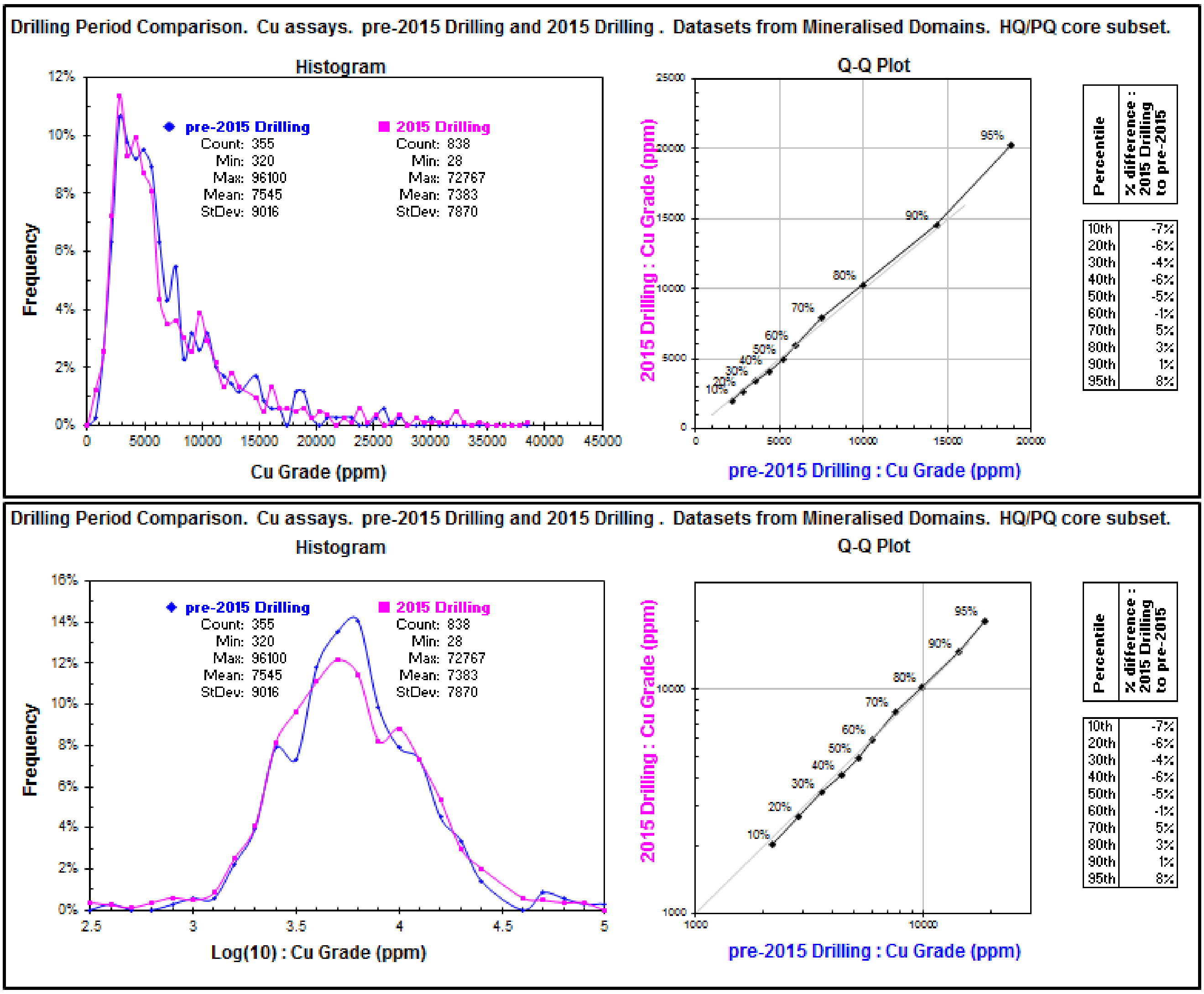
Top 30 Pairs (%AMPD) : >2000 Cu (ppm)

Original ITS	Duplicate GA	%AMPD	Batch
3210	2466	26%	BKM0004
5580	4952	12%	BKM0007
10400	11600	11%	BKM0016
12300	13600	10%	BKM0009
5200	4705	10%	BKM0008
8110	7373	10%	BKM0007
10200	9298	9%	BKM0016
3950	3605	9%	BKM0028
7680	7052	9%	BKM0019
7740	7164	8%	BKM0003
4560	4221	8%	BKM0004
2570	2393	7%	BKM0008
7580	7101	7%	BKM0024
3980	3737	6%	BKM0007
13900	14800	6%	BKM0009
4810	4533	6%	BKM0004
5990	5671	5%	BKM0009
8330	7892	5%	BKM0004
10400	9925	5%	BKM0003
4170	4006	4%	BKM0019
5270	5094	3%	BKM0028
3560	3681	3%	BKM0008
15400	15900	3%	BKM0009
7860	7644	3%	BKM0003
19500	19000	3%	BKM0024
23000	22500	2%	BKM0028
2760	2710	2%	BKM0003
15300	15500	1%	BKM0028
7330	7250	1%	BKM0003
6170	6225	1%	BKM0025

12.2.4 Comparison of Copper Assays; 2015 with pre-2015

The 2015 copper assays can be combined with the pre-2015 assays for estimating grades at BKM as they show the comparable population distributions (Figure 12-13).

Figure 12-13 Comparison of 2015 and pre-2015 copper dataset populations.



12.2.5 Copper Grade Relationship with Core Recovery

Core recoveries by length are good for the BKM mineralization with over 90% of the intervals within the mineralized domains returning >90% recovery (Table 12-16). There is also no observed relationship between length recovery and copper grade. H&A observed minor occurrences of washing or scrubbing of core from the drilling and core sawing processes and although not measured, the scrubbing is likely to impact on copper grade as the soft covellite and chalcocite will preferentially wash away over more resilient minerals such as pyrite and quartz. It is anticipated that this action will reduce the copper content in the assay sample, though core photo observations indicate any effect will be minimal.

Table 12-16 Copper grades split by recovery categories. a.) pre-2015 drilling. b.) 2015 drilling

a. pre-2015 Drilling			b. 2015 Drilling			
Logged Core Recovery within domained mineralization			Recovery (%)	Percent of Dataset	Av drill run-length (m)	Av Cu Grade (ppm)
Core Recovery (%)	Count	Av. Cu grade (ppm)				
0 to 10	5	12120	0 to 10	0.1%	1.00	5900
10 to 20	6	5250	10 to 20	0.1%	0.13	400
20 to 30	14	4605	20 to 30	0.2%	0.17	4133
30 to 40	13	5994	30 to 40	0.7%	0.23	2400
40 to 50	26	4702	40 to 50	0.5%	0.29	4790
50 to 60	16	6791	50 to 60	1.4%	0.31	3707
60 to 70	23	5904	60 to 70	1.0%	0.33	5600
70 to 80	24	5905	70 to 80	2.3%	0.43	9402
80 to 90	37	7197	80 to 90	2.0%	0.64	10950
90 to 100	997	6918	90 to 100	91.8%	1.21	7703
All	1161	6810	Total	100%	1.10	7665

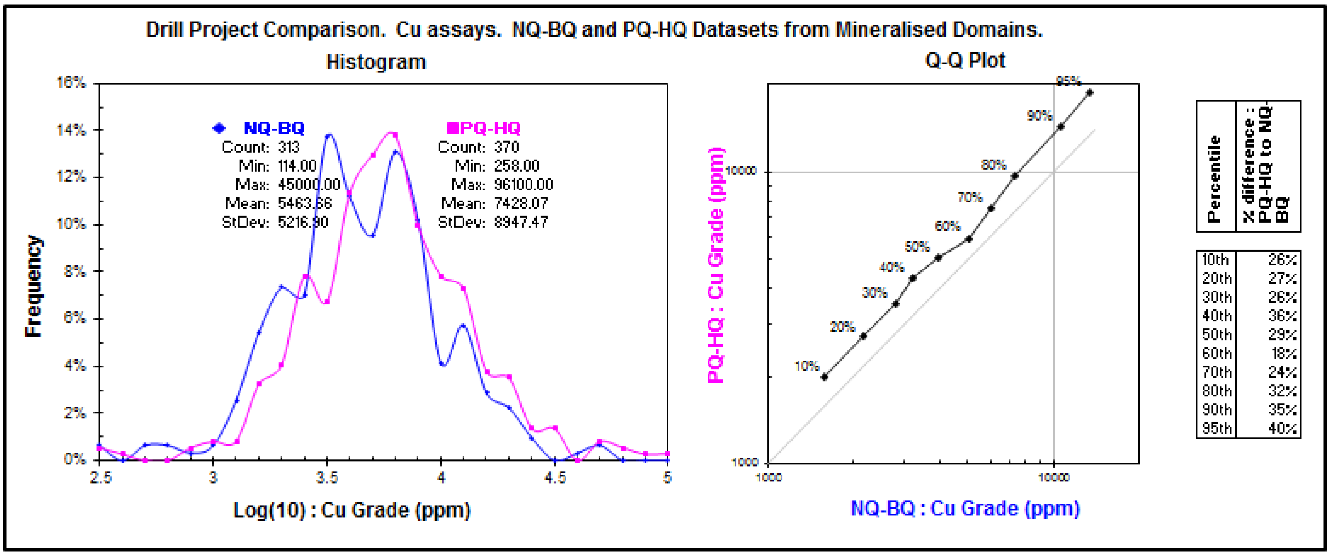
12.2.6 Copper Grade Relationship with Primary Sample Orientation and Size

The analysis of copper grade versus the primary sample size shows that the average grade for NQ-BQ drill core samples is 26% lower than the average grade for the PQ-HQ drill core samples. Figure 12-14 shows that this is because there is a population shift in copper grade-tenor of approximately this amount between the two datasets. The PQ-HQ dataset also shows a greater range of copper values and a higher maximum value. A population shift such as the one shown in Figure 12-14 can be due to one or both of the following:

- PQ and HQ drilling samples mineralization closer to the surface than NQ and BQ drilling (average depths are shown at Table 10-2). The grade differential may be due to primary zonation within the BKM mineralization.
- The (relatively) thin and low density copper mineralization veining intersected in the BKM drilling can be considered similar to results anticipated from nuggetty gold mineralization (though not as severe). In these cases the most common effect of decreasing the primary sample size is to decrease the assay grades to levels less than the actual grade of the mineralization being sampled. It is unknown if this is happening at BKM, however H&A suspects that it is likely (especially if the copper mineralization is not related to supergene events) and that the estimated copper grade of the deposit may be compromized by the NQ and BQ drilling.

All drilling in the 2015 program was undertaken employing HQ triple tube diamond core. The same baseshift in copper assays is observed when comparing the 2015 copper assays with the pre-2015 NQ-BQ sample assays. There is a good comparison with the pre-2015 PQ-HQ core sample copper assays and the 2015 copper assays (Figure 12-13).

Figure 12-14 Copper grade comparison between PQ-HQ drilling and NQ-BQ drilling (pre-2015 data).



All holes that intersect mineralization are drilled westerly except BK-02, BK044-02, KBK-0023 and KBD03-02 which are drilled to the east and BK-01, BK-04 and BK-06 which are vertical holes. There is no discernible copper grade differential between easterly and vertical holes and those drilled towards the west. The small number of available easterly and vertical holes is not a good sample size to determine what the ideal drilling direction will be for generating a robust resource estimate at BKM. Further work is required in investigating optimum drilling directions throughout the mineralization.

12.2.7 Tonnage Factor Determination

The 2015 DBD measurements were used to determine tonnage factors for the 2015 resource estimate. Four domains were employed to stamp DBD values onto the block model for the 2015 resource estimate. These are:

- mineralized domains within 17m of surface (or 20 downhole from collar) = 2.70t/m3
- mineralized domains > 17m deep = 2.85 t/m3
- non- mineralized material within 17m of surface = 2.33 t/m3
- non- mineralized material deeper than 17m = 2.77 t/m3

In determining the tonnage factors the data was coded as follows and assessed (Table 12-17):

- Being within or beyond 17m vertically from surface (20m downhole),
- within and outside of the resource grade domains (>0.2%Cu threshold) and
- the sample having been dispatched to the laboratory for drying (porous and permeable samples, water soaking) or as having been determined for DBS on site without drying (competent and sealed core).

The 17m vertical split was established by visible observations from the dataset and core photos and it can be seen from the average DBDs for the lab dried core (Table 12-17) that significantly lower values are recorded from samples within the shallow zone than those from the deeper zone. Although the 17m divider will not fit throughout all areas of the deposit it is considered a reasonable estimation of the change between material affected by surface weathering (and other processes) and the better preserved material at depth.

The surface “base_2-70_DBD_2015.00t” and mineralized domains were utilized to assign tonnage factors into the 2015 BKM block model.

Table 12-17 DBD and tonnage factor evaluation

Classification		Within Mineralized Domains		Outside of Mineralized Domains	
Sample Depth Range	Core Condition	Number of Samples	Average DBD	Number of Samples	Average DBD
Within 17m of Topo Surface	Lab Dried (water soaking)	43	2.30	110	1.96
	Non-water soaking	99	2.88	112	2.69
	Combined	142	2.70	222	2.33
Deeper than 17m from Topo Surface	Lab Dried (water soaking)	69	2.63	206	2.64
	Non-water soaking	524	2.88	590	2.81
	Combined	593	2.85	796	2.77
All Samples		**735**	**2.82**	**1018**	**2.67**

13 MINERAL PROCESSING AND METALLURGICAL TESTING

Miller Metallurgical Services (MMS) has undertaken an analysis of the preliminary metallurgical characterization tests on the BKM Project. The testing has been undertaken at two laboratories; Intertek in Indonesia and Core Metallurgy in Australia. Full details of this testwork are provided in the report "*Beruang Kanan Main Copper Project - Heap Leach Metallurgical Test Interpretation Report*" (Miller Metallurgical Services, 2016).

The results and interpretation of the tests are summarized here.

13.1 GEOLOGY

The geological interpretation of mineralized material types has been extended to include the sequential assay results of the copper mineralized material. The geological data base has been modified as above and used in the PEA mine planning and scheduling. The characterization of leachable copper content is an integral part of this data (refer to Section 14.11).

13.2 MINERALOGY

Copper minerals are dominated by:

- Chalcocite in most areas
- Mixtures of chalcocite, covellite and bornite
- Chalcopyrite is present in most mineralized zones to some extent
- There appears to be some non-sulphide copper that has been allocated to silicates. This material is not yet confirmed due to dispersed nature of the copper

The copper minerals are widely dispersed in pyrite in veins within the rock matrix and fine crushing may be required. Pyrite is the dominant sulphide with high Fe:Cu ratios.

13.3 WEATHERING

There is little meteoric weathering except for a shallow 'soil' layer of a few meters thickness which also contains some copper.

13.4 OXIDATION

Other than the soil layer there is no significant oxidation of the sulphides.

13.5 ROCK TYPES

The geological rock types have been used to characterize the temporal pattern for plant feed based on the mine plan. The geo-mechanical characteristics of unleached and leached rocks may predominate over the geological typing. The presence of clays is noted in the logs. The proportion of fines and high retained moisture in leached material are indicators that the leached rock physical characters may be the prime driver for leaching kinetics.

13.6 COMMINUTION

Little comminution testing has been done. The general rock properties are not considered hard.

13.7 CRUSH SIZE SELECTION

The finer crush size leached slower than the coarser crush size in the two column tests. The crush size will be a balance between the finely disseminated copper minerals, the stringer like pyrite and the rock geo-mechanical characteristics. It is likely that crushing in the 12 mm to 19 mm range may be needed.

13.8 HYDROLOGICAL / GEOMECHANICAL / GEOTECHNICAL

The hydrology and geotechnical characteristics of the mineralized materials are yet to be fully determined. The rocks appear to have variable clay content and to be somewhat friable; as shown by the fines generation during the excess sample composite blending time. The leach rocks appear to have generated significantly more fines than the head samples. The residual moisture in the Intertek columns is high from the high fines content.

Based on the above observations, the various rock types require further evaluation. Some care will be needed in sample preparation and internal procedures to more closely model the likely field operational process (i.e. crushing, blending, agglomeration, placement (in the test columns) and leaching conditions).

13.9 LEACHING

Higher leach recoveries than expected have been achieved without overt signs of biological assistance. It is proposed that the process is being dominated by a galvanic leaching process similar to the Galvanox™ process, whereby copper minerals are galvanically leached by the high pyrite to copper ratio in the mineralization.
Details of the test work undertaken are provided in the following sub-sections.

13.9.1 Shake Flask Tests

Preliminary shake flask tests on ground mineralized material were conducted to provide information on the bacterial/ferric leaching of the ore. These were conducted only at Core Metallurgy. Leaching was conducted at:

- Three different grind sizes
- Two different initial pH's
- Using Core standard bacterial culture addition.

The results of the leaching Oxidation Reduction Potential (ORP) and pH are provided in Figure 13-1 to Figure 13-3 below.

Items of interest from these tests are:

- The starting pH of 1.8 always dropped to around 1.5 of less
- The ORP for the higher grade samples (BKM0059 & 60) initially rose to + 450 mV SHE but then fell to +350 mV
- The OPR for the low grade (close to average mined grade) rose and stabilized at +350 mV SHE.
- There are no discernible trends with either starting pH or grind size.

Figure 13-1 Shake Flask Test BKM0060 - ORP Profile

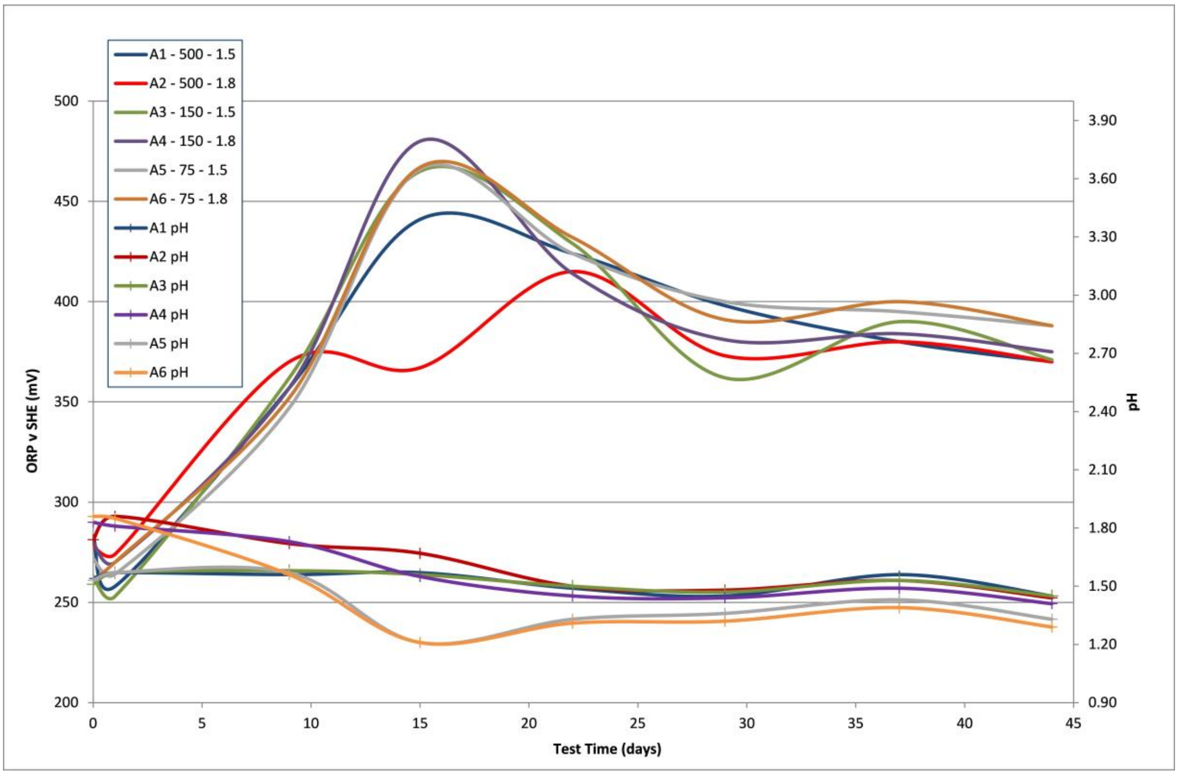


Figure 13-2 Shake Flask Test BKM0059 - ORP Profile

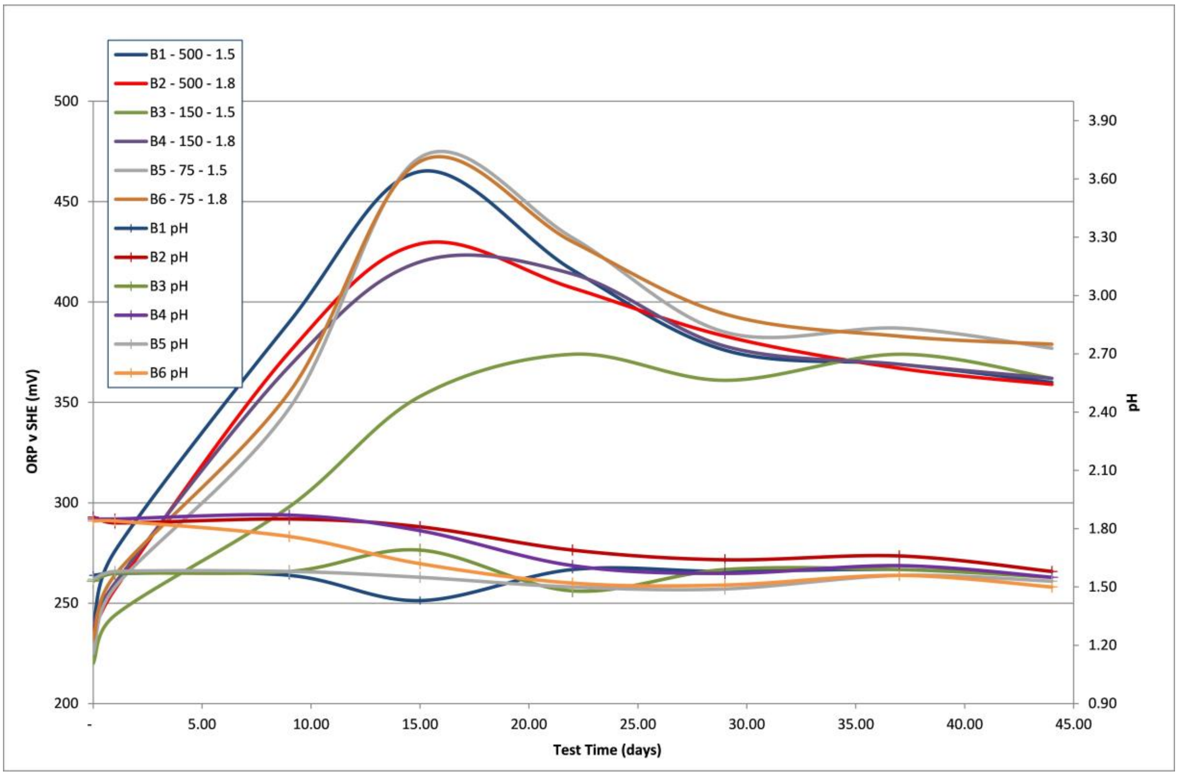
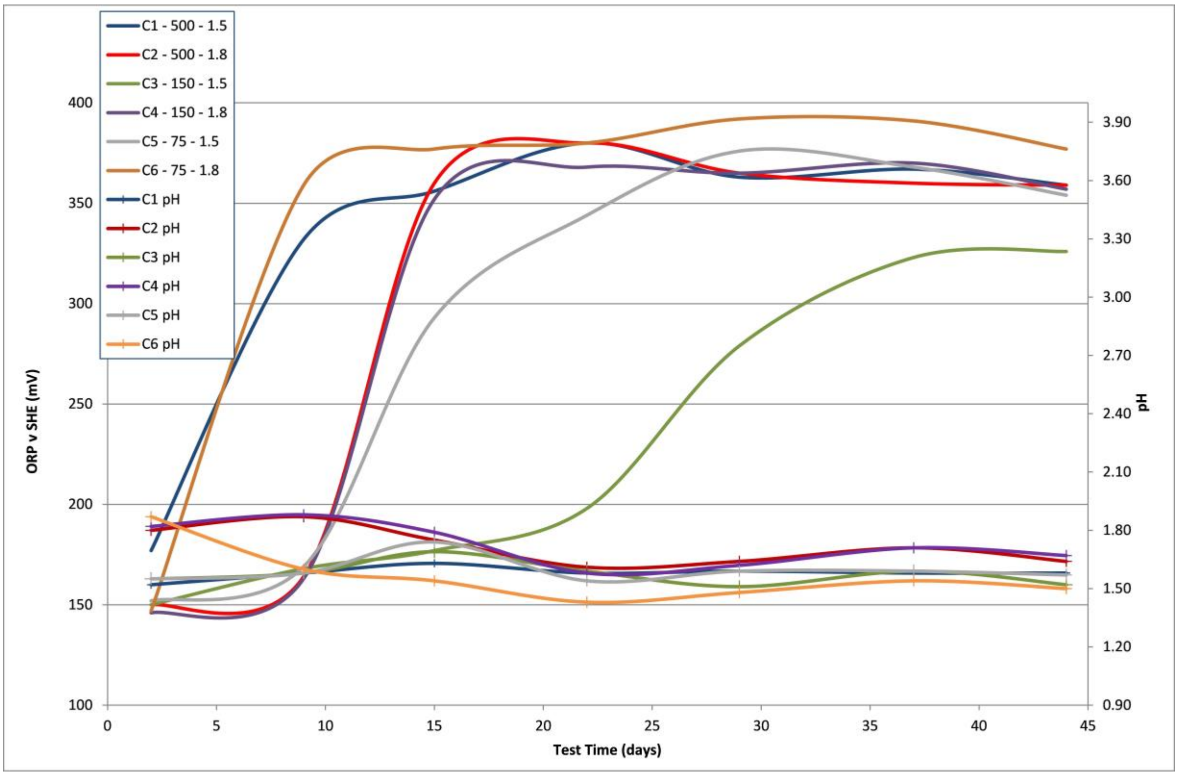


Figure 13-3 Shake Flask Test BKM0058 - ORP Profile



The recovery of copper from the tests is provided in Table 13-1. There are two alternate recovery calculations made:

- Head assay and tail assay
- Tail assay and copper in solution (which also provides a back calculated head grade).

The results indicate:

- There are no discernible trends of recovery with grind size or initial pH. Although there may be a mild increase of recovery with finer grind size.
- The back calculated head for the low grade (BKM 0058) is higher than the assay head provided by Intertek. This results in a lower recovery than indicated by the head/tail assays alone
- Absolute level of recovery is relatively high considering the low ORP that was achieved in the testing.
  - 74% to 80% for high grade (BKM0058)
  - 72% - 73% for medium grade (BKM0059)
  - 75% - 77% for low grade (BKM0060)
- The high recoveries are not consistent with conventional bacteria leaching alone
- It is likely that high pyrite content and high pyrite:copper ratio are providing an alternate ‘direct’ galvanic leaching environment.

Table 13-1 Shake Flask Tests Copper Recovery

Test Ref	Test Conditions			Final Solution Cu (mg/L)	Cu Residue Grades (ppm)				Cu_{TOT} Recovery (%)	
	Sample	P_{80} (µm)	pH		Cu_{Total}	Cu_{HAS}	Cu_{CNS}	Cu_{Res}	Solids' Assay[1]	Calc. Head
1023C1	BKM0058	500	1.5	1700	5765	275	2900	2590	83.6	79.8
1023C2		500	1.8	1680	6457	272	4120	2060	81.6	76.6
1023C3		150	1.5	2140	7840	411	5850	1580	77.6	74.2
1023C4		150	1.8	2380	7766	352	4610	2805	77.8	75.4
1023C5		75	1.5	2310	6850	338	4420	2090	80.5	78.2
1023C6		75	1.8	2300	6245	331	3120	2800	82.2	79.5
1023C6 (2)		75	1.8	2330	6263	287	3160	2820	82.1	80.9
1023B1	BKM0059	500	1.5	697	2649	174	2480	0	72.6	71.7
1023B2		500	1.8	737	2643	133	2460	53	72.7	71.6
1023B2 (2)		150	1.5	663	2436	124	2210	99	74.8	71.8
1023B3		150	1.8	717	2629	147	2480	0	72.8	72.4
1023B3 (2)		75	1.5	695	2627	160	2430	38	72.9	72.4
1023B4		75	1.8	680	2614	116	2500	0	73.0	72.6
1023B5		500	1.8	600	2645	163	2480	0	72.7	73.5
1023B6		150	1.5	623	2548	122	2430	0	73.7	72.9
1023A1	BKM0060	500	1.5	505	1425	127	865	433	76.4	75.9
1023A2		500	1.8	524	1459	114	893	453	75.8	76.3
1023A3		150	1.5	418	1437	170	864	404	76.2	74.6
1023A4		150	1.8	463	1497	164	904	429	75.2	73.9
1023A5		75	1.5	418	1435	154	850	431	76.2	76.8
1023A6		75	1.8	421	1449	157	889	404	76.0	75.8
1023A6 (2)		75	1.8	446	1462	175	908	380	75.8	75.6

(1) Neglecting Mass Loss

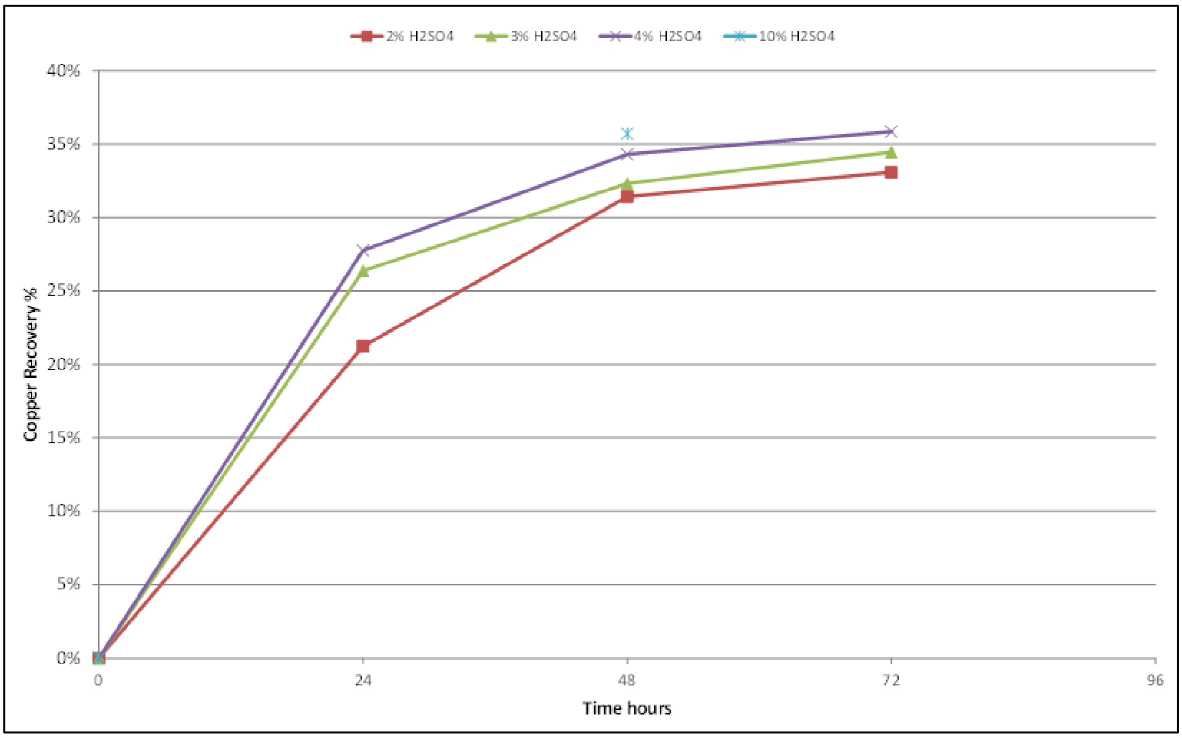
13.9.2 Bottle Roll Tests

PT Intertek Utama Services conducted bottle roll tests on the one sample but crushed to -3.25 mm. The leaching time is short at 72 hours. The leach results are shown graphically in Figure 13-4 below.

There is only a mild increase in recovery with increasing acid concentration; indicating that mild acid conditions will be sufficient for column leaching of the ore.

The leach recovery is significantly higher than the acid soluble copper content would indicate. This provides another set of evidence for a non-bacterial galvanic leach pathway.

Figure 13-4 Intertek Bottle Roll Test



Core Metallurgy undertook bottle roll tests using:

- As received material (AR) crushed at Intertek to -13 mm
- Crushed to – 8 mm
- Inoculated with Core standard biomass

The leaching results for the three sample composites are provided in Figure 13-5 to Figure 13-7 below.

Figure 13-5 Core Bottle Roll Test BKM0060 – Leach Performance

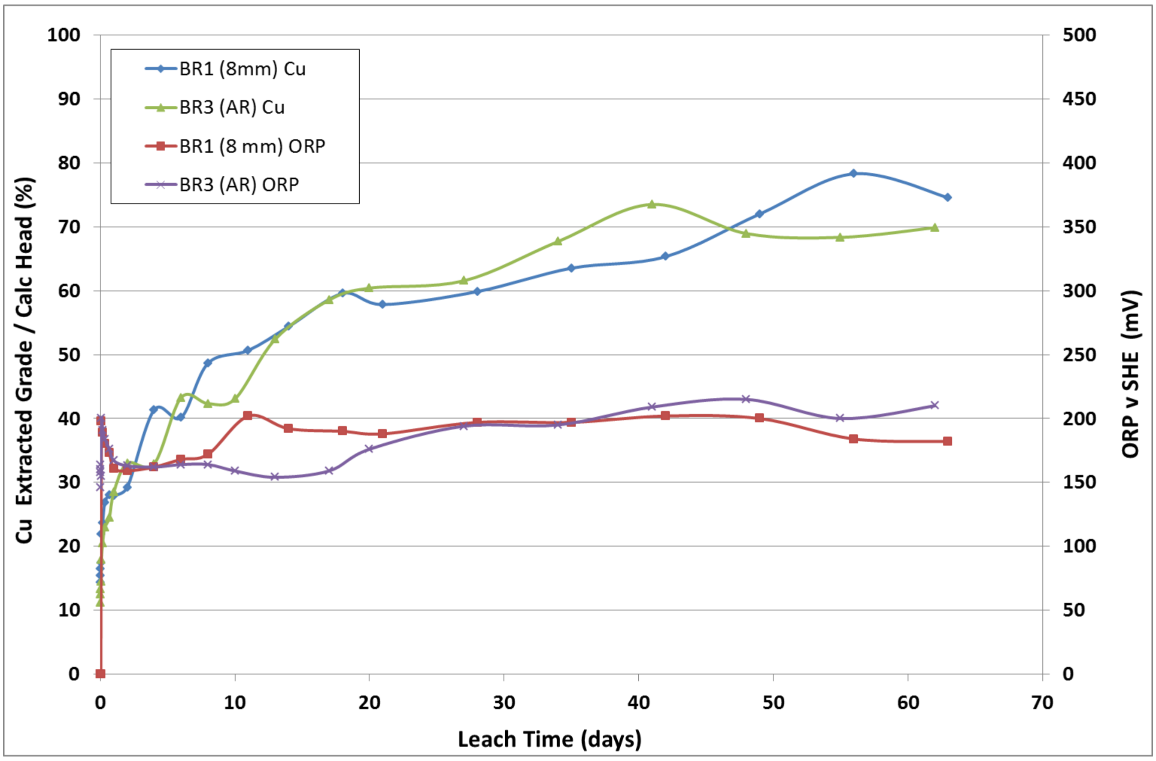


Figure 13-6 Core Bottle Roll Test BKM0059 - Leach Performance

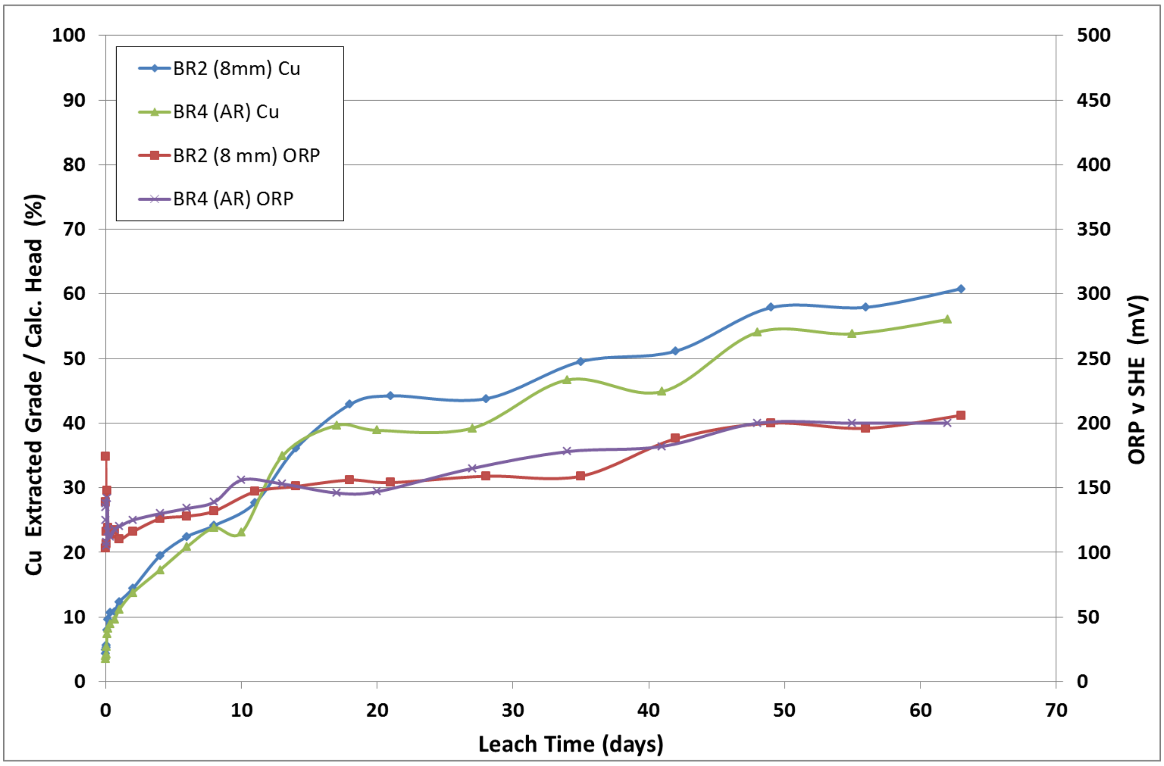
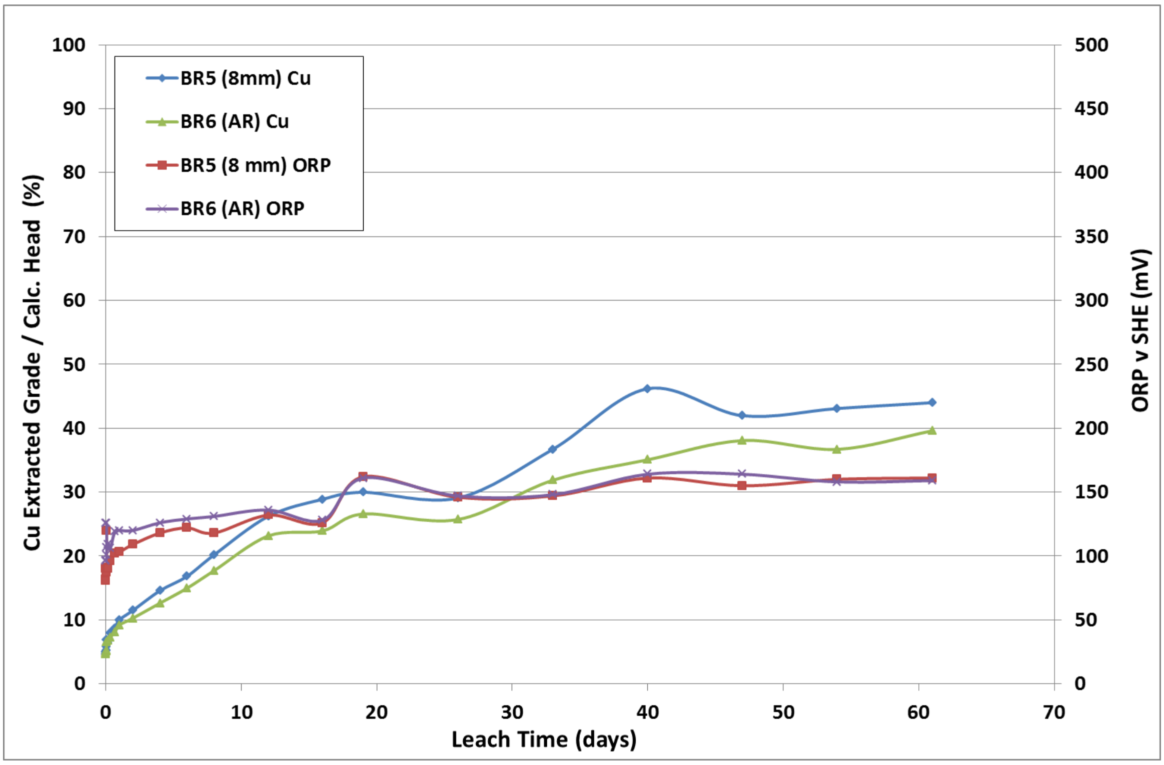


Figure 13-7 Core Bottle Roll Test BKM0058 - Leach Performance



The results are summarised in below.

Table 13-2 13.8.1 Shake Flask Tests Copper Recovery

Parameter	BKM 0058		BKM 0059		BKM 0060	
	AR	-8 mm	AR	-8 mm	AR	-8 mm
Recovery % Cu tot	39.6	44	56	60.8	68.9	74.6
Final ORP mV SHE	159	159	200	206	182	210
Recovery Difference		4.4		4.8		5.7
Total Cu leached: q	36.0	NA	15.9	16.0	12.0	12.6

The lower grade sample has the highest recovery of 74.6% with the highest (yet still very low) ORP of 210 mV SHE. As the grade rises, the leach recovery drops but the total copper leached increases substantially. All tests were continuing to leach at test termination.

The low ORP and, the lower ORP with higher grade relationship, is consistent with pyrite galvanic leaching of the copper minerals. The pyrite concentration is well above the minimum 2:1 ratio found to be needed in the Galvanox™ process. This process also operates at low ORP and is able to leach all copper minerals including chalcopyrite.

13.9.3 Column Tests

Intertek undertook two short column tests were conducted on their low grade sample.

- Two crush sizes were used at -13 mm and -6.5mm
- The leach was initially open circuit with a high acid concentration of 20 g/L

- After the column collapse the leach was changed to closed circuit with a 10 g/L acid in the feed.

The leach recovery curves are shown in Figure 13-8 and Figure 13-9. The recovery has been normalised to the head/tail calculated recovery as the data on column exit solution volumes is not available.

Figure 13-8 Column Test Normalized Head/Tail Recovery - 13mm

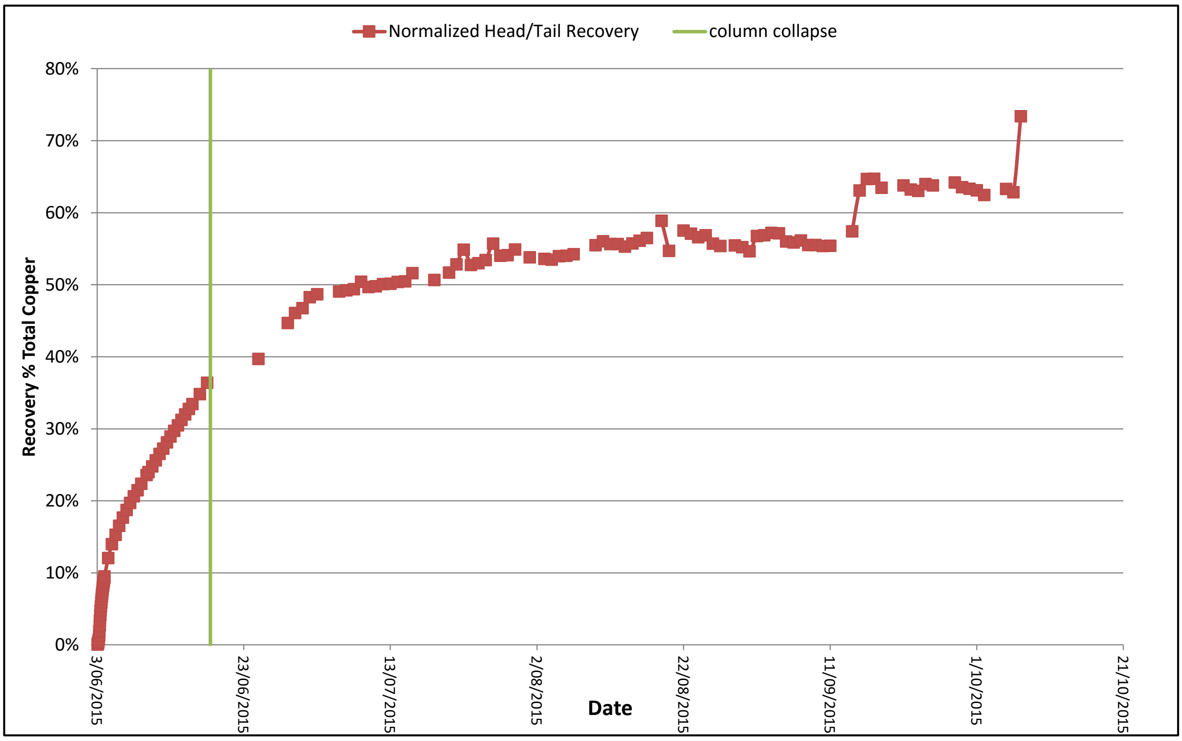
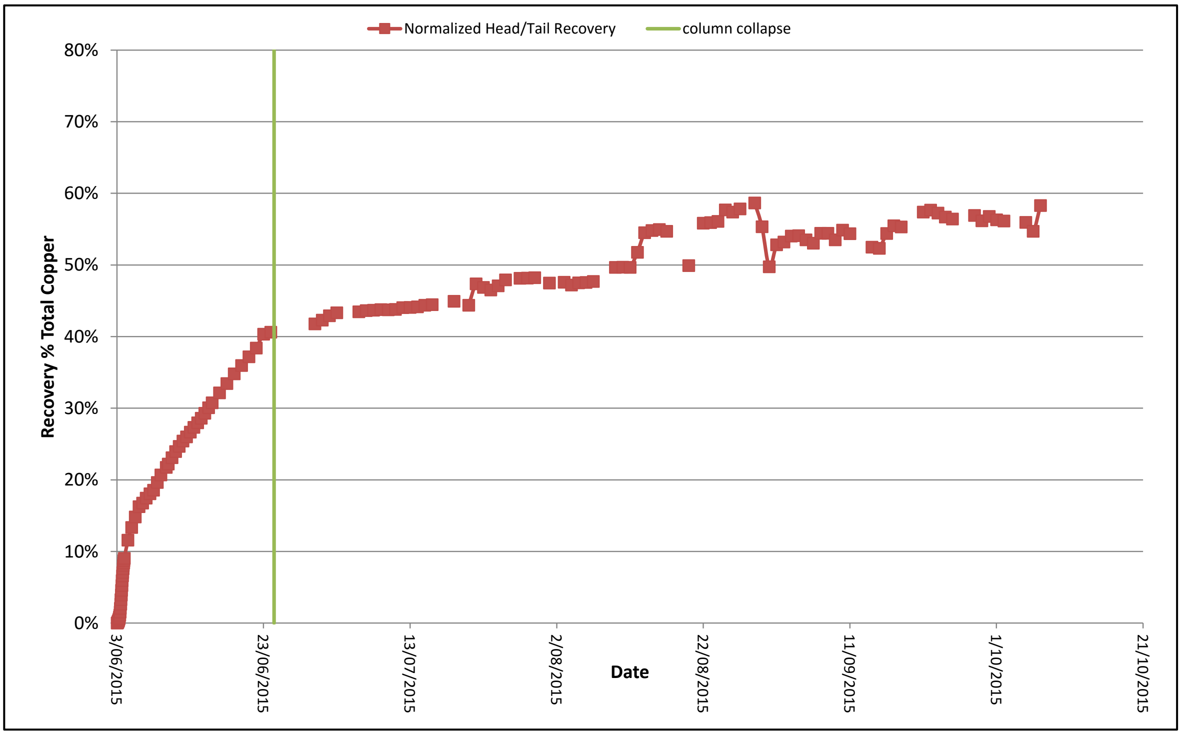


Figure 13-9 Column Test Normalized Head/Tail Recovery - 8mm



The column collapse has been indicated with the change in leach conditions (recirculated and lower acid concentration).

- Before the column collapse (day 14 for the 13 mm column and day 22 for the -6.5 mm column) approximately 40% of the copper leached
- At column closure (120 days) the -13 mm column had achieved 73% total copper recovery while the -6.5 mm column achieved only 58% recovery
- Both columns were showing continued recovery profiles with increased leaching time
- The kick up in recovery at the termination of the leach is due to the inclusion in the accounting of the copper contained in the feed and discharge solution containers

The short initial period of higher acid concentration shows a steep leach recovery rate. The lower acid concentration after the columns' collapse has a slower leach rate. This is consistent with the diffusion control model.

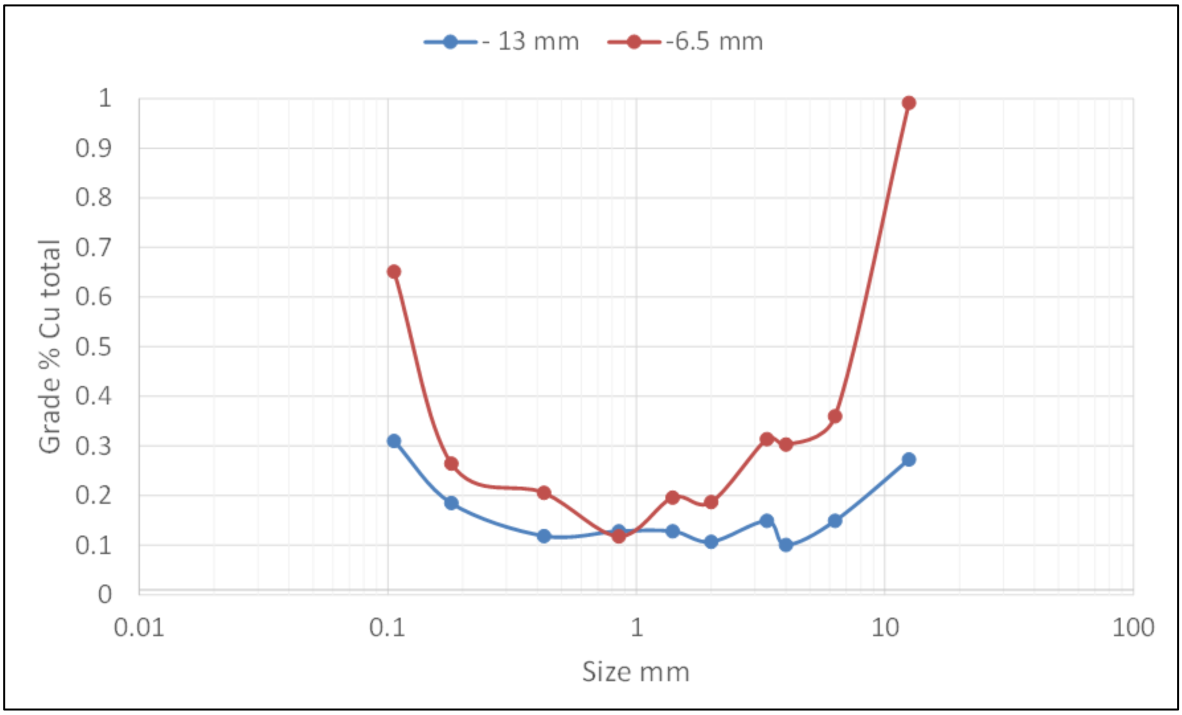
The disseminated nature of the mineralisation has been noted on the mineralogy. At this stage there is no firm evidence that there is a major crush size dependence on leaching rate and recovery. On the contrary the column tests show the reverse where the coarser crush is leaching faster and to higher recoveries than the finer crush. This counter-intuitive result has been shown to be the case in a number of commercial operations. It relates back to the combination of the geomechanical nature of the rock mass and the significant effect of agglomeration.

- The feed to these columns were NOT agglomerated and slow leaching is an expected outcome.
- The extra fines created by the finer crushing also cause higher bulk density along with longer diffusion path lengths; that in turn result in slower leaching.
- Further the long blending time have created more fines again – especially for the doubly blended – 6.5 mm column.

Any further column testing will be conducted on agglomerated material that has been appropriately prepared.

The leached material grade from the column tests show the effect of the above situation on the size by size leaching response.

Figure 13-10 Column test size by size leached material assays



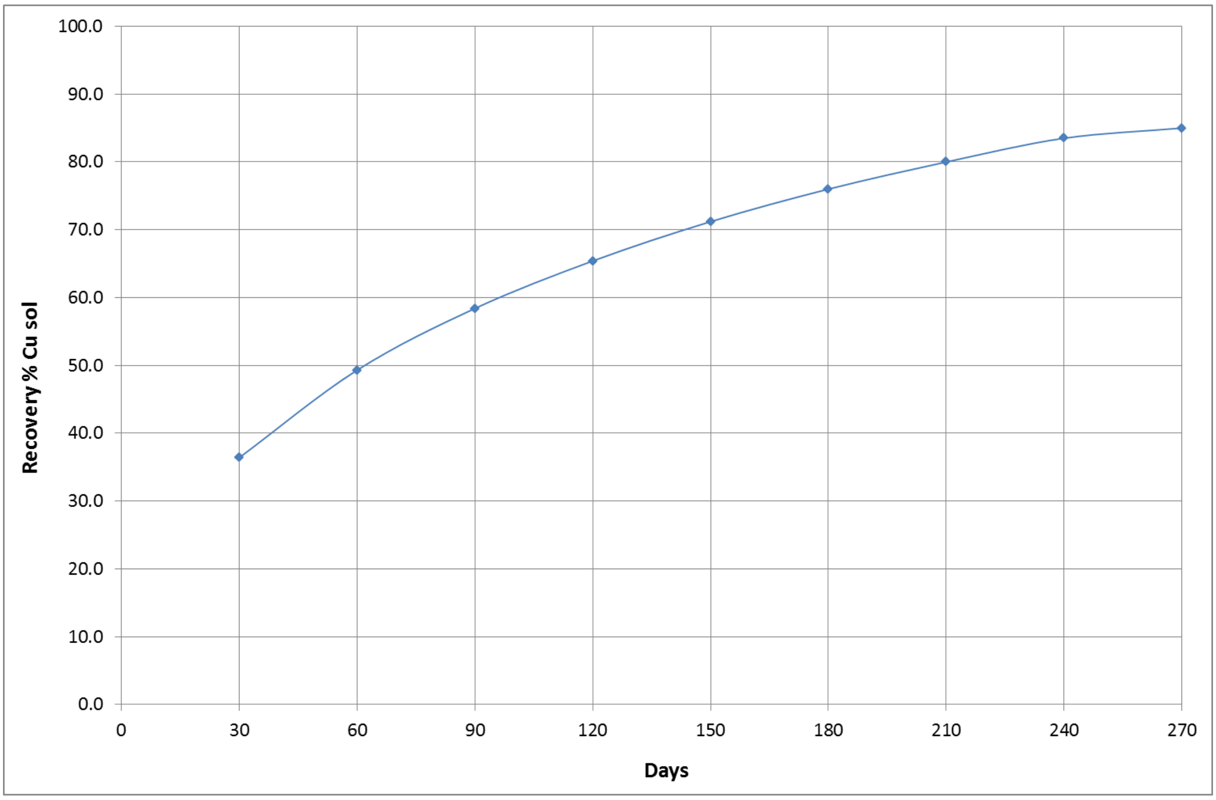
The -13 mm material shows a relatively consistent low grade leached material assay across all sizes. On the other hand the -6.5 mm shows elevated assay for both the coarse and the finer sizes. The results from the -6.5mm fine fraction are consistent with the higher moisture content of the leached material (15.2% compared to 12.1% in the -13 mm). The diffusion happens in the retained moisture; with higher moisture content indicating longer diffusion path lengths; giving the high grade in the fines in particular. The same argument explains the slightly rising leached material grade in the -13 mm with decreasing size.

The high grade in the -6.5 mm coarse size is also a response from the longer diffusion path lengths in the higher retained moisture. It is not uncommon to see these different responses with the finer crush size leaching to higher grades in the coarse sizes.

13.9.4 Leaching Response Curve

A maximum of 85% recovery of soluble copper has been set to provide some further margin for less than ideal operational control. The level of recovery is consistent with operations heap leaching similar secondary sulphide minerals. Leaching rates for heap leaching have not yet been determined. However a base line leaching response (recovery vs time) has been developed based on the proposed leach chemistry and operational experience from similar types of copper mineralization. This is effectively an 85% recovery of soluble copper in 270 days for a 0.43% soluble copper head grade as shown in Figure 13-11.

Figure 13-11 Assumed Leaching Response Curve



13.10 ACID CONSUMPTION

The acid consumption is likely to be low or negative. For this study an acid production rate of 30 kg/t has been assumed to size a neutralization facility. A small amount of acid will be used to make up for electrolyte bleed from the electrowinning plant (refer to Section 17.6).

13.11 LEACHATE CHEMISTRY

The leachate chemistry shows no issues for the production of high grade copper metal from the ions in solution using the solvent extraction – electrowinning process. Heavy metal contamination of the solutions will require precipitation of any discharge of leach liquors from the site.

The probable leaching mechanism of pyrite galvanic oxidation of the copper minerals has a low Oxidation Reduction Potential and little potential to precipitate ferric-arsenate within the heap. A neutralization and heavy metal precipitation plant is required to both manage the acid production and to precipitate some heavy metals from the recirculating process solution.

13.12 ECONOMIC LIMIT TO LEACHING

With a potential for acid production, there will be no economic limit to leaching and leach times can potentially be extended to achieve higher than the 85% target soluble copper recovery.

13.13 LEACH PLANNING

A preliminary leach plan using heap leaching has been selected based on the:

- Poor flotation response (due primarily to the very fine copper minerals disseminated within the pyrite matrix).
- Reasonably high total ‘leachable’ copper
- Geological assessment of rock competency
- Preliminary results from test work (i.e. columns, bottle rolls and shake flask tests).

14 MINERAL RESOURCE ESTIMATES

The BKM 2015 mineral resource estimate was undertaken utilizing MineSight™ software for domaining and Vulcan™ software for block modeling and grade interpolation. This section lists the processes and parameters used in generating the estimate.

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

14.1 RESOURCE DOMAINING

The methods involved in identifying and generating the copper grade interpolation domains is outlined in Section 12.2.2. These domains and details are listed in Table 14-1. The domain triangulations are grouped according to their composite search ellipsoid parameters (Figure 14-1).

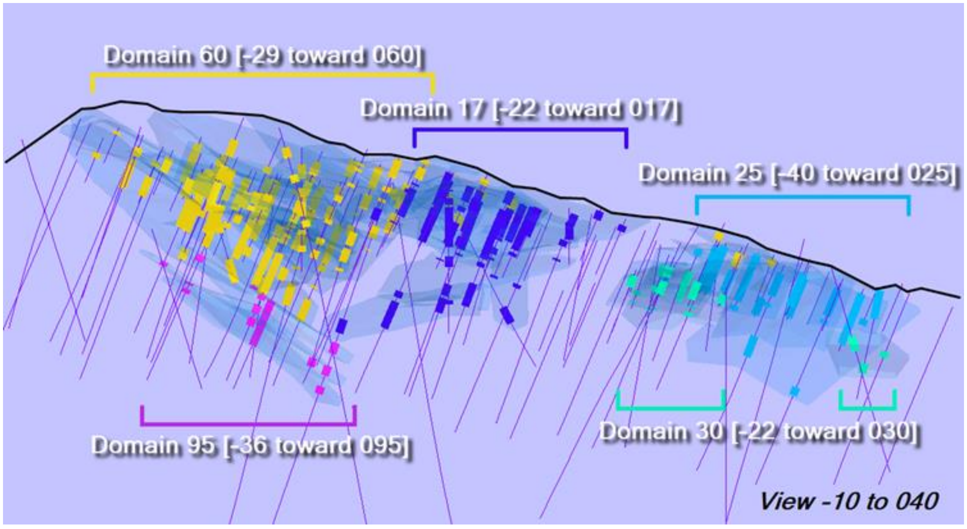
The domains have been utilized as hard boundaries for copper grade interpolation in the BKM 2015 Resource Estimate.

Table 14-1 Resource domain TIN files, block model coding details and composite search parameters for copper grade interpolation

Triangulation	BM Variable	Value	Priority	Z Axis Inversion	Composite Search Ellipsoid		
					Bearing (Z)	Plunge (Y)	Dip (X)
29_060_Solid_min_1.00t	estdom	60	1	None	60	-30	-15
29_060_Solid_min_2.00t	estdom						
29_060_Solid_min_3.00t	estdom						
29_060_Solid_min_4.00t	estdom						
29_060_Solid_min_5.00t	estdom						
29_060_Solid_min_6.00t	estdom						
29_060_Solid_min_7.00t	estdom						
29_060_Solid_min_22.00t	estdom						
29_060_Solid_min_23.00t	estdom						
29_060_Solid_min_24.00t	estdom						
36_095_Solid_min_8.00t	estdom	95	1	None	95	-36	0
36_095_Solid_min_9.00t	estdom						
36_095_Solid_min_10.00t	estdom						
22_030_Solid_min_19.00t	estdom	30	1	None	30	-22	0

Triangulation	BM Variable	Value	Priority	Z Axis Inversion	Composite Search Ellipsoid		
					Bearing (Z)	Plunge (Y)	Dip (X)
22_030_Solid_min_20.00t 22_030_Solid_min_21.00t 22_030_Solid_min_25.00t	estdom estdom estdom						
40_025_Solid_min_16.00t 40_025_Solid_min_17.00t 40_025_Solid_min_18.00t	estdom estdom estdom	25	1	None	25	-40	0
22_017_Solid_min_11.00t 22_017_Solid_min_12.00t 22_017_Solid_min_13.00t 22_017_Solid_min_14.00t 22_017_Solid_min_15.00t	estdom estdom estdom estdom estdom	17	1	None	30	-22	-10
Trim_BM.00t	estdom	3	3	Complete	*Utilised to remove distal blocks from BM*		
DTM-BK-Lidar_C.00t	estdom	2	2	Partial	*Utilised to clip domains at topo*		

Figure 14-1 Mineralization Domains (View in dip direction)



14.2 COPPER ASSAY COMPOSITING

Compositing was undertaken utilizing the Vulcan™ run-length routine. Composites were checked visually on screen and against original sample intervals/grades to ensure that domain contacts and sample interval breaks were honored.

The nominal primary sample interval of 3.0m dictates that the compositing interval cannot be less than this length. The BKM 2015 Resource Estimate is underpinned by 1,672 mathematical composites within mineralized domains and designed to have a nominal length of 3 meters. A total of 152 of these composites are less than three meters in length, with 34 being less than one meter in length. The 34 short composites (<1m) were checked against the original assay dataset which confirmed that compositing had been undertaken as intended. The short composite intervals are the result of the irregular original sampling intervals where the compositing routine generates a remainder-interval to accommodate the additional sample lengths between the last 3m composite and the domain boundary. As there is no discernible copper grade differential across the mineralization (Table 12.3) the short edge-composites were not excluded from the composite dataset used for grade interpolation.

A total of 8006 nominal 3m composites are located outside of the mineralized domains.

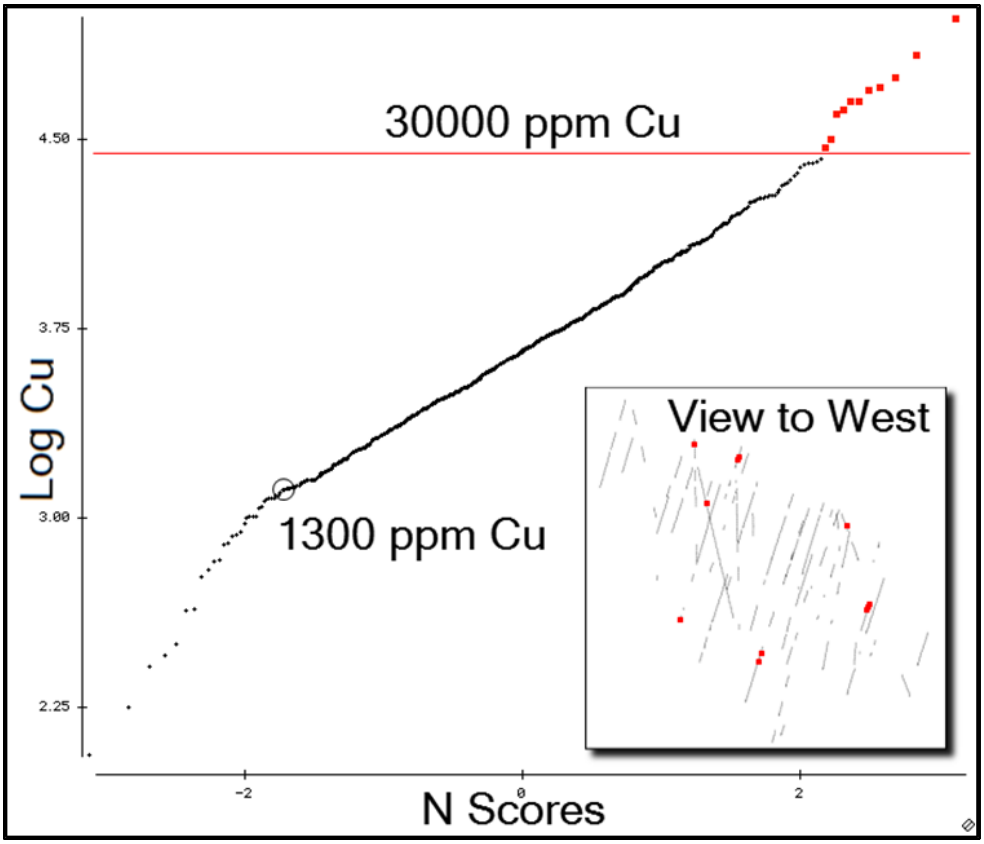
The copper composite data distribution is shown in Figure 14-4. The population has a mean of 6844ppmCu.

14.3 HIGH GRADE COPPER TREATMENT

A review of the copper composite data was undertaken to identify any outlier assays that may require consideration during grade interpolation. The 3m copper composites within mineralized domains were log10 transformed and plotted as a log-probability graph (Figure 14-2). A clear continuum in the graph between 1300ppm and 30000ppm copper supports the observations from core and made during resource domaining, that being that the copper mineralization appears to be of the same event and that more intense veining leads to higher grades. The thirty composites with grades greater than 30000ppm that plot as outliers in the log-probability also plot spatially as individual and dual/triple samples. These outliers were selected for high-grade treatment during grade interpolation.

The high grade copper composites have been used uncut in grade interpolation however their area of influence has been restricted to a 50m x 50m x 25m volume surrounding their location (Figure 14-2). This action will preserve high grades within the estimate and will reflect the geological reason for their presence (locally intense veining).

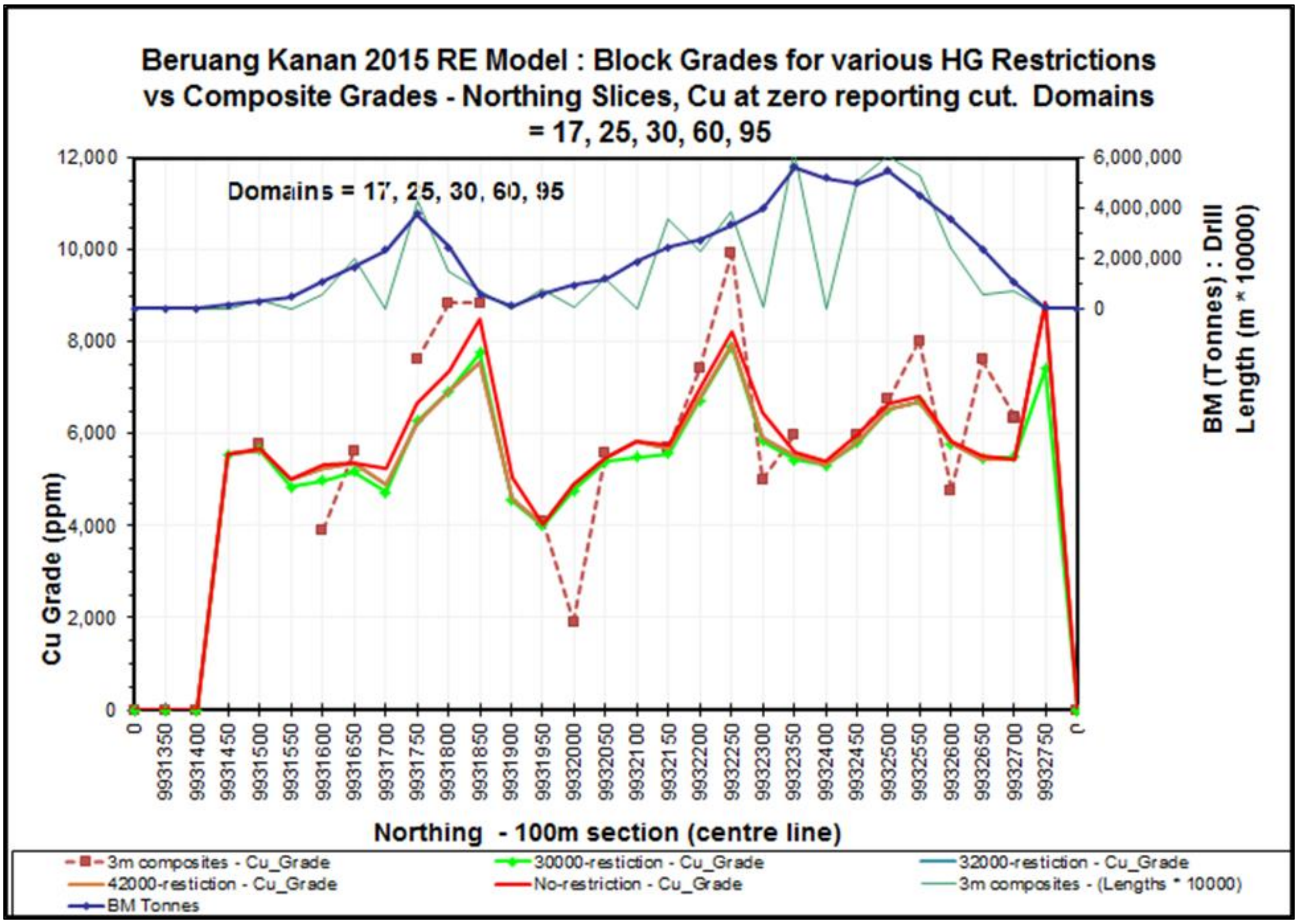
Figure 14-2 Log-probability plot (copper composites)



Note: High grade treatment threshold set at 30000ppm Cu.

As validation that 30000ppm Cu is a reasonable cut, three check interpolation runs were undertaken with restrictions set at 32000ppmCu, 42000ppmCu and at no-restriction. Swath plots presented at Figure 14-3 show that 30000ppmCu is a reasonable level to apply the restriction as only the unrestricted model shows any significant deviation of grade from the other trials and only on sections where drilling meters are low.

Figure 14-3 Swath plot of high grade Cu restriction threshold trials



14.4 BLOCK MODEL DETAILS

Details of the BKM 2015 Resource Estimate Vulcan™ block model are listed below. Domain triangulations and the block coding details are listed at Table 14-1.

```
Block Model Details

Model name           : BK_postestimate30000_Oct2015
Format               : extended
Structure            : non-regular
Number of blocks     : 172720
Number of variables  : 12
Number of schemas    : 2
Origin               : 0.000000 0.000000 0.000000
Bearing/Dip/Plunge   : 90.000000 0.000000 0.000000
Created on           : Thu Oct 15 09:23:46 2015
Last modified on     : Thu Oct 15 09:23:46 2015
Model is indexed.

Variables       Default    Type       Description
-------------------------------------------------------
estdom          5          short      Estimation domains
cuid2           -99.0      short      Cu ppm ID2 estimate
flagid2         -99.0      short      flag if estimated in ID2 1pass 2pass 3outsidedoms
cuok            -99.0      short      Cu ppm OK estimate
flagok          -99.0      short      Flag if estimated in OK 1pass 2pass 3outsidedoms
kvar            -99.0      float      OK variance
numbsamp        -99.0      short      Number of composites OK
numbdhs         -99.0      short      Number of holes OK
avdist          -99.0      short      Av distance of selected samples OK
class           -99.0      short      Classification 2Indic 3infer 4potential
dbd             -99.0      float      DBD
base270dbd      -99        integer    base of 2.70 dbd 17m below topo surface

Schema <parent>
Offset minimum : 768300.000000 9931200.000000 100.000000
       maximum : 769250.000000 9932800.000000 600.000000
Blocks minimum : 25.000000 25.000000 10.000000
       maximum : 25.000000 25.000000 10.000000
  No of blocks : 38 64 50

Schema <subblock>
Offset minimum : 768300.000000 9931200.000000 100.000000
       maximum : 769250.000000 9932800.000000 600.000000
Blocks minimum : 5.000000 5.000000 2.000000
       maximum : 25.000000 25.000000 10.000000
  No of blocks : 190 320 250
```

14.5 COPPER GRADE INTERPOLATION

Ordinary Kriging was employed as the copper interpolation method. For the kriging neighborhood investigation and experimental variography report refer to Appendix 16 of the resource report (Hackman, 2015). Key features of the mineralization and domaining identified in the investigation are:

- The general consistency of copper grades within and between the domains
- All domains show comparable copper grade populations distributions, with the exception of:
    - domain 95 which has a clear positive base shift in grade tenor and
    - domain 25 which shows deviations at grades <2000ppm (greater percentage of population than other domains) and >20000ppm (also greater percentage of population than within other domains)

- Experimental semi-variograms were assessed for all domains combined and for domains 95 and 25 individually. One spherical structure fits all semi-variograms and primary directions reflect the overall geometries of the modeled domains.

Copper grade interpolation was undertaken in two passes for each domain, reflecting the block proximity to drilling data and block relationship with mineralization domains. Details of the runs are listed in Table 14-2. In summary:

- **Pass 1:** Within modeled mineralized domains and a search radius of 100mX100mX25m (runs ok017a, ok025a, ok030a, ok060a and ok095a). Composites within domains can inform blocks within domains, composites outside of domains are not used. Five search ellipsoids orientations are employed, each reflecting the overall geometry of the domains they best fit (as shown in Table 14-1 and Figure 14-1). A minimum of 5 and maximum of 24 composites are used to generate block grades. Octant search parameters are employed with a minimum of 6 octants to be informed before a grade is interpolated. Copper grades greater than 30000ppm are restricted to estimate blocks within a radius of 50mX50mX25m.
- **Pass 2:** Within modeled mineralized domains and a search radius of 200mX200mX50m (runs ok017b, ok025b, ok030b and ok095b) and 300mX300mX50m (run ok060a). All other parameters are the same as for Pass 1 except a maximum of 16 composites and minimum of 4 octants criteria is applied.
- **Pass 3:** Within modeled mineralized domains a relaxed run (okfill) was employed to fill blocks at the extremity of the model.
- **Pass 4:** Outside of modeled mineralized domains, sample selection of only those composites with greater than 2000ppm copper grades, outside of the modeled mineralized domains and within a search radius of 25mX25mX10m (run ok5a). All other parameters are the same as for the Pass 1 except a maximum of 10 composites applied and the octant search criteria removed.

Table 14-3 shows that grade interpolation process ran as planned with 98% of the blocks within the mineralized domains being estimated in passes 1 and 2 and 306 blocks estimated outside of the mineralized domains (pass 4).

Table 14-2 Copper Grade Interpolation (Estimation Run Details)

Criteria	Default for estimation runs (those not listed to right)	Specific to individual estimation runs	
		Estimation Run	Details
Estimation_File	bkcuid20k.bef		
Estimation_Type	Ordinary Block Krigin		
Block Model	BK_postestimate30000_Oct2015.bmf		
Estimation Variable	cuok:Default value -99		
Composite_File	BKCU_3M.MAP		
Input Variable	CUPPM		
Composite Selection Criteria	ignore default domain code values (-99)	ok5a	CUPPM >=2000 and GECOD = "5.000"
Maximum Number of Composites	10		
Minimum Number of Composites	5		
Sample Upper Cuts	not cut		
High Grade Cu Restriction Threshold	30000		
Restriction Major Axis (m) Within	50		
Restriction Semi-Major Axis (m) Within	50		
Restriction Minor Axis (m) Within	25		
Bearing (Rotation around Z): Composite Selection, High Grade Restriction and Kriging Structure 1		ok017a; ok017b; ok030a; ok030b	30
		ok025a; ok025b	25
		ok060a; ok060b; okfill; ok5a	60
		ok095a; ok095b	95
Plunge (Rotation around Y): Composite Selection High Grade Restriction and Kriging Structure 1		ok017a; ok017b; ok030a; ok030b	-22
		ok025a; ok025b	-40
		ok060a; ok060b; okfill; ok5a	-30
		ok095a; ok095b	-36
Dip (Rotation around X): Composite Selection High Grade Restriction and Kriging Structure 1		ok017a; ok017b	-10
		ok025a; ok025b; ok030a; ok030b; ok095a; ok095b	0
		ok060a; ok060b; okfill; ok5a	-15
Composite Selection Ellipsoid Major Axis (m)		ok017a; ok025a; 0k030a; ok060a; ok095a	100
		ok017b; ok025b; 0k030b; ok095b; okfill	200
		ok060b	300
		ok5a	25
Composite Selection Ellipsoid Semi-Major Axis (m)		ok017a; ok025a; 0k030a; ok060a; ok095a	100
		ok017b; ok025b; 0k030b; ok095b; okfill	200
		ok060b	300
		ok5a	25
Composite Selection Ellipsoid Minor Axis (m)		ok017a; ok025a; 0k030a; ok060a; ok095a	25
		ok017b; ok025b; 0k030b; ok060b; ok095b	50
		okfill	200
		ok5a	10
Ordinary Kriging Model	Spherical - single structure		
Nugget	0.3		
Sill Differential	0.7		
Range - Major Axis	65		
Range - Semi-Major Axis	40		
Range - Minor Axis	30		
Composites required to make an estimate	Minimum 5 (except ok5a = 3)	ok017a; ok025a; 0k030a; ok060a; ok095a	Maximum 24
		ok017b; ok025b; 0k030b; ok060b; ok095b	Maximum 16
		okfill; ok5a	Maximum 10

Criteria	Default for estimation runs (those not listed to right)	Specific to individual estimation runs	
		Estimation Run	Details
Octant base composite search (matches search ellipsoid)		ok017a; ok025a; ok060a; ok095a	Minimum of 6 octants filled; min 1 max 4 comps per octant
		ok017b; ok025b; ok060b; ok095b	Minimum of 4 octants filled; min 1 max 4 comps per octant
Block Selection		ok017a; ok017b	estdom eq 17 and FLAGOK it 0
		ok025a; ok025b	estdom eq 25 and FLAGOK it 0
		ok030a; ok030b	estdom eq 30 and FLAGOK it 0
		ok060a; ok060b	estdom eq 60 and FLAGOK it 0
		ok095a; ok095b	estdom eq 95 and FLAGOK it 0
		okfill	estdom gt 6 and FLAGOK it 0
		ok5a	estdom eq 5 and FLAGOK it 0
Block Discretization	5X; 5Y; 2Z		
Estimation Centroid	parent block centroid		
Flag if Estimated		ok017a; ok025a; 0k030a; ok060a; ok095a	1
		ok017b; ok025b; 0k030b; ok060b; ok095b	2
		okfill	3
		ok5a	4

Table 14-3 Copper Grade Interpolation (Estimation Run Performances)

Criteria	Estimation Run	Result
Selected Blocks	ok017a	11667
	ok017b	6675
	ok025a	6113
	ok025b	4581
	ok030a	4854
	ok030b	221
	ok060a	26754
	ok060b	12618
	ok095a	2661
	ok095b	1810
	okfill	1037
	ok5a	94762
Estimated Blocks	ok017a	4992
	ok017b	6516
	ok025a	1532
	ok025b	4324
	ok030a	4633
	ok030b	221
	ok060a	14136
	ok060b	12164
	ok095a	851
	ok095b	1643
	okfill	1017
	ok5a	306
Percent Estimated	ok017a	0.428
	ok017b	0.976
	ok025a	0.251
	ok025b	0.944
	ok030a	0.954
	ok030b	1
	ok060a	0.528
	ok060b	0.964
	ok095a	0.32
	ok095b	0.908
	okfill	0.98
	ok5a	0.003

14.6 TONNAGE FACTORS

The tonnage factors were stamped onto the model according to the following:

- mineralized domains within 17m of surface (or 20 downhole from collar) = 2.70t/m3
- mineralized domains > 17m deep = 2.85 t/m3
- non-mineralized material within 17m of surface = 2.33 t/m3
- non-mineralized material deeper than 17m = 2.77 t/m3

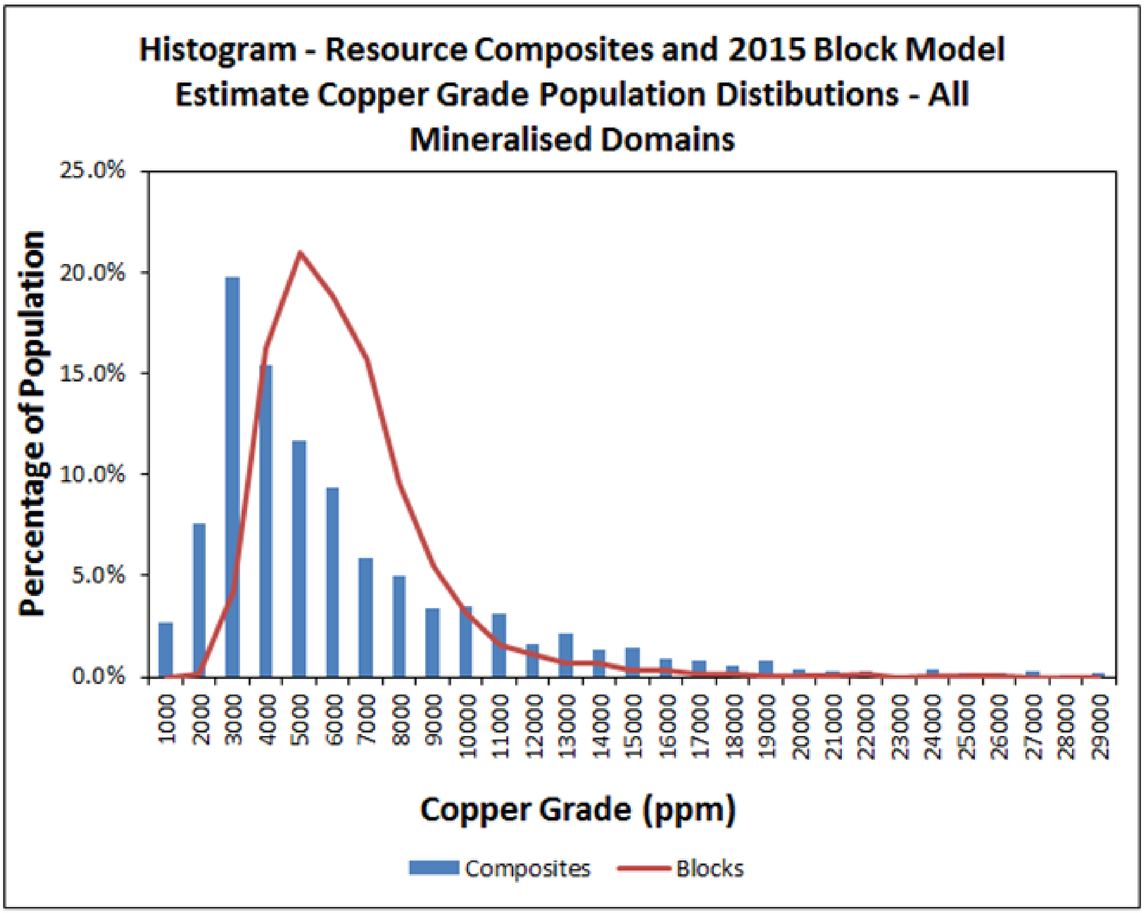
The surface “base_2-70_DBD_2015.00t” and mineralized domains (Table 14-1) were utilized to assign tonnage factors into the 2015 BKM block model.

14.7 MODEL VALIDATION

The resource block model coding was validated visually against both the mineralization domain models and the coded composites.

The copper grade interpolation was cross-checked against the composite data both statistically (Figure 14-4) and spatially on screen and by swath plots (Figure 14-5). An ID2 check estimate and a composite selection methodology check estimate (octant search parameters removed) were generated and correlate well with the grade distribution of the BKM 2015 resource block model. The BKM copper grade interpolation strategy has produced a resource model that adequately reflects the grade distribution identified in the broad spaced drilling of the project area.

Figure 14-4 Histograms showing comparison between 2015 Resource Model Copper grades and Composite Copper grades.



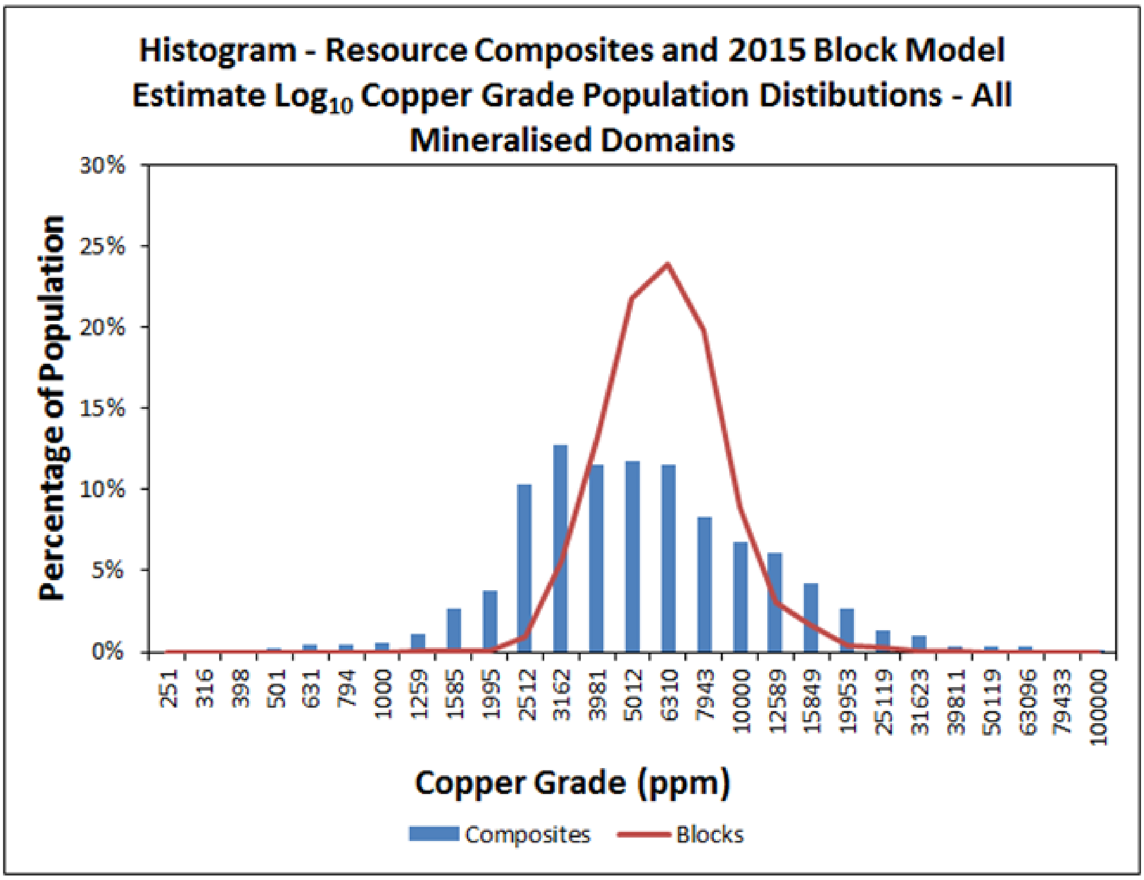
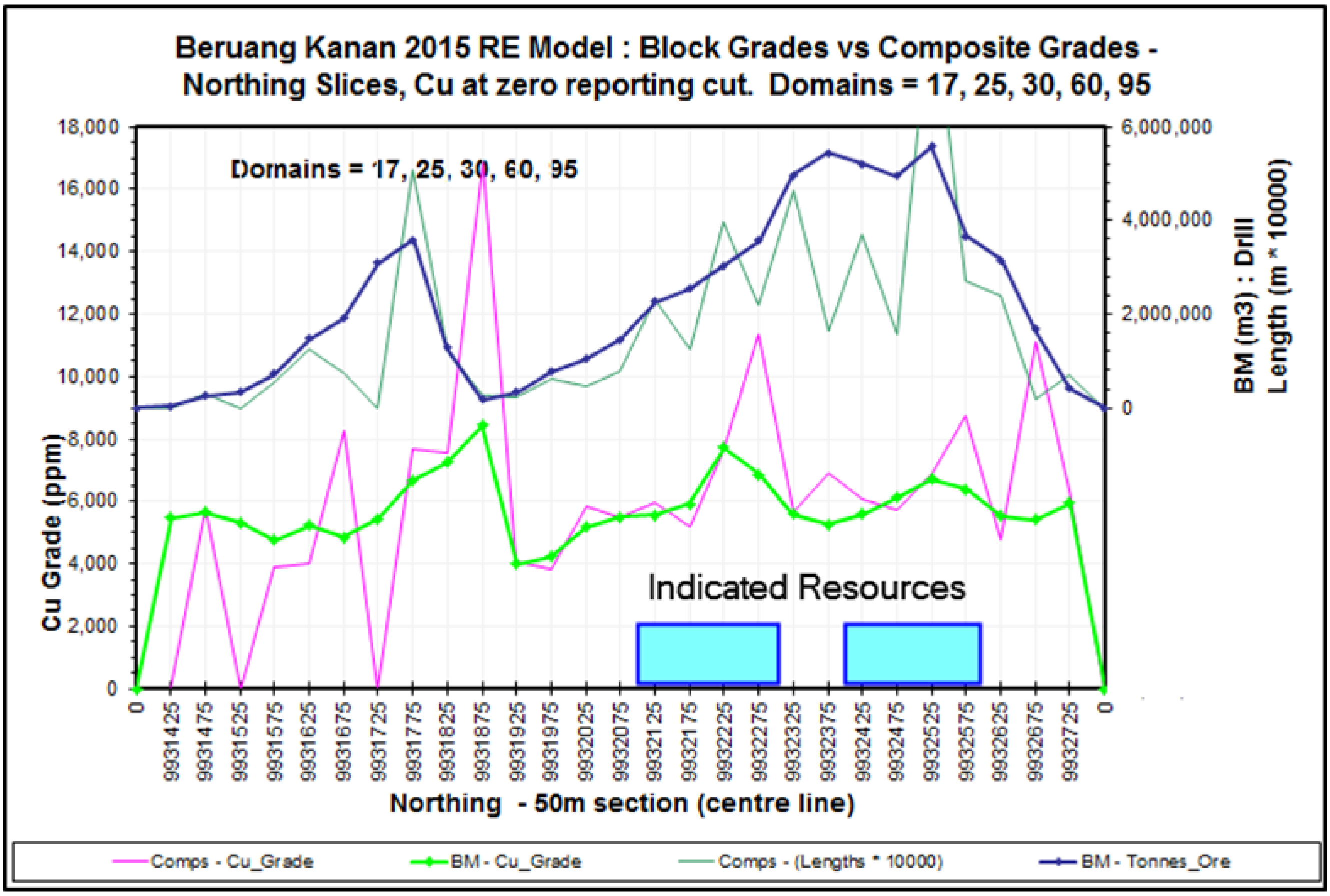


Figure 14-5 Block Model Validation (Northing Swath Plots)



14.8 CLASSIFICATION

The resources at Beruang Kanan as estimated in 2015, being the subject of this report, are classified as Indicated and Inferred Resources under guidelines set out in the Canadian National Instrument 43-101. The key considerations in assigning this classification are as follows and risk reduction associated with these criteria will assist with expanding the Indicated Resources and assigning higher classifications in future estimates:

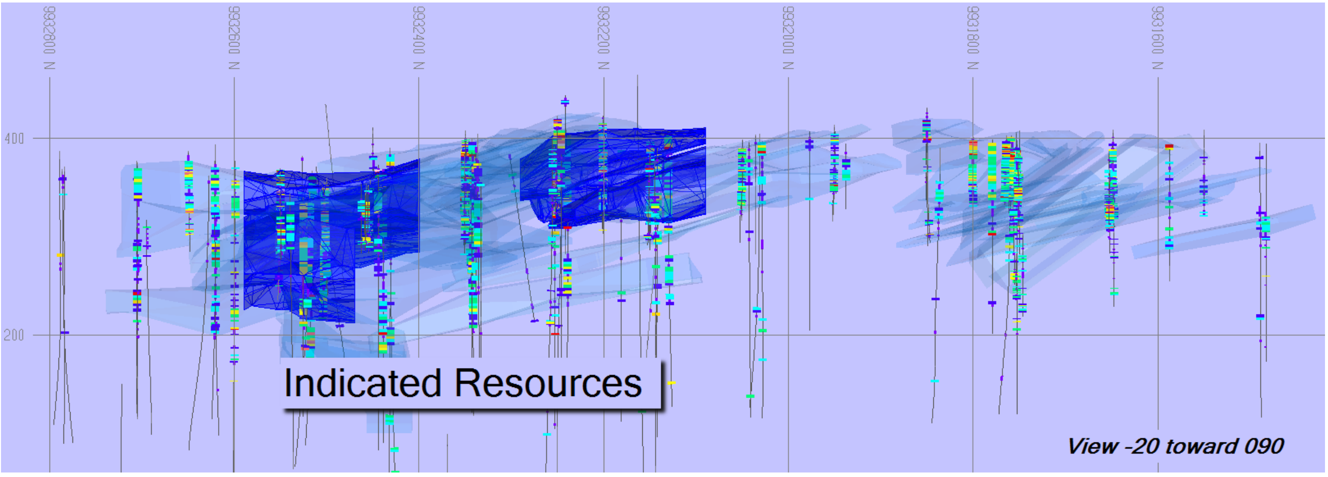
- Moderate risk associated with the current drill spacing and orientation in reliably testing the/all mineralization style(s)
- Moderate to high risk associated with drill core diameters and primary sampling intervals suitability in dealing with the fundamental sampling errors anticipated with vein style mineralization
- Moderate risk associated with the limited understanding in the geological and grade continuity of the mineralization to guide domaining and inform modeling and interpolation design
- Low risk associated with the unknown suitability of the sample comminution and sub-sampling strategy employed by all workers
- Low risk associated with inability to directly validate historic data

Two areas in the central and northern reaches of the BKM mineralization, where overall mineralization thickened and pre-2015 drilling, suggested reliable continuity in both tonnes and grade were targeted for infill drilling in 2015. The 2015 infill holes confirmed as anticipated that grade and thickness in these areas was relatively well understood to the level that Indicated Resource Classification could be considered. Two zones of mineralization were modeled centered on these areas (Figure 14-6). The areas are confined to the thicker 2000ppm Cu interpolation domains and from:

- 9932090N and 9932290N; 768780E and 768950E
- 9932400N and 9932590N; 768610E and 768880E

Table 14-4 shows that copper grades for 90% of the Indicated Resources were interpolated in the first pass of the estimation runs. This pass has most stringent criteria in selecting samples for estimating block grades (Table 14.2) as reflected by the statistics listed in Table 14-4. In contrast 46% of Inferred resources were interpolated in the first pass of estimation runs.

Figure 14-6 Location of Indicated Resources



Note: Remaining mineralization classified as Inferred Resources

Table 14-4 Statistics on data selection criteria for Indicated and Inferred resources

Interpolation Run Number		Indicated Resources	Inferred Resources	Total
Interpolation Cu Grade (%)	1	0.7	0.6	0.7
	2	0.6	0.5	0.6
	3		0.5	0.5
	Total	0.7	0.5	0.6
Tonnes Estimated (MT)	1	13.6	23.1	36.7
	2	1.5	23.9	25.4
	3	0	2.7	2.7
	Total	15.1	49.7	64.8
Average Number of DHs in Estimate	1	4.7	4.5	4.6
	2	4.2	4	4.1
	3		1.3	1.3
	Total	4.5	3.3	3.7
Average Number of Composites in Estimate	1	23.6	22.1	22.9
	2	16	15.9	16
	3		5.2	5.2
	Total	19.8	14.4	16.6
Average distance to Composites in Estimate	1	47.4	60.8	54.1
	2	56	79.7	67.9
	3		42.3	42.3
	Total	51.7	60.9	57.2

The Beruang Kanan 2015 Resource Estimate and Block Model are known at an Indicated and Inferred level of confidence (NI 43-101) at a global or overall scale. The Inferred resources are not suitable for any detailed studies or investigations requiring a high degree of local resource confidence (such as engineering or metallurgical studies) other than for the preparation of a Preliminary Economic Assessment Study and for planning purposes for programs designed in improving local and global resource confidence or resource expansion.

The following details the technical areas considered in classifying the BKM 2015 copper resource estimate.

14.8.1 Geological Understanding (Geological and Copper Grade Continuity):

KSK and joint venture workers have not (yet) undertaken sufficient work to fully understand the style(s) of mineralization at BKM. The geological and grade continuity is based on recent interpretations undertaken as part of this and the previous (2014) resource estimation processes. The core logging and other observations fit with the interpretation that mineralization is vein style and hosted mostly within a structurally complex zone (shear or thrust coupling/ramping/divergence). Of concern regarding confidence in the resource estimate is that:

- Surface mapping does not recognize the interpreted major thrust directions of ~090 and ~110 degrees used in directing the overall geometry of mineralization.
- The vein mineralization continuity is not understood and may be at orientations other than that described by the overall geometry of the mineralization.

14.8.2 Drilling Density and Configuration

The drilling is mostly oriented at -60 degrees towards 270 degrees and at nominal 50m centers along 100m spaced grid lines over the main zone of mineralization (closing to 50m by 50m where Indicted Resources are classified). Of concern regarding confidence in the resource estimate is that:

- There has been no investigation into attitude of the mineralized veins/vein-sets and therefor no evaluation as to the suitability of drill hole orientation with respect to the mineralization.
- The drill density is such that, given the vein-style mineralization hosting copper, the estimate can only be considered for classification at a global scale.

14.8.3 Sample Location

The collar locations of holes are considered well known. Down hole survey information is lacking for 30 of the 54 holes drilled into the BKM Main Zone mineralization. Of concern regarding confidence in the resource estimate is that:

- Although the locations of samples from half of the pre 2015 drill holes delineating the mineralization cannot be validated, the reasonable predictability of hole trace locations for those with survey information lends support to the reliability of hole traces defined by a single collar survey azimuth and declination. The 2015 drilling results support the earlier hole results indicating that collar location issues are likely to pose only a minor risk to the estimate. The sample locations are considered well enough established to consider the BKM resource estimate for classification at a global scale.

14.8.4 Primary Sample Size

The mineralization has been tested with core drilled at sizes of PQ (3%), HQ (73%), NQ (18%), BQ (2%), with 4% of core sizes not recoded. Workers for the pre 2015 drilling have employed a nominal 3m sample interval (20% of samples within mineralization) and a significant number of 2m intervals were sampled by workers in the 2015 drilling campaign (17% of samples within mineralization). Of concern regarding confidence in the resource estimate is that:

- There is an observed copper grade tenor shift of 26% between the NQ-BQ drill core samples (lower) and the PQ-HQ drill core samples. This may be due to natural grade variability throughout the mineralization or to the fundamental sampling error in dealing inherent heterogeneity of the mineralization. In general the fundamental sampling issue diminishes with increase in the primary sample size; therefor it is likely that the grade of the copper for the global resource estimate is negatively biased, if this issue exists.

- The large primary sample size and the sample comminution and reduction process employed are not theoretically ideal (according to Gy's generalized sampling nomogram) however the relatively narrow band of copper assays within the mineralization suggests that any issues may not be of significance when the risk is assessed at the global scale.

14.8.5 Sample Preparation and Assay

Large mineralized samples (2m and 3m lengths) were crushed to -4mm (3m samples) and -2mm (2m samples) before being sub-sampled to 1kg for pulverizing. All digests were conducted by 3 acid digest. Of concern regarding confidence in the resource estimate is that:

- The sample comminution and reduction process employed are not theoretically ideal (according to Gy's generalized sampling nomogram) however the relatively narrow band of copper assays within the mineralization suggests that any issues may not be of significance when the risk is assessed at the global scale. The QC evaluation of the coarse crush and split duplicates undertaken during the 2015 drilling campaign uncovered possible native copper which appears to be poorly handled in the comminution and reduction process employed.
- Three acid digests are akin to total digests. This is only an issue if copper silicates are present within the mineralization at BKM. There is one recording of the copper silicate, chrysocolla, in an early thin section report. More work is required on the copper mineralogy, however the assaying is acceptable for classifying the BKM resource at a global scale. In addition three acid digests will give total copper content of samples. Sequential digests are required of mineralized samples to obtain recoverable copper assays for use in future resource estimates.

14.8.6 Assay Data Quality

The 2015 assay QC program and QC work undertaken by ENJ-KSK contains sufficient quality control samples to assess reliability of the copper assays. Earlier work by OX-KSK contained limited quality control samples and there were no quality control samples submitted with assays for the early work undertaken by KSK (pre 2002). Of concern regarding confidence in the resource estimate is that:

- Quality control samples submitted with the 2015 KSK program shows that the copper assaying for this period is of acceptable quality for classifying global resources.
- Quality control samples submitted with the ENJ-KSK program shows that the copper assaying for this period is of acceptable quality for classifying global resources.
- Quality control samples submitted with the OX-KSK program show that there may be issues with copper assays from early batches of their work, however only one hole is affected by this issue and therefor assays from this period are of acceptable quality for classifying global resources.

- The copper assays data population from the early OX-KSK and early KSK work is comparable with the assay population from the 2015 KSK and ENJ-KSK work, leading H&A to conclude that, even though there is limited/no quality control on the early work, the copper assays from these periods are suitable for inclusion in the BKM 2015 Resource Estimate and acceptable for classifying global resources.

14.8.7 Tonnage Factors

Dry Bulk Density measurements were taken from core during KSK 2015 program. Of concern regarding confidence in the resource estimate is that:

- DBD measurements are reliable and suitable for estimating tonnages at BKM. The use of a rudimentary surface (topography translated -17m) in defining the base of surface effects and applying lower tonnage factors to resources above this surface will add risk to the resource estimate however it is considered minimal with respect to classifying global resources at BKM.

14.8.8 Resource Copper Grade Interpolation

The copper grade has been estimated by ordinary kriging interpolation methods. Of concern regarding confidence in the resource estimate is that:

- The resource estimate reconciles well with the source (composite) dataset and compares well with alternative estimates utilizing ID2 methodologies and various check high grade restriction and composite selection strategies. The copper grade interpolation strategies are robust for the BKM estimate and acceptable for classifying the resource at global scale.

14.9 COPPER RESOURCE TABLE AND GT CURVES

The Indicated and Inferred Copper Resource at BKM is tabulated at Table 14-6 and presented in the Grade-Tonnage curve in Figure 14-7.

The base case resource is reported at a 0.2% copper cut. A 0.2% copper cut (following rounding to reflect confidence) approximates the calculated break-even (BE) cut-off grade value of 0.21% copper derived from an initial economic evaluation of BKM. This economic evaluation was designed to provide a preliminary assessment of the viability of the project prior to the PEA proper. Details of the calculated of BE cut-off are detailed Table 14.5.

Table 14-5 Resource cut-off grade calculation

Parameter	Unit	Value
Dilution	%	5%
Processing	$ / tonne leach feed	$9.50
Mining Premium	$ / tonne leach feed	$0.22
Total Leach Feed Cost	$ / tonne leach feed	$9.72
Recovery	$	80%
Base Copper Price	$ / lb.	3.00
	$ / tonne	6613.86
Royalty	%	4%
NSR	%	1%
Transport	$ / t / km	$0.15
Km	km	400
	$ / t copper	$60.00
Net Price	$ / t copper	$6,223.17
Breakeven Cut-off Grade	calculation	$\frac{\text{Total Leach Feed Cost x (1 + Dilution)}}{\text{Recovery x Net Price}}$
	% Total Cu	0.21%

The parameters used to derive this cut-off were developed at the beginning of the PEA process and were based on initial estimates and assumptions. The parameters were further refined over the course of the study and therefore the associated cut-off grade was also updated. The mine plan detailed in Section 16.5 is based on a Leachable Copper Grade (Cu Leach). The break-even cut-off grade for the project, based on the final economic parameters detailed in Section 21, is approximately 0.09% Cu Leach, which equates to approximately 0.16% Total Copper (Cu Total - refer to Table 25-2). The reduction in cut-off grade was primarily driven by a price increase from $3.00/lb. to $3.25/lb. and a reduction in total ore cost from $9.72/tonne to $6.10/tonne.

However the LOM schedule has been developed utilizing a variable elevated cut-off grade strategy that is optimized over time to maximize the project value. The optimized cut-off grade ranges between the BE cut-off of 0.09% Cu Leach and an elevated cut-off up to 0.11% Cu Leach over the life of the project. This equates to a Cu Total cut-off grade range of approximately 0.16% to 0.20%. Therefore the use of a resource cut-off of 0.2% Cu Leach can be considered appropriate for the total resource.

The PEA pit design includes >95% of the Indicated Resource material.

In addition, H&A has reviewed parameters utilized for determining reporting cuts from similar deposits and uncovered that, utilizing a similar approach and parameters:

- GeoVector Management Inc. determined a 0.2% copper reporting cut for the Las Posadas Copper Deposit, Chile, as part of PEA prepared for Global Hunter Corp. (October 2012).
- Tetra Tech Inc. determined a 0.25% copper reporting cut for the Zonia Copper-Oxide Deposit, Arizona, USA, as part of a resource report prepared for Cardero Resource Corp. (December 2015).

H&A is of the opinion that 0.2% Cu is an appropriate base case reporting cut in stating the BKM mineral resources and that any upward movement in reporting cut to 0.3%Cu (based on any sensitivity studies) would not alter the reported Indicated Resources and reduce the reported Inferred Resources by 2.8Mt (refer to Figure 14-7).

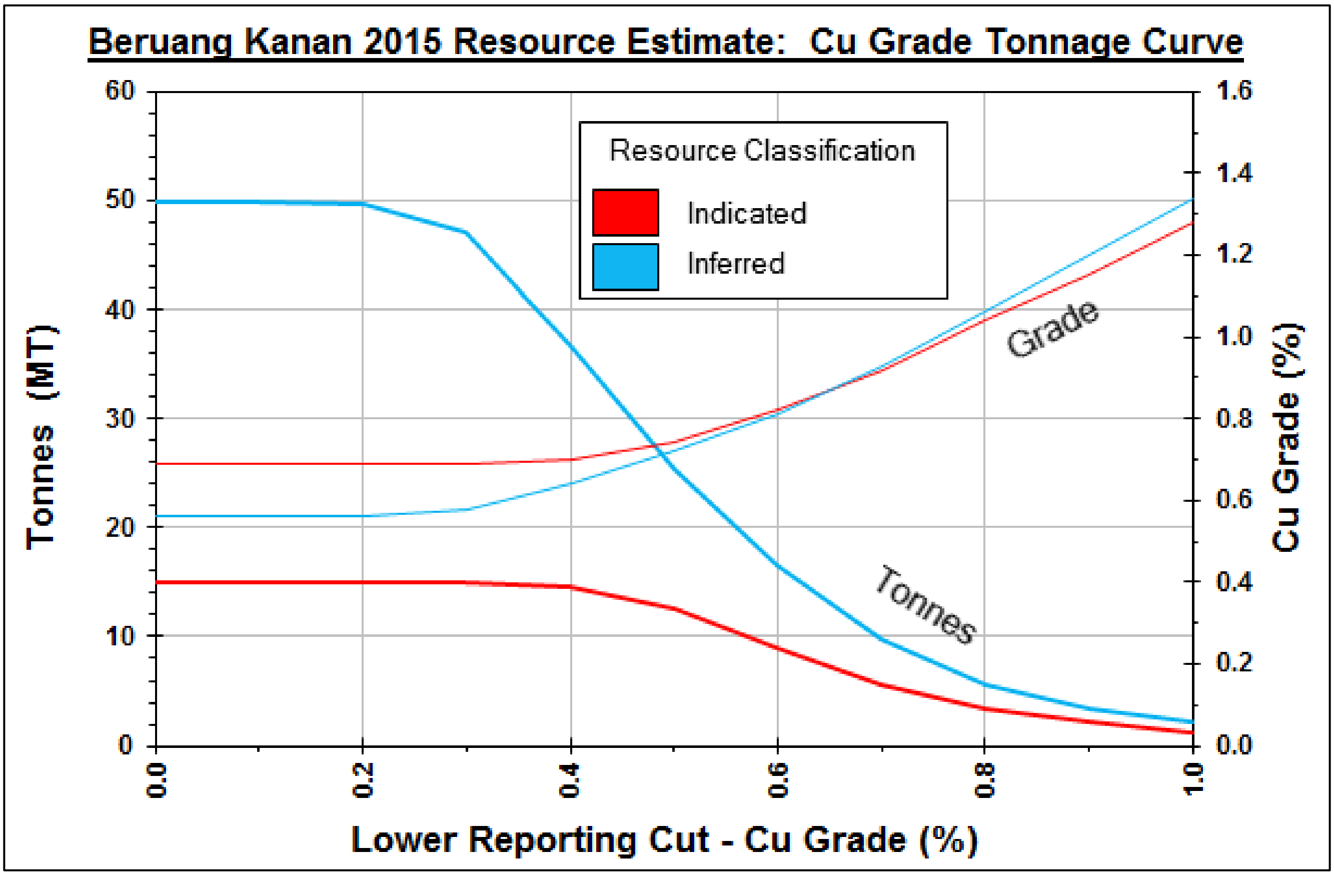
Table 14-6 Tabulated Copper Resources - Beruang Kanan [Indicated and Inferred Classified Resources reported separately].

Indicated Mineral Resources				
Reporting cut (Cu %)	Tonnes (‘000)	Cu Grade (Cu %)	Contained Cu (‘000 tonnes)	Contained Cu (‘000,000 lbs)
0.2	15,000	0.7	105	231
0.5	12,600	0.7	88	194
0.7	5,600	0.9	50	110

Inferred Mineral Resources				
Reporting cut (Cu %)	Tonnes (‘000)	Cu Grade (Cu %)	Contained Cu (‘000 tonnes)	Contained Cu (‘000,000 lbs)
0.2	49,700	0.6	298	657
0.5	25,300	0.7	177	390
0.7	9,800	0.9	88	194

Notes: *Mineral Resources for the Beruang Kanan mineralization have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. In the opinion of Duncan Hackman, the block model Resource estimate and Resource classification reported herein are a reasonable representation of the copper Mineral Resources found in the defined area of the Beruang Kanan Main mineralization. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserve. Computational discrepancies in the table and the body of the Release are the result of rounding.*

Figure 14-7 Copper Grade Tonnage Curves (Beruang Kanan Resources)



H&A details the current tenure and permitting status of the KSK CoW (incorporating the Beruang Kanan Main Zone mineralization) in Section 4.2. KSK has signed a MOU with the Government of Indonesia covering items to amend in the KSK CoW, a requirement stipulated by the Government to align existing CoWs with the current Indonesian Mining Law. All details of the amendments have not been finalized; however the continuation of the CoW is clearly stated in the MOU.

H&A is not aware of any current legal, political, environmental, permitting, taxation, socio-economic, marketing or other risks that could materially affect the potential development of the mineral resources at BKM.

14.10 Comparison with 2014 Resource Estimate

The 2014 resource estimate was reported as 47MT @ 0.6%Cu or 280KT of contained copper at a 0.2% reporting cut.

The 2015 resource estimate shows an improved level of confidence in 15MT of the former resources which are now reported as Indicated Resources (15MT @ 0.7%Cu, or 105KT of contained copper at a 0.2% reporting cut). The expansion drilling has replaced and marginally increased the upgraded Inferred Resources which now stand at 49.7MT @ 0.6%Cu, or 298KT of contained copper at a 0.2% reporting cut.

14.11 MODELLING OF LEACHABLE COPPER

14.11.1 Leachable Copper

Only a proportion of the total copper present in the mineralized zone can actually be qualified as “leachable” for the purposes of recovery through a heap leach process. This proportion is required to effectively evaluate the economic viability of the various mineralized material types. Therefore, as part of the PEA process, it was agreed that the proportion of total copper that can be considered leachable should be modelled as a new resource model item.

A Percent Leachable Copper Model was generated and stored as a modifying variable within the 2015 resource model prepared for engineering studies. The leachable copper model reflects the spatial distribution of the percentage of copper soluble in combined sulphuric acid and cyanide solution as indicated by the assays obtained from a sequential soluble copper analytical program undertaken on the 2015 KSK drill core pulps.

The final model supplied to engineers was the 2015 resource model, renamed to “BK_postestimate30000_Oct2015_S_mat_leach.bmf” which reflects the additional variables estimated post resource reporting in November 2015, these being the percent leachable copper, total sulphur grade and the host rock material type. The total sulphur and host rock material type variables are not utilized or reported in the PEA and therefor are not described in this document.

14.11.2 Data Description

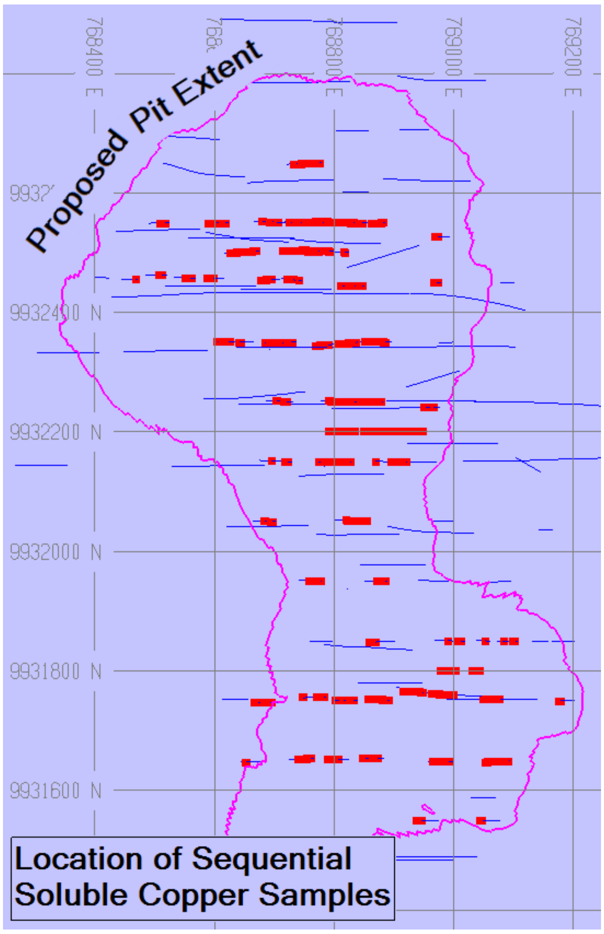
1918 samples from 59 drillholes drilled by KSK in 2015 were selected from pulps stored at the Intertek Utama Services laboratory in Jakarta (ITS) and submitted for sequential soluble copper assays at ITS.

Table 14-7 lists the number of samples selected from each hole and Figure 14-8 shows the spatial distribution of holes with sequential soluble copper assays. There is a uniform horizontal and vertical coverage of samples selected for these assays which, although less in number, reflects the relative sample distribution underpinning the Copper Resource Estimate. Samples were not selected from pre 2015 holes as, although not proven, it is highly probable that pulps from these holes will be compromised as the sulphides are likely to have oxidized in the tropical climate.

Table 14-7 Sequential Soluble Copper Assays selected from holes drilled by KSK in 2015

Hole	Number	Hole	Number	Hole	Number	Hole	Number
BK31550-01	6	BK31750-09	9	BK32200-02	53	BK32450-06	8
BK31550-02	6	BK31800-01	50	BK32200-03	54	BK32450-07	1
BK31650-01	4	BK31800-02	18	BK32250-01	7	BK32450-10	5
BK31650-02	19	BK31850-02	13	BK32250-02	59	BK32500-01	76
BK31650-03	22	BK31850-03	16	BK32250-03	85	BK32500-02	86
BK31650-04	13	BK31850-04	9	BK32250-04	6	BK32500-03	65
BK31650-05	27	BK31850-07	1	BK32350-02	57	BK32550-01	7
BK31650-06	45	BK31950-03	19	BK32350-03	64	BK32550-02	16
BK31750-01	53	BK31950-04	32	BK32350-04	38	BK32550-03	40
BK31750-02	62	BK32050-01	14	BK32350-05	23	BK32550-04	58
BK31750-03	64	BK32050-02	56	BK32450-01	35	BK32550-05	8
BK31750-04	56	BK32150-02	31	BK32450-02	21	BK32550-06	22
BK31750-05	20	BK32150-03	54	BK32450-03	25	BK32550-07	94
BK31750-06	18	BK32150-04	10	BK32450-04	9	BK32650-01	59
BK31750-07	21	BK32200-01	57	BK32450-05	12	**TOTAL**	**1918**

Figure 14-8 Location of sequential soluble copper samples



14.11.3 Laboratory Procedures

ITS was charged with retrieving sample and QC pulps from 50 original 2015 batches and amalgamating these into 12 batches for sequential soluble copper analysis according to the following procedure:

- GA40: Sample digested in 5% sulfuric acid for 1 hour. Copper in the solution is then determined in a dilute nitric acid solution by AAS. ITS quotes a precision of ±10% for this method.
- CN06: Centrifuged residual of GA40 analysis is washed then digested in 0.3% NaCN (in 0.3% NaOH) for 1 hour. The liquid is separated by centrifuging and the copper content of the solution determined by AAS. ITS quotes a precision of ±15% for this method.
- GA30: Centrifuged residual of CN06 is washed then 0.25g subjected to three acid digest to insipient dryness (hydrochloric, perchloric and nitric), washed in concentrated hydrochloric acid and diluted to 100ml. Copper is determined by AAS. ITS quotes a precision of ±5% for this method.

14.11.4 Quality Control Evaluation

Quality control samples included in batches allowed the evaluation of:

- Client and laboratory standards and blanks
- Laboratory repeats and second samples (for each assay in the analytical sequence)
- Total copper grade comparison (between Seq. Sol Cu and original GA30 Cu assay of pulp)
- Review of repeatability of assay results for each assay in the analytical sequence for client and laboratory standards

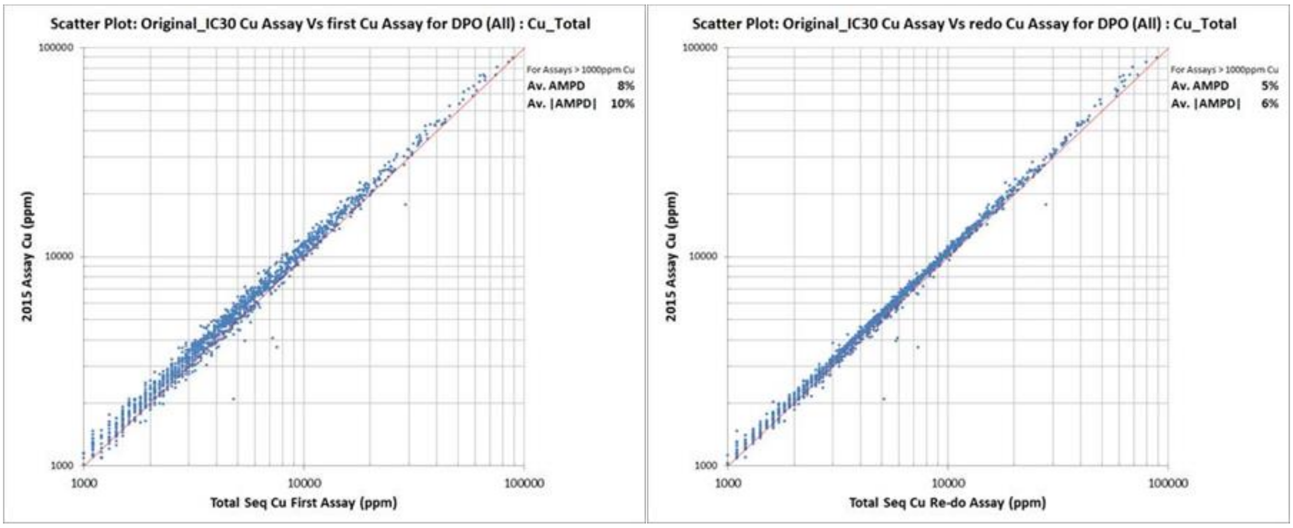
A summary of the QC analysis of the sequential soluble copper analyses undertaken by ITS in December 2015 highlighted that:

- There is serious concern with the laboratories capability or performance in generating reliable sequential assays as evidenced in both the Lab and Client standards performance.
- The laboratory shows competency in overall repeatability of assays from pulps with the REP (second charge from pulp undertaken at end of batch) results being generally of lower correlation than the SS results (second charge taken and run in sequence following first charge). The exception to this is the poor repeatability of the REP results for the CN and Residual Copper assays. The breakdown in precision in the REP pairs compared with the SS pairs suggests that there are procedural, competency or instrumental consistency issues that are more likely to vary more as time intervals between assays increase.
- There is a systematic minus 10% relative bias in the total SEQ Cu assay (GA40+CN06+GA30) compared with the original 3 acid copper assays utilized in the resource estimate.

H&A considers that the total leachable copper (GA40+CN06) and the total SEQ copper assays are unsuitable for determining the percentage leachable copper within the resource/reserve model and that, in the interim, while ITS was investigating and re-assaying the samples, that the percent leachable copper be determined by use of the original three acid digest copper result (pre November 2015) and the GA30 copper assay result from the residual material (December 2015). The assays for CRM client standards OREAS 151A and OREAS 50C, where copper is contained in bornite and chalcopyrite, add support that the GA30 assays are reliable.

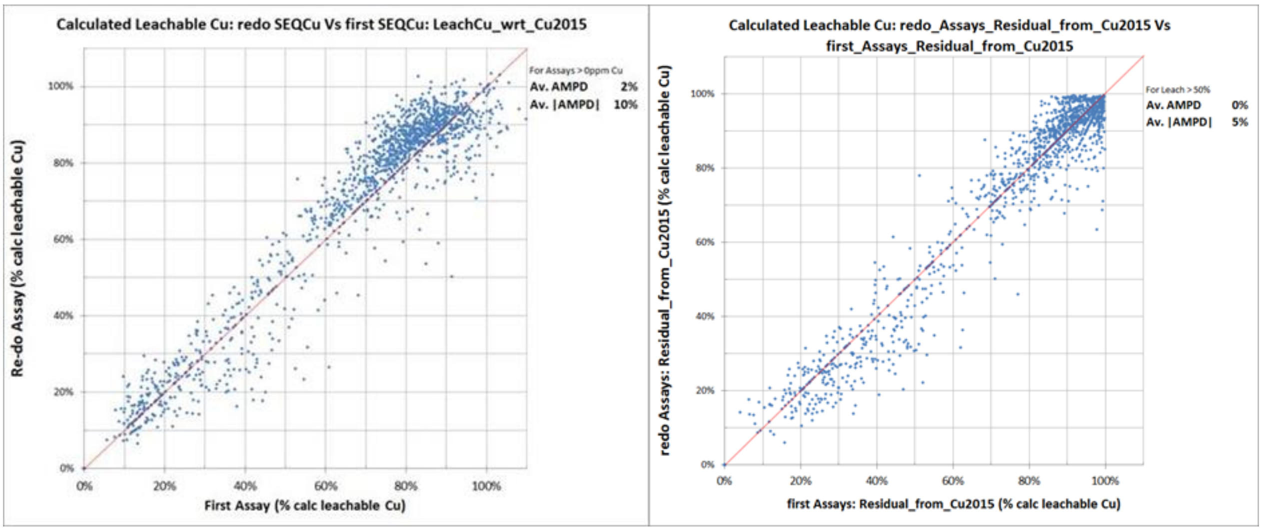
ITS re-assayed the samples utilizing an updated procedural change where the GA40 digest was extended to 2 hours and the residue removed from contact with the solution at the time of taking an aliquot for analysis in both the GA40 and CN06 digests. QC results for the re-assays (February/March 2016) show improved reliability of the results and the correlation between the combined soluble copper assays and the resource estimate copper assays has improved, though a negative bias still exists in the combined soluble copper assay (refer toFigure 14-9). Figure 14-10 shows that the upgraded laboratory procedures has increased the percent leachable copper without affecting the copper grades of the residual material between the first and redo assays (given the precision levels stated by ITS). Those samples with >50% leachable copper show an Av. AMPD of 0% and those in the <50% leachable copper range show a 10% reduced Av. AMPD in the redo assays vs the first assays.

Figure 14-9 First and re-do sequential soluble copper assay comparison.



Scatter plots and AMPD show improved performance with respect to the original pre November 2015 GA30 Cu assays.

Figure 14-10 First and re-do sequential soluble copper assay comparison.



Note: Left; GA40+CN06 (acid and cyanide soluble). Right; residual copper as a percentage of original pre November 2015 copper grade.

Work with ITS is continuing in improving the reliability of the sequential soluble copper assays. H&A considers that, by using the residual copper assay (i.e. the non-leachable component) and the original 2015 copper assay that a reasonable estimate of the percent leachable copper can be produced.

14.11.5 Interpolation and verification

The percent leachable copper is determined by the following formulae:

percent leachable copper = (2015_Cu_assay - Seq_Sol_Cu_residual_assay) / 2015_Cu_assay

The percent leachable copper variable was interpolated into the resource model utilizing the following parameters:

- Methodology: Inverse Distance Squared
- Composite length: 3m (percent leachable copper from first assays undertaken in December 2015)
- Hard boundary control: Domains 17, 25, 30, 60 and 90 (those utilized in 2015 resource estimate).
- BM variable: Leach3
- Interpolation was undertaken in two passes for each domain, reflecting the block proximity to drilling data and block relationship with mineralization domains:
  - Pass 1: Within modeled mineralized domains and a search radius of 100mX100mX25m (runs id017a, id025a, id030a, id060a and id095a). Composites within domains can inform blocks within domains, composites outside of domains are not used. Five search ellipsoids orientations are employed, each reflecting the overall geometry of the domains they best fit (refer Section 14.1). A minimum of 5 and maximum of 24 composites are used to generate block grades. Octant search parameters are employed with a minimum of 6 octants to be informed before a grade is interpolated.

- Pass 2: Within modeled mineralized domains and a search radius of 200mX200mX50m (runs id017b, id025b, id030b and id095b) and 300mX300mX50m (run id060a). All other parameters are the same as for Pass 1 except a maximum of 16 composites and minimum of 4 octants criteria is applied.
- Pass 3: Within modeled mineralized domains a relaxed run (idfill) was employed to fill blocks at the extremity of the model.
- Pass 4: Outside of modeled mineralized domains and within a search radius of 200mX200mX50m (run id5a). All other parameters are the same as for the Pass 1 except a maximum of 10 composites applied and the octant search criteria removed.

15 MINERAL RESERVE ESTIMATES

As there is no Mineral Reserve associated with this study, the term "ore" has not been used to avoid any implication of levels of certainty. The that has been material determined as economically and metallurgically suitable for processing, and has been included in the mine plan developed for this study, is referred to as "leach feed inventory". This has been abbreviated to LFI for the remainder of this document.

16 MINING METHODS

16.1 OPERATING PARAMETERS AND ASSUMPTIONS

The BKM prospect covers the 1000m north-south strike extent and 950m width of a vein style deposit of predominately leachable mineralogy (i.e. covellite and chalcocite). As such it is amenable to a low strip ratio open pit mining approach utilizing a conventional truck and shovel based mining methodology.

The PEA mining evaluation was based around the following initial assumptions:

- A mining fleet of relatively small equipment (i.e. 100 – 120 tonne excavators and 40 tonne Articulated Dump Trucks (ADT's)). This was based on:
    - A relatively low annual total material movement (TMM)
    - The steep terrain in which the BKM is situated (refer to Section 0)
    - The high rainfall combined with relatively weathered open pit materials

Mining activities would be undertaken by a local contractor in order to:

- Minimize upfront capital
- Utilize their local knowledge to accelerate project start-up
- Maximize utilization of the local workforce

On this basis the mining component of this PEA comprised:

1. Initial high level open pit optimization and design to:
    a. Confirm the project order of magnitude for scale
    b. Assess potential pit ramp configurations to confirm final wall slopes
2. First pass strategic schedule to:
    a. Provide to three (3) local mining contractors for a budgetary mining cost estimate
    b. Provide the PEA team with an initial indication of project metrics
3. Final open pit optimization based on preferred mining contract submission and latest CAPEX / OPEX for processing and infrastructure
4. Mine design and mining general arrangement
5. Final Life of Mine (LOM) schedule and associated mining cost estimate
6. Risk Assessment and identification of opportunities and areas for further evaluation

16.2 INITIAL STRATEGIC EVALUATION AND MINING CONTRACT QUOTATION

The first pass open pit optimization was carried out using very high level parameters primarily for the purpose of determining an approximate open pit footprint and inventory. This inventory was then used to generate a LOM schedule, and the resulting physicals and general layout passed to three (3) Indonesian based mining contractors for a budgetary cost estimate. A summary of the quotation received are provided in Table 16-1 below. Given the commercially sensitive nature of the estimates, the names of the contracting groups are not disclosed here.

Table 16-1 Mining Contractor Cost Estimate – Submissions Received

Tenderer	CAPEX ($M)			OPEX	
	Site Establishment	Mob/De-Mob	Total	$M	$/t
Contractor No. 1	$2.21M	$2.38M	$4.59M	$252.34M	$2.33
Contractor No. 2	$3.20M	$2.24M	$5.45M	$259.69M	$2.39
Contractor No. 3	$0.25M	$1.21M	$1.47M	$232.27M	$2.14

The Contractor No. 3 quote was selected, primarily on a lowest cost basis, and the price breakdown associated with this quote was utilized for all subsequent estimations.

16.3 OPEN PIT OPTIMIZATION

This phase of the study was carried out using Geovia's WHITTLE™ optimization software (Whittle), a ubiquitous industry tool for open pit optimization. Input parameters for the optimization were based on robust calculations and / or estimates wherever possible. Where no definitive data number was available high level assumptions were made based on comparable operations / projects or the collective experience of the project team.

16.3.1 Input Parameters and Assumptions

16.3.1.a Mining Model

Orelogy was provided with the 2014 Beruang Kanan Resource model (*BK_postestimate30000_Oct 2015.bmf*). The model was provided as a block model developed utilizing the Maptek Vulcan™ (Vulcan) modelling software. Orelogy also utilized Vulcan to undertake any subsequent coding of the model for optimization purposes, thereby maintaining the integrity of the model.

In order to correctly evaluate the economic viability of the material types, H&A modelled a leachable copper attribute which was then added to the resource model. This took account of the proportion of total copper that could be considered "leachable" in terms of the proposed heap leach process. This approach provides a more accurate reflection of the processing performance of the deposit.

16.3.1.b Pit Wall Slope Assumptions

No geotechnical assessment has been carried out for the BKM open pit. Therefore Orelogy made broad assumptions on acceptable wall slope angles based on:

- The degree of weathering of the material
- Limited RQD data
- Practical experience

A Rock Quality Designation (RQD) was evaluated for ~6500m of core drilling. The resulting Rock Mass Quality of 65% indicates "fair" rock conditions. On this basis Orelogy would not anticipate an inter-ramp angle of steeper than 55 degrees. In addition, as the western wall extends to well over 200 meters vertically in some areas, a significant geotechnical berm will need to be implemented over an acceptable stack height to ensure:

- Acceptable overall slope angle for an extended wall height with no ramps, to minimize risk of large scale catastrophic wall failure
- A suitable sized catch berm to arrest any debris from small scale failures and ensure any material does not propagate down the entire face

Table 16-2 shows the calculated wall parameters selected for the initial optimization, and final selected optimization slopes which were based on assessment of the first pass designs.

Table 16-2 Wall Slope Parameters

Item	Unit	Western	Eastern
Face angle	°	75	
Bench height	m	10	
Berm width	m	5	
Inter-Ramp Angle (IRA)	°	52.5	
Geotechnical Berm (or Ramp)	m	25	N/A
Wall Height	m	200	100
Ramp Width	m	16	16
Number of Ramps on Slope	#	2	1
Overall Slope Angle (OSA)	°	43.5	47.1
Slope angle applied to final optimisation based on pit designs	°	45	

16.3.1.c Mining Dilution and Feed Losses

During mining, some waste dilution and leach feed loss will occur. The amount of dilution that occurs will be dependent on the nature of the mineralized zones being mined. At this time, a detailed dilution assessment has not been completed and factors were used. An average 5% dilution was applied to head grade. Mineralization losses were assumed at 5%. This dilution and mineralization loss reflects delineation errors and mixing along mineralization / waste contacts caused by potential blast movements and ability of the mining equipment to mine precisely along these contacts.

16.3.1.d Input Parameters

Whittle optimization parameters are summarized in Table 16-3.

Table 16-3 Whittle Parameters

Parameter		Unit	Value
Geotechnical	Overall Slope Angle	degrees	45
Inventory Losses	Mining Recovery	%	95
	Mining Dilution	%	5
Mining Cost	Drill and Blast Cost	US$/t waste	0.329
	Pit/Dump Clearing and Road Construction (Included in waste mining cost only)	US$/t waste	0.325
	Fixed Contractor Costs (i.e. Overheads/Salaries)	US$/t waste	0.029
	Road Construction	US$/t waste	0.010
	Final Wall Pre-split Blasting	US$/t waste	0.006
	Mining Cost - reference cost at reference level 405 m	US$/t total	1.64
	Vertical haulage cost differential below 405 m	US$/t total /10m'	0.08
	Vertical haulage cost differential above 405 m	US$/t total /10m'	0.05
Leach Feed Premium	Fixed Owner Mining Costs (i.e. Mine Services/Overheads/Salaries)	US$/t feed	0.29
	Grade Control	US$/t feed	0.19
	Feed Rehandling	US$/t feed	0.47
	Total Leach Feed Mining Premium	US$/t feed	0.95
Waste only cost	Rehabilitation Cost	US$/t waste	0.1
Processing	Production Rate	kt (Cu Cathode)	25
	Processing Cost	US$/t feed	5.15
	Copper Process Recovery	%	85
	Copper (modelled)	%	Var.
Financials	CAPEX	$M	164
	Discount Rate per Period	%	10
	Product Pricing		
	Copper	US$/lb.	3.25
	Selling Costs		
	Transport Haulage	US$/t/km of product	0.15
	Royalties		
	Government Royalty	%	4
	NSR	%	1

16.3.2 Results

The Base Case optimization was based on a single set of input variable values, as outlined in Section 16.3.1.c.

This section summarizes the Whittle output, comprising a series of nested shells, and describes the selection of a single shell as a basis for sensitivity comparisons.

The Whittle pit optimization software estimates a "best case" and a "worst case" discounted value. The best case requires that each shell be mined sequentially while the worst case mines the deposit on a bench by bench basis. The best case is generally impracticable as shell increments can be very small and therefore un-mineable by themselves. The worst case is always achievable but gives much lower values. In practice, a compromise between the two cases is generally achieved by staging the pit using suitable cut-backs.

The cash flow estimates generated by Whittle include provision for initial capital cost and time cost (G&A cost) but do not include taxation or other accounting treatments.

The economic output from Whittle has been considered only as a relative indicator of overall project value to determine an economic Whittle shell to be used as a guide for pit design. The final economical evaluation of the BKM pit inventory has been undertaken within a financial model, running multiple iterations using economic parameters.

Table 16-4 below summarizes the results of the Base Case optimization. In Table 16-4 the "Total" tonnes are a measure of the shell size generated and includes all material mined (i.e. leach feed material + waste).

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Table 16-4 Whittle Optimization Results – Base Case Scenario

Rev. Factor	Shell	Total	Waste	Leach Feed	Strip Ratio	Incr. SR	Cu Total	Cu Leach	Revenue	Processing Cost	Mining Cost	Selling Cost	Cash flow	DCF		Mine Life	Cu Cathode Produced
														Best Case	Worst Case		
		Mt	Mt	Mt	t:t	W:O	%	%	$M	$M	$M	$M	$M	$M	$M	Yrs.	kt
0.20	1	3.2	1.0	2.2	0.5	0.5	1.25	1.01	128.3	-11.3	-5.9	-3.4	-56.2	-66.2	-66.2	1.1	18.8
0.25	2	9.3	3.0	6.3	0.5	0.5	0.97	0.78	285.2	-32.4	-16.7	-7.5	64.7	32.4	31.9	2.0	41.9
0.30	3	20.2	7.2	13.0	0.6	0.6	0.82	0.67	504.4	-67.0	-36.6	-13.2	223.7	150.4	148.4	3.3	74.1
0.35	4	27.2	10.5	16.7	0.6	0.9	0.77	0.63	611.8	-86.2	-49.6	-16.0	296.0	198.1	194.7	4.0	89.9
0.40	5	34.0	13.9	20.1	0.7	1.0	0.74	0.60	698.9	-103.5	-62.7	-18.3	350.5	233.5	228.7	4.5	102.7
0.45	6	43.3	18.3	25.0	0.7	0.9	0.70	0.56	805.6	-128.8	-80.6	-21.1	411.1	269.4	261.2	5.1	118.3
0.50	7	52.3	23.1	29.3	0.8	1.1	0.67	0.53	893.5	-150.7	-98.5	-23.4	456.9	295.9	284.6	5.6	131.3
0.55	8	57.8	26.2	31.6	0.8	1.3	0.66	0.51	939.3	-162.8	-109.4	-24.6	478.5	307.4	294.3	5.9	138.0
0.60	9	66.6	31.1	35.6	0.9	1.2	0.63	0.49	1,010.0	-183.2	-127.7	-26.4	508.6	323.5	306.4	6.3	148.4
0.65	10	71.7	34.4	37.2	0.9	2.1	0.63	0.48	1,041.3	-191.7	-138.2	-27.2	520.2	329.6	311.1	6.5	153.0
0.70	11	77.6	38.0	39.6	1.0	1.5	0.62	0.47	1,079.5	-204.2	-150.4	-28.2	532.7	335.7	315.1	6.7	158.6
0.75	12	82.5	40.9	41.5	1.0	1.5	0.61	0.46	1,107.3	-214.0	-160.2	-29.0	540.2	339.2	316.1	6.9	162.7
0.80	13	91.5	47.2	44.3	1.1	2.3	0.60	0.45	1,151.2	-228.0	-179.2	-30.1	550.0	343.9	317.9	7.1	169.1
0.85	14	94.5	49.1	45.4	1.1	1.7	0.59	0.44	1,166.3	-233.6	-185.7	-30.5	552.5	345.1	318.2	7.2	171.3
0.90	15	98.9	51.8	47.1	1.1	1.5	0.59	0.44	1,186.8	-242.5	-194.5	-31.0	554.8	346.2	316.8	7.3	174.4
0.95	16	101.8	54.0	47.8	1.1	3.3	0.58	0.43	1,197.4	-246.0	-200.6	-31.3	555.5	346.5	316.3	7.4	175.9
1.00	17	102.7	54.8	47.9	1.1	4.3	0.58	0.43	1,200.6	-246.9	-202.7	-31.4	555.6	346.5	316.1	7.4	176.4
1.05	18	115.4	65.4	50.0	1.3	5.2	0.58	0.43	1,240.3	-257.6	-231.3	-32.4	555.0	346.1	313.4	7.7	182.2
1.10	19	120.1	68.9	51.2	1.4	3.1	0.58	0.42	1,256.1	-263.5	-241.9	-32.8	553.8	345.5	311.6	7.7	184.5
1.15	20	122.4	70.7	51.7	1.4	3.4	0.57	0.42	1,263.0	-266.2	-246.9	-33.0	553.0	345.0	310.3	7.8	185.6
1.20	21	126.3	74.0	52.2	1.4	5.9	0.57	0.42	1,273.3	-269.1	-255.9	-33.3	551.1	344.1	308.6	7.9	187.1
1.25	22	137.7	83.8	53.9	1.6	6.0	0.57	0.42	1,302.4	-277.4	-282.0	-34.1	544.9	341.2	303.1	8.0	191.3
1.30	23	138.5	84.5	54.0	1.6	4.0	0.57	0.42	1,304.5	-278.2	-283.8	-34.1	544.4	340.9	302.6	8.0	191.6
1.35	24	142.1	87.5	54.5	1.6	5.9	0.57	0.42	1,312.7	-280.9	-291.7	-34.3	541.8	339.7	300.2	8.1	192.8
1.40	25	146.2	91.2	55.1	1.7	7.0	0.57	0.42	1,321.6	-283.6	-301.1	-34.6	538.4	338.1	297.7	8.1	194.2
1.45	26	151.8	96.1	55.7	1.7	8.2	0.57	0.41	1,332.4	-286.7	-313.0	-34.8	533.9	336.1	293.5	8.2	195.7
1.50	27	156.7	100.5	56.2	1.8	7.6	0.57	0.41	1,342.0	-289.6	-324.0	-35.1	529.3	334.0	290.1	8.3	197.2
1.55	28	160.2	103.6	56.6	1.8	8.5	0.57	0.41	1,349.3	-291.5	-333.0	-35.3	525.5	332.3	288.0	8.3	198.2
1.60	29	163.3	106.3	57.0	1.9	7.8	0.57	0.41	1,355.0	-293.3	-340.1	-35.4	522.2	330.9	285.7	8.3	199.1
1.65	30	166.1	108.8	57.3	1.9	8.2	0.57	0.41	1,360.2	-294.9	-346.8	-35.6	519.0	329.4	283.9	8.4	199.8
1.70	31	166.5	109.2	57.3	1.9	6.9	0.57	0.41	1,361.1	-295.2	-348.0	-35.6	518.3	329.2	283.6	8.4	200.0
1.75	32	167.3	109.9	57.4	1.9	8.8	0.57	0.41	1,362.3	-295.6	-349.6	-35.6	517.5	328.8	283.0	8.4	200.1
1.80	33	167.7	110.3	57.4	1.9	7.7	0.57	0.41	1,363.0	-295.9	-350.7	-35.6	516.8	328.5	282.5	8.4	200.2
1.85	34	170.4	112.7	57.7	2.0	10.8	0.57	0.41	1,367.1	-297.0	-356.8	-35.7	513.5	327.0	280.4	8.4	200.8
1.90	35	173.1	115.2	57.9	2.0	9.6	0.57	0.41	1,371.1	-298.3	-363.1	-35.9	509.8	325.4	278.2	8.4	201.4
1.95	36	173.4	115.5	58.0	2.0	6.5	0.57	0.41	1,371.7	-298.6	-364.0	-35.9	509.3	325.2	277.9	8.4	201.5
2.00	37	173.6	115.6	58.0	2.0	5.2	0.57	0.41	1,371.8	-298.7	-364.3	-35.9	509.0	325.0	277.8	8.4	201.5

16.3.3 Shell Selection

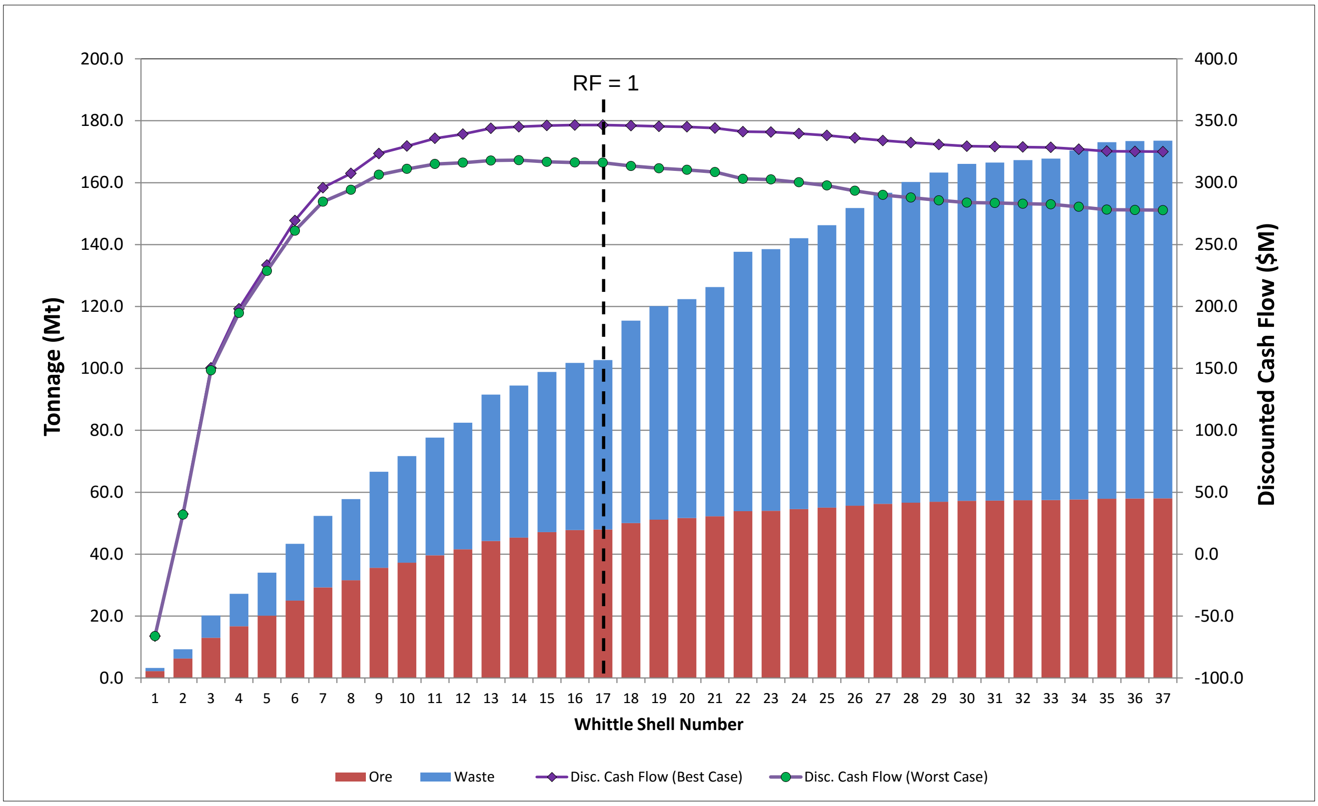
The main factors in the shell selection were the business objectives of:

- Acceptable feed grade and maintaining production targets
- Obtaining a low stripping ratio low while
- Generating an acceptable mine life

An integral part of open pit optimization is deciding which section of the ultimate pit to mine during a specific period.

Figure 16-1 summarizes the optimization results in graphical format. While the Best Case Schedule discounted cash flow curve peaks at Shell 17, there is little additional value after Shell 14. It was concluded that the Worst Case shell would be a much more robust economic shell. Therefore Shell 14 (RF 0.85) was selected on the basis of the shell returning the highest Worst Case Discounted Cash flow.

Figure 16-1 Whittle Pit by Pit Graph



Mine designs based on the shells will typically add extra waste with some potential loss of feed material. This is due to the requirement to account for:

- Access requirements
- Minimum practical mining width
- Geotechnically rational wall geometry
- Other practical mining constraints.

Typical open pit optimization takes no account of mining width in the generation of pits. Consequently, an optimized pit which is used as a starting point for the design of a final pit can have:

- distances from a pit wall to the topographical "daylight" that are too narrow to mine safely and practically or,
- a pit floor that is too constricted to access

Early iterations of the BKM designs indicated that the steep topography was exacerbating issues with the optimal shells related to mining width. Therefore ORELOGY utilized the "Mining Width" approach in Whittle to produce a more realistic pit shape suitable for design purposes.

The preferred Shell 14 was selected for application of the "Mining Width" functionality. Shell 18 was selected as an ultimate push back extent within which a practical design can lie without compromising the results of the selection of an economic shell.

Table 16-9 details optimization results using Whittle's Minimum Mining Width (MMW) module. Whittle shells 14 and 18 (Figure 16-1) have been used for pushback definition within the module. The minimum mining width of two blocks by two blocks (50m x 50m) was applied. The application of MMW has added 22% more waste to the final shell and 7% more feed material but retained overall cash flow and reduced DCF by no more than 3%.

Table 16-5 Effect of Applying MMW

Parameter	Unit	Before MMW	After MMW	% Variation from Before Case
Rock	Mt	94.5	108.2	14%
Waste	Mt	49.1	59.9	22%
Feed Material	Mt	45.4	48.4	7%
SR	T:t	1.1	1.24	13%
Grade Input	TCu %	0.59	0.58	-2%
Grade Input	Cu (sol) %	0.44	0.43	-2%
Cu Cathodes Produced	kt	171.3	177.1	3%
Revenue	$M	1166	1,205	3%
Processing Cost	$M	-233.6	-249.1	7%
Mining Cost	$M	-185.7	-213.8	15%
Product/Selling Cost	$M	-30.5	-31.5	3%
Cash flow	$M	552.5	549.0	-1%
Worst DCF	$M	318.2	310.1	-3%
Mine Life	Years	7.2	7.5	4%

16.3.4 Sensitivity Analysis

In order to assess the robustness of the project, optimization sensitivities were undertaken for BKM deposit. A range of scenarios were evaluated and are summarized in Table 16-6.

Table 16-6 Optimization Scenarios

Scenario	Description
Scenario 1	Base Case
Scenario 2	Price +10%
Scenario 3	Price +20%
Scenario 4	Price -10%
Scenario 5	Price -20%
Scenario 6	Processing Cost +10%
Scenario 7	Processing Cost +20%
Scenario 8	Processing Cost -10%
Scenario 9	Processing Cost -20%
Scenario 10	Mining Cost +10%
Scenario 11	Mining Cost +20%
Scenario 12	Mining Cost -10%
Scenario 13	Mining Cost -20%

A summary of the sensitivity results for BKM deposit is highlighted in Table 16-7, Table 16-8 and Table 16-9 which show the following:

- The project is sensitive to increases/reductions in price, with cash flow varying significantly.
- The overall size of the best case shell is moderately sensitive to other modifying factors.

Table 16-7 Tonnage Variation (Worst Case Schedule)

Tonnage Variation	Description	% Change from Base Case		
		Total Tonnes	Leach Feed Tonnes	Discounted Cash-flow
>15%	Price +20%	17.0%	8.0%	53.1%
	Price -20%	-15.2%	-9.3%	-48.1%
	Mining Cost +20%	-15.0%	8.2%	-5.9%
5%-15%	Price +10%	11.3%	5.6%	26.9%
	Price -10%	-8.4%	-3.2%	-23.9%
	Processing Cost +10%	-5.3%	-1.6%	-3.7%
	Processing Cost +20%	-13.8%	-8.5%	-7.8%
	Processing Cost -10%	-6.7%	-1.8%	7.0%
	Processing Cost -20%	-5.4%	-1.0%	12.4%
	Mining Cost +10%	-12.6%	-6.5%	-1.8%
	Mining Cost -10%	-5.6%	-1.7%	6.1%
	Mining Cost -20%	-5.1%	-1.6%	10.5%

Table 16-8 Discounted Cash Flow Variation (Worst Case Scenario)

DCF	Description	% Change from Base Case		
		Total Tonnes	Feed Tonnes	Discounted Cash-flow
>15%	Price +10%	11.3%	5.6%	26.9%
	Price +20%	17.0%	8.0%	53.1%
	Price -10%	-8.4%	-3.2%	-23.9%
	Price -20%	-15.2%	-9.3%	-48.1%
5%-15%	Processing Cost +20%	-13.8%	-8.5%	-7.8%
	Processing Cost -10%	-6.7%	-1.8%	7.0%
	Processing Cost -20%	-5.4%	-1.0%	12.4%
	Mining Cost +20%	-15.4%	-8.2%	-5.9%
	Mining Cost -10%	-5.6%	-1.7%	6.1%
	Mining Cost -20%	-5.1%	-1.6%	10.5%
<5%	Processing Cost +10%	-5.3%	-1.6%	-3.7%
	Mining Cost +10%	-12.6%	-6.5%	-1.8%

Table 16-9 Open Pit Optimization Sensitivities

Parameter Changed	Rock	Waste	Leach Feed	SR	Grade Input	Grade Input	Metal Output	Revenue	Processing Cost	Mining Cost	Product/ Selling Cost	Cash flow UCF	Best Case DCF	Worst Case DCF	IRR Best Case	IRR Worst Case	Mine Life	Rock	Waste	Leach Feed	SR	Best Case DCF	Worst Case DCF	Mine Life
	Mt	Mt	Mt	t:t	TCu %	Cu(sol) %	t	$M	$M	$M	$M	$M	$M	$M	%	%	Years	Mt	Mt	Mt	t/t	$M	$M	Years
Base Case	107.9	59.4	48.5	1.22	0.58	0.43	177,380	1,207.4	-249.8	-213.1	-31.6	548.9	311.3	311.1	48%	48%	7.5							
Price +10%	120.1	68.9	51.2	1.34	0.58	0.42	184,578	1,382.1	-263.8	-241.9	-32.9	679.5	428.0	394.8	72%	56%	7.8	11.3%	16.0%	5.6%	9.8%	37.5%	26.9%	3.9%
Price +20%	126.3	73.9	52.4	1.41	0.57	0.42	187,155	1,528.8	-269.7	-255.9	-33.3	805.8	511.3	476.5	81%	63%	7.9	17.0%	24.5%	8.0%	15.6%	64.2%	53.1%	5.2%
Price -10%	98.9	51.9	47.0	1.11	0.59	0.44	174,249	1,067.5	-241.9	-194.5	-31.0	436.1	265.9	236.9	54%	40%	7.3	-8.4%	-12.6%	-3.2%	-9.0%	-14.6%	-23.9%	-1.7%
Price -20%	91.5	47.5	44.0	1.08	0.60	0.45	168,884	919.7	-226.6	-179.2	-30.1	319.9	187.2	161.6	43%	32%	7.1	-15.2%	-20.0%	-9.3%	-11.5%	-39.9%	-48.1%	-4.6%
Processing Cost +10%	102.1	54.4	47.7	1.14	0.59	0.43	176,014	1,198.1	-270.4	-201.4	-31.3	531.0	330.9	299.6	62%	47%	7.4	-5.3%	-8.4%	-1.6%	-6.6%	6.3%	-3.7%	-0.7%
Processing Cost +20%	93.0	48.7	44.4	1.10	0.60	0.45	169,917	1,156.6	-274.3	-182.8	-30.2	505.3	314.9	287.0	61%	45%	7.2	-13.8%	-18.1%	-8.5%	-9.8%	1.1%	-7.8%	-4.0%
Processing Cost -10%	100.6	53.0	47.6	1.11	0.58	0.43	175,393	1,193.9	-220.7	-198.1	-31.2	579.8	361.2	333.0	65%	50%	7.4	-6.7%	-10.7%	-1.8%	-9.0%	16.0%	7.0%	-1.1%
Processing Cost -20%	102.1	54.1	48.0	1.13	0.58	0.43	176,190	1,199.3	-197.8	-201.2	-31.4	605.0	377.1	349.7	66%	51%	7.4	-5.4%	-9.0%	-1.0%	-7.4%	21.1%	12.4%	-0.6%
Mining Cost +10%	94.3	48.9	45.4	1.08	0.59	0.44	171,263	1,165.8	-233.6	-203.8	-30.5	533.9	333.5	305.5	63%	47%	7.2	-12.6%	-17.6%	-6.5%	-11.5%	7.1%	-1.8%	-3.3%
Mining Cost +20%	91.3	46.8	44.5	1.05	0.60	0.45	169,233	1,152.0	-229.2	-214.3	-30.1	514.3	322.0	292.9	62%	46%	7.1	-15.4%	-21.3%	-8.2%	-13.9%	3.4%	-5.9%	-4.4%
Mining Cost -10%	101.9	54.2	47.7	1.14	0.59	0.43	175,875	1,197.2	-245.5	-180.8	-31.3	575.6	358.0	330.1	64%	49%	7.4	-5.6%	-8.8%	-1.7%	-6.6%	15.0%	6.1%	-0.8%
Mining Cost -20%	102.4	54.7	47.7	1.15	0.59	0.43	176,107	1,198.8	-245.8	-161.7	-31.3	595.9	369.9	343.9	65%	51%	7.4	-5.1%	-7.9%	-1.6%	-5.7%	18.8%	10.5%	-0.7%

16.4 PIT AND WASTE ROCK STORAGE DESIGN

The conceptual pit design was developed using the optimized shell as the template.

16.4.1 Pit Design Criteria

The pits will be mined using 2.5 meter flitches for leach feed and waste respectively. This height gives reasonable production efficiency while keeping dilution to a minimum. In waste, the flitch height could be increased to improve efficiency within the limits of the equipment size. Table 16-10 details the design parameters that have been used for the BKM pit designs.

Table 16-10 Pit Design Parameters

Parameter	Unit	Value
Flitch height	m	2.5
Blast Bench Height	m	5
Batter Angle	Degrees	75
Batter Height	m	10
Berm Width	m	5
Inter-ramp Angle (IRA)	Degrees	52.5
Geotech Stack Height on West wall	m	100
Geotech Berm Width	m	25
Overall Slope Angle (OSA) – West Wall	Degrees	44.2
Ramp Width	m	16

16.4.2 Minimum Mining Widths and Standoff Distances

The aim for pit designs, whether they are final or interim stages, is to ensure that there is sufficient space so that, in the bottom of each stage and also while excavating a subsequent stage, these areas can be mined in an efficient and safe manner.

16.4.3 Pit Design & Inventory

Table 16-11 gives the material balance within the pit design and compares it to the selected Whittle shell. The conversion from Whittle to design is very good adding 0.6 Mt of mineralized material with rock tonnage matching closely. The pit design inventory has been reported at a fix breakeven cut-off grade of 0.09% to replicate the inventory generated by Whittle.

The scheduling inventory used in the mine plan (refer to Section 16.5) utilizes a cut-off grade strategy. The strategy classifies LFI material by a Cu Leach cut-off grade that varies over time from 0.09% Cu Leach to 0.11% Cu Leach.

Table 16-12 provides the scheduling inventory utilized for the LOM plan presented in the resource classifications of Indicated and Inferred. The scheduling model is based on regularized block geometry and therefore some smearing between the two categories has occurred. However the total amount of material, and contained metal, within the combined Indicated and Inferred categories has been replicated exactly from the resource model.

It should be noted that while the cut-off grades applied within the mine plan are marginally lower than the initial cut-off grade estimate used to define the resource (refer to Section 14.9) the amount of material in the mine plan that falls below the resource cut-off of 0.2% Cu Total is insignificant, amounting to <50 kt or <0.1% of the total scheduling inventory.

Table 16-11 Comparison between Whittle Shell and Final Pit Design

Source	Rock	Waste	Leach Feed[1]	SR	Cu Total	Cu Leach	Cu Cathodes Produced
	Mt	Mt	Mt	t:t	%	%	kt
Whittle Shell	108.2	59.9	48.4	1.24	0.58	0.43	177.1
Pit Design	108.6	59.6	49.0	1.22	0.58	0.43	177.5
Difference	+0.4%	-0.5%	+1.2%	-1.6%	0%	0%	+0.2%

[1] Breakeven cut-off grade of 0.09% Cu Leach

Table 16-12 Final Pit Design Scheduling Inventory by Resource Classification1

Classification	Leach Feed[1]	Cu Total	Cu Leach
	Mt	%	%
Indicated	14.2	0.66	0.52
Inferred	34.5	0.55	0.39
Total LFI	**48.7**	**0.58**	**0.43**
Waste	**59.8**		
Total	**108.6**		

[1] Based on variable cut-off grade

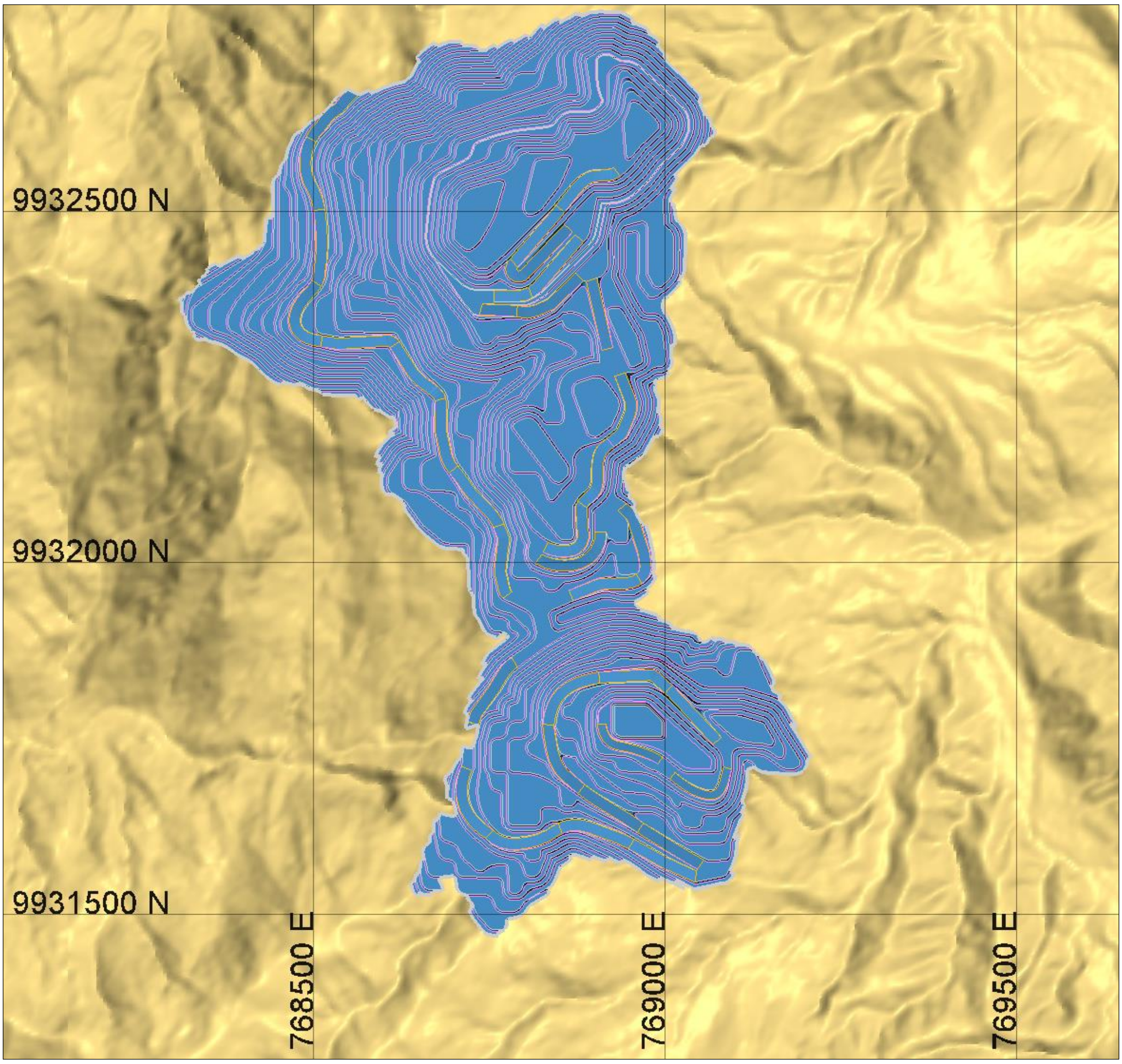
Figure 16-2 presents a plan view of the ultimate pit.

A number of design iterations will need to be completed at the feasibility study stage to test the assumptions used and the designs developed. Designs need to consider long term access to the various stages of the pit and wherever appropriate should include the use of backfill to reduce ex-pit dumping requirements and material movement costs.

For the southern section of the pit, the mine life is much shorter and as such, a more “aggressive” approach was taken to mine design. For example:

- Reducing ramp width to single access only with passing bays over the last 20 vertical metres
- Reducing the minimum mining width at the base of the pit
- Including good-bye cuts

Figure 16-2 Ultimate Pit Plan View



16.4.4 Staging and Pit Development

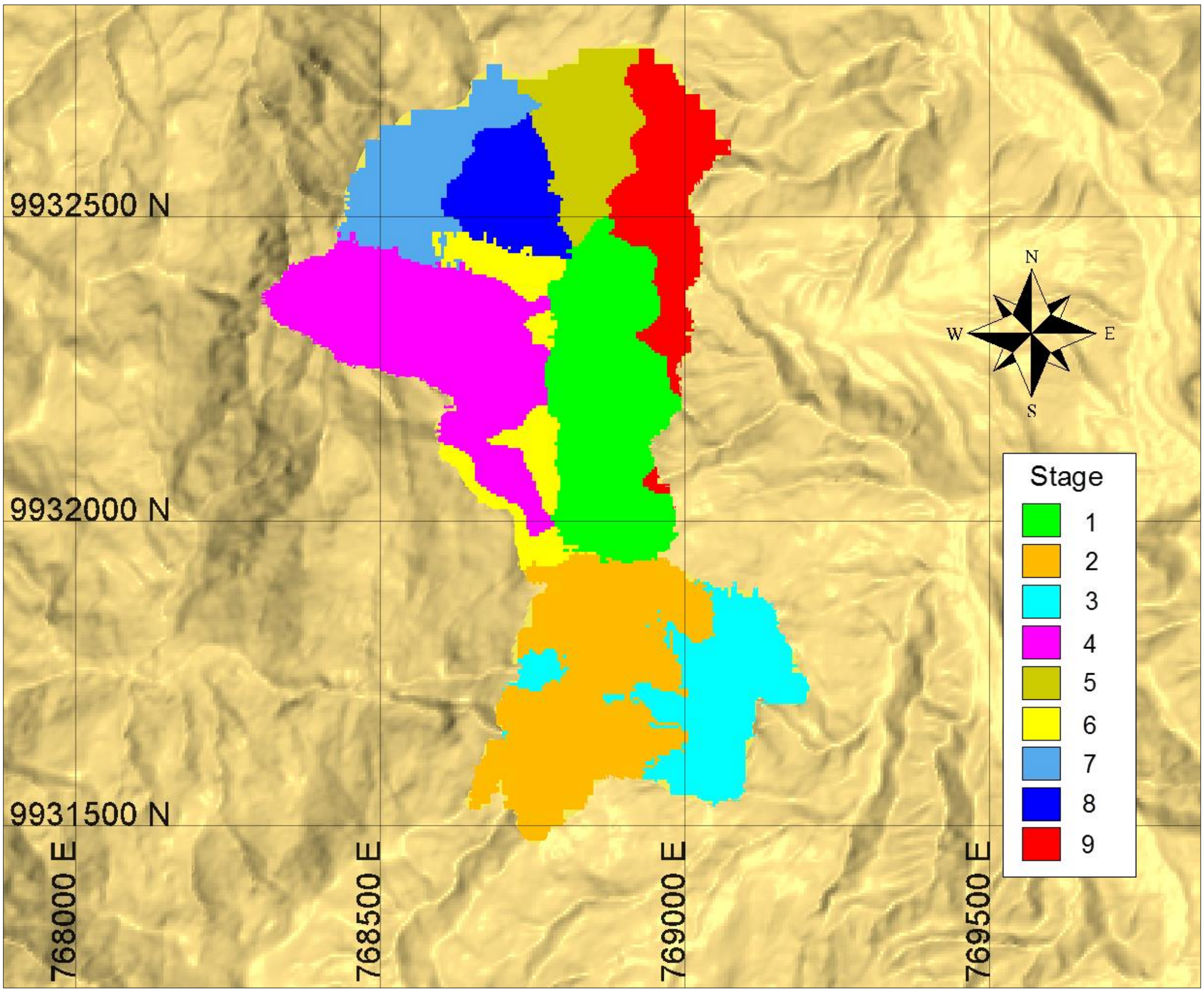
Given the constricted nature of the current financing markets, one of the key mining opportunities was to assess the benefit of a more focused mining strategy; the aim being to maximize returns generated over the first 5 years of the project without significantly affecting the overall project NPV. This period is generally the focus for potential financers as it would cover any normal loan payback period. As opposed to the top-down mining approach for the Worst Case optimization shell, scheduling a series of cutbacks or stages provides a method to delay waste and access higher grade leach feed earlier in the mine schedule.

In order to achieve this goal, the pit was sub-divided into several stages for scheduling purposes. Multiple iterations were assessed before selecting the final solution as shown in Figure 16-3. The large northern section of pit was subdivided into seven mining stages and the smaller southern section was sub-divided into two mining stages. The tonnages contained within each stage are summarized in Table 16-13. The correct push-back distance or waste cut was determined by looking at a number of factors such as equipment size, final planned depth, production rate, ramp geometry etc.

Table 16-13 Tonnage by Stages

Stage	Tonnes
1	8,937,711
2	5,440,170
3	18,608,724
4	9,614,457
5	2,732,689
6	15,740,672
7	2,107,128
8	9,414,105
9	5,974,773
Total	108,570,429

Figure 16-3 Pit Staging



16.4.5 Pit Ramp Design

All haul roads used by mine equipment have been design to accommodate articulated 40t dump trucks. It is industry practice to design the main ramps with a pavement width of 3-3.5 times the truck width using a 10% gradient. This allows for safe passing of trucks exclusive of wall side drainage and pit side bunding. For the benches at the pit bottom (up to 20 meter overall height), a single lane ramp pavement width of 2.0-2.5 times the truck width was adopted at 10% gradient to reflect the lower traffic intensity on this section of the ramp and to minimize waste development. External roads will be designed with a bund on either side and with drainage gaps at regular intervals.

The shoulder barrier or safety bund on the outside edge will be constructed to a height equal to the rolling radius of the largest tire of the trucks using the ramp. The rolling radius of the 40t dump truck tire is 0.93 m. These shoulder barriers will be designed at 1.1H: 1V width 0.5m flat on top of the bund.

A ditch planned on the highwall side of ramps will capture run-off water from the pit wall surface and assure proper drainage of the running surface. The ditch will be 1.0 m wide. To facilitate drainage of the roadway, a 2% cross slope on the ramp is planned.

The haul road design specifications are detailed in Table 16-14. The dual lane ramp width is designed to be 16m wide (Figure 16-4) and the single lane ramp width is designed to be 12 m wide (Figure 16-5).

Table 16-14 Haul Road Design Criteria

Component	Dual Lane	Single Lane	Ex-pit Road
Truck Width (m)	4.1	4.1	4.1
Drain Width (m)	1.0	1.0	-
Windrow Base Width (m)	2.5	2.5	2.5 + 2.5
Pavement Width (m)	12.5	8.5	15.5
Overall Road Width	16	12	20.5
Gradient (max)	10%	10%	-

Figure 16-4 Dual Lane Ramp Design

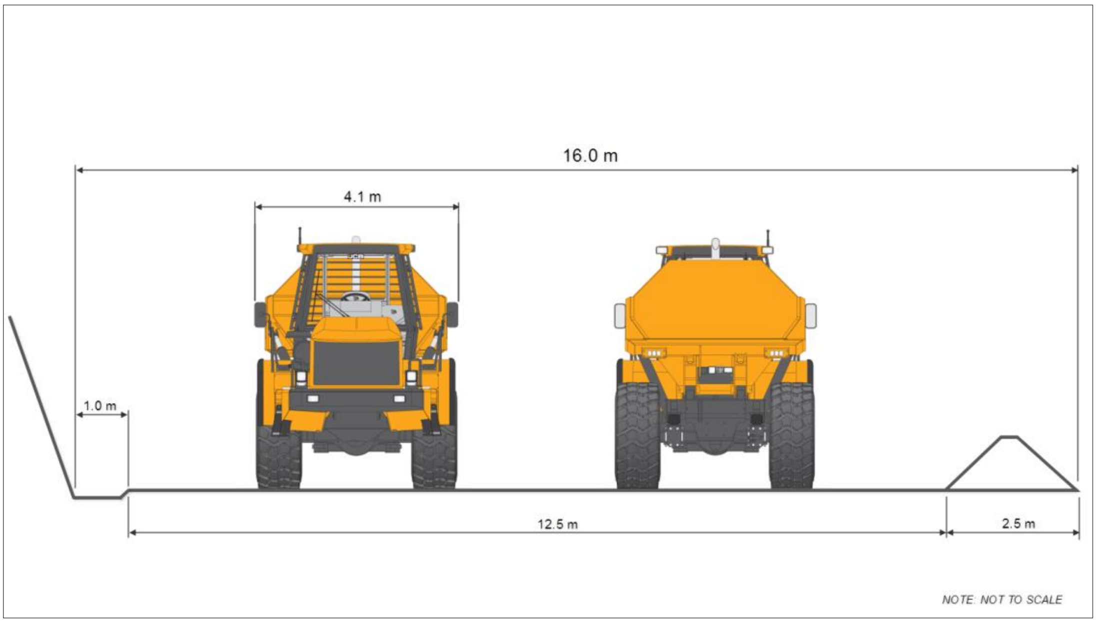
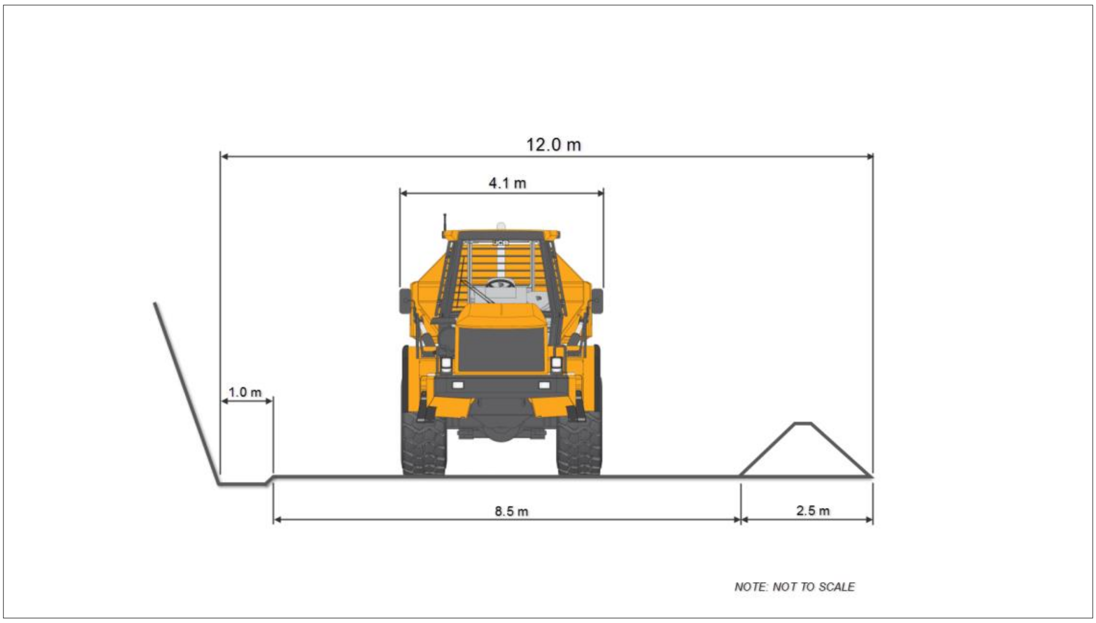
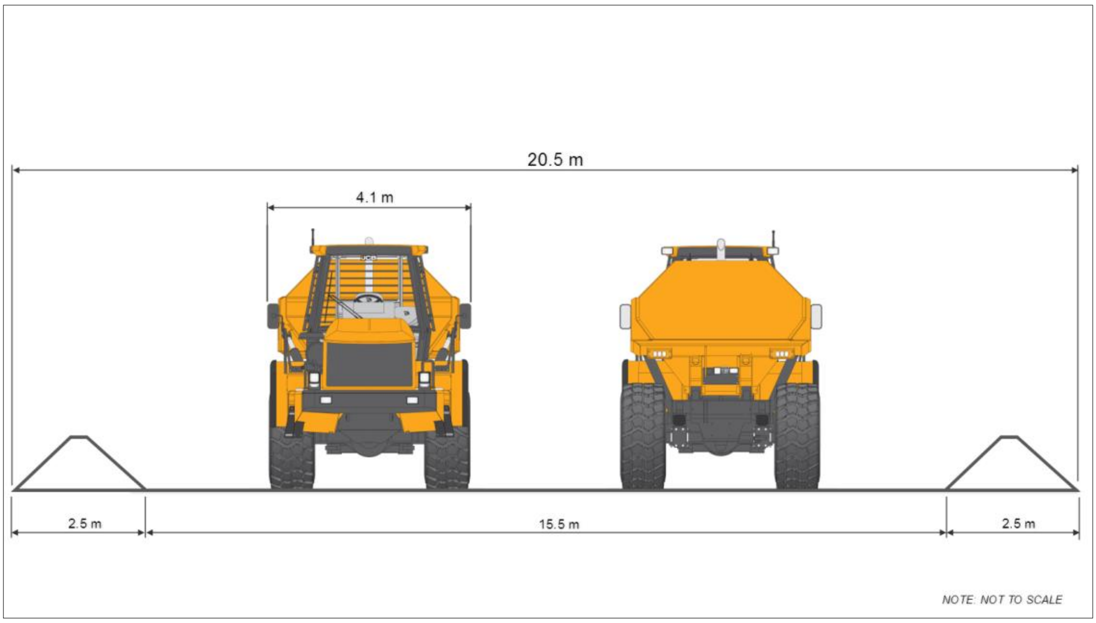


Figure 16-5 Single Lane Ramp Design



The out-of-pit double lane haul road with a berm on each side will measure 20.5 m wide (Figure 16-6).

Figure 16-6 Dual Lane Out of Pit Haul Road Profile



16.4.6 Waste Rock Dump Design

Waste material will be disposed in dumps adjacent to the pit. Steep terrain and unfavorable waste haul routes, which tend to be both uphill and downhill, complicated selection for potential waste dump locations. Two possible waste dump locations (NW & SW) were selected from a number of sites assessed. The final volume of waste material required for dedicated waste rock disposal is dependent on:

- Leach Pad construction requirements
- Pit backfill early access
- Cut-off grade used to determine mineable resources and subsequent waste quantities

The selected waste dumps for the PEA were designed in the area southwest and northwest of the ultimate mine pit to accommodate approximately 32.7Mbcm of waste rock. A total of 23.2Mbcm of waste are to be placed on the northern dump and 9.5Mbcm on the southern dump. The dumps will be constructed in 10 m lifts with an overall slope angle of 17-19 degrees.

In areas where the topography is unfavorable making bottom up construction too difficult, the dumps are built by end dumping from higher lifts and re-profiled to their final configuration at closure.

The northern dump will be constructed so that the top of the dump will be at lower elevation (~470mRL) than the pit entrance (~500mRL). The ramp will be dual lane and all waste haulage will effectively be downhill from the pit edge to dump edge at elevation ~430mRL. The section of the dump above 430mRL would be filled only if additional space is required. At the entrance of the waste dumps, a single dual lane ramp with a 10% gradient will be developed.

The waste dump, located southwest of the pit will have a capacity of 9.5Mbcm. The proposed access route to southern dump follows the natural topography and contours so as to reduce the amount of cut and fill needed and consequently reduce both road failure potential and costs. This road was designed as a dual lane road with a 10% gradient.

It is estimated that approximately 5Mbcm of waste material could be used for pit backfill. Backfilling can significantly reduce the areas of land left in a disturbed state (post-closure), related closure rehabilitation costs (e.g. ongoing water management), and the safety issues associated with leaving an open pit. Where pit design is amenable, the open pit will be progressively backfilled with waste rock during mining.

The waste dump locations are illustrated in Figure 16-7. The waste dump construction and final landform are based on the following criteria:

- The swell factors utilized to calculate the placed material in the waste dump (i.e. excavated swell + compaction in the dump) was 30%.
- All waste dumps were designed with dual lane ramps of minimum 16 meters width and 10% gradient.
- Dump face heights of 10m in the final landform (may be higher during construction).
- Dump face angle is 37 degrees during construction and 20 degrees in the final landform.
- Berm width is 21.7m during construction and 7.5m in the final landform.
- A 50m wide berm every 5 benches (50m height).

Figure 16-8 below provides a schematic cross section through the dump profile.

Evaluation of the potential for Acid Rock Drainage (ARD) from the waste dumps was not evaluated as there is no waste rock characterization data available. A Metal Leaching / Acid Rock Drainage (ML/ARD) management procedure is detailed in Section 20.4.2.

Figure 16-7 Waste Dump Locations (NW, SW & In-pit Backfill)

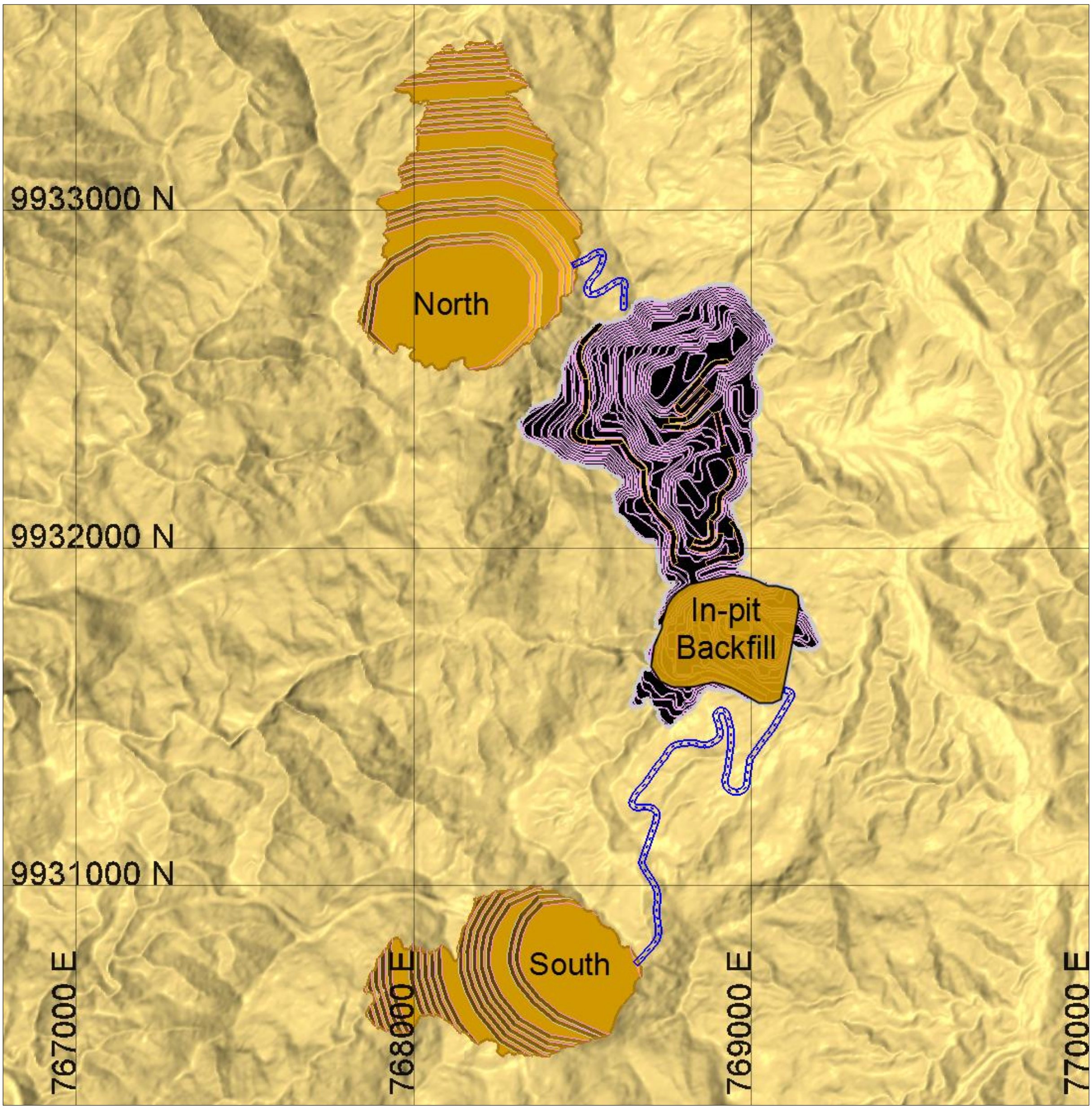
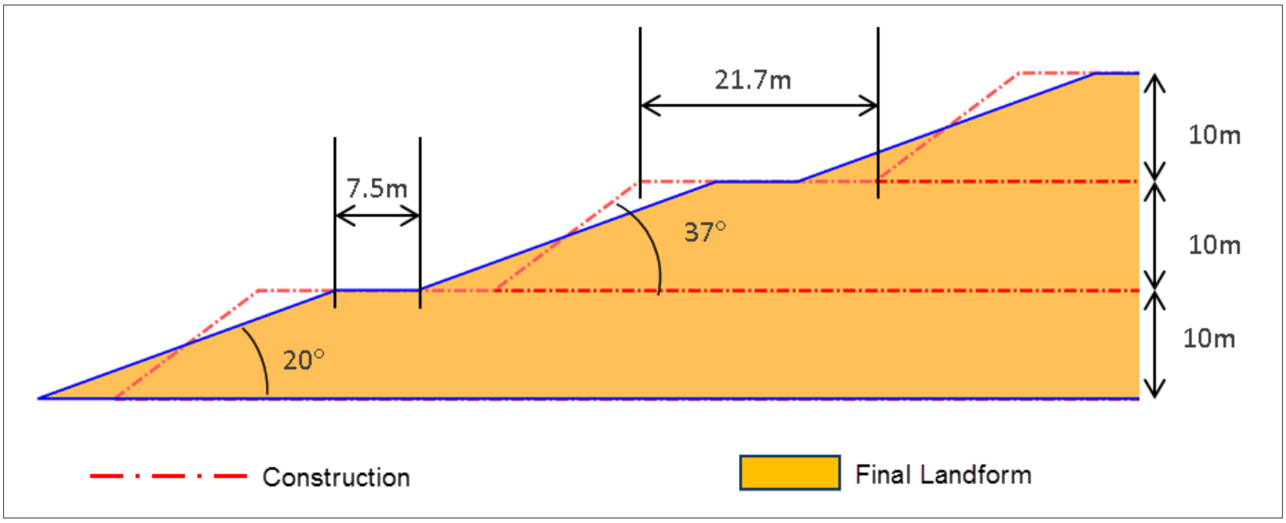


Figure 16-8 Schematic Cross-section of waste dump design



16.5 LIFE OF MINE SCHEDULE

16.5.1 Strategic Mine Scheduling

The objective of strategic mine scheduling was to develop a high level mine plan that meets the business objectives of KSK within a set of operational parameters. This usually requires a trade-off between a number of conflicting requirements such as:

- Maximizing project cash-flow and / or Net Present Value (NPV)
- Maximizing resource utilization and/or mine life
- Minimizing operating costs / fleet size / pre-strip

These requirements can in turn be constrained by various factors such as:

- Mining capacity
- Mill throughput restrictions
- Blending or product quality requirements
- Stockpile limitations

Strategic scheduling was undertaken using Maptek Evolution® software (EVO) which uses evolutionary algorithms to determine the solution that balances these drivers and returns the highest DCF. The schedules were generated on an annual calendar basis with the aim to identify the maximum value schedule from a number of different project options (scenarios).

EVO has distinct modules, each one with a specific purpose. The module utilized in this study was Strategy, which has the following characteristics:

- Generates schedules maximizing project DCF
- Keeps all the constraints within specified ranges
- Optimizes the stage development sequence
- Optimizes cut-off grade per period

The primary objective of cut-off grade optimization is to apply a variable cut-off grade over time to increase the NPV of the project by leach feed material at different cut off grades period by period.

EVO was used to define robust solutions for each option selected from 1000's of feasible schedules. Rather than use a simple bench average, EVO utilizes the grade tonnage curve for each bench by stage to optimize the cut-off grade and pushback mining sequence, and thereby improve DCF. The evolutionary algorithms inside EVO quickly generate a range of feasible schedules over a number of generations to find the best fit solution. Each generation commences with 200 unique schedules, following which only the best schedules are selected as the basis for the subsequent generations, eventually converging on an optimal solution from a number of directions.

By assessing multiple schedule options in EVO, that apply different trade-offs for the desired objectives, a best fit solution was ultimately identified.

16.5.2 Input Parameters and Assumptions

To facilitate scheduling, the original sub-block model was reblocked into a 25x25x10m regularized block model containing proportions of leach feed and waste so that no further dilution was applied. This was done before coding the model with stages. The leach feed material within the model was defined using Indicated and Inferred resources and Cu Leach grade greater than or equal to 0.09%.

The terrain at the BKM site does not lend itself to the formation of low grade stockpiles for later reclamation, therefore, during the scheduling and cut-off grade optimization process, any low grade material with a grade below the calculated cut-off grade for each period will be discarded as waste.

The pit optimization process illustrated that the sequencing was more important than the final pit shell, and therefore to develop a more robust strategic schedule for the BKM Project, the stage designs defined in Section 16.4.4 were used in this process.

The schedule was developed to keep open the multiple working areas as much as possible during each period in order to provide higher Cu grade and satisfy leach pad feed requirements.

The 100-120t excavator productivity was assumed to be 4.5Mtpa based on 5000 operating hours at 900t/op.hr. Total mining production was limited to four excavators based on the size of the benches and available working area.

A nominal production start-up year of 2019 was selected as being realistic and achievable for the project, providing 1½ years for project financing and detailed engineering, with a 1 to 1½ year timeframe for construction inclusive of some overlap with engineering and procurement.

16.5.3 Scheduling Targets and Constraints

The key drivers for the strategic mine scheduling were:

- Copper cathode production rate of 25ktpa
- Plant feed quality based on leachable Cu grade
- Financial parameters inputs as per the pit optimization (Table 16-3)

The objective was to produce a realistic and practical schedule that maximizes the project NPV by:

- Minimizing the pre-stripping period
- Deferring waste mining wherever possible
- Targeting higher feed grade early in the project

The following constraints were applied:

- No stockpiles

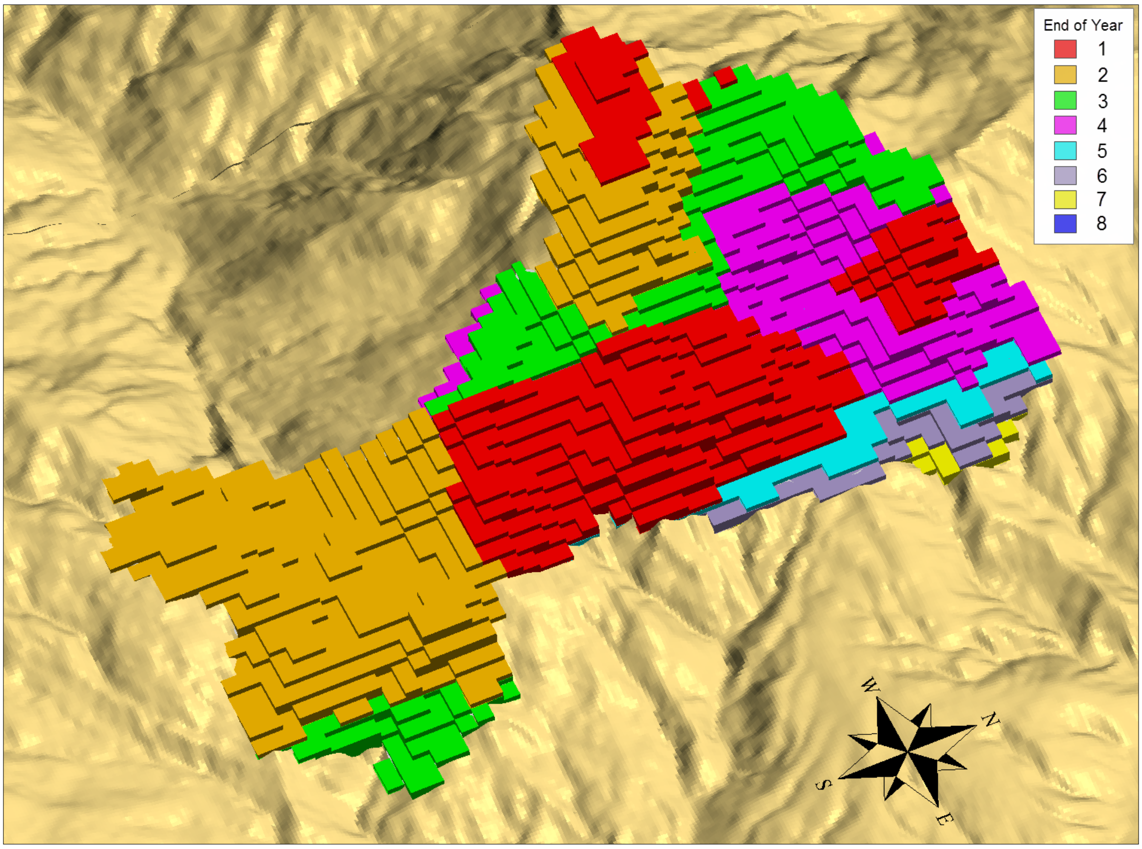
- Bench turnover limit of 60m (12 x 5m blast benches)
- Minimum 0.09% Cu (leachable) grade was applied
- Annual Mining limit of 18.0Mt

16.5.4 Final Results

An optimized mine plan was developed for the life of the deposit that meets the production targets. The pit has been divided into several stages/phases which will be mined simultaneously. These stages are wide enough to provide sufficient working space for the mine equipment. The pushback mining technique was used to balance leach feed and mining production rate, stripping waste material in the following stage while mineralized material is being mined in the current stage.

In order to facilitate in-pit backfilling of waste, mining begins within the high grade and low strip central pit area, and progresses towards to the southern end of the pit. As the pit floor is exposed in southern end of the pit, the waste backfilling process is commenced. During the initial stages of mine development, mining will be focused on exposing leach feed material through opening up multiple faces within more than one stage, located at the central and southern section of the ultimate pit design as shown in Figure 16-9.

Figure 16-9 Period Plans – Plan View



Stage 1 will be mined over a period of two years. This would expose the high grade mineralized zone with minimal waste stripping. During the first year pre-stripping will occur in stage 4, 5 and 7. In Year 2, Stage 2 is stripped to expose high grade material in Stage 4. The waste strip ratio will increase from 0.7 to 1.6 as mining operations advance from Year 1 to Year 4. Ongoing mining operations will occur in multiple stages each period as shown in Figure 16-10.

Figure 16-10 Pit Phases progression over LoM

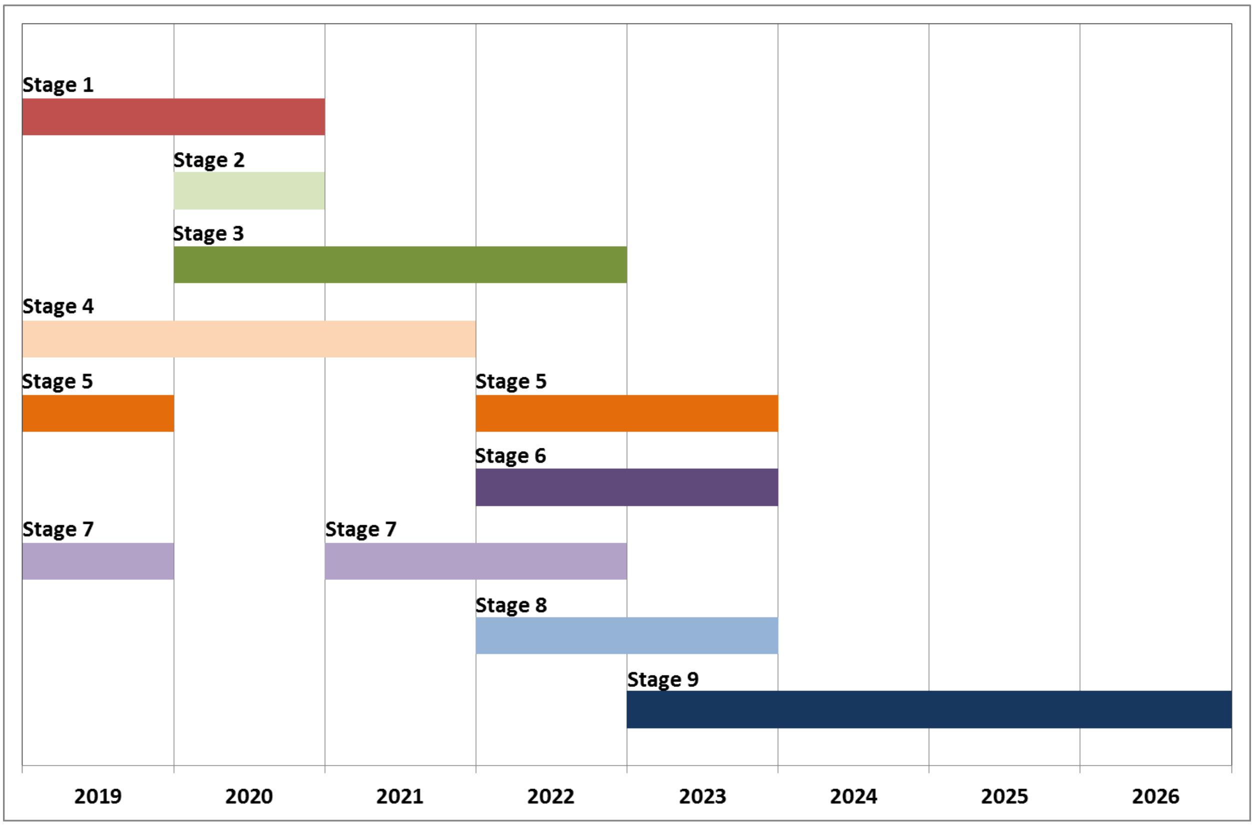


Figure 16-11 to Figure 16-18 show the resultant end of year face positions for BKM Project from year 1 to end of scheduled inventory at mid-year 8.

Figure 16-11 Mine Layout at end of Year 1

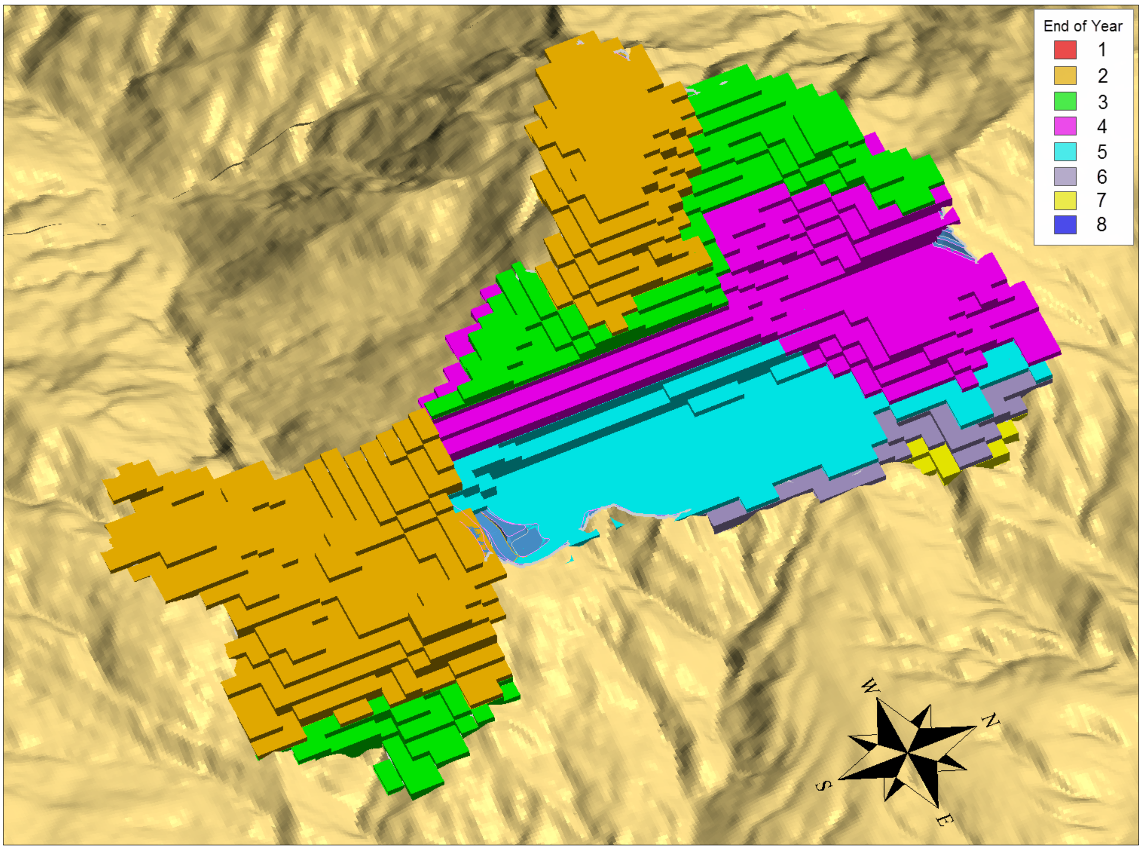


Figure 16-12 Mine Layout at end of Year 2

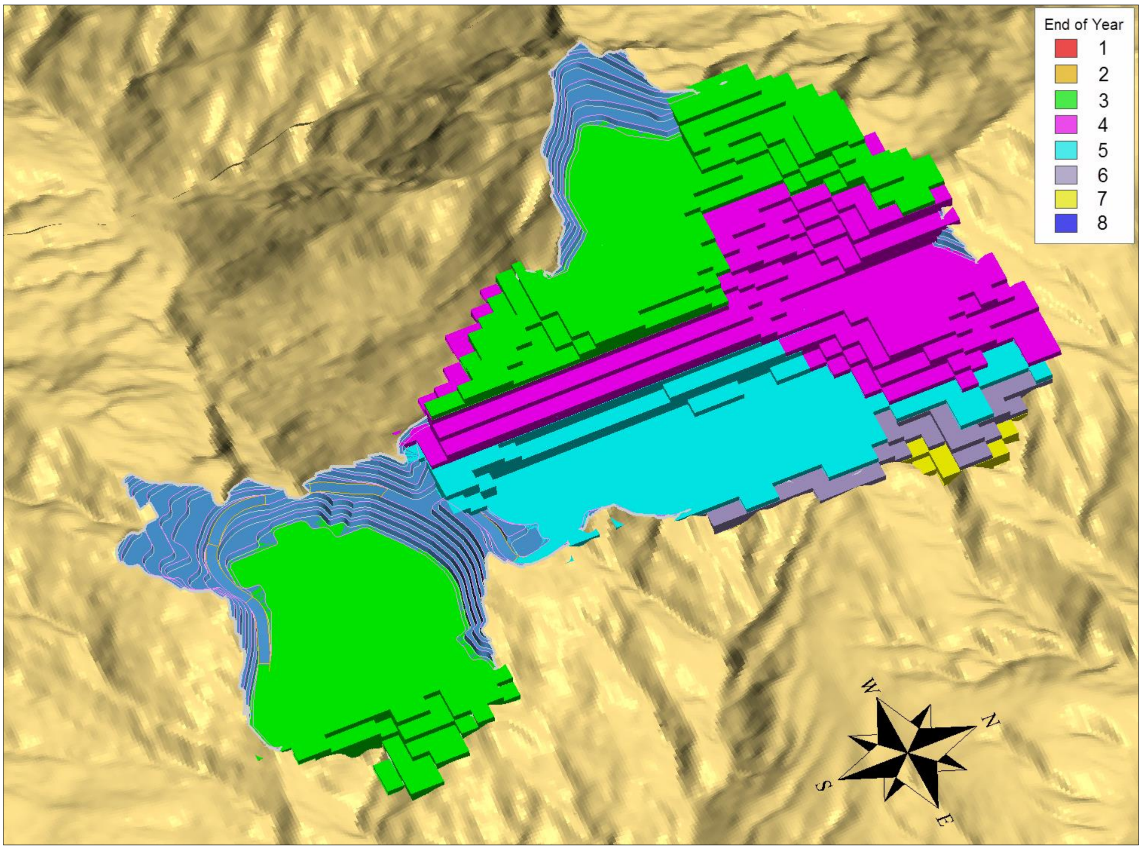


Figure 16-13 Mine Layout at end of Year 3

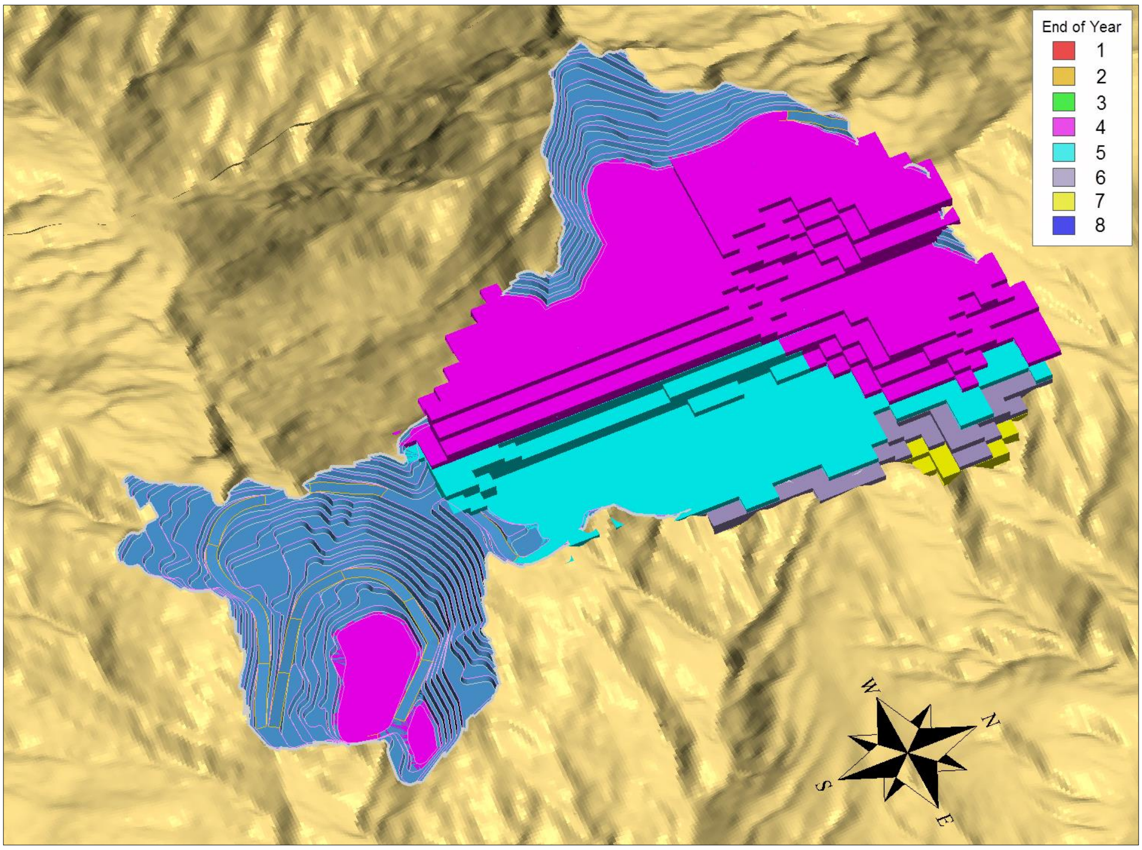


Figure 16-14 Mine Layout at end of Year 4

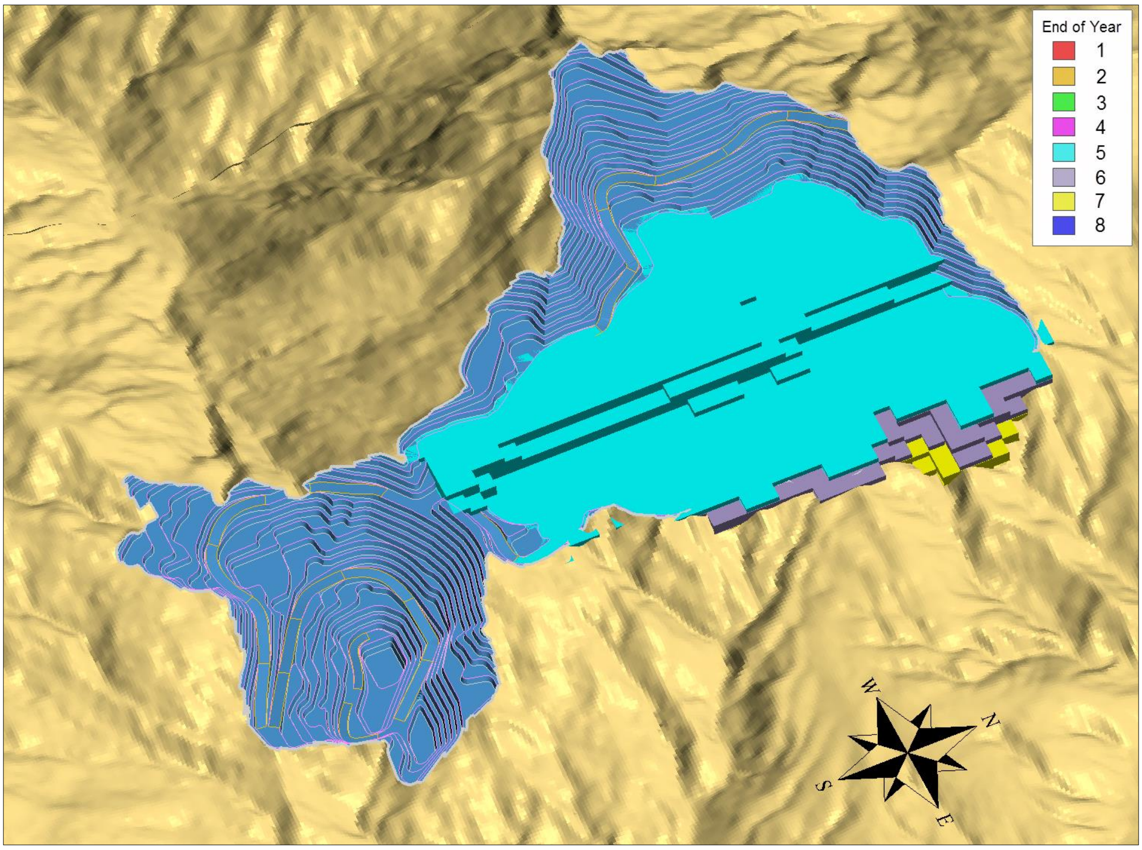


Figure 16-15 Mine Layout at end of Year 5

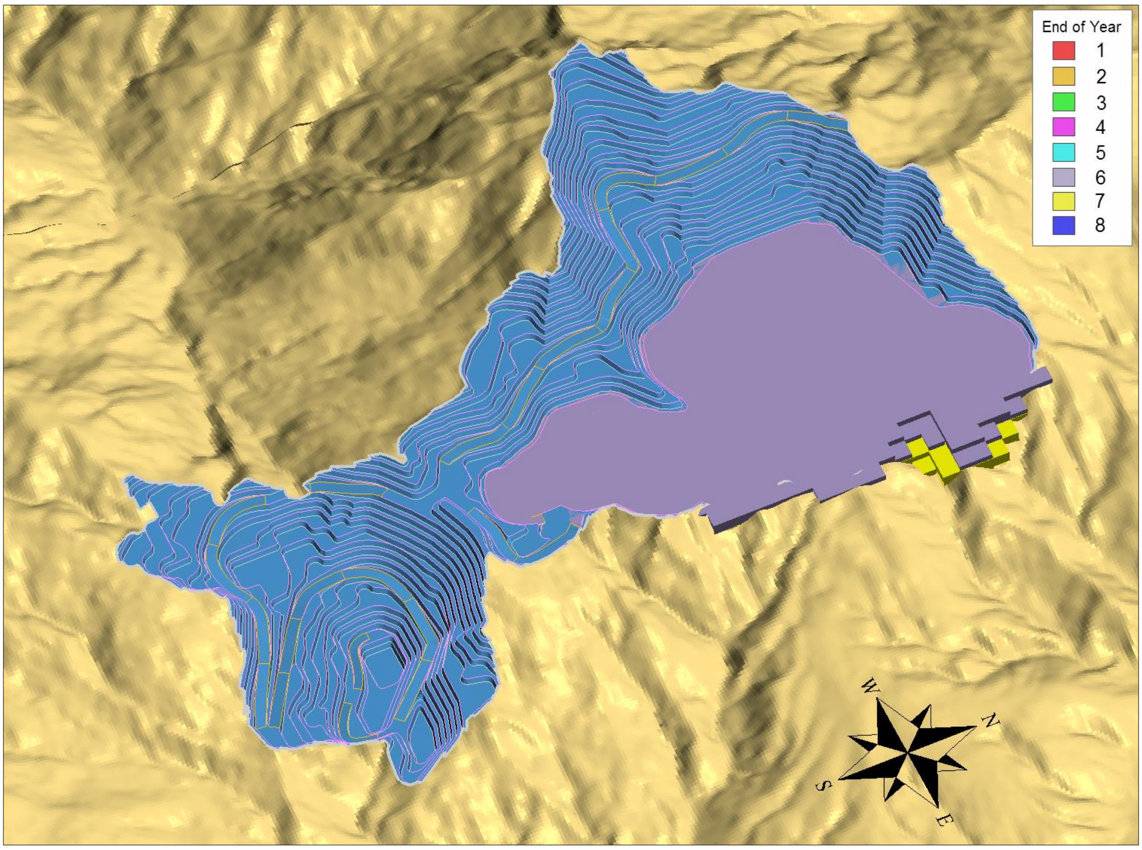


Figure 16-16 Mine Layout at end of Year 6

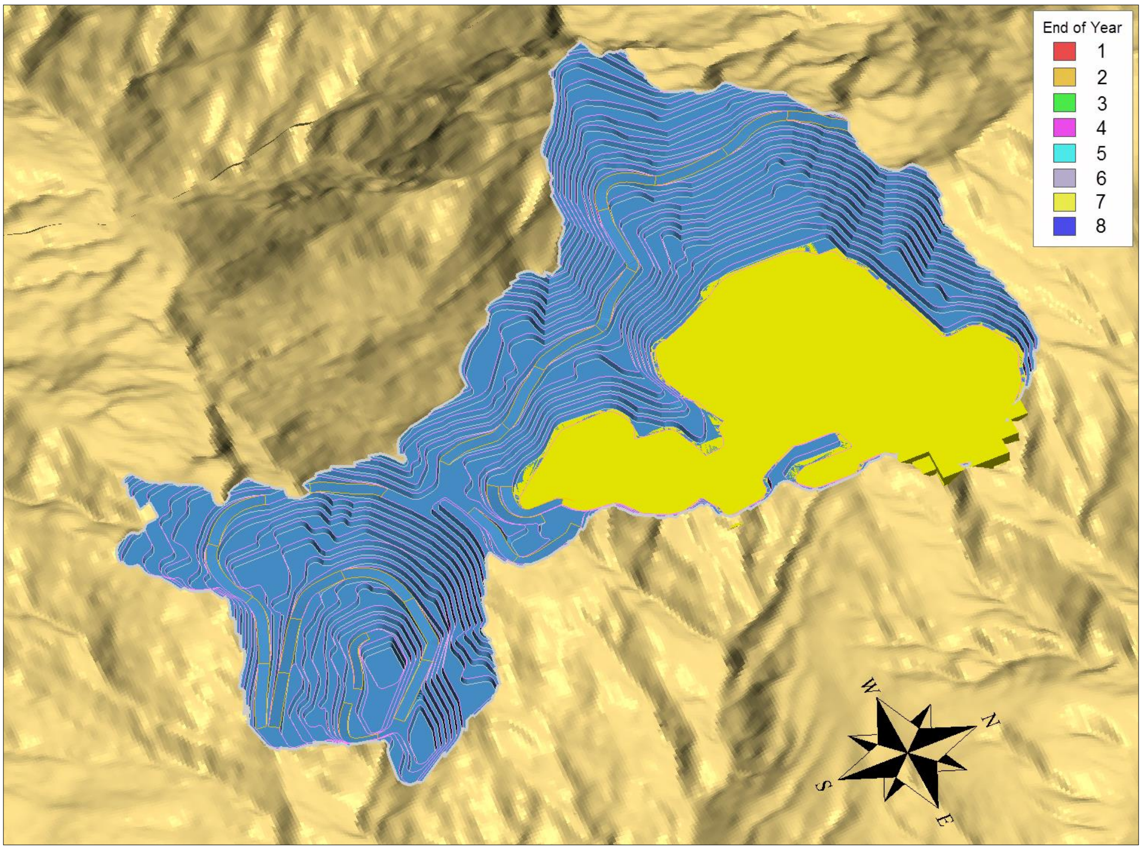


Figure 16-17 Mine Layout at end of Year 7

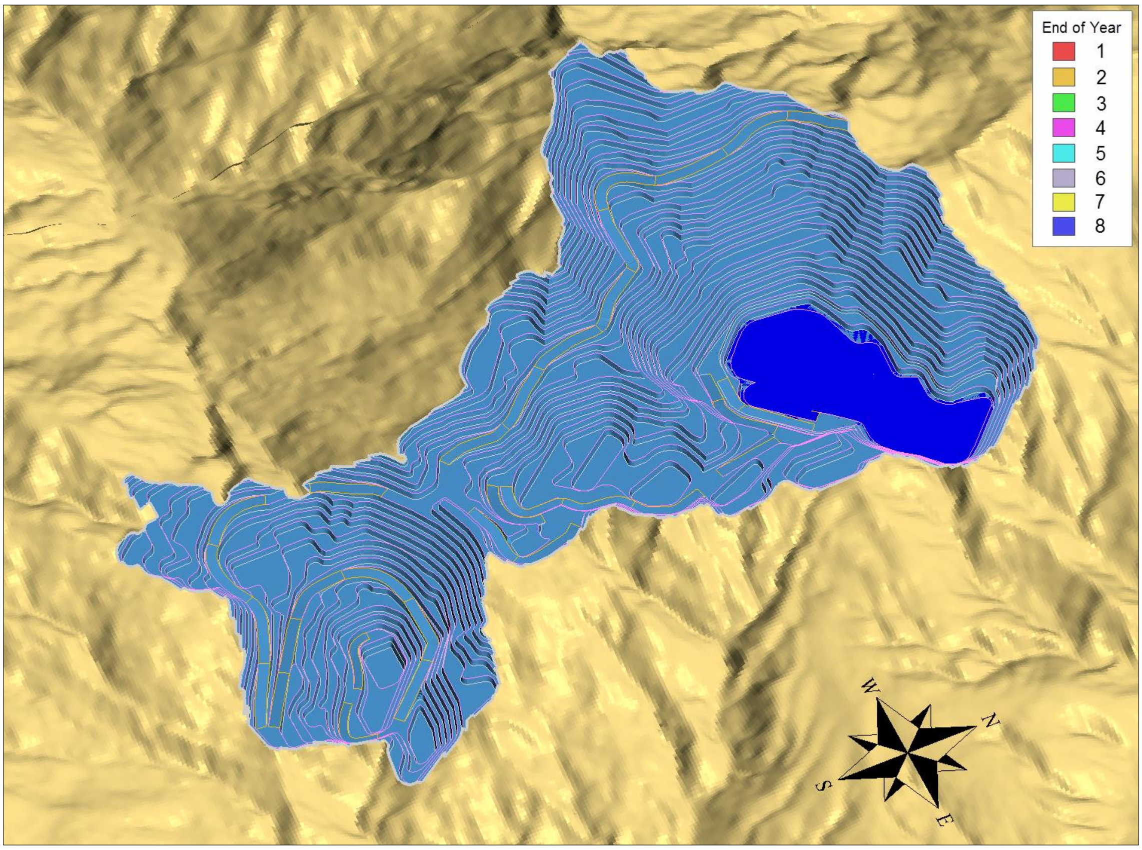


Figure 16-18 Mine Layout at end of Year 8

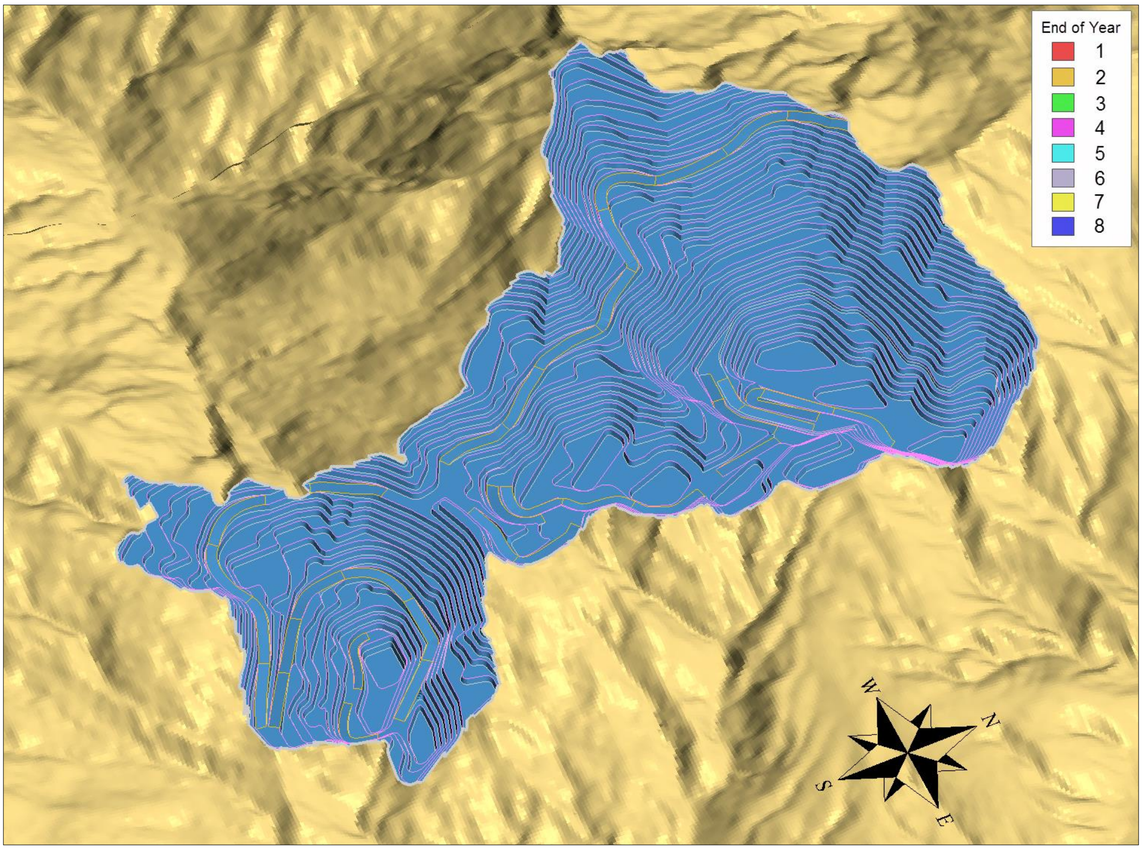


Figure 16-19 shows the Total Material Movement (TMM) by period for the mining schedule.

Figure 16-19 TMM Schedule

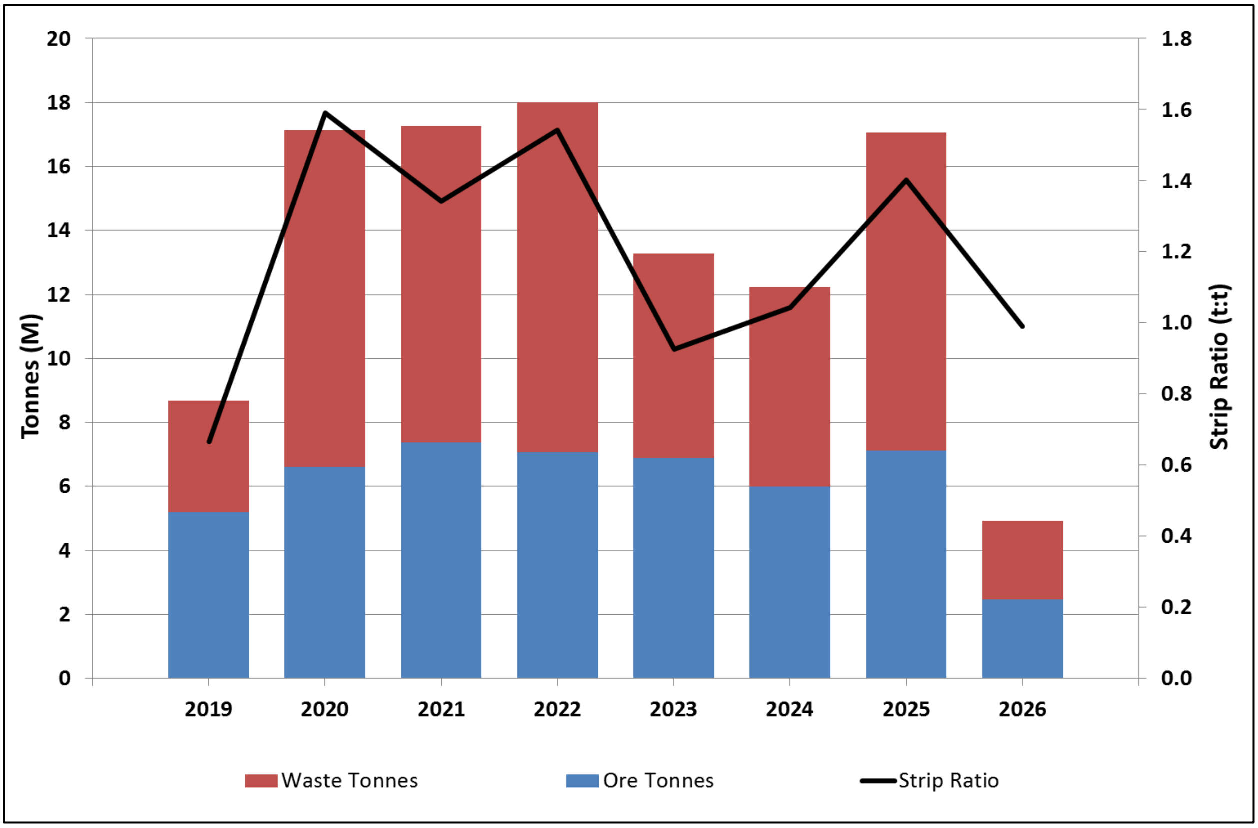
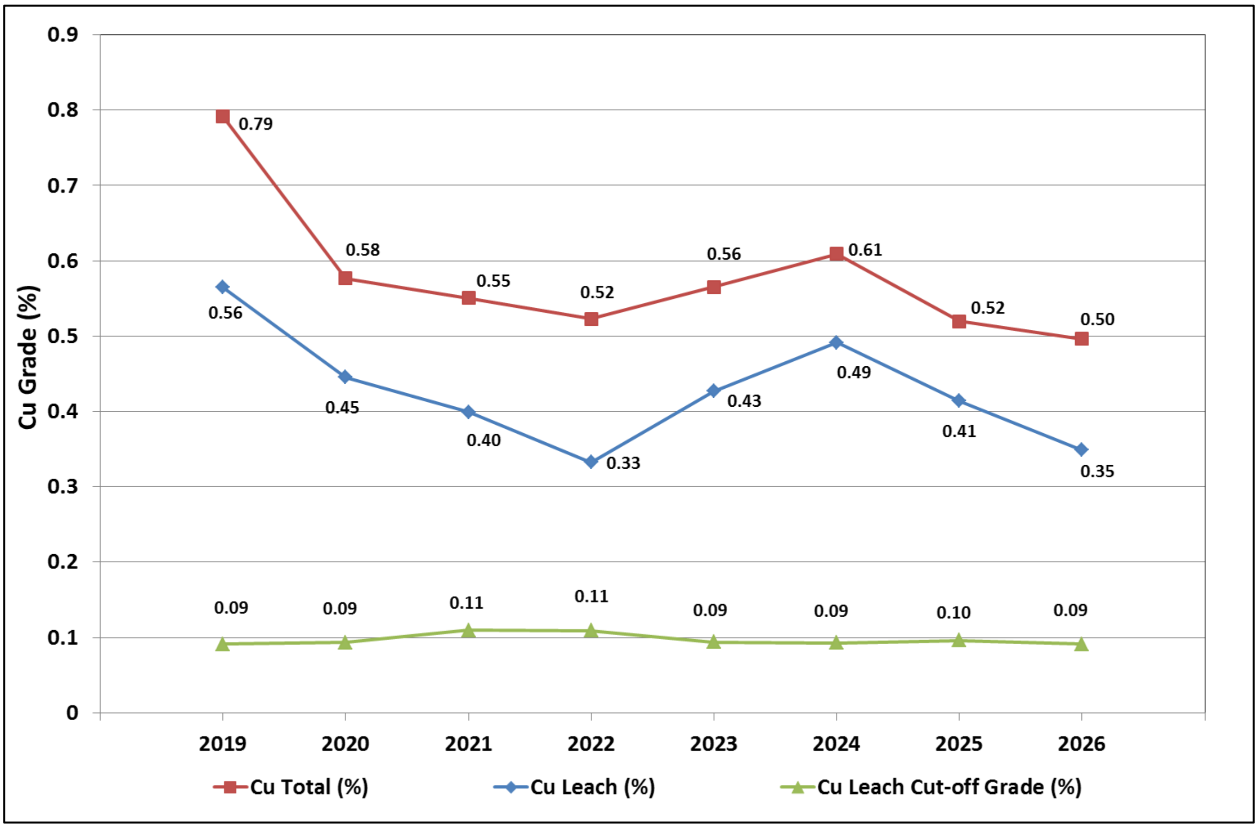


Figure 16-20 below shows the Cu Leach Cut-off grade applied per period and total Cu and Cu (leachable) grade variation on annual basis.

Figure 16-20 Cu grade (Total and Leachable)



The TMM, grades and Cu cathodes production have been summarized in Table 16-15. Cathode production utilizes the Assumed Leaching Response Curve discussed in Section 13.9 and displayed in Figure 13-11. On the basis of this curve, of the recoverable metal delivered to the heap in any given year, 63.3% of it is leached and produced as cathode in that year, and 36.7% in the following year. As such the copper cathode production will continue after mining has ceased as shown in Table 16-15.

Table 16-15 Production Schedule Summary

Item	Unit	Totals	2019	2020	2021	2022	2023	2024	2025	2026	2027
Mined Tonnes	Mt	108.6	8.7	17.1	17.3	18.0	13.3	12.2	17.1	4.9	
Mill-Feed Tonnes	Mt	48.7	5.2	6.6	7.4	7.1	6.9	6.0	7.1	2.5	
Waste Tonnes	Mt	59.8	3.5	10.5	9.9	10.9	6.4	6.2	10.0	2.5	
Strip Ratio	(Waste t : Ore t)	1.23	0.67	1.59	1.34	1.54	0.93	1.04	1.40	0.99	
Cu Total	%	0.58	0.79	0.58	0.55	0.52	0.56	0.61	0.52	0.50	
Cu Leach	%	0.50	0.66	0.52	0.47	0.39	0.50	0.58	0.49	0.41	
Cu Leach Recovery	%	85	85	85	85	85	85	85	85	85	
Cu Leach COG	%		0.09	0.09	0.11	0.11	0.09	0.09	0.10	0.09	
Insitu Metal	kt	208.6	29.4	29.4	29.4	23.5	29.4	29.4	29.4	8.6	
Recovered Metal Delivered	kt	177.3	25.0	25.0	25.0	20.0	25.0	25.0	25.0	7.0	
Copper Cathode Produced	kt	177.3	15.8	25.0	25.0	21.8	23.2	25.0	25.0	13.8	2.7
	lb. M	391.0	34.9	55.1	55.1	48.1	51.1	55.1	55.1	30.5	5.9

16.6 MINE OPERATIONS

The proposed Project will be a conventional open-pit operation with approximately 109Mt of material mined over the LoM. It is assumed that all mining activities will be undertaken using mining contractors. While owner mining may be an option in the future, this option was not considered at this time.

16.6.1 Clearing and Soil Stockpiles

The removal of topsoil and clearing of native flora will be carried out by dozers, front end loader and dump trucks.

Initial topsoil stripping will focus on mine start-up areas within the pits and selected areas inside the waste dump footprint. Progressive topsoil stripping will occur as new areas are developed. The top 300 mm of topsoil will be stripped and stored in stockpiles for future site rehabilitation.

The native flora will be stockpiled and used for site rehabilitation either progressively or after mine closure.

16.6.2 Drill and Blast

This section of the report considers the design of the drill and blast method within the mining process. At this time there is little data to determine drill and blast parameters. The comments and assumptions below are conceptual in nature and based on industry experience. It is assumed that all mined material below weathered zone will require drilling and blasting.

16.6.2.a Drill Design and Selection

Drill pattern design and drill type selection is a function of the rock type, rock strength, strata dip, joint spacing and frequency, bedding thickness and required production profile, no one of which were known at the time of this study.

The proposed drill specifications used in this assessment were:

- Drilling holes of 102 mm to 115 mm
- Drilling holes up to 30 degrees from the vertical
- Drill depth range 5 -10 m

16.6.2.b Blast design and explosive selection

The following is a summary of the blast design criteria:

- Powder factor (PF) of approximately 0.65 kg/m3
- Bulk explosive will be pumped emulsion blends
- Pre-splitting of interim or final walls.
- Cushion or trim blasting patterns adjacent to interim or final design batters
- Blasts heave and throw control to minimize feed contamination and sterilization
- Implementation of Maximum Instantaneous Charge (MIC) to control excessive ground vibrations on interim and final walls

Preliminary calculations for generic blast design patterns are shown in Table 16-16.

Table 16-16 Production Blast Parameters

Blasting Parameter	Value	Unit
Spacing	3.5	m
Burden	4	m
Bench Height	5	m
Rock density	2.74	t/m^3
Hole depth	5.7	m
Subdrill	0.7	m
Hole diameter	102	mm
Charge weight per hole	45.5	kg exp/hole
PF	0.65	kg/m^3
Explosive density	1.25	t/m^3
Stemming	1.3	m
Blasted Material	70.4	m^3/hole

16.6.2.c Wall control blasting

As the BKM open pit operation will reach depths of over 200 m, it will be important to implement wall control blast designs from the commencement of mining operations. A wide variety of options are available which could be implemented to reduce vibration and back break and improve overall wall control. These include:

- Cushion blasting
- Blast vibration monitoring ground and walls
- Maximum instantaneous charge size.

It is recommended that wall control systems are evaluated at the next phase of study.

16.6.3 Load and Haul

Diesel powered hydraulic backhoe excavators of 100-120t operating weight with heavy rock buckets of approximately 6m^3 capacity will be used to dig the blasted rock. The excavators will load the 40-tonne ADT dump trucks in a 4-6 pass loading.

Loaded trucks will haul material to the following destinations:

- All waste will be hauled directly to the nearest waste dump.
- All leach feed material will be hauled directly to the primary crusher and direct tipped into the hopper or placed on a stockpile located on the ROM pad.

16.6.4 Auxiliary Pit Services and Support Equipment

The primary mining operations will be supported by a fleet of support equipment. This make up of this fleet will be determined by the selected contractor but would generally consist of:

- Bulldozers of 40 tonne operating weight (Caterpillar D8 or similar)
- Motor graders of 20 tonne operating weight (Caterpillar 14M or similar)
- Water trucks of 20kL capacity
- Front-end loaders of 10 - 12 tonne rated payload (Caterpillar 988 or similar)
- Maintenance vehicles and service vehicles

The bulldozers will be utilized for floor control on the active faces and also for pushing tipheads on the waste rock dumps. The graders will be utilized for grading roads, working areas to reduce tire damage and blast areas before each drill campaign. The water carts will be used for dust suppression.

Maintenance and other service vehicles will be used for servicing and refueling of tracked equipment on the pit floors. Trucks will be serviced at the mine workshop and refueled at the fuel storage facility located in the mine laydown area.

16.6.5 Stockpile Management

The mine schedule has been modified to maximize the in-pit blending from the active faces being extracted and minimize re-handle of feed material from ROM stockpile. Only minor blending will be obtained from day to day management of the ROM stockpile as there is little space on the ROM for storage; due to the steep nature of the site. Overall less than 10% of the leach feed material is expected to be re-handled. Reclaim of stockpiled leach feed material will be conducted using the front-end loader.

16.6.6 Mine Layout

The mine design for the BKM consists of a conventional open pit layout with multiple entry access ramps. The dimensions of the designed pit are 1300m north to south and on average 600m from west to east. The main pit access is planned from the west of the deposit on the same side as the RoM pad. All leach feed material will exit the pit via this ramp. The waste will be placed into hillside waste dumps located northwest and southwest of the pit via dedicated ramps at the north and south of the pit respectively.

The area can be described as rolling to rugged. The Mine site is surrounded by rugged mountain country. Some of the minor creeks have formed deep gullies. The nature of the overlying topography will have considerable impact upon access road design and waste storage positioning. Careful study of the topography is required to ensure proper pit access points. Change in equipment size frequently is a cause of road modification, particularly width.

A general mine site layout has been developed that includes:

- the open pit area and waste rock dumps
- mining workshop and laydown area
- internal and external road network and ROM Pad
- Leach Pad
- SX/EW and administrative infrastructure
- Ponds and stormwater storage
- Accommodation camp

This layout design is illustrated in Figure 16-21. 3D visualizations of this layout are provided in Figure 16-22 and Figure 16-23.

Figure 16-21 Mine Site Layout

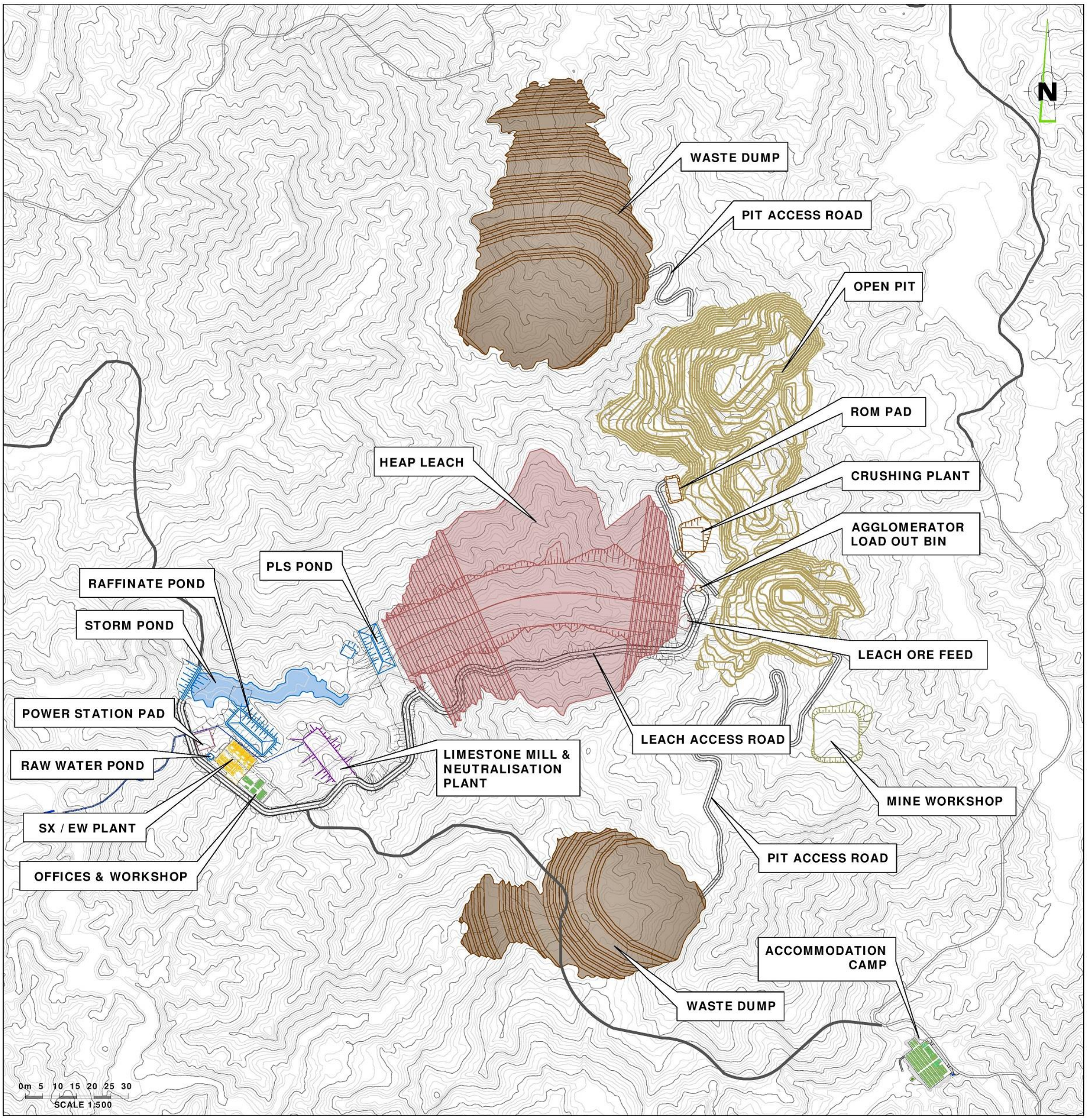


Figure 16-22 Mine Site Layout – Perspective View looking North

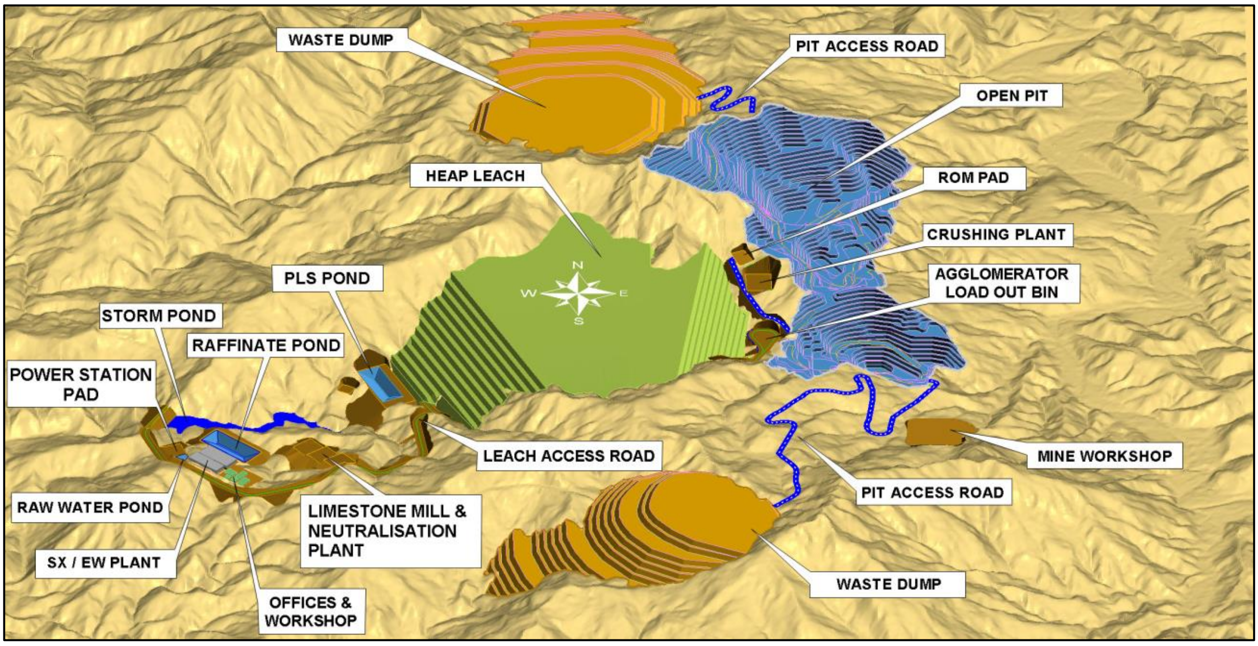
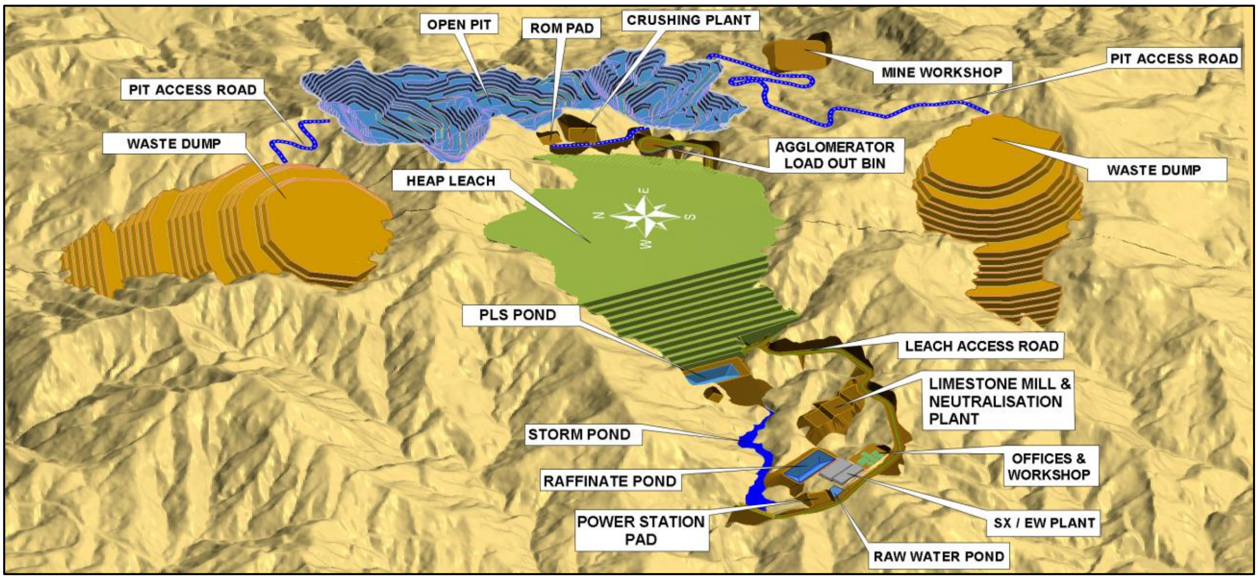


Figure 16-23 Mine Site Layout – Perspective View looking East



16.6.7 Surface Water Management

Key components of the water management system within the mine include:

- Implementation of bench drains within the pit and waste rock dump and storage (sedimentation) channels and ponds to manage surface water runoff and pit (de)water within the site in a controlled manner.
- Divert clean water around mine operation to prevent contamination by mining activities.
- Reducing the discharge of pollutants from the mine to the environment through use of oil traps in permanent maintenance areas and minimizing oil spillage during on-site fueling and maintenance.

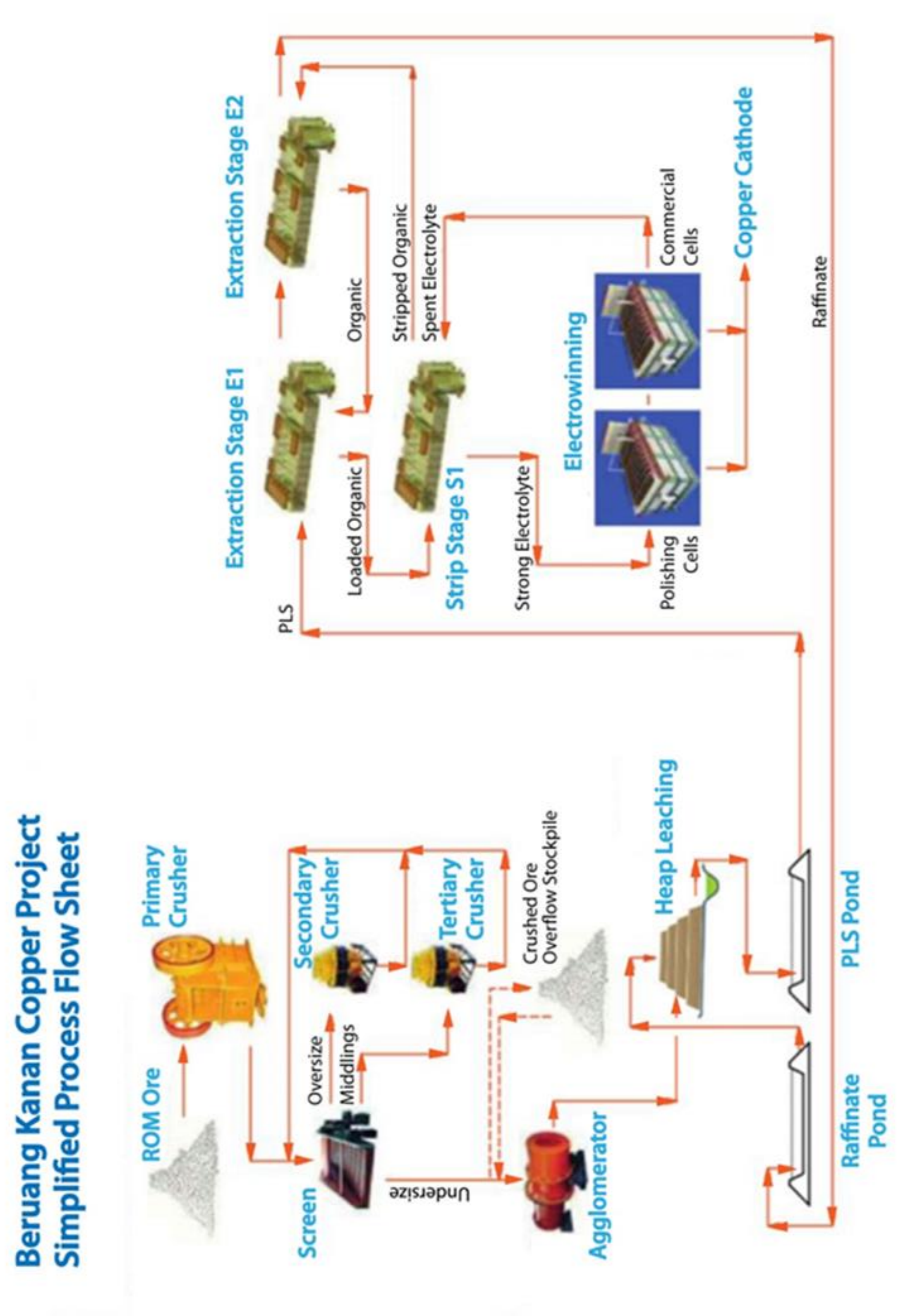
17 RECOVERY METHODS

17.1 INTRODUCTION

The overall processing scheme consists of the following assumptions:

- Crushing – Primary crushing to reduce the ROM LFI to a P80 of between 19 mm to 8 mm in a three-stage closed circuit crushing plant.
- Agglomeration – binding crushed LFI fines with larger fragments to form more uniform particles, which will assist stabilization of fines in the heap and also allow the intimate mixing of sulphuric acid and raffinate with the LFI.
- Stacking – placing of agglomerated crushed LFI on the heap leach pad incorporating transportation to a radial stacker (with stinger). This will allow agglomerated LFI to be stacked in heaps of nominal 6 m, but from 2 m to 8 m high.
- Heap Leaching – whereby copper is leached from a heap, or pad, of crushed LFI by leaching solutions percolating down through the heap and then collected from a sloping, impermeable liner below. The pad construction will be carried out in a single stage operation. Raffinate from the Solvent Extraction Plant (SX) will be used to irrigate LFI. Pregnant Leach Solution (PLS) containing copper as well as iron and other impurities will be delivered to the SX Plant.
- Solvent Extraction (SX) – whereby the PLS and electrolyte from the Electrowinning (EW) Plant are contacted with an organic extractant (LIX984NC or equivalent) that selectively transfers copper from the PLS to electrolyte; and increases its concentration in the process. Raffinate (PLS with low copper concentration) will be returned to the leach circuit.
- Electrowinning (EW) – whereby a direct electrical current is applied to the circulating electrolyte in cells containing multiple lead anodes and stainless steel cathode mother plates. Oxygen is formed at the anode and copper metal is plated on the stainless steel mother-plates. This copper deposit will be stripped from the cathode mother plates and bundled and strapped for sale.
- Reagent Storage, Preparation and Distribution – management of reagents such as sulphuric acid, diluent, extractant, salt, cobalt sulphate and guar etc.
- Plant Services – including a compressed air system, water reticulation, fire services and a hot water heating system.

Figure 17-1 Generalized Project Process Flow Sheet

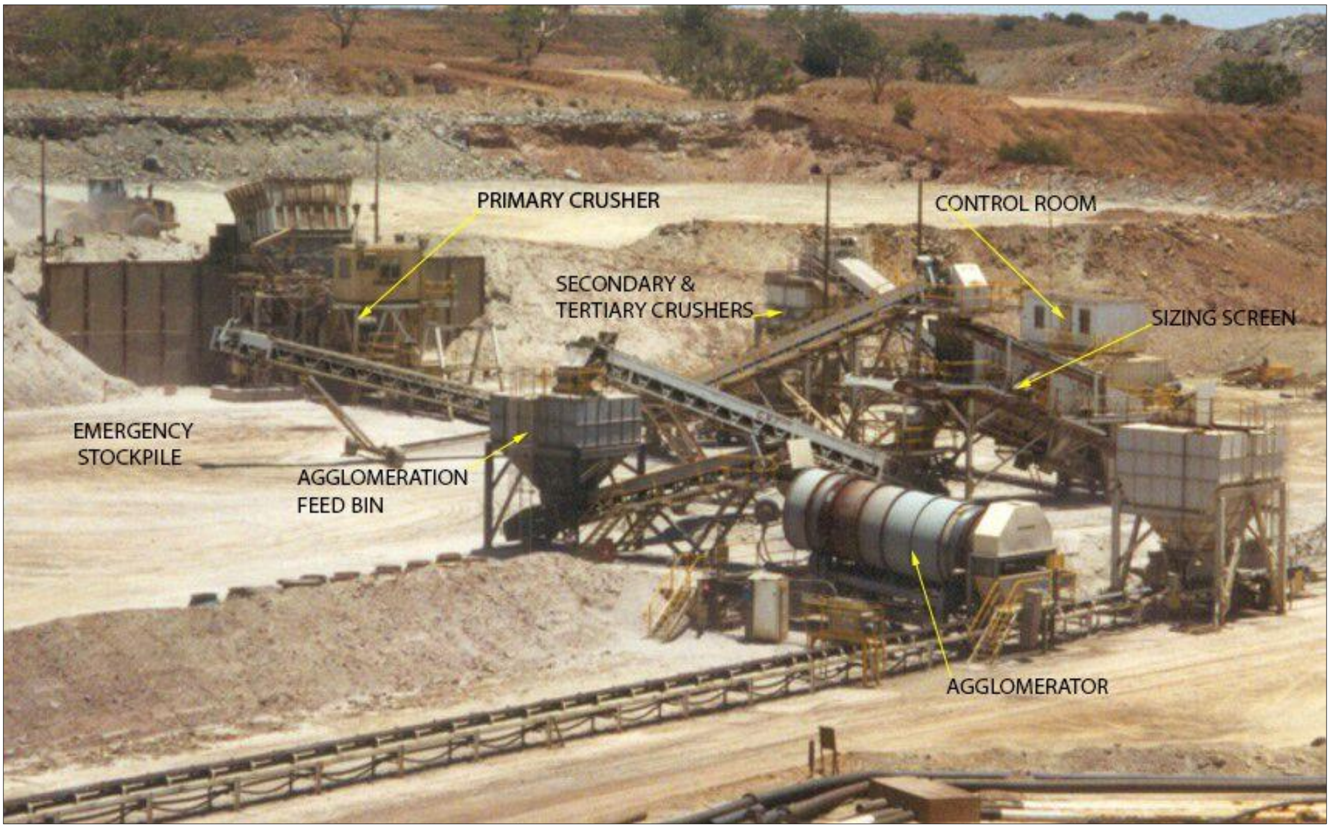


17.2 CRUSHING

Crushing, agglomeration and stacking will be carried out on a 24 hour per day basis. LFI will be hauled from the pit and dumped into the (run of mine) ROM dump hopper. No allowance has been made for significant blending of LFI material on the ROM stockpile as space and capacity is extremely limited. Some blending of LFI types will occur as part of the mining approach as a result of day to day management of the active faces being extracted. The following description should be read in conjunction with Figure 17-19 Crushing and Heap Leaching PFD.

ROM LFI is withdrawn from the dump hopper and primary crushed. The primary crushed LFI is conveyed to a double deck screen(s) to remove the material smaller than the required product size. Product will then be conveyed to the agglomerator feed bin. A bypass conveyor to a stockpile is provided to allow crushing plant operation to continue when the stacking system is not in operation. A reclaim hopper and conveyor are provided to allow operation of the agglomeration and stacking system when the crushing plant is not in operation.

Figure 17-2 Typical small crushing and agglomeration plant



Oversize from the top deck will be conveyed to the secondary crusher feed bin. Oversize from the bottom deck will be sent to the tertiary crushers' feed bins.

Material is withdrawn from the secondary and tertiary crusher feed bins and crushed. The crushed LFI is added to the screen feed conveyor for recovery of the correctly size product; and recycle of the oversize back to crushing.

Dust control is provided using fogging sprays at the dump hopper(s), enclosures (chutes, bins, screens and hoppers) and fine sprays at transfer points.

17.3 AGGLOMERATION AND STACKING

The following description should be read in conjunction with Figure 17-19 Crushing and Heap Leaching PFD.

Crushed LFI will be conveyed from the crushing plant to the agglomeration plant. The agglomerator will be fed at constant rate from a surge bin. Sulphuric acid and raffinate are added under ratio control from a weightometer on the agglomerator feed conveyor. The agglomerator will include a number of on-line measurement and control elements to provide good quality control of the agglomerates produced including.

- Feed mass and moisture measurement – These will be used to calculate the amount of moisture to be added to the LFI.
- Agglomerated LFI moisture / particle size distribution (PSD) / and surface reflectance – The PSD will be used to adjust the moisture addition rate; while surface reflectance will be used to detect an excess moisture addition.

Figure 17-3 Typical Agglomeration Plant (with sampling system)

Figure 17-4 High Quality Agglomerates with all fines bound to coarser particles.

Process acid tanks and dosing pumps will be installed adjacent to the agglomerator to ensure accurate acid addition to the LFI prior to stacking. Acid will be transferred to the leach circuit and EW Plant from these storage tanks according to process requirements.

Crushed and agglomerated LFI is stacked on the leach pad using a mobile stacker. The stacker is fed with trucks (due to the necessary flexibility needed in the steep terrain at BKM). Stacking is under the control of the local operator with no automatic systems other than emergency shut down on the materials handling train. The stacked height is selected by the process engineer and managed by the local operator; to a height of nominally 6 m (range 2 m to 8 m).

Figure 17-5 Stacker and dump hopper (ready for relocation)

17.4 HEAP LEACHING

The following description should be read in conjunction with Figure 17-19 Crushing and Heap Leaching PFD.

The heap leach pad design is based on single stage leaching of LFI. The “valley fill” style of operation is dictated by the steep topography (refer to Figure 17-6). This does not allow for a two stage leaching process to be effective. Due to the likely acid generation of the ore, there is no limit to economic leaching and it can continue when over-stacked with more LFI. In theory, the leach can continue until all the leachable copper is extracted. In practice there is a limit within the lifetime of the heap to the slow leaching final copper.

Leach solutions are applied with “Dripper” style irrigation systems in the wet season (refer to Figure 17-7) and “Wobbler” style irrigation in the dry season (refer to Figure 17-8). Wobblers will maximize evaporation of the water; and assist in the water balance management. During the wet season the drippers will be covered with a thin plastic HDPE (High Density Poly-Ethylene) sheet (“raincoat”) to shed rainwater away from the process to a separate diversion drain around the process site. The process flow for this is part of the Process Water Management flowsheet as shown in Figure 17-22.

The irrigation and raincoat are laid out as the pad is stacked. This allows the agglomerate to be placed under irrigation within two days of placement and leaching to begin within a few days. This keeps the agglomerates intact and assists with leaching kinetics.

Figure 17-6 Heap Leach Valley Fill Design

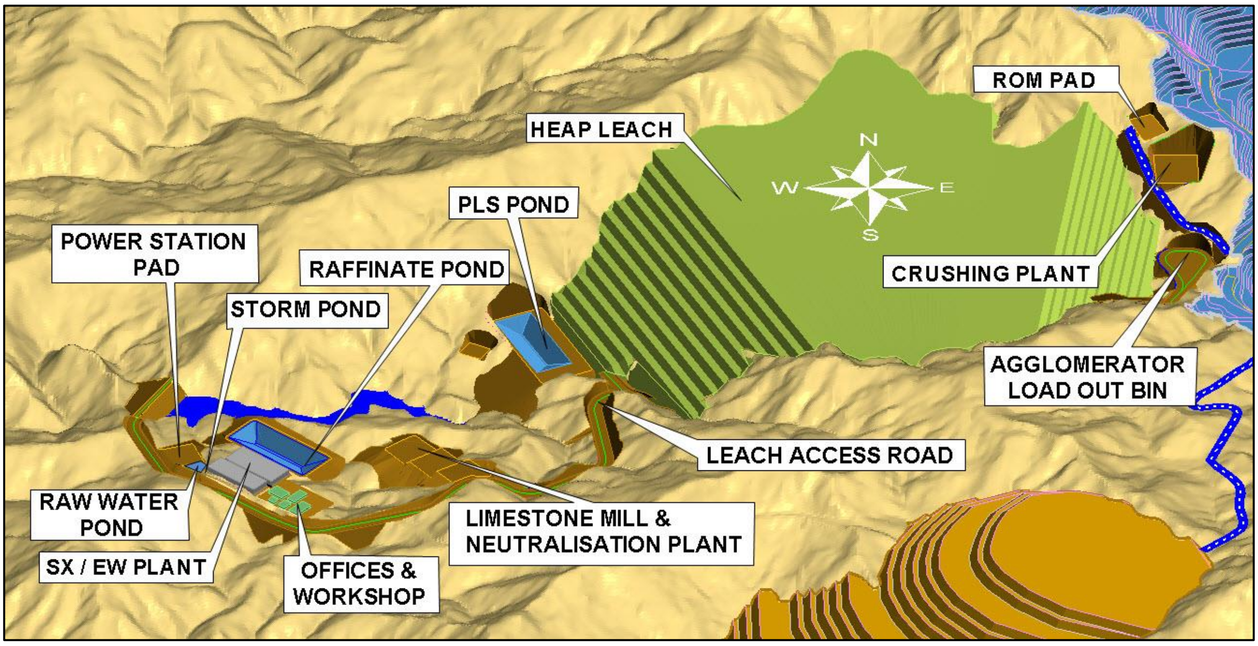


Figure 17-7 Dripper Irrigation

Figure 17-8 Wobbler Irrigation

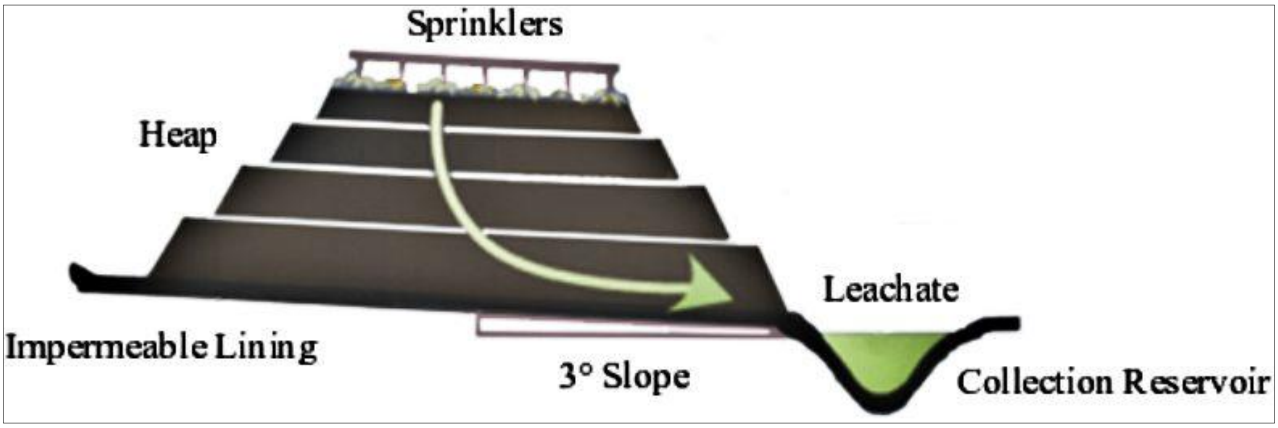
The irrigation system is installed by a dedicated crew of artisans and operators. On completion of leaching the main piping is recovered for re-use while the dripper line is left in place as recovery of this is not an economic proposition.

Figure 17-9 Stacker and Dozer Ripping the compacted surface

Leached LFI is drained for a period to allow the surface to dry out sufficiently to allow the stacker and trucks to operate successfully on the heaps top surface. The LFI will be acid producing, and it can be stacked without an intermediate low permeability barrier. This allows longer leaching times to be used on the over-stacked ore to gain improved final leach recovery. The compacted surface under the stacker wheels will be ripped by a small dozer to improve the through percolation; and the leaching of the over-stacked LFI.

Leach solution trickles through the agglomerated LFI dissolving copper and other elements. It is collected in a series of slotted drainage pipes, which direct it to the solution collection area in the downstream portion of the leach pad. From here the solution is piped to the collection drain. Due to the nature of the valley fill operation there is only single stage, over-stacked leaching; which produces a single, low grade pregnant liquor solution (PLS) for recovery of the copper.

Figure 17-10 Heap Leach Process Schematic

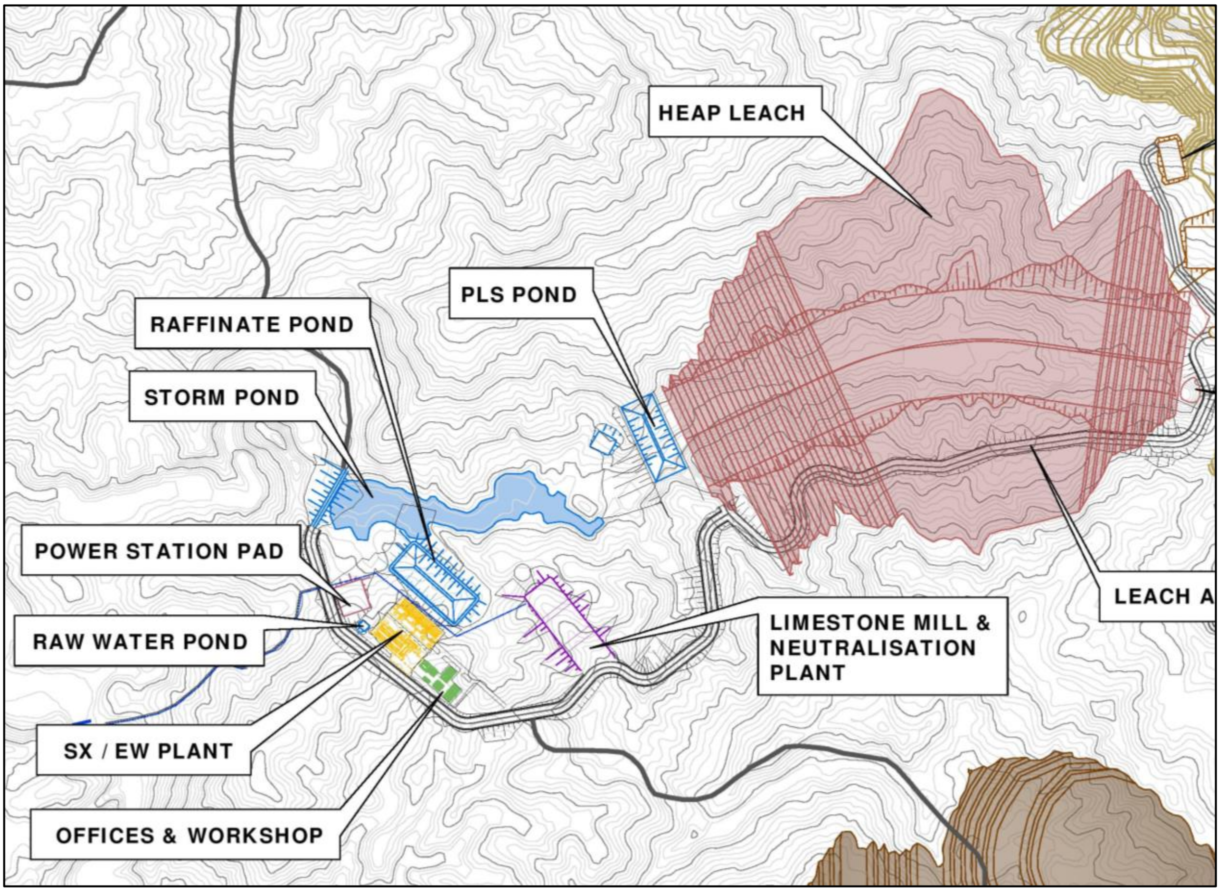


PLS then drains from the heap via the collection drains into the PLS pond where it is pumped with the PLS pump to the Solvent Extraction plant (SX) for recovery of the copper content. The copper depleted solution from the SX is referred to as raffinate. It has the acid content increased to the operating target as part of the SX plant control. The PLS pump is under the control of the SX plant.

Raffinate is collected in the raffinate pond and recirculated by the raffinate pump back onto the leach area under irrigation. The raffinate pump is under the control of the Central Control Room (CCR) operator who is in consultation with the leach field operator. Selection of which heap areas to irrigate with raffinate is under the control of the Heap Leach field operator.

Any overflow from the PLS pond or raffinate pond, due to wet weather or prolonged shut down, will be directed to the storm catchment pond (refer to Figure 17-11). The storm pond retains all the water and solutions from the design rain events for recovery of the valuable copper and reuse of the water. The excess contaminated solution will be released to the environment after passing through a Neutralization Plant. The storm water return pump is under the control of the CCR operator, who uses this supply as the first preference for make-up water into the leach solution system. Depending on the copper concentration it can be added to either the PLS or the raffinate ponds.

Figure 17-11 Overall Process Area Development Plan



The water balance of the heap is the most important parameter to control as part of the overall site environmental management system. Excess water will need to be disposed of by a number of potential methods:

- Neutralization of the acid and precipitation of metal ions before direct discharge to the surrounding water catchment.
- On site evaporation by use of Wobbler irrigation and 'water cannons' (refer to Figure 17-12). The water cannon units create a fine water aerosol that can evaporate large quantities of water. The units are in use in similar heap leach applications in many countries.

Figure 17-12 Water cannon in operation evaporating water.

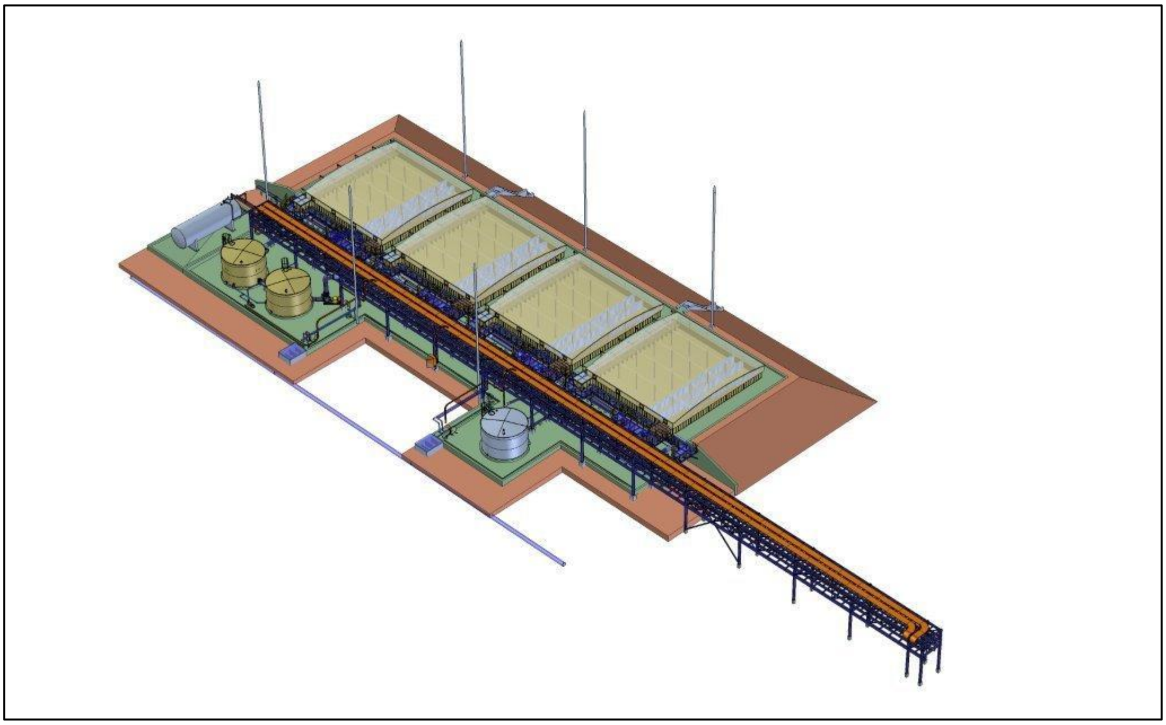
17.5 SOLVENT EXTRACTION

The following description should be read in conjunction with Figure 17-20 Solvent Extraction and Electrowinning PFD.

The objective of the Solvent Extraction plant (SX) is to transfer copper from the PLS solution to a solution (advance electrolyte) containing 50g/L Cu; while preventing the transfer of deleterious elements such as iron (Fe), manganese (Mn), and silica (SiO_2). To achieve this, a copper selective extractant is used.

The solvent extraction plant consists of a single train of extraction and stripping mixer-settlers. The train is designed to allow the bulk removal of copper in the extraction section. In the extraction section up to 92% of the copper is removed in two series extraction stages, plus one parallel extraction stage. All of the copper poor solution (raffinate) with the remaining 8% of the copper, at around 0.26 g/L, is recycled to the leach pads to be used for irrigation to enable further copper pick up.

Figure 17-13 Solvent Extraction Plant with Four Mixer-Settler Units



The minimum design capacity of the EW circuit is 25,000 tpa. Therefore the SX circuit is designed to be capable of transferring up to 25,000 tpa of copper to the EW facility when fed with appropriate PLS grades.

The SX process is described below and is shown pictorially in Figure 17-20.

- The SX will be fed by a single PLS pump supplying pregnant leach solution (PLS) from the PLS Pond.
- The PLS will be split into two streams. One stream will be fed to two series extraction stages (E1 & E2). The other portion of the PLS will be fed to a single parallel stage of extraction (E3p).
- The PLS is combined in the mixer of the first Extraction Stage (E1) with an organic reagent discharging from the second Extraction Stage (E2). The organic reagent has been partially loaded with copper in the E2 stage.
- Further copper is extracted in E1 onto the partially loaded organic reagent. The now Loaded Organic (LO) is then discharged from E1.
- In E2 further copper is extracted from the aqueous phase with partially loaded organic from E3p.
- In E3p copper is also extracted from the separate PLS feed aqueous phase by stripped organic from the strip stage S1.
- The E2 & E3p stages operate in an organic continuous mode so as to minimize the organic losses to the aqueous phase (raffinate).
- The aqueous phase from E2 and E3p reports to the raffinate pond as raffinate.

Due to the presence of silica in the PLS, the E1 stage will need to be operated in organic continuity to minimize the creation of floating crud, from the silica content. This can give uncontrolled transfer of PLS to the strip stage via the LO. The entrainment of PLS in the LO is high from E1. This will be removed in a specially designed coalescer where +90% will be recovered.

LO with low PLS entrainment is pumped by the LO pump to the strip stage (S1) where most of the copper is removed from the organic with lean electrolyte from the Electrowinning (EW) containing 35g/L Cu and 180g/L sulphuric acid. The stronger acid concentration strips the copper from the organic phase raising the aqueous phase overall to 50g/L Cu and reducing the sulphuric acid content to 157g/L. The aqueous phase leaving S1 is referred to as Advance Electrolyte (AE) and is gravity fed to the AE tank; while the organic advances to E3p.

AE will then be directed from the SX plant to the EW facility. The AE contains small amounts of entrained organic, solids and crud. If allowed to enter the EW circuit, the contaminants would affect the cathode quality and therefore multimedia filters are installed to remove entrained contaminants from the AE. The filters are comprised of a bed of coarse anthracite on top of a bed of fine garnet and the solution flows into the top of the filters and down through the beds, trapping any organics. Periodically the filters are drained of accumulated organic, air scoured and back washed (i.e. solution flowing upwards) to remove any trapped organic or solid material from the anthracite and garnet. The backwash solution is discharged to the backwash tank to recover the valuable electrolyte.

Figure 17-14 Multimedia Filters

Formation of crud in the SX plant is typically caused by the presence of suspended solids and silica in the PLS. Predicting the quantities of crud likely to be generated in a new circuit is difficult. However several measures will be put in place to manage crud, including:

- Use of a simple integrated crud removal system;
- Safe access to settlers to facilitate crud removal; and
- Separation of crud using a tri-canter centrifuge.

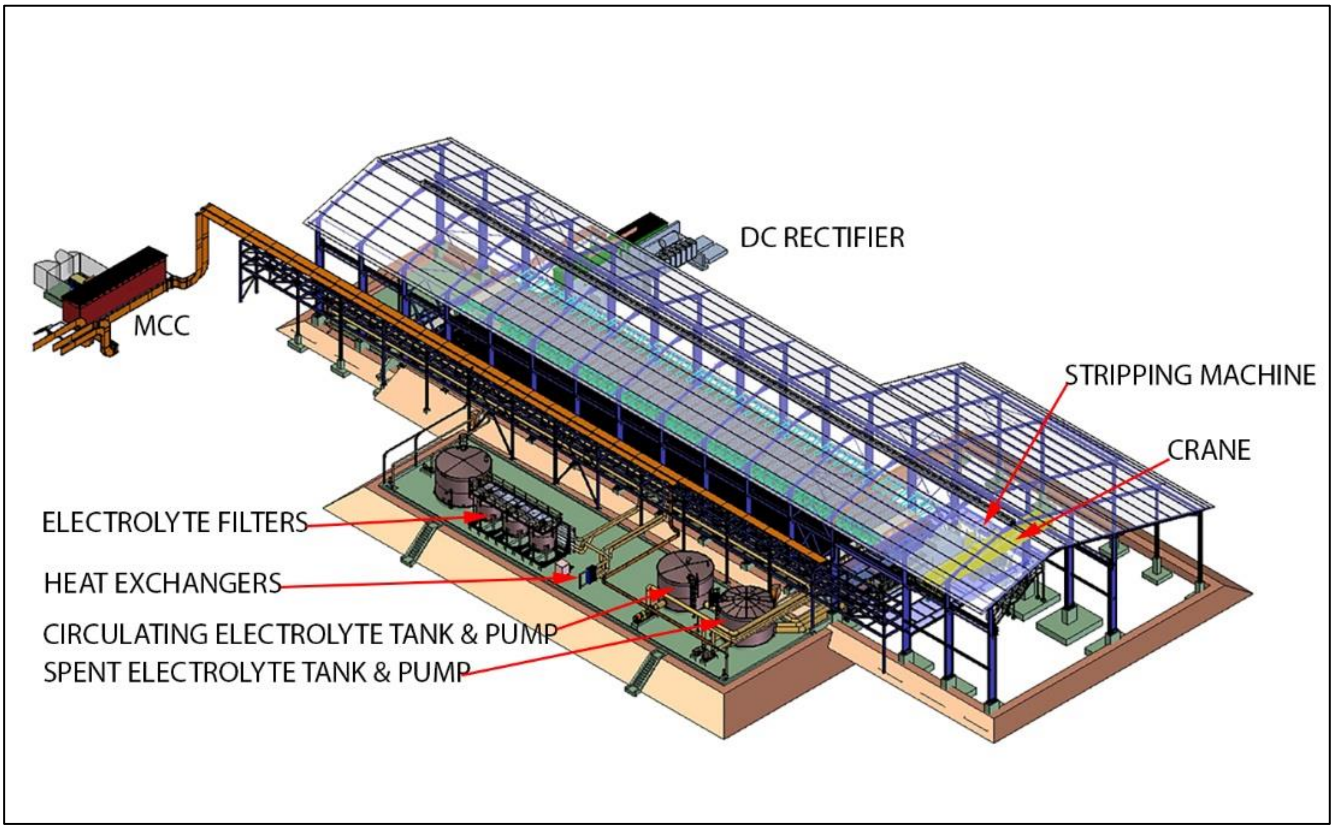
Regular clay treatment of the organic will be required to maintain the performance of the extractant. The crud tank and crud centrifuge will be used for this purpose.

17.6 ELECTROWINNING

The following description should be read in conjunction with Figure 17-20 Solvent Extraction and Electrowinning PFD.

The AE needs to be heated from the SX operating temperature of around 30 degrees, to the EW operating temperature of around 45 degrees. A recovery heat exchanger is used to recover heat from the hot, Spent Electrolyte (SE) being pumped to the solvent extraction circuit. During electrowinning the electrolyte is heated by energy provided by the rectifier that is not used in the actual deposition of copper. About 30% of the total rectifier energy is used in electrolyte heating. In addition to the heat recovery from the SE, the AE can be further heated using circulating hot water in a trimming heat exchanger.

Figure 17-15 Typical Electrowinning Plant Layout



The cleaned heated AE is circulated to polishing EW cells. These polishing cells help to remove the last traces of organic from the AE and protect the other commercial cells from organic breaking through the multimedia filter.

Polishing cell discharge flows to the circulating electrolyte tank where it is joined by the residual SE. The fresh circulating electrolyte is pumped (by the circulating electrolyte pump) to the remaining commercial cells for deposition of the copper. Overflow from the commercial cells is SE .

SE discharging from the commercial cells is returned to the SE tank. Excess SE flows into the circulating electrolyte tank for further copper addition via the stronger electrolyte returned from the polishing cell overflows.

Each EW cell contains 48 stainless steel cathodes and 49 lead anodes. Copper is deposited on the cathodes electrolytically.

A small stream is taken out of the SE flow from the backwash return pump. This stream is known as the "Iron Bleed" and is used to control the build-up of iron and other impurities in the EW circuit. The volume of electrolyte lost in the iron bleed is made up by potable water. Concentrated sulphuric acid (H_2SO_4) is added to the SE to maintain an acid level of 180 g/l to compensate for the acid loss caused by the iron bleed.

The copper growth cycle is generally between six and eight days with an average of seven days. Operation at high current density will give a shorter growth cycle while operating at low current density will give a longer growth cycle.

Figure 17-16 Internal View of Electrowinning Plant

Cobalt sulphate and salt solution will be added to the electrolyte in order to maintain a concentration of up to 200 ppm cobalt. This helps to protect the anodes from corrosion. Polyacrylamide solution will also be added to the electrolyte where it acts as a growth modifier helping to create dense flat deposits of copper. Potable quality water will be added to the spent electrolyte tank to make up for the iron control bleed and the water lost in production of oxygen at the anode.

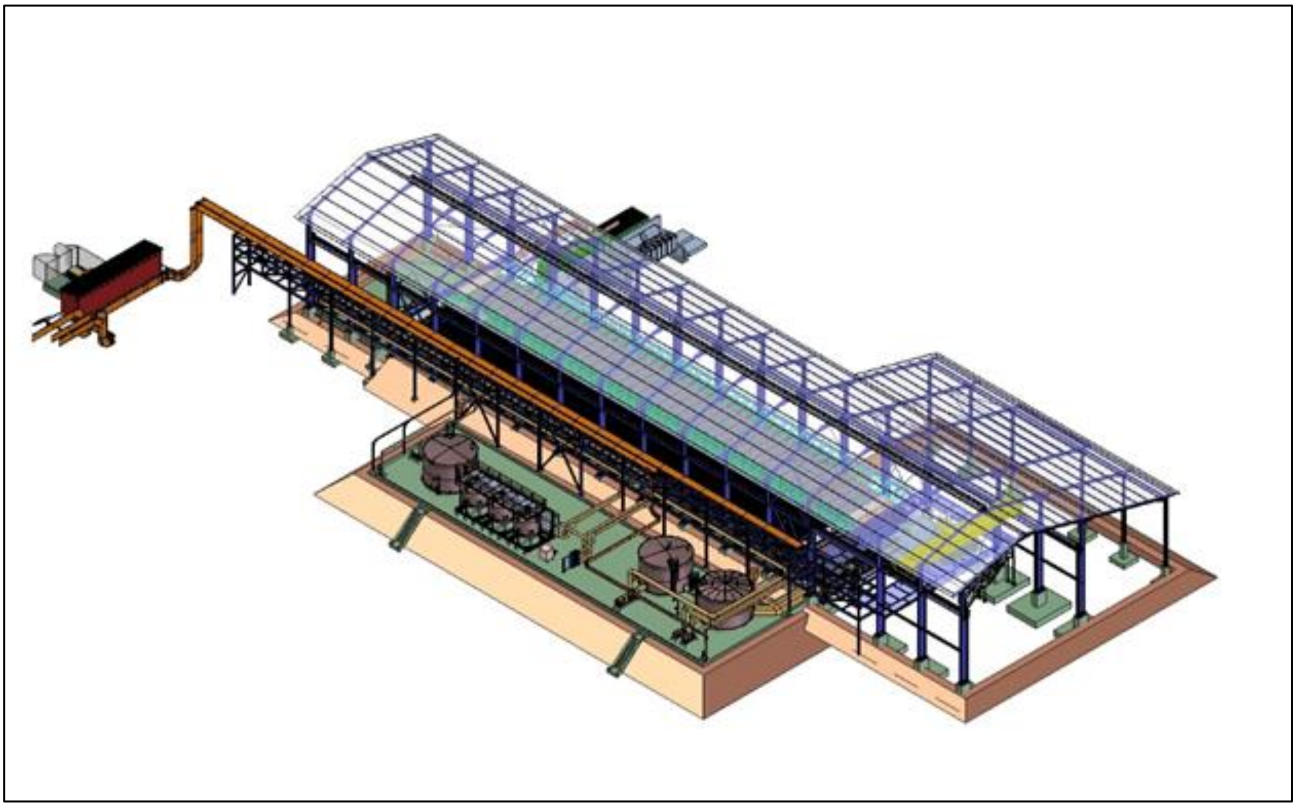
A DC current will be applied to the electrolytic cells that contain lead anodes and stainless steel cathode mother plates. Oxygen will be liberated at the anodes and copper will be deposited at the cathodes.

The cathode mother plates will be removed and transported by overhead crane to the semi-automatic cathode stripping machine where adherent electrolyte is washed from the copper deposit with hot water. The cathode mother plates will then be delivered one at a time to a flexing-stripping station where the copper deposits will be removed.

Copper will be harvested on a seven day *growth* cycle from the electrowinning cells. One third of the cathodes (every third cathode) are removed from a cell by an overhead crane that transports the cathode mother plates to the washing / stripping machine. The cathode stripping machine will be semi-automatic and contain a flexing stripping and knifing station. Cathode mother plates will be delivered to the feed-in conveyor of the machine and indexed automatically through the wash station section where adherent copper sulphate electrolyte will be washed off. The stripping machine operator will pick up a single electrode from the wash station using the stripping machine hoist and deliver it to the flexing/stripping station. The flexing stripping station operation will be initiated by the operator and the cathode deposit will be automatically removed from the mother plate and delivered to the bundling station for manual strapping. The freshly stripped mother plate will be delivered to the feed-out conveyor, and the mother plate will be automatically spaced for return to the cells.

The copper cathode will be bundled, weighed, marked and strapped ready for shipment to export customers. Shipment will be by truck to an export terminal.

Figure 17-17 Typical 25 000 tpa EW building



17.7 NEUTRALIZATION

The following description should be read in conjunction with Figure 17-21 Neutralization PFD.

The neutralization plant is used during operation for both the reduction in solution acid concentration and precipitation of metals from water to be discharged from the site. This is required to allow the solvent extraction plant to operate effectively in extracting the copper from the leach solution. It is also required to precipitate the metals and sulphates to levels that are acceptable for solution discharge as part of the overall site water management.

Limestone is crushed in an adjacent quarry (approx. 15km from site) and delivered to the neutralization plant stockpile. From here it is reclaimed by wheel loader and delivered at a controlled rate to the limestone mill. The limestone is ground to a fine slurry and the slurry stored in agitated tanks.

The limestone is added to the SX raffinate solution in such a way as to neutralize the acid while both maximizing the dissolution of the limestone and minimizing the production of gypsum scale in the downstream process equipment.

Crystallization control is achieved by separating the raffinate from the neutralizing agent into two separate reactors and adjusting the recycle flow between the two reactors. Limestone is mixed with the solids recycled from the product thickener to produce a conditioned sludge in the sludge reactor. Sufficient time is allowed for the limestone to be adsorbed onto the surface of the recycled metal hydroxides, leaving very little limestone left free in the solution. When the conditioned sludge is added to the neutralization reactor, the free acid is the first chemical species to react with it. The acid neutralization process is stopped at this point. The sludge is thickened and the thickened solids both recycled and sent to sludge filtration. A recycle rate for sludge solids of about 250% is used for most metallurgical plant operations. The sludge solids recycle minimizes the reagent consumption, reduces the amount of gypsum that can scale downstream equipment, and grows larger crystalline precipitates that are easier to thicken and filter.

For water balance management, the plant is used for metal precipitation from excess process solutions. After the acid is neutralized the metal sulfates react with hydrated lime to precipitate further metal hydroxides. Since the majority of the limestone is adsorbed on the surface of the sludge particles, the reactions take place on the particle surface, rather than out in free solution. This layer by layer growth results in a smoother particle surface, with less void space. Hence, less entrapped water and a denser particle. The particle surfaces also tend to repel water bonding, allowing metal solids to be more easily captured.

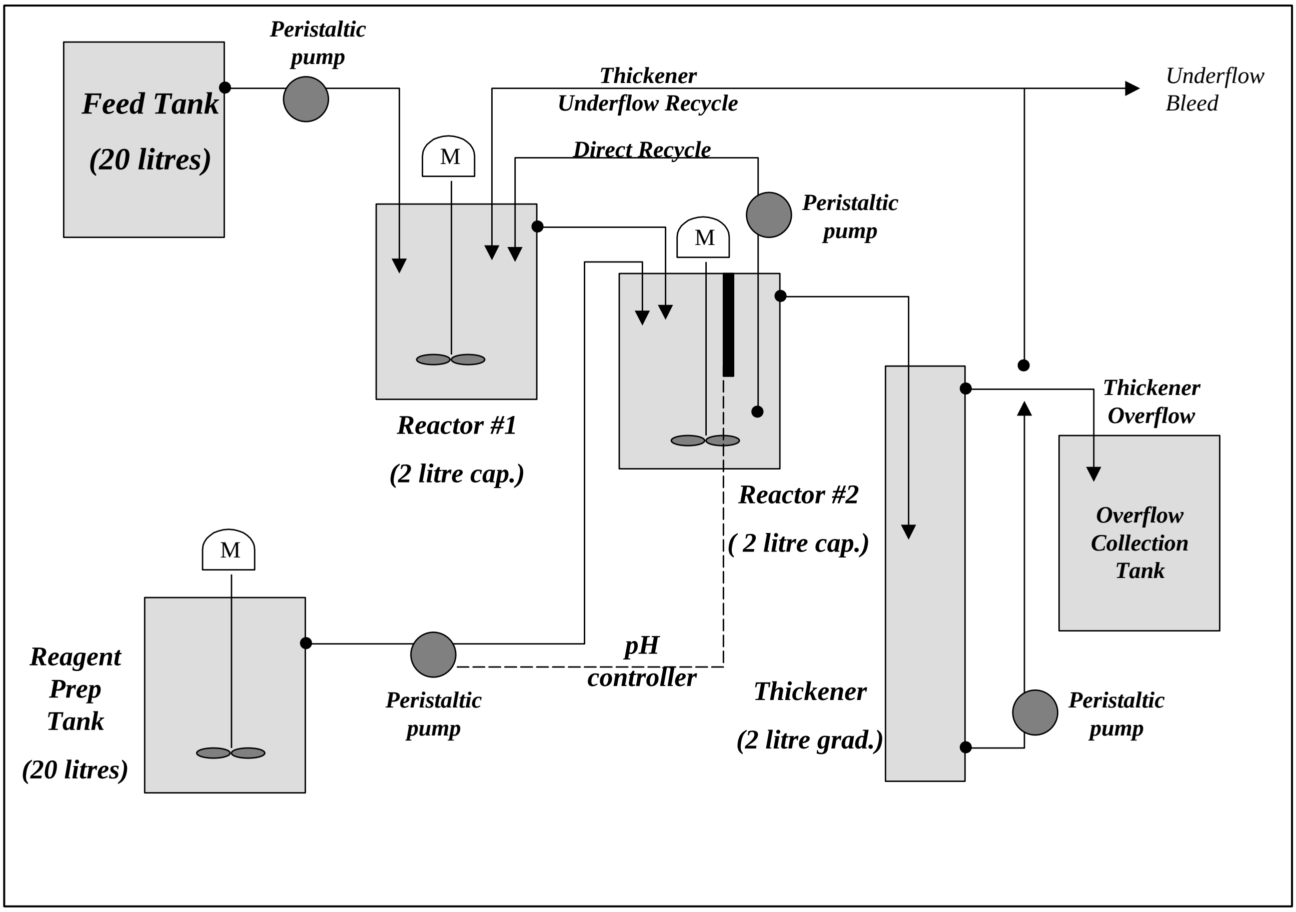
The recycled sludge from the thickener has a lower metal ion concentration, and will dilute the metal ion concentration of the feed stream in the first vessel. This dilution effect will discourage new particle formation in the first vessel, and therefore encourages precipitation on the existing particles for denser particles. In the event that particle growth needs to be improved further, thickener underflow can be recycled back into the reactor system.

The sludge from this process tend to have a lower neutralizing capacity than conventional sludge. The Net Neutralization Potential (NNP) of this sludge typically range from 80–300 tonnes $CaCO_3$/1000 tonnes of sludge compared to a conventional alkaline neutralization sludge that have NNP between 160 – 800 tonnes $CaCO_3$/1000 tonnes of sludge. This indicates that the alkali use is more efficient and less un-reacted limestone and lime are present in the HDS sludge. The amount of excess dissolved gypsum in solution is also significantly reduced resulting in almost no scale build-up in the process equipment.

The thickened sludge is filtered in pressure filters to remove the moisture from the sludge. The solution is returned to the process for re-use. The filter cake is used as fill in the quarry from which the limestone was mined or will be disposed with the mine waste rock dumps.

A simplified laboratory pilot circuit is shown in Figure 17-18. A full flowsheet is provided in Figure 17-21.

Figure 17-18 Simplified Flowsheet of a laboratory sludge recycle neutralization process



17.8 PROCESS FLOW DIAGRAMS

The following to provide a pictorial representation of the process described in the previous sections 17.2 to 17.7. These are referred to as Process Flow Diagrams, or PFD's.

Figure 17-19 Crushing and Heap Leaching PFD

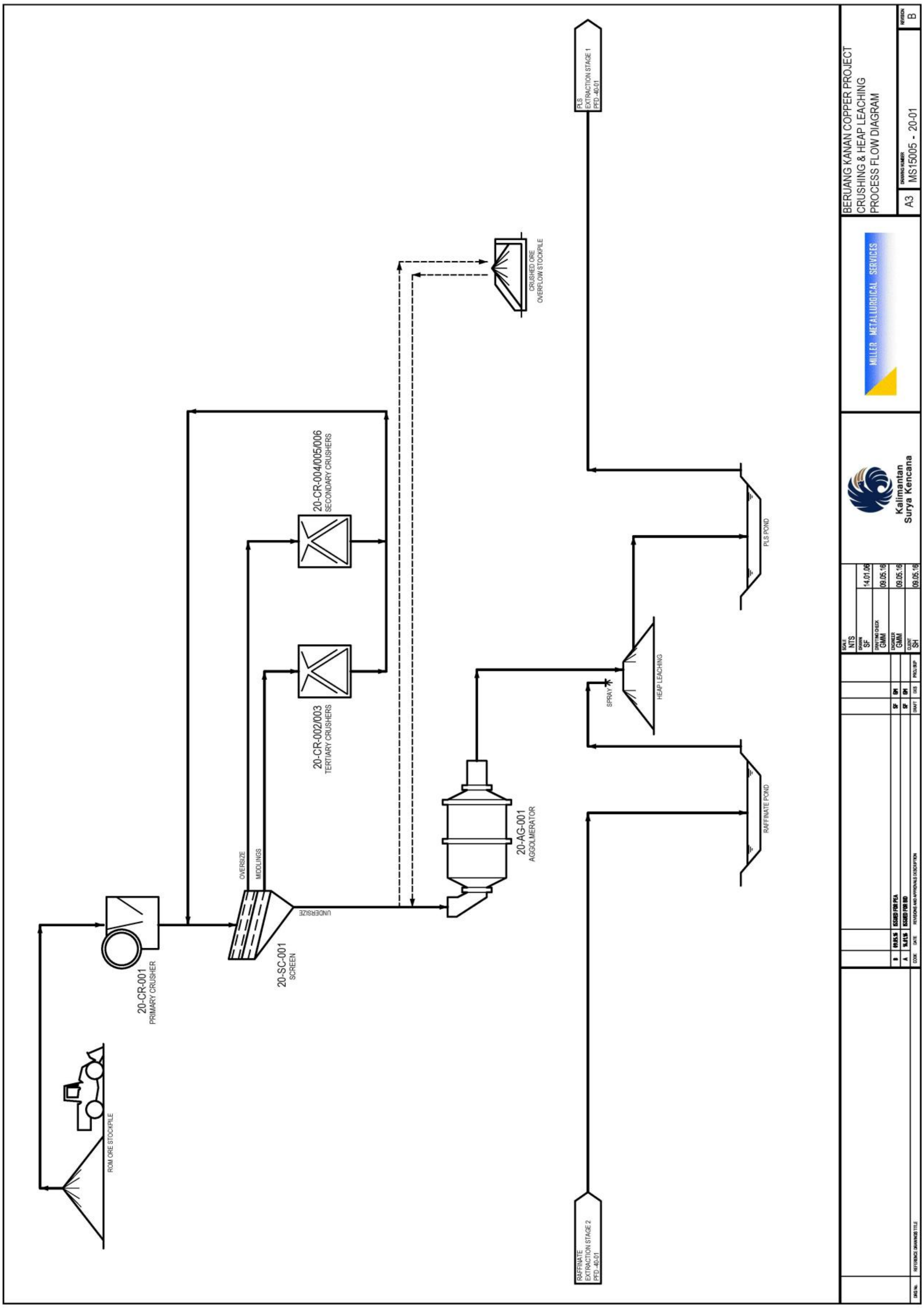


Figure 17-20 Solvent Extraction and Electrowinning PFD

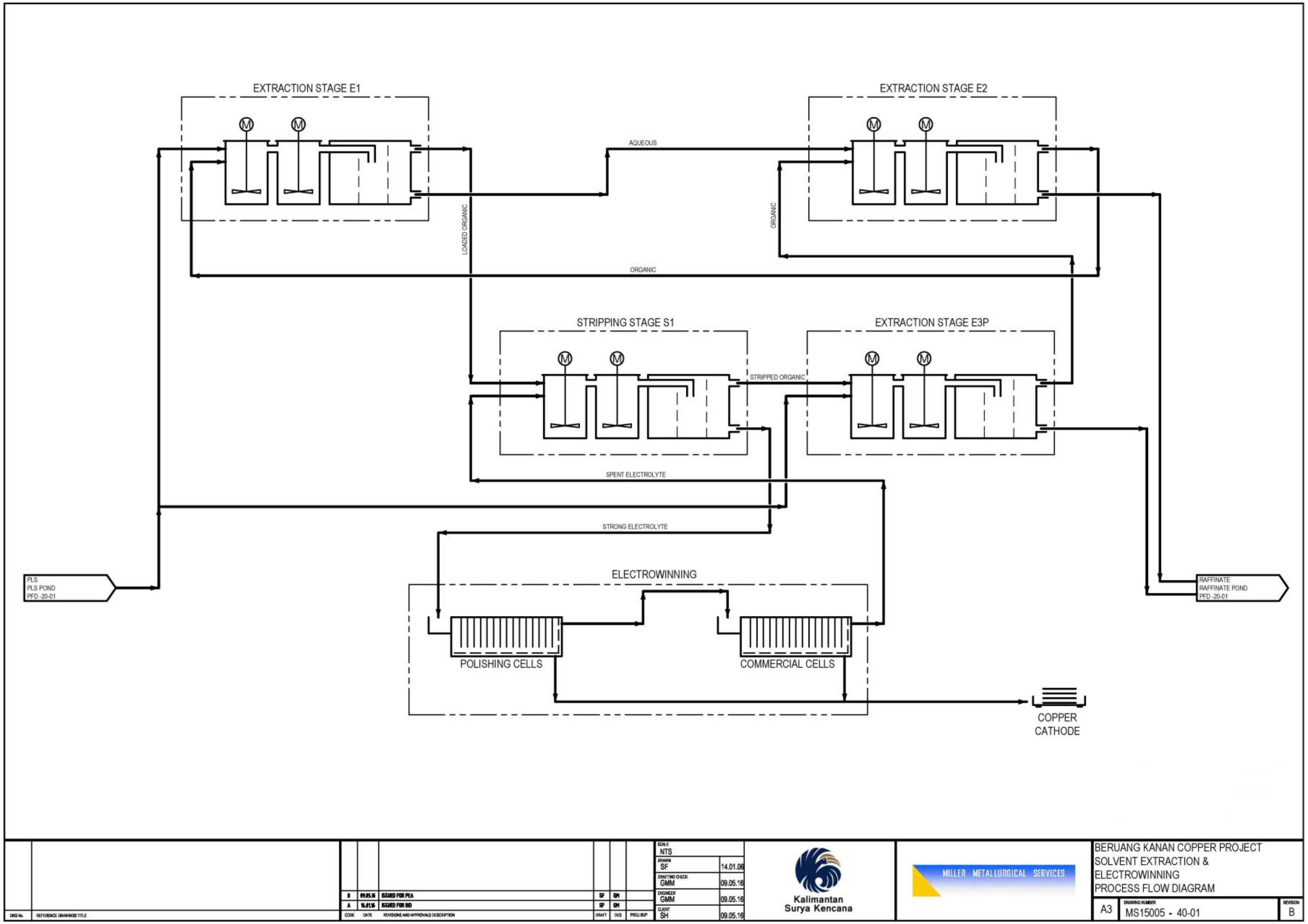


Figure 17-21 Neutralization PFD

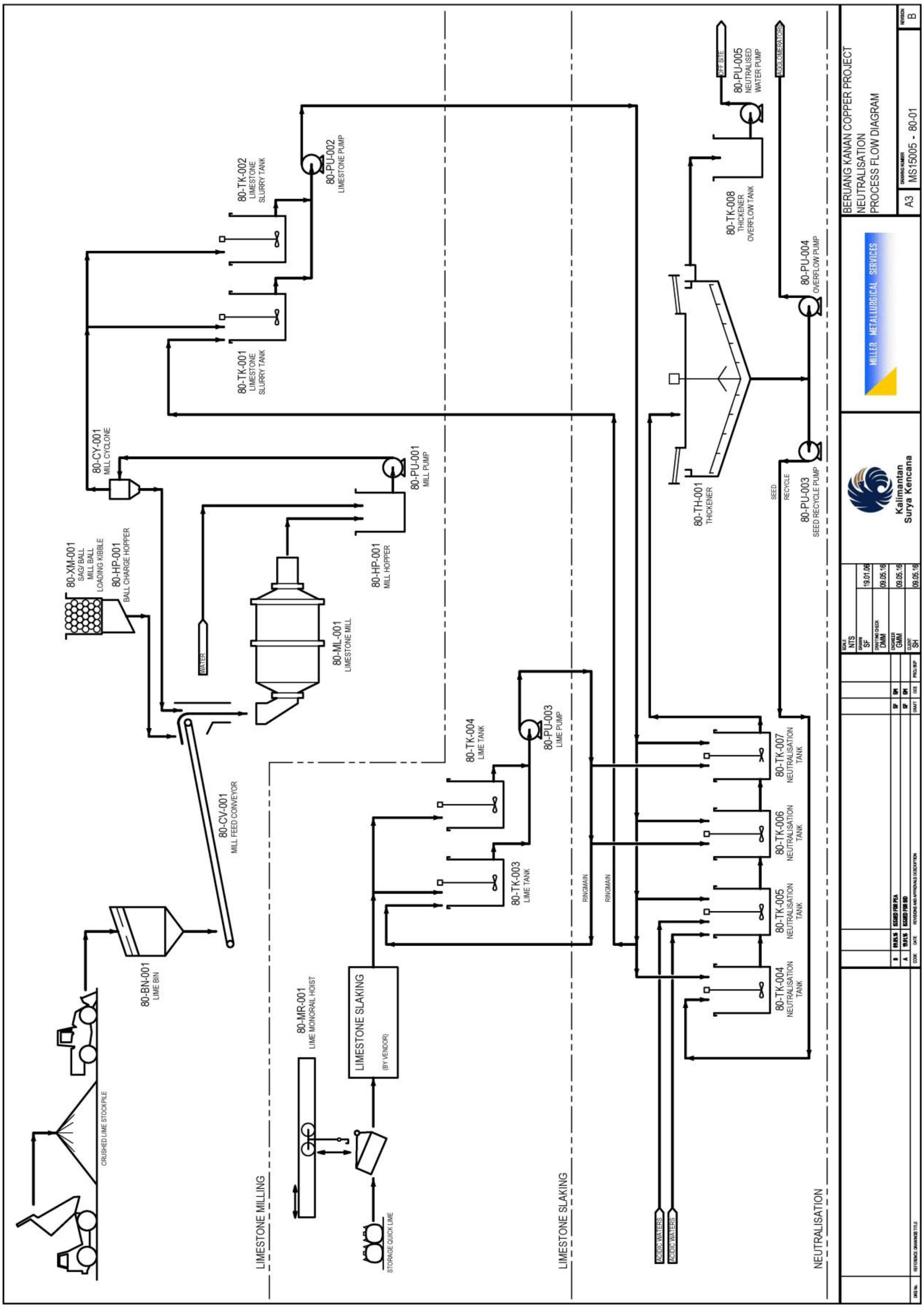
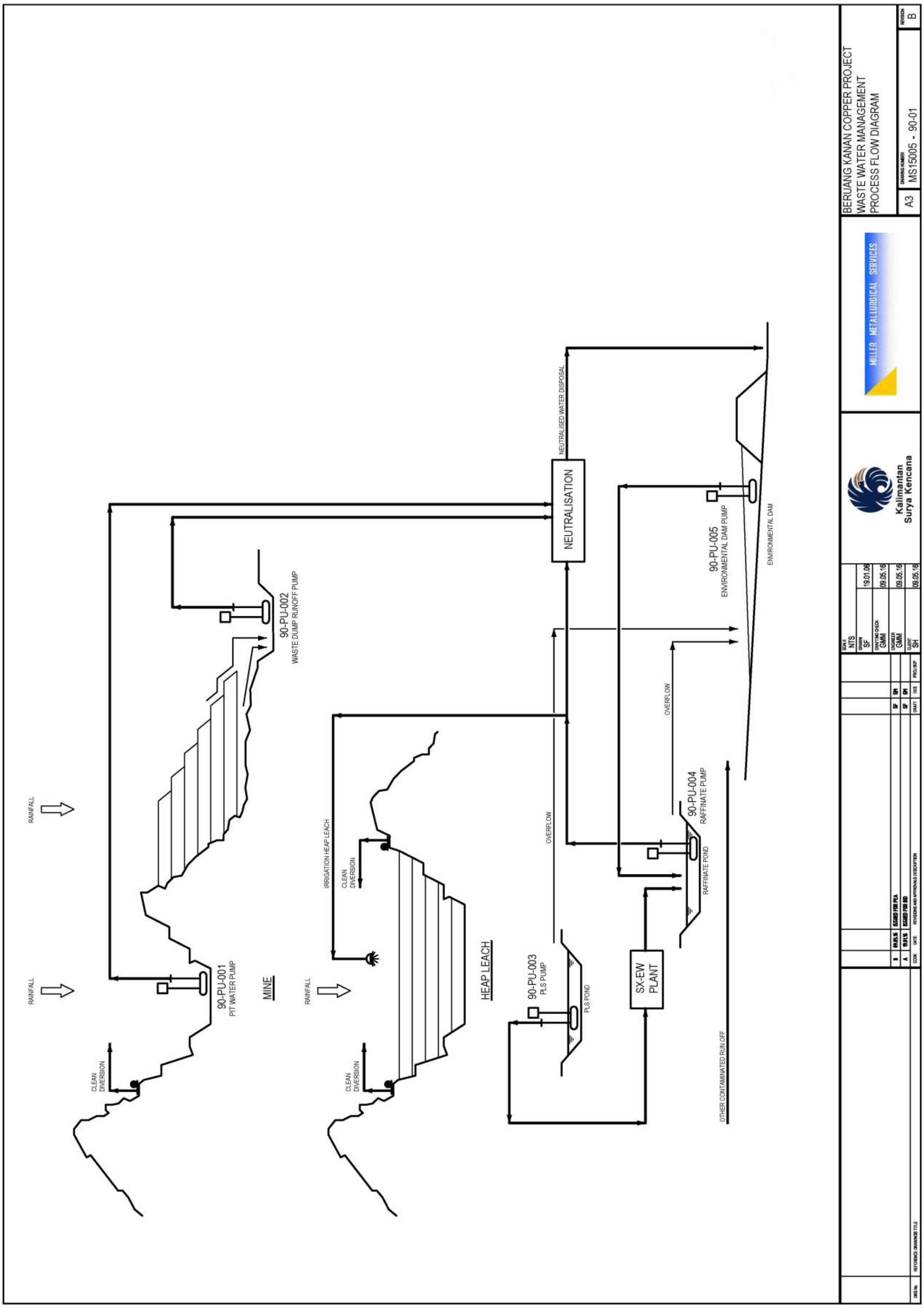


Figure 17-22 Process Water Management PFD



17.9 Reagent / Services

17.9.1 Sulphuric Acid

Concentrated sulphuric acid (93% to 98%) is delivered to the site storage tanks via truck transportable tank-isotainers. The acid is off-loaded and stored in mild steel tanks. The concentrated acid is pumped to the EW acid mixer to maintain the required electrolyte acid concentrations. Some acid is required for start of leaching and will be added to the agglomerator drum using local temporary pumps direct from a tank-isotainer.

Concentrated sulphuric acid piping is Schedule 80 carbon steel. At dilution points piping is grade 904L stainless steel.

17.9.2 Cobalt Sulphate

Cobalt sulphate will be delivered via road in one (1) tonne bags to an on-site make-up plant. It will be added to the EW to maintain a minimum of 200 ppm cobalt. Some cobalt is lost to the leach solution in the EW bleed that is used to control the iron in electrolyte.

17.9.3 Extractant

Extractant is transported in Intermediate Bulk Containers (IBC's) and is delivered via road, loaded in 20 ft. shipping containers.

The plant fork lift will handle the IBC from the shipping container or site storage warehouse to the SX plant where it is emptied directly into the SX sump pump by a flexible hose. A rinse with diluent will also be required to flush the last of the extractant from the IBC. Disposal of the empty IBC's will be done using the site protocols for hydrocarbon management.

17.9.4 Diluent

Diluent is delivered in 22 m^3 tank-isotainers. The isotainer will be emptied by pumping the diluent into the storage tank located next to the extraction settler E1 in the SX plant. Diluent is dosed by the diluent pump to the organic launder of the extraction settler E1. The same pump is used for both off-loading the delivery tanks and discharging from the storage tank.

17.9.5 Hot Water

Hot water will be supplied from a dedicated diesel / electric fired hot water heater for process heating in the EW plant. It will be reticulated around the plant.

Hot water is used in:

- EW plant for trimming of the electrolyte temperature
- EW plant for electrolyte heating from a 'cold start'
- Wash water heating in EW.

17.9.6 Compressed Air

Plant compressed air is reticulated from the site supply compressors. All the air will be dried to instrument quality. Only one air quality will be used on site.

Plant air use is for running of portable tools, and supply to utility hose points. Instrument air is taken from the single supply header into plant area sub-mains.

A stand-by compressor will have sufficient capacity to supply instrument air and critical process air.

17.9.7 Water Systems

A raw water pond will be provided on the site to receive untreated, uncontaminated water from the supply system.

A raw water pump located at the raw water pond will feed a simple water treatment plant (filtration and chlorination) to maintain an acceptable level in the potable water tank. This tank will be sized to provide potable water requirements for the safety showers, offices and ablutions and process plant make up.

The process water pump will deliver raw water to the crushing plant for dust suppression, the raffinate pond to provide for evaporation losses in the heap leach; and to utility hose stations throughout the Plant.

A fire water system will be installed that includes the regulatory instrumentation, jacking pump, electric pump and diesel fire pump to maintain a set discharge pressure at all times to hydrants located in the process plant.

Contaminated water from the heap leach storm pond or from open de-watering will be delivered directly to the raffinate pond for inventory and evaporation make up. Neither of these waters will be delivered to the raw water pond. Excess contaminated water will be neutralized and discharged from site.

17.10 FIRE PROTECTION

A number of hand held foaming units will be provided for use in the SX plant area.

Electrical rooms are fitted with hand held carbon dioxide (CO_2) fire extinguishers.

Historically SX facilities have experienced significant fire events due to the large quantities of combustible organic solvent being treated, with SX fires proving extremely difficult to extinguish once established. In these cases the most effective means of fire control for an SX facility is to provide the following:

- Minimum separation of +30 m from other valuable assets
- Means of cooling the external surfaces of the other assets if they are closer than this
- Ability to channel the SX plant contents to a nearby emergency pond that is remote from the other assets
- Address the known fire initiation factors and eliminate them from the plant design.

The separation distance issue is one of the main drivers to the site selection. The nearest significant assets are the EW and crud plants which have been located more than 30 m from the SX plant as shown on the Overall Process Area Development Plan (refer to Figure 17-11).

The SX plant includes one overflow pond. In the event of a fire, excess solvent is discharged to the pond via a buried pipe that includes a fire break box to eliminate any active flame propagating to the pond. With this arrangement only a minor amount of potential liquid organic fuel will be left in the SX area.

The plant will utilize high flash point diluent specifically formulated to minimize fire risk in the SX duty.

Lightning protection for the SX plant has been provided for, with the provision of lighting masts and finials as well as bonding conductors for the building structures and surrounds. Other buildings will be protected as required by local regulations.

17.11 BASIC CONTROL PHILOSOPHY

The process plant control system will be based on a network of PLC's with a SCADA operator interface. There will be an operator interface station in the crushing area and in the process plant control room located in the SX / EW plant.

A more detailed control philosophy will be included in any subsequent feasibility level study. This will describe the:

- Instruments included, their function and control
- Description of interlocks and safety trips
- Description of the sequences for starting, stopping and tripping the plant in a safe manner
- Interfaces with vendor packages

17.12 PROCESS SITE WATER MANAGEMENT

The site water balance has not been developed due to the lack of appropriate data on key parameters:

- Rainfall design events and return intervals
- Evaporation rates
- Catchment yield estimate

All of these items will be addressed as part of the next phase of study for the project.

The water management plan is shown in Figure 17-22 and is based on similar plans for operations in tropical monsoon type climates. The plan assumes that rain over the potentially contaminated catchment will generate surface water over and above the make-up needs for the process and the inventory in the leached materials. As a result a number of actions will be taken to minimize the input of rain water to the potentially contaminated catchment and to manage the waters that are accumulated there.

- Wherever possible the clean run off will be diverted around the storm water catchments through:
    - Use of contour drains to run around the heap leach and the storm water dam
    - Use of 'rain coats' on the heap leach during the wet season; to capture clean rain water and divert it into the diversion drains
    - Minimization of the total footprint of the plant and leach pads that need to be captured in the storm water dam.
- Maximize the use of storm water for process input to the leaching solutions, the crushed material inventory and any other potential use.
- Use of heap leach irrigation systems that will maximize the evaporation of water
- Potential to use enhanced evaporation techniques within the storm pond catchment to eliminate as much storm water which will reduce the requirement for neutralization.
- Provision of a neutralization and precipitation plant to allow preparation of a treated water stream that will be suitable for discharge from the site.

These principals have been applied to at least one other heap leach project in Indonesia, at the Wetar operation, and to a number of operations in equatorial Africa and Central America.

17.13 PROCESS SITE SELECTION

The process site selected for the project is shown in Figure 17-11. The site was selected to address the following criteria

- Minimum haul distance from the mine
- Ability to install a leach pad of the capacity to treat all the mined material
- In a catchment that has a limited upstream or uphill catchment
- Can construct a storm dam with large containment volume without a major dam construction.
- Is not at risk of back flow from other streams during extreme rain events
- Has sufficient sites that can be constructed for the major plant and infrastructure
    - Crushing plant
    - Heap leach pad
    - Solution ponds

- SX-EW recovery plant
- Neutralization plant
- Power station
- General services and infrastructure

The current site selected addresses all of these requirements.

Two potential alternate sites were considered to the east of the proposed pits:

1. Between the pit workings and the significant stream that runs approximately 500 m east of the pits
2. To the east of the stream.

Both of these sites had issues that precluded them from further consideration, including:

- Longer haul distances for LFI from the pits
- Requirement to construct a bridge over the stream. This would require an expensive high level bridge to maintain access during larger rain events.
- A large amount of catchment diversions to limit the inflow of clean water.
- Larger storm pond dams due to the limited topographical features that could be exploited
- For one site there was not enough space for the (relatively) shallow storm pond, the heap leach pad and the process plant
- For the other site the space was available but it was very crowded and unlikely to be readily operable.
- Both sites were on similar levels to the major stream and would have had higher risk profiles for backflow from that stream.

18 PROJECT INFRASTRUCTURE

18.1 SITE DEVELOPMENT

18.1.1 Bulk Earthworks

Bulk earthworks are required in the following areas:

- Haul Roads
- Diversion Channels
- Heap Leach Pad
- ROM Pad/Secondary Crusher
- SX-EW pad
- Raffinate pond
- Power Station Pad
- Lime Pad
- Pump Station Pad
- Environmental Dam
- Accommodation Camp Pad
- Stormwater HDPE Lined Dam

All water ponds will be constructed from compacted earth fill embankments and will be seepage sealed and protected with a 1.5mm thick HDPE lining.

Provisions have been made for the ROM pad and brake test ramp.

No geotechnical, hydrology or hydrogeological information was available for the PEA and therefore conditions were assumed.

18.1.2 Stormwater Management

The principle employed to manage storm water includes two classifications namely, contaminated and uncontaminated stormwater.

18.1.2.a Contaminated Stormwater

All stormwater which falls onto the mining or process areas including stockpiles will be contained and re-used wherever possible, in line with a zero discharge philosophy. Stormwater control ponds will be constructed to collect contaminated stormwater for return to the process circuit.

Contaminated stormwater will be collected in a pollution/stormwater control facility through a network of surface/stormwater cut off drains and sub-surface drains. All major stormwater systems will be designed to accommodate a 1 in 50 year flood event.

18.1.2.b Uncontaminated Stormwater

Stormwater cut-off berms will be constructed around the mining pit, plant areas and other infrastructure facilities to prevent stormwater run-off from higher laying areas becoming contaminated by being exposed to mining or process related activities or infrastructure.

The heap leach will be provided with plastic covers (rain coats) to minimize infiltration of rain water. The clean run off from the raincoats will be collected separately and discharged from the site.

18.1.2.c River Diversions

No provision has been made for river / stream diversion as the preliminary site layout (refer to Figure 16-21) indicates that the major infrastructure items are located outside of the major watercourse running to the east of the project area.

A detailed hydrology (surface water) study will be required to accurately assess the stormwater management infrastructure requirements and impacts on the natural water courses within the project area.

18.2 ROADS

18.2.1 Access Road

A river navigable by barge is located within 3 hours of the BKM Project. This will provide the most efficient and lower risk option for transporting materials to and from the site.

Due to limited information regarding the condition of the existing local road network, a high level desktop review was conducted of the proposed access route from Palangkaraya to site using Google Earth.

A detailed route survey will need to be conducted during subsequent phases of the project to ascertain the condition of the road and accurately assess any upgrades necessary.

18.2.2 Mine Haul Roads

Refer to Section 16.4.5.

18.2.3 Internal Service Roads

The project is serviced by an internal road network servicing the plant and mining area as well as the outlying infrastructure facilities on site. The internal service roads were assumed as 6m wide and to have a gravel wearing course.

The sections of road highlighted in red in Figure 18-1 require upgrading. Road 1, running NW to SE is in an apparent bad state and needs rebuilding. The road rehabilitation assumptions are:

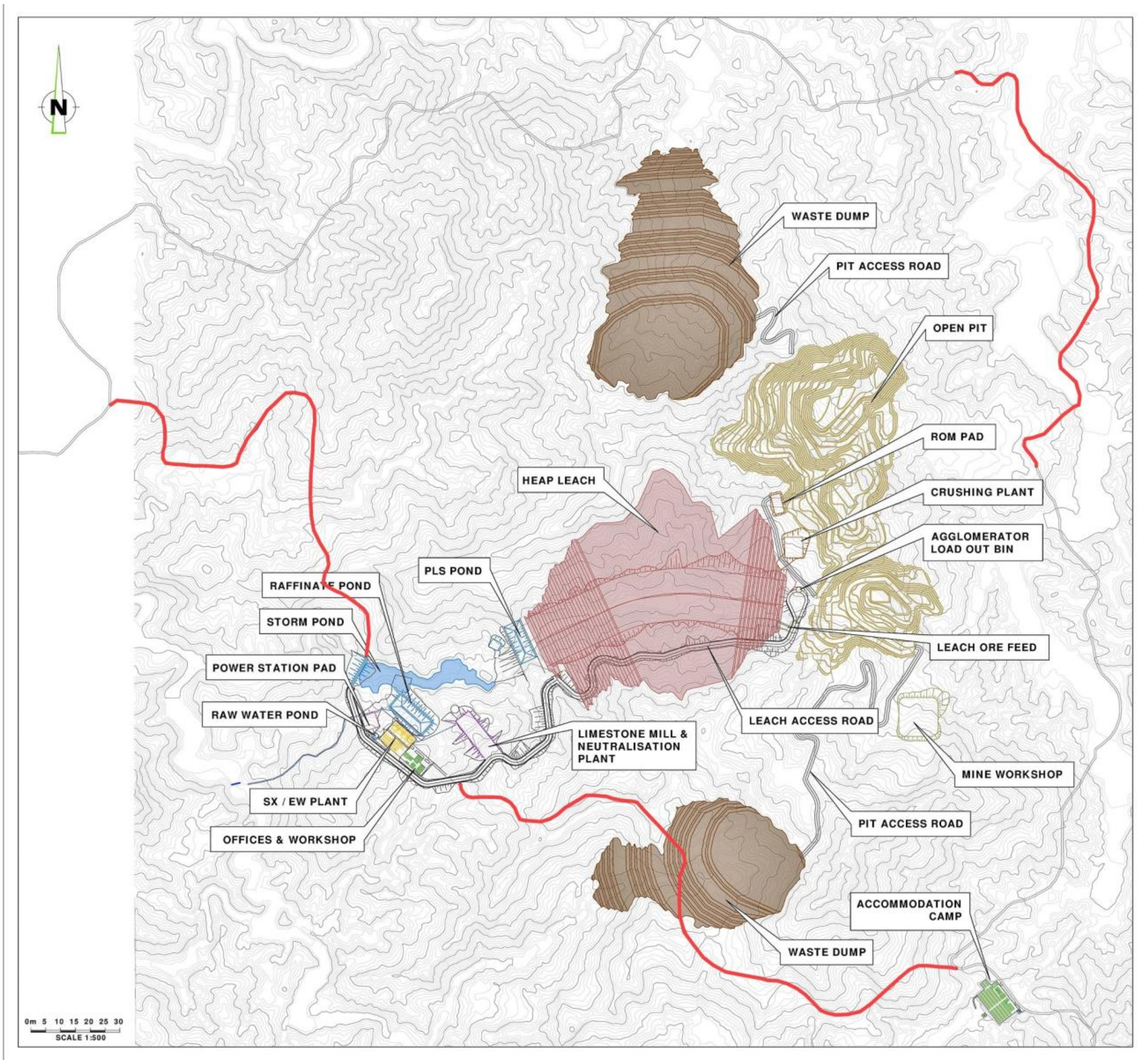
- 450mm to be excavated

- Followed by compaction of 200mm of in-situ material
- Followed by three layers of compacted backfilling, each 150mm class B1, D2 and D2, will be required.

Road 2, running N-S as shown in Figure 18-1, is apparently in a slightly better state and only requires backfilling. The road rehabilitation assumptions are:

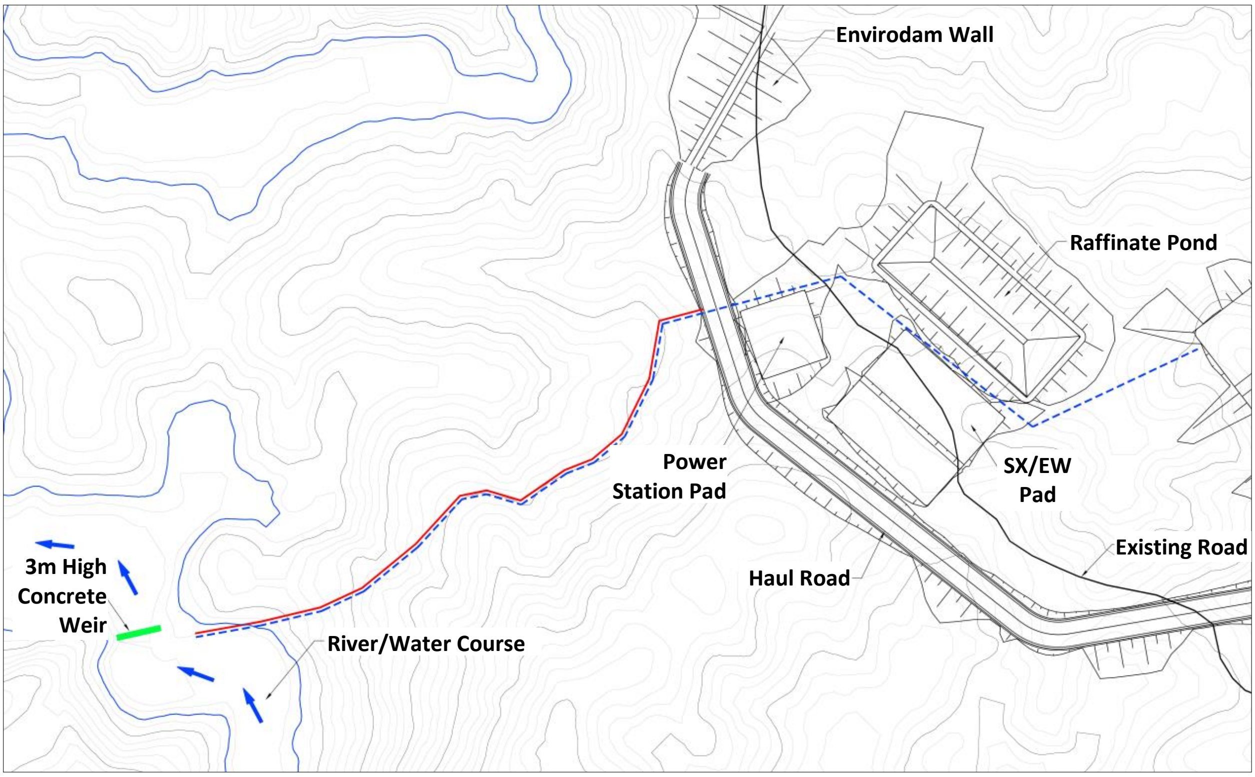
- 200mm of material will be ripped to spoil
- Followed by compaction of 200mm of in-situ material
- Followed by backfilling and compaction of 200mm of D2 material.

Figure 18-1 Internal roads potentially requiring rehabilitation



A service road for the pipe line from the river to the plant pad has also been included as shown in Figure 18-2.

Figure 18-2 Water pipeline service road



18.3 POWER SUPPLY

The BKM Copper Project will require approximately 20 MW of peak load power for 25,000 tonne-per-annum operation demand. A key infrastructure component is the supply of electrical power as this constitutes 25% of the total operating cost. An established Indonesian power supply company has provided initial cost estimates to KSK for a power rental and supply arrangement based on an appropriately sized liquefied natural gas fired power plant.

Power will be distributed from the power station via an 11kV overhead line. A ring main unit (RMU) located near the ROM and secondary crusher will provide isolation from the step down transformers. Similarly, an RMU at the limestone mill will accept 11kV, while a mini-substation located at the SX-EW plant will convert the 11kV to 400V for plant use.

The main 11kV substation will be directly fed from the power station as it is close by. This substation will serve the SX-EW plant and the raffinate pond pump.

The accommodation camp will be serviced by two separate diesel generators, one duty and one standby. This was weighed against the potential issues that can be experienced with distributing 11kV from the plant power station over the challenging terrain to the camp. Each generator is a 250kVA, 400V unit, with a self-contained diesel day-tank.

A third 150 kV generator will provide power to the water pump down by the river. A trade-off should be performed that explores the possibility of this pump being powered from the main power station once the exact locations of the plant and pump station are known.

18.3.1 Fuel Storage

Minimal on-site fuel storage will be required due to the provision of electrical power from an external provider. Mine fleet consumption is assumed to be catered for by the mining contractor and excluded from this figure.

18.4 WATER SUPPLY

The philosophy is to provide the majority of the raw water requirements from the storm water control ponds or from the mining pit dewatering pond. In the event that these water sources are inadequate, raw water will be pumped from the river, as indicated in Figure 18-2.

A HDPE lined raw water pond located at the process plant will provide two days raw water buffer capacity for the project.

18.4.1 Pit Dewatering

Pit water will be pumped to a either an ex-pit surface dam or an in-pit settlement pond to allow settlement of suspended solids. From these storage facilities the water will then be pumped and or gravity drained to the main envirodam for controlled discharge. The mine water storage facilities will be regularly cleaned of silt and the silt will be disposed or inside the mine waste rock dumps.

When required, pit water will be pumped from pit sumps into water carts and used for dust suppression. However as rainfall is relatively high and evaporation rates low for the BKM site the requirement for dust suppression will be minimal relative to the pit water inflows.

Surface water will be drained away from the pits to avoid inundation of the pits by surface flood water.

A detailed hydrogeology (ground water) study will be required to determine the extent of pit dewatering required.

18.5 MINE SITE INFRASTRUCTURE

BKM will be mined by a contract mining company in order to minimize the mining capital cost of the project. The mining contractor will be responsible for the supply and operation of the mining related infrastructure which typically includes:

- Offices, lunchrooms & ablutions for contractor personnel.
- Workshop for all mining and ancillary equipment and the contractor light vehicle fleet, including wash down bay, welding area and tire storage area.
- Fuel and oil storage facilities.
- A storage facility and explosives magazines.
- Pit dewatering equipment.

18.6 PROCESS INFRASTRUCTURE

The following infrastructure buildings will be located nearby the process plant facility:

- Plant Motor Control Centers (MCC's),
- Plant control room,
- Plant laboratory & equipment,
- Plant gate house (pre-fabricated or timber construction),
- Plant workshop & equipment,
- Plant store & equipment,
- Plant offices & furniture (pre-fabricated or timber construction),
- Plant ablutions (pre-fabricated or timber construction).

A budgetary cost estimate from a local contractor has been sourced based on a layout provided by DRA. These units will be constructed using timber material and employing the local workforce which is the most cost-effective approach.

18.7 ACCOMMODATION CAMP

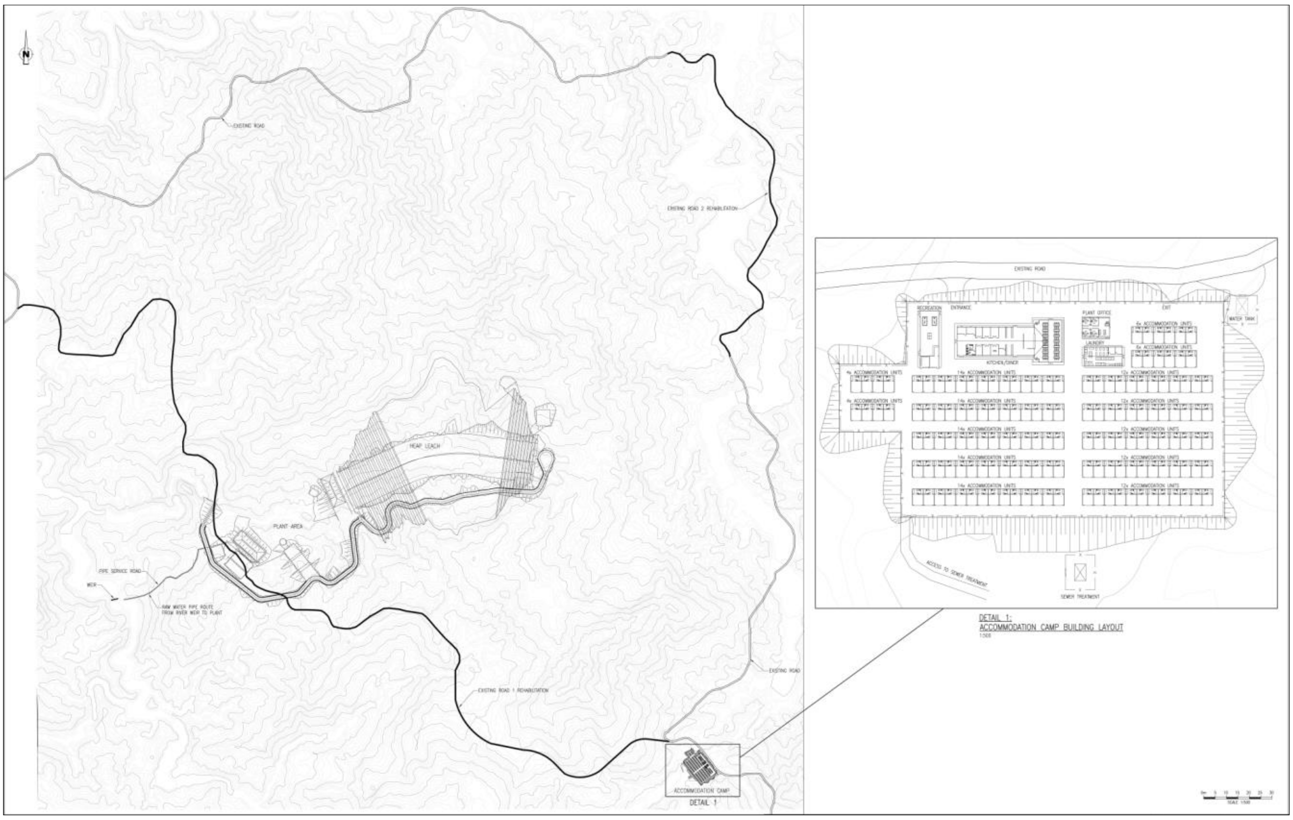
Due to the remote location of the project, all operational and construction staff will be housed on site in an accommodation camp.

The permanent operations camp will consist of one-hundred pre-fabricated (or timber constructed) single-quarter units. There is also a camp office, kitchen and diner, recreation building and laundry.

The camp will be commissioned early on so it can be used during the construction phase by senior project staff members. Temporary tented accommodation will compliment any additional labor requirements during the construction phase of the project.

The location of the camp is shown in Figure 18-3, with the key plan insert showing typical layout.

Figure 18-3 Accommodation Village Location and Layout



18.8 SITE UTILITIES & SERVICES

18.8.1 Potable Water

Raw water will be treated using a potable water treatment package plant and discharged into storage tanks located at the accommodation camp. The potable water will then be pumped to elevated holding tanks located on the mining, plant, camp and office terraces.

The reticulation network consists of buried uPVC piping of various size diameters (according to flow rates required). The elevated tanks will provide sufficient water pressure for domestic and safety shower use.

18.8.2 Sewage Treatment System

Sewage from the mining, plant office terrace areas will be reticulated to a localized septic tank at each area. Each system is based on the “soak away” principle. The assumption is that the soak away drain pipe will not be within one meter of the groundwater or rock.

18.8.3 Communications

The site will be equipped with IT/ VOIP communication networks connected to a satellite link for communications with the outside world.

18.8.4 Fencing and Access Control

18.8.4.a Fencing

Fencing will be provided around the mining and plant platform areas, the office area and all the water containment ponds. No perimeter fencing is required as the area is remote and is densely vegetated.

All fencing on site is assumed to be of the diamond mesh type – 1.8m high.

18.8.4.b Access Control

Manned access control will be provided at the main entrance and the SX plant entrance using a portable pre-fabricated (or locally constructed timber) security guard house. This will allow the manual checking of vehicles, documentation and persons entering at this point. It will also allow for persons who have not been through induction etc. to be facilitated. Visitors will be instructed in the mine's safety requirements via a safety induction conducted from within the main security office. Safety equipment will be issued, as necessary, before being given access to plant or mining areas.

18.8.5 Weighbridge

Bundled copper cathode will be weighed before transporting by truck to an export terminal. A 3.2m wide weighbridge will be constructed for this purpose and will be positioned between the road and the process plant terrace to allow optimized flow of trucks in both directions.

19 MARKET STUDIES AND CONTRACTS

19.1 COPPER MARKET

Copper is a commodity traded on a global scale between producers and consumers and is sold in three main forms: refined copper cathode; copper concentrate; and blister copper.

Cathode is the purest form of copper and is the feedstock used to produce copper wire, cable, sheet, strip, tube, etc. It is also used in the production of alloys such as brass and bronze.

The SX/EW process to be employed at BKM is anticipated to produce copper cathode that attracts an LME (London Metal Exchange) “A” Grade Copper Price and assumes that it will met the required quality and physical properties to command the associated price premium. However if the product is not LME accredited, a discount to the copper price may be negotiated by a future buyer. There are no other deductions to the value of the contained copper.

19.2 COMMODITY PRICE PROJECTIONS

Copper prices are predominantly affected by global market supply and demand dynamics.

For the purposes of this study it was assumed that the BKM project will commence production in 2019 and KSK, through Asiamet Resources Ltd, sourced an independent copper price forecast from Wood Mackenzie (Woodmac). Woodmac are a recognized international consulting and research group providing market analysis across a broad spectrum of commodities. The forecast copper price schedule provided is detailed in Table 19-1 below.

Table 19-1 Copper Price Project

Year	Price (USD/lb)
2019	2.60
2020	2.95
2021	3.25
2022	3.65
2023	3.60
2024	3.40
2025	3.30
Average	**3.25**

Source: Wood Mackenzie, March 2016

. In line with KSK’s view on copper price, the average of the Woodmac forecast LME copper price (Constant 2016 $US), over the seven years 2019 to 2025, of $3.25/lb. was applied for the financial analysis of this study.

19.3 CONTRACTS

There are currently no contracts in place between KSK and other entities regarding the off-take for the BKM Project.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The KSK CoW comprises two blocks, A and B with total area 61,003 ha. The BKM deposit is located within Block B, which is situated in Gunung Mas Regency, Central Kalimantan Province.

The senior management team at KSK has extensive experience with mining projects in Indonesia and fully understands the importance of conducting all aspects of mining activities in an environmentally and socially responsible manner to ensure the success of the project. Specifically KSK is committed to:

- Complying with all applicable Indonesian environmental and social laws and regulations pertaining to mining operations;
- Adopting an inclusive and transparent approach with all stakeholders, with a focus on local communities and Indigenous Peoples;
- Appling Best Available Technology (BAT) and Good International Industry Practice (GIIP) to the design and operation of mining activities at the BKM Project; and,
- Leaving a positive legacy subsequent to cessation of mining activities.

Environmental and social considerations for the project are discussed in this section. Following the description of the environmental and social setting in Section 20.2, the potential environmental and social effects are detailed in Section 20.3. Management strategies to address any significant environmental or social effects are described in Section 20.4 and Section 20.5 respectively. Permitting requirements for the BKM Project are described in Section 20.6 and reclamation and mine closure is discussed in Section 20.7.

20.2 ENVIRONMENTAL AND SOCIAL SETTING

20.2.1 Environmental Setting

Preliminary environmental baseline studies and monitoring programs have been conducted at the KSK CoW area since 2012. The main site-specific environmental baseline data collected at site include meteorology, surface water quality and terrestrial ecology. Climate, surface water quality and terrestrial ecology for the project area are described in the following sections.

20.2.1.a Climate

A manual rain gauge was deployed at site between April and December 2015. Daily rainfall measurements were recorded during this period. An automated weather station (Davis-VantagePro2) was installed in BKM in October 2012. The weather station was operational until early June 2013, at which time it was lost. Quality assurance/quality control information on the data collected is not available and therefore the site-specific climate data summarized in this section should be viewed in that context.

Monthly rainfall in the BKM area for the April to December 2015 period of record ranged from 3.5 mm/month to 608.6 mm/month (refer to Figure 20-1). The lowest rainfall occurred in September and the highest rainfall in November. The average monthly rainfall in dry season (from May to October) was 101.2 mm/month.

Average temperature at BKM for the period of record (October 2012 to June 2013) was 25°C. The highest temperature (25.6°C) occurred on May and the lowest (24.4°C) occurred in December (refer to Figure 20-2).

Wind speed in the BKM area during the period of record ranged from 0 m/s to 6.4 m/s (12.44 knot), with the highest wind speeds occurring in December. Based on Beaufort scale, this range of wind speeds is categorized as calm to moderate breeze.

Wind direction in BKM in the wet season is predominantly from the North East (refer to Figure 20-3).

Figure 20-1 BKM Rainfall (2015)

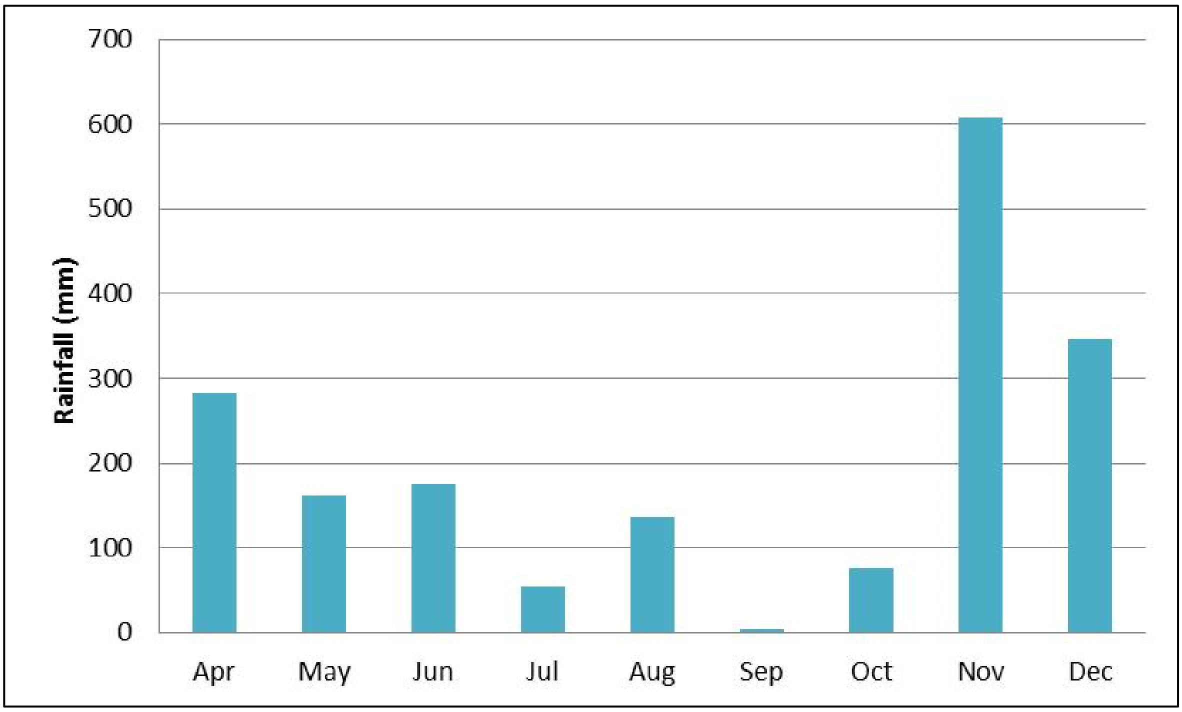


Figure 20-2 BKM Temperature (October 2012 to June 2013)

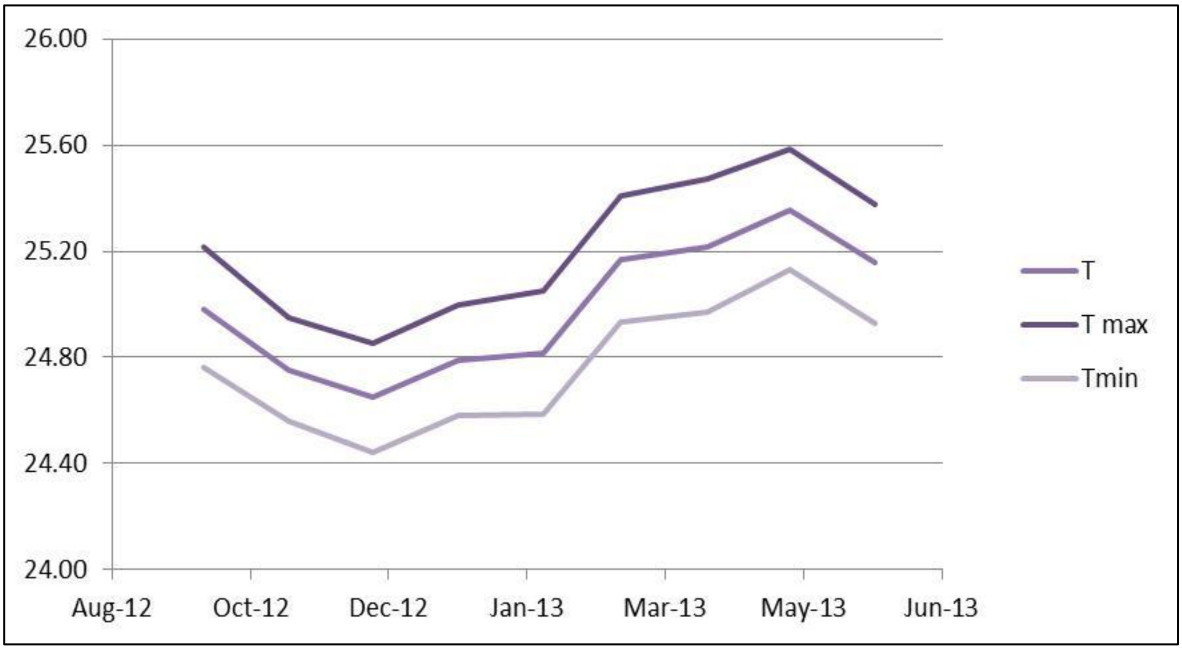
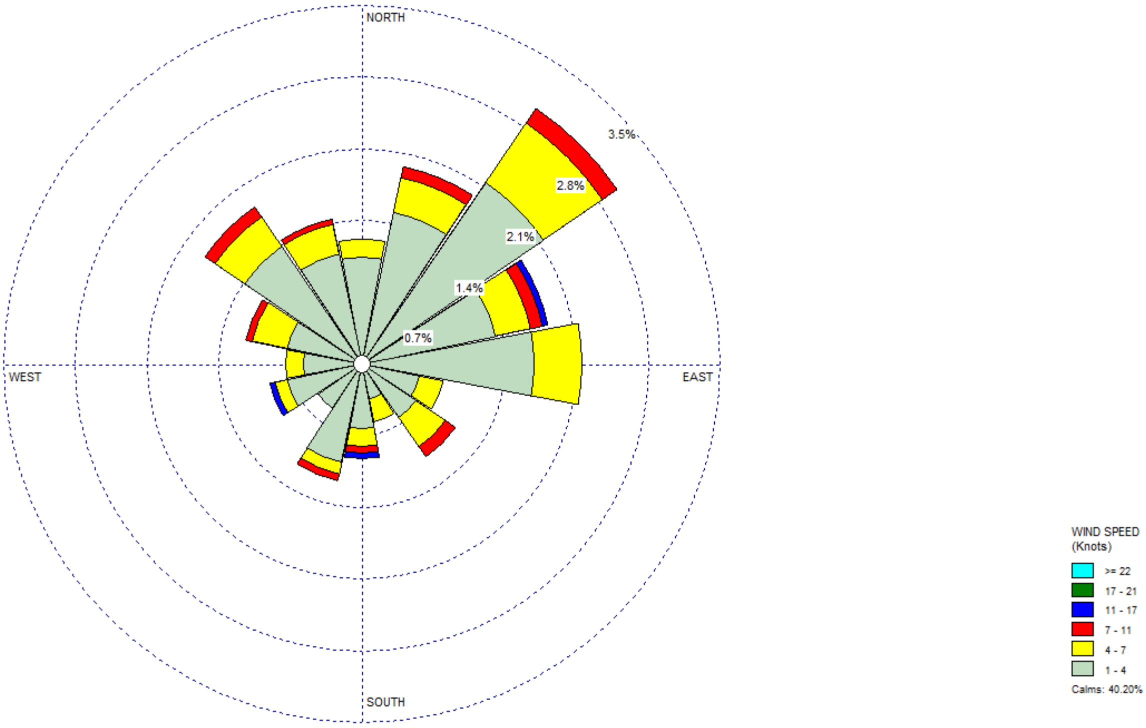
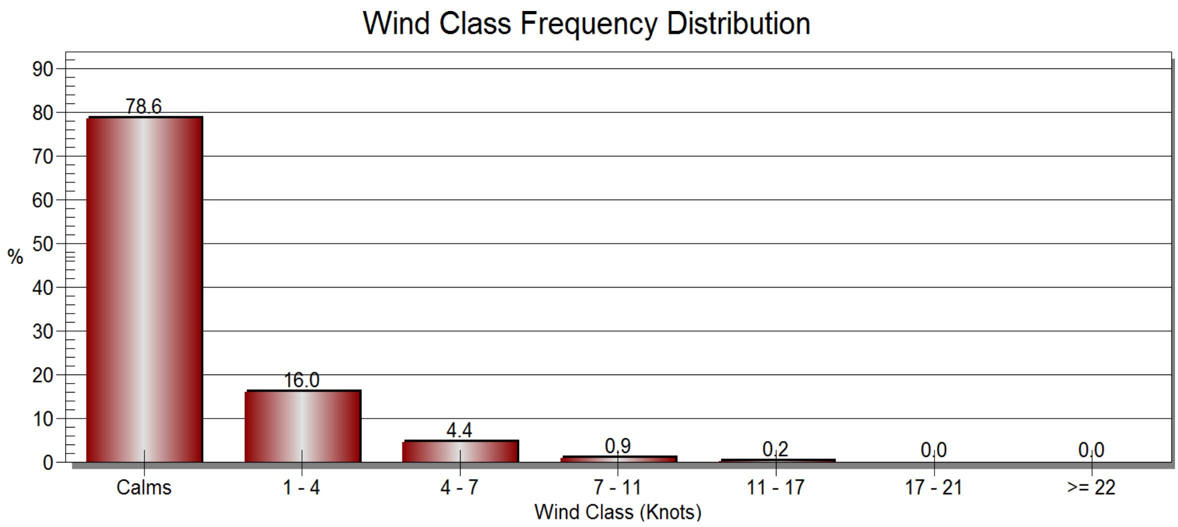


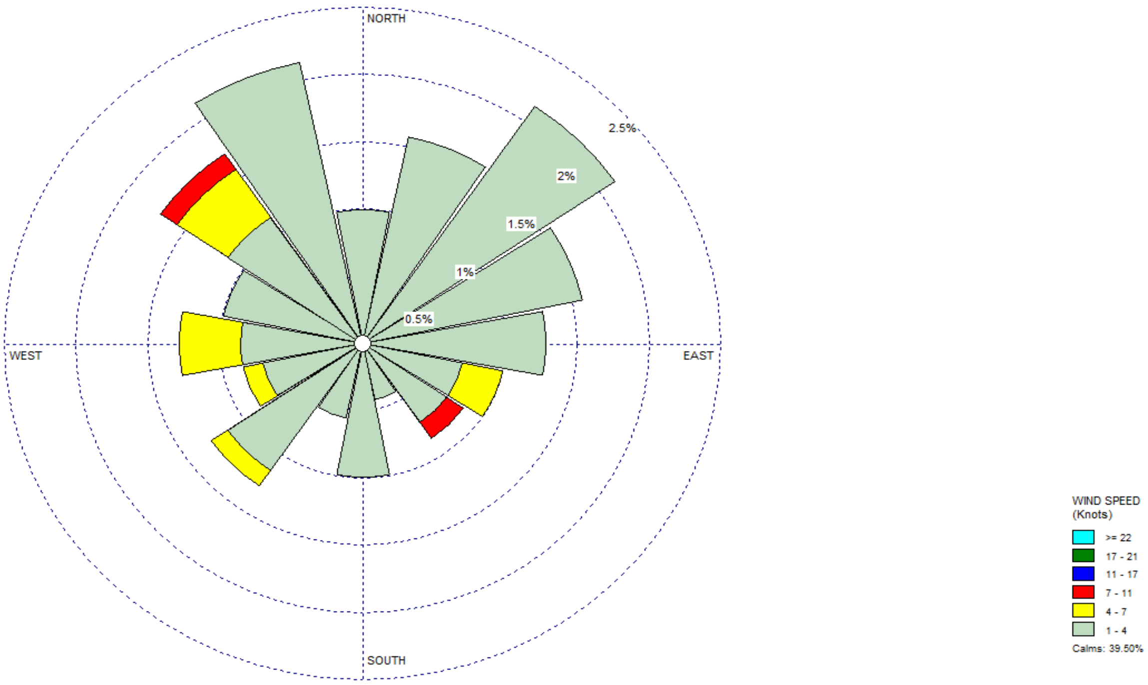
Figure 20-3 BKM Wind Speed and Direction in Wet Season

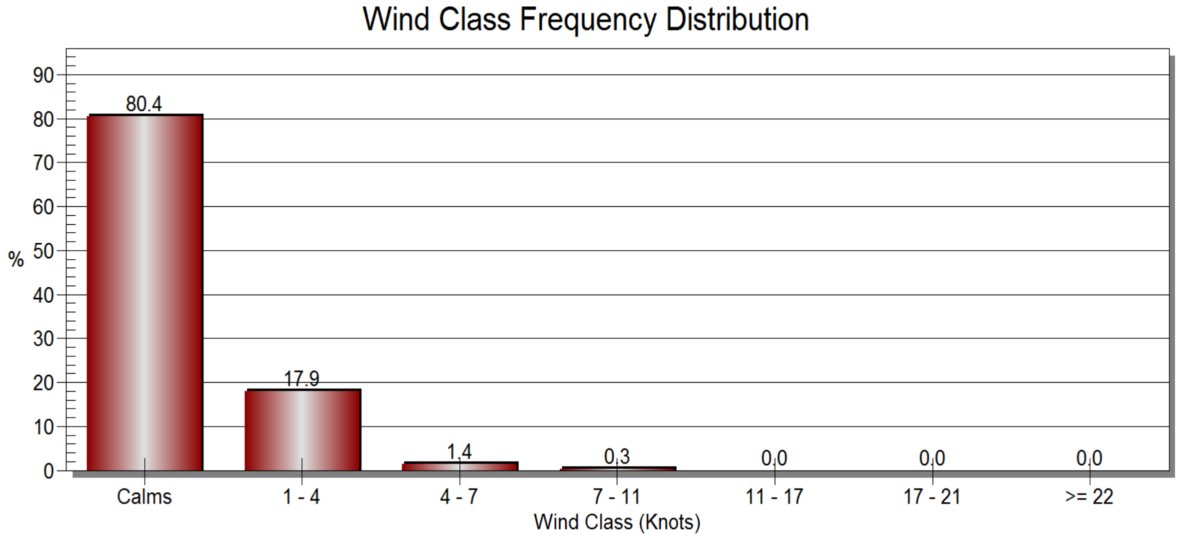




Wind speed and direction in the dry season is presented in Figure 20-4. Wind speed is typically lower in the dry season, with predominant wind directions being from the North East and North North West.

Figure 20-4 BKM Wind Speed and Direction in Dry Season





Available regional climate data were analyzed from the National Oceanic and Atmospheric Association (NOAA) station at Tjilik Riwut (approximately 190 km to the South East of Beruang Kanan) and are presented here to characterize regional climate data. Monthly climate data from January 2012 up to January 2016 from the NOAA regional station are summarized and presented in the Table 20-1.

Table 20-1 Monthly Climate Data at Tjilik Riwut Station (January 2012-2016)

Month	Temperature (°C)			Precipitation (mm/month)			Wind Speed (knot)		
	Average	Max	Min	Average	Max	Min	Average	Max	Min
January	27.01	35.22	21.72	185.12	269.75	45.47	3.34	8.4	0.5
February	26.84	35.28	21.61	166.18	230.89	32.51	3.2	6.7	1.2
March	27.05	36.22	19.61	207.45	329.18	90.17	3.11	7.9	0.2
April	27.38	35.61	22	195.96	393.95	62.99	3.06	9.4	0
May	27.54	34.28	21.28	142.05	250.44	9.65	2.84	6.1	0
June	27.41	35	21.22	113.54	162.05	6.35	2.64	6.1	0
July	26.83	35	20.61	69.28	197.61	8.13	3.46	6.3	0.8
August	27.01	35.39	20	32.7	56.64	9.14	4.02	6.1	0.5
September	27.35	35.28	19.61	39.56	100.08	0	3.79	6.8	1
October	27.71	35.72	19.28	130.24	203.2	22.61	3.66	6.5	1.4
November	27.33	35.78	21.39	178.63	329.95	19.56	3.24	6.7	0.5
December	27.05	35.78	22.72	294.83	442.98	185.42	3.26	6.9	0.4

Source: NOAA, 2016

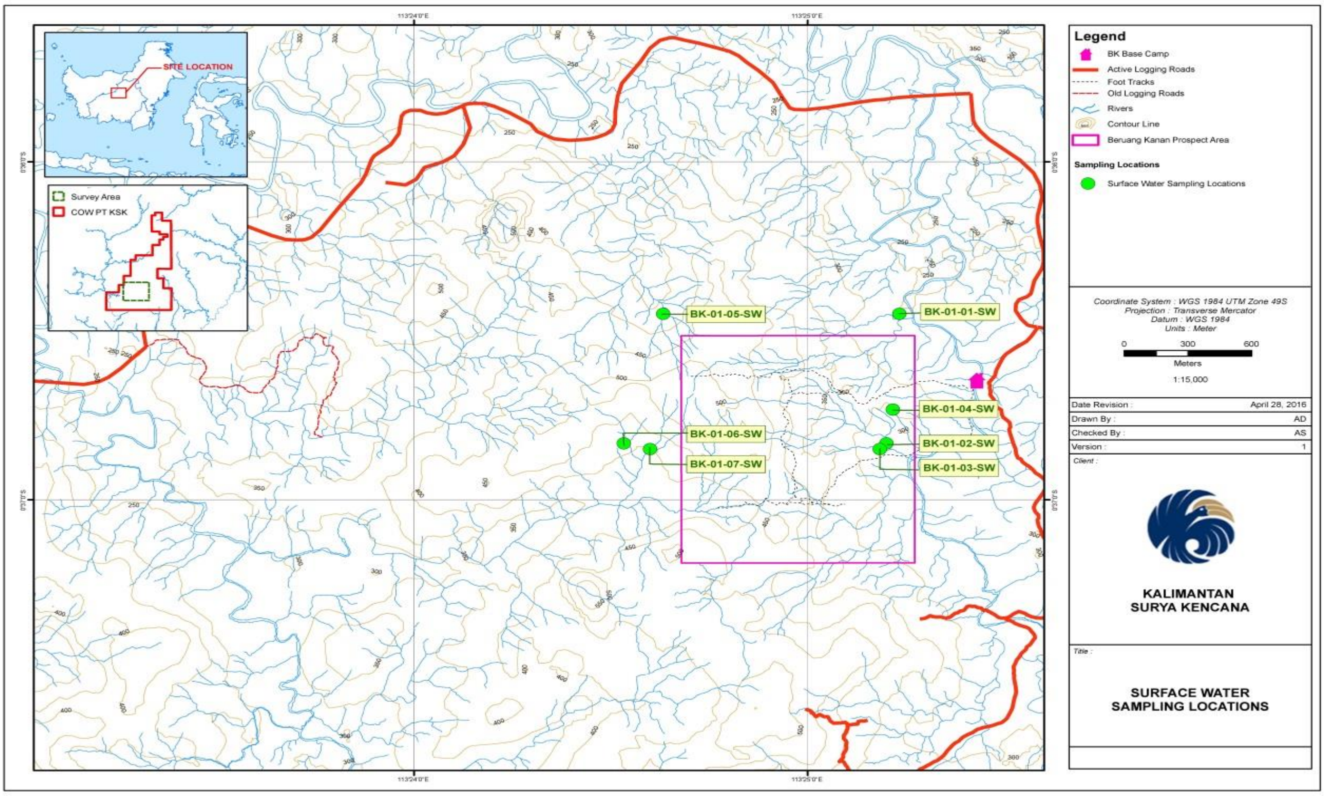
20.2.1.b Surface Water Quality

A baseline surface water quality monitoring program was conducted by KSK in May, July, and October 2012. These represent dry season and dry-wet season transition periods in the project area. A total of 213 river water samples from 89 locations in the CoW were collected and analyzed. The water quality analyses were conducted by Intertek Laboratory, Jakarta, which is an accredited environmental analytical laboratory.

From the total number of samples, water quality from 7 samples are presented and discussed in this section. These samples collected in May 2012 are from locations most proximal to the BKM area (refer to Figure 20-5) and have been analyzed for the full suite of analytical parameters necessary for comparison to national water quality standards.

The water quality results are discussed with reference to the standard of Government of Indonesia Regulation (PP No 82/2001 Class II) regarding “water quality management and water contamination control for water for recreational/infrastructures, freshwater fish cultivations, cattle breeding, agricultural irrigation, and/or other usages that requires the same water quality”, which is the applicable ambient water quality standard for the BKM Project.

Figure 20-5 Surface Water Sampling Locations



In situ water quality parameters (temperature, electrical conductivity, and pH) were measured in the field and are shown in Table 20-2. Based on the in situ measurement results, all parameters were within the applicable standard except for pH. The pH values show an acidic pH at all sampling locations, ranging from 2.33 to 4.74. These values are lower than the GoI Regulation No. 82/2001 standard (pH 6-9).

Table 20-2 In Situ Water Quality Field Measurement Results

PARAMETER	UNIT	STANDARD[1]	BK-01-01-SW	BK-01-02-SW	BK-01-03-SW
			May 2012	May 2012	May 2012
Conductivity	uS/cm	NP	17	51	28
pH	pH units	6 - 9	3.98	3.65	4.74
Temperature	˚C	± 3	24.4	24.7	24.5

Note: [1]GoI Regulation No. 82/2001 Class II; yellow shaded: Exceed the standard; NP: Non Proposed

Table 20-3 In Situ Water Quality Field Measurement Results (Continued)

PARAMETER	UNIT	STANDARD[1]	BK-01-04-SW	BK-01-05-SW	BK-01-06-SW	BK-01-07-SW
			May 2012	May 2012	May 2012	May 2012
Conductivity	uS/cm	NP	17	153.6	98.9	110.8
pH	pH units	6.5 - 9	3.29	3.18	2.33	3.43
Temperature	˚C	± 3	24.6	23.2	23.3	23.5

Note: [1]GoI Regulation No. 82/2001 Class II; yellow shaded: Exceed the standard; NP: Non Proposed

Laboratory analytical results for the same 7 samples are presented in Table 20-3. Similar to in situ measurement results, the laboratory result also show low pH values which ranged from 3.34 to 5.94 and are below the Indonesian water quality standard. Total dissolved solids (TDS) and conductivity were relatively low except at BK-01-05-SW, BK-01-06-SW, BK-01-07-SW, which have higher TDS and conductivity. Total Suspended Solids (TSS) concentration were low at all locations ranging from <1 to 7 mg/L, due to low stream flow during the dry season.

Nutrients such as ammonia, nitrate, nitrite, and total phosphate were all below the standard. In general, dissolved metal concentrations in surface waters were relatively low and below the Indonesian standard except for copper which has exceeded the Indonesian standard at 4 out of the 7 locations.

Laboratory analytical results for E.Coli were below the standard at all locations. While, from the 7 samples analyzed for microbiology (total coliform) 2 samples (>2,420 MPN/100mL) exceeded the analytical limit (which is not suitable for comparison to the standard, which is higher than the maximum analytical limit).

The naturally depressed pH and elevated copper concentrations in area streams reflect the natural acid rock drainage/metal leaching of the surficial oxidized/supergene deposit. The local aquatic ecology would have adapted to these site-specific water quality conditions, in the immediate vicinity of the BKM deposit.

Table 20-4 Surface Water Quality Results

Parameters	Units	Standard[1]	BK-01-01-SW May 2012	BK-01-02-SW May 2012	BK-01-03-SW May 2012	BK-01-04-SW May 2012	BK-01-05-SW May 2012	BK-01-06-SW May 2012	BK-01-07-SW May 2012
Physical									
pH	-	6.0-9.0	5.63	5.10	5.94	5.73	3.34	3.56	3.41
TDS	mg/L	1000	14	42	24	14	110	68	77
TSS	mg/L	50	<1	<1	<1	3	2	2	<1
Turbidity	NTU	NP	0.6	<0.5	<0.5	<0.5	1.8	2.4	0.6
Anions									
Acidity as $CaCO_3$	mg/L	NP	24	11	4	6	25	15	19
Alkalinity as $CaCO_3$	mg/L	NP	4	40	2	31	<1	<1	<1
Chloride, Cl-	mg/L	NP	<0.5	<0.5	<0.5	<0.5	0.6	<0.5	<0.5
Fluoride, F-	mg/L	1.5	0.02	<0.02	<0.02	<0.02	<0.02	0.03	0.02
Sulphate, SO_4^{2-}	mg/L	NP	8	24	17	6	30	32	20
Sulphide as H_2S	mg/L	0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002
Total Cyanide, CN	mg/L	0.02	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Nutrients									
Ammonia, NH_3-N	mg/L	NP	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02	<0.02
Nitrate, NO_3-N	mg/L	10	0.046	0.108	0.090	0.061	0.052	0.074	0.129
Nitrite, NO_2-N	mg/L	0.06	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Total Phosphorus as P	mg/L	0.2	0.007	<0.005	<0.005	<0.005	<0.005	<0.005	<0.005
Dissolved Metals									
Aluminium, Al	mg/L	NP	0.07	0.67	0.11	0.13	1.96	1.25	1.32
Antimony, Sb	mg/L	NP	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
Arsenic, As	mg/L	1	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	0.0012
Barium, Ba	mg/L	NP	<0.01	0.02	0.02	<0.01	0.01	<0.01	0.02
Boron, B	mg/L	1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Cadmium, Cd	mg/L	0.01	<0.0001	0.0002	<0.0001	<0.0001	<0.0001	<0.0001	<0.0001
Calcium, Ca	mg/L	NP	1.30	1.32	1.48	0.94	0.64	0.31	0.26
Chromium (VI), Cr^{6+}	mg/L	0.05	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002	<0.002

Parameters	Units	Standard[1]	BK-01-01-SW May 2012	BK-01-02-SW May 2012	BK-01-03-SW May 2012	BK-01-04-SW May 2012	BK-01-05-SW May 2012	BK-01-06-SW May 2012	BK-01-07-SW May 2012
Chromium, Cr	mg/L	NP	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Cobalt, Co	mg/L	0.2	<0.001	0.002	0.001	<0.001	0.002	0.002	0.001
Copper, Cu	mg/L	0.02	0.024	0.156	0.030	0.020	0.021	0.007	0.004
Iron, Fe	mg/L	NP	0.021	0.06	<0.005	0.22	0.299	0.058	0.204
Lead, Pb	mg/L	0.03	<0.001	0.004	0.001	<0.001	0.002	0.003	<0.001
Magnesium, Mg	mg/L	NP	0.59	0.66	0.94	0.35	0.85	2.74	1.10
Manganese, Mn	mg/L	NP	0.013	0.027	0.036	0.025	0.026	0.105	0.040
Mercury, Hg	mg/L	0.002	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005	<0.00005
Potassium, K	mg/L	NP	0.22	0.33	0.38	0.14	0.37	0.20	0.29
Selenium, Se	mg/L	0.05	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005	<0.0005
Sodium, Na	mg/L	NP	1.29	0.70	2.08	0.51	0.14	0.65	0.09
Zinc, Zn	mg/L	0.05	<0.005	<0.005	<0.005	<0.005	0.031	0.022	0.022
Miscellaneous									
BOD	mg/L	3	<2	<2	<2	2	2	2	2
Chlorine, Cl_2	mg/L	0.03	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
COD	mg/L	25	6	4	3	10	4	3	5
Surfactants, MBAS	µg/L	200	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Oil & Grease	µg/L	1000	<1	<1	<1	<1	<1	<1	<1
Total Phenols	µg/L	1	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Microbiology									
E.Coli	MPN/100mL	1000	60	5	4	32	2	6	3
Total Coliform	MPN/100mL	5000	1120	920	517	>2420	1730	>2420	435

Note: [1]GoI Regulation No. 82/2001 Class II; yellow shaded: Exceed the RI[1] standard; NP: Non Proposed.

20.2.1.c Flora and Fauna

A terrestrial flora and fauna survey of the BKM area was conducted in January 2016 by PT Lorax Indonesia. The complete flora and fauna baseline report is titled *PT Kalimantan Surya Kencana Flora and Fauna Ecology Study* (PT Lorax Indonesia, Jan 2016).

Objectives of the study were to:

i) characterize the baseline terrestrial ecology of the project area;
ii) provide input to development of a site-specific biodiversity management plan; and,
iii) provide a general reference for future environmental management strategies.

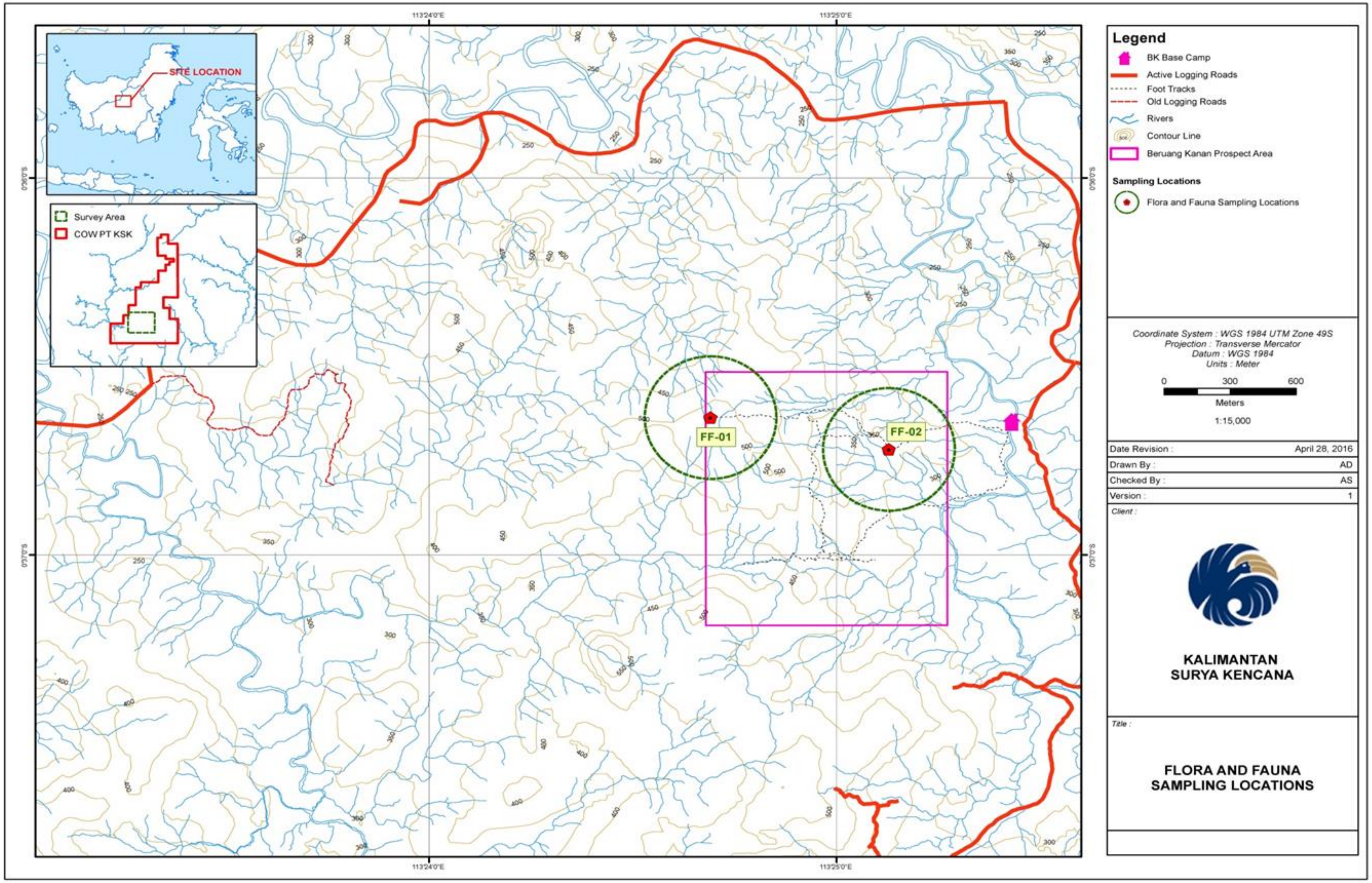
Flora and fauna observations were held at 2 locations in the project area (refer to Figure 20-6). The survey timing (January) corresponded to the wet (rainy) season. The locations were selected in order to cover the range of altitude in the project area and the locations of future mining development. The survey locations are: FF1 – Beruang 1 (500 m ASL) and FF2 – Beruang 2 (300 m ASL).

Flora survey was conducted using belt transect (combination of line and plot transects). Data collection for each location was conducted by observation of plant species found in the predetermined transects as well as in the surrounding area and photo-documentation of the terrestrial habit. The species were identified with reference to published literature Quantitative method for data calculation include analysis of Important Value Index (IVI), Species Richness Index (DMg), Diversity Index (H'), Evenness Index (E) and Dominancy Index (C).

Fauna data collection included name of species, number of individual species, and locations. Mammals and birds data were collected by recording species along a total of 1,000 m trackline, placement of 1 mist net, and recording of species outside of established sampling locations (opportunistic observation). Data were analyzed by qualitative and quantitative descriptive methods. Quantitative analysis for mammals, birds and herpetofauna include species Richness Index (DMg), Diversity Index (H'), and Evenness Index (E).

Conservation status of encountered flora and fauna species were assessed against international and national conservation regulations. National standard refers to Indonesian Government Regulation No. 7/1999 on Plants and Animals Conservation. International references were the International Union for the Conservation of Nature (IUCN) Red List and Convention on International Trade in Endangered Species (CITES) Appendices.

Figure 20-6 Flora and Fauna Survey Study Locations



Flora

FF1 is located in western part of BKM area within a hilly-steep area (slope of 40%). The area is characterized by secondary forest with some patches of primary forest. FF2 is located in an ex logged forest area within the lower undulating topography (slope of 4%), characterized by secondary forest vegetation.

Flora encountered in the study area from total of 24 plots were 187 species from 43 families. Density index shows vegetation structure at the sampling locations is in a healthy condition. Most of all stages of plants have high species diversity. Potency of total stands at FF2 has greater volume value than FF1. The number of individuals/ha for FF1 is higher than FF2. In general, the composition of plants in the study area is dominated by the family of Dipterocarpaceae (dipterocarps). The flora assemblage at FF1 is more diverse and rich than at FF2. Index of similarity shows a low similarity in vegetation types between the two habitats.

Fauna

Twenty-one mammal species from 14 families were recorded during the survey. FF2 has the higher richness and diversity of mammal species compared to FF1. In general, the abundance and diversity of mammals in the study area is relatively low. Ungulates (i.e., deer) encountered indicates food for these types of mammals in the study area is abundant. The presence of primates indicates the secondary forest in the area is in good condition. High number of large trees with continuous canopy provides an appropriate habitat for primates.

Eighty-six bird species from 33 families were encountered in the study area. Encountered bird species can be divided into 4 group of habitat: 47 species were forest birds, 29 species were bush birds, 2 species were Riparian birds, and 8 species were canopy top birds. Location FF2 has higher richness and diversity of bird species than FF1. Abundance of birds are high both in FF1 and FF2.

Conservation Status

None of the encountered flora species are listed in CITES appendices. Twenty-five species are included in IUCN categories and 4 species included in PP No.7/1999 (Table 20-4). Twenty-five species included in IUCN falls into 4 categories: 7 species as Critically Endangered (CE), 2 species as Endangered (E), 4 species as Vulnerable (Vu) and 10 species as Low Risk/Least Concern (LR/LC). Four flora species that are protected under Indonesian Regulation No.7/1999 are *Nepenthes amabilis, Shorea beccariana, Shorea palembanica,* and *Shorea pinanga*.

Table 20-5 Conservation Status of Encountered Flora

No.	Species	IUCN Conversation Status	PP No. 7/1999
1	*Durio acutifolius*	Vu	
2	*Santiria laevigata*	LR/LC	
3	*Calophyllum soullattri*	LR/LC	
4	*Hopea beccariana*	CE	
5	*Shorea cordata*	CE	
6	*Shorea johorensis*	CE	
7	*Shorea laevis*	LR/LC	
8	*Shorea lamellata*	CE	
9	*Shorea leprosula*	E	
10	*Shorea palembanica*	CE	+
11	*Shorea pauciflora*	E	
12	*Shorea peltata*	CE	
13	*Shorea sagittata*	CE	
14	*Shorea uliginosa*	Vu	
15	*Shorea multiflora*	LR/LC	
16	*Gonystylus bancanus*	Vu	
17	*Engelhardtia serrata*	LR/LC	
18	*Aglaia tomentosa*	LR/LC	
19	*Dysoxylum alliaceum*	LR/LC	
20	*Memecylon myrsinoides*	Vu	
21	*Horsfieldia irya*	LR/LC	
22	*Dacrydium elatum*	LC	
23	*Anisophyllea disticha*	LR/LC	
24	*Prunus arborea*	LR/LC	
25	*Ternstroemia penangiana*	Vu	
26	*Nepenthes amabilis*	N.A.	+
27	*Shorea beccariana*	N.A.	+
28	*Shorea pinanga*	N.A.	+

Note: CE = Critically Endangered; E = Endangered; LC = Least Concern; LR = Low Risk; Vu = Vulnerable; N.A. = Not Available; + = Protected under PP no.7/1999

Seven of 21 mammals species encountered in the study area have conservation status (Table 20-5). Five species are included in IUCN Red List, five species are listed in CITES appendices and 4 species are protected under Government of Indonesia Regulation No. 7/1999. Species included in all three IUCN, CITES, and Indonesia Regulation are Bornean Gibbon (*Hylobates muelleri*), Sun Bear (*Helarctos malayanus*), and Sambar Deer (*Rusa unicolor*).

Twenty-one of 86 bird species have conservation status (Table 20-5). Eighteen species are listed on IUCN Red List, 6 species are listed in CITES Appendix II, and 11 species are protected according to the Government of Indonesia Regulation No. 7/1999. Great Argus (*Argusianus argus*) is included in all three conservation references.

Table 20-6 Conservation Status of Encountered Fauna (Mammals and Birds)

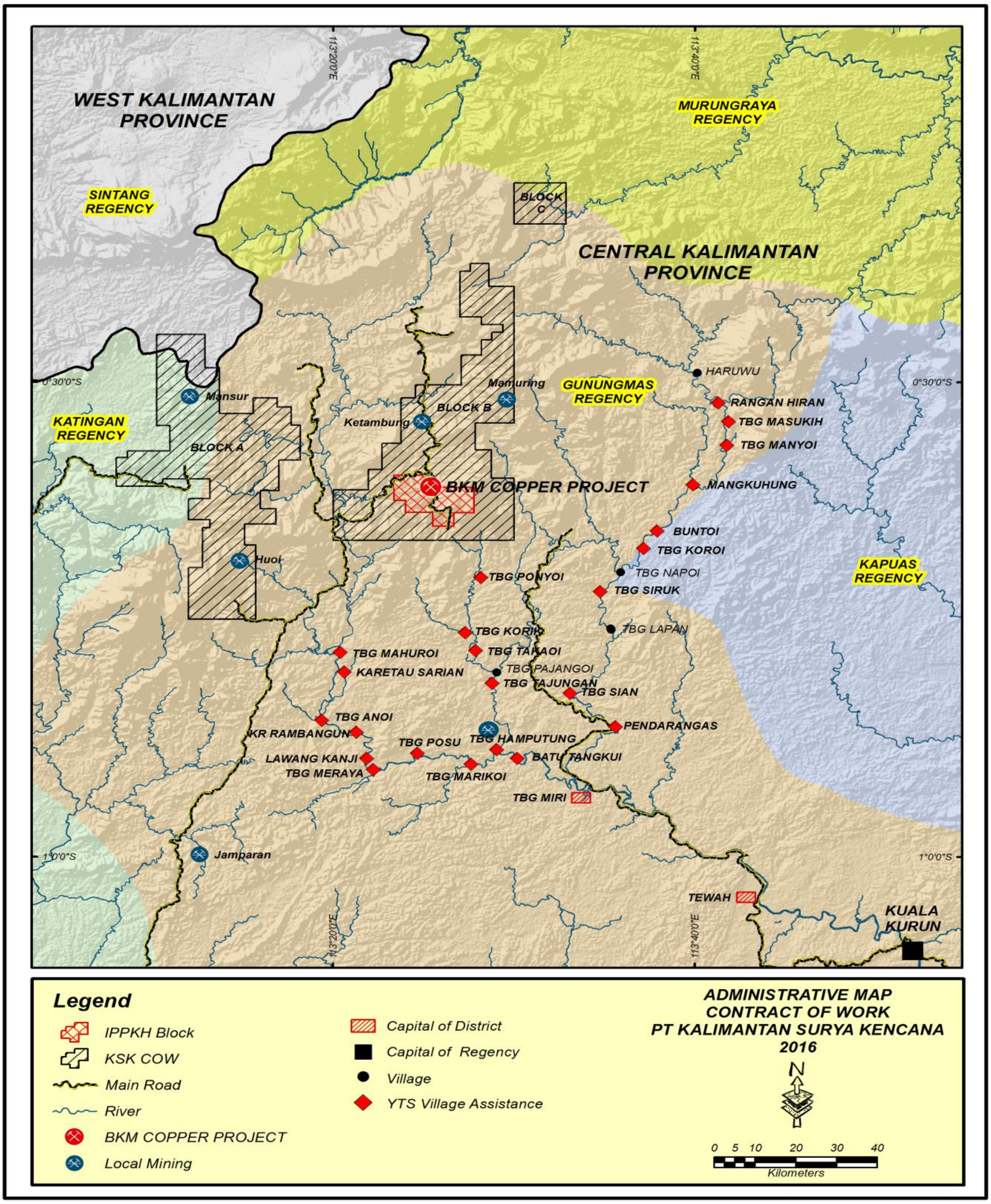
No.	Species	General Name	Status		
			IUCN	CITES	PP
Mammals					
1	*Macaca fascicularis*	Long-tailed Macaque		App II	
2	*Macaca nemestrina*	Pig-tailed Macaque	Vu	App II	
3	*Hylobates muelleri*	Bornean Gibbon	EN	App I	+
4	*Helarctos malayanus*	Sun Bear	Vu	App I	+
5	*Sus barbatus*	Bearded Piq	Vu		
6	*Muntiacus muntjak*	Red Muntjac			+
7	*Rusa unicolor*	Sambar Deer	Vu	App II	+
Birds					
1	*Spilornis cheela*	Crested Serpent Eagle		App II	+
2	*Argusianus argus*	Great Argus	NT	App II	+
3	*Treron capellei*	Large Green Pigeon	Vu		
4	*Loriculus galgulus*	Blue-crowned Hanging Parrot		App II	
5	*Rhopodytes diardi*	Black-bellied Malkoha	NT		
6	*Rhyticeros undulatus*	Wreathed Hornbill		App II	+
7	*Anthracoceros albirostris*	Oriental Pied Hornbill		App II	+
8	*Megalaima rafflesii*	Red-crowned Barbet	NT		
9	*Megalaima mystacophanos*	Red-throated Barbet	NT		
10	*Dinopium rafflesi*	Olive-backed Woodpecker	NT		
11	*Eurylaimus ochromalus*	Black and yellow Broadbill	NT		
12	*Chloropsis cyanopogon*	Lesser Green Leafbird	NT		
13	*Pycnonotus cyaniventris*	Grey-bellied Bulbul	NT		
14	*Setornis criniger*	Hook-billed Bulbul	Vu		
15	*Pityriasis gymnocephala*	Bornean Bristlehead	NT		
16	*Malacopteron affine*	Sooty-capped Babbler	NT		
17	*Napothera atrigularis*	Black-throated Wren-Babbler	NT		
18	*Alcippe brunneicauda*	Brown Fulvetta	NT		
19	*Rhinomyias umbratilis*	Grey-chested Jungle Flycatcher	NT		
20	*Ficedula dumetoria*	Rufous-chested Flycatcher	NT		
21	*Philentoma velatum*	Maroon-breasted Philentoma	NT		
22	*Gracula religiosa*	Common Hill Myna		App II	+
23	*Anthreptes singalensis*	Ruby-cheeked Sunbird			+
24	*Hypogramma hypogrammicum*	Purple-naped Sunbird			+
25	*Cinnyris jugularis*	Olive-backed Sunbird			+
26	*Aethopyga siparaja*	Crimson Sunbird			+
27	*Arachnothera longirostra*	Little Spiderhunter			+
28	*Arachnothera flavigaster*	Spectacled Spiderhunter			+
29	*Prionochilus thoracicus*	Scarlet-breasted Flowerpecker	NT		

Note: NT= Near Threatened; Vu= Vulnerable; EN= Endangered; App I= Appendix CITES I; App II= Appendix CITES II; + = Protected under PP no.7/1999

20.2.2 Social Setting

The latest census (2013) for Gunung Mas shows the regency (*Kabupaten*) has a population of 154,084 persons. The regency is divided into 12 subdistricts (*Kecamatan*) with 12 towns and 115 villages. The exploration block (Block B) falls within the area of three subdistricts: Damang Batu, Kahayan Hulu Utara, and Miri Manasa (refer to Figure 20-7).

Figure 20-7 Project Location, Settlements and Artisanal (Local) Mining Sites



There are no permanent settlements within the exploration block, and the nearest established villages lie to the South and Southeast, along the Kahayan, Hamputung and Miri rivers. These villages are inhabited by indigenous Dayak communities that are predominantly of Christian faith but also adhere to the traditional Kaharingan belief system. The tribal and language group in this area is known as *Ot Danum*. In accordance with the traditional belief system, this cultural group maintains a close connection with nature and deep respect for the spirits of the ancestors. Sacred sites include ancestral sites (*Kaleka*), the residence places of natural spirits (*Keramat*), and burial sites (*Sandung*), among others. Although such sacred sites may often lie at a considerable distance from settlements, there is no evidence of the existence of any heritage sites with archeological interest or significance within the KSK CoW.

These indigenous communities practice shifting rice cultivation and cultivate jungle rubber. Traditionally, the forests in the exploration area have been used for seasonal hunting as well as harvesting of non-timber forest products. However, there has been a significant shift away from traditional livelihood activities over the last twenty years: towards far greater reliance on artisanal gold mining as a primary source of income. As a result, there are great numbers of alluvial gold miners working along the upper Kahayan River and its tributaries. There may be more than 1,000 dredges active along these streams (Figure 20-7). These are all highly mobile units, and many are seasonally active within the company's exploration area.

Traditionally, the indigenous peoples of Central Kalimantan have had free access to the land, water, forests, and other natural resources in their territories. With the advent of large scale industrial development this has radically changed, particularly in recent years with the development of palm oil plantations. Access to land and forest resources has become much more restricted. In reaction to this, and to ensure that indigenous people continue to have access to land surrounding their villages, the provincial government has instituted initiatives to guarantee indigenous communities retain free access to local land resources.

In Central Kalimantan, the Governor's Regulation No. 13/2009 provides for individual and communal rights to *"manage, harvest and utilize the natural resources and/or the products, on the ground or in the surface in the forest area outside of the customary land."* The Provincial Government has also introduced several regulations that enable indigenous communities to gain access to forest and non-timber forest products.

In addition, the Central Kalimantan Government Regulation No. 5/2015 concerns the Central Kalimantan Spatial Plan for 2015-2035, including customary rights on land: namely 600,000 ha of customary forest as part of the protected territories of the province, 900,000 ha customary land as part of the agricultural cultivation area and Dayak Misik program, and 300,000 ha as customary residential area. This land is to be directly managed by Dayak people.

20.2.3 Future Environmental and Social Studies in Support of Permitting

Extensive additional environmental and social studies at site are required in support of project permitting. Permitting requirements are described in Section 20.6. Studies and assessments summarized in this section will provide the requisite input to the Feasibility Study, Environmental, Social and Health Impact Assessment (ESHIA) and Closure and Reclamation plans for the BKM Project.

At a minimum, future environmental studies will include:

- *Meteorology* – full suite of climate data (rainfall, wins speed and direction, temperature, solar radiation, humidity and evaporation) need to be collected for a minimum of one year (preferably longer) as input to site water balance and development of water management plans for the site.
- *Regular surface water quality monitoring* – expansion and continuation of surface water quality monitoring program is required for establishing site-specific ambient water quality compliance standards and as input to the water management plan.
- *Hydrology* – stream flow/discharge measurements need to be collected (concurrent with meteorological data collection) as input to the site water balance and design of water management structures.
- *Hydrogeology* – determination of the physical and chemical hydrogeology of the area to determine groundwater yields (potential source of water for the project), establishing baseline chemical groundwater quality and associated site-specific compliance criteria and as input to the water management plan.
- *Acid rock drainage/metal leaching (ARD/ML)* – A full geochemical characterization study consisting of static testing, metal leaching and kinetic laboratory and field tests is needed to determine source terms (waste rock, ore and pit walls) as input to water quality modelling and development of waste and water management plans.
- *Soils* – the physical and chemical characterization of the soil is needed as input to the top soil management plan and reclamation planning.
- *Dry season flora and fauna monitoring* – the dry season survey would complement the wet season flora and fauna survey (January 2016) to provide a seasonal assessment of flora and fauna in the project site and as input to the biodiversity management plan.
- *Dry and wet season aquatic ecology* – seasonal aquatic ecology (algae, plankton, benthos, fish and fish tissue metal contents) surveys are required to establish the baseline aquatic ecology of project area streams and as input to the aquatic resources management plan and water management plan.
- *Dry and wet season air quality and noise* – baseline air quality and noise characterization of the project area is needed as input to the air quality and noise management plans.

At a minimum, future social studies will include:

- *Demographics* - statistical data for local residents are required for the social impact assessment.
- *Livelihood* – sources of income/livelihood data are required for the social impact assessment and development of CSR programs.
- *Economics* – economic data for the area are required for the social impact assessment and development of CSR programs.

- *Cultural heritage* – a cultural-heritage assessment is required in support of the social impact assessment and as input into the cultural heritage plan.

At a minimum, future community health studies will include:

- *Public health* – An assessment of prevalent diseases in the area is necessary as input to community health management plan and for development of targeted CSR programs.
- *Biomarker study* – an assessment of selected (with a focus on mercury) metals in local residents is needed to establish baseline levels (which may be elevated due to artisanal mining activities) and as input to the community health management plan.

In addition to the above studies at site, environmental and social studies would need to be conducted in the area of the proposed port/LCT landing. These would include riverine physical, chemical and biological studies, terrestrial studies in the vicinity of associated on-land infrastructure and social assessment in nearby settlements in the proposed port area.

20.3 Environmental and Social Effects

A comprehensive assessment of the potential environmental and social impacts (both negative and positive) from the project will be determined subsequent to further project definition, collection of requisite baseline data (refer to Section 20.2.3) and compilation of the Environmental, Social, and Health Impact Assessment (ESHIA – refer to Section 20.6). Significant anticipated environmental and social impacts are presented in sections 20.3.1 and 20.3.2, respectively.

20.3.1 Environmental Effects

Potential environmental effects from the project include:

- Visual changes to landscape, due to changes in topography (pit, waste dump, heap leach etc.)
- Impacts to air quality from dust and movable (vehicles) and immovable emissions
- Noise from construction and operations
- Stream flow (hydrology) pattern changes due to diversions
- Impacts on aquatic resources (surface water, groundwater and aquatic biota)
- Impacts on terrestrial ecology (flora and fauna)

Significant impacts anticipated at this stage of the project are:

1) sedimentation and increase in total suspended solids in local streams;
2) ARD/ML and;
3) localized loss of flora and fauna.

Each is described in more detail below.

20.3.1.a Sedimentation

Clearing of vegetation from the land for the proposed mine will result in increased surface runoff, soil erosion and increase sediment load in the local streams which may impact aquatic resources. This is a well-known consequence of deforestation and is most pronounced during heavy rains in the wet season. In regards to the BKM Project, this impact is expected to be most significant during the early stages of construction, as the full water management structures (environmental control dam(s)) would not be operational and progressive reclamation would not have been initiated.

Management strategies to address this potential impact are described in Section 20.4.

20.3.1.b ARD/ML

The disturbance of bedrock for mining purposes exposes fresh rock surfaces to oxidizing conditions. Under these conditions, weathering of Fe-sulphide minerals may lead to the release of acidity causing ARD. The most common sulphide mineral is pyrite which weathers according to the following reaction:

$$FeS_2 + 15/4\ O_2 + 7/2\ H_2O = Fe(OH)_3 + 4\ H^+ + 2\ SO_4^{2-}$$

Other, less common sulphide phases such as pyrrhotite and chalcopyrite also release acid upon oxidation, albeit at lower concentrations. The oxidation of non-Fe sulphide minerals may not directly result in ARD, however, leaching of high concentrations of metals from these minerals may still be of environmental concern.

Conversely, certain mineral classes have the capability of neutralizing acidity when dissolved and their presence and abundance controls the likelihood and timing of ARD onset. Due to their relatively high dissolution rates, carbonate minerals are by far the most important group of neutralizing agents commonly observed in most environments. This process can generally be described by two reactions:

at pH<6.3:

$$FeS_2 + 15/4\ O_2 + 7/2\ H_2O + 2\ [Ca, Mg]CO_3 = Fe(OH)_3 + 2\ SO_4^{2-} + 2\ H_2CO_3^{-} + 2\ [Ca^{2+}, Mg^{2+}]$$

at 6.3<pH<10.3:

$$FeS_2 + 15/4\ O_2 + 7/2\ H_2O + 4\ [Ca, Mg]CO_3 = Fe(OH)_3 + 2\ SO_4^{2-} + 4\ HCO_3^{-} + 4\ [Ca^{2+}, Mg^{2+}]$$

Kinetically much slower-dissolving silicate minerals may additionally counteract ARD, particularly in carbonate-deplete regimes with slow acid production. It is important to note that while low pHs generally enhance the solubility of many species of environmental concern (e.g., Cu, Zn, Ni, Pb, etc.), certain elements are similarly or more mobile under neutral conditions (e.g., Mo, Se, As, Sb) causing potentially deleterious drainage in pH-neutral environments.

Potential sources of ARD/ML are from the pit walls and waste dumps during operations and closure and from the heap leach post-closure. Management strategies to address this potential impact are described in Section 20.4.

20.3.1.c Terrestrial Biodiversity

Clearing of vegetation during construction will result in a localized impact to native terrestrial flora and fauna species. As discussed in Section 20.2.1.c, the BKM areas is a degraded environment with significant prior impact (deforestation) by human activity, however there are species present which are of biodiversity value and must be protected. Mobile fauna will avoid land clearing activities but flora on the footprint of the proposed mine would be temporally lost. Progressive and post-closure reclamation will restore the vegetation on most of the site, with the exception of the open pit that will form a pit lake at closure (Section 20.7).

Management strategies to address this potential impact are described in Section 20.4.

20.3.2 Social Effects

Mine development will have positive as well as potential negative social impacts for local communities and the region. Positive social impacts include:

- Increased revenue for local, provincial and central government of Indonesia though payments of taxes and royalties.
- Increase employment opportunities for local residents and mining professional from other regions in Indonesia.
- Opportunities for local contractors and supplier as a part of KSK's supply chain.
- Improvements to infrastructure and services - whereas the district, provincial, and national governments have the overall responsibility to provide water, roads, schools and health clinics, the project will nonetheless enhance infrastructure and services in alignment with government plans through a community infrastructure fund.
- Training and education - benefits will also result from CSR programs that seek to strengthen management bodies in the community, invest in training and education, and build capacity through partnerships.
- Community Health - benefits will also result from CSR programs that seek to improve healthcare and disease prevention, including awareness programs regarding the hazards of mercury use for artisanal miners.

Potential negative social impacts include:

- Loss of access to forest resources - although the development of a mine may restrict some access and harvesting rights to forest resources, this is only true for a small area of the exploration block. Most of the area of the concession lends itself to customary use as described previously. Continued access rights to these areas will foster good relations and the trust of the local Dayak communities.
- In migration and social conflict - greater road access may also create opportunities for the influx of newcomers, leading to potential social conflicts between locals and non-locals. Uncontrolled in-migration will therefore be reduced by minimizing new road construction and by locating company workcamps at a considerable distance from local villages ensuring less social impact and greater preservation of cultural identity.
- Social conflict due to perceptions of unfair hiring/termination practices.

Management strategies to address these potential impacts are described in Section 20.5.

20.4 ENVIRONMENTAL MANAGEMENT

Potential management strategies to eliminate, and where not possible, mitigate anticipated significant environmental impacts described in Section 20.3.1 is described in this section.

20.4.1 Sediment Control

The following are potential measures to mitigate potential impacts of increased turbidity and suspended solids in local streams associated with construction and mining activities:

- Schedule the majority of construction to be conducted during the dry season;
- Minimize land clearing and disturbance to the minimum required for safe production purposes;
- Diversion of non-impacted runoff around areas of disturbance via engineered drainage systems that minimize the erosive energy of the contained water flows;
- Containment of impacted water within sumps and conveyance to the environmental control dam(s) to enable adequate settling of suspended materials prior to use or discharge;
- Deployment of localized erosion and sediment control techniques (check-dams in streams and Reno mattresses on stream banks);
- Deployment of static and active water treatment processes within drainage channels, sumps, ponds and the environmental control dam(s) that promote the rapid settlement of suspended solids; and,
- Progressive revegetation and reclamation of disturbed areas and topsoil stockpiles as quickly as possible.

20.4.2 ARD/ML Management

The primary objective of waste rock ML/ARD management is to eliminate acid generation, and where not possible, to improve the quality of infiltration and run-off water that comes into contact with the waste rock.

These objectives are generally achieved by configuring long-term storage facilities that reduce the exposure of potential acid forming (PAF) rock to infiltrating waters and/or limit the oxidation of sulphidic waste rock and control the discharge of contact water. Best management practices that may be appropriate at the BKM project to limit the effects of ML/ARD on mine site contact water are outlined below.

20.4.2.a Non-Acid Forming Rock Use for Outer Shell and Base Construction

Constructing portions of the waste dump that have a high proportion of contact with water with inert or non-acid forming (NAF) rock. Portions of waste rock dumps that have the greatest interaction with water include the outer shell of the dump, the base of the dump and toe of the dump. Specifically:

- Restricting contact water at the base of the dump can be achieved by the selective handling and placement of NAF materials for construction of dump toe and lower lift of the dump. The required thickness of the NAF layer is dependent on site-specific conditions.
- Restricting contact water on the surface of the dump can be achieved by placement of NAF materials with a minimum thickness of 5 m perpendicular thickness on the outer shell to prevent direct contact of runoff water with the underlying potentially acid forming (PAF) material. Placement of NAF layers on the outer shell of the dump may also limit oxygen penetration into PAF materials in some instances, which will increase lag time before onset of acid conditions. However, effectively limiting oxygen penetration is dependent on site climate and material characteristics and cannot be expected in most situations without detailed engineering design and managed cover placement procedures. Designing a waste dump to limit water infiltration can be achieved by reshaping, drainage construction, topsoil placement and progressive revegetation of the dump. More comprehensive cover systems can also be considered for situations where infiltration rates <10% precipitation are targeted.

20.4.2.b PAF Encapsulation Cells

Isolation of PAF in cells within the waste rock dump with containment cell perimeter walls constructed of NAF rock, specifically:

- Placement of PAF materials within the walled cell in lifts of not greater than ten (10) meters thickness; and,
- Upon reaching design height for PAF storage, provision of a compacted clay or mudstone to provide oxidation control of PAF materials.

20.4.2.c Blending of PAF and NAF Materials

Blending of PAF material with acid neutralizing materials (NAF/limestone) to prevent the onset of acid conditions. In order to implement a blending strategy, a source of material with a high buffering capacity such as limestone or lime would need to be established.

20.4.3 Mine Water Monitoring and Management System

Develop and implement a mine water quality monitoring system for mine water generated from open pit and waste dump areas. The mine water generated from these areas should have mine water management infrastructure developed before discharging into an environmental control dam (or other management dams) to minimize the dilution and greater volume of contaminated water. If the quality of the mine water falls within the required standard for mine water discharge quality parameters, it can be released. If the quality exceeds the required standard mine water quality discharge parameters, the mine water should be managed in a separate mine water treatment system that either allows for reuse in the operation and/or treatment and discharge to the environment.

20.4.4 Biodiversity Management

Subsequent to completion of seasonal flora and fauna surveys at site and compilation of the ESHIA, KSK will develop a Biodiversity Management Plan for the project. This plan will adopt a hierarchical management approach, consisting of: 1) avoidance/minimization of impacts; 2) site reclamation/restoration; 3) biodiversity offsets; and, 4) environmental awareness programs. Each of the above management elements is summarized below.

- Avoidance/minimization of impacts by minimizing land clearing and disturbance to the minimum required for safe production purposes and by developing migration corridors for fauna, as necessary.
- Site reclamation/restoration – progressive reclamation of areas within the site will be conducted at the earliest opportunity in order to reestablish the native vegetation and thereby promote the recolonization of flora and fauna.
- Biodiversity offsets – KSK will look for opportunities to work with experts and conservation groups to contribute to offset programs that ideally result in measurable net gain of biodiversity in the central Kalimantan region.
- Environmental awareness programs - efforts in the conservation of biological diversity will involve environmental campaigns which are designed to raise public environmental awareness among KSK employees, subcontractors and the local communities. These programs will focus on the importance of conservation of biological diversity and will encourage the cessation of hunting/harvesting of endangered and threatened species. Research and collaboration activities with NGOs, universities and conservation agencies will be conducted as part of the overall program.

20.5 SOCIAL MANAGEMENT

KSK recognizes that a social license to operate a mine in Indonesia is of paramount importance to its sustainability. Potential management strategies to eliminate, and where not possible, mitigate anticipated significant negative social impacts and to enhance positive impacts described in Section 20.3.2, are described in this section.

The company will seek to avoid adverse social, economic and cultural impacts wherever possible and strive to deliver net benefits to affected indigenous communities through planned impact management and benefit sharing schemes. Benefit sharing will promote broader economic participation in projects, and provision of direct and indirect employment and business opportunities. Service provision work will also be an important benefit in an area where there are few sources of paid work. The company will seek to strengthen the community asset base and general well-being by providing jobs that will build skills and increase mobility. Having a local content plan in place will ensure that benefit-sharing takes place at the local level and has positive impacts on local society, economy and culture. KSK will be transparent in his hiring policies and practices, in order to mitigate perceptions of unfair hiring practices.

In 1997, the company established a community development foundation called Yayasan Tambuhak Sinta (YTS) with the purpose of ensuring the local people would benefit from any mineral development in the area. Since then, the foundation has actively engaged with both communities and local governments to strengthen their capacity to function more effectively. The foundation has been active in providing support to strengthen community, government, and other institutions, resulting in smoother running communities and stronger relationships with district government. There is now a strong connection between both levels and good support from government for community development initiatives. For the last four years, YTS has been providing technical assistance to district government to assist in improving their planning and budgeting processes, and their service delivery to communities. This has resulted in improved economic livelihoods, as well as social services, such as health and education. It has also strengthened the ties between district government and the communities where YTS has been working.

The YTS community development program presently engages with 328 households out of a total of 2155 households in the three subdistricts closest to the exploration block. YTS is now working in 22 of the 30 villages in this area, providing support for livelihood activities, such as fish farming, pig rearing, rubber cultivation, and vegetable growing. Some communities have been providing vegetables, fish and other local produce to the company’s exploration camps. The overall impact of the development program has been to improve community relations both with the company and the local government. There is now a strong and widespread support for the company and its activities.

The YTS community development program is conducted with the input, support and participation of both local communities and local government. The focus is on ‘community empowerment’ – i.e., to help communities gain the capacity to self-govern in an effective manner and be able to act as equal partners in the regional development process. Having Dayak leaders involved in decision-making is an important aspect of the community engagement process. Having a good understanding of the philosophy and approach of government also assists the company to navigate changes in the policy and regulatory environment for

Indigenous Peoples and their customary lands. The involvement of YTS ensures that impact mitigation and enhancement will be informed by working knowledge of the communities and will be conducted with their input, support and participation. Good program planning, with regular monitoring and evaluation, will ensure that any negative impacts of development will be mitigated by the community development program.

The company benefits the communities by providing employment and training, and by developing local supply chains to increase purchases from local suppliers in accordance with a local content plan. The company will continue to provide skill and enterprise development for employees, the wider community, and local suppliers. It will make a commitment to indigenous employment, and ensure that an effective grievance mechanism will be put in place to deal with community concerns. CSR programs will continue to provide grants and enterprise development opportunities in the neighboring communities as well as greater linkage to and leveraging of government resources to widen coverage of existing programs. Lastly, benefits will result from greater revenues provided to local and national governments with monitoring and support to local governments regarding the ways in which they use the revenue provided.

20.6 PERMITTING

20.6.1 Introduction

Key environmentally related approvals required in support of the construction permit for the BKM Project are described in this section. The BKM Project is located in an area designated as Production Forest, which allows development of open pit mining. Furthermore, given the final product from the mine will be copper cathode, Ministry of Energy and Mineral Resources (MEMR) Regulation No. 11 of 2012 regarding Value Added Mineral Activity Through Mineral Processing and Refinement (commonly known as the raw mineral export ban), which applies to unprocessed and semi-processed (concentrate) mining products, does not apply to the BKM Project. There are therefore no regulatory hindrances for the permitting, construction, operation and export of copper for the proposed BKM Project.

Key laws and regulations applicable to mine permitting in Indonesia are presented in Table 20-7. However the four key approvals in support of project permitting are:

1) Feasibility Study
2) ESHIA (called AMDAL in Indonesia)
3) Mandatory 5-Year Reclamation and Mine Closure Plans; and
4) Borrow-to-Use Forestry Permit (IPPKH).

Each of these requirements is described in the sections below.

Table 20-7 Relevant Legislation and Requirements

Relevant Legislation	Legislative Requirement
Law	
Local Government Act, No. 23/2014	This law states that local governments have the authority in terms of efficient use of natural resources and conservation efforts including authorization for permit approval (i.e. feasibility study, AMDAL, reclamation & closure mine, etc.).
Environmental Protection and Management Act, No. 32/2009	If conducting a business or activity that could potentially have a significant impact on the environment an Environmental Impact Analysis (*AMDAL*) process must be completed and an Environmental License obtained. Environmental License holders must manage their activities so as to ensure appropriate standards of environmental protection and management to maintain the sustainability of the environment and control pollution and/or environmental degradation. Environmental license holders must obtain appropriate permits for the disposal of waste and use of hazardous or toxic materials. Environmental license holders must conduct regular environmental audits and report periodically on environmental management activities and performance.
Health Act, No. 36/2009	This law regulates health issues, health practitioners, health facilities, including public health and labour.
Mineral and Coal Mining Act No. 4/2009	Replacement of CoW with Mining Business Licence (IUP). IUP holders shall: Develop and empower the local community; Guarantee the application of environmental quality standards consistent with the ecological characteristics of the area of operations; Preserve the function of water resources and carrying capacity in accordance with the applicable laws; Obtain rights to land with the title holder; Prepare and submit reclamation and closure plans and provide a guarantee fund for these activities; Prepare and submit periodic work plans and budgets and implementation reports.
Waste Management Act, No. 18/2008	Reference for domestic or solid waste management. Domestic waste management effort including 3R and public participation.
Spatial Planning Act, No. 26/2007	As a reference to the arrangement and management of spatial plans in Indonesia. This law states that spatial planning shall be based on the principle of integration; harmony, balance and sustainability.
Forest Act No. 41/1999 Jo No. 19/2004	Business entities must be granted a lease and permit for the purpose of exploration, survey or exploitation within forest areas and are obliged to rehabilitate the areas of disturbance progressively.
Water Resources Act, No. 7/2004	A license must be obtained for the construction of water resource infrastructure and the use of water resources for business purposes. Water resources must be protected from pollution and must be utilized efficiently.
Manpower Act, No. 13/ 2003	Manpower regulation regarding right of employees, company, work roster, age limitation, termination, salaries and benefit. Also regulates training, manpower placement, and foreign workers.
Building Act No. 28/2002	Safe construction of structures so as not to endanger persons, property or the environment.
Occupational Health and Safety Act, No. 1/1997	Employers must implement controls to prevent a person's death, injury or illness being caused by the workplace or activities.
Natural Resources and Ecosystem Act, No.5/1990	Reference implementation of the activities that the utilization of natural resources can take place in the best way possible, so that conservation measures remain to be implemented and the natural resources and ecosystems are maintained.

Relevant Legislation	Legislative Requirement
Government Regulation	
Government Regulation concerning the Management of Hazardous and Toxic Materials, No 101/2014	A business entity that is a producer, transporter, user or processor of hazardous and/or toxic waste shall complete a permitting process, prevent contamination or damage to the environment by the business activities and routinely report on the waste activities.
Government Regulation concerning Domestic Waste Management, No. 81/2012	Domestic waste management including 3R, transportation, temporary collection station, rural landfill and people's participation.
Government Regulation concerning Environmental and Social Responsibilities of Limited Liability Companies, No. 47/2012	Undertake and report environmental and social responsibilities programs in accordance with the company's annual work plan and budget.
Government Regulation concerning Environmental Licences No. 27/2012	Ensure that the planned business/activity adheres to the spatial plan. Prepare and submit an AMDAL for the proposed project. Prepare an AMDAL in accordance with the Ministerial guidelines.
Government Regulation concerning Implementation of Occupational Health and Safety Management Systems, No. 50/2012	Establish and implement a formal OHS Management system to ensure that OHS risks are appropriately managed to minimize workplace injury and illness.
Government Regulation concerning Reclamation and Post-Mining, No. 78/2010	Mining Production Operations Businesses must prepare and submit a 5 year Reclamation Plan and a Closure Plan for approval and provide a financial guarantee.
Government Regulation concerning Utilization of Forest Areas, No. 24/2010	Entities granted a Forest Area Borrow-to-Use License must comply with the obligations.
Government Regulation concerning the Implementation of Mineral and Coal Mining Businesses, No. 23/2010	Ensure that mining businesses are conducted in accordance with the issued Mining Business Licence (IUP). Provide compensation to land owners based on mutual agreement.
Government Regulation concerning Marine Environment Protection, No. 21/2010	Reference in carrying out efforts to prevent and control pollution of the aquatic environment originating from activities associated with shipping
Government Regulation concerning Forest Rehabilitation and Reclamation, No. 76/2008	Reference in reclamation and rehabilitation program with principles that cover: a. continuous budgeting system (multi-year); b. clarity of authority; c. understanding of the tenure system; d. share of costs (cost sharing); e. implementation of incentive systems; f. community empowerment and capacity institutional; g. participatory approach; and h. transparency and accountability.
Government Regulation concerning National Spatial Plan, No. 26/2008	The reference in determining the location of the implementation of the action plan in accordance with the applicable National Spatial Plan.
Government Regulation concerning Government Sharing Issue Between the Provincial Government and District / City Government, 38/2007	The division of government affairs, especially in the environmental, including study of environmental impact control and conservation of natural resources.
Government Regulation concerning Water Quality Management and Pollution Control, No. 82/2001	Maintain water quality and control water pollution at its source. Obtain a license for the disposal of waste water prior to undertaking this activity and comply with the applicable water quality standards.
Government Regulation concerning Hazardous Material Management, No. 74/2001	Guidance for management of Hazardous material includes generation, transporting, distribution storing, dispose and registration.
Government Regulation concerning Government Authority and Provincial Authority as Autonomous Region, No. 25/2000	Reference for local government authority / autonomous including permits approval. The Government's authority includes the authority in the field of foreign policy, defense and security, justice, monetary and fiscal, religion and authorities in other fields (utilization of natural resources and strategic high technology, conservation and national standardization).
Government Regulation concerning Air Pollution Control, No. 41/1999	Air pollution control includes the prevention and control of pollution, as well as the restoration of air quality by prevention of pollution from mobile and stationary sources. Provides ambient air quality standards.
Presidential Decree	

Relevant Legislation	Legislative Requirement
Presidential Decree Concerning Explosive Materials, No. 125/1999	To ensure that Companies have a legitimate purpose for the purchase and use of explosives and obtain a permit to do so. Ensure the safe utilization, handling, storage, transport and disposal of explosives so as not to endanger persons, property or the environment.
Presidential Decree Concerning Protected Areas Management, No. 32/1990	Management of protected areas includes the establishment of protected areas, the type of protected area, what can and cannot be done in protected areas.
Ministry Regulation	
Regulation of Minister Energy and Mineral Resources concerning Reclamation and Closure Mine for Mineral and Coal Mining Business No. 07/2014	Guidance for management of reclamation and mine closure for mineral and coal mining companies. The reclamation cost calculation and success criteria are also reviewed in this guidance.
Regulation of the Minister Forestry concerning Guidelines of Forest Borrow-to-Use, No. P.16Menhut-II/2014	Guidance for document of forest Pinjam Pakai (borrow-to-use) application, procedures and requirement of Pinjam Pakai document, criteria of offset program and tax.
Regulation of Minister Energy and Mineral Resources concerning Increasing The Value Added Minerals Activity Through Mineral Processing And Refinement, No. 11/2012	Changes of The Minister Of Energy And Resources Mineral Number 07 Of 2012 Concerning Increasing Value Added Activity Through Mineral Processing And Refinement. Each type of certain metal mineral mining, non-metal mineral and rock commodities shall be processed and / or purified according to the minimum limit of processing and / or purification as listed in Annex I, Annex II and Annex III respectively.
Regulation of the Minister for Environment concerning Public involvement in environmental permit, No. 17/2012	Guidelines for Community Involvement In The Process Of Environmental Impact Assessment (AMDAL) and Environmental Permit.
Regulation of the Minister for Environment concerning Guidelines for the compilation of Environmental Document, No. 16/2012	Guidance of environmental permit documents including AMDAL, UKL-UPL and SPPL. Prepare and submit an ANDAL to provide a reference framework for consideration and prepare the AMDAL in accordance with agreed reference framework. Include an analysis of the social aspects and involve the community in the process.
Regulation of Minister of Environment concerning Types of Business Plan and / or Activities Requiring Have AMDAL, No. 05 of 2012.	The reference in determination list of business and / or activities that are required to have an AMDAL document.
Regulation of the Minister for Forestry concerning Planting Guidelines for Holders of Forest Area Lease Utilization Permits for the purpose of Watershed Rehabilitation, No. 63/2011	Rehabilitation of the forest lease and utilization area shall be consistent with adjacent forest areas. Annual rehabilitation plans must be prepared, submitted for approval and implemented at the leaseholders cost.
Regulation of the Minister Forestry concerning Guidelines for Forest Reclamation, No. P.4/Menhut-II/2011	Guidance and reference for implementation of forest reclamation including 5-year reclamation planning. Forest reclamation program, includes preparation of forest area, setting landform / land arrangement, erosion control and sedimentation, management layer of top soil, revegetation and security.
Regulation of the Minister of Forestry, No. P.70/Menhut-II/2010	Changes in Regulation of the Minister of Forestry No. P.70 / Menhut-II / 2008 on Guidelines for Forest and Land Rehabilitation
Regulation of the Minister of Forestry concerning Changing Forest Function Status, No. P. 34/Menhut-II/2010	Guidance and procedure for changing function of forest status.
Regulation of the Minister Forestry concerning Guidelines for Forest Reclamation Success Assessment, No. P.60/Menhut-II/2009	Guidance for reclamation success criteria and that should be achieved at mine closure.
Regulation of the Minister for Environment concerning Terms and	Guidelines for licensing requirements and in waste management, especially in the case of discharges to the sea.

Relevant Legislation	Legislative Requirement
Procedures for Licensing of Waste Water Disposal into the Sea, No. 12 of 2006	
Decree of the Minister of Environment concerning Waste Water Quality Standards for Gold or Copper Mining Businesses and/or Activities, No. 202/2004	Ensure that water discharged in accordance with the issued permit complies with the discharge water quality standards.
Decree of the Minister of Energy and Mineral Resources concerning Guidelines for Feasibility Study Compilation, No. 1453.K/29/MEM/2000	Guidelines for compilation of Exploitation/Production Feasibility Study. The document covering technical and economic aspects including but limited to environmental setting, geology, mineralization, resources and reserve, geotechnical, hydrology and hydrogeological, mining plan, process plant, logistic and accessibility, HSE, organization, marketing and economic feasibility.
Decree of Minister of Environment concerning Vibration, No. 49/1996	Reference for vibration standard.
Decree of Minister of Environment concerning Noise, No. 48/1996	Reference for noise standard.
Regulation of the Minister of Manpower concerning Health and Safety Management System, No. PER.05/MEN/1996	Guidance for Occupational Health and Safety Management System, includes organizational structure, planning, responsibility, implementation, implementation, achievement, assessment and maintenance of safety and health policy work in order to control the risks associated with work activities in order to create workplaces that are safe, efficient and productive.
Decree of the Minister for Mining and Energy concerning Occupational Health and Safety in Mining, No. 555/1995	Mining companies must ensure the safety and health of all persons (employees, contractors or visitors) involved in the mining operations activities.
Decree of the Minister for Mining and Energy Concerning Prevention And Control Destruction And Pollution Environment In General Mining Activities, No. 1211.K/008/M.PE/1995	Prevention and control of destruction and environmental pollution in the implementation and management of environmental monitoring so as to achieve the goals, regulation, maintenance, monitoring, control, recovery and development of the environment in the mining business activities.
Decree of the Minster of Transmigration, No. 01/1980 Safety and Health in Construction	To ensure that construction equipment is fit for purpose and construction work is carried out so as to ensure the health and safety of employees.
Joint Decision – Others	
The Joint Decision of the Minister for Mining and Energy and Minister for Public Work concerning Water Utilization, No. 04/1991 and 76/1991	Regulation pertaining to water usage and or water source for Mining Operations.
The Joint Decision of the Minister for Mining and Energy and Minister for Forestry concerning Guidelines for Mining Activities and Energy in the Forest Area, No. 969/1989	Mining entities must obtain a license for mining in forest areas, pay market value stumpage compensation and rehabilitate the area to its original condition.
Decree of the Mining Director-General Concerning Reclamation Bond, No. Kep-336.K/271/DDJP/1996	Guidance and procedures for reclamation guarantee, cost, placement of guarantee and guarantee release.
Decree of the Mining Director-General Concerning Guidelines for Storing Explosive Materials in General Mining Activities, No 316/1990	Guidance regulation for storing and utilization of explosive material in general mining. This decree refer to Presidential Decree Concerning Explosive Materials, No. 125/1999
Regulation of the Head of National Police Concerning Monitoring, Control and Security of Commercial Explosives, No. 02/2008	Regulation related to monitoring, controlling and security of commercial explosive. This regulation refer to Presidential Decree Concerning Explosive Materials, No. 125/1999
Kep.Ka. BAPEDAL No. 056 of 1994 on Guidelines for the Determination of	The determination method used to assess whether a proposed business or activity can have a significant impact on the environment.

Relevant Legislation	Legislative Requirement
Significant Impacts.	
No. KEP-01 / BAPEDAL / 09/1995 on Procedures and Technical Requirements for Storage and Collection of Hazardous and Toxic wastes	Technical study regarding the procedures for the storage and collection of hazardous and toxic wastes.
No. KEP-02 / BAPEDAL / 09/1995 on Document Hazardous and Toxic	A reference for the management of hazardous and toxic wastes.
No. KEP-03 / BAPEDAL / 09/1995 on Technical Requirements of Hazardous and Toxic Wastes	Technical guidance regarding the treatment of hazardous and toxic wastes.
No. KEP-04 / BAPEDAL / 09/1995 on Procedures for Processing Requirements Stockpiling, Processing Terms Used Location, and Location Former Landfill of Hazardous and Toxic wastes	Technical guidance regarding the procedures / processing results to the terms landfilling of hazardous and toxic wastes.
No. KEP-05 / BAPEDAL / 09/1995 on Symbols and Labels of Hazardous and Toxic wastes.	Reference / ordinances concerning symbols and labelling of hazardous and toxic wastes.
Kep.Ka. BAPEDAL No. 124 of 1997 on Public Health Guide in the Environmental Impact Assessment.	Guide / assistance in conducting a study to assess the public health aspects in the preparation of an AMDAL.

20.6.2 Feasibility Study

The purpose of the Feasibility Study is to provide an assessment of the project's technical and economic feasibility. This document is to contain all reports or studies related to technical and economic matters, covering resources, reserves, geology, geotechnics, metallurgy, mine planning, processing, reclamation and mine closure, environment, social and economic feasibility.

The Feasibility Study report compilation is to refer to the Minister Energy and Mineral Resources Decree No. 1453K/29/MEM/2000, Annex XIIIb pertaining Guidelines for Preparation of Feasibility Study Report, Exploitation and Production. KSK holds a CoW and therefore the lead agency in charge of Feasibility Study approval will be the MEMR at the central Government level.

20.6.3 AMDAL

The regulation governing the obligations of the AMDAL or UKL-UPL is Minister of Environment No. 5 Year 2012 pertaining Type of Planned Business and/ or Activities Requiring Environmental Impact Assessment. Due to the nature and scale of the proposed activity the BKM Project will require an AMDAL. Key components of the AMDAL are:

- Project description;
- Environmental and social baseline description;
- Stakeholder consultation (local communities and government agencies);
- Environmental and Social Impact Assessment (ANDAL); and,

- Environmental and Social Management and Monitoring Plans (RKL/RPL).

In accordance with Regulation of the Minister of Environment Number 08 Year 2013 pertaining to Procedure Assessment and Inspection Document Environment Environmental Permit and Issuance of Environmental Permit in Annex III, the authority for assessment of BKM AMDAL is the AMDAL Appraisal Committee (KPA) is at the Central Kalimantan Province.

20.6.4 5-Year Reclamation and Mine Closure Plans

Submission of 5-Year Reclamation Plan (progressive reclamation) and Mine Closure Plan are mandatory in support of project permitting. The applicable regulations are:

- Government Regulation No. 78 Year 2010 pertaining to Reclamation and Mine Closure;
- Minister of Energy and Mineral Resources Regulation No. 7 Year 2014 pertaining to Implementation of Reclamation and Mine Closure on Mineral and Coal Mining Business Activities; and,
- Forestry Minister Regulation No. P.4 / Menhut-II / 2011 pertaining to Guidelines for Forest Reclamation.

The 5-Year Reclamation document will need to include:

- Description of area to be disturbed in the first 5 years of operations;
- Area to be reclaimed in the first 5 years of operations;
- Reclamation methods;
- Reclamation success criteria; and,
- Reclamation costs for the first 5 years of operations.

Once completed and approved, KSK will need to lodge a Guarantee/Bond to cover the costs for the 5-year reclamation program. The guarantee is reduced/reimbursed annually to the Company as it demonstrates completion of the reclamation works described in the 5-year Reclamation Plan. Upon completion of the first 5 years of operations, a second 5-year reclamation plan will need to be compiled and approved. This process is continued for the life of mine.

The Mine Closure Plan (covering closure works subsequent to cessation of mining/processing operations) requirements are:

- Description of environmental and social setting prior to mining;
- Description of mining and processing activities;
- Description of environmental and social setting at the end of mining operations;
- Stakeholder consultations;
- Mine closure plan;

- Mine closure management team and schedule;
- Closure success criteria;
- Post-closure environmental and social monitoring; and,
- Closure costs.

Once completed and approved, KSK will need to lodge a cash deposit as Guarantee to cover the costs for the mine closure program. The cash guarantee will be reimbursed to the Company as it demonstrates completion of the closure works and achievement of closure success criteria, described in the Mine Closure Plan.

KSK holds a CoW and therefore the lead agency in charge of 5-Year Reclamation and Mine Closure plans approval will be the MEMR at the central Government level.

20.6.5 Borrow-To-Use Forestry Permit (IPPKH)

The final major permit required in support of issuance of the mine construction permit is the Forestry Borrow-to-Use Permit. This permit is granted subsequent to the approval of the Feasibility Study and AMDAL and submission of the mandatory 5-Year Reclamation and Mine Closure plans. The application for, and issuance of, the Forestry Borrow-to-Use Permit is regulated by Minister Forestry Decree No. P.16/Menhut-II/2014 pertaining to Forestry Borrow-to-Use Guidelines. The objective of this decree is to regulate forest utilization for development purposes outside of forestry activities.

The issuance of the Forestry Borrow-to-Use Permit, which is an administrative rather than a technical undertaking, is a 2-stage, sequential process. The first stage is issuance of a Principle Permit (Ijin Prinsip) that covers the general total area of disturbance followed by the detailed Borrow-to-Use (IPPKH) Permit, which includes details regarding exact area to be cleared and the number of trees to be cut. The Company as an IPPKH Permit holder will have certain obligation including payment for trees to be cut, tax payments and requirements for reclamation and rehabilitation of mining area at closure.

Approval/issuance of the Forestry Borrow-to-Use Permit is the authority of the Ministry of Environment and Forestry at the central Government level.

20.7 MINE RECLAMATION AND CLOSURE

20.7.1 Introduction

Regulatory requirements related to mine closure and progressive reclamation were described in Section 20.6.4. Key objectives of progressive reclamation and mine closure plans are:

- Comply with applicable laws and regulations;
- Protect public and employee health, safety and welfare;

- Limit or mitigate adverse impact on the environment, biodiversity and community;
- Ensure that reclamation is conducted progressively;
- Achieve post-mine land use conditions in alignment with those promulgated in the applicable Spatial Plans of Gunung Mas Regency and Central Kalimantan Province;
- Ensure that stakeholders' needs, concerns and aspirations are taken into account when considering mine closure;
- Establish environmental and social closure success criteria;
- Implement a post-closure environmental and social monitoring program to demonstrate safe closure and achievement of closure success criteria; and,
- Provide a reasonable basis on which reclamation and closure costs can be estimated and managed.

Strategies for progressive reclamation, final mine closure and unplanned closure are described in sections 20.7.2, 20.7.3 and 20.7.4, respectively. Given the early stage of the project and the need for flexibility, the proposed approaches are general in nature and will need to be further defined through studies during the permitting as well as operational phase of the proposed mining activity.

20.7.2 Progressive Reclamation

Progressive (or concurrent) reclamation which is a regulatory requirement in Indonesia and GIIP, involves the gradual closure of project areas, as activity in that area is completed, throughout the mining operations. This will ensure physical and chemical stability of reclaimed areas and reestablishment of the terrestrial and aquatic habitats at the earliest opportunity.

Typically in Indonesia, the Forestry Borrow-to-Use Permit required mine sites to be reclaimed to the original forest function (Production Forest). In regards to site reclamation/revegetation, the following tasks will be conducted:

- Land contouring including grading and compaction to create a stable landforms that compatible with the surrounding environment
- Topsoil will be stripped and stockpiled during mining operations prior to waste rock excavation. It will be used later for rehabilitation where topsoil from the stockpiles will be applied to the reclaimed areas surface to a nominal depth of up to 30 cm, providing media for vegetation establishment.
- Revegetation with plant species aims to control erosion, provide vegetative diversity and contribute to a stable and compatible ecosystem. The revegetation program will involve:
    - Installation of brush fencing or straw-roll energy-breaks at regular intervals. The straw rolls will be placed in shallow, lateral trenches with half of the roll below the surface level. The rolls are then anchored with stakes of tree cuttings.

- Check revegetated slopes for erosion immediately after heavy rainfall. Repair gullies as soon as possible by filling with soil, applying seed and jute netting anchored with bamboo stakes.
- Plant trees as deemed appropriate. Species selection for revegetation serves both long- and short-term purposes. The initial mandate is simply to physically stabilize the new landform with regard to erosion. The long-term objective relates more to ultimate land-use. A number of grass species will be planted as first colonizers along with understory and native trees species local to the area. Suitable vegetation for slope stabilization includes cover crops (grasses and legumes) which can be hand- or hydro-seeded and fast-growing fuel or forage trees.

- Reinstatement of natural drainage patterns where they have been altered or impaired and construction of new drainage channels around the reclaimed areas to limit surface runoff and erosion.

In addition to site reclamation/revegetation, equipment and infrastructure will be demobilized and decommissioned progressively. Specifically:

- Demobilization of Equipment - any mobile or stationary equipment that is no longer necessary will be cleaned and disassembled and/or removed from site.
- Closure of Access and roads - Access roads to parts of the project no longer in operation and not needed as access for future monitoring programs will be closed permanently and reclaimed.
- Demolition and disposal of Facilities – buildings and associated infrastructure that are no longer in use for the project will be decommissioned. Depending on the facility, it will be dismantled and the materials will then be reused, donated, sold, or disposed of in an environmentally-safe manner.

Monitoring of the physical and chemical stability of progressively reclaimed areas will occur through the life of the project. Components to be monitored include soils, surface water quality, flora, fauna, aquatic ecology and geotechnical characteristics.

20.7.3 Mine Closure

The final mine closure plan will be developed based on following considerations:

- Community and stakeholder participation – the local community and related government stakeholders to be consulted in developing this post-mine program. Community and stakeholders' interests were identified during the plan development process, and duly considered in the post-mine plan.
- Post-closure land use – disruption of local community access to their traditional livelihoods to be minimized.
- Stable landforms – all landforms to be returned to a safe, stable condition, and be compatible with the surrounding landscape.

- Stable ecosystems – appropriate ecosystems to be re-established and the capacity of the site to support the pre-development level of biodiversity to be restored.
- Environmental quality – air, land, surface water, groundwater, and vegetation to be restored and comply with standards stipulated by the applicable regulations and closure success criteria, with no residual long-term liability.

In the following sections conceptual approaches to final site reclamation/vegetation, closure of mining facilities (open pit and waste dump), processing facilities (heap leach, process ponds and process plant) and supporting infrastructure (raw water and environmental control dams, roads, buildings, explosive magazine, emulsion process plant, agglomeration plant, ROM pad, etc.) are described. In addition, contaminated site assessment and clean up (as necessary), social post-closure programs and environmental and social monitoring programs are outlined.

20.7.3.a Site Reclamation/Revegetation

The general method for site reclamation/revegetation, which includes: application and spreading of topsoil, energy breaker placement, hand or hydro seeding of appropriate grass and legume and finally transplanting of local trees and wild seedlings from adjacent forest areas, is the same as for progressive reclamation which is described in Section 20.7.2. Targeting the revegetation program towards the establishment of a secondary forest (original pre-mining state) affords a robust and stable biological framework very similar to the natural pre-mining conditions without precluding the opportunity for other use should land-use requirements change.

All land areas described in the following sections will be revegetated using these methods subsequent to physical stabilization, decommissioning and clean up.

20.7.3.b Closure of Mining Facilities

Open pit

At the end of mine operation, the open pit will form a pit lake. The final closure of the pit will include the slope stabilization, establishment of safety berms, fences and warning signage installation, rehabilitation of haul roads accesses, spillway construction and channel armoring.

Waste Dump

Reclamation of the waste dumps will be ongoing throughout the mine life. The final closure and reclamation process may involve two stages. The first stage is a recontouring of the dump surface (as necessary) to achieve both geotechnical stability of the dump under static and seismic conditions in addition to visually "blending" the terrain into the existing topography. Once reshaping of an area is complete, the external surfaces of the waste dump will be covered with a low permeability layer of compacted mine waste to minimize water infiltration into the underlying waste rock. On the top surface of the low permeability layer will be placed 0.3 m of top soil to provide a growing medium for the establishment of a sustainable vegetative cover.

20.7.3.c Closure of Processing Facilities

Heap Leach

Objectives of the heap leach closure are as follows:

- Stabilize the landform and slopes for geotechnical stability;
- Minimize infiltration into the heap through an engineered cover system;
- Stabilize slopes and soils to erosion;
- Establish self-sustaining growth of natural vegetation; and,
- Manage post-closure seepage to ensure compliance with Indonesian discharge standards.

An engineered low permeability cover will be constructed at the top and surface of the heap leach to reduce the amount of infiltration into the surface of the pad, and consequently reduce the long-term post-closure seepage volume that has to be managed. The surface of heap leach will be sloped (~3%) to promote rapid drainage off the facility (through the drain layer) and to prevent surface ponding of water and the development of a head over the barrier layer. Excessive runoff water will be directed down to the base of the pad via a rip rap drain to minimize erosion. Active and/or passive (wetlands) treatment of residual seepage from the heap leach will be implemented, as necessary, until the seepage quality meets Indonesian water quality discharge standards.

Process and Stormwater Ponds

Process and stormwater ponds may be maintained for a time subsequent to closure for the management and treatment of the heap leach draindown solution. Ultimately the ponds may be converted to wetlands to passively treat residual seepage from the heap leach or decommissioned, filled and revegetated.

Process Plant

Processing plant and associated facilities will be dismantled and/or destroyed at mine closure. All main processing/refining facilities and all equipment used will be decontaminated, dismantled and removed from site. Concrete foundations will be broken up in-situ and buried.

20.7.3.d Closure of Supporting Facilities

At the end of mine operations, all of infrastructure and equipment will be dismantled and removed from site. The main objective for dismantling and removing infrastructure is to stabilize soil surface and to undertake reclamation in disturbed areas consistent with future land use (return to forest function).

The supporting facilities will be dismantled and removed as part of the post-mine program. Before removal, all unused chemicals and explosives will be removed and transported outside the mine area for reuse or resale. All hazardous material will be handled according to applicable regulations.

Every demolished structure that may be recycled or material of a surplus value will be removed and included into a recycle program. Non-hazardous material unsuitable for recycling or have no surplus value will be cut up, demolished and if necessary buried in the waste dump or landfill. Concrete foundations will be broken up in-situ and buried.

All roads in the Forestry Borrow-to-Use area will be removed and reclaimed, unless otherwise stipulated by government authorities and/or based on agreement with stakeholders, including local administration agencies.

20.7.3.e Contaminated Site Assessment and Clean Up

A site assessment will be undertaken, involving review of operational documents and visual inspections related to the areas where equipment and machinery were stored and serviced, the areas around chemical storage tanks, fuel tanks, heap leach, process ponds, water treatment plant and any other maintenance areas where soil may have been in contact with hydrocarbons and/or chemical compounds associated with mine operations.

Following the site assessment described above, a detailed site investigation will be undertaken to sample and analyze soil in any location where any contamination is identified. The soil analytical results will determine whether clean up actions, including on-site bioremediation, would be required to ensure protection of human health and biota post-closure.

20.7.3.f Post-Closure Social Programs

The post-mine social programs will be the continuation of the community development programs instituted during the operation. The implementation of these programs will be in cooperation with the Regional Government agencies. In addition, cooperation and coordination will also be implemented with local corporate and/or NGO partners, in order to develop and enhance the potential of surrounding communities.

The post-mine community development programs will be designed to be self-sustaining (i.e., from, by and for the community) as they would be based on aspirations of local residents who will be key stakeholders in the process and will participate in the programs.

The details of the post-closure community development programs will be determined through discussions involving the Company as well as local government and community stakeholders prior to, and during, mine operations. These programs will be assessed on a regular basis during the course of their implementation to ensure benefits for the stakeholders mentioned earlier. The following is a summary of potential post-mine social programs:

- Local Economic Development
- Training and Education
- Community Health
- Culture and Youth

20.7.3.g Post-Closure Environmental and Social Monitoring

A post-closure environmental and social monitoring program will be implemented to assess closure performance and compliance with previously establish closure success criteria. Environmental monitoring components will be surface and groundwater quality, air quality and noise, soils, flora and fauna, aquatic ecology and geotechnical stability. Social monitoring will focus on the implementation and success of post-closure community development programs.

20.7.4 Unplanned/Temporary Closure

Temporary closure due to economic, political or any other reason is discussed in this section. The main objectives of temporary closure are similar to those for final closure, which are to protect public health and safety, and to ensure both physical and chemical stability of the site. Key elements for temporary closure planning are:

- Heap leach seepage management – The heap leach will be covered (jackets) to minimize infiltration and seepage and key water management structures, pumps and the treatment plant will remain in operation in order to ensure all discharges from the site during temporary closure meet the Indonesian discharge standard.
- Progressive reclamation - Restoration, revegetation, and rehabilitation of the land will occur as a continuous process throughout mining operations. In the event of temporary closure, progressive reclamation already underway will be completed.
- Demobilization of Equipment - Both mobile and stationary equipment will be cleaned and purged of all chemicals and fuels, with all containers being properly labelled. All equipment that can be stored will remain in storage until the resumption of mining activities.
- Closure of Access and Pathways - All access roads to the project site and selected haul roads (access to open pit) will be temporally closed to prevent any foreign vehicles from entering the site and approaching hazard areas such as the open pit, heap leach and waste dumps.
- Monitoring – Environmental monitoring will be performed during times of temporary closure to ensure compliance with applicable regulations and to determine corrective actions plans, if necessary.

21 CAPITAL AND OPERATING COSTS

21.1 CAPITAL COST ESTIMATE

The total initial and sustaining capital costs developed for the BKM Project are summarized in Table 21-1 below.

Table 21-1 Summary of Capital Cost Estimate

Item		USD M
Initial	Mining	$1.7
	Primary Crusher + Agglomerator	$24.6
	Leach Pads	$31.3
	SX/EW (Incl. Neutralization)	$82.7
	Infrastructure	$2.1
	Subtotal	$142.4
	Contingency @ 15%	$21.4
	Total	**$163.8**
Sustaining and Closure	Mining	$5.0
	Leach Pads	$1.0
	SX/EW	$1.6
	Subtotal	$7.6
	Contingency @ 15%	$1.1
	Total	**$8.7**
Grand Total		**$172.5**

21.1.1 Project Capital Cost Assumptions

The capital cost estimate covers all major capital cost items required to commence and maintain operations at BKM.

All costs are provided in US dollars (USD) and an exchange rate for Indonesian Rupiah of 13,400 IDR per 1 USD.

The accuracy range for the project estimate is ±35%. The estimate makes no provision for future escalation and currency exchange rate fluctuations beyond the basis date. Any other estimate qualifications, assumptions and exclusions are summarized in the relevant sections below.

All mining related costs are based on the preferred submission to a Request for Quotation for mining contract services (refer to Section 16.2).

Heap leach pad earthworks and construction is based on a similar request for budget quotation pricing.

The crushing / agglomeration / stacking and SX/EW plant capital cost estimate have been developed by MillerMet and are based on their database of costs and previous experience with similar plants and projects.

Project infrastructure capital is a combination of budget quotation pricing sought from suitable local construction / engineering groups or first principal estimates developed by DRA, the study infrastructure engineering consultant.

No allowance has been made for import duties, taxes, insurances etc. However a global contingency of 15% has been allowed on all capital items.

21.1.2 Mine Capital Cost Estimate

21.1.2.a Mine Infrastructure and Establishment

Mine infrastructure costs were provided as part of the mining contractor submissions and are detailed in Table 21-2.

Table 21-2 Mine Infrastructure and Establishment Capital Cost Estimate

Item		Cost (USD '000)
Heavy Mining Equipment Mobilization		$63.5
Ancillary Equipment Mobilization		$22.6
Infrastructure	Workshop & Washbay	$80.1
	Fuel & Oil Storage Facility	$19.6
	Store	$14.4
	Office	$96.1
	Lunchroom	$7.2
	Ablutions	$6.4
	AN Storage Facility	$29.3
	Magazines	$35.2
Subtotal		**$374.3**
Contingency @ 15%		**$56.1**
Total		**$480.5**

21.1.2.b Mine Pre-Production Costs

Due to the fact that ore presents effectively from surface there is no pre-production mining per-se. The pre-production costs accounted for consist of stripping / clearing of the pits and waste dumps, and building of ex-pit roads. Again these costs were provided as part of the mining contractor submissions, with the exception on the estimate for initial drainage works which is a provisional sum estimated by Orelogy. The costs are detailed in Table 21-3.

Table 21-3 Mine Pre-production Capital Cost Estimate

Item	Cost (USD '000)
Stripping / Clearing	$603.1
Roadbuilding	$664.4
Perimeter Drainage	$50.0
Subtotal	**$1,317.5**
Contingency @ 15%	$197.6
Total	**$1,515.2**

21.1.3 Crushing / Agglomerating / Stacking Capital Cost Estimate

The capital cost for this component was provided by MillerMet on the basis of a recently sourced cost for a comparable project. The original cost was based on a 5 Mtpa throughput arrangement, and this cost was escalated to an equivalent 7.5Mtpa set-up in line with the peak schedule LFI throughput. The cost calculated was $24.6M. The 15% contingency adds a further $3.7M for a total capital cost of $28.3M.

21.1.4 Heap Leach Capital Cost Estimate

A detailed material take-off (MTO) for the leach pad civil construction was developed by MillerMet. This breakdown was provided to two suitably qualified Indonesian groups for the purposes of obtaining a budget price for this work. The capital estimate based on the preferred submission is detailed in Table 21-4.

Table 21-4 Heap Leach Pad Construction Capital Cost Estimate

Item	Cost (USD '000)
Preliminaries and General	$977.3
Bulk Earthworks	$14,590.6
Roadworks	$970.2
Drainage	$8,800.7
Liner Installation / Retaining Walls	$5,930.8
Subtotal	**$31,269.7**
Contingency @ 15%	$4,690.4
Total	**$35,960.1**

21.1.5 SX/EW Plant Capital Cost Estimate

The capital cost estimate for the SX/EW plant is based on the experience of MillerMet personnel and has been estimated at $82.7M, inclusive of $7.7M for a limestone mill and neutralization plant. The 15% contingency adds a further $12.4M for a total capital cost of $95.1M.

21.1.6 Infrastructure Capital Cost Estimate

Table 21-5 details the capital cost estimate for project infrastructure and the source of those costs.

Table 21-5 Infrastructure Capital Cost Estimate

	Item	Source	Cost (USD '000)
Camp Facilities	Office, Accommodation, Canteen & Security Post	Budget quote	$214.0
	Camp Clearing and Earthworks	DRA	$300.0
	Warehouse	Budget quote	$58.2
	Workshop		$275.1
	Facility Access Road (with base course)		$279.9
	Security Fences (Plant)		$20.6
	Security Fences (Leach Pad & Dam)		$67.2
	Waste Management		$104.5
		Subtotal	**$1,319.4**
Power Supply		Based on rental	**$0.0**
Site Office and Facilities	Main Office / Locker Room	Budget quote	$479.2
	Cranage	Budget quote	$130.6
	Water Treatment Plant	DRA	$60.0
	Sewage Treatment	DRA	$15.0
	Communication System	DRA	$20.0
	Fencing	DRA	$35.0
	Weighbridge	DRA	$90.0
		Subtotal	**$829.8**
Subtotal			**$2,149.2**
Contingency @ 15%			$322.4
Total			**$2,471.6**

21.1.7 Power Supply Capital Cost Estimate

Power supply is assumed to be provided under a facility rental and supply arrangement, as provided to KSK by PT. Prastiwahyu Trimitra Engineering (PTE). PTE have a proven track record in the provision of electrical power, supplying upwards of 260MW across 17 sites and 7 islands in Indonesia. The costs are detailed in the Operating Costs Section 21.2.4.

21.1.8 Mine Development, Rehabilitation and Closure Capital Cost Estimate

The following costs are applied as capital items to generate the project cash flow detailed in the project economic analysis (Section 22). The costs are applied over time. However the total amounts are summarized in Table 21-6.

Table 21-6 Mine Development, Rehabilitation and Closure Capital Cost Estimate

Item			Cost (USD '000)
Sustaining	**Mining**	Stripping / Clearing	$671.1
		Roadbuilding	$118.7
		Subtotal	**$789.8**
Closure	**Mining**	De-Mob/Dis-establishment	$230.2
		Rehabilitation	$2,040.9
		Waste Dump Capping	$1,900.0
	Leach Pads	Capping and Remediation	$1,010.0
	SX/EW	Dis-establishment	$1,590.0
		Subtotal	**$6,771.1**
Subtotal			**$7,560.9**
Contingency @ 15%			$1,134.1
Total			**$8,695.0**

21.1.9 Capital Cost Estimate Contingency

A contingency of 15% has been applied across all capital components.

21.1.10 Capital Cost Estimate Exclusions and Qualifications

The following is a summary of exclusions and qualifications from the capital cost estimate that may or may not have been noted previously:

- Scope or schedule changes
- Force majeure occurrences or cost exclusions
- Hazardous waste remediation
- Financing cost (interest during construction)
- Indirect allowances for sustaining capital estimate, excluding closure costs
- Future currency exchange rates beyond first quarter of 2016
- Any costs related to lost production
- Sunk costs such as previous studies and previously constructed facilities
- Future escalation beyond first quarter of 2016
- Value added or sales taxes
- Surplus material or equipment salvage value credits

21.2 Operating Cost Estimate

21.2.1 Project Operating Cost Assumptions

The operating cost estimate covers all operating cost centers required to commence and maintain operations at BKM.

The exchange rate for Indonesian Rupiah is 13,400 IDR per $1 US.

The accuracy range for the project estimate is ±35%. The estimate makes no provision for future escalation and currency exchange rate fluctuations beyond the basis date. Any other estimate qualifications, assumptions and exclusions are summarized in the relevant sections below.

All mining related costs are based on the preferred submission to a Request for Quotation for mining contract services (refer to Section 16.2).

The crushing / agglomeration / stacking and SX/EW plant capital cost estimate have been developed by MillerMet and id based on a first principal cost model.

Project infrastructure capital is a combination of budget quotation pricing sought from suitable local construction / engineering groups or first principal estimates developed by DRA, the study infrastructure engineering consultant.

No allowance has been made for import duties, taxes, insurances etc. for consumables or parts.

21.2.2 Mining Operating Cost Estimate

The overall mine operating costs, as summarized in Table 21-7 below, include the following cost components:

- Contract mining variable costs inclusive of:
  - Load and Haul
  - Drill and Blast
  - Support and auxiliary Fleet
  - Management and supervision
- Contract ore rehandle cost from agglomerator stockpile to stacker feed bin
- Grade control based on advance R/C drilling program
- Owner costs related to management and technical support personnel

Table 21-7 Mine Operating Estimate

Item	$M	$ / tonne total	$ / tonne ore	$/ rec. lb.
Contract Mining Cost	$200.1	$1.84	$4.11	$0.51
Rehandle	$22.7	$0.21	$0.47	$0.06
Grade Control	$6.7	$0.06	$0.14	$0.02
Owners Mining Cost	$3.5	$0.03	$0.07	$0.01
TOTAL	$233.0	$2.15	$4.78	$0.60

21.2.3 Process Operating Cost Estimate

21.2.3.a Heap Leach Operating Estimate

The operating costs for the heap leach are a combination of the following cost components:

- Reagents
  - Extractant
  - Diluent
  - Clay
- Labor (refer to Table 21-11)
- Power as a variable component, based on supplied cost (refer to Section 21.2.4) and consumption rates for:
  - Crushing, Agglomeration & Stacking
  - Heap Leach
- Consumable parts for:
  - Crushing, Agglomeration & Stacking
  - Heap Leaching

The overall costs are summarized in Table 21-8 below

Table 21-8 Heap Leach Operating Estimate

Item	$M	$ / tonne total	$ / tonne ore	$/ rec. lb.
Reagents	$6.7	$0.06	$0.14	$0.02
Labor	$3.6	$0.03	$0.07	$0.01
Power	$38.6	$0.36	$0.79	$0.10
Consumables	$34.8	$0.32	$0.71	$0.09
TOTAL	$83.7	$0.77	$1.72	$0.21

21.2.3.b SX/EW Plant Operating Estimate

The mine operating cost is a combination of the following cost components:

- Reagents
  - Limestone ($CaCO_3$)
  - Cobalt Sulphate
  - Guar
  - Acid
- Labor (refer to Table 21-11)
- Power as fixed and variable components, based on supplied cost (refer to Section 21.2.4) and consumption rates for:
  - Solvent Extraction
  - Electrowinning
  - Services incl. Water Supply
  - $CaCo_3$ mill
  - General light & power including camp
- Contract laboratory

- Consumable parts for:
  - Solvent Extraction
  - Electrowinning
  - Services
  - Neutralization
- Miscellaneous costs
  - Off-site metallurgical testwork
  - Consultants

The overall costs are summarized in Table 21-9 below

Table 21-9 SX/EW Operating Estimate

Item	$M	$ / tonne total	$ / tonne ore	$/ rec. lb.
Reagents	$38.8	$0.36	$0.80	$0.10
Labor	$8.8	$0.08	$0.18	$0.02
Power	$93.0	$0.86	$1.91	$0.24
Contractors	$1.2	$0.01	$0.02	$0.00
Consumables	$17.2	$0.16	$0.35	$0.04
Miscellaneous	$1.6	$0.01	$0.03	$0.00
TOTAL	$160.7	$1.48	$3.30	$0.41

21.2.3.c Total Process Operating Cost

Table 21-10 below summarizes the total processing cost of heap leach and SX/EW combined.

Table 21-10 Total Process Operating Estimate

Item	$M	$ / tonne total	$ / tonne ore	$/ rec. lb.
Reagents	$45.5	$0.42	$0.93	$0.12
Labor	$12.4	$0.11	$0.25	$0.03
Power	$131.6	$1.21	$2.70	$0.34
Contractors	$1.2	$0.01	$0.02	$0.00
Consumables	$52.0	$0.48	$1.07	$0.13
Miscellaneous	$1.6	$0.01	$0.03	$0.00
TOTAL	$244.3	$2.25	$5.02	$0.62

21.2.4 Power Supply Operating Cost Estimate

The budgetary electrical power supply cost provided by PT. Prastiwahyu Trimitra Engineering (PTE) is on the basis of a facility rental and additional cost of fuel. It assumes the BKM Project will utilize 20MW of power over a six year rental term.

The fuel usage cost is based on a specific gas consumption of 9600 British Thermal Units (BTU)/ kWh and a gas price of $11 US/MBTU. This equates to a fuel cost of $0.1056 US/ kWh.

The quoted rental cost is $0.045 US/ kWh.

Therefore total power cost to the project is $0.1507 US/ kWh. The total LoM power cost of $131.6 M US equates to 27% of total direct operating costs.

21.2.5 General and Administrative Operating Cost Estimate

An organization chart for the owner's team was developed and associated costs generated as shown in Table 21-11. In addition allowances were made for:

- Development and geotechnical drilling
- Office equipment and consumables
- Consulting
- Training and travel
- Light vehicle costs

These costs were then distributed to the relevant cost center as shown in Table 21-12. An adjusted number was also calculated for the post-mining period when only the heap leach and SX/EW are operating based on reduced personnel numbers.

Table 21-11 Proposed KSK Workforce

Personnel					Base Annual Salary	# per Crew	# on Site	Total #	Total Cost per Year
1	General and Administration	1.1	Management	General Manager	$260,000	1	1	1	$275,036
				GM Personal Assistant	$20,000	1	1	1	$22,024
				HRD Manager	$20,000	1	1	1	$22,024
				HRD Assistant	$12,000	2	2	2	$28,048
				Government Relations	$10,000	1	1	1	$12,024
				Community Relations	$10,000	2	2	2	$24,048
				Admin / Secretary	$7,500	2	2	4	$38,096
		1.2	Financial / Purchasing	Financial Manager	$30,000	1	1	1	$32,024
				Senior Accounts Controller	$20,000	1	1	1	$22,024
				Accounts Controller	$10,000	1	1	2	$24,048
				Logistic Officer	$10,000	2	2	4	$48,096
				Purchasing Officer	$10,000	1	1	2	$24,048
		1.3	Safety / Security / Medical	SSM Supervisor	$12,000	1	1	1	$14,024
				Security Officer	$4,000	3	9	12	$66,216
				Medical Officer	$20,000	1	1	2	$44,048
		1.4	Office Services	Services Supervisor	$12,000	1	1	1	$14,024
				Office Care	$3,600	2	2	2	$11,248
				IT	$12,000	3	3	3	$42,072
		1.4	Camp Services	Camp Coordinator	$12,000	1	1	1	$14,024
				Cleaner	$3,600	8	8	8	$44,992
				Cooks	$3,800	8	8	8	$46,592
2	Process and Infrastructure	2.1	Management and Supervision	Process Manager	$200,000	1	1	1	$215,036
				Senior Process Engineer	$21,000	1	1	1	$23,024
				Process Engineer	$20,000	2	2	2	$44,048
				Graduate Process Engineer	$12,000	1	1	1	$14,024
				Process Superintendent	$30,000	1	1	1	$32,024
				Process Supervisor (Shift)	$24,000	1	3	4	$102,072
				Laboratory Technician (Shift)	$21,000	3	9	12	$270,216
				Clerk	$7,500	1	1	2	$19,048
		2.2	Crushing / Agglom. / Stacking	Senior Operator (Shift)	$8,100	1	3	4	$38,472
				Process Operator (Shift)	$7,800	3	9	12	$111,816
				Samplers	$7,800	1	3	4	$37,272
				Clean-up Crew	$7,500	2	6	8	$72,144
		2.3	Leach	Senior Operator (LH)	$8,100	1	3	4	$38,472
				Process Operator	$7,800	3	9	12	$111,816
				Laborers	$4,000	2	6	8	$44,144
		2.4	SX-EW	Senior Operator (LH)	$8,100	2	6	8	$76,944
				Process Operator (Shift)	$7,800	2	6	8	$74,544
				Stripping crew	$3,800	3	9	12	$63,816
		2.5	Services & Neutraltn.	Senior Operator (LH)	$8,100	1	3	4	$38,472
				Process Operator	$7,800	3	9	12	$111,816
				Laborers	$4,000	3	9	12	$66,216
		2.6	Maintenance	Maintenance Supervisor	$21,000	1	1	1	$23,024
				Electrician (Shift)	$21,000	1	3	4	$90,072
				Fitter (Shift and Workshop)	$8,100	3	9	12	$115,416
				Boilermaker (Shift and Workshop)	$8,100	3	9	12	$115,416
				Maintenance Assistant	$8,100	3	9	12	$115,416
				Warehouse Supervisor	$20,000	1	1	1	$22,024
				Warehouse Assistant	$24,000	2	2	4	$104,096
3	Mining	3.1	Mgmt. and Supervision	Mine Manager (Registered)	$50,000	1	1	1	$52,024
				Mine Clerk	$7,500	1	1	1	$9,524
				Contract Controller	$12,000	1	1	1	$14,024
				Contract Supervisors	$24,000	1	3	4	$102,072
		3.2	Technical Services	Senior Engineer	$24,000	1	1	1	$26,024
				Planning Engineer	$20,000	2	2	2	$44,048
				Graduate Engineer	$12,000	1	1	1	$14,024
				Senior Geologist	$24,000	1	1	1	$26,024
				Shift Geologist	$12,000	1	3	4	$54,072
				Graduate Geologist	$10,000	1	1	1	$12,024
				Pit Technician	$4,000	2	2	4	$24,096
				Senior Surveyor	$12,000	1	1	1	$14,024
				Surveyor	$10,000	1	1	1	$12,024
				Surveyor Assistant	$4,000	1	1	1	$6,024
TOTAL							**204**	**262**	**$3,469,708**

Table 21-12 General and Administrative Operating Estimate (USD '000 per Year)

Cost Centre	Mining			Post Mining		
	Light Vehicles	Personnel	TOTAL	Light Vehicles	Personnel	TOTAL
Site G&A	$27.0	$1,667.7	$1,694.7	$24.0	$1,368.0	$1,392.0
Crushing / Agglomeration. / Stacking	$18.0	$476.8	$494.8	$0.0	$0.0	$0.0
SX/EW	$9.0	$1,068.1	$1,077.1	$9.0	$958.8	$967.8
Mining	$57.0	$820.5	$877.5	$0.0	$0.0	$0.0
TOTAL	$111.0	$4,033.2	$4,144.2	$33.0	$2,326.8	$2,359.8

21.2.6 C1 Cash Operating Cost

For this study it is assumed the C1 cost includes sustaining costs (i.e. Mine Development, Rehabilitation and Closure , Section 21.1.8) but excludes off–site transport (i.e. transport of cathode, refer Section 21.2.7 below). Details of the project C1 Cash Operating Cost are provided in Table 21-13.

Table 21-13 C1 Cash Operating Cost

Item	$M	$ / tonne total	$ / tonne ore	$/ rec. lb.
Mining	$233.0	$2.15	$4.78	$0.60
Crushing / Stacking	$66.0	$0.61	$1.35	$0.17
SX/EW Processing	$47.2	$0.44	$0.97	$0.12
Power	$131.2	$1.21	$2.69	$0.34
G&A and Support	$13.5	$0.12	$0.28	$0.03
Sustaining	$8.7	$0.08	$0.18	$0.02
C1 Cash Cost	$499.5	$4.60	$10.25	$1.28

21.2.7 Transport and Logistics Operating Cost Estimate

Bulk materials movement is proposed to be via a combination of trucking and low-cost barge transportation. Trucking will utilize the current all-weather unsealed road route extending approximately 120 km north from the Kasongan River Port to the BKM site. The working assumption in this PEA study is for KSK to negotiate an ongoing access and maintenance agreement with the company that constructed and maintains this all-weather logging road (KSK currently has an access agreement in place for the use of this road route).

The proposed barging route is from the Kasongan port to the Java Sea and onto Banjarmasin Sea Port, a distance of approximately 510km. The Banjarmasin Sea Port is a deep water port located 25km from the Java Sea and caters to international vessels. It has a large container handling and storage service together with a customs clearance/bonded area for international freight. Marine barging would utilize self-propelled Landing Craft Tanks (LCTs) that are highly suited for the upper reaches of the Katingan river system and commonly used to transport bulk materials, predominately timber, equipment and fuel, to and from Kasongan. Copper cathode will be packed in 20 tonne containers at the process plant, locked and tagged, then transported via road and LCT barge to Banjarmasin Port where the copper cathode will be trans-shipped in line with sales contracts. All transport, logistics and product security will be outsourced to specialist contractors.

Assessment of bulk materials transport and logistics for the project, and the associated capital and operating costs, has been completed by PT. Resindo Resources & Energy (RRE), a highly experienced provider of similar infrastructure solutions throughout Indonesia.

The cost calculated by REE for the transport approach detailed above totaled $111.680 / tonne. This equates to $19.8M US of cathode transport costs over LoM.

21.2.8 Operating Cost Estimate Exclusions

The following are items are excluded from the cost estimate:

- Operating and sustaining costs associated with maintenance of off-site infrastructure.
- Transportation costs beyond Banjarmasin Port.
- All Owner's costs, including but not limited to:
    - GST, import duties and/or any other statutory taxation, levies and/or national and local institutions.
    - Contributions to health and social welfare programs.
    - Head office costs, administration charges, insurance premiums, interest payments, payroll, etc.
    - EIA, EMPR, contributions to rehabilitation funds, environmental monitoring and conformance to environmental requirements.
    - Working capital, bridging finance and costs associated with finance raisings.

22 ECONOMIC ANALYSIS

22.1 INTRODUCTION

This Technical Report, and this section in particular, contains information and statements that are "forward-looking" in nature. Any forward-looking statements in this Report are necessarily based on opinions and estimates made by the relevant Qualified Persons responsible for the statement or any associated Independent Expert, at the date the statements were made. Forward-looking statements include, but are not limited to, statements with respect to:

- the Mineral Resource estimates
- the cost and timing of execution of the BKM Project
- the proposed mine plan
- the proposed processing methodology
- associated production and throughput rates

- metallurgical recovery and overall processing performance
- infrastructure requirements
- cost estimates (capital, operating and sustaining)
- commodity price projections
- the Net Present Value (NPV) and Internal Rate of Return (IRR) and payback period of capital

No allowance or assessment has been taken of:

- the timing of the environmental assessment process, or any results of this process that may result in changes to the proposed Project
- government regulations and permitting timelines
- estimates of reclamation obligations
- requirements for additional capital
- environmental risks
- general business and economic conditions.

Although Orelogy, and other Qualified Persons, believe the expectations reflected in the forward-looking statements to be reasonable, they collectively do not guarantee future results, levels of activity, performance or achievements.

The mine plan and associated economic analysis presented in the following sections are presented with conceptual years shown. Orelogy consider the timing shown to be an “earliest start date”. The sensitivity analysis (Section 22.5) provides some indication of the effect of a delay in project execution.

Additional mining, technical, and engineering studies will be undertaken which may alter the Project assumptions as discussed in this Report, and may result in changes to the calendar timelines presented.

22.2 METHODOLOGY

The BKM Project has been valued using a discounted cash flow (DCF) approach. Estimates have been prepared for all the various components of cash revenue and expenditures for ongoing operations.

Capital cost estimates have been prepared for initial development and construction of the BKM Project, starting in Year -1. In addition to the initial capital cost, a sustaining capital component was included periodically from Year 1.

The resulting net annual cash flows are discounted back to a nominal Year -1 basis, on the assumption the first year of production (Year 1) is 2019. The base currency used throughout this report is USD based at Q1 2016, considering that the estimation was developed during the 1st Quarter of 2016 and the copper prices provided by Wood Mackenzie are on 2016 dollar basis. The IRR is defined as the discount rate that yields a zero NPV. The payback period is calculated as the time needed to recover the initial capital costs, in terms of undiscounted cash flow.

22.3 FINANCIAL MODEL INPUTS

22.3.1 Mine Plan

A mine production schedule was developed for the project as detailed in Section 16.5.4. The inventory used for generating this schedule is calculated within an operable pit design and is a sub-set of the overall mineral resource.

Economic leach feed material is present from surface and therefore there is effectively no ramp-up or pre-strip period associated with the mine plan. Consequently there is no requirement for “capitalizing” any initial mine operating cost component.

The mine life of the proposed mine plan is approximately eight (8) years.

22.3.2 Processing Performance

The basis of the metallurgical recovery is detailed in Section 0.

It includes:

- modelling of the soluble or “leachable’ copper content of the material
- an 85% recovery of the leachable component
- a leach cycle over time to attain the final 85% recovery

22.3.3 Metal Price and Revenue

The metal price for copper cathode is based on a price forecast sourced from Wood Mackenzie, as detailed in Section 19.2. Although the pricing provided by Wood Mackenzie varied over time, for the purposes of this study an average price was applied as a constraint over the duration of the mine life.

Annual revenue is calculated by applying estimated metal prices to the annual cathode production estimated for each operating year. The price has been applied to all life of mine production without escalation or hedging.

22.3.4 Capital Costs

Capital costs are discussed in Section 21.1.

For the purposes of this analysis any expenditure prior to start of the full project period has been treated as “sunk” cost. This would include such items as:

- Further resource confirmation or extension drilling
- Subsequent pre-feasibility and /or feasibility studies
- Front End Engineering and Design (FEED)
- KSK corporate, administrative and personnel costs

22.3.5 Operating Costs

Operating costs are discussed in Section 21.2.

22.3.6 Transport Costs

Transport costs are discussed in Section 21.2.7.

22.3.7 Royalties

A government royalty of 4% on the copper price has been applied.

In addition Freeport-McMoRan Exploration Corporation holds a 1% NSR royalty. This is capped at $35.4 M, which reflects the investment made by Freeport prior to the agreement with KSK.

22.3.8 Closure and Salvage

A nominal amount of US$6.8 M has been allowed for site closure at the end of the mine life. This amount is treated as a negative capital cost and is spread over the last four (4) years of the mine life. A sensitivity of project NPV to this component is included in Section 22.5.

As noted in Section 21.1.10, no salvage value has been allocated.

22.3.9 Taxes and Depreciation

The current CoW stipulates a 30% company tax rate. However this agreement is still under negotiation and the current tax laws within Indonesia are in a state of flux, with opportunities for "tax holidays" and a number of other provisions becoming available.

Therefore the application of tax is relatively simplistic and has been based on the prevailing company tax rate of 25%. A sensitivity based on the CoW requirement of 30% company tax is included in Section 22.5.

On the basis of an eight (8) year project life, depreciation of capital assets has been applied using a declining balance approach at 25%. Depreciation has been applied on the basis of the matrix shown in Table 22-1.

Table 22-1 Depreciation Matrix

Asset Life (yrs)	Declining-Balance	Straight-Line
4	50.0%	25.0%
8	25.0%	12.5%
16	12.5%	6.3%
20	10.0%	5.0%

No allowance has been made for:

- Import and / or export duties
- Stamp duties
- Withholding tax

- Any other taxes or levies

22.3.10 Discount Rate and Inflation

A discount rate of 10% has been applied, with additional sensitivity analysis in Section 22.5.

No allowance for inflation has been included.

22.4 ECONOMIC ANALYSIS RESULTS

The results of the Base Case economic analysis, and the key underlying assumptions, are provided in Table 22-2 and presented over time in Table 22-4 and Figure 22-1.

Table 22-2 BKM Project Economic Analysis Results

<table>
<tr><th colspan="2">Economic Summary</th><th>Unit</th><th>Base Case</th></tr>
<tr><td colspan="2">Life of Mine (LOM)</td><td>Years</td><td>8</td></tr>
<tr><td colspan="2">Copper Cathode Sold</td><td>Million lbs</td><td>391.0</td></tr>
<tr><td colspan="2">Copper Price (LOM Average)</td><td>$US/lb</td><td>3.25</td></tr>
<tr><td colspan="2">Gross Revenue</td><td>$US</td><td>1,270.6 M</td></tr>
<tr><td colspan="2">LOM C1 Operating Costs</td><td>$US</td><td>499.5 M</td></tr>
<tr><td colspan="2">LOM C1 Operating Cost (recovered copper)</td><td>$US/lb</td><td>1.28</td></tr>
<tr><td colspan="2">Royalties</td><td>$US</td><td>63.5 M</td></tr>
<tr><td colspan="2">Off-site transport</td><td>$US</td><td>19.8 M</td></tr>
<tr><td colspan="2" rowspan="2">LOM All In Operating Cost</td><td>$US</td><td>582.8 M</td></tr>
<tr><td>$US/lb</td><td>1.49</td></tr>
<tr><td colspan="2">Initial Capital Cost (including a 15% Contingency)</td><td>$US</td><td>163.8 M</td></tr>
<tr><td colspan="2">Taxes</td><td>$US</td><td>136.6 M</td></tr>
<tr><td colspan="4">NPV and IRR (Base Case)</td></tr>
<tr><td colspan="2">Discount Rate</td><td>Percent (%)</td><td>10</td></tr>
<tr><td rowspan="4">Pre-Tax</td><td>Net Free Cash Flow(including royalties)</td><td>$US</td><td>524.0 M</td></tr>
<tr><td>NPV</td><td>$US</td><td>290.7 M</td></tr>
<tr><td>IRR</td><td>%</td><td>47.5</td></tr>
<tr><td>Payback Period</td><td>Years</td><td>2.1</td></tr>
<tr><td rowspan="4">After Tax</td><td>Net Free Cash Flow (incl. royalties)</td><td>$US</td><td>387.5 M</td></tr>
<tr><td>NPV</td><td>$US</td><td>204.3 M</td></tr>
<tr><td>IRR</td><td>%</td><td>38.7</td></tr>
<tr><td>Payback Period</td><td>Years</td><td>2.4</td></tr>
</table>

Table 22-3 C1 Operating Cost[1]

Component	LOM Total $'000	$ / tonne mined	$ / tonne ore	$ / lb cathode
Mining	$232,969	$2.15	$4.78	$0.60
Crushing / Stacking	$65,985	$0.61	$1.35	$0.17
SX/EW Processing	$47,273	$0.44	$0.97	$0.12
Power	$131,064	$1.21	$2.69	$0.34
G&A and Support	$13,526	$0.12	$0.28	$0.03
Sustaining	$8,695	$0.08	$0.18	$0.02
C1 Cash Cost	**$499,512**	**$4.60**	**$10.25**	**$1.28**

1 Excludes off-site transport and royalties. Includes sustain costs

Figure 22-1 Financials by Year

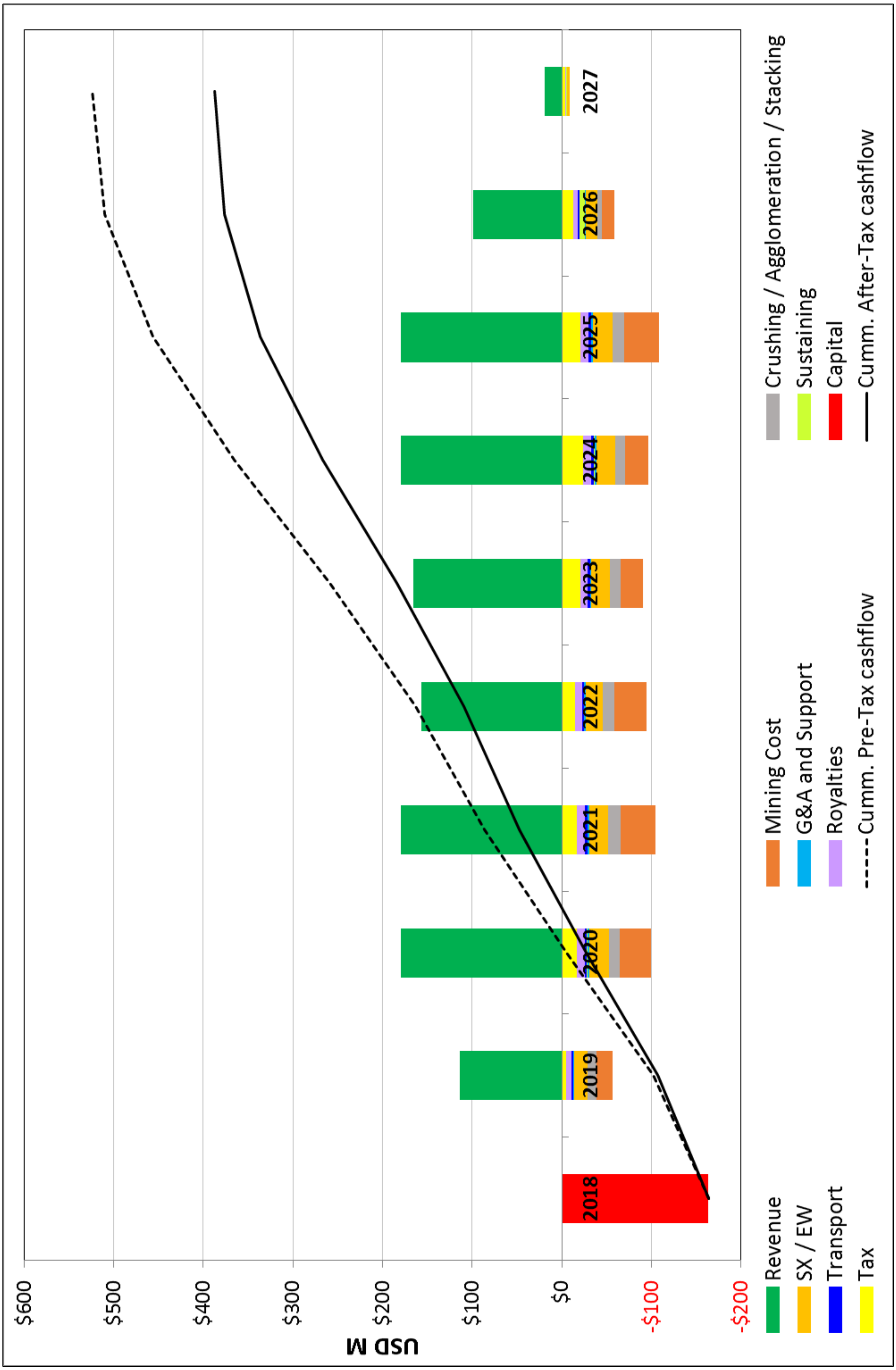


Table 22-4 Annualized Cashflow (2018 to 2027)

Item	units	Total	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027
Leach Pad Feed	kt	48,722	0	5,211	6,609	7,371	7,076	6,889	5,983	7,108	2,476	0
Total Cu	%	0.58%		0.79%	0.58%	0.55%	0.52%	0.56%	0.61%	0.52%	0.50%	0.00%
Leachable Cu	%	0.43%		0.56%	0.45%	0.40%	0.33%	0.43%	0.49%	0.41%	0.35%	0.00%
Metal Produced	kt	177		15,831	25,000	25,000	21,834	23,166	25,000	25,000	13,817	2,692
	M lbs	391.0	0.0	34.9	55.1	55.1	48.1	51.1	55.1	55.1	30.5	5.9
Cashstream 1: Revenue	**USD M**	**$1,270.6**	**$0.0**	**$113.4**	**$179.1**	**$179.1**	**$156.4**	**$166.0**	**$179.1**	**$179.1**	**$99.0**	**$19.3**
Initial Capital Costs	USD M	$163.8	$163.8									
Sustaining Costs	USD M	$0.9		$0.0	$0.9	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Closure Costs	USD M	$7.8	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.6	$0.6	$6.0	$0.6
Cashstream 2: Capital Cost	**USD M**	**$172.5**	**$163.8**	**$0.0**	**$0.9**	**$0.0**	**$0.0**	**$0.0**	**$0.6**	**$0.6**	**$6.0**	**$0.6**
Total Mining Cost	USD M	$233.0	$0.0	$18.2	$34.7	$39.4	$35.9	$25.1	$25.8	$39.6	$14.3	$0.0
Total Crushing / Stacking Costs	USD	$87.4	$0.0	$9.5	$11.9	$13.2	$12.7	$12.3	$10.8	$12.7	$4.4	$0.0
Total SX/EW Processing Costs	USD M	$156.9	$0.0	$14.8	$21.7	$21.7	$19.3	$20.3	$21.7	$21.7	$13.3	$2.6
Total G&A and Support Costs	USD M	$13.5	$0.0	$1.7	$1.7	$1.7	$1.7	$1.7	$1.7	$1.7	$1.4	$0.3
Total Transport Costs	USD M	$19.8	$0.0	$1.8	$2.8	$2.8	$2.4	$2.6	$2.8	$2.8	$1.5	$0.3
Total Government Royalties	USD M	$50.8	$0.0	$4.5	$7.2	$7.2	$6.3	$6.6	$7.2	$7.2	$4.0	$0.8
Total Vendor Royalties	USD M	$12.7	$0.0	$1.1	$1.8	$1.8	$1.6	$1.7	$1.8	$1.8	$1.0	$0.2
Cashstream 3: Operating Costs	**USD M**	**$574.2**	**$0.0**	**$51.6**	**$81.6**	**$87.6**	**$79.8**	**$70.3**	**$71.7**	**$87.4**	**$39.9**	**$4.1**
Operating Cost per lb. rec.	USD/lb	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0	$0.0
Pre-Tax Net Free Cashflow	USD M	$524.0										
Depreciation	USD M	$150.3	$0.0	$40.9	$30.9	$23.2	$17.4	$13.1	$9.9	$7.6	$7.2	$0.0
Cashstream 4: Taxes	**USD M**	**$136.6**	**$0.0**	**$5.2**	**$16.6**	**$17.1**	**$14.8**	**$20.7**	**$24.4**	**$21.0**	**$13.0**	**$3.8**
After-Tax Net Free Cashflow	USD M	$387.5	-$163.8	$56.6	$79.9	$74.4	$61.8	$75.0	$82.5	$70.1	$40.1	$10.8
Cumulative Cashflow	**USD M**		**-$163.8**	**-$107.2**	**-$27.2**	**$47.2**	**$109.0**	**$184.0**	**$266.5**	**$336.6**	**$376.7**	**$387.5**
Discounted Cashflow	USD M	$204.3	-$163.8	$51.4	$66.1	$55.9	$42.2	$46.6	$46.5	$36.0	$18.7	$4.6
Cumulative NPV	**USD M**		**-$163.8**	**-$112.3**	**-$46.3**	**$9.7**	**$51.9**	**$98.5**	**$145.0**	**$181.0**	**$199.7**	**$204.3**

22.5 SENSITIVITY ANALYSIS

The primary sensitivity scenario run was for copper price, which effectively replicates a variation in grade and recovery as it is a variation in revenue. The results of this sensitivity are presented in Table 22-5. The sensitivity to price and costs are presented in Table 22-6 and graphically in Figure 22-2.

Other specific sensitivities run were as follows:

- Discount Rate at 8% and 15%
- Closure Costs x 2 and x 3
- Income Tax @ 30%

These results are detailed in Table 22-7.

Table 22-5 Sensitivity Analysis – Price

Copper Base Price		$2.75	$3.00	$3.25	$3.50	$3.75
		-15.5%	-7.5%	(Base Case)	7.5%	15.5%
Net Price after Royalty		$2.61	$2.85	$3.09	$3.33	$3.56
Pre-Tax	NPV @ 10%	$161.0 M	$225.9 M	$290.7 M	$355.6 M	$420.4 M
	I.R.R.	32.3%	40.1%	47.5%	54.7%	61.7%
	Pay-back (Undisc.)	2.8	2.4	2.1	1.8	1.7
Post-Tax	NPV @ 10%	$74.6 M	$139.4 M	$204.3 M	$269.1 M	$334.0 M
	I.R.R.	21.6%	30.5%	38.7%	46.4%	53.8%
	Pay-back (Undisc.)	3.7	2.8	2.4	2.1	1.8

Figure 22-2 NPV Sensitivity by Variation in Price and Costs

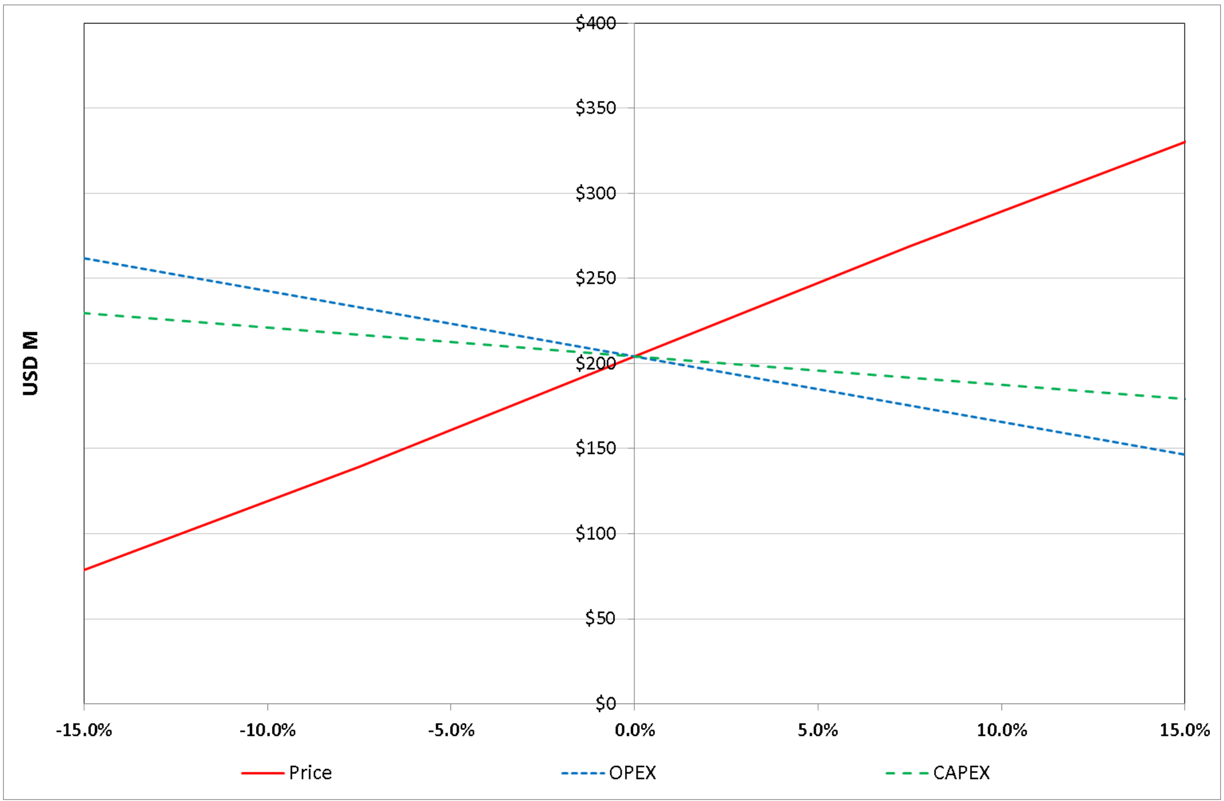


Table 22-6 Sensitivity Analysis – % Variation in NPV10 by % Variation in Key Driver

Key Driver		% Variation in Key Driver		
		-15.5%	Base Case	+15.5%
NPV10	Price	$74.6 M	$204.3 M	$334.0 M
	OPEX	$263.8 M		$144.7 M
	CAPEX	$230.3 M		$178.2 M
IRR	Price	21.6%	38.7%	53.8%
	OPEX	45.9%		31.1%
	CAPEX	47.0%		32.4%

Table 22-7 Sensitivity Analysis – Specific Cases

Parameter	NPV10	IRR
Base Case	$204.3 M	38.7%
Discount rate @ 8%	$232.7 M	38.7%
Discount rate @ 15%	$145.6 M	38.7%
Closure Cost x 2	$200.6 M	38.5%
Closure Cost x 3	$196.9 M	38.3%
Income Tax @ 30%	$187.0 M	36.8%

23 ADJACENT PROPERTIES

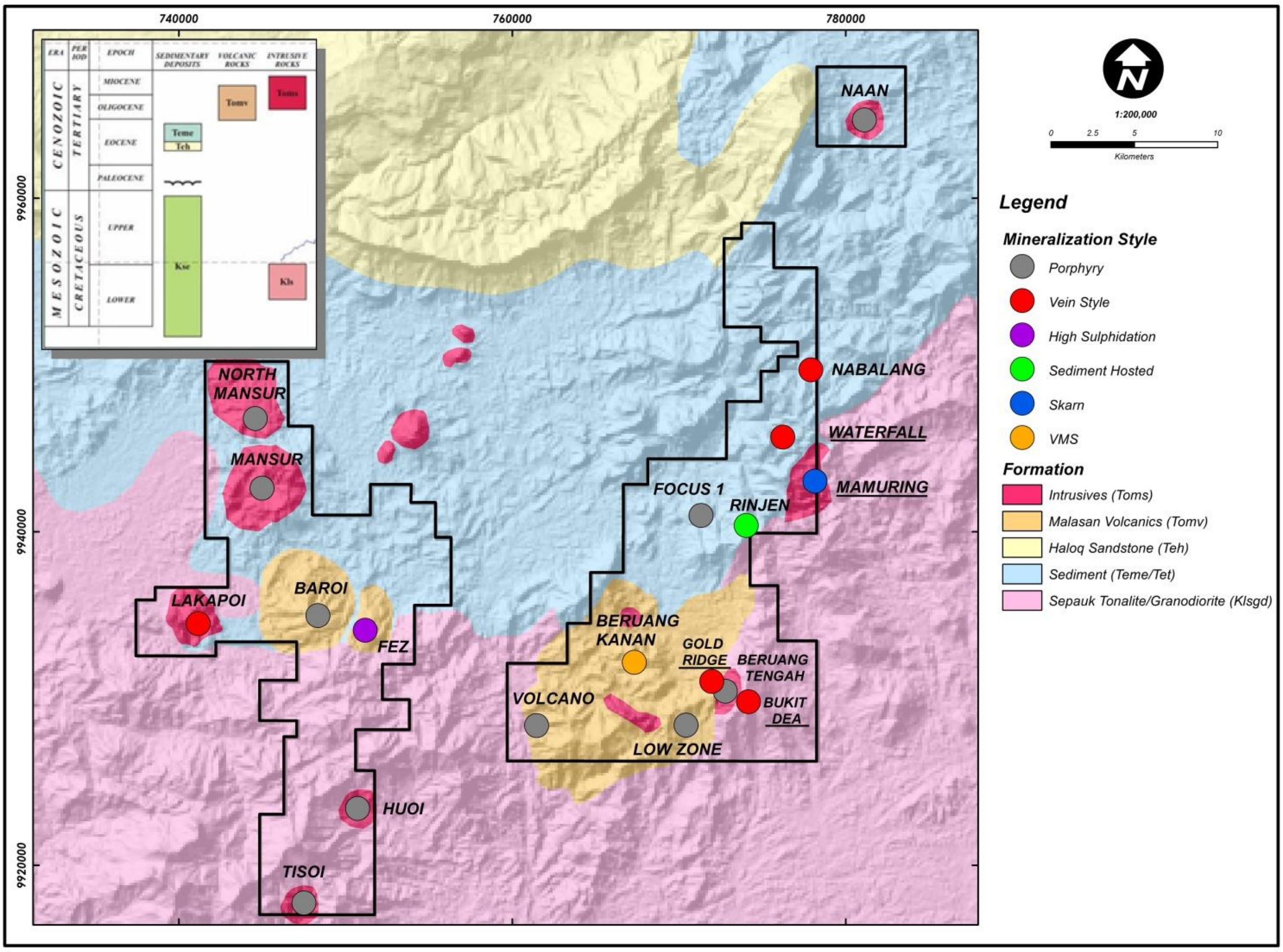
23.1 KSK PROJECTS

The following main projects have also been the subject of exploration activities within the KSK CoW (refer Figure 23-1 for location with respect to BKM):

- Baroi - Central and Far-East Prospects (Au, Cu, Ag, Mo): Vein-hosted mineralization in volcanic and sedimentary rocks. Veined material consistently returns copper grades of 1-5% and elevated Zn, Ag and Pb. The Central zone also contains elevated Au grades (1-3g/t). Baroi is 17km from BKM.
- Beruang Tengah - Eastern (Au, Cu) and Western (Au) Prospects. Beruang Tengah is 4km from BKM.
- Mansur Prospect (Cu Au). Mansur is 25km from BKM.
- BKM - North Polymetallic (Pb, Zn, Cu, Ag, Au) and South (Cu, Au) Prospects. These prospects are proximal to BKM.

These prospects have geological, geochemical and alteration characteristics that are consistent with mineralization being associated with porphyry intrusives and intrusive related vein systems (Munroe S, and Clayton M, 2006). Mineralization has been identified at all of these prospects and they are the target ongoing exploration. To this date none of the prospects have been subjected to drilling to the extent of that directed at BKM. Detailed descriptions of the geology and mineralization are reported by (Munroe S, and Clayton M, 2006).

Figure 23-1 Geological setting and location of prospects and main geochemical anomalies in the KSK CoW



Modified after (Munroe S, and Clayton M, 2006)

23.2 COMPANIES WORKING WITHIN THE KSK COW REGENCY

Two IUPs abut the KSK CoW; to the east, the tenure is held by PT. Kahayan Mineral, and to the north by PT. Persada Makmur Sejahtera. There is no information readily available on the activities undertaken by these companies.

There are further IUPs to the south of the KSK CoW. These do not adjoin the CoW

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information needed for the description of the PEA study in this Technical Report other than an evaluation of identified project risks and opportunities.

The BKM Project is subject to the same inherent risks and opportunities as any extractive industry project. This includes, but is not limited to, positive and negative variations in:

- Estimated Resources
- Copper price
- Geotechnical assessment of open pits, waste rock dumps and heap leach
- Capital and operating cost

- Inflation
- Fiscal / royalty / taxation regimes
- Land tenure and environmental permitting
- Impacts on terrestrial and aquatic ecology
- Continuity and effectiveness of community relations programs

24.1 OPPORTUNITIES

Opportunities include, but are not limited to:

- Extensions of the orebody along strike or to depth, or inclusion of adjacent deposits within the KSK CoW
- Increased recovery of leachable copper above the 85% assumed
- Potential to reduce power costs further by utilizing hydro generation
- Improved community relations through increased opportunities for employment and provision of services/good by local residents

24.2 RISKS

Specific risks that future works will need to focus on are:

- Uncertainties in long term management of acid rock drainage and metal leaching from mine workings, mine waste and the exhausted heap leach
- Site water balance and associated water management plan is not yet defined or quantified due to the current lack of data. This may result in:
    - Increased sediment loads in area streams during construction
    - Overtopping of stormwater retention pond due to extreme rain event
- Deterioration in community relations associated with in migration and perceived unfair hiring practices

25 INTERPRETATION AND CONCLUSIONS

25.1 GENERAL

Orelogy has prepared a PEA for the BKM Project and a supporting technical report prepared in accordance with NI 43-101 and Form 43-101F1. The PEA describes the potential technical and economic viability of establishing a conventional open pit mine, a heap leach / SX / EW operation and related infrastructure to process high sulphidation style copper mineralization from the BKM Deposit. Based on the work carried out in this PEA and the resultant economic evaluation, this study should be followed by further technical and economic studies leading to a prefeasibility study.

25.2 PROPERTY DESCRIPTION AND LOCATION

Information from KSK supports that the mining tenure held is valid and is sufficient to support declaration of mineral resources. Tenures have been surveyed in accordance with appropriate regulatory requirements and all applicable permits for current work activities are valid. There are no known environmental liabilities due to previous works or ongoing KSK exploration activities at BKM.

25.3 GEOLOGY

The BKM deposit is a covellite, chalcocite and chalcopyrite vein style copper mineralized system hosted in sheared sediments and volcanics within an interpreted thrust fault-coupling or ramping zone. Knowledge of the deposit settings, lithologies, structural and alteration controls on mineralization and rock density, and the mineralization style and setting is sufficient to support mineral resource estimation of Inferred and Indicated Classifications (CIM guidelines).

The exploration and drilling programs completed to date are appropriate to the type of the deposit. The exploration, drilling, geological modelling and research work supports ore genesis interpretations that satisfy Inferred and Indicated resource classifications (CIM guidelines).

Hackman & Associates (H&A), the QP undertaking the geological evaluation, has reviewed the quantity and quality of the data collected by KSK for defining a mineral resource and found it reliable and suitable for defining Indicated and Inferred Resources and for the purposes of this study. Review of the QA/QC dataset and reports found KSK’s sample preparation, analysis and security procedures appropriate and adequate for their latter work and, from favorable observations when comparing this dataset with earlier datasets, H&A has no reason to suspect any deterioration of diligence or security in the earlier work.

It is the opinion of H&A that the drill database for the BKM Project is reliable and sufficient to support the purpose of this technical report and the November 30, 2015 mineral resource estimate.

25.4 MINERAL RESOURCE ESTIMATION

The mineral resource estimate, as shown in

Table 25-1, which formed the basis of this PEA was completed by Mr. Duncan Hackman (B.App.Sc., MSc., MAIG) of Hackman & Associates, an independent QP as set forth by NI 43-101. The effective date of this resource estimate is November 30, 2015. Estimations of mineral resources for the BKM Project are considered appropriate for the form and styles of copper mineralization at BKM and follow the guidelines set out in CIM (2010).

Table 25-1 Tabulated Copper Resources - Beruang Kanan [Indicated and Inferred Classified Resources reported separately].

Indicated Mineral Resources				
Reporting cut (Cu %)	Tonnes ('000)	Cu Grade (Cu %)	Contained Cu ('000 tonnes)	Contained Cu ('000,000 lbs)
0.2	15,000	0.7	105	231
0.5	12,600	0.7	88	194
0.7	5,600	0.9	50	110

Inferred Mineral Resources				
Reporting cut (Cu %)	Tonnes ('000)	Cu Grade (Cu %)	Contained Cu ('000 tonnes)	Contained Cu ('000,000 lbs)
0.2	49,700	0.6	298	657
0.5	25,300	0.7	177	390
0.7	9,800	0.9	88	194

Notes 1: *Mineral Resources for the Beruang Kanan mineralization have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. In the opinion of Duncan Hackman, the block model Resource estimate and Resource classification reported herein are a reasonable representation of the copper Mineral Resources found in the defined area of the Beruang Kanan Main mineralization. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserve. Computational discrepancies in the table and the body of the Release are the result of rounding.*

Notes 2: *Mineral Resources are reported based on a Total Copper cut-off grade of 0.2% based on initial economic, geotechnical and metallurgical parameters which are considered reasonable. The subsequent mine plan detailed in Section 16.5 is based on a Recoverable Copper Grade which is Total Copper x Leachable Conversion x Process Recovery (@85%). The break-even cut-off grade for the project, based on the final economic parameters detailed in Section 21, is approximately 0.08% Recoverable Copper, which equates to approximately 0.11% Total Copper (refer to Table 25 2).*

However the LOM schedule has been developed utilizing a variable elevated cut-off grade strategy that is optimized over time to maximize the project value. The optimized cut-off grade ranges between 0.09% and 0.11% Recoverable Copper over the life of the project. This equates to a Total Copper cut-off grade range of approximately 0.12% to 0.15%. Therefore the use of a resource cut-off of 0.2% can be considered appropriate for the total resource.

Notes 3: *Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content.*

Factors which may affect the Mineral Resource estimate include:

- Advancements / changes to the geological, geotechnical and geometallurgical models
- Infill drilling to convert mineral resources to a higher classification
- Extension drilling to test for extensions of known resources to depth or along strike

Additional factors which may affect the open pit extraction scenario used to constrain the estimates are:

- Commodity price and exchange rate assumptions
- Assumptions used to estimate metallurgical recoveries
- Pit slope angles

It should be noted that all factors pose potential risks and opportunities to the current mineral resource. Recommendations to address these risks and opportunities are provided in Section 26.

25.5 METALLURGICAL TEST WORK AND PROCESS DESIGN

Metallurgical test work and associated analytical procedures were performed by recognized testing facilities, and the tests performed were appropriate to the mineralization type. The samples selected for testing were representative some of the various types and styles of mineralization present at the BKM Deposit.

The testwork completed is of a preliminary nature. The results provide sufficient data, in conjunction with appropriate industry experience, to make informed assumptions on leaching recovery and acid balance.

A process had been designed for the BKM Project to support a copper cathode production of 25,000 tonnes per year. The proposed process plant consists of primary crushing, agglomeration, truck haulage to a stacker, heap leaching, solvent Extraction (SX) and electrowinning (EW). The operating philosophy and selected equipment is in line with industry expectations.

Table 25-2 summarizes estimated total copper grades, leachable copper grade and leaching recoveries over time.

Table 25-2 Metallurgical Recoveries by Year

Item	Unit	Totals	2019	2020	2021	2022	2023	2024	2025	2026
Cu Total Head Grade	%	**0.58**	0.79	0.58	0.55	0.52	0.56	0.61	0.52	0.50
Conversion Cu Total to Cu Leach	%	**74.1**	71.3	77.2	72.5	63.6	75.6	80.7	79.6	70.3
Cu Leach Head Grade	%	**0.43**	0.56	0.45	0.40	0.33	0.43	0.49	0.41	0.35
Recovery of Cu Leachable	%	**85**	85	85	85	85	85	85	85	85
Overall Recovery	%	**63.0**	**60.6**	**65.6**	**61.6**	**54.1**	**64.2**	**68.6**	**67.7**	**59.7**

Further process optimization will be undertaken, that incorporates site specific conditions and ongoing metallurgical test work results, to maximize project economics.

25.6 MINING METHODS

The BKM Project's total project life is approximately 8 ½ years, including approximately 1 year of leaching / cathode production after mining has ceased.

The mine utilizes a conventional truck-and-shovel fleet and the proposed open pit operations will provide leach plant feed at an average annual production rate of approximately 15 Mtpa. Peak total material movement is approximately 18 Mtpa. The pit design incorporates a bench height of 5 m and an inter-ramp angle of 52.5°. Pit optimization and design results are based on scoping level geotechnical and operational information. The mining schedule utilizing nine (9) internal mining stages to balance material movement, strip ratio and head grade requirements.

Factors which may affect the mine plan include advancements and/or changes to:

- the geotechnical and hydrological assumptions
- commodity price
- metallurgical recoveries

The use of an experienced mining contractor reduces some of the risks associated with the project start-up, particularly any complexities associated with initial pioneering development on the steeper topography.

All factors and assumptions used to develop the PEA mine plan contain inherent potential risks and opportunities. Recommendations to address these risks and opportunities are provided in Section 26.

25.7 PROJECT INFRASTRUCTURE

The BKM Project infrastructure includes building locations which are based on the assumption that subsequent geotechnical evaluation is favorable. However further investigation could result in modifications to the planned infrastructure layout.

One of the key infrastructure components to be further reviewed is the exact location, layout and ancillary infrastructure requirements of the "rental" power station proposed by PT. Prastiwahyu Trimitra Engineering (PTE). Discussions between PTE and KSK are on-going to confirm this arrangement.

Further detail will also be sourced from the mining contractors on the requirements for their infrastructure, and all associated inclusions and exclusions.

25.8 WATER MANAGEMENT

The work completed to date suggests that the current water management concept is practicable and can, with further optimization, be carried forward to the next level of study. The surficial geology and geotechnical conditions should be investigated and the design basis and operating criteria should be further refined.

25.9 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The BKM Project will be subject to a mine permitting process typical for a mine of its size in Indonesia.

The four key approvals in support of project permitting are:

1) Feasibility Study
2) ESHIA (called AMDAL in Indonesia)
3) Mandatory 5-Year Reclamation and Mine Closure Plans; and
4) Borrow-to-Use Forestry Permit (IPPKH).

The BKM Project is located in an area designated as Production Forest, which allows development of open pit mining. Furthermore, given the final product from the mine will be copper cathode, MEMR Regulation No. 11 of 2012, commonly known as the raw mineral export ban, does not apply to the BKM Project. There are therefore no regulatory hindrances for the permitting, construction, operation and export of copper for the proposed BKM Project.

25.10 ECONOMIC ANALYSIS

Orelogy prepared an economic evaluation of the BKM Project based on a pre-tax and post-tax financial model. NPV was estimated at based on initial capital being un-discounted and discounting of cashflow from the first year of production. Economic performance measures such as NPV and IRR were estimated on both pre-tax and post-tax basis for the Base Case and four alternate metal prices of +/- 15.5 and +/- 7.5.

Sensitivity analyses was also conducted to analyze the sensitivity of the BKM Project to the key economic inputs. Using the Base Case as a reference, the key variables of price, OPEX and CAPEX were changed between -15.5 and +15.5 while holding the other variables constant. The BKM Project's NPV, calculated at a 10 discount rate, was found to be most sensitive to copper price and, in decreasing order OPEX and then CAPEX. The IRR was found to also be most sensitive to the copper price but marginally more sensitive to CAPEX than OPEX.

Using the base inputs as described in Section 22.3, the pre-tax NPV at a 10 discount rate was \$290.7 M and IRR was 47.5. After-tax NPV^{10} was \$204.3 M and after-tax IRR was 38.7. The estimated payback period for the project is 2.4 years on an after-tax basis.

26 RECOMMENDATIONS

26.1 GENERAL

The Qualified Person's that have undertaken this PEA collectively recommend the actions and activities detailed in the following sections to support the advancement of the BKM Project.

This assumes KSK decide to proceed to the next level of project assessment.

The following additional works are recommended to progress the BKM Project:

- Geotechnical assessment
- Metallurgical evaluation
- Mining and process engineering studies
- Waste-rock characterization
- Additional baseline studies and environmental permitting activities
- Marketing studies
- Various trade-off studies.

As part of the recommended work program, additional exploration and geotechnical drilling will be required to support some of the further works. Additional areas for work are also likely to be identified as activities progress.

The following sections detail the recommendations for future activities to progress the BKM Project. The cost associated with these additional works programs has not been included in this PEA.

26.2 GEOLOGY

Duncan Hackman & Associates makes the following recommendations:

- Waste rock characterization including:
  - geomechanical modelling to establish blasting performance and diggability
  - geochemical modelling to establish potential acidity of surface run-off over the temporary waste dumps and pits

26.3 MINERAL PROCESSING AND METALLURGICAL TESTING

Miller Metallurgical Services recommends that a comprehensive suite of metallurgical testing is commenced in order to arrive at key process design criteria. The items to be assessed include

- Appropriate crush size for leaching kinetics
- Recovery of leachable copper from the various mineral and rock mixtures to be treated
- Acid balance during leaching
- Confirmation of the proposed leaching pathway (i.e. combination of galvanic and biological assisted leaching)

This work can be undertaken in a phased manner over time as the project progresses towards successive levels of study.

26.4 MINING METHODS

Orelogy makes the following recommendations for future work:

- Conduct a geotechnical evaluation to define appropriate pit slope parameters. This will constitute a component of a larger site-wide geotechnical assessment.
- Conduct a hydrology / hydrogeology study to determine correct levels of pit in-flow and develop a detailed water management plan. This will constitute a component of a larger site-wide hydro assessment.
- Utilize results of waste rock characterization study to determine mine waste rock management protocols particularly for ML/ARD issues, and identify potential NAF or NAR materials
- Assess opportunities to backfill pits rather than dump waste rock ex-pit
- Undertake a trade-off study to determine the optimum SX / EW production rate and associated crushing /stacking capacity.
- Determine appropriate blending requirements based on updated metallurgical test work and develop methodology to deliver required blend (i.e. stockpiling, in-pit blending or both)
- Optimize pit staging size and sequence

26.5 PROCESS PLANT DESIGN

Miller Metallurgical Services makes the following recommendations:

- Conduct a hydrology / hydrogeology study to accurately develop an overall site water balance and associated management strategy (refer to Section 26.4 above)
- Geotechnical evaluation of the stacked LFI to fully assess the overall heap leach stability (refer to Section 26.4 above)
- Commence a phased, comprehensive suite of materials testing to arrive a key process design criteria including:
    - Crushing characteristics
    - Materials handling characteristics
    - Geo-mechanical and hydrological characterization of unleached and leach LTI for stacking and irrigation planning
    - Identification of suitable materials for heap and base construction

26.6 PROJECT INFRASTRUCTURE

DRA Pacific Pty Ltd and Orelogy make the following recommendations:

- Conduct site ground investigation study to determine suitability of current infrastructure locations. This will also assess any geo-hazards related to insitu slope stability.
- Establishment and maintenance of a suitable site access road will be outsources to an Indonesian logistic provider.
- Negotiations with a preferred electrical power provider will need to be advanced to ensure and site specific studies or assessments are undertaken in a timely manner, as this is a key infrastructure

component for the project. An assessment of the viability of hydro-generated power should be included as part of this process.

- Baseline work should be conducted to support design parameters that will support the design of the proposed water management infrastructure. This will include detailed field data acquisition and laboratory testing programs to characterize hydrological, hydrogeological and geo-environmental conditions for the proposed project.
- Conduct a detailed route survey to ascertain the condition of the roads and refine upgrade criteria

26.7 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

PT Lorax Indonesia makes the following recommendations as detailed in Section 20.2.3:

- Conduct comprehensive environmental baseline (taking into account seasonal variability)
- Conduct comprehensive ARD/ML, and social and public health baseline studies

26.8 CAPITAL AND OPERATING COSTS

Further detail and refinement of the project costs will be developed by all contributors to the next phase of study on the

26.9 ECONOMIC ANALYSIS

Further detailed assessment of appropriate allowances for taxation, depreciation and inflation are required, along with a schedule of capital expenditure prior to production start-up.

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APPENDICES

APPENDIX A Qualified Person Statements

CERTIFICATE OF QUALIFIED PERSON

I, Ross Cheyne, (F.AusIMM, BE Mining) am employed as Managing Director and Principal Consultant by Orelogy Consulting Pty Ltd (Orelogy), U1/i62 Colin St, West Perth, Western Australia 6005. AUSTRALIA.

This certificate applies to the technical report titled "*Beruang Kanan Main Copper Project Central Kalimantan, Indonesia Ni 43-101 Independent Technical Report - Preliminary Economic Assessment*" that has an effective date of 19 May, 2016 (the "technical report") and supports Asiamet's press releases entitled "*Asiamet BKM deposit PEA delivers US$204m NPV10 and 39% IRR*", dated 5 April, 2016.

I am a Fellow of the Australian Institute of Mining and Metallurgy (F.AusIMM 109345). I graduated from the University of Auckland, New Zealand in 1989 with a Bachelor of Mining Engineering (Hons).

I have practiced my profession for 25 years. I have been directly involved in precious and base metals, mining, consulting, and project management in Australia, Ghana, Malaysia, Myanmar, China, Kazakhstan and Namibia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not undertaken a site visit to the BKM Project.

I am responsible for the following sections of the report:

Section 1 to 3
Section 15 to 16
Section 18.3
Section 18.5
Section 19
Section 21.1.1 to 21.1.2
Section 21.1.6 to 21.1.10
Section 21.2.1 to 21.2.2
Section 21.2.4 to 21.2.8
Section 22
Section 24
Section 25.1
Section 25.6
Section 25.8
Section 25.10
Section 26.1
Section 26.4
Section 26.8
Section 26.9
Section 27

I am independent of PT Kalimantan Surya Kencana (KSK) and Asiamet Resources Ltd (ARS) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the BKM Project during the preparation of the Preliminary Economic Assessment on which this technical report is based.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 May, 2016

Ross Cheyne
F.AusIMM, BE Mining

CERTIFICATE OF QUALIFIED PERSON

I, Duncan Hackman, (MIAG, B. App. Sc., MSc.) am employed as Principal Consultant by Hackman and Associates (H&A), 260A Crawford Rd, Inglewood, Western Australia 6052, AUSTRALIA.

This certificate applies to the technical report titled "*Beruang Kanan Main Copper Project Central Kalimantan, Indonesia NI 43-101 Independent Technical Report - Preliminary Economic Assessment*" that has an effective date of 19 May, 2016 (the "technical report") and supports Asiamet's press releases entitled "*Asiamet BKM deposit PEA delivers US$204m NPV10 and 39% IRR*", dated 5 April, 2016.

I hold a Bachelor of Applied Science, Queensland Institute of Technology (1985), and a Master of Science, James Cook University (1996).

I am a member of the Australian Institute of Geoscientists AIG (MAIG 1742). I have practiced my profession for 30 years. I have been directly involved for 12 years in copper resource evaluation and mining, including deposits such as Khanong (Laos), Prominent Hill (Australia) and Golden Grove (Australia).

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I have undertaken several site visits to the BKM Project, the most recent being between the 21st and 28th June, 2015.

I am responsible for the following sections of the report:
Section 4 to Section 12
Section 14
Section 23
Section 25.2 to Section 25.4
Section 26.2

I am independent of PT Kalimantan Surya Kencana (KSK) and Asiamet Resources Ltd (ARS) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the BKM Project during the preparation of the Preliminary Economic Assessment on which this technical report is based.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 May, 2016

Duncan Hackman
MIAG, B. App. Sc., MSc.

CERTIFICATE OF QUALIFIED PERSON

I, Graeme Miller (F.AusIMM, BE (Mech), CP-AusIMM) am a director of Miller Metallurgical Services Pty Ltd (MillerMet), 23 Stanfell St, Corinda, Queensland 4075, AUSTRALIA.

This certificate applies to the technical report titled "*Beruang Kanan Main Copper Project Central Kalimantan, Indonesia Ni 43-101 Independent Technical Report - Preliminary Economic Assessment*" that has an effective date of 19 May, 2016 (the "technical report") and supports Asiamet's press releases entitled "*Asiamet BKM deposit PEA delivers US$204m NPV10 and 39% IRR*", dated 5 April, 2016.

I am a Fellow of the Australian Institute of Mining and Metallurgy. I graduated from the University of Queensland with a Bachelor of Chemical Engineering 1974.

I have practiced my profession for 35 years. I have been directly involved in precious and base metals, processing, consulting, and engineering management in Australia, Namibia, Zambia, DRC, Myanmar, Kazakhstan and Indonesia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not undertaken a site visit to the BKM Project.

I am responsible for the DRA input into the following sections of the report:
Section 13
Section 17
Section 18.1
Section 21.1.3 to 21.1.5
Section 21.2.3
Section 25.5
Section 26.3
Section 26.5

I am independent of PT Kalimantan Surya Kencana (KSK) and Asiamet Resources Ltd (ARS) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the BKM Project during the preparation of the Preliminary Economic Assessment on which this technical report is based.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 May, 2016

G Miller

Graeme Miller
F.AusIMM, BE (Chem), CP-AusIMM

CERTIFICATE OF QUALIFIED PERSON

I, Ali Sahami (Ph.D) am the President Director of PT Lorax Indonesia (Lorax), Menara Rajawali, 22nd Floor, Jl. DR. Ide Anak Agung Gde Agung, Kawasan Mega Kuningan, Jakarta, 12950, INDONESIA .

This certificate applies to the technical report titled "*Beruang Kanan Main Copper Project* Central Kalimantan, Indonesia Ni 43-101 Independent Technical Report - Preliminary Economic Assessment" that has an effective date of 19 May, 2016 (the "technical report") and supports Asiamet's press releases entitled "*Asiamet BKM deposit PEA delivers US$204m NPV10 and 39% IRR*", dated 5 April, 2016.

I hold a Ph.D. in Geochemistry from University College London (UK). I have practiced my profession (environmental scientist) for the past 25 years. I have been directly involved in precious and base metals, mining, consulting and project management in Canada, USA, Mexico, Bolivia,, Columbia, Peru, Phillipines and Indonesia. I have worked on environmental aspects of metal and coal mining projects in Indonesia over the past 15 years.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not undertaken a site visit to the BKM Project.

I am responsible for the DRA input into the following sections of the report:
Section 20
Section 25.9
Section 26.7

I am independent of PT Kalimantan Surya Kencana (KSK) and Asiamet Resources Ltd (ARS) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the BKM Project during the preparation of the Preliminary Economic Assessment on which this technical report is based.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 May, 2016

Sahami

Ali Sahami
Ph.D

CERTIFICATE OF QUALIFIED PERSON

I, Johan Du Preez (BSc Eng., P.Eng) am employed as Senior Infrastructure Design Engineer by DRA Global Pty Ltd (DRA), 3 Inyanga Cl, Johannesburg, 2157, SOUTH AFRICA.

This certificate applies to the technical report titled "*Beruang Kanan Main Copper Project Central Kalimantan, Indonesia Ni 43-101 Independent Technical Report - Preliminary Economic Assessment*" that has an effective date of 19 May, 2016 (the "technical report") and supports Asiamet's press releases entitled "*Asiamet BKM deposit PEA delivers US$204m NPV10 and 39% IRR*", dated 5 April, 2016.

I graduated from the University of the Witwatersrand South Africa in 1977 with a Bachelor of Science (Engineering). I am a Profession Engineer with the Engineering Council of South Africa (Membership Number 870132).

I have practiced my profession for 39 years. I have been directly involved in precious and base metals, mining and consulting in Australia, Botswana, Ghana, Namibia, Tanzania, Mozambique, India, DRC, Zambia, Zimbabwe and South Africa.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not undertaken a site visit to the BKM Project.

I am responsible for the following sections of the report:
Section 18.2
Section 18.4
Section 18.6
Section 18.7
Section 18.8
Section 25.7 (excluding Electrical Engineering Input)
Section 26.6 (excluding Electrical Engineering input)

I am independent of PT Kalimantan Surya Kencana (KSK) and Asiamet Resources Ltd (ARS) as independence is described by Section 1.5 of NI 43–101.

I have been involved with the BKM Project during the preparation of the Preliminary Economic Assessment on which this technical report is based.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 19 May, 2016

Johan Du Preez
BSc Eng., P.Eng.