NI 43-101 Technical Report for the Kwanika Project Preliminary Economic Assessment Update 2017

Revision of Technical Report issued 19 April 2017

Near Fort St. James, British Columbia, Canada

Centred at 55’ 31’ N and 125’ 20’ W

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1 Summary

The Serengeti Resources Inc. Kwanika Project (the Project) involves the development of a copper-gold deposit located near Fort St. James, British Columbia, Canada.

This National Instrument 43-101 (NI 43-101) Technical Report on the Project has been prepared by Moose Mountain Technical Services (MMTS) and is based on work produced by the following independent consultants:

- SRK Consulting (Canada) Inc. (SRK)
- Moose Mountain Technical Services

The resource evaluation work of this technical report was completed by Marek Nowak, P.Eng and Chad Yuhasz, P.Geo of SRK, Mr. Nowak and Mr. Yuhasz are independent Qualified Persons for the geological resources presented in this Technical Report.

Mr. Jim Gray (P.Eng.) of MMTS visited the Project site on October 18, 2011. He is the independent Qualified Person for all matters relating to infrastructure, mining, mining capital costs, mining operating costs, financial evaluation, environmental and regulatory aspects, and overall report preparation.

Mr. Tracey Meintjes (P.Eng.) of MMTS is the independent Qualified Person for matters relating to mineral processing, mineral processing capital, mineral processing operating costs, and metallurgical testing.

The Preliminary Economic Assessment (PEA) is based on exploration and internal Serengeti studies since 2004. The Resource model used is most recently described in an NI 43-101 Technical Report dated December 2016, which is referenced in this report. This PEA adds a revised mine plan and infrastructure design at scoping level of accuracy to the 2016 Resource by SRK. This 2017 design work is a revision to the 2013 Technical Report. The mine plan includes an integrated open pit and underground production schedule using typical operating parameters for comparable projects. For the 2013 Technical Report a conceptual evaluation by AMEC indicated that Block Caving is a suitable underground mining method as applied to this study. The AMEC evaluation is still relied upon for this update (See Appendix B).

All dollar figures presented in this report are stated in Canadian dollars unless otherwise specified.

General Project Information is summarized in Table 1-1.

Table 1-1 General Project Information

Description	Unit	Amount
Estimated Mineral Resources (Measured + Indicated)*	Mt	131.2
Estimated Mineral Resources (Inferred)*	Mt	73.1
Life-of-mine (LOM)	years	15
Milling Rate	t/d	15,000
Strip Ratio – Open Pit (LoM)	t waste: t milled	1.7
Total Project Capital Cost	$ million	$476
Average Overall Operating Cost	$ /t milled	$21.15
Copper Price	US$/lb	$2.90
Gold Price	US$/oz	$1,270
Silver Price	US$/oz	$19.00
Internal Rate of Return (IRR) Pre Tax/Post Tax	%	21.1/16.6
Net Present Value at 7% Discount Rate Pre Tax/Post Tax	$ million	$324.4/$191
Payback Period Pre Tax/Post Tax	years	3.73/4.0
Cash Cost per lb Cu (net of Au & Ag credits) Yr 1 – 8 & LOM	US$	$0.70/$1.20

* Within a Designed Open Pit or Underground shape.

1.1 Property Description and Location

The Kwanika property in north central British Columbia is situated in the Omineca Mining Division, approximately 140km northwest (approximately 200km by road) of Fort St. James, located on NTS map sheets 93N06 and 93N11, at latitude 55º31’ N and longitude 125º20’ W. The property is accessible year-round by four-wheel-drive vehicle, provided there is active snow removal in winter.

The property is the host of two porphyry-style mineral deposits: the copper-gold-molybdenum-silver South Zone and the copper-gold-silver Central Zone, both of which encompass current Mineral Resources.

1.2 Property Ownership

The Kwanika property consists of 29 contiguous unpatented mineral claims covering an area of 9,418.4ha and is jointly owned by Serengeti Resources Inc. (95%) and Daewoo Minerals Canada Corporation (5%). It is not subject to any royalties or other outstanding liabilities. Serengeti acquired the current extent of the property through staking between 2004 and 2006.

1.3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Kwanika Property is located approximately 75km to the southwest of the Kemess power line, and CN Rail maintains an active rail line to Fort St. James. The Kwanika Project is also in close proximity to the well-serviced communities of Prince George, Smithers, Fort St. James, and Mackenzie. Access to the Kwanika Property from Fort St. James is via the all-weather Leo Creek and Driftwood forest service roads (FSR) and the 30km long Tsayta Lake Road. Other access infrastructure on the Kwanika Property consists of gravel logging roads and several kilometres of excavated trails. There is sufficient water available in the immediate vicinity of the property to support both exploration and potential mining activities.

Serengeti has developed a beneficial association with the local Takla Lake First Nation and there has been community support for the Kwanika Project and the potential employment that it would provide.

The average temperature for this area is 3.1°C, with a peak average monthly temperature of 21.9°C in July and an average monthly low of -15.8°C in January. The region receives an average of 295mm of rainfall and 192cm of snowfall annually, with 138 days per year where precipitation exceeds 0.2mm. The Kwanika property is snow-covered from late October to May.

1.4 History

Exploration on the Kwanika property dates back to the 1930's and 40's. Copper mineralization was first recognized along Kwanika Creek in 1964 by Hogan Mines. Between 1966 and 1976, exploration was carried out that included geological, geochemical, and geophysical surveys that resulted in an aggregate of 5,700m of percussion and diamond drilling. In 1976, a Mineral Resource estimate for the main (currently referred to as the South Zone) deposit was published.

Between 1981 and 1989, different operators (Placer Developments Ltd., Aume Resources Ltd. and Daren Resources Ltd., Eastfield Resources Ltd.), conducted geochemical surveys and sampled rock outcrops, as well as IP and drilling. The claims were allowed to lapse and, in 1995, the property was re-staked by Discovery Consultants (Discovery) who conducted additional heavy mineral stream sediment and rock sampling. No more work was done until Serengeti staked the property starting in 2004.

1.5 Geological Setting and Mineralization

1.5.1 Geology

The Kwanika property lies in the northern part of the Upper Triassic to Lower Jurassic Quesnellia Terrane (Quesnel Trough) which comprises a belt of Lower Mesozoic volcanic rocks and intrusions lying between highly deformed Proterozoic and Paleozoic strata to the east and deformed Upper Paleozoic strata to the west. The Quesnel Trough is the host of numerous alkalic and calc-alkalic porphyry copper-gold deposits within British Columbia. In the area around the Kwanika property, Quesnellia is bounded by the Pinchi fault on the west and by the Manson fault on the east.

The Kwanika Project consists of two mineralized areas: Central Zone and South Zone. In the Central Zone the most economically significant intrusive body is a north-northeast trending monzonite stock that dips shallowly to steeply to the west. The intrusion has a strike length of nearly 1.3km and a thickness of 50m to 350m. The high-grade copper-gold mineralization (>0.6% copper equivalent (CuEq) in the Central Zone is dominantly hosted within, and immediately adjacent to, the monzonite intrusive. Monzonite has also been intersected at depth in the western and southwestern parts of the Central Zone and is thought to connect to the sill-like body in the central part of the deposit, suggesting the possibility of deep Central Zone mineralization.

The South Zone occurs within a fault bounded sequence of strongly altered intrusive rocks of alkalic to intermediate composition. The host lithologies occur within a north-south trending structural corridor. This structural corridor is bounded by the West Fault to the west and by a similar fault zone termed the East Fault along the eastern boundary of the corridor. Coincident chargeability and resistivity anomalies form a geophysical domain that represents the fault-bounded South Zone corridor. This variably mineralized domain is 2,900m long and up to 500m wide.

1.5.2 Mineralization

Copper and gold mineralization in the Central Zone at Kwanika occurs primarily in potassically and albitically altered lithologies. Alteration and mineralization grade outwards from a strong to intensely potassically and albitically altered, strongly mineralized core zone to a variably propylitically altered, weakly mineralized periphery. Hypogene mineralization is controlled by several generations of quartz + sulphide veining, with the highest copper and gold grades occurring in areas of quartz stockwork. A supergene enrichment blanket has been superimposed on the upper surface of the hypogene mineralization in the Central Zone.

The South Zone is characterized by porphyry style copper + gold + molybdenum + silver mineralization within monzonite, quartz monzonite, and monzodiorite with primary mineralization comprised of fine to coarse grained chalcopyrite disseminations and molybdenite mineralization along fractures and quartz selvages and, less commonly, disseminated blebs associated with pyrite and chalcopyrite.

Enrichment is associated with brecciated zones that have undergone secondary K-feldspar flooding and/or intense pyrite + chlorite + silica alteration.

1.6 Deposit Type

The Central Zone deposit is similar in characteristics to both the classic alkali porphyries in that the mineralization is associated with an intrusive complex of alkali-feldspar-saturated monzonite and the calc-alkalic porphyry type deposits in that the mineralization is associated with strong quartz stockwork.

The South Zone deposit is a structurally controlled porphyry deposit with quartz monzonitic to quartz monzodioritic host lithologies.

1.7 Exploration

In 2005, Serengeti conducted a 530 line km airborne magnetic/radiometric survey on the Kwanika and Germansen properties to assist in porphyry target identification. The airborne survey identified a small magnetic anomaly on the east side of the previously known South zone porphyry copper-gold deposit, with similar anomalies trending to the north-northwest of the deposit, as well as to the south. Six of these anomalies are associated with weak K/Th anomalies, which are often associated with porphyry copper-gold deposits.

During 2006 and 2007, Walcott Geophysics (Walcott) was engaged by Serengeti to carry out several ground-based IP surveys in the vicinity of the Central and South Zone deposits. The results outlined a significant IP signature over the Kwanika South deposit as well as a continuation of this IP anomaly into a large, covered area to the north-northwest. The following year, Serengeti carried out a regional airborne magnetic and electromagnetic (EM) survey. The results yielded by the survey identified multiple high magnetic/low resistivity anomalies throughout the property, which outline a general north-northwest trend coincident with South Zone and Central Zone deposit areas.

In 2007, selected baseline environmental studies were initiated on the Kwanika property. In 2008, pole-dipole IP surveys were conducted from south of the two known deposits to the southern boundary of the Kwanika property. Several chargeability anomalies have been identified by the IP surveys and will be the basis for further investigation of the southern section of the Kwanika property. The 2009 drilling program established an exploration model for a structurally controlled porphyry deposit in the South Zone area. Analysis and reinterpretation of geophysical and geological data suggested that potential existed for a structurally bounded domain of mineralization measuring up to 2,900m x 500m. Past exploration at Kwanika has demonstrated a strong correlation between chargeability anomalies and copper mineralization.

In August of 2016 Serengeti contracted McElhanney to fly a LIDAR survey over the Central and South zones of the Kwanika project. The resulting data was used to create a high resolution topographic surface.

Between July 2006 and August 2016, Serengeti drilled 75,412m in 180 drillholes in the resource area. Initial indications of mineralization were identified at Central Zone in hole K-06-04 during the first drill program in the summer of 2006 and the actual discovery hole K-06-09, drilled later the same year.

1.8 Sample Preparation, Analyses and Security

Serengeti implemented typical industry procedures for all aspects of the drilling, collar and down hole surveying, core description and sampling, sample preparation and assaying. Sample intervals were based on contacts between lithology, alteration, structural features and mineralogy, up to a maximum of two metres. A majority of samples taken are two metres long. Mineralized core was split on-site using two diamond saws, while select, lower grade core was split using a blade splitter.

From 2006 to 2009 all assays from the Kwanika Project were sent to Global Discovery Labs (GDL) in Vancouver, British Columbia. GDL did not have ISO accreditation but did participate in the Proficiency Testing Program for Mineral Analysis Laboratories (PTP-MAL). PTP-MAL is an ISO 9001:2000 accredited program that is operated by the Canadian Certified Reference Materials Project (CCRMP), and meets recognized international standards for proficiency testing providers. From 2010 to 2012, sampling was carried out by Acme Labs which held ISO 9001 accreditation during this time. During the 2016 drilling program, Activation Labs of Kamloops, British Columbia was used to carry out assaying of the Kwanika project. Activation Labs is ISO 17025 accredited laboratory.

1.9 Data Verification

Serengeti has conducted an independent Quality Assurance/Quality Control (QA/QC) sampling program on the Kwanika Project. QA/QC samples were included in the sample stream for both the Central and South zones. SRK has compiled and reviewed the database and the results of the QA/QC sample program. The QA/QC samples include blanks, standard reference material, and field duplicates.

SRK validated the collar, survey, and assay data for both the Central and South Zones. SRK migrated all collars in the resource areas to a more accurate elevation using a high-resolution Lidar scan. SRK visually reviewed the downhole surveys to confirm that they were reasonable. In addition, SRK compared assay database to the original assay certificates. A total of 100% of the assay values were validated and only minor errors were found.

1.10 Mineral Processing and Metallurgical Testing

Copper-Gold mineralization in Kwanika has been identified as two main zones, Central Zone, and South Zone. Serengeti conducted preliminary metallurgical testing on samples from the Central Zone. Metallurgical testing of the South Zone has not been conducted.

Exploratory metallurgical test work conducted in 2008 and 2009 demonstrates that a conventional multistage copper flotation circuit can produce a sellable copper concentrate. A copper recovery of 91% and gold recovery of 75% to a concentrate with 24% copper has been assumed for the PEA. These assumptions are preliminary and may vary with future test work.

1.11 Mineral Resource Estimates

SRK estimated copper, gold and silver resources for the Central Zone and copper, gold, silver and molybdenum resources for the South Zone. The resource block models are based on 180 drillholes, 122 located in the Central Zone and 58 located in the South Zone. The Central Zone was estimated in five domains limited to a volume defined by a 0.1% copper equivalent grade shell. The South Zone was estimated in two domains limited to a volume defined by a 0.07% copper equivalent grade shell. Copper equivalent values for the design of the grade shells were calculated from copper and gold only.

SRK is of the opinion that the block model resource estimate and resource classification reported herein represent a reasonable estimation of the global mineral resources on the Kwanika Property. The mineral resources presented herein have been estimated in conformity with generally accepted CIM *"Estimation of Mineral Resource and Mineral Reserves Best Practices"* guidelines and are reported in accordance with Canadian Securities Administrators' National Instrument 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability.

The "reasonable prospects for eventual economic extraction" requirement for a mineral resource generally implies that the quantity and grade estimates meet certain economic thresholds, and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. To demonstrate the reasonable prospect of eventual economic extraction, SRK constrained the overall mineral resource with Whittle™ pit optimization software using the parameters shown in Table 1-2. The results were used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.

The reader is cautioned that the results from the pit optimization were used solely for testing the "reasonable prospects for eventual economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves.

Table 1-3 presents the Indicated and Inferred resources in the Central Zone within the Whittle shell reported at 0.13% copper equivalent cut-off and the Indicated and Inferred resource outside of the Whittle shell that may be amenable for underground mining by block caving reported at 0.27% copper equivalent cut-off.

Table 1-**4** presents the Inferred open pit resources in the South Zone within the Whittle shell reported at 0.13% copper equivalent cut-off.

The mineral resource estimates were completed by Marek Nowak, P.Eng, an independent qualified person as defined in National Instrument 43-101. All estimation domains used were designed by Tessa Scott. In addition, Tessa Scott validated the quality of the Kwanika Central and South databases.

Table 1-2 Whittle[TM] Optimization Parameters for Resource Estimation Constraint

Input for Pit Optimization	Cu	Au	Ag	Mo
Metal price (US dollars)	$3/lb	$1300/oz	$20/oz	$9/lb
Open pit mining cost - Plant feed and Waste (Canadian dollars)	$2/t mined			
G&A costs, Processing, Water treatment and Tailings Placement (Canadian dollars)	$10/t milled			
Mining Loss	5%			
Dilution	2%			
Metal Recoveries	89%	70%	75%	60%
Overall Slope Angle (degrees)	45			

Table 1-3 Mineral Resource Statement*, Central Zone of the Kwanika Project, British Columbia, Canada, SRK Consulting, effective date October 14, 2016

Category	Quantity (x1000 Tonnes)	Grade			Contained Metal		
		Cu (%)	Au (g/t)	Ag (g/t)	Cu (000's lb)	Au (000's oz)	Ag (000's oz)
Open Pit							
Indicated	101,500	0.31	0.32	0.96	697,200	1,040	3,120
Inferred	31,900	0.17	0.14	0.59	118,500	140	610
Underground							
Indicated	29,700	0.34	0.36	1.05	222,300	350	1,010
Inferred	7,900	0.23	0.17	0.68	39,800	40	170

*Open pit mineral resources are reported in relation to a conceptual pit shell. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate. ** Open pit mineral resources are reported at a copper equivalent cut-off of 0.13% and underground resources are reported at 0.27%. The cut-offs are based on prices of US$3.00 per lb of copper, US$1,300 per ounce of gold, US$20 per ounce of silver and assumed recoveries of 89% for copper, 70% for gold and 75% for silver.

Table 1-4 Mineral Resource Statement*, South Zone of the Kwanika Project, British Columbia, Canada, SRK Consulting, effective date October 14, 2016

Category	Quantity	Grade				Contained Metal			
	(x1000 Tonnes)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu (000's lb)	Au (000's oz)	Ag (000's oz)	Mo (000's lb)
Inferred	33,300	0.26	0.08	1.64	0.01	191,400	80	1,760	7,470

*Open pit mineral resources are reported in relation to a conceptual pit shell. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate. **Open pit mineral resources are reported at a copper equivalent cut-off of 0.13%. The cut-off is based on a price of US$3.00 per lb of copper, US$1,300 per ounce of gold, US$20 per ounce of silver, US$9.00 per lb of molybdenum. The assumed recoveries are for copper 89%, gold 70%, silver 75%, and molybdenum 60%.

1.11.1 Open Pit Mine Planning

MMTS produced a series of Lerchs-Grossman (LG) pit shell optimizations for the Kwanika deposit using mining, processing, tailings, general and administrative (G&A) costs, and process metal recoveries estimated from similar studies and from information available for the Kwanika project. Indicated and Inferred Resource classes are used in the economic pit optimization for this scoping level (PEA) study. The LG pit cases selected for the Project are discussed in Appendix C.

Cut-off Grade (COG) is determined using an estimated Net Smelter Return (NSR) in $/t, which is calculated using Net Smelter Prices (NSP). The NSR (net of offsite charges and mill recovery) is used as a cut-off item for break-even economic selection of mineralized material. The NSP includes metal prices, US$ exchange rate, and off-site transportation, smelting, and refining estimates. The base case metal prices are shown in Table 1-5.

Table 1-5 Metal Prices and NSP for LGs

Metal	Market Price	Unit	NSP for LGs	Unit
Copper	$2.75	US$/lb	$ 3.23	$/lb
Gold	$1230	US$/oz	$ 48.98	$/g
Silver	$17.75	US$/oz	$ 0.65	$/g
Moly	$8.49	US$/lb	$ 5.69	$/lb

MMTS notes that the economic pit limits are based on the Indicated and Inferred Resource classes as well as estimated mining unit costs at a PEA level of study. These costs are derived to meet the local condition for the Project and the specific project arrangements for mine rock management, water management, environmental, and reclamation to the PEA level of study, as well as certain input parameters, such as pit slope angles, process recoveries, environmental considerations, and reclamation requirements. All of these components affect the mining quantities and activities to release the specified mineralization and, as such, affect the economic pit limits. Some of the elements of this study require more sampling and evaluation for future pre-feasibility study (PFS) level work and changes to the general design concepts can impact the mine plan. Changes to these design elements and parameters will not only affect the cost estimates within the plan, but will also impact the economic mining limit in future studies.

1.11.2 Underground Mine Planning

There are sufficient tonnes and grade below the Central Open Pit to support an underground mine. Several different mining methods have been evaluated including block caving. AMEC has reviewed the drill core and the targeted higher grade mineralized zone and considers Block Caving to be a viable mining method. Accordingly, MMTS developed Block Cave stopes and infrastructure based on typical parameters which include caveability, requirements for rock pre-conditioning, geotechnical aspects of the rock, development rates and mining loss and dilution factors. These parameters will require refinement in future studies.

The underground designs include four stope outlines and access development ramping down from surface with the portal adjacent to the mill. Additional conceptual designs address ventilation, development, and undercut, drawbell, cross-cut and extraction level design for block cave mining. At this stage of planning, the stope outlines are conservative and planning includes an estimate of equipment and rate of development from general factors. Optimization will be needed in future designs to maximize the underground extraction and optimize the ROM head grades from the underground operations. The summarized underground mill feed for production scheduling is listed in Table 1-7.

1.12 Mineral Reserve Estimates

The current study is at a PEA level and therefore there are currently no Mineral Reserves estimated for the Kwanika Project.

1.13 Mining Methods

A production schedule based on a 15,000t/d mill feed rate at a preliminary assessment level has been developed for the Kwanika Project. The pit phases are engineered based on the results of an updated economic pit limit analysis. The underground stopes are engineered based on the results of a cut-off grade analysis. A summary of Indicated and Inferred open pit mill feed for production scheduling is provided in Table 1-6 using whole block grades with mining dilution and loss varying by extraction method (open pit versus underground).

The copper (Cu), gold (Au), and silver (Ag) grade items used in this study are based on the resource model provided by SRK as described in Section 14.

Table 1-6 Summarized Indicated and Inferred Pit Resource for Production Scheduling

Pit Phase	*Class*	*ktonnes*	*NSR $/t*	*Cu%*	*Au g/t*	*Ag g/t*
Central	IND	11,752	$37.75	0.372	0.387	1.076
	INF	208	$24.13	0.278	0.170	0.785
South	INF	24,819	$21.27	0.265	0.076	1.630

Note: Open Pit NSR cut-off used is $11.30/tonne with a provision for mining loss of 5% and dilution of 2%.

Table 1-7 Summarized Indicated and Inferred Underground Mill Feed for Production Scheduling

UG Phase	*Class*	*ktonnes*	*NSR $/t*	*Cu%*	*Au g/t*	*Ag g/t*
All Central	IND	41,410	$47.85	0.455	0.522	1.364
	INF	666	$23.56	0.271	0.168	0.720

Note: To account for mining loss and dilution all material within the stope shapes are included with no loss or dilution applied. The stope shapes are considered as the boundary of all the excavated material sent to the mill.
Note: The Open pit and Underground Mineral resources stated in Table 1-6 and Table 1-7 are preliminary in nature, and include 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 in future studies. There is no certainty that the results of the preliminary economic assessment will be realized.

1.13.1 Mining Operations

Mining at Kwanika occurs as an UG block-cave and an Open Pit operation. There are some years where UG and OP mining is operating concurrently. The Project construction phase begins with UG decline development in Year-2. Site infrastructure and process plant construction begin in Year-2. Open Pit pre-stripping begins in Year-1 in Central Pit and the first ore to the plant comes from Central Pit. The first ore from the block cave is also delivered in Year 1.

Both the UG and OP operations are designed as contractor-operated in this PEA. Contractor fleets require minimal capital for start-up. There are increased operating costs incurred that pay for the contractor fleet, margin, overhead, and fleet depreciation. Market conditions for project financing play a factor in deciding whether the operation is owner or contractor operated. Future studies will further test the economics of contractor versus owner operated open pit and underground mining.

The above mining resources (Table 1-6 and Table 1-7) are used to produce a consolidated production schedule. A summary of the production schedule is provided in Table 1-8.

Mining operations, methods, and equipment will be typical of open-pit and underground mining in Northern British Columbia. The Project will be a medium-capacity operation that utilizes appropriately sized equipment for the major operating areas in order to generate high productivities, and reduce unit and overall mining costs.

The mine plan and production schedule will undergo further refinement during the Pre-Feasibility Study. Additional information on underground footprint optimization should more accurately determine the underground stope geometry, projected caving rates, and optimal sequencing of the drawbells3. Further details on rock storage management, water management, and final land use will be developed for the Environmental Assessment application, the result of which may impact the mine plan. These elements, along with other optimization details, will be integrated into pre-feasibility stage mine planning.

Serengeti Resources Inc.
Kwanika Project

Table 1-8 Life of Mine Production Summary

Open Pit Production		Totals	MSSP Period / YEAR / Units		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
			YEAR	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
		Totals	Units																	
Total UG + OP Mill Feed		**78,855**	**kTonnes**		**-**	**5,401**	**5,400**	**5,400**	**5,400**	**5,400**	**5,400**	**5,400**	**5,400**	**5,401**	**5,401**	**5,401**	**5,401**	**5,401**	**5,401**	**3,248**
	Cu	0.381	%		-	0.558	0.529	0.434	0.441	0.441	0.441	0.440	0.441	0.408	0.283	0.307	0.268	0.259	0.209	0.173
	Au	0.357	g/tonne		-	0.599	0.577	0.510	0.526	0.525	0.526	0.522	0.526	0.376	0.130	0.059	0.080	0.097	0.106	0.095
	Ag	1.398	g/tonne		-	1.536	1.502	1.330	1.354	1.353	1.354	1.349	1.354	1.171	1.437	1.732	1.561	1.615	1.233	0.893
Mining Schedule by Phase																				
C621 Waste					**9,005**	**3,287**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
C621 Direct Mill Feed		**3,506**	**kTonnes**		**-**	**3,506**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Cu	**0.54**	**%**			0.545														
	Au	**0.57**	**g/tonne**			0.565														
	Ag	**1.48**	**g/tonne**			1.476														
C622i Waste					**4,235**	**5,471**	**1,441**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
C622i Direct Mill Feed		**1,507**	**kTonnes**		**-**	**156**	**1,350**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Cu	**0.45**	**%**			0.274	0.466													
	Au	**0.53**	**g/tonne**			0.495	0.536													
	Ag	**1.42**	**g/tonne**			1.418	1.425													
UG Development Mill Feed		**90**	**ktonnes**		**-**	**37**	**-**	**31**	**-**	**22**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Cu	0.456	**%**			0.515		0.441		0.377										
	Au	0.469	**g/tonne**			0.565		0.526		0.226										
	Ag	1.308	**g/tonne**			1.467		1.354		0.978										
UG Level Development Mill Feed		**730**	**ktonnes**		**-**	**135**	**140**	**125**	**105**	**33**	**114**	**78**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
	Cu	0.470	**%**			0.597	0.485	0.435	0.441	0.441	0.440	0.377								
	Au	0.512	**g/tonne**			0.622	0.545	0.512	0.526	0.526	0.520	0.226								
	Ag	1.368	**g/tonne**			1.606	1.416	1.333	1.354	1.354	1.346	0.978								
UG 470 Cave Production Mill Feed		**41,255**	**ktonnes**			**1,313**	**3,910**	**5,244**	**5,295**	**5,345**	**5,286**	**5,322**	**5,400**	**4,141**	**-**	**-**	**-**	**-**	**-**	**-**
	Cu	0.452	**%**			0.597	0.553	0.434	0.441	0.441	0.441	0.441	0.441	0.405						
	Au	0.516	**g/tonne**			0.622	0.592	0.510	0.526	0.526	0.526	0.526	0.526	0.354						
	Ag	1.354	**g/tonne**			1.606	1.532	1.329	1.354	1.354	1.354	1.354	1.354	1.139						
S621 Waste			**kTonnes**	**-**	**-**	**-**	**-**	**13**	**-**	**-**	**-**	**-**	**-**	**7,592**	**6,452**	**181**	**-**	**-**	**-**	**-**
S621 Direct Mill Feed		**5,777**	**kTonnes**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**2,949**	**2,828**	**-**	**-**	**-**	**-**
	Cu	**0.32**	**%**												0.301	0.343				
	Au	**0.05**	**g/tonne**												0.053	0.044				
	Ag	**2.00**	**g/tonne**												2.009	1.989				
S622 Waste					**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**2,484**	**7,093**	**1,124**	**92**	**-**	**-**
S622 Direct Mill Feed		**9,544**	**kTonnes**		**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**2,573**	**5,393**	**1,579**	**-**	**-**
	Cu	**0.26**	**%**													0.267	0.268	0.251		
	Au	**0.09**	**g/tonne**													0.075	0.080	0.120		
	Ag	**1.55**	**g/tonne**													1.450	1.561	1.704		
S623 Waste					**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**8,046**	**5,363**	**148**	**-**
S622 Direct Mill Feed		**5,272**	**kTonnes**		**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**8**	**3,795**	**1,469**	**-**
	Cu	**0.27**	**%**														0.230	0.263	0.302	
	Au	**0.10**	**g/tonne**														0.044	0.088	0.132	
	Ag	**1.74**	**g/tonne**														1.219	1.584	2.161	
Stk1-3	**Stockpile Mined**	**11,174**	**kTonnes**		**382**	**3,131**	**3,434**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**393**	**1,353**	**961**	**1,343**	**176**	**-**
	Stockpile Reclaimed	**11,174**	**kTonnes**		**-**	**253**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**1,260**	**2,452**	**-**	**-**	**28**	**3,932**	**3,248**
	NSR	**$18.46**	**$/Tonne**			$79.85								$12.26	$24.85			$15.74	$15.26	$15.16
	Cu	**0.232**	**%**			0.690								0.418	0.261			0.179	0.175	0.173
	Au	**0.184**	**g/tonne**			1.006								0.446	0.224			0.103	0.096	0.095
	Ag	**0.928**	**g/tonne**			2.048								1.278	0.748			0.851	0.886	0.893

1.14 Recovery Methods

A conventional copper-gold flotation process is assumed for the Kwanika project including crushing, grinding, and multi-stage froth flotation to produce a copper-gold concentrate.

1.15 Project Infrastructure

The Project site will be accessible by road from Fort St. James via the all-weather Leo Creek and Driftwood forest service roads (FSR) and the 30km long Tsayta Lake Road. Access to the Kemess Power Line is available 75km from the Project site.

The General Arrangement is shown in Figure 1-1.

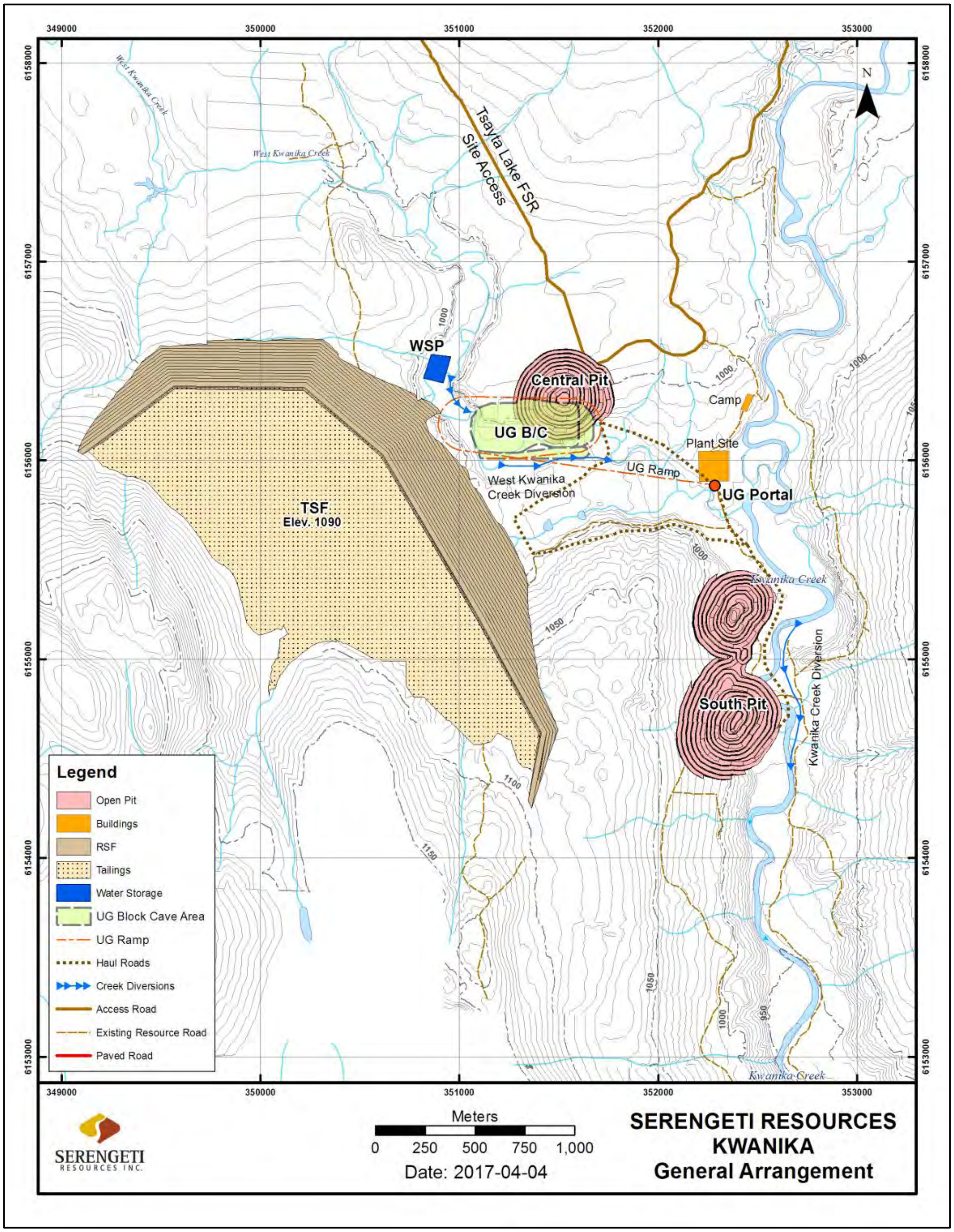


Figure 1-1 General Arrangement

The Project site will have open pit and underground mining related facilities, process related facilities, and a permanent camp. The general site layout is shown in Figure 1-1. On-site infrastructure includes:

- Power and energy supply (external power supply transmission line)
- Electrical substation
- Tailing Storage Facility (TSF)
- Water Storage Pond (WSP)
- Water Treatment Facility (WTF) associated with WSP
- Water Storage Pond/Water Treatment Facility (WSP/WTF)
- Fuel storage and dispensing (diesel)
- Maintenance and truck shop (supplied by open pit mining contractor)
- Administration/dry building
- Assay laboratory
- Cold storage
- Access roads
- Water supply
- Wastewater treatment systems
- Solid waste disposal facilities and sewage plant
- Communication systems
- Medical facilities
- Site support systems including workshops, maintenance shop, warehousing and security
- Permanent camp facility
- UG Portal
- UG Substation and electrical distribution systems
- UG air, compressors, and ventilation systems
- UG mine refuge stations
- UG trolley-assist system

Future studies will determine locations, sizes, and other specifications beyond the scope of this preliminary assessment.

1.15.1 On-Site Roads

On-site service roads will be constructed connecting to the main access road, the explosives storage and manufacturing facilities, tailings storage facility, processing plant, rock storage facility, and open pits.

1.15.2 Power Supply and Distribution

The selected power supply option for the Project includes extending a connection from the Kemess Power Line 75km to the Kwanika Project site. Site power will be distributed to various modular electrical rooms on site by means of an overhead line to the following areas:

- primary crushing
- tailings/water management
- explosive manufacturing
- permanent camp
- Maintenance/truck shop
- Central and South Pits
- U/G facilities

1.15.3 Mine Rock Storage Facilities (RSF)

Mined rock below cut-off grade is directly hauled from the pit and placed and compacted at the TSF Starter Dam. After the Starter Dam has been constructed, the waste rock will be end-dumped in suitable lifts building up and buttressing the downstream face of the TSF. Seepage and runoff from this facility will be collected and managed within the Water Storage Pond.

Allowances are made to address reclamation and post-closure requirements.

1.15.4 Tailings Storage Facility (TSF)

The flotation tailings will be pumped to the TSF to the south of the processing plant and southeast of the Central Pit area. The Starter Dam will consist of mine rock, and will have a core of low permeability till material. The Starter Dam will be constructed during pre-production mining in Year-2 and Year1. The Starter Dam will hold six months of tailing production and will be raised in advance of mill start-up and the production of the first concentrate. The Starter Dam will be approximately 25m high (including a freeboard of 5m) and contain approximately 11Mt of tailings.

The Ultimate Dam will be constructed by successive lifts of cyclone tailing sand, during production and will have capacity sufficient for containing approximately 80Mt of mill tailings. The Ultimate Dam will be approximately 80m high with a 5m freeboard.

The supernatant water that will accumulate within the TSF will be reclaimed by pumps mounted on a floating barge and pumped to the Water Storage Pond (WSP), and thereafter pumped back to the process facility. A seepage collection system to collect dam seepage will be included in the design. The TSF will include the ability to discharge to the environment when required, and, when water quality allows. A water treatment system will be utilized prior to discharge if water quality is not suitable.

1.15.5 Mine Area Water Management

Currently, West Kwanika Creek flows through the proposed areas of the TSF and Central Pit. A diversion is required below the WSP to route water away from the TSF, and then again to route water south of Central Pit. The main flow of Kwanika Creek requires a diversion east of the South Pit later in the mine life.

Fresh water diversion (e.g. around the TSF) will be used where possible to keep 'non-contact water' from entering active mining area. For contact water, it will be necessary to construct a Water Storage Pond (WSP) at the north end of the TSF to collect surface runoff from catchment areas and seepage water from the flotation tailings during mining operations and the initial stages of closure and reclamation. Water from the active open pit areas and from underground pumping will also be directed to a water storage pond for re-use or eventually treated and released to the environment. Details will be addressed as part of the overall site water balance and site water management plan in future studies.

1.15.6 Mine Area Closure Plan

At the cessation of mining operations, a closure plan will be implemented to return the operating area to a condition that will meet the end land use objectives.

The flotation tailings will be capped and the outer slopes of the RSF will be re-sloped to blend with the natural landscape and to enable access for wildlife. Natural seepage water collected within the water storage pond will be pumped and discharged to the open pit until the water quality meets discharge criteria.

The open pits will be allowed to fill through seepage and surface run-off. Stream run-off may be directed into the completed mining areas (open pit and Underground) to reduce the Acid Rock Drainage (ARD) and metal leaching potential as quickly as possible. The Kwanika Creek diversion channels will either continue to operate or will be decommissioned, as required by future testing work.

1.16 Market Studies and Contracts

Metal prices are reflective of industry consensus pricing and agreed to by MMTS and Serengeti at the effective date of this report with alternative prices for financial modeling at March 1 2017 spot prices and another case at 10% above the base case prices. No comprehensive market studies or contract agreements have been completed.

Table 1-9 Metal Prices

Metal	Metal Price (US$/unit)
Copper	US$2.90 / lb
Gold	US$1270 / oz
Silver	US$19.00 / oz

Concentrates will be sold into the general market. This will either be to North American, European, or Asian smelters and refineries.

1.17 Environmental Studies, Permitting and Social or Community Impact

1.17.1 Regulatory Framework

The Serengeti Kwanika Project falls within the category of a "reviewable project" of the British Columbia Environmental Assessment Act (BCEAA) and with proposed diversions of Kwanika Creek, a fish-bearing stream, will require a Federal Fisheries Act approval, and will trigger the requirements of the Canadian Environmental Assessment Act (CEAA). Other requirements of Provincial and Federal Acts and Regulations may also apply, depending upon final design components.

Future work will be required to conduct Environmental Assessment studies and to apply for an Environmental Assessment Certificate.

Consideration will be made in future engineering to mitigate issues that could trigger more restrictive regulatory requirements and improve Project economics.

1.17.2 Regional Land Use Processes

The Project is located within lands that have been dedicated in the Fort St. James Land and Resource Management Plan, approved by government in 1999. The Project area is within the Multi-Value Resource Management Zone Land Use designation, where lands are managed to integrate a wide range of resource values, including mining. A more detailed review of the present status of land designation and any additional and specific local land use plans is required.

1.17.3 Programs Already in Progress

In support of the exploration programs, Serengeti has been in consultation with the local Takla Lake First Nations (TLFN), providing jobs as well as starting base line environmental, archeological, weather and water studies including a project specific Valued Ecosystem Component (VEC) study. The positive relations to date, as well early baseline data for local environmental and water quality will be a benefit to a future PFS study and the Environmental Assessment process.

1.18 Capital and Operating Costs

1.18.1 Capital Costs

All currencies in this section are expressed in Canadian dollars. Costs in this report have been converted using a fixed currency exchange rate of US$0.77 to $1.00. The expected accuracy range of the capital cost estimate is +/- 40%.

Initial capital has been designated as all capital expenditures required producing copper concentrates for shipment to contract smelters. Sustaining capital includes underground mining equipment and infrastructure and mine closure and reclamation. A summary of the major capital costs is shown in Table 1-10 and Table 1-11.

This PEA estimate is prepared with a base date of Q1 2017 and does not include any escalation past this date.

Table 1-10 Initial Capital Cost Summary

Direct Costs	Initial Capital Cost ($X1000)
Overall Site	$15,700
Open Pit Mining – Pre- Production	$28,600
Open Pit Mining - Equipment	$2,500
Underground Mining - Development	$39,500
Underground Mining – Direct Level Development	$4,800
Underground Mining – Equipment and Infrastructure	$35,100
Processing Plant (including Ore Handling)	$120,000
Tailing Storage Facility	$35,000
Water Management	$23,000
On-Site Infrastructure	$38,300
Off-Site Infrastructure	$18,800
Sub-Total Direct Costs	**$361,300**
Indirect Costs	

Direct Costs	Initial Capital Cost ($X1000)
Project Indirects	$41,000
Owner's Costs	$13,000
Contingencies	$61,000
Sub-Total Indirect Costs	$115,000
Total Initial Capital Cost	**$476,300**

Table 1-11 Sustaining Capital Cost Summary

Sustaining Capital Cost Description	Capital Cost ($X1000)
Open Pit Mining – Sustaining	-
Underground Mining – Equipment and Infrastructure	$36,600
Closure	$46,300
Total Sustaining and Closure Capital Cost	**$82,900**

1.18.2 Operating Costs

The operating costs for the Project are shown in Table 1-12. The cost estimates in this section are based upon budget prices in Q1 2017 or based on the information from similar projects and MMTS' database. The expected accuracy range of the operating cost estimate is +/- 35%.

Table 1-12 Operating Cost Summary

	$/tonne Mined	$/tonne Milled
Mine (OP)*	$2.97	$7.98
Mine (UG)**	$11.73	$11.73
UG Ore Rehandle		$1.09
Mining Total* **		**$10.20**
Mill		**$9.00**
G & A******		**$1.95**
Tailing Treatment		Included in G&A
Water Treatment		Included in G&A
Total		**$21.15**

*Note: OP mining cost, divided by total OP tonnes mined / milled.

**Note: UG mining cost divided by total UG tonnes mined / milled.

***Note: Total mining cost (OP + UG) divided by total tonnes milled.

****Note: G&A Costs are applied at $2.02/tonne milled until mining ramps down in Year 14 and Year 15.

The operating costs are defined as the direct operating costs including mining, processing, tailings handling, water treatment, and G&A.

1.19 Economic Analysis

A Base Case economic evaluation has been undertaken incorporating consensus metal prices as of the effective date of this report. This approach is consistent with the guidance of the United States Securities and Exchange Commission, adheres to National Instrument 43-101 and is consistent with industry practice. The metal production values indicated in Table 1-13 are a summary of the results of the production schedule, which is used in the cash flow to determine projected revenues. The pre-tax economic results in Canadian dollars are listed in Table 1-14.

Table 1-13 Metal Production from Kwanika

	Years 1 to 5	LOM
Total Tonnes to Mill (000s)	27,001	78,855
Annual Tonnes to Mill (000s)	5,400	5,400
Average Grade		
Copper (%)	0.480	0.381
Gold (g/t)	0.547	0.357
Silver (g/t)	1.415	1.398
Total Production (after Recovery)		
Copper (000s lb)	260,260	600,635
Gold (000s oz)	356	676
Silver (000s oz)	921	2,659
Average Annual Production		
Copper (000s lb)	52,000	40,000
Gold (000s oz)	71	45
Silver (000s oz)	184	177

Table 1-14 Summary of the Economic Evaluation

	Unit	Base Case
Metal Price		
Copper	US$/lb	$2.90
Gold	US$/oz	$1270.00
Silver	US$/oz	$19.00
Exchange Rate	US$:CAD$	$0.77
Economic Results (Pre-Tax)		
Undiscounted cash flow	$ M	$710
NPV (at 5%)	$ M	$411
NPV (at 7%)	$ M	$324
NPV (at 8%)	$ M	$287
NPV (at 10%)	$ M	$220
IRR	%	21.1
Payback	years	3.73

1.19.1 Sensitivity Analysis

Sensitivity analyses have been carried out on the following parameters:

- Copper and gold metal price
- Exchange rate
- Initial capital expenditure
- On-site operating costs

The analyses are presented graphically as financial outcomes in terms of NPV and IRR. Both the Project NPV and IRR are most sensitive to copper price and exchange rate. The NPV and IRR sensitivities can be seen in Figure 1-2 and Figure 1-3. These results are presented graphically only to show trends for future evaluation. At a scoping level of engineering and costing the absolute values are not deemed relevant for economic evaluation.

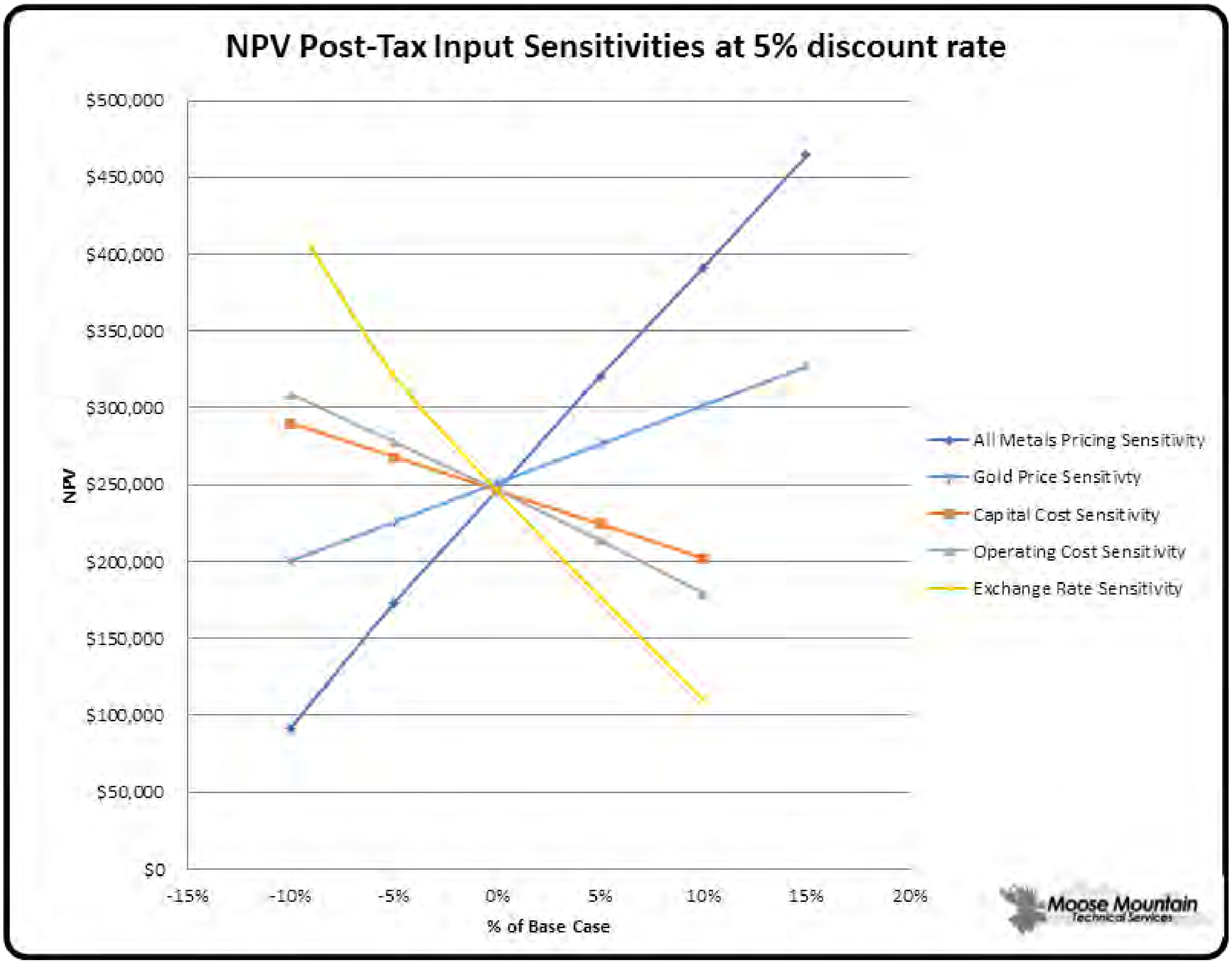


Figure 1-2 **Base Case Sensitivity to After-Tax NPV @ 5%**

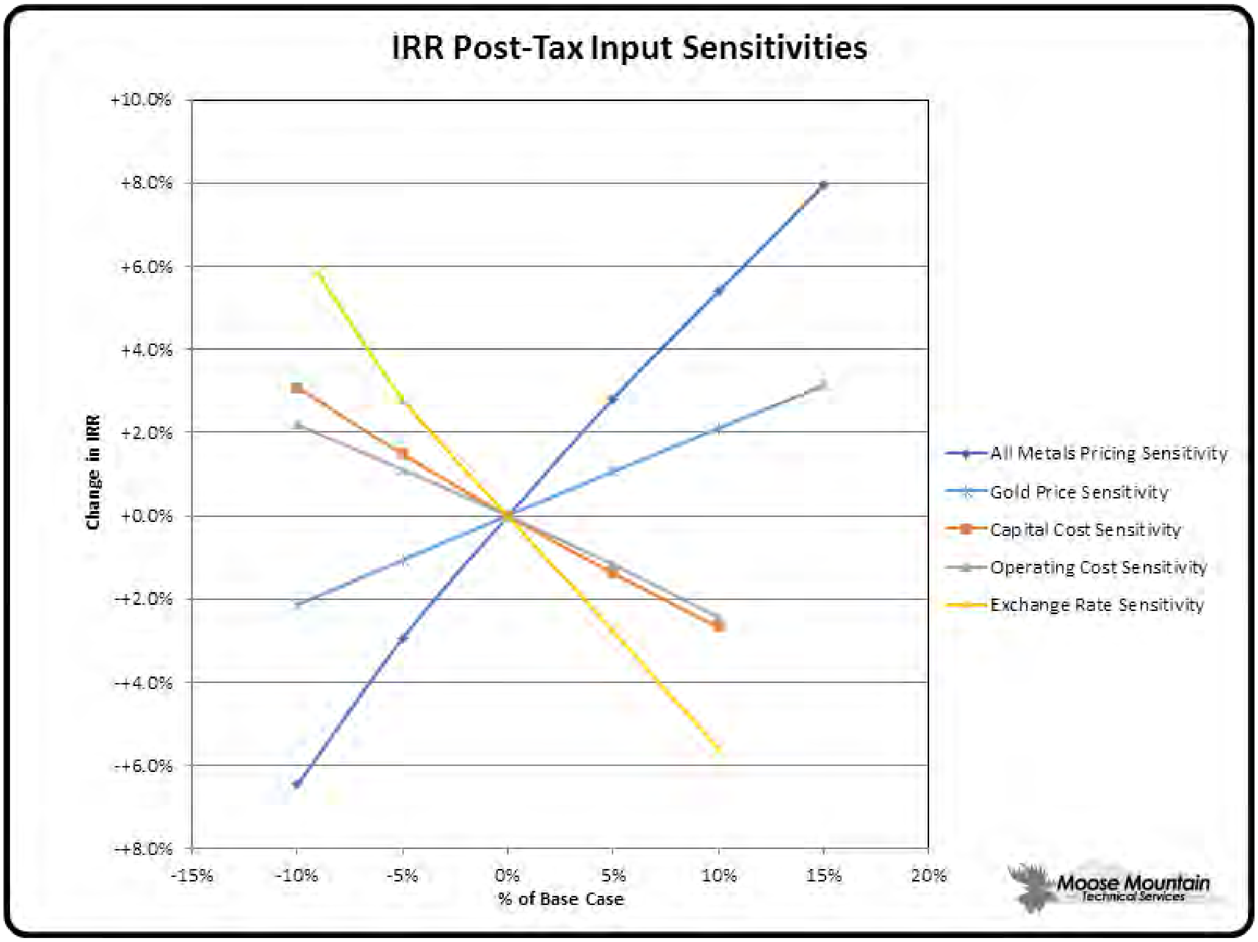


Figure 1-3 **Base Case Sensitivity to After-Tax IRR**

Note - The Mineral resources and economic results stated in this Preliminary Economic Assessment are preliminary in nature, and include 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 results of the preliminary economic assessment will be realized.

1.20 Adjacent Properties

1.20.1 Regional

The Quesnel Trough is the host to several other porphyry copper ± gold mines and significant deposits. These deposits include: the Mount Polley Mine, the former Kemess Mine and its related infrastructure located north of Kwanika, and the Mount Milligan Mine located approximately 85km south of Kwanika.

1.20.2 Local District

The adjacent Lustdust claims, owned by Alpha Gold Corporation, are located immediately to the north of the Kwanika property. The Lustdust property has been the subject of exploration for more than fifteen years on various precious and base metal vein and skarn occurrences and contains a small Indicated and Inferred copper-gold Mineral Resource known as the Canyon Creek Zone. The other significant prospect in the general vicinity of Kwanika is the Lorraine porphyry copper-gold property jointly controlled by Teck Corporation and Lorraine Copper Corp. which contains a modest, Indicated and Inferred Mineral Resource in two deposits. Lorraine Copper Corp. acquired the Lustdust property in June 2016.

1.21 Interpretation and Conclusions

The Kwanika deposit represents a copper-gold-silver deposit that is amenable to open pit and underground block caving and conventional milling consisting of flotation concentration.

1.21.1 Geology and Resource Modeling

SRK considers that the mineral resources for the Kwanika Project are appropriately reported. The Central Zone is reported at 0.13% copper equivalent cut-off grade for near surface mineralization and 0.27% copper equivalent cut-off grade for potential underground mining by block caving method. The South Zone is reported at 0.13% copper equivalent for open pit resources.

The volume of the supergene zone and related recoveries and the influence of faulting and barren dykes are two major factors that may affect the quality and quantity of the current estimates, and thereby are opportunities for improvement.

Both of these risks could be greatly reduced by re-logging drillholes and addition of several oriented core drillholes.

SRK is not aware of any potential significant risks and uncertainties that could affect the reliability or confidence on the reported resource.

1.21.2 Metallurgy

Limited metallurgical test work carried out on the Central Zone deposit indicates mineralization responds well to a process consisting of conventional multi-stage flotation and a typical process design for a copper porphyry in British Columbia is in order. A copper recovery of 91%, with gold and silver recovery of 75% has been estimated to a concentrate grading 24% copper. Processing operating costs are bench marked to similar mills in the area.

No metallurgical test work has been conducted on the south deposit so typical recoveries have been assumed.

A mill throughput of 15,000 tonnes per day is proposed for the Project.

1.21.3 Mining

The Kwanika Central Zone at depth is deemed to be amenable to the low-cost block caving mining method. The current work indicates that the revised Block Cave design adds significant positive contribution to the financial results of the Project both due to higher mill feed grades and lower mining costs. More evaluation is required in more advanced levels of study to assess the geotechnical characteristics for the proposed Block Cave and an optimized Block Cave mine plan is needed.

The Central and South pits have been designed to provide low cost mining for the near surface material. A trade-off between the depth of open pit mining and extending the Block Cave stope upwards indicates an economic advantage to increasing the underground limits upwards rather than mining more from the open pit. The results of the trade-off will need to be confirmed in future studies as the resource model and geotechnical test results become available.

The South Zone pit phases are marginally economic in the base case and future resource modeling, with more drilling may extend the economic mining limits and add some future mineable resources and reserves that are currently deemed sub-economic.

Extensive use of mining contractors in this study is due to the need to alternate between open pit and underground mining with specific equipment and expertise required for relatively short periods of time. Costs reflect minimal equipment purchase and contractor mark-ups on operating costs.

The geotechnical design of the TSF has not been completed at this level of study but the use of the open pit rock as a buttress will improve the design safety of the TSF which will help meet future permitting requirements.

1.21.4 Infrastructure

Regional Advantage – Access and Power

The project site has logistical advantage over many greenfield projects due to its location. The local area is a mining district with established services and supplies. Local roads can be upgraded and year-round truck access is available for construction, operating supplies, and personnel as required. The road network connects into existing rail lines and ports.

The regional power grid is only 75km away which will mean a low capital cost to connect into a low cost supply of energy. This provides a distinct advantage over remote sites dependent on onsite power generation. Maximizing the use of electrical equipment will reduce energy costs, and will also gain significant carbon credits with benefits through the permitting process.

This location advantage reduces the capital and overhead costs that relate to remote sites for access and power.

Site Services

The relatively short mine life and use of mining contractors' favours the use of modular buildings where possible. Some facilities will be permanent but using predominately modular construction with will allow lower building costs. Contractors with short term needs can supply their own temporary shops and offices.

On-sight accommodations will be required. The site is close enough to regional facilities so that medical emergencies can be evacuated so an onsite first aid facility will suffice, and staff doctors and nurses will not be required.

There is sufficient water onsite to meet industrial and personnel requirements. Water management will handle water storage, diversion, and treatment before discharge.

Tailings and Water Management

The recently revised BC Regulations require a higher factor of safety and use of Best Available Technology (BAT) and Best Available Practice (BAP) in TSF design. A geotechnical study has not been undertaken for this study but the design features included in this TSF design will provide the BAT/BAP provision to meet the revised regulations.

1.21.1 Regulatory, Environment, and Permitting

This scoping level project design, considering the knowledge of land use expectations, and the regulatory process in British Columbia, is a normal mining operation in this jurisdiction.

The Project lies within an area designated for multiple land uses, including mining. Provincial and Federal Environmental Assessments and Certificates will be required due to the nature and scope of the

Project. The project will need to demonstrate the ability to manage for ARD concerns during and following mining. Significant environmental issues such as fish stream diversions; ARD potential and wildlife habitat are expected to be manageable. Reclamation of all site disturbances is expected to be completed within industry norms.

At this time, MMTS is not aware of any constraints in this regard that may prevent the Mineral Resources at Kwanika from being exploited.

1.21.2 Economic and Financial

The financial results from the base case plan used in this report are based on preliminary parameters and assumptions. There is financial sensitivity to these which has been indicated. As noted, there are areas in the mine plan that needs additional evaluation that could increase or decrease the economic mining limits. To advance the Project, a PFS is required. This will require significant site investigation work and testing, including exploration drilling, geotechnical analysis and testing, metallurgical sampling and testing, and base line environmental studies. This new field data will provide PFS designs that may improve the economic certainty of the estimate. It also provides the mine plan required for the Project Description in the permitting process.

1.21.3 Opportunities

- This PEA study provides a scoping level basis for a viable operation with the opportunity to add more economic resources both on site and in the local area.
- There are other properties in the local area that have the potential of using the Kwanika facilities on a contract or joint venture basis.
- An expanded resource in the local area could use the facilities and infrastructure from this study.

1.22 Recommendations

It is recommended to advance the Project to a higher level of study to continue towards an eventual production decision. The recommended studies will include field investigations to:

- Gather environmental background information
- Waste rock characterization for ARD and metal leaching
- Surface and groundwater observations and monitoring
- Geotechnical drilling and sampling
- Infill and step-out exploration drilling,

This information will be needed to advance the engineering and environmental studies to a PFS stage that can be used for the Project Description required in the permitting process. Future studies will progress the Project to a Feasibility Study level and will complete the permitting process.

1.23 Exploration Drilling

A two-phase drill program is proposed to upgrade the current Central and South zone resources enabling the Project to move to a PFS stage. This will include a 13,200m drill program on the Central and South zones.

The first phase is proposed to upgrade Central Zone Inferred resource to Indicated, to test for a deep mineralized system in Central Zone, and to provide metallurgical and geotechnical information for future studies.

The second phase will be focused on South Zone to upgrade the Inferred resources and to test for increased resources below the current South pit designs.

1.24 Updated Resource Models

With the proposed drill programs the resource models will need to be updated. The Central Zone model will be updated with the Phase 1 drilling program, the South zone with the Phase 2 program.

1.25 Underground Block Cave Mining

The following studies are required for UG mining to advance to a PFS level:

- Undertake a geotechnical assessment of the proposed Central zone Block Cave to confirm cavebility, fragmentation and caving rate for the mine design.
- Optimize the footprint of a Block Cave design and the sequence of the draw point schedule
- Redesign the rest of the underground facilities to a PFS level

1.26 Open Pit Mining

The following recommendations for OP mining to advance to a PFS level:

- Undertake a geotechnical investigation for pit slope design.
- Re-design the Central and South Zone pits phases based on the updated resource model and other updated information

1.27 Metallurgical Testing

Significant metallurgical test work is required to provide suitable representative information for the Project. This will include testing on drill core samples from fresh core.

Additional metallurgical evaluation will need to be conducted to test the variability of the deposit and a metallurgical assessment program is required for the South Zone. An initial program for Central and South Zones will be associated with PFS level planning. Further test work will be warranted to support a future Feasibility study.

1.28 Infrastructure

General Site Studies

Additional planning and geotechnical studies for the surface facilities and structure on the site will be required to a PFS level for the Project Description in the permit application. This will include site layout optimization, geotechnical investigation for foundations, sources of fill and construction materials, and water management facilities.

TSF and Water Management

The TSF geotechnical and water management investigations need to consider the requirements of the BC regulations, alternatives assessments, ARD/Metal Leaching issues, and a site wide water balance.

1.29 Environmental Assessment

To advance the Project, the following actions are recommended for regulatory and permitting work. The timelines for environmental baseline studies and requisite permitting can be varied, and as such, nearly all items related to environmental studies are on the Project Execution Plan critical path.

- Required environmental baseline studies, many of which require two years of data
- Support for the Project from local communities and First Nations should be solicited and participation encouraged.

- Detailed requirements for land use, Environmental Assessment processes, and detailed fisheries, wildlife, ARD and other issues should begin immediately to provide guidance to mine planning and capital cost requirements.

1.30 Cost of Recommended Work up to PFS*

To advance the Project to PFS level the following approximate costs will be incurred:

Table 1-15 Recommendations and Future Study Costs

Recommendations and Future Study Costs:		
Exploration Drilling	Phase 1	$2,000,000
Exploration Drilling	Phase 2	$1,000,000
Geology and Resource Model Updates	PH1 and PH2	$120,000
OP Mining Prefeasibility Study	Geotech	$150,000
OP Mining Prefeasibility Study	Pit Designs	$80,000
UG Mining Prefeasibility Study	Geotech	$180,000
UG Mining Prefeasibility Study	Block Cave Footprint Finder	$25,000
UG Mining Prefeasibility Study	Stope and Development Design	$80,000
Metallurgical Testwork Program		$350,000
General Site Infrastructure		$250,000
TSF and Water/Waste Management		$250,000
Environmental and Permitting*	Baseline Studies	$2,250,000
Environmental and Permitting*	Regulatory Coordination and Report	$250,000
Total		**$6,985,000**
*This includes permitting work up to Prefeasibility/Project Description. An additional $2,500,000 is estimated to complete the permitting process.		

2 Introduction

The Serengeti Resources Inc. Kwanika Project (the Project) involves the development of a copper-gold deposit located near Fort St. James, British Columbia, Canada.

This National Instrument 43-101 (NI 43-101) compliant report on the Project has been prepared by Moose Mountain Technical Services (MMTS) and is based on work produced by the following independent consultants:

- SRK Consulting (Canada) Inc.
- Moose Mountain Technical Services

Mr. Chad Yuhasz (PGeo) of SRK Consulting visited the Kwanika Project on August 7th to 9th 2016. He is the QP for all matters relating to geology and resource estimation in this report.

Mr. Marek Nowak, (P.Eng.) of SRK Consulting is the QP for all matters relating to geology and resource estimation in this report.

Mr. Jim Gray (P.Eng.) of MMTS visited the Project site on October 18, 2011. He is the QP for all matters relating to infrastructure, mining, mining capital costs, mine operating costs, financial evaluation, and overall report preparation.

Mr. Tracey Meintjes (P.Eng.) of MMTS is the QP for matters relating to mineral processing, mineral processing capital, mineral processing operating costs, and metallurgical testing.

The Preliminary Economic Assessment (PEA) is based on exploration and internal Serengeti studies since 2005. The resource model used is most recently described in an NI 43-101 Technical Report dated December 2016 which is included in this report (Kwanika_NI43101_2CS054.000_20170103_ FINAL.docx). The effective date of the Resource model is October 14, 2016.

This PEA with an effective date of 3 April 2017, adds a mine plan and infrastructure bringing it to a scoping level of accuracy. The mine plan includes an integrated open pit and underground production schedule using typical operating parameters. A conceptual evaluation by AMEC Americas Ltd. (AMEC) has indicated that Block Caving is a suitable underground mining method as applied to this study, see Appendix B.

3 Reliance on Other Experts

AMEC Americas Ltd., of Vancouver, British Columbia reported on matters pertaining to the block caveability of the Central Zone (See Appendix B).

4 Property Location and Description

The Kwanika property is located in north central British Columbia, in the Omineca Mining Division, approximately 140km northwest (approximately 200km by road) of Fort St. James (Figure 4-1). The project area is on NTS map sheets 93N06 and 93N11, at latitude 55º30’ N and longitude 125º18’ W.

4.1 Mineral Tenure

The Kwanika property consists of 29 contiguous unpatented mineral claims covering an area of 9418.4 ha and is jointly owned by Serengeti Resources Inc. – 95% and Daewoo Minerals Canada Corporation – 5%. The property is not subject to any royalties or other outstanding liabilities. Serengeti acquired the claims through staking between 2004 and 2006.

Table 4-1 lists the claims for the Kwanika Project area and Figure 4-2 shows the claim map. The resource outlined in this report is contained within claims 501733, 514432, 514433, and 502953.

The Kwanika property is not subject to any known environmental liabilities and all required permits have been obtained and are in good standing.

Figure 4-1 Kwanika Property Location Map

Table 4-1 Mineral Tenure Information for the Kwanika Project

Tenure #	Claim Name	Hectares	Expiry Date	NTS	Record Date	Mining	Owner
501733	KWANIKA 1	457.642	04-Dec-2023	093N054	12-Jan-2005	OMENICA	SIR/DMCC
502953	KWANIKA 4	73.296	04-Dec-2023	093N054	13-Jan-2005	OMENICA	SIR/DMCC
505271		458.168	04-Dec-2023	093N044	31-Jan-2005	OMENICA	SIR/DMCC
505277	KWANIKA 5	458.450	04-Dec-2023	093N044	31-Jan-2005	OMENICA	SIR/DMCC
506007	KWANIKA 7	458.624	04-Dec-2023	093N044	6-Feb-2005	OMENICA	SIR/DMCC
514432		439.522	19-Nov-2023	093N054	19-Nov-2004	OMENICA	SIR/DMCC
514433		403.038	19-Nov-2023	093N054	19-Nov-2004	OMENICA	SIR/DMCC
514455	KWANIKA 8	18.316	13-Jun-2023	093N054	13-Jun-2005	OMENICA	SIR/DMCC
546495	Kwanika 9	458.767	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546496	Kwanika 10	458.884	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546497	Kwanika 11	458.982	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546498		459.078	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546500	Kwanika 13	459.184	04-Dec-2023	093N034,044	4-Dec-2006	OMENICA	SIR/DMCC
546501	Kwanika 14	459.285	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546502	Kwanika 15	459.394	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546503	Kwanika 16	459.506	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546507		459.650	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546508	Kwanika 18	459.810	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546509	Kwanika 19	460.016	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546510	Kwanika 20	460.215	04-Dec-2023	093N034,035	4-Dec-2006	OMENICA	SIR/DMCC
546511	Kwanika 21	460.385	04-Dec-2023	093N034,035	4-Dec-2006	OMENICA	SIR/DMCC
546512	Kwanika 22	18.422	04-Dec-2023	093N024	4-Dec-2006	OMENICA	SIR/DMCC
546553	Kwanika 24	18.329	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546554	Kwanika 25	36.661	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546555	Kwanika 26	36.670	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546556	Kwanika 27	55.032	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546557	Kwanika 28	36.697	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
546558	Kwanika 29	18.352	04-Dec-2023	093N044	4-Dec-2006	OMENICA	SIR/DMCC
1044440	KGV	457.996	30-Jun-2017	093N	30-May-2016	OMENICA	SIR/DMCC
Total	29 claims	9,418.370					

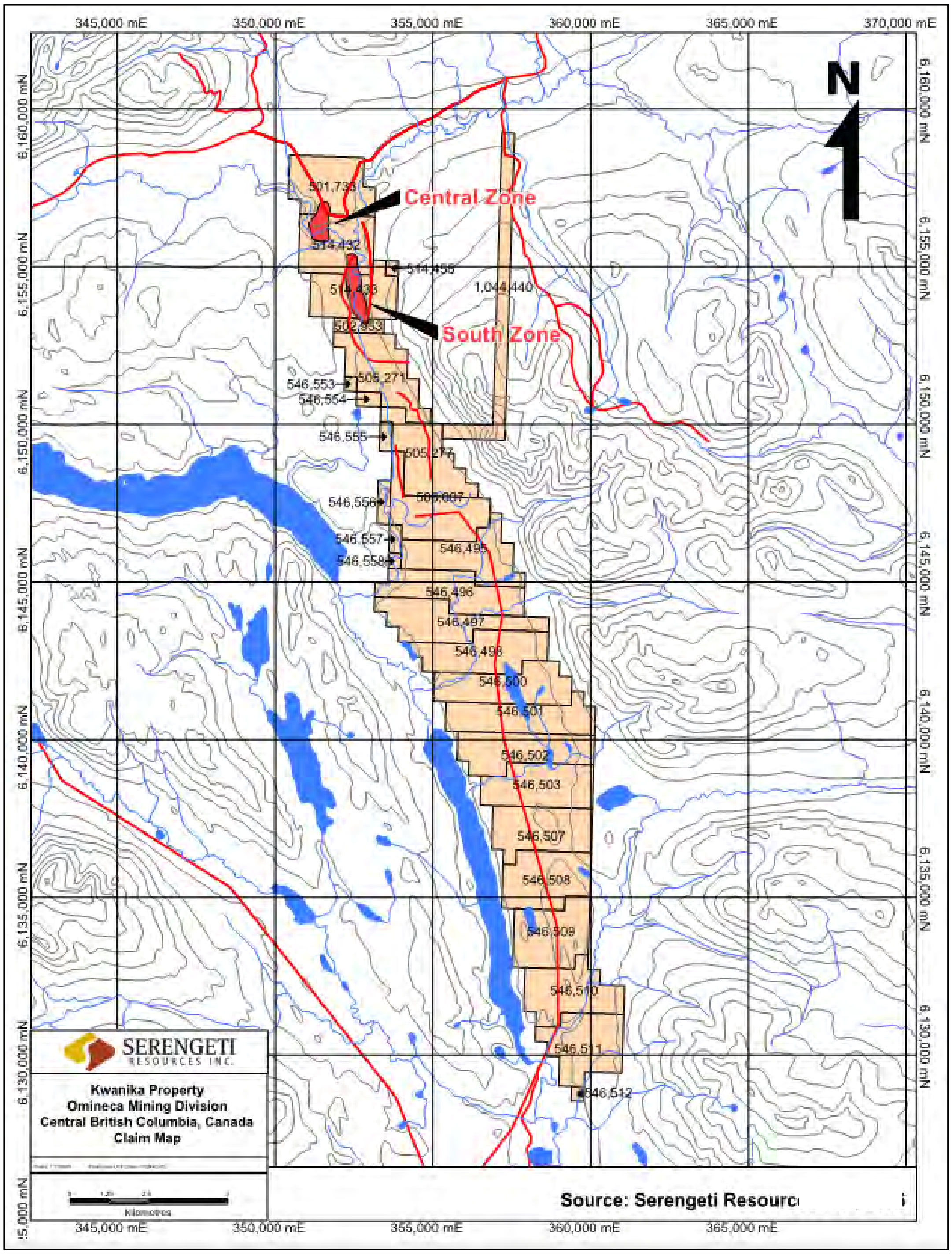


Figure 4-2 Kwanika Claim Map

4.2 Underlying Agreements

The Kwanika property is currently owned as to 95% Serengeti Resources Inc. and 5% Daewoo Minerals Canada Corporation and is subject to agreement whereby the later may earn up to a 35% interest by making aggregate expenditures, including cash payments to Serengeti in an aggregate of $8.2 million in an agreementl announced March 7, 2016.

4.3 Permits and Authorization

Exploration on the property is authorized by the British Columbia Ministry of Energy and Mines by permit number MX-13-113, most recently dated January 18, 2016 and covering the period through December 31, 2019.

4.4 Environmental Considerations

Serengeti completed a Valued Ecosystem Component study for the Project in 2008 with input from the Takla Lake First Nation.

4.5 Mining Rights in British Columbia

Subject to British Columbia law, Serengeti as valid mineral tenure holder has the sole and pre-emptive right to apply for mining rights on the Kwanika property.

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The Kwanika Property is located 140km northeast of Fort St. James in north central British Columbia. It is accessible by the well-maintained, all-weather Leo Creek Forest Service Road (FSR) and Driftwood FSR (Figure 5-1). The Driftwood FSR services the nearby town of Takla Landing and is maintained year-round by the British Columbia Forestry Service to within 29km of the site. The final 29km of access is via the Fall-Tsayta Lake FSR which is suitable for passage of four-wheel-drive vehicles in all seasons (pending snow removal) and has been maintained seasonally by Serengeti since the fall of 2006. The road is snow-free from May to October. Serengeti has developed and expanded a network of pre-existing exploration trails covering the northern end of the property.

5.2 Climate

The average temperature for this area (based on data from Fort St. James) is 3.1°C, with a peak average monthly temperature of 21.9°C in July and an average monthly low of -15.8°C in January. The region receives an average of 295mm of rainfall and 192cm of snowfall annually, with 138 days per year where precipitation exceeds 0.2mm. The Kwanika property is snow-covered from late October to May.

5.3 Local Resources and Infrastructure

The Kwanika Project is in close proximity to the well-serviced communities of Prince George, Smithers, and Fort St. James. These established centres can provide skilled labour for mine construction and operation and are presently a source of an extensive workforce pool for exploration. The property is 200km by road from the Mt Milligan mine which started production in 2014.

Serengeti reports that it has developed a beneficial association with the local Takla Lake First Nation, and that there is general community support for the Kwanika Project and the potential employment that it would provide.

5.4 Physiography

The property occupies a broad, till-blanketed valley which ranges in elevation from 900m to 1,200m. The local topography is gently to moderately sloping, with sparse bedrock exposure (Figure 5-2). The only observable rock outcrops on the property are along the meandering Kwanika Creek, where fluvial processes have locally eroded the till blanket.

Kwanika Creek lies east of the Pacific divide, draining southward into the Nation Lakes chain, and eventually into the Arctic Ocean. The property is moderately forested with spruce and lodgepole pine, broadleaf deciduous trees and shrubs, such as alder, birch and aspen, and underlying lichen and mosses.

The extent of the property is sufficient to support a mining operation with a potential of power supply from the Kemess power line 75km to the Kwanika Project site.

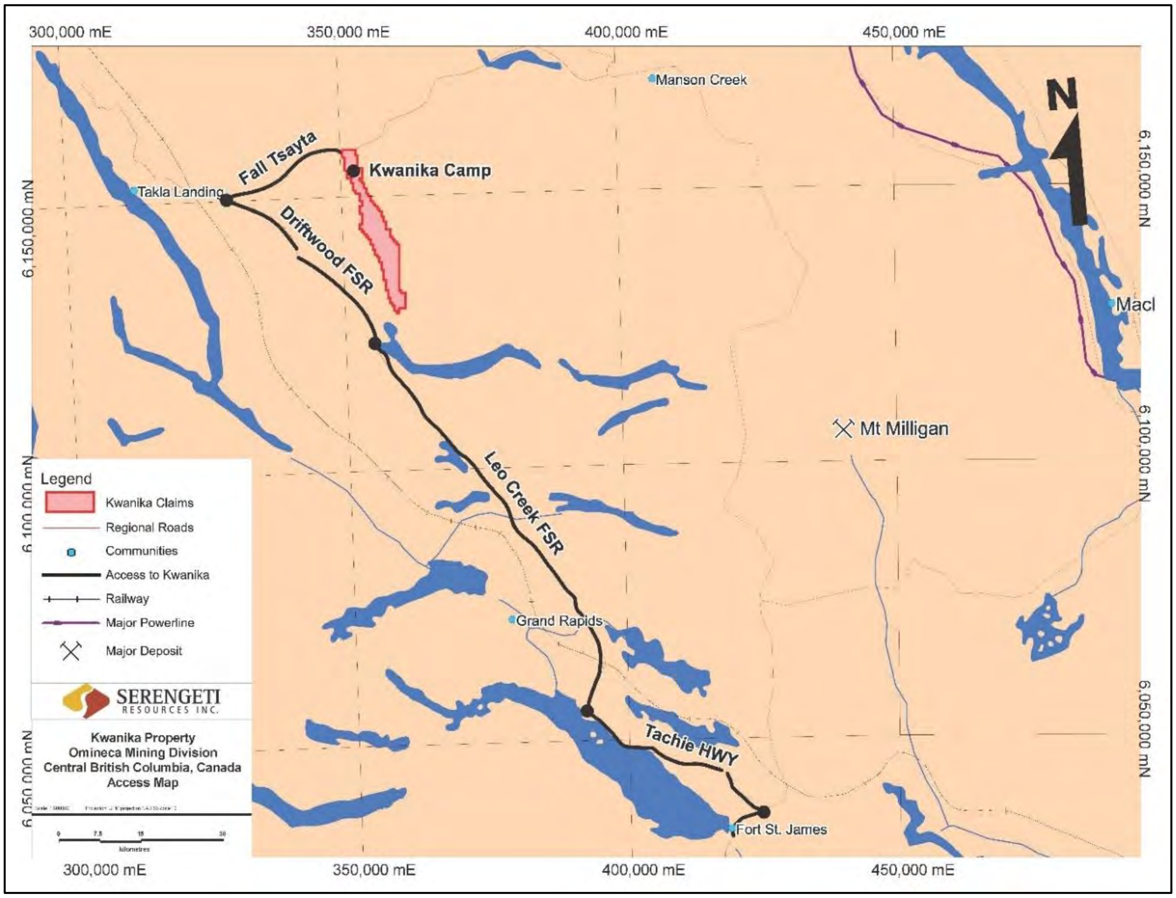


Figure 5-1 Project Access Map

Figure 5-2 Picture of the Kwanika Property

6 History

The first exploration on the Kwanika property occurred in the 1930s and 1940s following the discovery of mercury at Pinchi Lake. Initial exploration concentrated on prospecting for mercury mineralization along the Pinchi fault and for placer gold in Kwanika Creek.

Copper mineralization was first recognized along Kwanika Creek by prospectors Almond and Thurber in 1964. A. Hodgson and G. Bleiler were first to stake the property for Hogan Mines Ltd. (Hogan) in 1965. During that year, Hogan conducted a small X-ray drilling program (27.4 m) as well as a trenching and geochemical program (Macdonald, 1965; Buskas, Garrett & Morton, 1989). Geochemical results of a typical exposure yielded 0.25% Cu and 0.01% MoS_2 over 3.4m. More copper mineralized samples yielded 0.94% Cu and 0.01% MoS_2 over 2.3m.

The property was subsequently optioned to Canex Aerial Exploration Ltd. (Canex) in 1966 (Pentland, 1966; Sawyer 1969). Canex's work included geological, geochemical (sediment and water, parameters not defined) and magnetic/induced polarization (IP) surveys on a 67.6km cut grid, as well as drilling eleven diamond drillholes (856 m). The geophysics identified an IP anomaly coincident with mineralized outcrops along Kwanika Creek. Drilling confirmed that this IP anomaly was caused by sulphide mineralization that comprised up to 5% of the rock mass. A second IP anomaly with a coincident 300 gamma magnetic response and a frequency effect of 3% was also identified to the west of Kwanika Creek. It remained untested as it was thought to be located in a sedimentary environment and within the Pinchi fault zone.

The Canex option was terminated and the property was acquired by Great Plains Development Company of Canada (Great Plains) in 1969. Great Plains conducted a magnetic survey and drilled seven diamond drillholes (1,320 m) to test the previously identified IP and magnetic low anomalies (Sawyer, 1969; Buskas, Garrett & Morton, 1989). Results for drillholes DDH# B-1, B-2, and B-4 showed the best copper mineralization at the bottom of the holes, with 0.10% Cu to 0.21% Cu in the top 45m to 0.21% Cu to 0.41% Cu at 91m to 101m. The drilling program outlined an area about 490m by 300m of low grade copper mineralization, grading approximately 0.20% Cu. No gold analysis was done and molybdenum was analyzed only in selected sections.

In 1972, Bow River Resources Ltd. (Bow River) mapped the property and drilled six percussion holes for a total of 549 m, (Buskas, Garrett & Morton, 1989). An analysis of the drillhole logs reveals 0.15% Cu to 0.17% Cu over the full length of three holes (9m to 91m depth).

Pechiney Developments Ltd. (Pechiney) optioned the property in 1973 and conducted a 64.4km grid IP and resistivity survey (Hallof & Goudie, 1973). When the results were interpreted with previous drillhole data, it was determined that the best copper grades corresponded to anomalies with frequency effects over 3% and resistivities over 100 ohm-m. In 1974, Pechiney conducted a 30 hole, 2,993m percussion drilling program (Guelpa, 1974); however, assay results for this work are not available.

In 1981, Placer Developments Ltd. conducted a geochemical survey further south which consisted of 35 soil samples and 16 rock samples (Bulmer, 1981). Soil samples were collected from a grid with 100m sampling interval and a line spacing of 200m. Rock samples were collected from outcrops on the soil grid as well as along Kwanika Creek. The survey identified anomalous copper (up to 2,520ppm), molybdenum (up to 730ppm) and mercury (up to 90 ppb) values occurring within cataclased granite along Kwanika Creek, near the Pinchi fault.

In 1983, Aume Resources Ltd. conducted a geochemical survey at the northern end of the Kwanika property to investigate the gold content of mercury mineralization associated with the Pinchi fault (Culbert, 1983). The survey consisted of 43 soil samples, 37 stream sediment samples and 12 rock samples, which were collected during line traverses and included samples collected outside the property boundaries. Assay results supported the high concentration of mercury associated with the Pinchi fault (up to 6,400 ppb), however, Au and Ag values were not anomalous.

In 1986, Daren Resources Ltd. conducted a geochemical survey in the northwest corner of the Kwanika property, which included work on the northwestern and western periphery of the property (Christoffersen, 1986). The regional survey consisted of 96 soil samples, 14 silt samples, and 15 rock samples. The results obtained from this survey confirmed previously identified low order gold, silver, and arsenic anomalies, with the best sample grading 275ppb Au, 58ppm As, and 1.1ppm Ag.

In 1989, W. Halleran staked the Swan property, located in the northern portion of the Kwanika claims at 55°30’N, 125°19’W (Carpenter, 1999), on ground previously abandoned by Bow River. Halleran was able to demonstrate the association of gold with the copper mineralization and subsequently optioned the property to Eastfield Resources Ltd. (Eastfield) (Buskas, Garrett & Morton, 1989). During 1989, Eastfield conducted an extensive exploration program which consisted of cutting 22.6km of grid lines, a geochemical survey (55 soils at 50m intervals, 143 stream sediments on Kwanika Creek tributaries and 162 rock samples), and a 23.3km IP survey. Work conducted during this period also consisted of geological mapping, prospecting and resampling historical core. Results from the geochemical survey indicated that the highest and most consistent copper-gold anomalies were restricted to the North copper zone (values up to 9,462ppm Cu and up to 1,227ppb Au). A comprehensive analysis of the geophysical chargeability results in conjunction with geochemical, drillhole and geological surveying data yielded six targets for future exploration which extended throughout the property. Furthermore, it was determined that the best copper mineralization was not always associated with the strongest sulphide mineralization, suggesting that significant copper mineralization may be associated with less intense IP anomalies.

Eastfield also carried out a small drilling program in 1991 consisting of four diamond drillholes, totalling 549m (Morton, 1991). The program intended to test geophysical targets to the north and west of the Pechiney 1974 percussion holes. The drilling program failed to identify new zones of significant mineralization.

Discovery Consultants (Discovery) re-staked the Swan property and continued exploration in 1995 with a limited heavy mineral stream sediment (two samples) and rock (15 samples) geochemical program (Carpenter, 1996). The heavy mineral stream sediment samples from the west edge of the property yielded anomalous gold values of 3,180ppb and 4,580ppb, while the rock samples had values up to 73ppb Au and 2,607ppm Cu. In 1999, Discovery obtained an additional three heavy mineral stream sediment samples from the east side of the property which yielded anomalous gold values of 7,450ppb and 1,730ppb (Carpenter, 1999).

A historical Mineral Resource estimate for what is currently referred to as the South Zone deposit was produced in 1976. The estimate stated a Mineral Resource of 36Mt grading 0.20% Cu (Pilcher and McDougall, 1976). No mention was made of the source of this estimate or how it was done, however, Serengeti was able to obtain a similar result using the same dataset and a polygonal method. Note that this is an historical estimate as defined in NI 43-101. The historical estimate doesn't use mineral resource categories as outlined in NI43-101. The estimate is only referenced for historical completeness and it should not be relied upon as it is superseded by the mineral resource estimates presented in Section 14 of this report.

No further work was done on the property until Serengeti acquired it in 2004. Subsequent work carried out is described in Section 9, Exploration.

7 Geological Setting and Mineralization

7.1 Regional Geology

The Kwanika property lies in the northern part of the Upper Triassic to Lower Jurassic Quesnellia Terrane (Quesnel Trough) which comprises a belt of Lower Mesozoic volcanic rocks and intrusions lying between highly deformed Proterozoic and Paleozoic strata to the east and deformed Upper Paleozoic strata to the west (Garnett, 1978). The Quesnel Trough is the host of numerous alkalic and calc-alkalic porphyry copper-gold deposits within British Columbia. In the area around the Kwanika property, Quesnellia is bounded by the Pinchi fault on the west and by the Manson fault on the east. The Pinchi fault separates Permian rocks of the Cache Creek Terrane to the west from the Upper Triassic Takla Group to the east (Garnett, 1978).

The porphyry deposits in the general vicinity of the property (Mt. Milligan and Lorraine) are associated with potassically altered diorite, monzodiorite, monzonite, and syenite plugs and stocks, as well as coeval andesitic volcanic rocks. The significant deposits in the region are associated with strong aeromagnetic features that trend both east-west and northwest, and with strong copper-gold stream sediment anomalies.

Garnett (1978) separated the Hogem Batholith into three major intrusive phases based on both age and lithology (Table 7-1 and Figure 7-1).

Table 7-1 Regional Geology Setting

Division of Hogem Batholith Intrusive Suite		
Intrusive Phase	**Phase Divisions**	**Rock Varieties**
PHASE III: Lower Cretaceous		Leucocratic Granite, Alaskite
PHASE II: Mid to Lower Jurassic	Chuchi syenite Duckling Creek Syenite Complex	Leucocratic Syenite, Quartz Syenite Leucocratic Syenite Foliated Syenite
PHASE I: Lower Jurassic to Upper Triassic	Hogem Granodiorite Hogem Basic Suite	Granodiorite, Quartz Monzonite, minor Tonalite, Quartz Diorite, Quartz Monzonite, Granite Monzonite to Quartz Monzonite Monzodiorite to Quartz Monzodiorite Nation Lakes Plagioclase Porphyry Monzonite Monzodiorite Diorite, minor Gabbro, Pyroxenite, Hornblendite

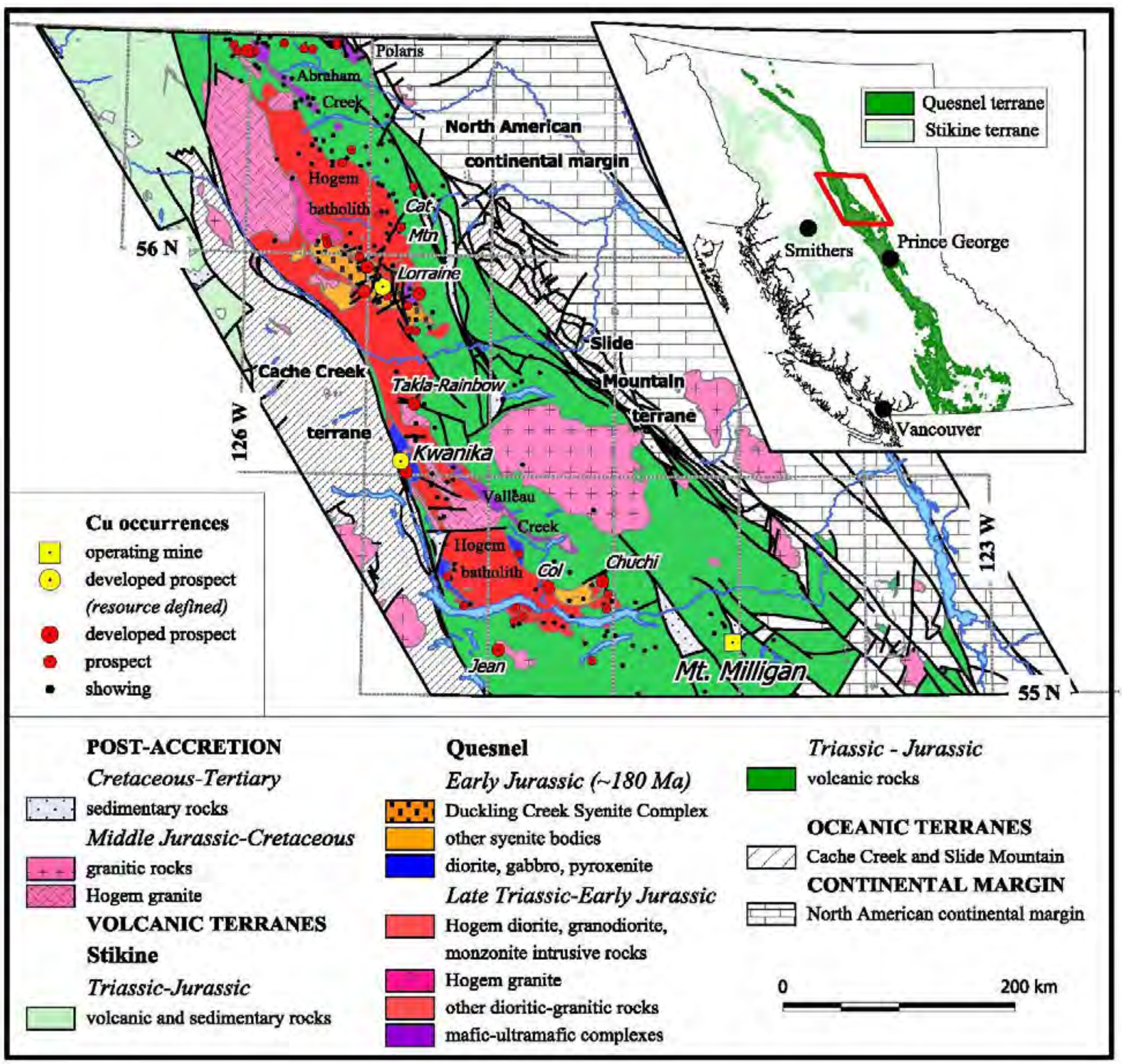


Figure 7-1 Regional Geology

7.1.1 Structure

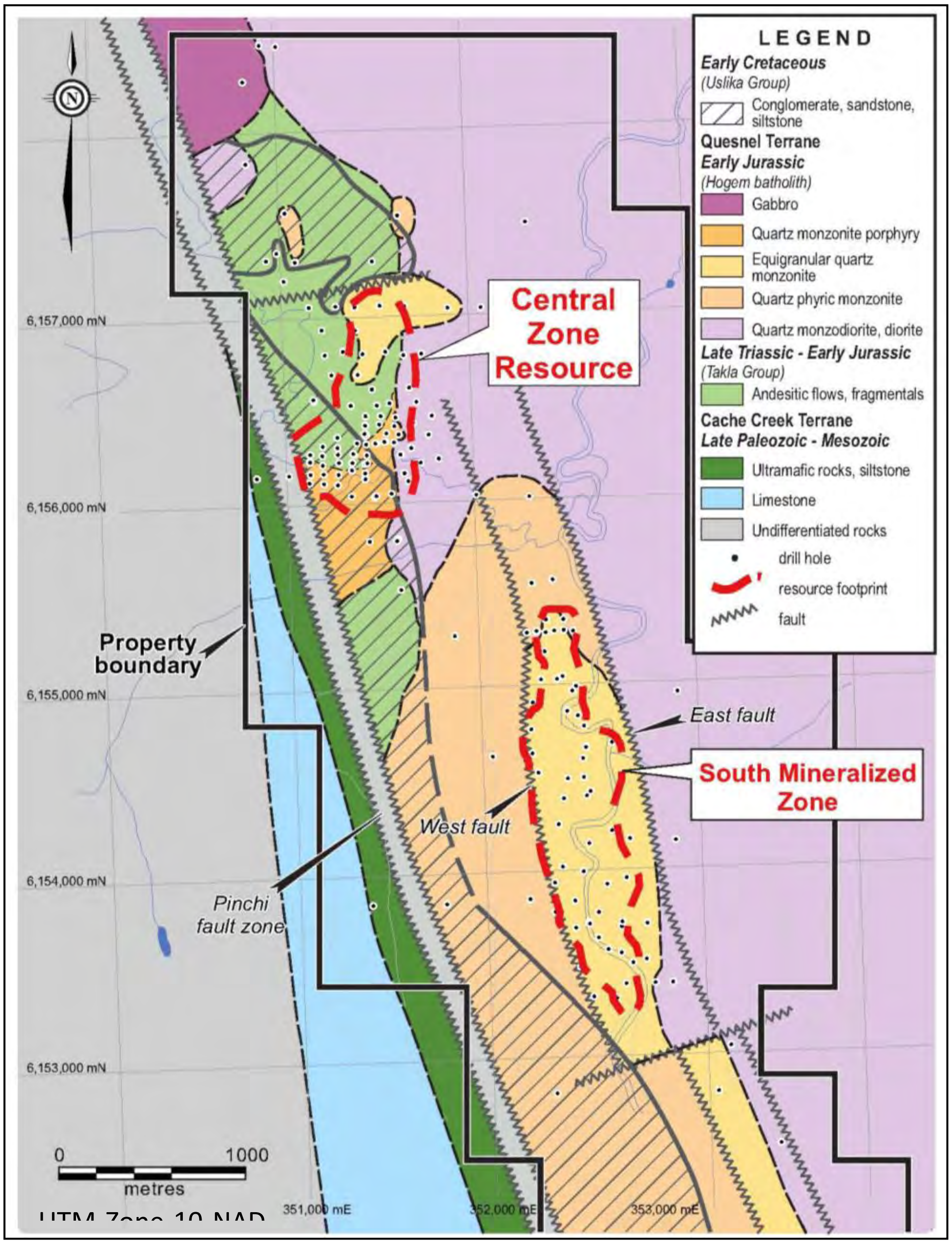
The following description of the regional tectonic and structural setting of Kwanika is taken from Osatenko, 2016.

The Kwanika deposits occupy the western margin of the Late Triassic to Jurassic composite Hogem batholith that at the latitude of the Kwanika deposits, defines the western margin of Quesnel terrane. Faulting and variable motion on post-mineral, north, north-west trending arc-parallel faults separate Quesnel terrane from the Cache Creek terrane at Kwanika. Repeated motion on these Mesozoic through Tertiary age faults is responsible for the uplift, erosion and development of the supergene zone, (locally present at the upper surface of central zone), its subsequent preservation within a fault-bound Early Cretaceous basin and later dissection and removal of the western portion of the Central Zone during Eocene dextral trans-tension.

A north-trending, steep west-dipping fault locally demarks the eastern edge of an Early Cretaceous sedimentary basin that formed above the Kwanika Central Zone. Normal displacement on this fault and subsidence (Osatenko and Moore (in prep)) produced the current geometry of the Central Zone and the basin for Early Cretaceous sedimentary rocks that were deposited non-conformably on Early Jurassic quartz monzonite of the Central Zone. Generation of the pull-apart basin is constrained only by the age of sedimentary rocks preserved. The presence of behemoth sized mineralized intrusive slabs within the basin suggests a steep fault-scarp topography and periods of tectonic instability to periodically introduce such large blocks. Shearing is common along the basal contact that separates the Early Cretaceous sedimentary rocks from the highly oxidized supergene zone of the Central Zone and is thought to reflect inversion of the basin probably during northeast-directed compression in the Mid-Cretaceous during development of the Skeena fold and thrust belt on Stikine terrane. Following the Mid-Cretaceous, motion along major strike slip faults in the Canadian Cordillera including the Pinchi Fault was primarily dextral.

7.2 Property Geology

The Kwanika Project consists of two mineralized areas: the Central Zone and the South Zone. The geology and alteration for each zone are described independently. Figure 7-2 shows the interpreted geology around the Central and South zones.



(Source: Serengeti 2016)

Figure 7-2 Local Property Geology

7.2.1 Property Structure

On a property scale, three major NNW oriented fault structures have been identified largely from drillhole data; the Central Zone fault, West Kwanika fault and East Kwanika fault. These faults are interpreted to display a variable amount of dip slip (generally west side down) and strike slip movement. The most laterally persistent planar feature is the West fault which bounds the west side of the South mineralized zone, and extends to and beyond the north property limit. It is possible that the South and Central Zones were once part of one large mineralized system, offset dextrally along this structure, although this has not been conclusively demonstrated. The East fault is interpreted to bound the east side of the South zone and has also been traced to the north where it may form the eastern limit of a sub-basin of Cretaceous sedimentary rocks, located near the north end of the property.

Faulting within the Central zone is more complex and requires further study to gain a more complete understanding. However, the most persistent structural feature within the zone is a steep west-dipping, NNW oriented fault referred to as the Central Zone fault. It separates an eastern domain characterized by upright-oriented lithological contacts and grade distribution boundaries from a western domain characterized by sub-horizontal orientation of lithologies and grade boundaries. This same fault locally over-steepens the unconformity at the eastern limit of the Cretaceous basin which overlies and preserved (from removal by glaciation) the supergene copper zone present locally at the upper surface of the Central zone.

The sub-vertically dipping Pinchi fault lying near the west property boundary, truncates the Central Zone between 500 and 750 meters below surface, with rocks of the Cache Creek Terrane lying to the west of it.

7.2.2 Central Zone Geology

The Central Zone deposit is characterized by the presence of two major and several minor intrusive bodies of the multi-phase Hogem Batholith intruding into successions of andesitic rocks of the Takla Volcanic Group.

The most economically significant intrusive body is a north-northeast trending monzonite stock that dips shallowly to steeply to the west. The intrusion has a strike length of nearly 1.3km and a thickness of 50m to 350m. The monzonite is a medium-grained, equigranular to feldspar porphyritic rock, consisting of plagioclase and K-feldspar, with lesser amounts of amphibole, biotite, quartz, and minor tourmaline in veins and narrow breccias. The high grade copper-gold mineralization in the Central Zone is dominantly hosted within, and immediately adjacent to, the monzonite intrusive. Monzonite has also been intersected at depth in the western and southwestern parts of the Central Zone and is thought to connect to the sill-like body in the central part of the deposit, suggesting the possibility of deep Central Zone mineralization.

In hand specimen, the monzonite is cream to orangey in colour, and rarely pale grey. Locally, it is distinguishable by a “rice-grain porphyry” texture, marked by coarse-grained, somewhat oval-shaped plagioclase phenocrysts. Thin sections of the equigranular phase of the intrusion show textures ranging from non-distinct to trachytoid, with aligned subhedral plagioclase phenocrysts. The porphyritic phase comprises mainly plagioclase crystals occurring in a fine-grained matrix of quartz and K-feldspar (Le Couteur, 2008). Albite, hematite, sericite, K-feldspar, silica, and secondary biotite alteration overprint primary textures and compositions.

Where strongly mineralized, the unit commonly displays quartz stockwork and hydrothermal brecciation. The strongest mineralization occurs within zones of strong to intense, texture-destructive albite + hematite alteration, commonly occurring at the stratigraphic top of the hypogene mineralized zone.

The second prominent intrusive body in the Central Zone comprises monzodiorite and diorite and lies both to the east of and below the monzonite unit. Whereas core logging and mapping have divided the body into two zones, the zones have strikingly similar magnetic susceptibility profiles and grade characteristics, suggesting that they may be variably altered parts of the same intrusive body. This body is the largest of the intrusive phases encountered on the property. Its strike length has been demonstrated to be at least two kilometres, and the body has been encountered in varying thicknesses in nearly all the drillholes in the Central Zone. Lateral extent and thickness of this body is not known as it remains open to the east and at depth as the majority of drillholes have terminated within the monzodiorite and diorite units.

7.2.3 Central Zone Alteration

Potassic alteration is the most widespread facies encountered within the Central Zone area. Concentric zoning of the potassic alteration is evident as intensity grades from strongly pervasive and texture destructive styles in the mineralized core of the deposit to a widespread envelope of weak, fracture- and vein-related alteration. Strongly potassically altered zones often coincide with local, pervasive silica flooding. These potassic + silica altered zones are commonly mineralized. K-feldspar is the dominant mineral in the intensely potassic-altered core, whereas secondary biotite is generally more common at depth and peripherally. Potassic alteration is observed in all the major units; however, it is most common in the monzonite, monzodiorite, and diorite units.

Albitic alteration is significant in the Central Zone as there is a strong correlation with increasing intensity of albite alteration and high grade mineralization. The albite alteration facies is identified solely by the presence of albite; however, the assemblage of albite + hematite ± silica veinlets is often observed where albite is present. The albite alteration progression is characterized by the following sequence: a strong to intensely altered and texture destructive albitic zone in relatively sharp contact with the underlying, strongly to intensely potassic altered interval, all of which is surrounded by a broad zone with irregular, patchy occurrences of vein- and fracture-related albite alteration. While albitic alteration occurs in all lithologies in the Central Zone, its intensity is generally strongest in the monzonite, where it occurs at the top of the hypogene zone. The strongly albitic core in the hypogene zone is also concurrent with a weak to strong stockwork of quartz ± albite veins. This albite-altered, quartz-stockwork zone is among the most important indications of high-grade copper-gold mineralization in the Central Zone.

Propylitic alteration is widespread in the Central Zone. As is typical in many copper-gold porphyry deposits, the propylitic assemblage occurs at the fringes of the deposit and is generally associated with lower grade copper-gold mineralization. Propylitic alteration is most intense at the outer boundary of the potassically altered core. The intensity of alteration decreases away from this potassic-propylitic boundary. Overprinting by other alteration assemblages is common, often in complex relationships. Propylitic alteration is most prevalent in the andesite and diorite units; however, it affects all the major lithologies in the Central Zone. There is also evidence of regional scale, low grade propylitic alteration on the Kwanika property, as several of the drillholes testing locales outside the deposit area have encountered primarily propylitic altered lithologies. Table 7-2 shows the primary alteration assemblages in the Central Zone.

Table 7-2 Alteration Assemblages in the Central Zone

Central Zone Alteration Assemblages	
Alteration Type	**Mineral Assemblage**
Potassic	K-feldspar + secondary biotite ± gypsum/anhydrite veining
Albitic	Albite, albite + hematite ± silica
Propylitic	Chlorite + epidote + pyrite ± sericite ± carbonate

7.2.4 South Zone Geology

The South Zone deposits occur within a fault bounded sequence of strongly altered intrusive rocks of alkalic to intermediate composition. These intrusive lithologies have been previously described by Garnett (1972), Garnett (1978), and by Eastfield as belonging to the various intrusive phases of the Upper Mesozoic Hogem Batholith.

In hand specimen, the host intrusive lithologies are characterized by their reddish-orange colour, porphyritic texture, and relative silica undersaturation. They are described as quartz-bearing feldspar porphyritic monzonites and monzodiorites. Specific igneous compositions are difficult to determine due to pervasive alteration. Thin section examination has shown that the rocks are composed of feldspar phenocrysts occurring in a fine-grained matrix of quartz and feldspar, where feldspar includes plagioclase, K-feldspar, and less commonly albite (Le Couteur, 2009, McLeod, 2009). Secondary minerals include chlorite altered amphiboles and biotite, minor magnetite, as well as trace concentrations of apatite, titanate, rutile, ilmenite, and zircon. The majority of the samples studied have been subjected to brittle deformation. Some fragments to several millimetres in size are cemented by finer fragments of similar rock. There is evidence of more than one stage of fracturing as multi-lithic breccia fragments are present (McLeod, 2009).

Modal abundances of quartz, alkali feldspars, and plagioclase demonstrate that the primary South Zone lithologies are quartz monzonites to quartz monzodiorites. Le Couteur (2009) notes that even the 'freshest' samples are strongly altered, thus, composition (i.e., name) of the parent magma is difficult to identify.

The South Zone deposit area is transected by post-mineral to late-mineral dykes. The majority of the dykes are andesitic in composition with less common strongly sericite + K-feldspar altered, post-mineral monzonite (?) dykes. All dykes are in sharp to locally faulted contact with the main intrusive units. Typical thicknesses encountered are in the order of one metre to five meters. There is a noted increase in abundance of dykes to the east, but further drilling is required to fully understand the dimension and location of these bodies.

The host lithologies of the South Zone deposits occur within a north-south trending structural corridor. This structural corridor is bounded by the West Fault to the west and by a similar fault zone termed the East Fault Zone along the eastern boundary of the corridor. Coincident chargeability and resistivity anomalies form a geophysical domain that represents the fault-bounded South Zone corridor. This variably mineralized domain is 2,900m long and up to 500m wide.

The West Fault has been encountered in ten drillholes over 350m and its geophysical signature can be traced over the majority of the 2,900m long geophysical domain. Near surface, the West Fault is a three to five metre wide foliated cataclasite. At depth, the fault is represented by a crush zone that has an inferred true width up to 75m. The West Fault is an important structure for three principal reasons:

- Throughout the northern-most 750m of the South Zone deposit, there is commonly a marked increase in grade of copper and molybdenite in a corridor within approximately 100m to 150m of the West Fault.
- Immediately east of the West Fault, the mineralized system is observed up to 600m below surface and is open to depth.
- The West Fault is believed to have played an important role in the formation of significant portions of the South Zone deposit as it may have been the primary pathway for fluid flow, which would help to explain the first two observations.

There is strong geophysical and geological evidence for the existence of an 'East Fault Zone' that parallels the West Fault. Drilling completed in 2010 encountered broad zones of shearing and sericite alteration bounding the mineralization to the east, which are thought to represent the regionally extensive East Fault Zone.

A major body of leucocratic quartz-phyric monzonite unit occurs to the west of the West Fault. This quartz-phyric monzonite does not contain any significant base or precious metal mineralization.

7.2.5 South Zone Alteration

There are several alteration facies affecting the lithologies in the South Zone. K-feldspar alteration is the most widespread facies and affects virtually all rocks observed in drill core and outcrop. K-feldspar alteration varies from halos around veins and fractures to more commonly occurring pervasive flooding. Sericite alteration also occurs throughout much of the South Zone. Sericite is observed in thin section as fine to coarse patches, altering or replacing feldspars (Le Couteur, 2009). As noted in the above section, K-feldspar and sericite alteration commonly destroys igneous textures and original compositions. This pervasive alteration leads to difficulties in the identification of the parent magma of intrusive lithologies. The most intensely altered sections observed in thin section are associated with zones of brittle deformation.

An iron-rich alteration assemblage consisting of chlorite + pyrite + silica ± secondary biotite also occurs frequently within the South Zone and is observed overprinting earlier K-feldspar and sericite alteration. The iron-rich alteration assemblage is associated with zones of brittle deformation that are up to several metres in thickness. This type of alteration is typically texture destructive, forming alteration pseudo-breccias to completely replacing original igneous textures and compositions. Chalcopyrite and molybdenite, as well as elevated precious metal concentrations, occur within these zones.

South Zone lithologies have also been affected by several less important alteration facies, including epidote and hematite alteration associated with strong fracturing, chlorite alteration and replacement of amphibole and biotite, carbonate veining and flooding, and late-stage silica flooding and quartz veining. The intensities of these secondary alterations are highly variable and their occurrence is localized.

7.2.6 Central Zone Mineralization

Copper and gold mineralization in the Central Zone at Kwanika occurs primarily in potassic and albitic altered lithologies. Alteration and mineralization grade outwards from a strong to intensely potassic and albitic alteration, strongly mineralized core zone, to a variably propylitic altered, weakly mineralized periphery. Analysis of the Au/Cu ratio demonstrates a gold-enriched Central Zone core, surrounded by a broad, copper-enriched margin. Stronger mineralization mainly occurs in the monzonite, monzodiorite, and diorite units, but is also present at generally lower grades within the andesite. Whereas higher grade mineralization is locally found in all lithologies, there is a clear correlation between the monzonite and copper-gold enrichment. The emplacement of this monzonite unit is thought to be a primary control on mineralization at Kwanika. Other zones of high grade mineralization are closely spatially related and/or grade outwards from the monzonite.

Hypogene mineralization is strongly controlled by several generations of quartz + sulphide veining, with the highest copper and gold grades occurring in areas of quartz stockwork. The majority of the vein complexes and the structures (i.e., fractures, faults) observed in drill core are steeply dipping to subvertical, which suggests these structures may be controlling hypogene mineralization in the Central Zone. At the top of the hypogene zone, well mineralized quartz stockwork zones are associated with strongly to intensely albite altered wall rocks. At greater depths in the system, mineralized quartz veins commonly occur within narrow, potassic altered vein halos. Chalcopyrite and bornite are the important copper minerals. There is a consistent trend of mineralization in the Central Zone, described by the following top-down progression.

- bornite-rich zone
- chalcopyrite + bornite zone
- chalcopyrite + rare bornite zone

Chalcopyrite occurs as fine to coarse clots within veins and, less commonly, as fine to coarse disseminations. Bornite is commonly observed as rims to chalcopyrite grains and, in many instances, it appears that chalcopyrite has been altered to bornite (Le Couteur, 2008). Rare cases of coarse aggregates of bornite (greater than one centimetre) occur in the albite-altered zone. Disseminated mineralization is of lesser importance in the upper part of the deposit but increases in significance at depth within the system. Pyrite occurs throughout the Central Zone as disseminations within veinlets and wall rocks. Overall, pyrite comprises 2% to 3% of the rock in the Central Zone.

A supergene enrichment blanket has been superimposed on the upper surface of the hypogene mineralization in the Central Zone. A conformable supergene profile has been preserved beneath the west-dipping sedimentary basin cover. Thickness of the supergene profile is highly variable due to the influence of local structures. It ranges from five metres up to 70m in thickness and extends laterally for 500m.

Two distinct assemblages of supergene mineralization are observed in the Central Zone resource area:

- supergene oxide (native copper)
- supergene sulphide (chalcocite, covellite)

Supergene oxide mineralization is most commonly observed overlying the supergene sulphide assemblage. Native copper in the supergene oxide zone occurs mainly as wires along fractures, suggesting copper transport in solution. Phase diagram evidence of native copper stability in meteoric waters suggests that native copper was deposited by descending sulphate solutions from the oxidized upper part of the system. Thin section analysis has shown that chalcocite and bornite in the Central Zone have similar habits and grain sizes as the chalcopyrite, suggesting that secondary copper sulphides occupy pre-existing chalcopyrite sites. These textural observations in conjunction with the fact that there is an increased copper content of native copper (100% Cu), chalcocite (79% Cu), and bornite (63% Cu) as opposed to chalcopyrite (33% Cu), suggests that supergene sulphide enrichment might be due to replacement of chalcopyrite by secondary minerals with a higher copper content, rather than deposition at new sites (Le Couteur, 2008).

7.2.7 South Zone Mineralization

Mineralization in the Kwanika South Zone deposit area is different in type and mode of occurrence as compared to mineralization in the Kwanika Central Zone. The South Zone is characterized by porphyry style copper + gold + molybdenum + silver mineralization within monzonite, quartz monzonite, and monzodiorite. Copper enrichment is associated with brecciated zones that have undergone secondary K-feldspar flooding and/or intense pyrite + chlorite + silica alteration. McLeod (2009) suggested a sequence of mineralization whereby alkalic igneous rocks are intruded followed by hydrothermal/ hydropressure injections producing potash alteration (K-spar flooding, sericitization) and subsequent healing/deposition of calcite ± silica and sulphide. No significant supergene enrichment has been observed to date in the South Zone.

The primary economic minerals are chalcopyrite and molybdenite. Chalcopyrite occurs as fine to coarse-grained (greater than one centimetre) disseminations along fractures and within zones of intense silica flooding. Molybdenite occurs primarily along fractures and quartz vein selvages, and less commonly as disseminated blebs associated with pyrite and chalcopyrite. While there is a clear association of molybdenite enrichment with copper mineralization, the relationship is not linear. Zones of molybdenum enrichment commonly occur within strongly fractured and/or quartz veined domains, which are located both within and peripheral to copper + gold mineralized zones. These observations suggest that molybdenite deposition was either associated with a separate (interpreted as later) mineralizing event or it was remobilized and deposited along fracture and/or vein selvages. Less important economic minerals in the South Zone include sooty grey chalcocite occurring along fractures and bornite rimming chalcopyrite. Trace amounts of enargite, tetrahedrite, sphalerite, galena, and rare greenockite are present.

Carbonate flooding and veining occurs throughout the sulphide enriched regions. McLeod (2009) notes a sequence of calcite flooding and sulphide mineralization occurring after brecciation of the host lithologies. Quartz + sulphide veinlets are relatively minor in the South Zone, although they are more abundant with increased copper-molybdenite grades at depth along the West Fault.

8 Deposit Types

Porphyry copper-gold deposits in British Columbia occur in both the Quesnellia and Stikinia terrains and in post-accretionary settings. They are classified into three types: Alkalic, Transitional and Calc-Alkalic, based on the composition of the host rocks, Cu/Au metal ratios, alteration types, and presence or absence of quartz stockworks. Each of the three types of porphyry copper-gold deposits is represented in British Columbia by at least one very significant deposit (Figure 8-1).

The Central Zone deposit is similar to the classic alkali porphyry model in that the mineralization is associated with an intrusive complex of alkali-feldspar-saturated monzonite. However, the deposit differs from that alkali porphyry model, being associated with strong quartz stockwork. In this regard, it is similar to the calc-alkalic porphyry type deposits. Therefore, in the opinion of Serengeti geologists, the Central Zone deposit may in fact be transitional in nature between alkalic and calc-alkalic types.

The South Zone deposit is a structurally controlled porphyry deposit. Host lithologies are quartz monzonitic to quartz monzodioritic in composition. Thin section analysis has determined that copper-gold-silver-molybdenite mineralization is associated with zones of brittle deformation that have been inundated by intense K-spar ± silica flooding. The structures that bound the deposit to the east and to the west are interpreted to be both the causes of this brittle deformation and the conduits for fluid flow.

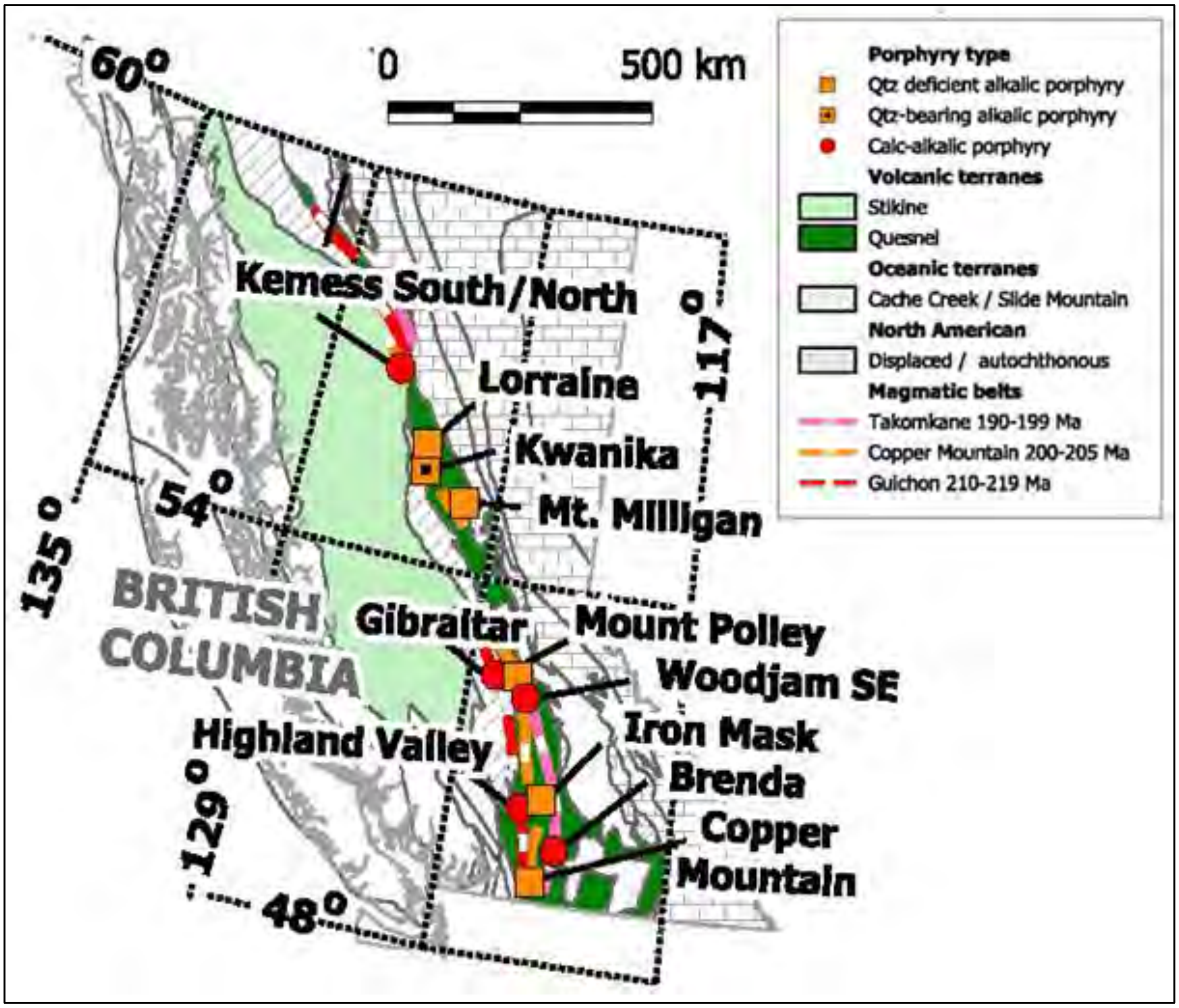


Figure 8-1 **Porphyry Deposits in British Columbia**

9 Exploration

In 2005, Serengeti conducted a 530km airborne magnetic/radiometric survey and collected eleven rock samples on the Kwanika and Germansen properties to assist in porphyry target identification (Osatenko, 2005). The airborne survey identified a small magnetic anomaly on the east side of the known porphyry copper-gold deposit, with similar anomalies trending to the north-northwest of the deposit, as well as to the south. Six of these anomalies are associated with weak K/Th anomalies, which are often associated with porphyry copper-gold deposits. The copper, gold, and molybdenum values in rock samples associated with the deposit outcrops along Kwanika Creek ranged from 507ppm to 10,740ppm Cu, 22ppb to 416ppb Au, and 2ppm to 533ppm Mo.

During 2006 and 2007, Walcott Geophysics (Walcott) was engaged by Serengeti to carry out several ground-based IP surveys in the vicinity of the Central and South Zone deposits. In 2006, Serengeti conducted a magnetic and IP survey over 26.9km of geophysical lines. The results outlined a significant IP signature over the Kwanika deposit as well as a continuation of this IP anomaly into a large, covered area to the north-northwest.

The following year, Serengeti carried out a regional airborne magnetic and electromagnetic (EM) survey, totalling 320 line-km, over the Kwanika property (Figure 9-1). The purpose of the survey was to detect zones of conductive sulphide mineralization, to outline any porphyry-style intrusive complexes, and to provide information that could be used to map the geology and structure of the survey areas. The results yielded by the survey identified multiple high magnetic/low resistivity anomalies throughout the property, which outline a general north-northwest trend coincident with South Zone and Central Zone deposit areas.

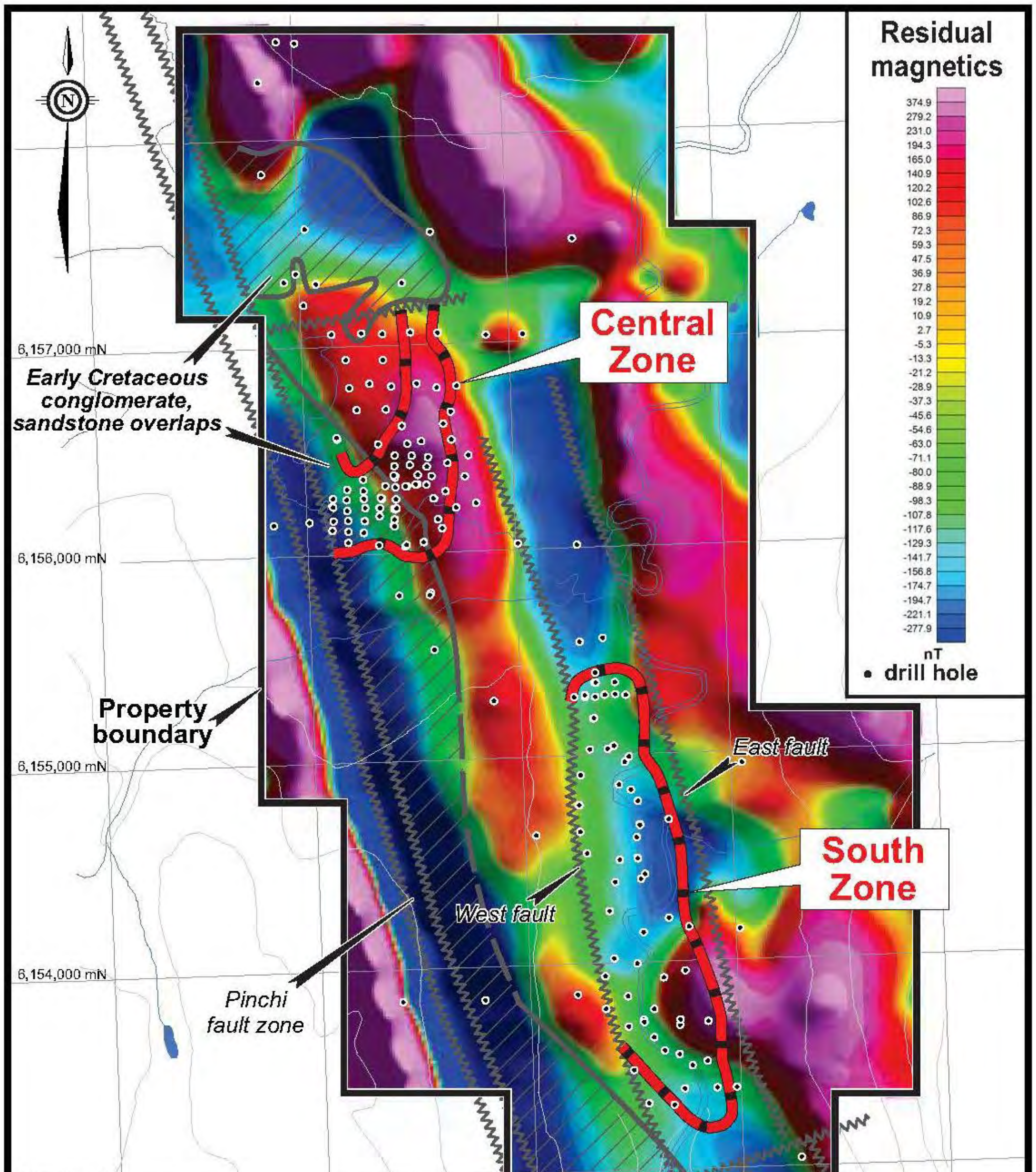
The IP work has included 50m, 100m, and 200m dipole spacing's in surveys carried out over 22 lines, covering 87.5 line-km (Figure 9-2). The results of the various surveys have outlined an area of anomalous chargeability (i.e., greater than 12mV/V) over an area measuring 5.5km long by 300m to 500m wide in the northern section of the Kwanika property. The shape of this anomaly is directly coincident with the outline of the currently known, near surface (i.e., within approximately 200m) copper-gold ± molybdenite mineralization in the Central and South Zone deposits. Drilling by Serengeti and earlier operators has shown that strong chargeability anomalies (i.e., greater than 20 mV/V) are commonly coincident with zones of higher grade, near-surface copper-gold ± molybdenite mineralization.

In 2007, selected baseline environmental studies were initiated on the Kwanika property by Ecofor Consulting Ltd. This phase of work was concluded in November 2008 and included measuring stream discharge levels, water quality, and other pertinent hydrological data.

In the summer and fall of 2008, Walcott was contracted to conduct 70 line-km of 100m spaced dipole IP surveys over 22 lines from south of the two known deposits to the southern boundary of the Kwanika property, a north-south distance of approximately 23km. Several chargeability anomalies have been identified by the IP surveys and will be the basis for further investigation of the southern section of the Kwanika property.

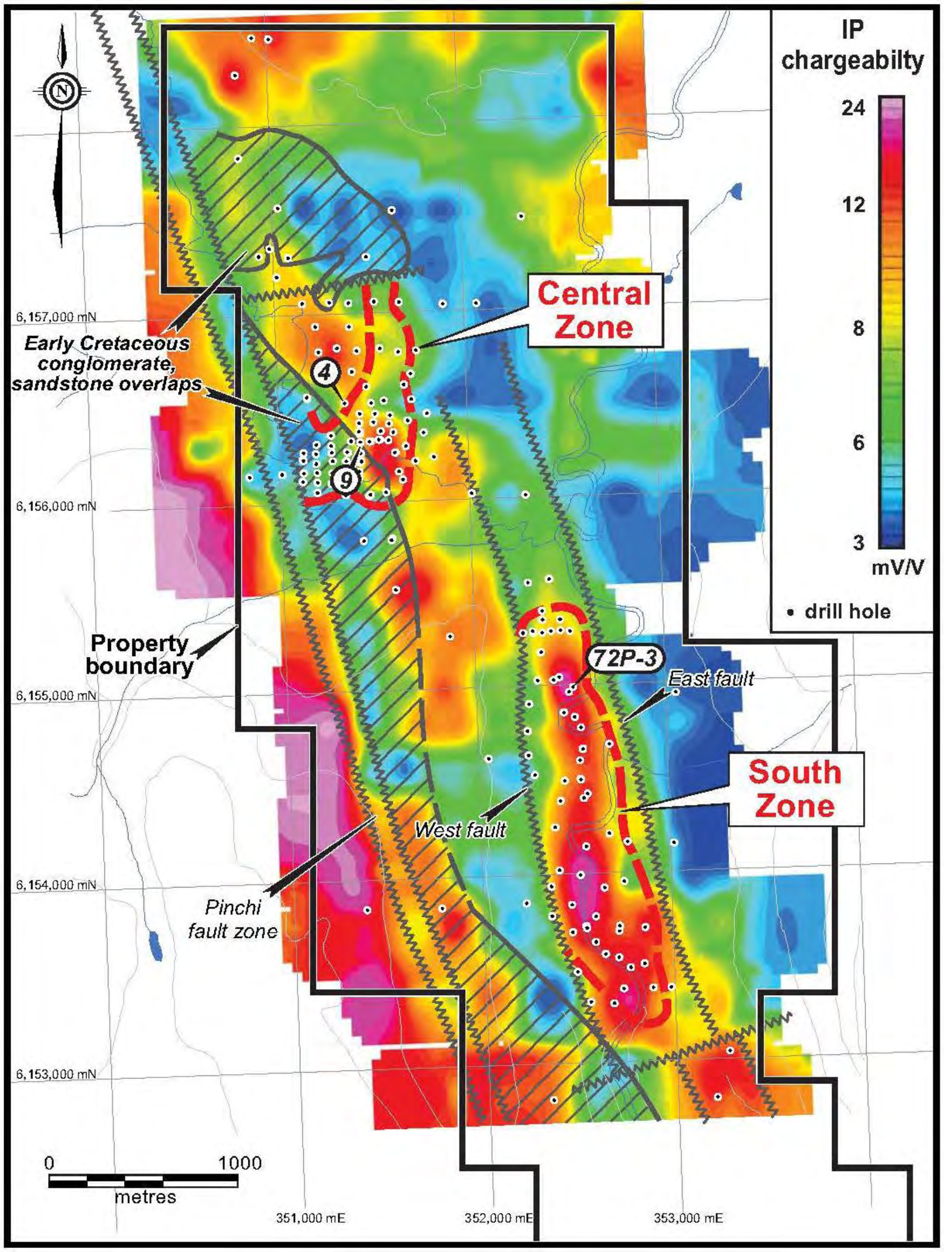
The 2009 drilling program established an exploration model for a structurally controlled porphyry deposit in the South Zone area. Analysis and reinterpretation of geophysical and geological data suggested that potential existed for a structurally bounded domain of mineralization measuring up to 2,900m x 500m. This favourable structural setting was coincident with a +12mV/V chargeability anomaly. Past exploration at Kwanika has demonstrated a strong correlation between chargeability anomalies and copper mineralization.

In August of 2016 Serengeti contracted McElhanney to fly a LIDAR survey over the Central and South zones of the Kwanika project. The resulting data was used to create a high resolution topographic surface.



(Source: Serengeti 2016)

Figure 9-1 Map of the Residual Magnetics



(Source: Serengeti 2016)

Figure 9-2 Map of the IP Chargeability

10 Drilling

In the resource area, a total of 75,412m of diamond drilling in 180 holes was carried out on the Kwanika property from July 2006 to August 2016. Drilling on the Central Zone totalled 57,454m in 122 holes, while drilling at the South Zone totalled 17,958m in 58 holes. There is one additional exploration hole outside of the resource area that is far to the south and not included in the summary. The results of this drilling have achieved three main goals:

- delineated Indicated and Inferred Mineral Resources on the Central Zone deposit, which was initially discovered by Serengeti in late 2006;
- delineated an Inferred Mineral Resource on the South Zone deposit; and
- tested several geophysical anomalies on the Kwanika property to explore for possible extensions of the Central Zone deposit.

All but the first five drillholes were surveyed for downhole azimuth and dip using a Reflex EZ shot tool, generally at 50-60m intervals, either during drilling or upon completion. For all programs from 2006 - 2010, Allnorth Consultants Limited was contracted to carry out a differential GPS (DGPS) survey of the drillhole collar locations on the Kwanika property. Drilling from 2011-2012 were surveyed using handheld GPS units and drilling collars from 2016 were surveyed using a Reflex APS GPS unit.

A LIDAR survey flown in 2016 has been used to verify all collar elevations. SRK compared the drillhole collar elevations to the new 2016 LIDAR surface topography and found that the elevations for some of the holes were not in agreement with the high accuracy surface. SRK adjusted all collars to conform to the 2016 LIDAR topography. The collars were only modified near the resource areas.

All drill core was logged for geological and geotechnical characteristics (geotechnical logging included rock quality designation (RQD), magnetic susceptibility, and specific gravity), and was photographed, sampled, and split by diamond saw or core splitter. The majority of drill core collected by Serengeti on the Kwanika property was NQ (4.76 cm) size. In rare cases, BQ size (3.64 cm dia.) core was drilled when core size had to be reduced due to ground conditions. HQ size (6.35 cm dia.) core was drilled at the top of several holes that were collared in the sedimentary basin in the Central Zone as well as for deep drilling in the 2016 drilling campaign.

SRK inspected the core logging facility at the Kwanika camp and reviewed the core handling procedures, and considers them to be reasonable and consistent with common industry practice. The drilling was observed to be well-managed, using equipment appropriate for the Project. The core is currently stored in conex bins or cross-piled at the Kwanika camp. Figure 10-1 to Figure 10-3 show the drilling for the Central and South Zones in plan and section views.

10.1 Historic Drilling

The South Zone area at Kwanika was drill tested during the period 1965-1991 by 30 historical diamond and percussion drillholes. The historic data are not included in this data compilation. Serengeti has confirmed and expanded this mineralized zone with drilling that replaces the historic data. The historic drillholes prior to 2006 are discussed in the History section and are not included in the resource estimation for the South Zone.

10.2 Serengeti Diamond Drilling Campaigns

The intersections given here are capped average values within the modeled high grade shells.

Phase I: In the summer of 2006 five diamond drillholes (K-06-01 to K-06-05, 659.6m) were drilled to follow up on an IP anomaly. These holes confirmed the copper grade of the previously known mineralization and identified a new zone some distance to the north of the South Zone.

Phase II: In November and December 2006, five diamond drillholes (1,214.7m) were drilled in the vicinity of hole K-06-04, resulting in the discovery hole for the Central Zone, K-06-09 (0.69% Cu and 0.54g/t Au over 111m).

Phase III: Subsequent to the discovery of the Central Zone deposit in the fall 2006/winter 2007, Serengeti initiated the third phase of the diamond drill program to define the new deposit. An all-weather, 30-man camp was constructed in March 2007. Coast Mountain Geological Ltd. (CMG), a Vancouver-based geological consulting firm, was contracted to manage the drill project. Diamond drilling was carried out by Cyr Drilling International Ltd. of Winnipeg, Manitoba.

The Phase III drill program on the Kwanika property was conducted from March 2007 to August 2008. During this period, a total of 113 diamond drillholes, with an aggregate length of 53,646.3m, were drilled on the property. These drillholes were primarily designed to delineate the mineralization in the Central Zone, explore the South Zone, as well as to test geophysical anomalies and possible extensions to the Central Zone mineralization.

Examples of significant drill intersections encountered include K-07-15 (0.60% Cu and 0.73 g/t Au over 323m) and K-08-113 (0.73% Cu and 1.36g/t Au over 280.5m). The significant grades and widths of copper and gold mineralization encountered confirmed the existence of a previously unknown porphyry copper-gold deposit.

The South Zone drilling campaign during 2007 and 2008 comprised 16 diamond drillholes for an aggregate length of 4,935.4m. Several holes in the South Zone encountered a strongly mineralized copper-gold-molybdenite-silver porphyry system that had not been fully recognized by past exploration. Examples of drill intersections include K-08-110 (0.26% Cu, 0.14g/t Au, and 0.007% Mo over 233m) and K-08-116 (0.34% Cu, 0.09g/t Au, and 0.012% Mo over 129m).

Phase IV: This phase of drilling was conducted from June to September 2009. During this period, a total of 17 diamond drillholes were completed on the property with an aggregate length of 6,249.1m. This phase of exploration was primarily designed to follow up several encouraging intersections obtained during 2008 drilling in the underexplored South Zone area. Significant drill intersections encountered included:

K-09-124 (0.41% Cu, 0.05g/t Au, and 0.019% Mo over 231m)

K-09-126 (0.51% Cu, 0.14g/t Au, and 0.023% Mo over 143m)

Drilling was successful in delineating and expanding a copper-gold-molybdenite-silver resource in the South Zone.

Phase V: The Phase V drill program on the Kwanika property was conducted from June to August 2010. During this period, a total of 28 diamond drillholes were completed on the property with an aggregate length of 7,619m. This phase of exploration consisted of step-out drilling intended to expand the existing South Zone resource reported in March 2010. A series of in-fill drillholes were also completed in order to gain further understanding of the mineralization associated with the West Fault. The Phase V drilling was successful in both expanding the mineralized envelope to the north of the historical resource area of the South Zone deposit and adding important geological information to the exploration model.

Phase VI: From June to July of 2011 a total of 5 drillholes were completed with an aggregate length of 1,724m. This phase of exploration was carried out to test IP-chargeability and Ah-horizon soil exploration targets to the east and northeast of the Central Zone.

Phase VII: The Phase VII drilling program was completed in August to September of 2012. During this period, a total of 4 drillholes were completed to an aggregate length of 1,494m. Holes K-12-174 to K-12-176 tested IP-chargeability targets to the north of the Central Zone deposit. One additional drillhole was drilled at the south end of the property to test a deep IP-chargeability anomaly. Three line kilometres of IP was also completed in 2012 to test the existence of a chargeability anomaly to the east of the Central Zone resource area.

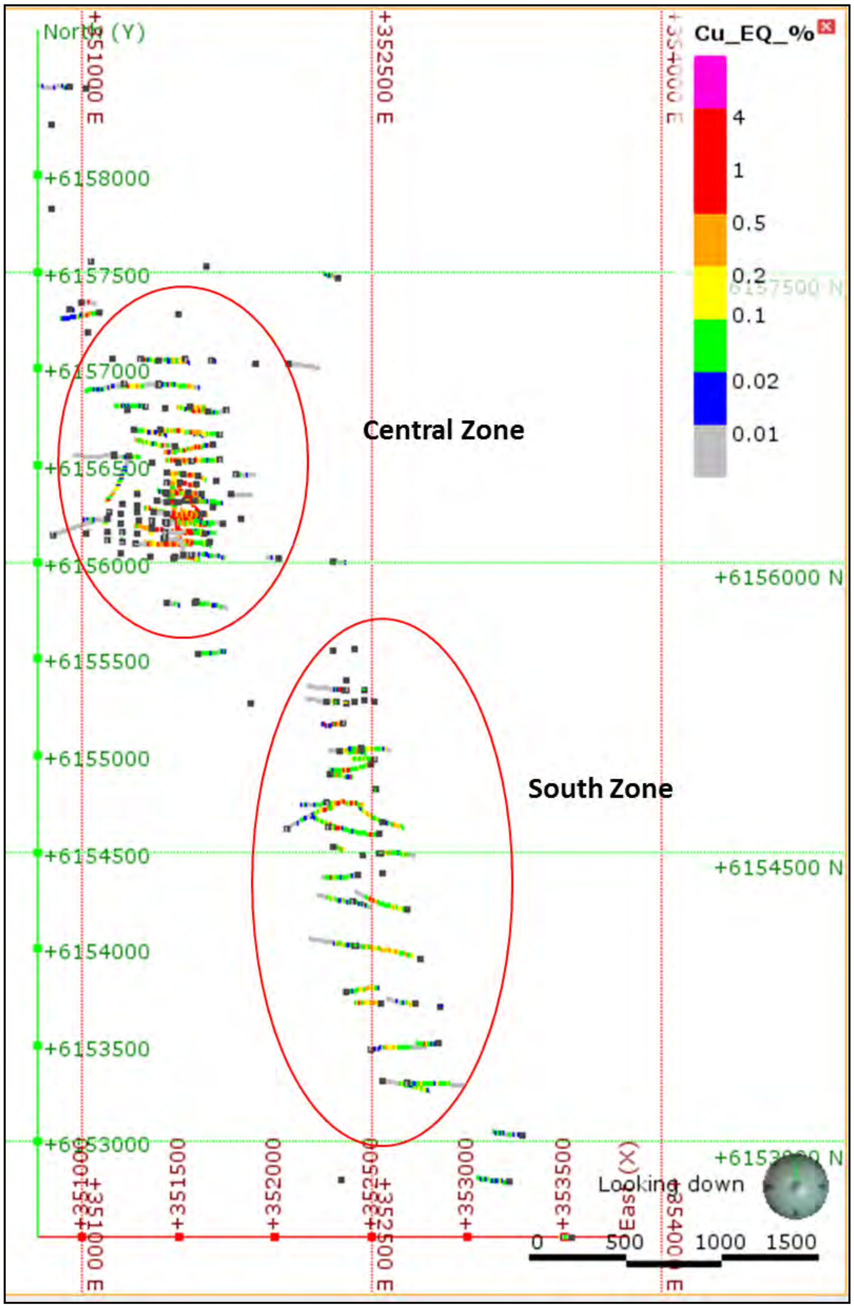
Phase VIII: This drilling campaign took place from July to August of 2016 during a joint exploration program funded by Daewoo Minerals Canada Corporation. A total of 3 deep drillholes where completed with an aggregate length of 2,445m to test the deep roots of the Central Zone as well as an IP-chargeability anomaly to the north of the Central Zone. Hole K-16-177 penetrated the Central Zone producing significant results within the deposit. Highlights include:

- K-16-177 (0.85% Cu, 1.14g/t Au over 259m)

K-16-179 tested the northern deep extent of the Central Zone and showed significant grade at depth indicating the potential for further deep exploration. K-16-178 tested the northern deep chargeability anomaly and intersected significant lengths of highly altered andesite with moderate mineralization.

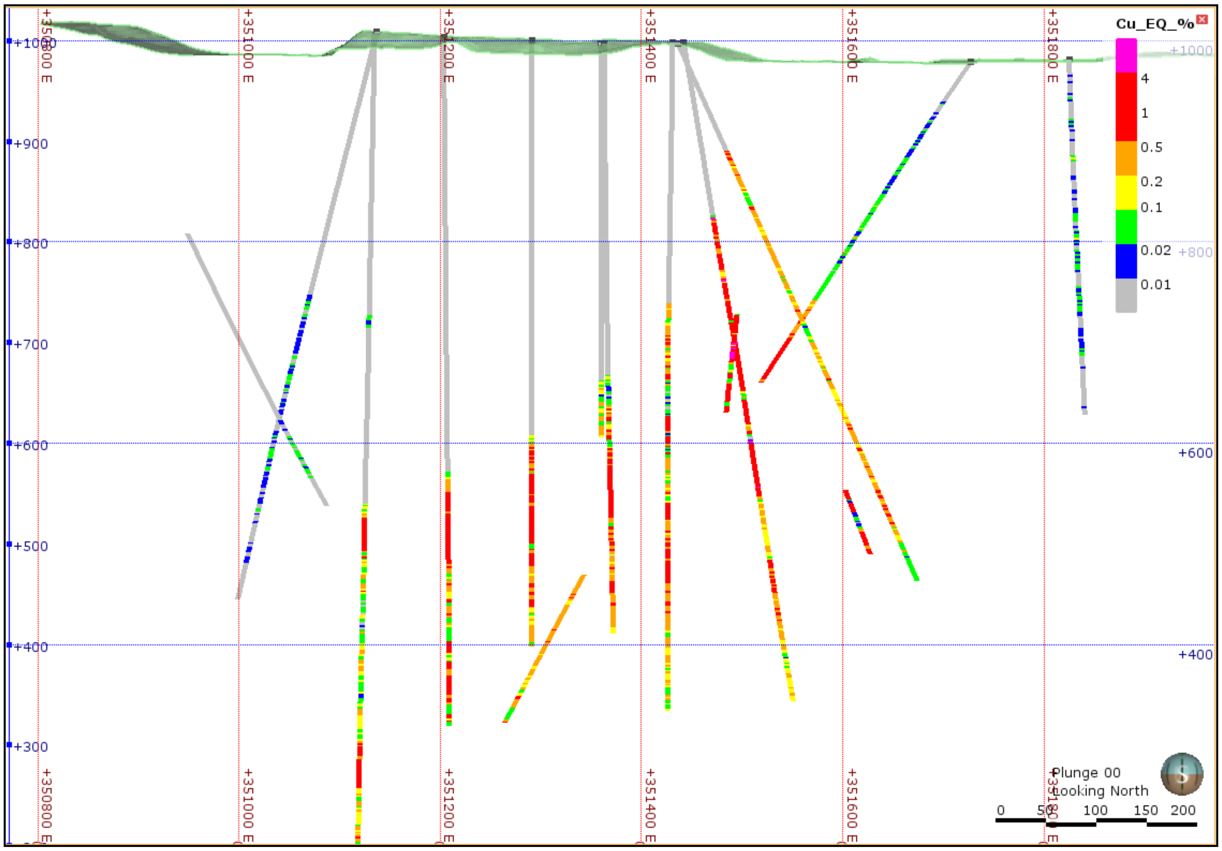
Figure 10-1 shows a plan view of the drilling for the Central and South Zones. Figure 10-2 and Figure 10-3 show drilling cross sections through each of the zones.

SRK reviewed the drilling and is of the opinion it is suitable for use in the resource statement.



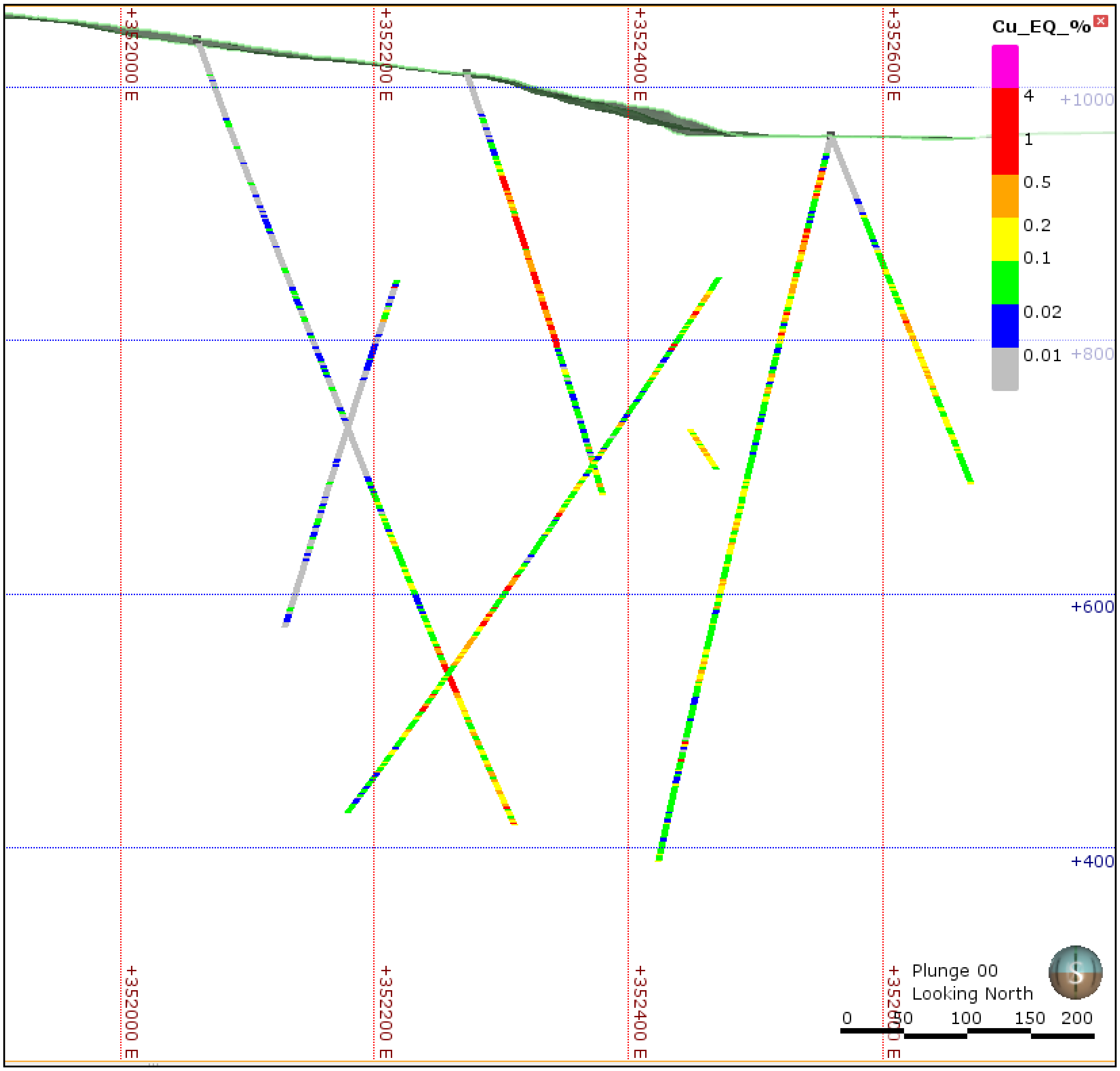
(Source: SRK 2016)

Figure 10-1 Map showing the Distribution of Drilling



(Source: SRK 2016)

Figure 10-2 Cross section in the Central Zone at 6156200 N, looking north, 50m thick section



(Source: SRK 2016)

Figure 10-3 Cross section in the South Zone at 6154670 N, looking north, 150m thick section

11 Sample Preparation, Analyses and Security

11.1 Core Logging

The drill programs at the Kwanika property were managed by Coast Mountain Geological (CMG) from 2006 to 2008 and by Serengeti staff from 2009 to 2016. The methodology for core handling and sampling, since 2006, is described below.

The core was transported from the drills to the camp at each drilling shift change, once in the morning and once in the evening. Each morning, the core drilled during the previous day was quick-logged by a geologist. The quick log involved a brief description of lithology, alteration, and mineralogy, as well as a description of any significant structural characteristics. A copper grade based on visual approximations of mineralization was assigned to each interval, where:

- grade 0 indicated <0.1% Cu,
- grade 1 indicated 0.1% Cu to 0.25% Cu,
- grade 2 indicated 0.25% Cu to 0.5% Cu, and
- grade 3 indicated >0.5% Cu.

From February 2008 to August 2016, a Niton handheld X-ray fluorescence (XRF) tool was used to aid in the initial grade estimation.

Once quick-logged, the core was stacked on-site pending detailed logging. The logging included a detailed description of the lithology, alteration, structural features, and mineralogy. Sample intervals were divided based on contacts between these characteristics, to a maximum of two metres. The overlying conglomerate, encountered at the top of many of the holes drilled in the Central Zone, was not sampled unless copper mineralization was observed. Once the sampled intervals were established, each interval was assigned a unique sample number. Alternating blanks, reference standards, and core duplicates were inserted every 15 samples. Each sample was identified with a three part tag. One tag was kept in the camp library for reference. The other two tags were stapled to the base of the core box with a written metal tag at the start of the corresponding interval. Mineralized core was split on-site using two diamond saws, while select, lower grade core was split using a blade splitter.

Geotechnicians determined the recovery, rock quality designation (RQD), specific gravity, magnetic susceptibility and conductivity of the rock. The recovery and RQD was determined by measuring the total length between two blocks, which represents one tube of core. This length should normally be ten feet, or 3.048m. The percent recovery was determined by dividing the length measured by the expected length. Next, the geotechnician added the total length of pieces of core which were longer than 10cm within the standard run length. This length was divided by the total length measured to give the percent RQD.

The magnetic susceptibility and conductivity was determined using a multi-parameter probe. A reading was taken every 1.5m directly on the core surface. Recovery and RQD was completed for the full length of the holes, while specific gravity and magnetic susceptibility were measured only for sampled intervals.

11.2 Core Sampling

After logging, the core was split under the supervision of project geologists. Two diamond core saws and a mechanical splitter were used to split the core. Most samples were taken with the saws, however, in zones observed by the geologists to be low grade, the mechanical splitter was used. The diamond core saws used clean, uncirculated water to aid in cutting, and were cleaned regularly to avoid contamination. The mechanical splitter was cleaned thoroughly after each sample was split.

Once split, half of the core was left in the core box for reference, and the other half was sent for analysis. Samples were placed in labelled plastic bags with the corresponding sample tag and sealed with zip ties. The standards and blanks were also put in a labelled plastic bag with a sample tag. These plastic bags were placed in numbered rice sacks, which were sealed by heavy duty zip ties and given a numbered tamper-proof security tag.

Samples were transported via truck by a local third party expediting and freight company. To ensure that samples were not tampered with during transport to the laboratory, the number of each security tag and its associated rice sack number were recorded by the geologist at the Kwanika site. A list of each bag and its unique security tag number was forwarded to GDL/ACME/ACT, which then confirmed that each security tag matched its correct rice sack.

11.3 Core Preparation and Analysis

11.3.1 Sampling by Global Discovery Labs (2006 - June of 2009)

From 2006 to 2009 all assays from the Kwanika Project were sent to Global Discovery Labs (GDL) in Vancouver, British Columbia. GDL did not have ISO accreditation but did participate in the Proficiency Testing Program for Mineral Analysis Laboratories (PTP-MAL). PTP-MAL is an ISO 9001:2000 accredited program that is operated by the Canadian Certified Reference Materials Project (CCRMP), and meets recognized international standards for proficiency testing providers.

Samples sent to Global Discovery Labs were passed through a two stage crushing process reducing the material to 90% minus 2mm in size. The crushed material was split in a Jones Riffle to a subsample measuring 250g to 300g. The samples were pulverized in a ring-and-puck mill to 95% passing a 150 mesh screen.

The shipped samples were divided into two groups: samples with an assumed grade less than 0.2% Cu and samples with an assumed grade of greater than 0.2% Cu, as determined by the Project geologist. All samples were subject to aqua regia digestion and then run for 28 elements using Inductively Coupled Plasma (ICP) spectrometry (Package ICP-OES). Samples with greater than 2,000 ppm Cu or 100 ppb Au were rerun for Au, Cu, Pb, Zn and Fe by Atomic Absorption (AA). Dissolution of the samples for the base metal determinations was done using aqua regia, while for the gold it was aqua regia followed by 2, 6-Dimethyl-4-heptanone.

Samples assaying greater than 0.2g/t Au in the ICP or AA analyses were rerun using fire assay and AA finish. These assays were carried out on a 30g (one assay-ton) aliquot.

11.3.2 Sampling by Acme Labs (July of 2009 - 2012)

From 2009 to 2012, sampling was carried out by Acme Labs which acquired GDL in July of 2009. Acme Laboratories held ISO 9001 accreditation during this time. The assay prep and processing remained the same from 2009-2012 after Acme took over GDL. Refer to section 10.3.1.

11.3.3 Sampling by Activation Labs (2016)

During the 2016 drilling program, Activation Labs of Kamloops, British Columbia was used to carry out assaying of the Kwanika project. Activation Labs is ISO 17025 accredited laboratory.

Once samples were received at the lab they were weighed, and then crushed up to 90% passing 10 mesh, riffle split (250 g) and then pulverized to 95% passing minus 150 mesh including cleaning of the pulveriser bowl after each sample. Prepared samples were assayed for a suite of 38 elements including Selenium by aqua regia digestions and Inductive Coupled Plasma (ICP) spectrometry. All Au analysis was carried out by 30 g fire assay and Atomic Absorption.

Samples greater than 2500ppm Cu were rerun by assay grade aqua-regia digestion and ICP spectrometry. Au results greater than 3.0g/t were rerun by 30g Fire Assay and a gravimetric finish.

11.4 Specific Gravity Data

Specific gravity data was collected using whole core measurements carried out onsite before core was sent for assay. This was done using a water immersion method. This data was recorded in a density log within the drilling template. Specific gravity was not collected for some early drilling campaigns.

The specific gravity was determined by taking every fourth sample and first determining the weight of that sample in air and then the weight of the sample in water. The volume of the sample was determined by subtracting its weight in air from its weight in water. Specific gravity was found by dividing the sample's weight in air by its volume.

11.5 Quality Assurance and Quality Control Programs

An independent assay QA/QC program has been in place throughout the drilling campaigns carried out by Serengeti. Control samples were inserted at a rate of two commercial Certified Reference Materials (CRMs), one blank, and one duplicate for every 60 core samples (for a frequency of one QA/QC sample for each 15 core samples). Serengeti used CRMs prepared by CDN Resource Labs Ltd. (CDN) of Langley, BC. One standard, CGS-18, the manufacturer has not fully certified the Au assay, and this standard is deemed "Provisional". CDN warns that provisional standards cannot be used to monitor accuracy with a high degree of certainty.

Blank material comprised packets of pulverized barren material, similar to the standards.

Duplicates were produced by Serengeti at the Kwanika property by cutting the initial core sample interval in half and leaving one half in the core box. The half that was to be sent to the laboratory for analyses was then quartered by cutting each piece in half again and putting one side of the core in one sample bag and the other side of the core in a separate sample bag.

11.6 SRK Comments

In the opinion of SRK the sampling preparation, security and analytical procedures used by Serengeti are consistent with generally accepted industry best practices and are therefore adequate.

12 Data Verification

Serengeti has conducted and independent QA/QC sampling program on the Kwanika Project. QA/QC samples were included in the sample stream for both the Central and South zones. SRK has compiled and reviewed the database and the results of the QA/QC sample program. A total of 100% of the resource database was validated against assay certificates provided by the lab. The QA/QC samples include blanks, standard reference material, and field duplicates.

12.1 Verifications by SRK

12.1.1 Site Visit

Chad Yuhasz, PGeo, conducted a site visit on August 7th to 9th 2016. Procedures and protocols were discussed and reviewed, however no independent sampling for verification purposes was undertaken.

12.1.2 Database Validation

SRK validated the collar, survey, and assay data for both the Central and South zones. Nearly all of the drillholes were surveyed with high accuracy equipment. When compared to the historic topography surface, the drillholes often fell below the surface. This is attributed to elevation errors in the previously generated "tree-top" photogrammetric basemap used to create the historic topographic surface. To rectify this, a LIDAR survey was flown in September 2016 and the collars were migrated to the new LIDAR surface to obtain more accurate elevation values. The drillhole traces were visually checked to validate the downhole surveys.

The assay database was compared against the assay certificates. The assay certificates from 2006 to 2015 were provided by Bureau Veritas and for the 2016 assays from Activation Laboratories. Bureau Veritas has had three owners; the lab was originally Teck Global Discovery Labs, then Acme Labs, and finally Bureau Veritas Labs. QA/QC samples were included during all years of drilling, from 2006 to 2016. A total of 100% of the assay values were validated and only minor transcription errors were found. All errors were corrected in the assay database before use for the resource.

12.1.3 Verifications of Analytical Quality Control Data

QA/QC samples were incorporated into the sample stream in the field. QA/QC samples were included as blanks, standard reference material, and field duplicates.

Blank and standard reference materials were provided by CDN Resources Laboratories Inc. Field duplicates represented quartered core samples.

Standard CDN-CSG-18 has been removed from the QA/QC analysis. The results from the lab are reported as the same number for a large portion of the samples. SRK discusses this standard in the 2009 NI 43-101 report and was able to correct it with the lab as it appears to be a data entry error (Rennie, 2009). SRK was not able to confirm SRK's results and has removed the standard from the QA/QC analysis for this review.

12.2 Central Zone

Table 12-1 shows a summary of the QA/QC samples. The samples were inserted into the sample stream at approximately 1 in every 35 assays.

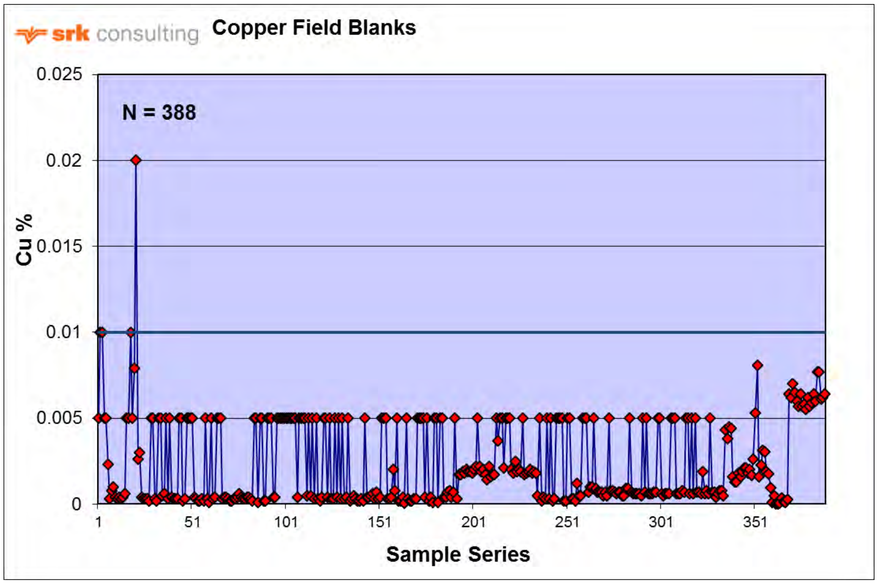
Table 12-1 Central Zone QA/QC Sample Summary

Sampling Program	Count	(%)
Sample Count	21,445	
Field Blanks	388	2%
Standard Samples	668	3%
Field Duplicates	377	2%
Total QC Samples	**1,433**	**7%**

12.2.1 Blanks

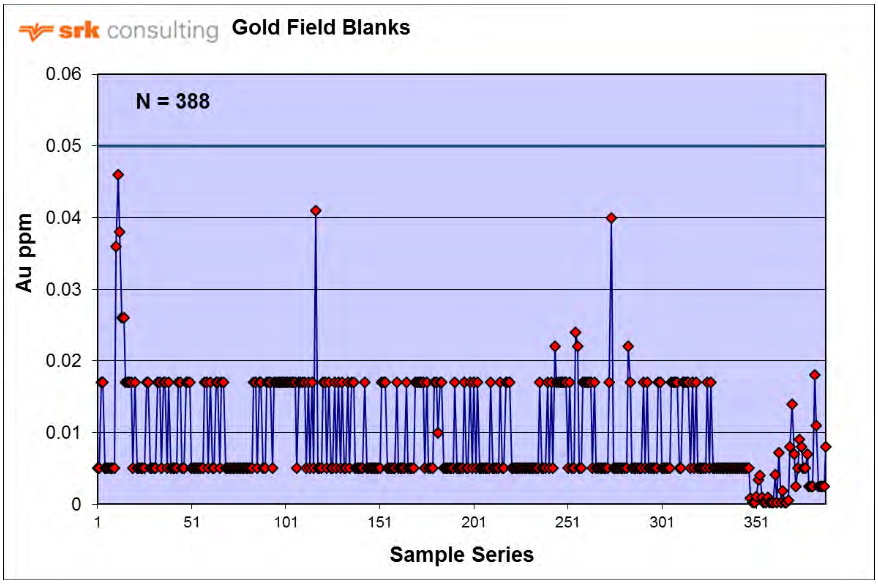
A total of 388 field blanks were included in the QA/QC samples from 2006 to 2016. The blank material is a pre-crushed prepackaged blank from CDN Resources Laboratories Inc. Blanks perform very well and there is only one sample for copper returning a value greater than five times the detection limit. Gold and silver do not have any failures.

The blank sample performance is acceptable. Figure 12-1 and Figure 12-2 show the copper and gold blank charts.



(Source: SRK 2016)

Figure 12-1 Copper Blanks



(Source: SRK 2016)

Figure 12-2 Gold Blanks

12.2.2 Standards

A total of 668 standard reference material samples were included in the QA/QC samples from 2006 to 2016. The standards are prepackaged envelopes of pulverized material from CDN Resource Laboratories Ltd. Of the nine standards utilized, three have more than 20 samples. Only standards with more than fifteen samples will be discussed in this report. All standard charts are provided in Appendix A. Figure 12-3 and Figure 12-4 show the plotted results for CDN-CSG-11 for copper and gold. Table 12-2 shows the standards used. Silver does not represent a significant portion of the resource and has not been reviewed in detail.

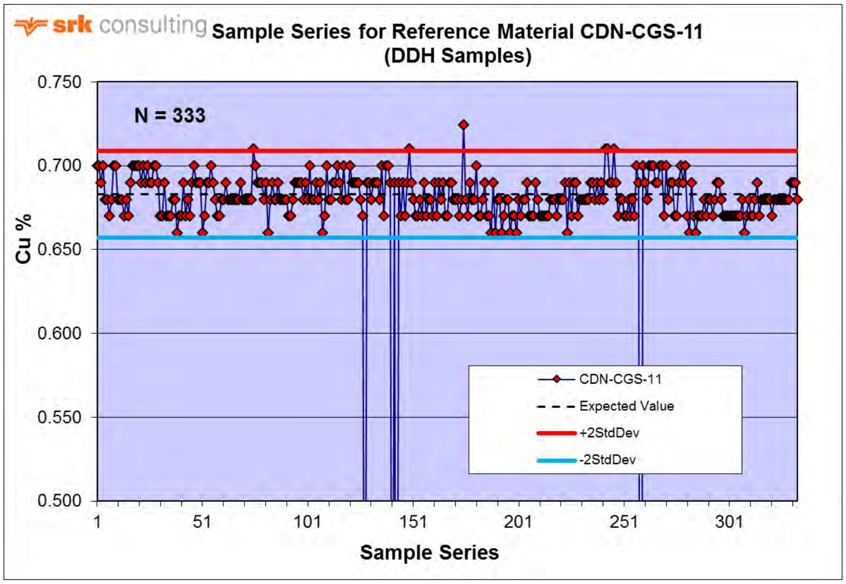
Table 12-2 Standard Reference Material Samples

Standards	Count
CDN-CGS-11	333
CDN-CGS-12	260
CDN-CGS-15	7
CDN-CGS-22	10
CDN-CGS-23	4
CDN-CM-23	21
CDN-CM-36	16
CDN-CM-5	13
CDN-CM-7	4

Standards CDN-CSG-11 and CDN-CSG-12 performed well for copper with only 3% of the samples falling outside of two standard deviations of the expected value and failing. Standard CDN-CM-23 performed very well for copper with no samples failing. Failed samples were not reanalyzed or reviewed for mislabeled labeled samples.

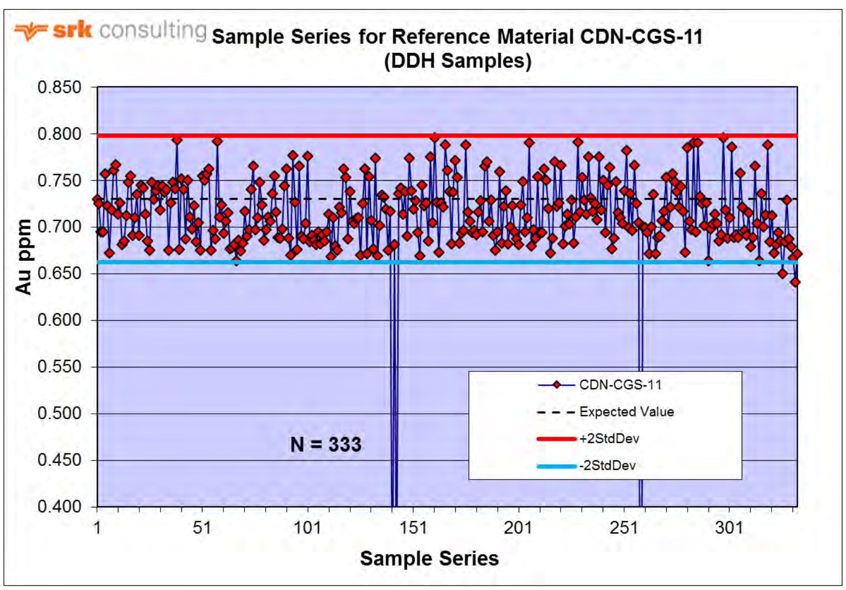
Standards CDN-CSG-11, CDN-CSG-12, and CDN-CM-23 all performed very well for gold with 5% or less of the samples falling outside of two standard deviations of the expected value and 2% or less of the samples outside of three standard deviations.

The standard sample performance is acceptable.



(Source: SRK 2016)

Figure 12-3 Standard CDN-CGS-11 for copper



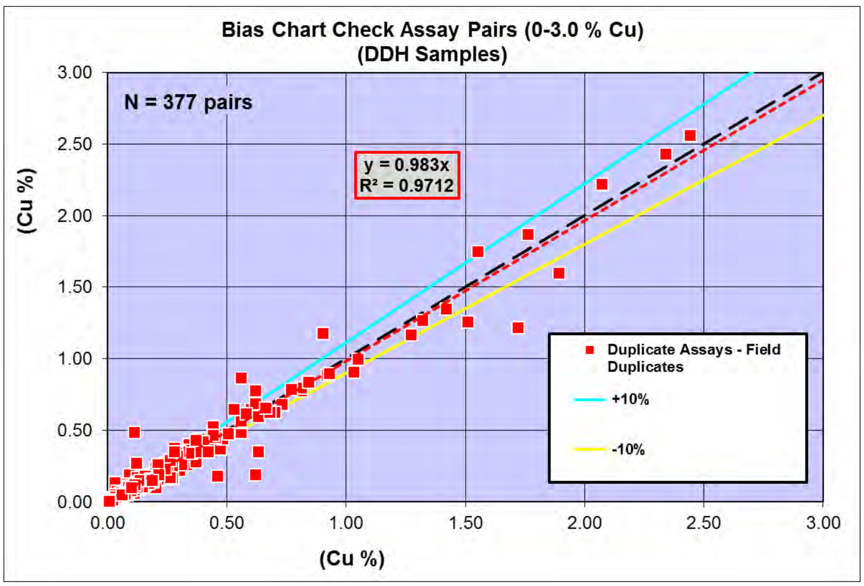
(Source: SRK 2016)

Figure 12-4 Standard CDN-CGS-11 for Gold

12.2.3 Field Duplicates

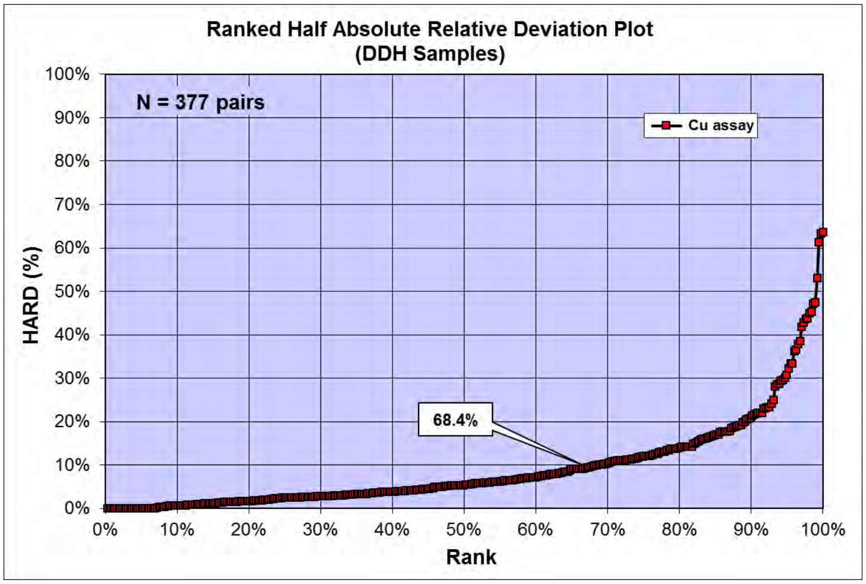
Serengeti sent field duplicates to the lab as part of the QA/QC sample procedure. The field duplicates are quarter core sawn samples. SRK has reviewed the duplicate samples for copper and gold. Figure 12-5 to Figure 12-8 show the paired data for copper and gold. For copper nearly 70% of the duplicates are within 10% of the original assay result. For gold nearly 65% are within 10%.

The field duplicate sample performance is acceptable.



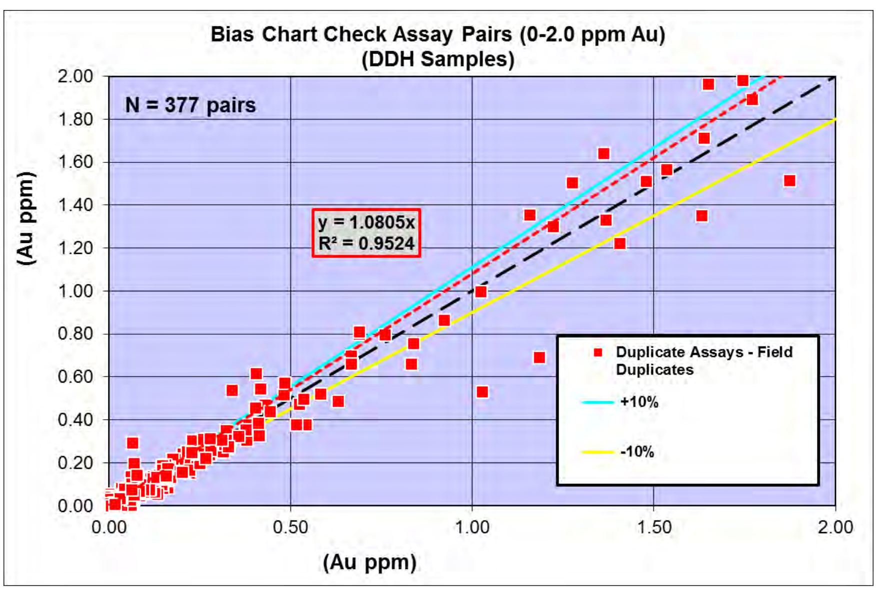
(Source: SRK 2016)

Figure 12-5 Paired Original and Duplicate Copper Samples



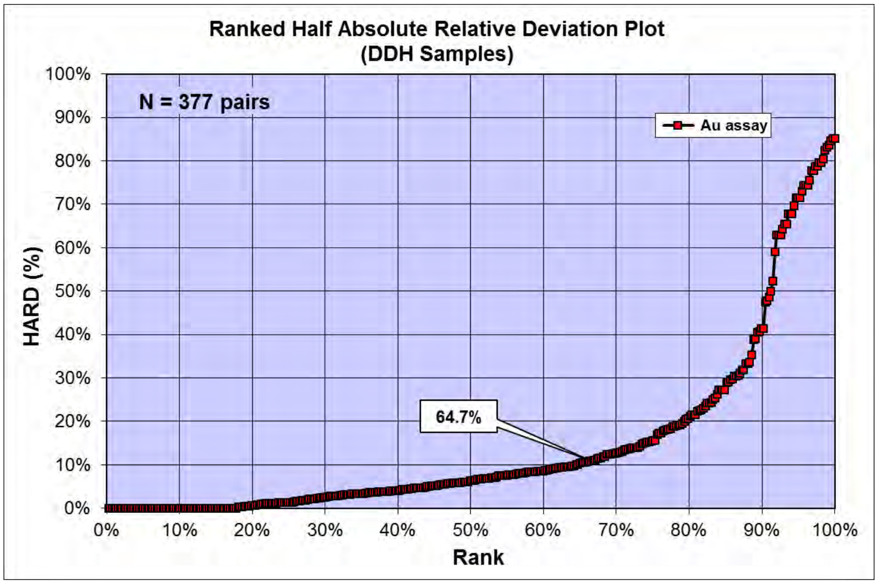
(Source: SRK 2016)

Figure 12-6 Copper Field Duplicates Plotted as Relative Deviation



(Source: SRK 2016)

Figure 12-7 Paired Original and Duplicate Gold Samples



(Source: SRK 2016)

Figure 12-8 Gold Field Duplicates Plotted as Relative Deviation

12.3 South Zone

The QA/QC samples consist of blanks, standards, and duplicates. Table 12-3 shows a summary of the QA/QC samples. The samples were inserted into the sample stream at approximately 1 in every 35 assays.

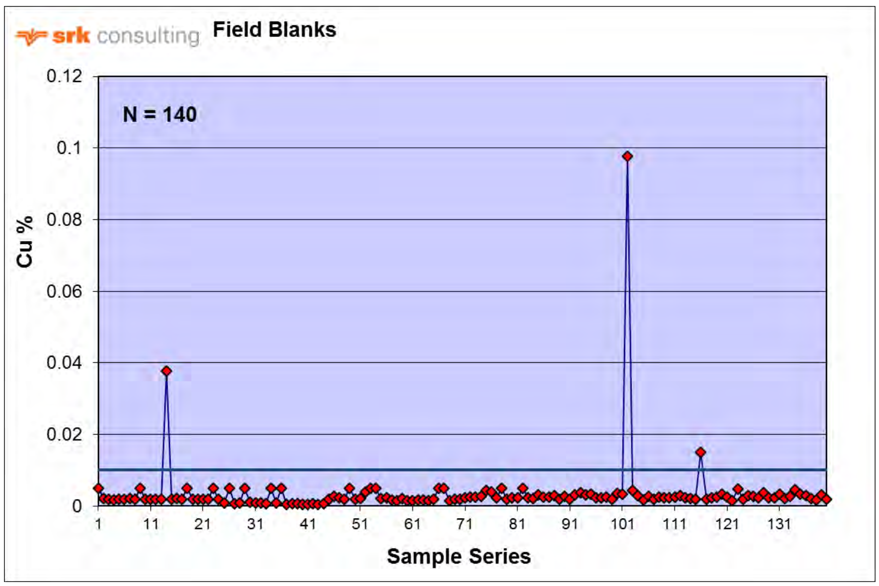
The 2011 SRK report describes an umpire sample program from 2010 to 2011 (Rennie, 2011). The samples were sent to ALS laboratories. SRK does not have all of the source data to review the umpire samples independently and has not included it in this report.

Table 12-3 South Zone QA/QC Sample Summary

Sampling Program	Count	(%)
Sample Count	8,065	
Field Blanks	140	2%
Standard Samples	198	3%
Field Duplicates	139	2%
Total QC Samples	477	6%

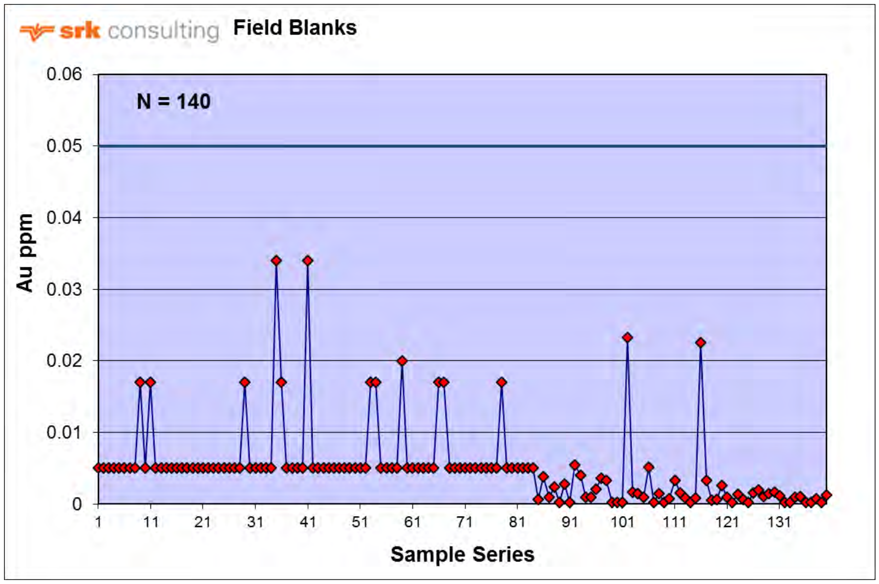
12.3.1 Blanks

A total of 140 field blanks were included in the QA/QC samples from 2008 to 2016. The blank material is a pre-crushed prepackaged blank from CDN Resources Laboratories Inc. Blanks perform very well overall. There are only three copper assays returning values above five times the detection limit. Gold and silver do not have any failures. Molybdenum has two samples failing. Figure 12-9 and Figure 12-10 show the copper and gold blank charts.



(Source: SRK 2016)

Figure 12-9 Copper Blanks



(Source: SRK 2016)

Figure 12-10 Gold Blanks

12.3.2 Standards

A total of 198 standard reference material samples were included in the QA/QC samples from 2008 to 2016. The standards are prepackaged envelopes of pulverized material from CDN Resource Laboratories Ltd. All standard charts are provided in Appendix A . Figure 12-11 and Figure 12-12 show the plotted results for CDN-CSG-11 for copper and gold. Table 12-4 shows the standards used. Silver and molybdenum are not a significant portion of the resource and have not been reviewed in detail.

Table 12-4 Standard Reference Material Samples

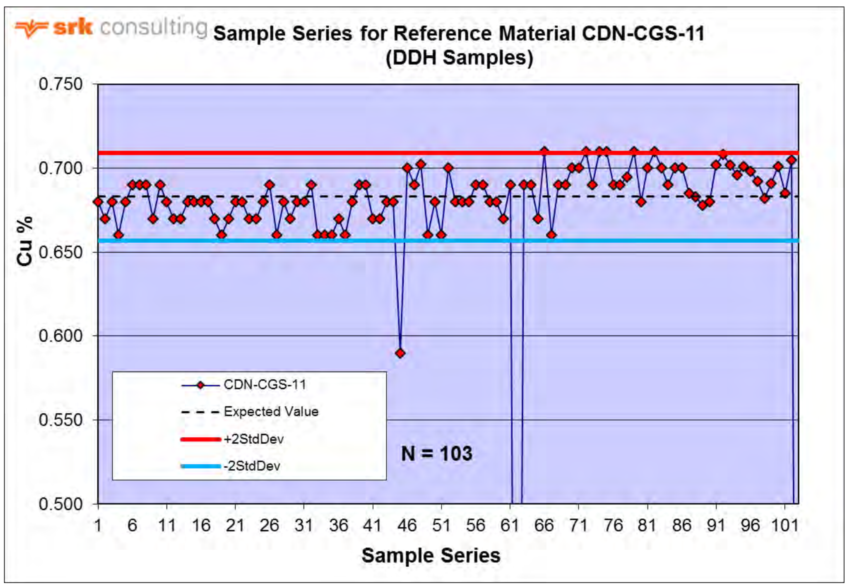
Standards	Count
CDN-CGS-11	103
CDN-CGS-12	20
CDN-CGS-23	36
CDN-CM-7	39

Standard CDN-CSG-11 and CDN-CM-7 perform well for copper with only 3% of the samples outside of two standard deviations from the expected value. Standard CDN-CSG-12 also performed acceptably with 5% of the samples falling outside of two standard deviations.

Standard CDN-CGS-23 did not perform well for copper with 39% of the samples falling outside of two standard deviations of the expected value. This sample overall reports lower than expected values.

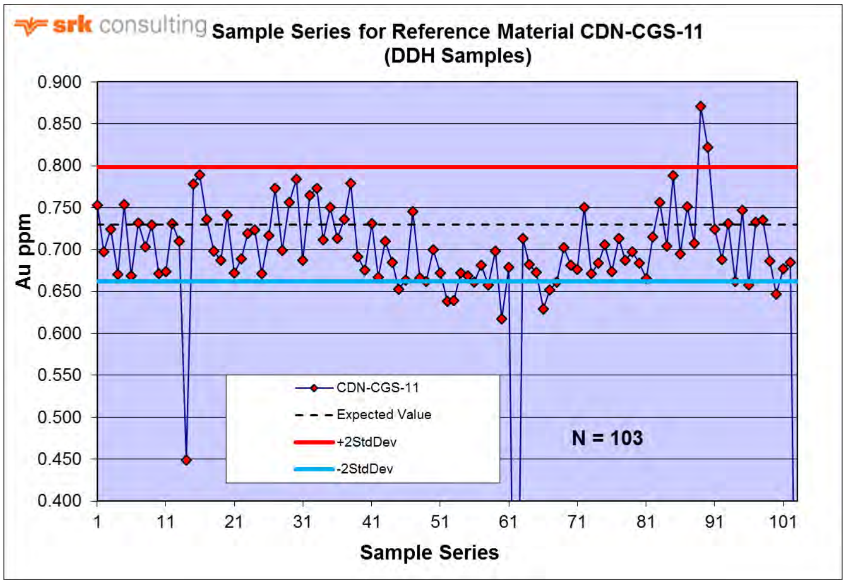
For gold, standards CDN-CSG-11 and CDN-CSG-12 performed very well with 5% or less of the samples falling outside of three standard deviations of the expected value. Standards CDN-CGS-23 and CDN-CM-7 had more samples with assays outside of three standard deviations, with 6% and 10% respectively.

The standard sample performance is acceptable.



(Source: SRK 2016)

Figure 12-11 Standard CDN-CGS-11 for Copper



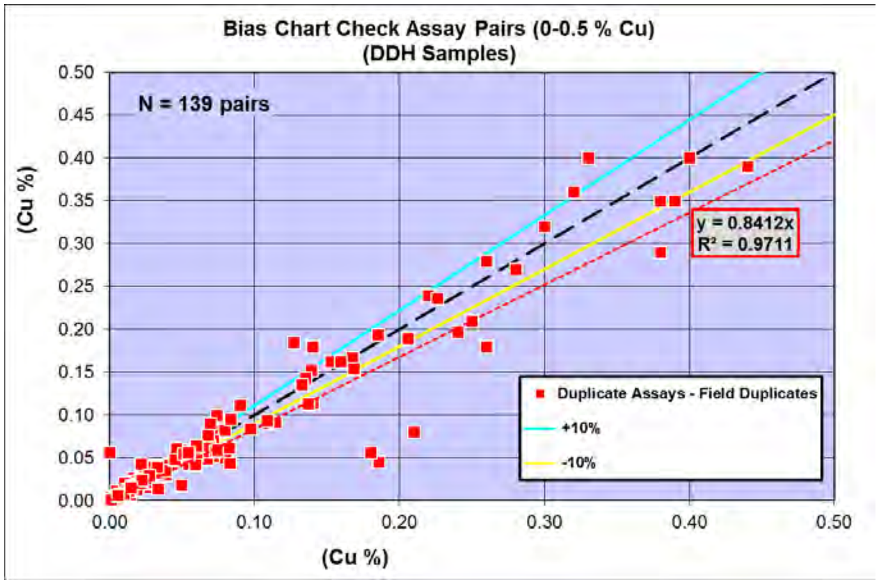
(Source: SRK 2016)

Figure 12-12 Standard CDN-CGS-11 for Gold

12.3.3 Field Duplicates

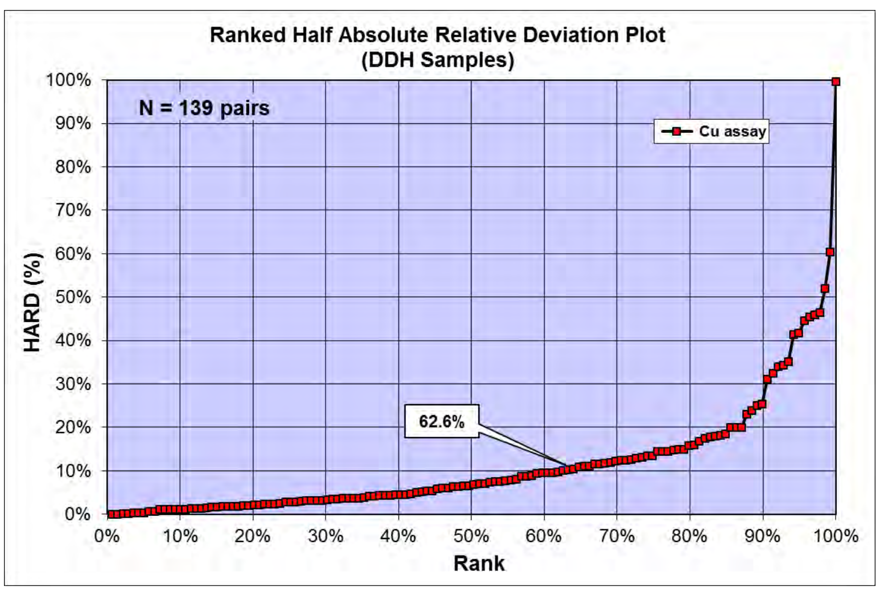
The field duplicates are quarter core sawn samples. SRK has reviewed the duplicate samples for copper and gold. Figure 12-13 to Figure 12-16 show the paired data for copper and gold. For copper approximately 60% of the duplicates are within 10% of the original assay result. For gold nearly 65% are within 10%.

The field duplicate sample performance is acceptable.



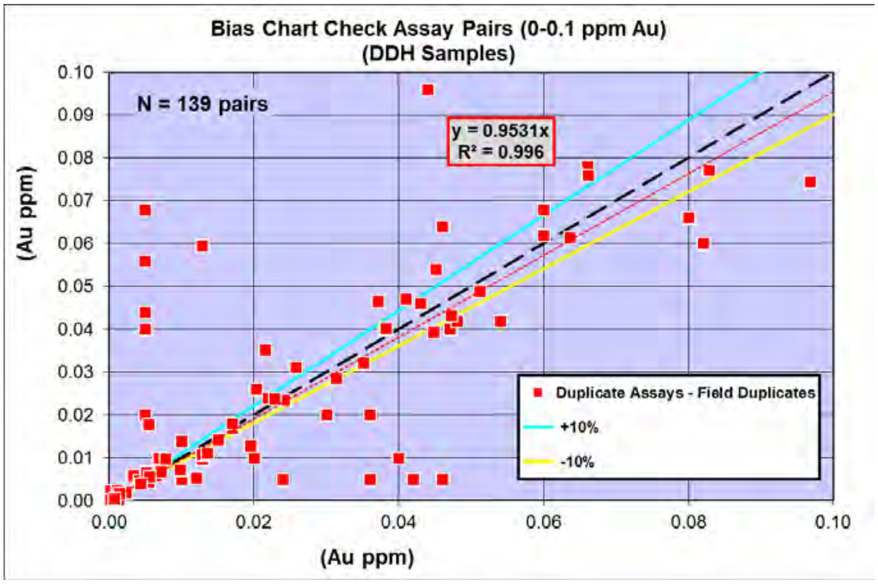
(Source: SRK 2016)

Figure 12-13 Paired Original and Duplicate Copper Samples



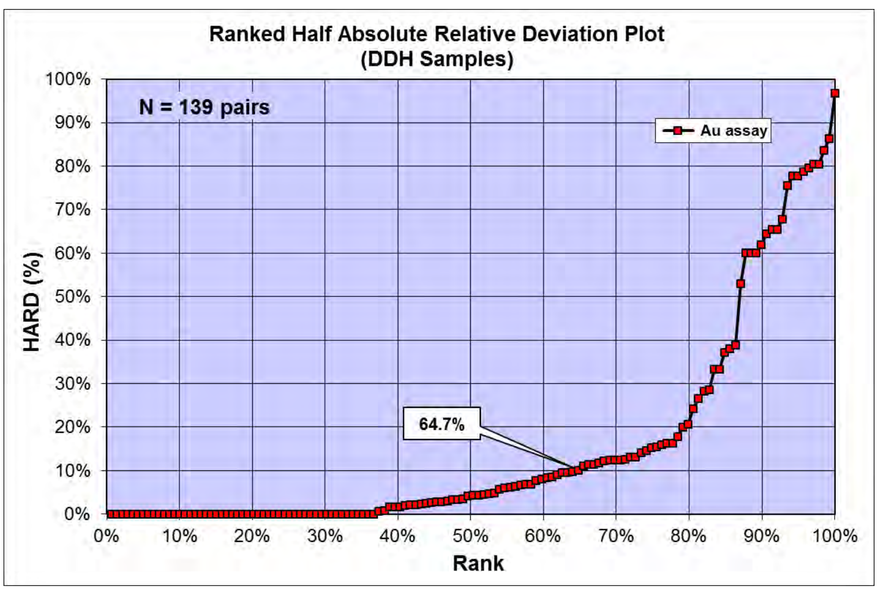
(Source: SRK 2016)

Figure 12-14 Copper Field Duplicates Plotted as Relative Deviation



(Source: SRK 2016)

Figure 12-15 Paired Original and Duplicate Gold Samples



(Source: SRK 2016)

Figure 12-16 Gold Field Duplicates Plotted as Relative Deviation

12.4 SRK Comments

SRK has determined that the QA/QC sample program is acceptable and shows no bias. SRK recommends an increase in QA/QC samples to 5% each for blanks, standards, and field duplicates, as well as an ongoing umpire lab check program. SRK also recommends constant monitoring of the QA/QC data and re-assaying the batches of samples when QA/QC samples fail.

13 Mineral Processing and Metallurgical Testing

Copper-gold mineralization in Kwanika has been identified as two main zones, Central Zone, and South Zone. Serengeti conducted preliminary metallurgical testing on samples from the Central Zone. Metallurgical testing of the South Zone has not been conducted.

An exploratory metallurgical test program was commenced on November 3, 2008 with SGS Metallurgical Services Ltd. (SGS). A total of 52 samples weighing 186kg were collected by Serengeti personnel from the Central Zone and sent to SGS where equal amounts of each sample were used to construct a 120kg master composite, with the remaining material being stored for later testing. The master composite sample assayed 0.66% Cu and 0.76g/t Au.

The Central Zone test work included chemical and mineralogical analyses, Bond Ball Work Index testing, gravity concentration, batch rougher and cleaner flotation tests, and a locked cycle flotation test. Eleven batch flotation tests included five rougher tests, and six cleaner tests. Primary grind P80 ranged from 133m to 75m and regrind P80 ranged from 32m to 20m. Highlights from the Central Zone test work include the following:

- Copper mineralization is mostly chalcopyrite, with minor content of bornite and other copper sulphides.
- Mineralization is finely disseminated
- Gold appears to be associated with sulphides including copper species and pyrite.
- Bond Work Index is approximately 16kWh/tonne.

The 2008 exploratory metallurgical test work concludes that a conventional concentration process with a primary grind of 80% passing 75µm, and regrinding of the rougher concentrate to 80% passing 26µm before feeding a three-stage cleaning flotation circuit could recover 88.5%Cu, and 65.2%Au with a concentrate grade of 27.7%Cu, and 20.9g/t Au. The final copper concentrate was found to be very clean, and the content of penalty elements such as As, Bi, Sb, and Hg is very low. Cleaner flotation tests demonstrate that a reduction of concentrate grade to 24% would significantly increase copper and gold recovery.

Examination of mass pull, grind size data from the 2008 exploratory test work showed that additional mass pull and grinding would significantly improve copper and gold recoveries.

Follow-up regrind and scavenger flotation test conducted on rougher tails conducted by SGS in April 2009 achieved a copper recovery of 94.7% and gold recovery of 82.9%. A copper recovery of 91% and gold recovery of 75% is estimated after accounting for losses in the cleaner circuit to produce a concentrate grade of 24% Copper.

The metallurgical test work is considered exploratory and indicative of potential recoveries. The composite samples were taken from a limited area within the Central Zone are not likely to be representative of the entire deposit. No test work has been carried out on the South Zone.

The master composite sample has higher grades than the average PEA mine plan mill feed grades. It is assumed that any recovery reduction in future test work associated with a reduction of head grade will be offset by recovery improvements from a more detailed process test work program.

Table 13-1 shows the metallurgical recovery assumptions used for the PEA. These assumptions are preliminary and will vary with future test work.

Table 13-1 Kwanika Process Recovery Assumptions

Parameter	Central Zone and UG Recovery	South Zone and Low-Grade Stockpile Recovery
Copper Recovery	91%	89%
Gold Recovery	75%	70%
Silver Recovery	75%	75%
Copper Concentrate	24% Cu	24% Cu

14 Mineral Resource Estimates

14.1 Introduction

The Mineral Resource Statement presented herein represents the copper, gold, silver, and molybdenum mineral resource evaluation prepared for the Kwanika Project in accordance with the Canadian Securities Administrators' National Instrument 43-101.

The mineral resource model prepared by SRK considers 180 core boreholes drilled by Serengeti Resources during the period of 2006 to 2016. The resource estimation work was completed by Marek Nowak, P.Eng, (APEGBC #16985) an appropriate "independent qualified person" as this term is defined in National Instrument 43-101. The effective date of the resource statement is October 14, 2016.

This section describes the resource estimation methodology and summarizes the key assumptions considered by SRK. In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the copper-gold-silver-molybdenum mineral resources found on the Kwanika Project at the current level of sampling. The mineral resources have been estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. 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.

The database used to estimate the Kwanika Project mineral resources was audited by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for copper-gold-silver-molybdenum mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

Leapfrog Geo™ 3.1.1 was used to construct the geological solids and GEOVIA GEMS™, was used to prepare assay data for geostatistical analysis, construct the block model, estimate metal grades and tabulate mineral resources. Statistical analysis was completed in non-commercial software and in SAGE for variography analysis.

14.2 Resource Estimation Procedures

The resource evaluation methodology involved the following procedures:

- Database compilation and verification;
- Construction of wireframe models for the boundaries of the mineralization;
- Definition of resource domains;
- Data conditioning (compositing and capping) for geostatistical analysis and variography;
- Block modelling and grade interpolation;
- Resource classification and validation;
- Assessment of “reasonable prospects for economic extraction” and selection of appropriate cut-off grades; and
- Preparation of the Mineral Resource Statement.

14.3 Resource Database

The Kwanika Project database used for the resource estimation comprises descriptive and assaying information for exploration drilling conducted by Serengeti from 2006 to 2016. The database was provided to SRK as an MS Access database. The resource block models for the Central and South deposits are based on 180 drillholes with 122 drillholes located in the Central Zone and 58 drillholes located in the South Zone. Table 14-1 provides a summary of the database used for the resource estimation.

Table 14-1 Exploration Data Used for Resource Estimation

Zone	Drill Type	Number of Drillholes	Total Metres Drilled	Number of Drill Samples
Central	Core	122	57,454	21,445
South	Core	58	17,958	8,065

14.4 Geology Modelling

14.4.1 Topographic Surfaces

A bare earth Lidar survey was flown over the property in September of 2016. A surface was created from the Lidar data points provided by McElhanney for both the Central and South zones. These surfaces were triangulated using all points for each zone. The collar data was pressed onto these high resolution surfaces.

A 5m resolution surface was interpolated to create a lower resolution surface for use in clipping the 3D models and informing the block models for the Central and South zones.

14.4.2 Central Zone

14.4.2.1 **Lithology Models**

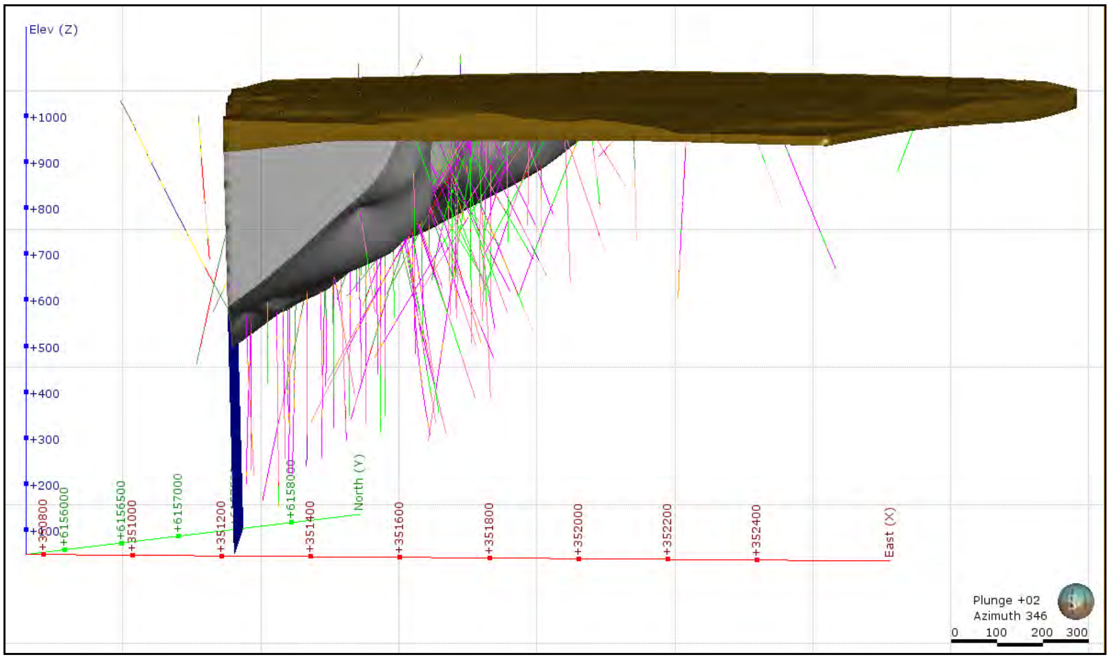
SRK modeled the major lithologies which influenced or limited the grade. These included the overburden, the unconformity surface and sediments, and the Pinchi Fault structure. SRK also reviewed the data available to model the mostly barren dykes that can cut through the deposit and determined that the dykes strike and dip were not understood well enough to model.

The unconformity and overlying sediments, overburden, and the Pinchi Fault were modeled to limit the grade from crossing these boundaries in the final resource model (Figure 14-1).

Statistics of the logged rock codes suggested that grades were generally higher in the logged monzonite lithology. After reviewing the grade data in 3D it was determined that the grade is not limited only to the monzonite and grade shells were modeled. A monzonite solid was created but not used in the resource modelling.

SRK also investigated different logged attributes which could control or influence grade in the deposit. Alteration, mineral types, fracturing, and veining were interrogated to identify possible links to grade. Visually in 3D there are distinct correlations between increased veining, chalcopyrite mineralization, albitic alteration, and increased grade. None of these features correlated sufficiently to use as a model. It was determined that a grade shell would incorporate all of these features while constraining the grade zones.

No supergene enrichment zone was modeled. After careful review of the available data and core intercepts it was determined that mineral identification was not consistently logged enough to model a high confidence zone of supergene enrichment. Native copper is present and further review and re-logging could be very helpful to define this zone for mining recovery and potential.



(Source: SRK 2016)

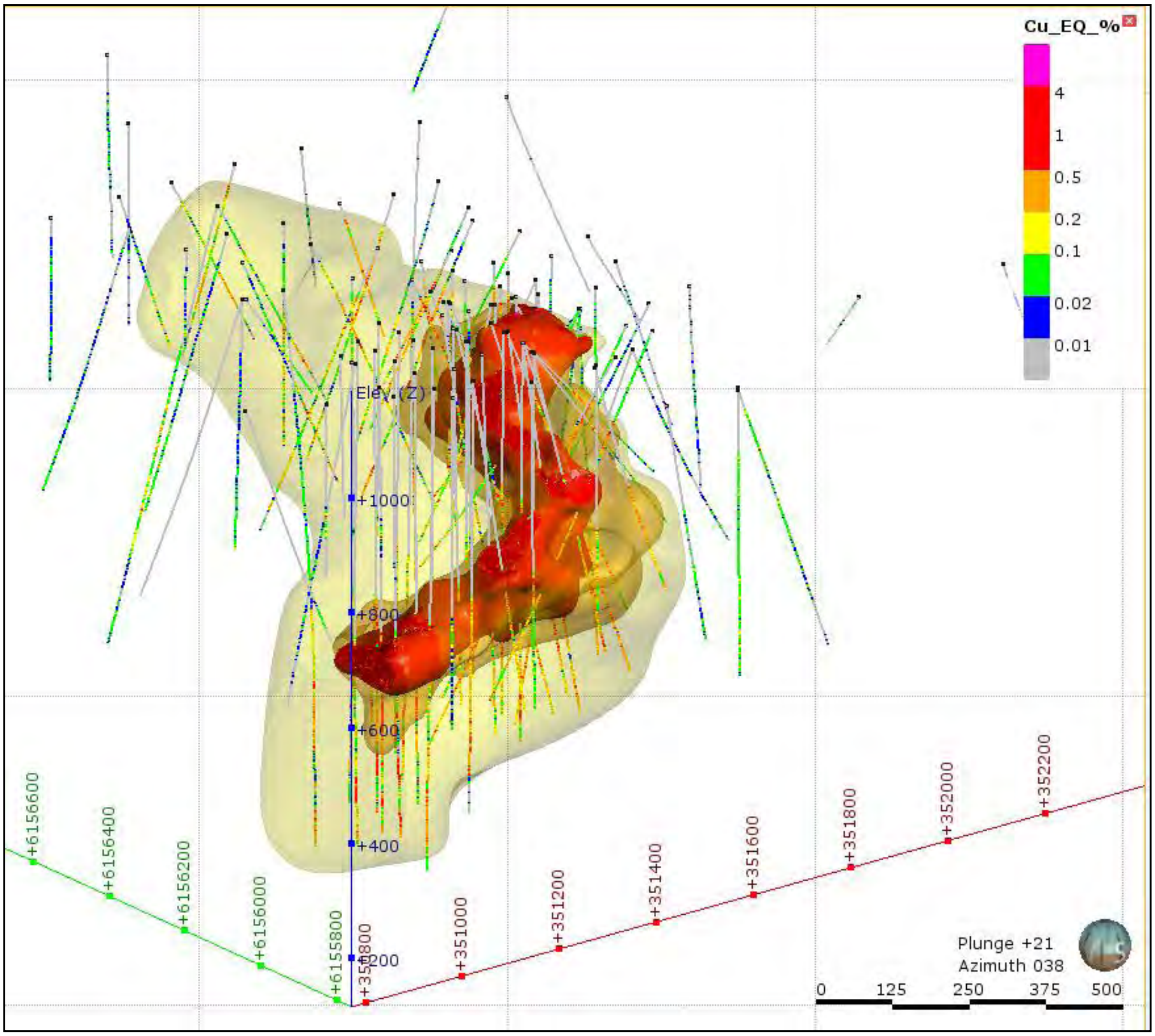
Figure 14-1 Modelled lithology solids, overburden (brown), sediments (grey), and the Pinchi Fault (blue). 3D view looking northwest

14.4.3 Estimation Domains

Three grade shell models were created in Leapfrog Geo™ 3.1.1. The shells were modeled at different copper equivalent cut-offs. The copper equivalent is based on $1130/oz gold and $2.77/lb copper. Silver is not a significant portion of the deposit and was excluded from the model. The three grade shells represent a high grade domain of 0.5% CuEq, a middle grade of 0.35% CuEq, and a low grade shell at 0.1% CuEq. The models incorporate internal dilution and are based on 5m composites. The shells were based on spheroidal indicator interpolants using 5m composites. The final shells were modeled by selecting drillhole assay intervals on section for a continuous solid which enveloped the target grade. They were modeled as geological solids in leapfrog and clipped against each other so the higher grade solids would not extend beyond the lower grade extent. The high grade shell overall correlates to the monzonite lithology, increased zone of veining, albitic alteration, and chalcopyrite mineralization.

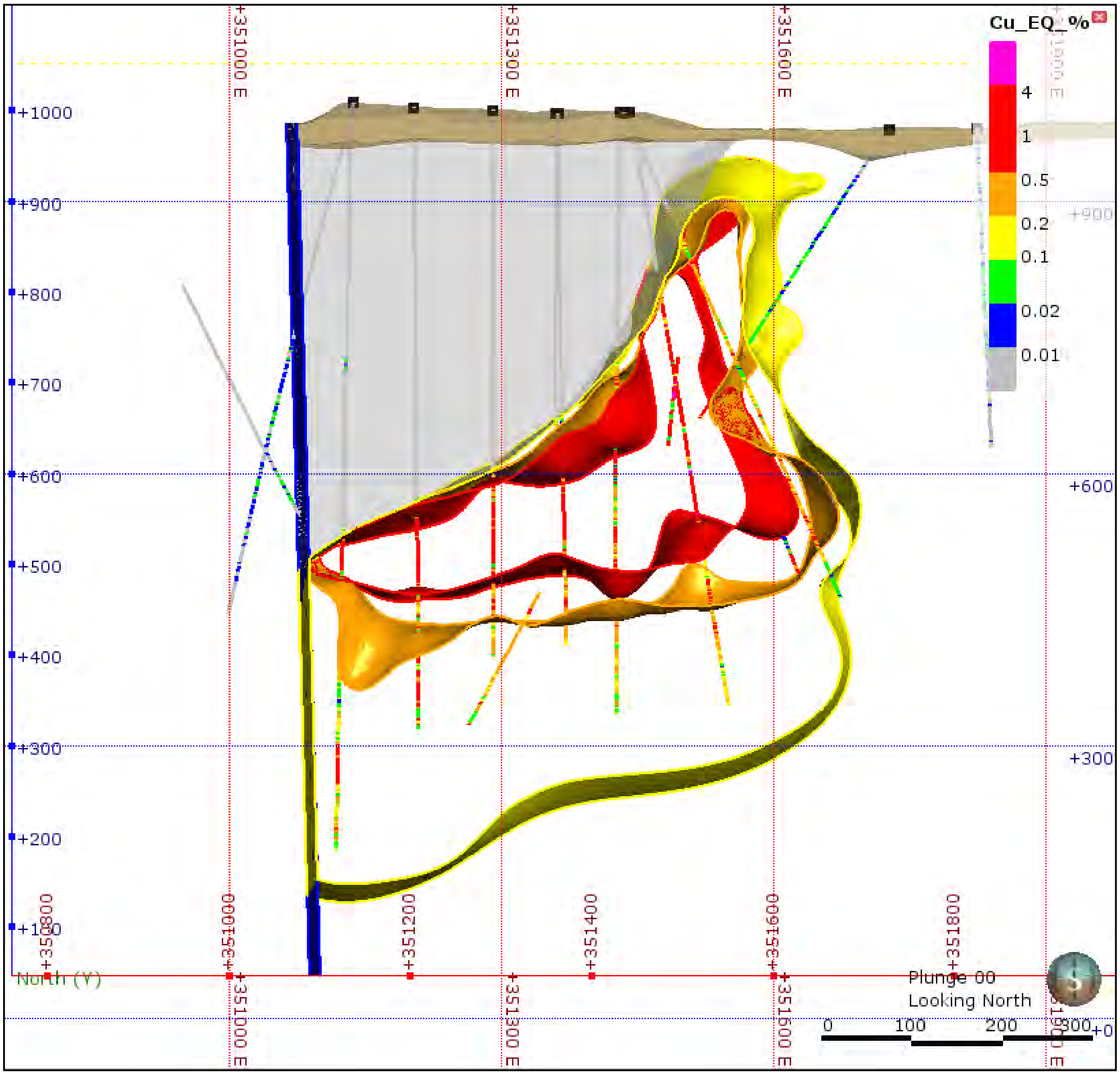
The grade shells were clipped to the unconformity surface, the overburden, the Pinchi Fault, and the topographic surface (Figure 14-2 and Figure 14-3).

For the estimation, the final grade 0.5% and 0.35% grade shells have been split into north and south domains due to the irregular shape of the grade shells. Figure 14-4 shows the 0.5% copper equivalent shell split into the north and south domains.



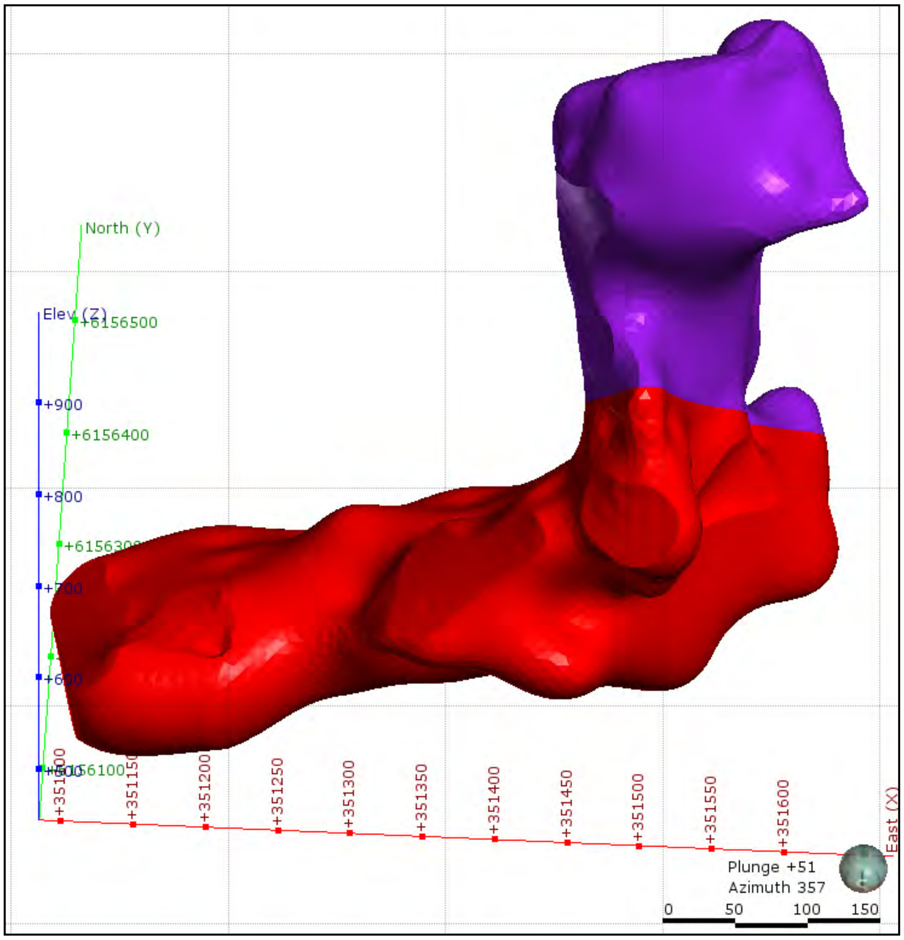
(Source: SRK 2016)

Figure 14-2 Copper Equivalent Grade Shells. 0.5% is red, 0.35% is orange, and 0.1% is yellow / 3D View Looking North East



(Source: SRK 2016)

Figure 14-3 Cross section showing the copper equivalent grade shells (red, orange, and yellow) with the overburden (brown), sediments (grey), and the Pinchi Fault (blue) with the drillholes / Section view looking north



(Source: SRK 2016)

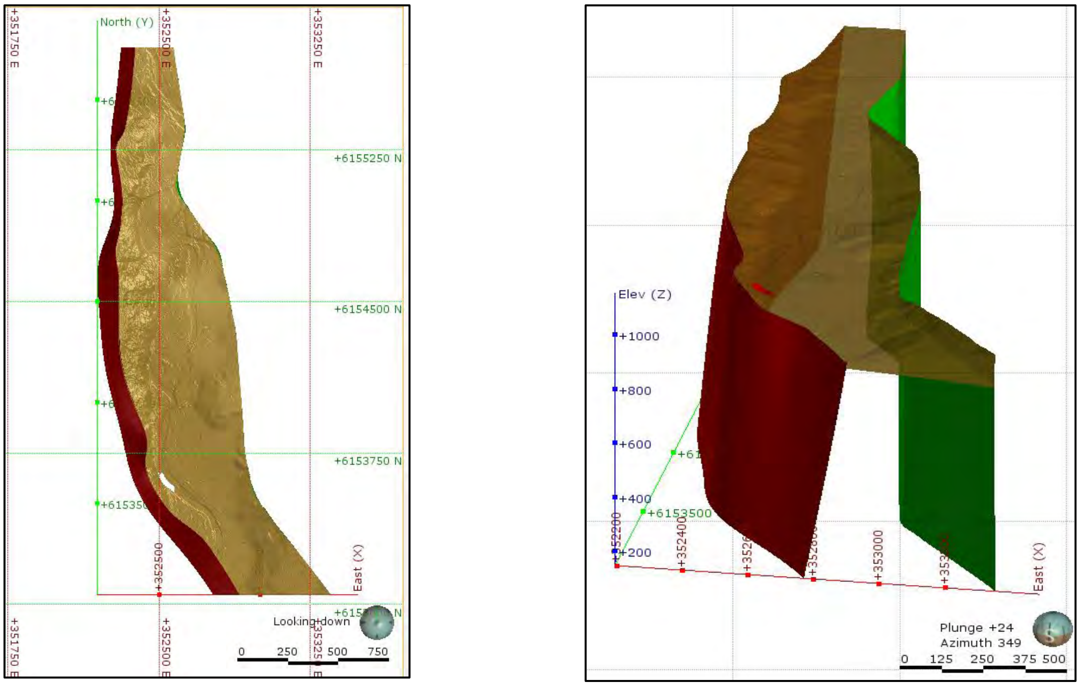
Figure 14-4 High grade 0.5% CuEq shell clipped into north (purple) and south (red) domains / 3D view looking northward

14.4.4 South Zone

14.4.4.1 Lithology Models

The South Zone contains a mix of dominantly monzonite and monzodiorite lithologies which are approximately bound by the West Fault. The grade appears to be structurally controlled and not bound by lithology or alteration. There is no clear correlation between grade and alteration, fracturing, or veining. The control of mineralization is not well understood in the South Zone.

The faults and the overburden were modeled for the South Zone. The West Fault was modelled as dipping steeply to the west based on logged fault interceptions. The East Fault was modelled from geophysical interpretation and drilling from outside the modelled area.



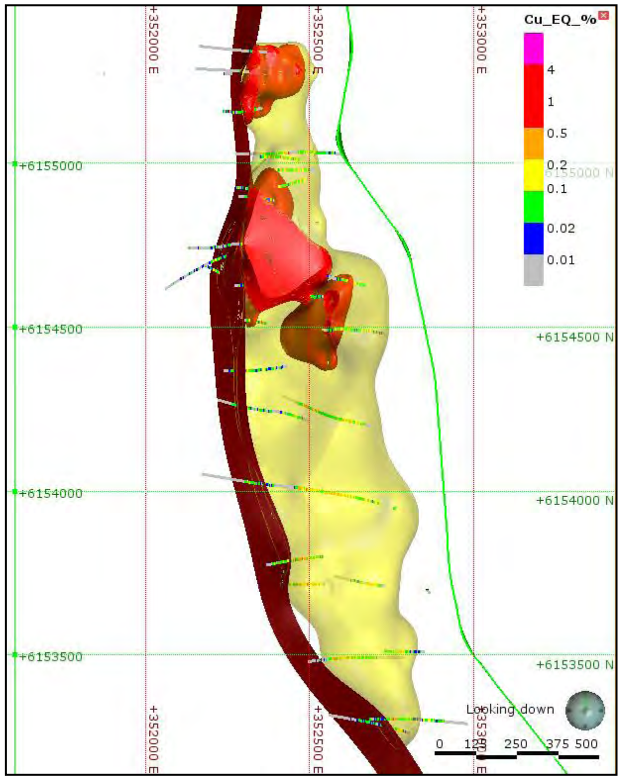
(Source: SRK 2016)

Figure 14-5 South Zone: West Fault (red) and East Fault (green) with overburden (brown) / Plan view looking down and 3D view looking northwest

14.4.5 Estimation Domains

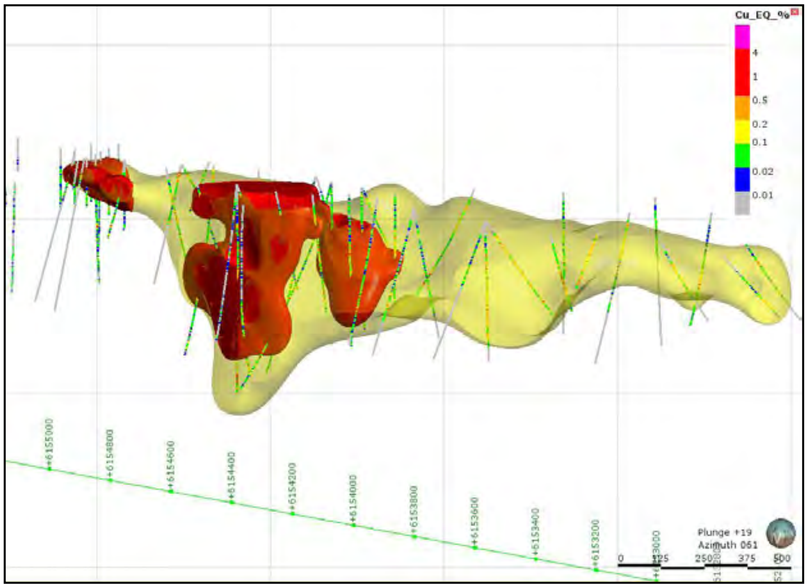
The South Zone mineralization is not well understood but tends to follow the eastern side of the West Fault. Two grade shell models were designed in Leapfrog Geo™ 3.1.1. The grade shells were modeled at different copper equivalent grades. The copper equivalent is based on $1,130/oz gold and $2.77/lb copper. Silver and molybdenum are not significant portions of the deposit and were excluded from the design of the shells. The models represent a high grade domain of 0.2% CuEq and a low grade shell at 0.07% CuEq. The models incorporate internal dilution and are based on 5m composites. The high grade domain was influenced with a structural trend following the West Fault with an area of flattening in the north end of the deposit. This structural trend was based on a simplified surface which followed the West Fault. The shells were based on indicator interpolants using 5m composites. The final shells were modeled by selecting drillhole assay intervals on section for a continuous solid which enveloped the target grade. They were modeled as geological solids in leapfrog and clipped against each other so the higher grade solids would not extend beyond the lower grade shell.

The grade shells were clipped to the overburden and the West Fault. The final clipped grade shells were used as estimation domains. Figure 14-6 and Figure 14-7 show the final estimation domain solids for the South Zone.



(Source: SRK 2016)

Figure 14-6 South Zone Estimation Domains between the West Fault (dark red) and the East Fault (green). Copper equivalent shells 0.07% CuEq (yellow) and 0.2% CuEq (red) / Plan view looking down



(Source: SRK 2016)

Figure 14-7 South Zone Estimation Domains, copper equivalent shells 0.07% CuEq (yellow) and 0.2% CuEq (red). 3D view looking northeast.

14.5 Compositing

14.5.1 Central Zone

Before compositing, SRK capped very few extreme assay values. In 0.5% CuEq south shell three copper assays were capped at 6% and in 0.5% CuEq north shell one copper assay was capped at 4%. Apart from the extreme value capping, the influence of a larger number of high grade assays has been further limited by a procedure described in Section 13.6. For the resource estimation, the assays were composited to 3m intervals, separately within each grade shell. Only composites longer than 1m were used in the estimation process.

14.5.2 South Zone

For the resource estimation, the assays were composited to 3m intervals, separately within each grade shell. Only composites longer than 1m were used in the estimation process.

14.6 Central Zone - Evaluation of High Grade Populations

Block grade estimates may be unduly affected by very high grade assays. As described in Section 13.5.1, very few extreme assays were capped. Furthermore, SRK elected to limit the influence of high grade assays assigned to high grade populations (Table 14-2 to Table 14-4). For each estimation domain, a probability plot of composite metal assay grades was used to select high grade population thresholds.

The composite assays from the high-grade populations were only used to estimate a block if they were located at small distances from the block.

In addition, for testing the estimation results from the high-grade restriction design, a typical capping, with no high grade restriction, was applied on the composited data. A test resource block model from the capped data was compared with the final high grade restriction resource model.

Table 14-2 Central Zone: Copper High Grade Population Thresholds

Domain Name	Max Composite Assay (%)	Capped Value for Testing Only	Number Capped	High Grade Threshold Defined from Composites	Ndat All Comps	Number above threshold	%Data above Threshold
SH05S	5.6	3.5	9	2.0	1,214	39	3
SH05N	3.2	2.0	7	1.5	1,067	22	2
SH035S	1.9	1.0	9	0.6	828	30	4
SH035N	1.9	1.3	3	0.5	696	50	7
SH01	1.2	0.6	26	0.4	4,815	164	3

Table 14-3 Central Zone: Gold High Grade Population Thresholds

Domain Name	Max Composite Assay (g/t)	Capped Value for Testing Only	Number Capped	High Grade Threshold Defined from Composites	Ndat All Comps	Number above threshold	%Data above Threshold
SH05S	5.4	4.5	8	3.0	1,214	36	3
SH05N	5.1	4.0	2	2.0	1,067	31	3
SH035S	1.7	1.0	7	0.7	828	21	3
SH035N	3.3	1.7	7	0.9	696	21	3
SH01	4.2	1.0	26	0.5	4,815	123	3

Table 14-4 Central Zone: Silver high grade population thresholds

Domain Name	Max Composite Assay (g/t)	Capped Value for Testing Only	Num Capped	High Grade Threshold defined from composites	Ndat All Comps	Number above threshold	%Data above Threshold
SH05S	24.5	9.0	6	6.0	1,214	21	2
SH05N	9.9	7.0	4	4.0	1,067	35	3
SH035S	4.7	4.0	3	3.0	828	13	2
SH035N	5.5	3.5	3	2.0	696	30	4
SH01	15.0	2.0	78	1.5	4,815	195	4

14.7 South Zone - Capping

Composite assay grades were capped as presented in Table 14-5.

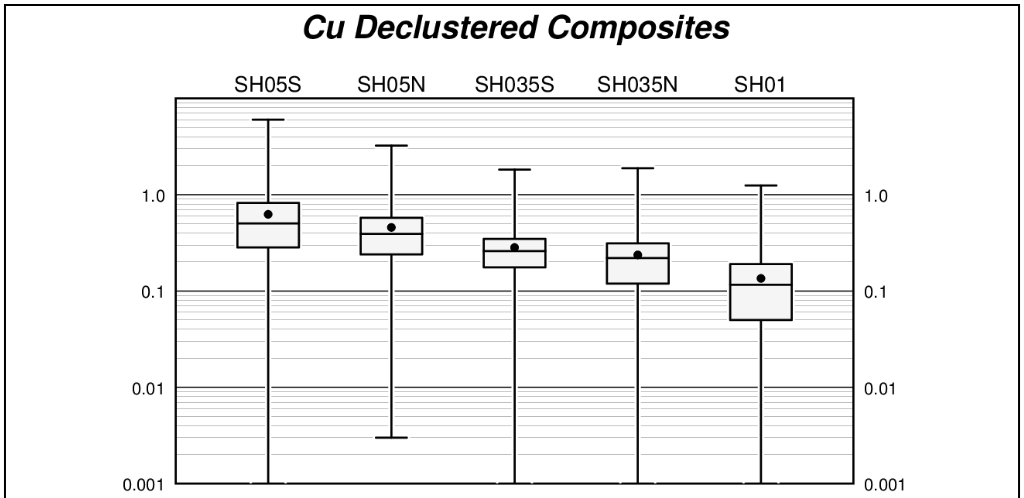
Table 14-5 South Zone: Capped composites

Metal	Estimation Domain	Max Composite Assay (%)	Number of Data	Capped Value	Num Capped
Cu	SH02	2.01	704	1.20	10
	SH007	1.31	2128	0.60	17
Au	SH02	5.45	704	0.50	5
	SH007	1.64	2128	0.50	20
Ag	SH02	12.6	704	8.00	7
	SH007	17.6	2128	4.00	26
Mo	SH02	0.210	704	0.08	5
	SH007	0.153	2128	0.06	8

14.8 Statistical Analysis

14.8.1 Central Zone

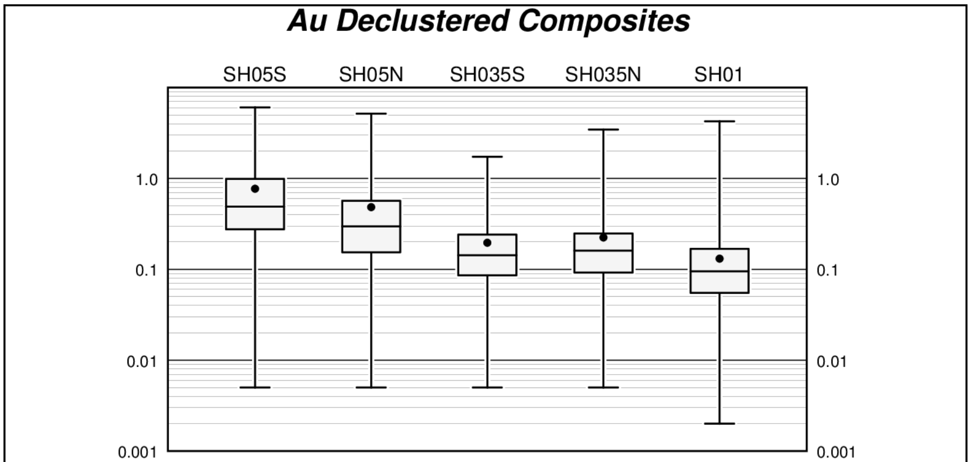
Statistics of polygonally declustered 3m composite grades for copper, gold and silver within each estimation domain are presented in Figure 14-8 to Figure 14-10. The highest metal grades are associated with the southern portion of the 0.5% CuEq grade shell.



	SH05S	SH05N	SH035S	SH035N	SH01	
Number of data	1233	1083	848	710	5003	Number of data
Mean	0.625	0.457	0.283	0.237	0.135	Mean
Std. Dev.	0.504	0.356	0.187	0.178	0.111	Std. Dev.
Coef. of Var.	0.807	0.777	0.659	0.75	0.819	Coef. of Var.
Maximum	6.0	3.231	1.825	1.877	1.246	Maximum
Upper quartile	0.821	0.575	0.347	0.312	0.19	Upper quartile
Median	0.504	0.392	0.26	0.22	0.116	Median
Lower quartile	0.283	0.24	0.176	0.119	0.05	Lower quartile
Minimum	0.001	0.003	0.001	0.001	0.001	Minimum

(Source: SRK, 2016)

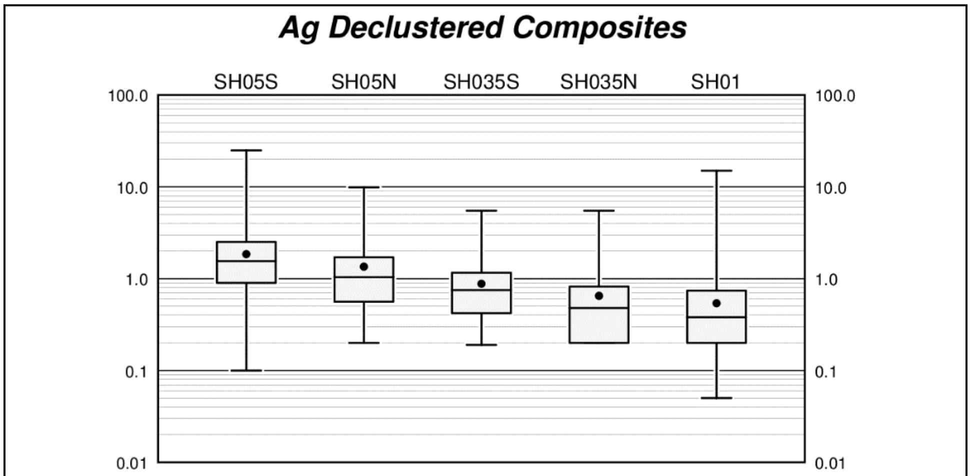
Figure 14-8 Basic Statistics for Declustered Copper Composite Assays (%) In The Central Zone Estimation Domains



	SH05S	SH05N	SH035S	SH035N	SH01	
Number of data	1233	1083	848	710	5003	Number of data
Mean	0.77	0.483	0.196	0.224	0.131	Mean
Std. Dev.	0.792	0.568	0.173	0.29	0.141	Std. Dev.
Coef. of Var.	1.028	1.177	0.882	1.297	1.08	Coef. of Var.
Maximum	6.048	5.152	1.729	3.443	4.257	Maximum
Upper quartile	0.99	0.568	0.241	0.248	0.168	Upper quartile
Median	0.49	0.297	0.143	0.16	0.095	Median
Lower quartile	0.275	0.154	0.086	0.092	0.055	Lower quartile
Minimum	0.005	0.005	0.005	0.005	0.002	Minimum

(Source: SRK, 2016)

Figure 14-9 Basic Statistics for Declustered Gold Composite Assays (G/T) In The Central Zone Estimation Domains



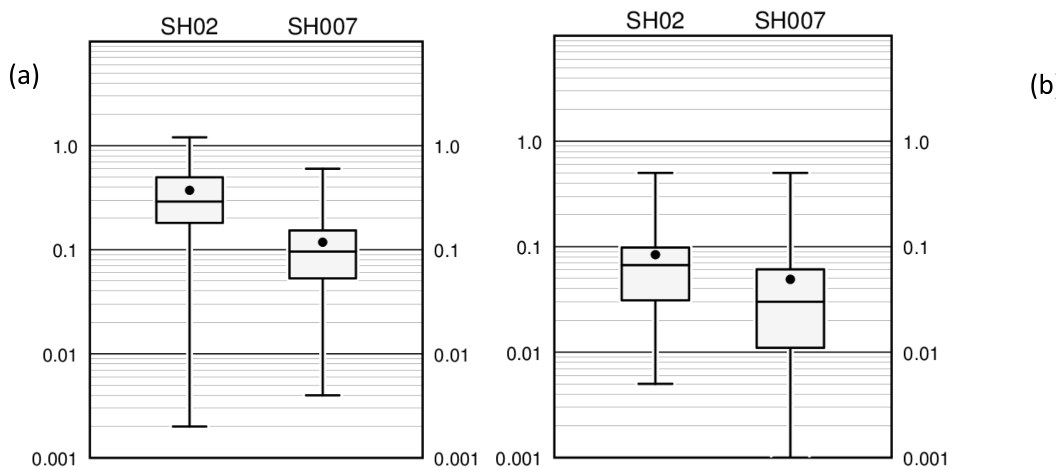
	SH05S	SH05N	SH035S	SH035N	SH01	
Number of data	1233	1083	848	710	5003	Number of data
Mean	1.85	1.35	0.88	0.65	0.54	Mean
Std. Dev.	1.39	1.17	0.63	0.57	0.55	Std. Dev.
Coef. of Var.	0.75	0.87	0.72	0.87	1.02	Coef. of Var.
Maximum	25.0	9.89	5.49	5.49	15.0	Maximum
Upper quartile	2.51	1.71	1.16	0.82	0.74	Upper quartile
Median	1.55	1.04	0.75	0.48	0.38	Median
Lower quartile	0.9	0.56	0.42	0.2	0.2	Lower quartile
Minimum	0.1	0.2	0.19	0.2	0.05	Minimum

(Source: SRK, 2016)

Figure 14-10 Basic Statistics for Declustered Silver Composite Assays (G/T) in the Central Zone Estimation Domains

14.8.2 South Zone

Statistics of polygonally declustered 3m composite capped grades for copper, gold, silver and molybdenum are presented in Figure 14-11 and Figure 14-12. Note generally lower gold grades in this zone when compared to the Central Zone.

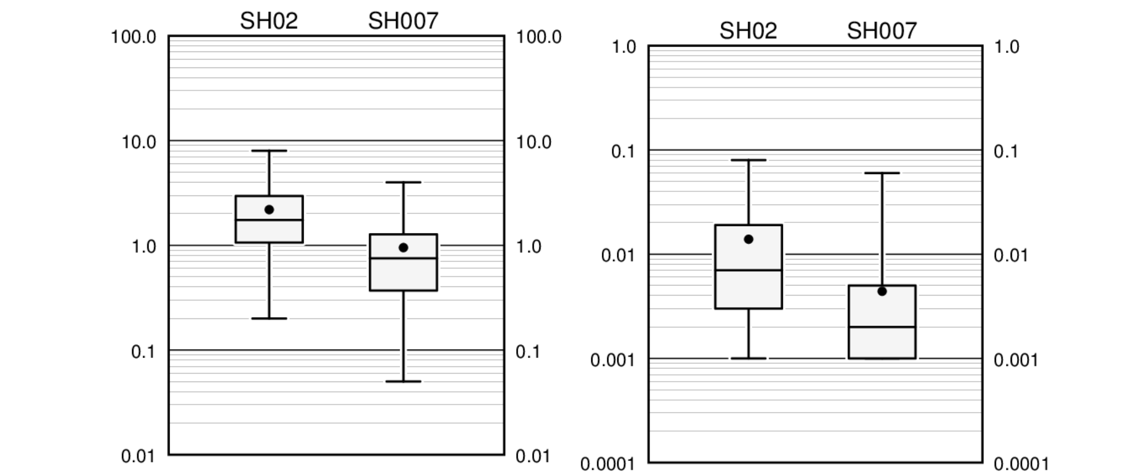


(a)	SH02	SH007
Number of data	704	2128
Mean	0.372	0.118
Std. Dev.	0.269	0.091
Coef. of Var.	0.724	0.772
Maximum	1.2	0.6
Upper quartile	0.495	0.153
Median	0.29	0.096
Lower quartile	0.181	0.053
Minimum	0.002	0.004

SH02	SH007	(b)
704	2128	Number of data
0.084	0.049	Mean
0.085	0.07	Std. Dev.
1.002	1.41	Coef. of Var.
0.5	0.5	Maximum
0.098	0.061	Upper quartile
0.067	0.03	Median
0.031	0.011	Lower quartile
0.005	0.001	Minimum

(Source: SRK, 2016)

Figure 14-11 Basic Statistics for Declustered (A) Copper (%) And (B) Gold (G/T) Composite Assays in the South Zone Estimation Domains



	SH02	SH007
Number of data	704	2128
Mean	2.19	0.95
Std. Dev.	1.6	0.77
Coef. of Var.	0.73	0.81
Maximum	8.0	4.0
Upper quartile	2.95	1.27
Median	1.74	0.75
Lower quartile	1.06	0.37
Minimum	0.2	0.05

SH02	SH007	
664	1801	Number of data
0.0139	0.0044	Mean
0.0164	0.0064	Std. Dev.
1.1783	1.447	Coef. of Var.
0.08	0.06	Maximum
0.019	0.005	Upper quartile
0.007	0.002	Median
0.003	0.001	Lower quartile
0.001	0.001	Minimum

(Source: SRK, 2016)

Figure 14-12 Basic Statistics for Declustered (A) Silver (G/T) and (B) Molybdenum (%) Composite Assays in the South Zone Estimation Domains

14.9 Variography

14.9.1 Central Zone

Correlogram models were designed for copper and gold from composited assay data in all estimation domains. Correlograms were selected because they tend to be less sensitive to very high values. Downhole correlograms were used to model nugget effects (i.e. assay variability at very close distances). Directional correlograms, supported by correlogram maps were used to model grade continuities.

The correlogram models used in the resource estimation are presented in Table 14-6 and Table 14-7.

Table 14-6 Correlograms of Copper Grades

Domain Name	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
SH05S	0.1	0.54	0	49	75	10	15	50
		0.36				95	70	350
SH05N	0.10	0.30	-55	-42	50	50	6	6
		0.60				68	100	140
SH035S	0.10	0.70	-84	-45	87	30	30	35
		0.20				280	400	80
SH035N	0.10	0.30	0	0	0	30	30	15
		0.60				70	200	150
SH01	0.10	0.70	-90	40	0	25	17	30
		0.20				90	300	230

Table 14-7 Correlograms of Gold Grades

Domain Name	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
SH05S	0.05	0.40	0	-32	-49	14	8	40
		0.55				55	70	200
SH05N	0.05	0.50	-17	21	-56	20	80	30
		0.45				40	370	55
SH035S	0.15	0.50	0	0	0	60	20	45
		0.35				350	200	310
SH035N	0.10	0.40	-15	-35	5	10	30	20
		0.50				70	150	50
SH01	0.30	0.47	-109	44	40	50	25	20
		0.23				110	400	230

14.9.2 South Zone

Correlogram models were designed for copper and gold from composited assay located within both the higher grade and low grade domains. Directional correlograms, supported by correlogram maps were used to model grade continuities (Table 14-8).

Table 14-8 Correlograms of Copper and Gold Grades

Metal	Nugget C_0	Sill C_1 and C_2	Gemcom Rotations (RRR rule)			Ranges a_1, a_2		
			around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
Copper	0.15	0.30	-30	0	0	25	25	15
		0.55				85	344	180
Gold	0.25	0.60	45	25	0	100	15	25
		0.15				450	280	300

14.10 Specific Gravity

14.10.1 Central Zone

A total of 2,535 specific gravity (SG) determinations are present in the Central Zone, 1,612 of these are located in the resource domains. The average SG values are very similar in all domains ranging between 2.74 in the low-grade domain and 2.76 in the high-grade domains. SRK elected to estimate block SG using the inversed distance squared interpolation method.

14.10.2 South Zone

A total of 1,141 specific gravity (SG) determinations are present in the South Zone with 574 being located in the resource domains. The average SG values are very similar in both higher and low grade domains with 2.66 in the higher-grade area and 2.68 in the lower grade area. SRK elected to estimate block SG using the inverse distance squared interpolation method.

14.11 Block Model and Grade Estimation Methodology

14.11.1 Central Zone

Resource estimation was completed in the Central Zone with the block model geometry and extents as presented in Table 14-9.

Table 14-9 Central Zone: Block Model Extents

Description	Easting	Northing	Elevation
	X (m)	Y (m)	Z (m)
Block Model Origin (Lower left corner)	350,800	6,155,500	180
Block Dimension	10	10	10
Number of Blocks	172	252	92

The resource estimation methodology was based on the following:

- Before compositing a few very extreme values were capped and any missing assays were assigned 0.0 grades.
- Assays were composited to 3.0m lengths.
- Grades were estimated by ordinary kriging for all metals. Based on high correlation between copper and silver, silver was estimated using the copper correlogram models.
- High grade restrictions were applied for all metals. High grade search ellipsoids were 65m in major and semi-major directions of continuity and 45m in a minor direction of continuity.
- A combination of hard and soft boundaries was applied during the estimation process according to a logic matrix presented in Table 14-10.
- All blocks estimated for copper were also estimated for silver.
- Specific gravity was estimated by an inverse distance squared procedure.

Table 14-10 Logic Matrix Defining Which Composites Were Used to Estimate Blocks in the Central Zone

		BLOCKS				
		SH05S	SH05N	SH035S	SH035N	SH01
DATA	SH05S	X	X	X	X	
	SH05N	X	X	X	X	
	SH035S	X	X	X	X	X
	SH035N	X	X	X	X	X
	SH01			X	X	X

The selection of the search radii and rotations of search ellipsoids were guided by copper correlogram models. In addition, the search radii were established to estimate a large portion of the blocks within the modelled area with limited extrapolation. The parameters were established by conducting repeated test resource estimates and reviewing the results as a series of plan views and sections. Identical search ellipsoid rotations and very similar search radii were used for gold and silver. This design ensured that all blocks estimated for copper were also estimated for the other two metals. Table 14-11 shows applied copper estimation parameters.

Table 14-11 Central Zone: Copper Estimation Parameters

Domain	Min Sample	Max Sample	Max per Hole	Gemcom Rotations (RRR rule)			Search Radii		
				around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
SH05S	6	16	4	0	49	75	95	65	130
SH05N	6	16	4	-55	-42	50	70	110	150
SH035S	6	16	4	-84	-45	87	90	130	70
SH035N	6	16	4	0	0	0	50	130	90
SH01	6	16	4	-109	40	40	100	200	200

South Zone

Resource estimation was completed in the South Zone with the block model geometry and extents as presented in Table 14-12.

Table 14-12 South Zone: Block Model Extents

Description	Easting	Northing	Elevation
	X (m)	Y (m)	Z (m)
Block Model Origin (Lower left corner)	350,800	6,155,500	180
Block Dimension	10	10	10
Number of Blocks	172	252	92

The resource estimation methodology was based on the following:

- Before compositing a few very extreme values were capped and any missing assays were assigned 0.0 grades.
- Assays were composited to 3.0m lengths and capped.
- Ordinary kriging was applied for copper, gold and silver. Based on high correlation between copper and silver, silver was estimated with copper variogram models. Molybdenum was estimated using inverse distance squared interpolation.
- Blocks in the 0.2% CuEq shell were estimated from composites within the shell and from composites located within 50m distance from the shell (buffer zone).
- Blocks in the buffer zone were estimated from all data.

- Blocks in the 0.07% CuEq shell were estimated from data located in the shell and from data located in the buffer area.
- Specific gravity was estimated by inverse distance squared interpolation method.

Resource estimation parameters in the South Zone are presented in Table 14-13.

Table 14-13 South Zone: Estimation Parameters

Metal	Min Sample	Max Sample	Max per hole	Gemcom Rotations (RRR rule)			Radii		
				around Z	around Y	around Z	X-Rot	Y-Rot	Z-Rot
Cu	6	24	6	-30	0	0	150	350	200
Au	6	24	6	45	25	0	350	200	200
Ag	6	24	6	-30	0	0	150	350	200
Mo	6	24	6	-30	0	0	150	350	200

14.12 Model Validation and Sensitivity

All estimated domains in both Central and South zones were validated by completing a series of visual inspections and by:

- Comparison of local “well-informed” block grades with composites contained within those blocks.
- Comparison of average assay grades with average block estimates along different directions – swath plots.

In addition, in the Central Zone SRK compared estimated block grades from the high grade restriction methodology and estimated block grades from capped data.

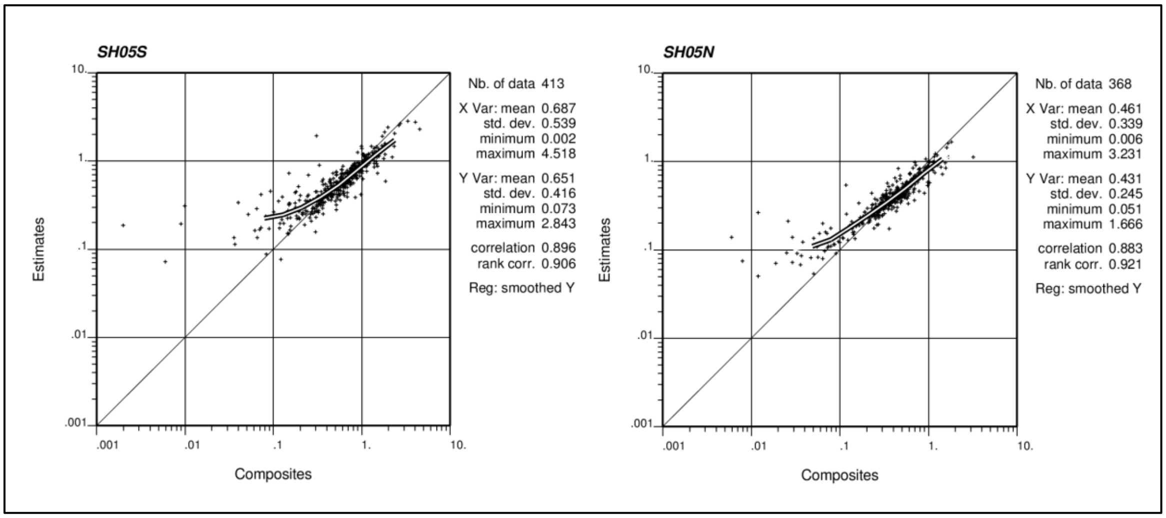
14.12.1 Central Zone

Table 14-14 shows the results of a comparison of the average grades from models estimated using restricted search methodology for high grade data with the simple capping without restricted search methodology. Ideally, at no cut-off, the model based on the capped data should be similar to the model based on the high grade restriction. The results indicate that no specific bias has been introduced by an application of high grade restriction to the estimation process.

Table 14-14 Comparisons of Average Grades at Zero Cut-Off from Different Block Models

Domain	Metal	Resource Model: Estimated Average from High Grade Restriction	Estimated Average from Capped Data
SH05S	Cu	0.56	0.56
SH05N		0.42	0.42
SH035S		0.29	0.3
SH035N		0.23	0.24
SH01		0.14	0.14
SH05S	Au	0.69	0.7
SH05N		0.46	0.46
SH035S		0.23	0.24
SH035N		0.24	0.25
SH01		0.13	0.14

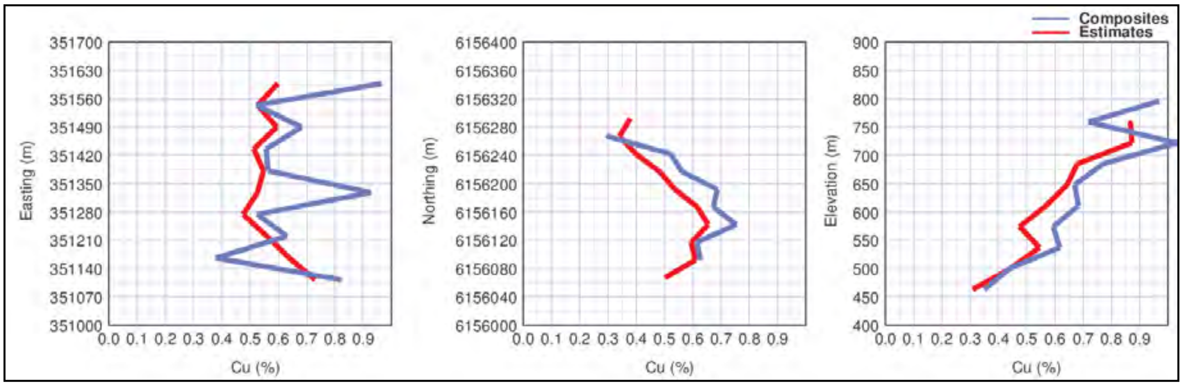
Figure 14-13 shows a comparison of estimated copper block grades with drillhole assay composite data contained within those blocks in the SH05S and SH05N domains. On average, the estimated blocks are slightly lower than actual data. This is a result of soft boundary applied between these high-grade domains and lower grade domains. The comparison is based on the data actually located in the high-grade domains and the estimated block grades influenced by both the high and the lower grade domains. Note very little scatter around the x = y line. This indicates that estimated block grades are quite variable and not over smoothed. Similar results were noted in other estimation domains and for other metals.



(Source: SRK, 2016)

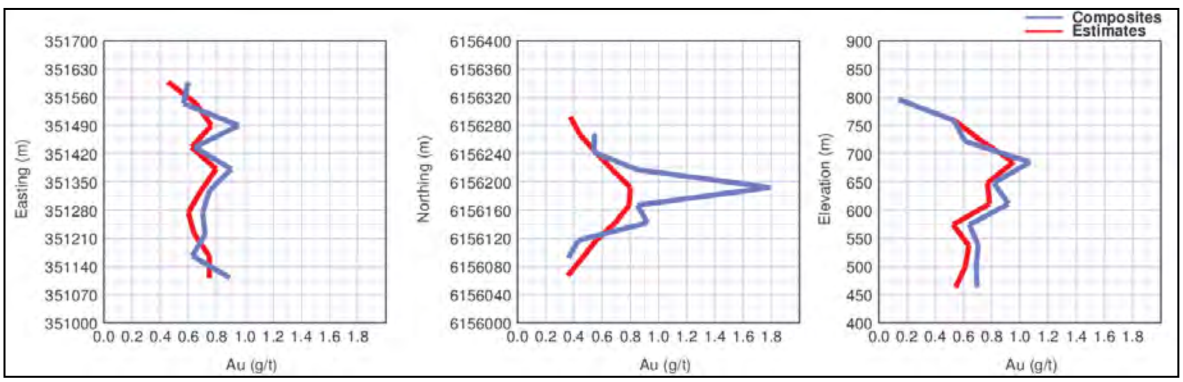
Figure 14-13 Comparison of Copper Block Estimates with Borehole Assay Data Contained Within Blocks in Two High Grade Estimation Domains

As a final check, average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparison with average block estimates along east-west, north-south, and horizontal swaths. Figure 14-14 and Figure 14-15 show the swath plots for copper and gold in the high-grade domain SH05S. Note that generally the block estimated grades are slightly lower than the declustered data within this domain. As already discussed, this is a result of estimating the block with soft boundaries between the high grade and the lower grade domains. In other domains, the differences are smaller. Overall, the validation shows that current resource estimates are a good reflection of drillhole assay data.



(Source: SRK, 2016)

Figure 14-14 SH05S Domain: Declustered Average Copper Composite Grades Compared to Copper Block Estimates



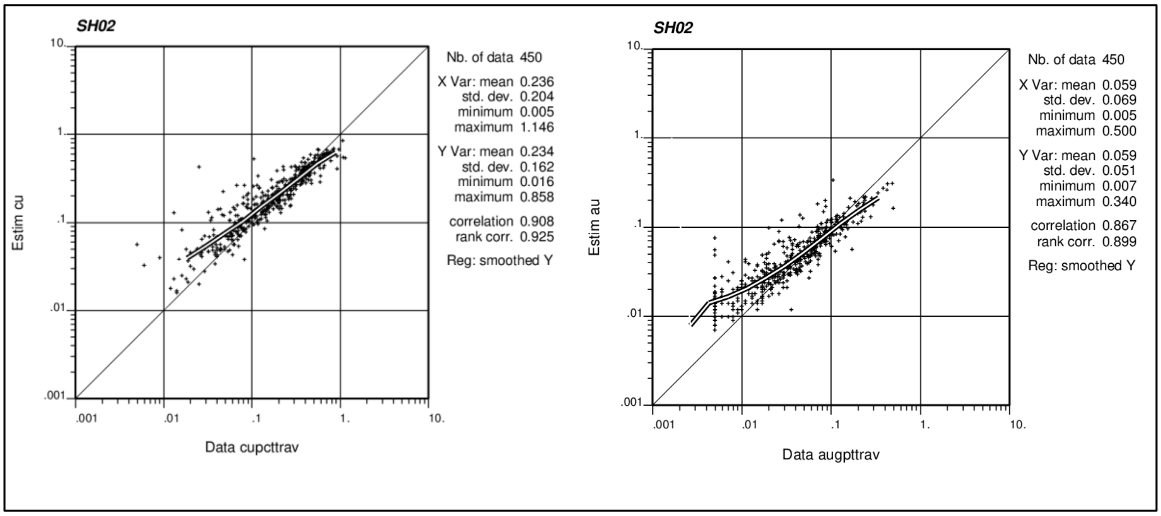
(Source: SRK, 2016)

Figure 14-15 SH05S Domain: De-clustered Average Gold Composite Grades Compared to Gold Block Estimates

14.12.2 South Zone

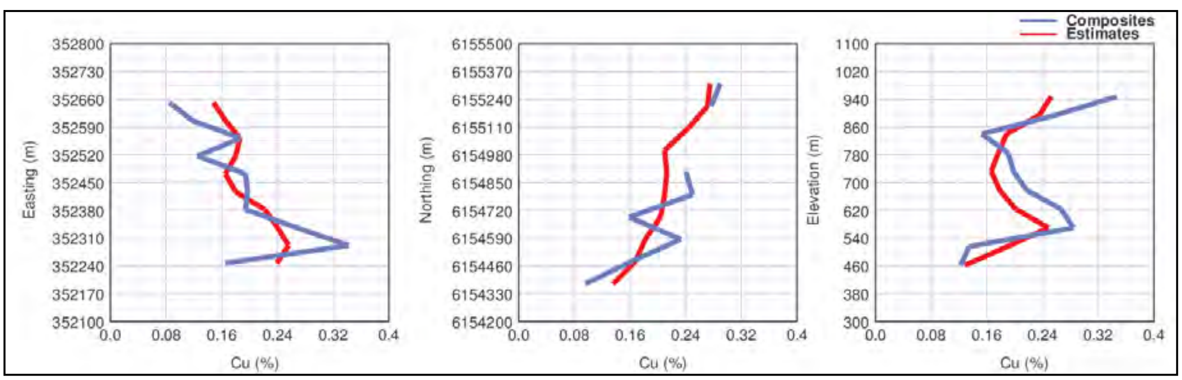
Figure 14-16 shows a comparison of estimated copper and gold block grades with drillhole assay composite data contained within those blocks in the higher grade 0.2% CuEq domain. On average, the estimated blocks are very similar to the actual data. Note very little scatter around the x = y line. This indicates that estimated block grades are quite variable and not over smoothed. Similar results were noted in other estimation domains and for other metals.

In another check, average composite grades and average block estimates were compared along different directions. This involved calculating de-clustered average composite grades and comparison with average block estimates along east-west, north-south, and horizontal swaths. Figure 14-17 shows the swath plots for copper in the higher grade area. Note that the block estimated grades are quite similar to the declustered data. Similar results were shown for other metals. Overall, the validation shows that current resource estimates are a good reflection of drillhole assay data.



(Source: SRK, 2016)

Figure 14-16 Comparison of Copper and Gold Block Estimates with Borehole Assay Data Contained within Blocks in the 0.2% CuEq Domain



(Source: SRK, 2016)

Figure 14-17 Declustered Average Copper Composite Grades Compared to Copper Block Estimates in the Higher Grade Area

14.13 Mineral Resource Classification

Block model quantities and grade estimates for the Kwanika Project were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May, 2014) by Marek Nowak, PEng. (APEGBC #16985), an appropriate independent qualified person for the purpose of National Instrument 43-101.

Mineral resource classification is typically a subjective concept, industry best practices suggest that resource classification should consider both the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating both concepts to delineate regular areas at similar resource classification.

SRK is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired by core drilling on sections spaced at approximately 50m for the Central Zone and 100m for the South Zone.

SRK assigned estimated blocks to Indicated Category if the following criteria were met:

- A block had to be estimated from at least three drillholes.
- Average distance of composites used to estimate a block was lower than 80m.
- At least eight composites had to be used to estimate a block grade.

Note that small clusters of blocks assigned to Indicated Category and located outside of the core of the deposit were re-assigned to an Inferred Category. All estimated blocks not assigned to Indicated Category were assigned to an Inferred Category.

14.14 Mineral Resource Statement

CIM Definition Standards for Mineral Resources and Mineral Reserves (May, 2014) defines a mineral resource as:

"A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."

To determine the quantities of material offering "reasonable prospects for eventual economic extraction" by an open pit, SRK used a Whittle pit optimizer and reasonable mining assumptions to evaluate the proportions of the block models that could be "reasonably expected" to be mined from an open pit. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade (Table 14-15). The Whittle shells were designed based on identical parameters for both Central and South zones.

The reader is cautioned that the results from the pit optimization are used solely for testing the "reasonable prospects for eventual economic extraction" by an open pit and do not represent an attempt to estimate mineral reserves.

Table 14-16 presents the Indicated and Inferred resources in the Central Zone within the Whittle shell at 0.13% copper equivalent cut-off and Indicated and Inferred resource outside of the Whittle shell possibly amenable to underground mining by block caving method reported at 0.27% copper equivalent cut-off. Figure 14-18 shows a designed Whittle shell and the area for potential underground mining within which the resources have been reported for the Central Zone.

Table 14-17 presents the Inferred open pit resources in the South Zone within the Whittle shell at 0.13% copper equivalent cut-off. Figure 14-19 and Figure 14-20 show a designed Whittle shell within which the resources have been reported for the South Zone.

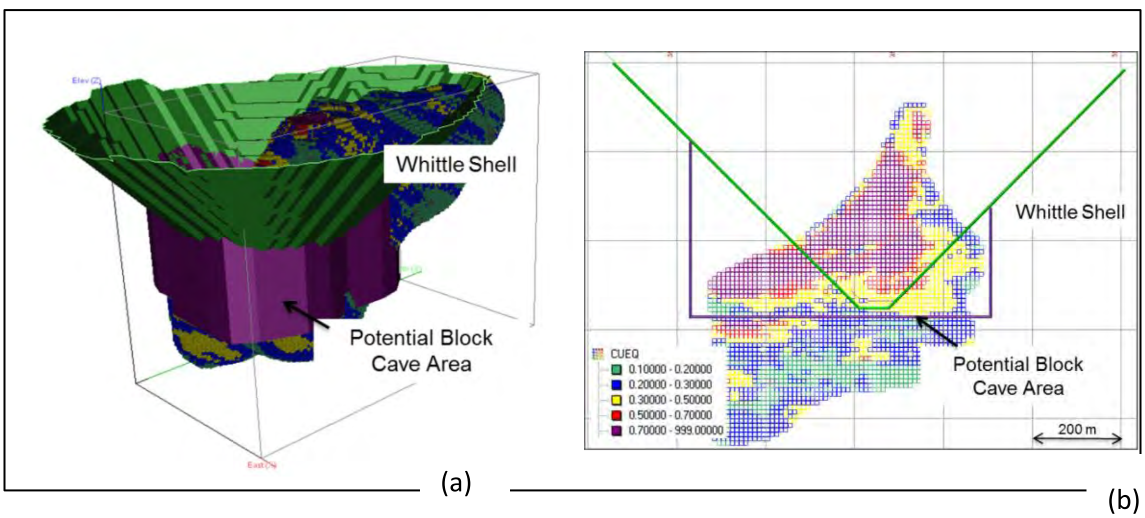


Figure 14-18 Central Zone: (a) North-west view of the Whittle shell and a potential block cave area, (b) East-west 6,156,175N section showing estimated copper equivalent block grades with the Whittle shell and the potential block cave area

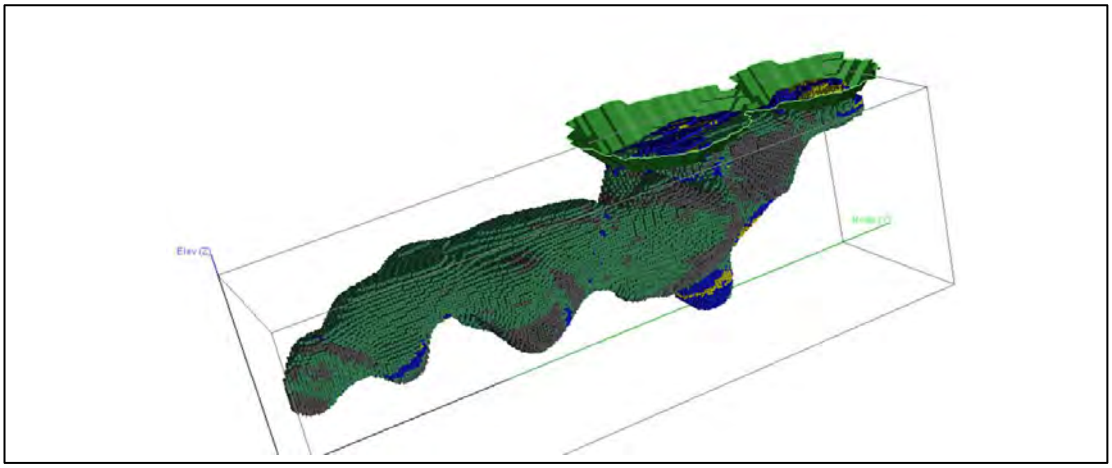


Figure 14-19 South Zone: North-west View of the Whittle shell

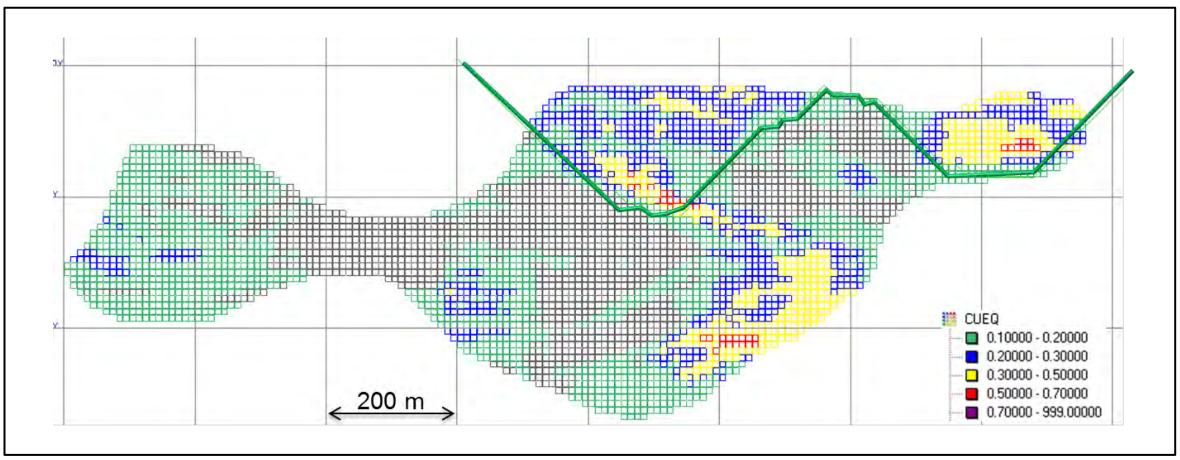


Figure 14-20 South Zone: North-south 352,390E section of the Whittle Shell and Estimated Copper Equivalent Block Grades

Table 14-15 Whittle™ Optimization Parameters for Resource Estimation Constraint

Input for Pit Optimization	Cu	Au	Ag	Mo
Metal Price (US dollars)	$3/lb	1300/oz	$20/oz	$9/lb
Open Pit Mining Cost - Plant feed and Waste (Canadian dollars)	$2/t mined			
G&A costs, Processing, Water treatment and Tailings Placement (Canadian dollars)	$10/t milled			
Mining Loss	5%			
Dilution	2%			
Metal Recoveries	89%	70%	75%	60%
Overall Slope Angle (degrees)	45			

Table 14-16 Mineral Resource Statement*, Central Zone of the Kwanika Project, British Columbia, Canada, SRK Consulting, effective date October 14, 2016

Category	Quantity (x1000 Tonnes)	Grade			Contained Metal		
		Cu (%)	Au (g/t)	Ag (g/t)	Cu (000's lb)	Au (000's oz)	Ag (000's oz)
Open Pit							
Indicated	101,500	0.31	0.32	0.96	697,200	1,040	3,120
Inferred	31,900	0.17	0.14	0.59	118,500	140	610
Underground							
Indicated	29,700	0.34	0.36	1.05	222,300	350	1,010
Inferred	7,900	0.23	0.17	0.68	39,800	40	170

*Open pit mineral resources are reported in relation to a conceptual pit shell. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.

** Open pit mineral resources are reported at 0.13% copper equivalent cut-off and underground resources reported at 0.27%. The cut-off is based on a price of US$3.00 per lb of copper, US$1,300 per ounce of gold, US$20 per ounce of silver. The assumed recoveries are for copper 89%, gold 70%, and silver 75%.

Table 14-17 Mineral Resource Statement*, South Zone of the Kwanika Project, British Columbia, Canada, SRK Consulting, effective date October 14, 2016

Category	Quantity	Grade				Contained Metal			
	(x1000 Tonnes)	Cu (%)	Au (g/t)	Ag (g/t)	Mo (%)	Cu (000's lb)	Au (000's oz)	Ag (000's oz)	Mo (000's lb)
Inferred	33,300	0.26	0.08	1.64	0.01	191,400	80	1,760	7,470

*Open pit mineral resources are reported in relation to a conceptual pit shell. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.

**Open pit mineral resources are reported at a copper equivalent cut-off of 0.13%. The cut-off is based on a price of US$3.00 per lb of copper, US$1,300 per ounce of gold, US$20 per ounce of silver. The assumed recoveries are for copper 89%, gold 70%, silver 75%, and molybdenum 60%.

14.15 Grade Sensitivity Analysis

The mineral resources at the Kwanika Property are sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the block model quantities and grade estimates are presented at various cut-offs in Table 14-18 and Table 14-19.

The presented tonnes and grade represent totals for both open pit and potential underground development. The reader is cautioned that the figures presented in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Figure 14-21 and Figure 14-22 present this sensitivity as grade tonnage curves in the Central Zone. Figure 14-23 presents grade tonnage curves in the South Zone within an open pit.

Table 14-18 Central Zone Block Model Quantities and Grade Estimates* on the Kwanika Property at various copper equivalent cut-off grades within a designed Whittle shell and within the area for potential underground development

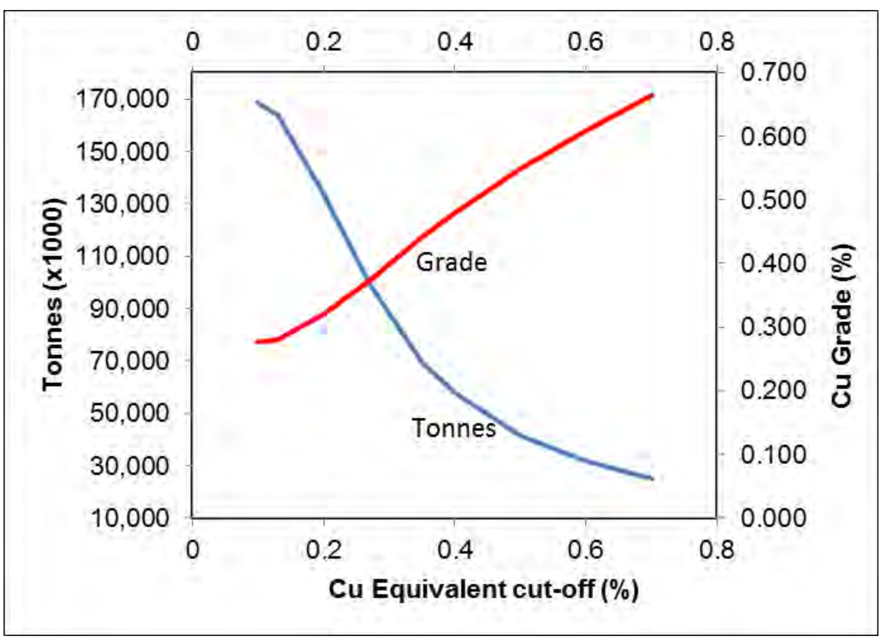
Category	Cut-off	Quantity	Grade			Contained Metal		
	CuEq (%)	(x1000 Tonnes)	Cu (%)	Au (g/t)	Ag (g/t)	Cu (000's lb)	Au (000's oz)	Ag (000's oz)
Indicated	0.70	24,800	0.66	0.83	1.90	363,100	660	1,510
	0.60	31,800	0.61	0.75	1.76	427,300	760	1,800
	0.50	41,400	0.55	0.66	1.62	502,400	880	2,150
	0.40	57,700	0.48	0.55	1.43	609,500	1,020	2,650
	0.35	69,600	0.44	0.49	1.32	677,600	1,110	2,950
	0.27	99,100	0.37	0.40	1.13	818,200	1,270	3,590
	0.20	133,500	0.32	0.33	0.98	941,800	1,420	4,210
	0.13	164,000	0.28	0.29	0.88	1,016,500	1,520	4,660
	0.10	168,600	0.28	0.28	0.87	1,024,100	1,530	4,710
Inferred	0.70	30	0.64	0.63	1.72	400	0	0
	0.60	100	0.49	0.49	1.40	1,100	0	0
	0.50	300	0.41	0.40	1.22	2,700	0	10
	0.40	1,300	0.34	0.26	0.95	9,900	10	40
	0.35	3,300	0.30	0.21	0.83	22,100	20	90
	0.27	17,000	0.23	0.17	0.68	87,200	90	370
	0.20	48,300	0.19	0.15	0.61	197,900	230	950
	0.13	76,900	0.16	0.13	0.56	268,400	330	1,380
	0.10	81,100	0.15	0.13	0.55	275,400	340	1,440

*The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Table 14-19 South Zone Block Model Quantities and Grade Estimates* on the Kwanika Property at various copper equivalent cut-off grades within a designed Whittle shell

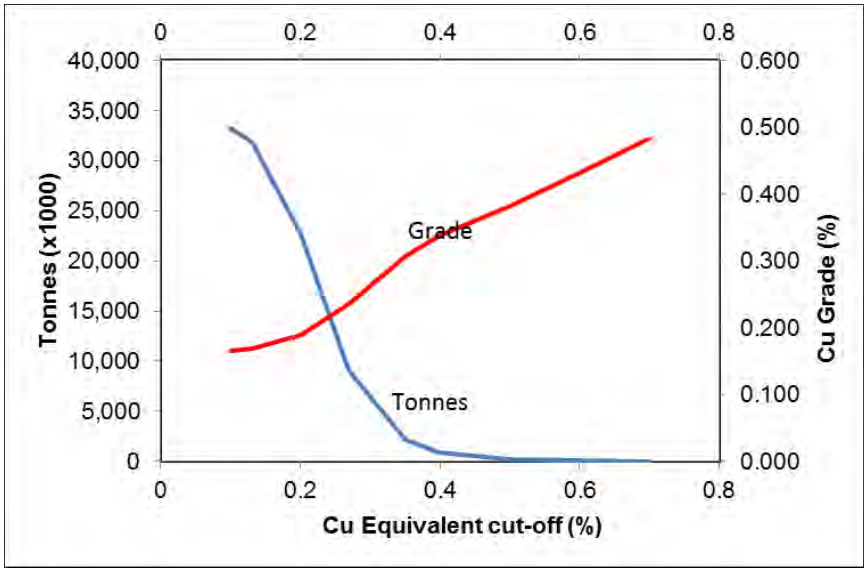
Category	Cut-off CuEq (%)	Quantity (x1000 Tonnes)	Grade Cu (%)	Grade Au (g/t)	Grade Ag (g/t)	Grade Mo (%)	Contained Metal Cu (000's lb)	Contained Metal Au (000's oz)	Contained Metal Ag (000's oz)	Contained Metal Mo (000's lb)
Inferred	0.70	100	0.59	0.18	3.23	0.02	1,400	0	10	50
	0.60	500	0.52	0.14	2.95	0.02	6,200	0	50	230
	0.50	2,400	0.45	0.11	2.70	0.02	23,600	10	200	910
	0.40	7,700	0.38	0.09	2.29	0.02	64,800	20	570	2,710
	0.35	13,100	0.35	0.09	2.09	0.01	99,800	40	880	4,120
	0.27	23,800	0.30	0.08	1.84	0.01	156,600	60	1,410	6,200
	0.20	30,500	0.27	0.08	1.71	0.01	183,600	80	1,670	7,180
	0.13	33,300	0.26	0.08	1.64	0.01	191,400	80	1,760	7,470
	0.10	33,800	0.26	0.08	1.63	0.01	192,400	80	1,770	7,540

*The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.



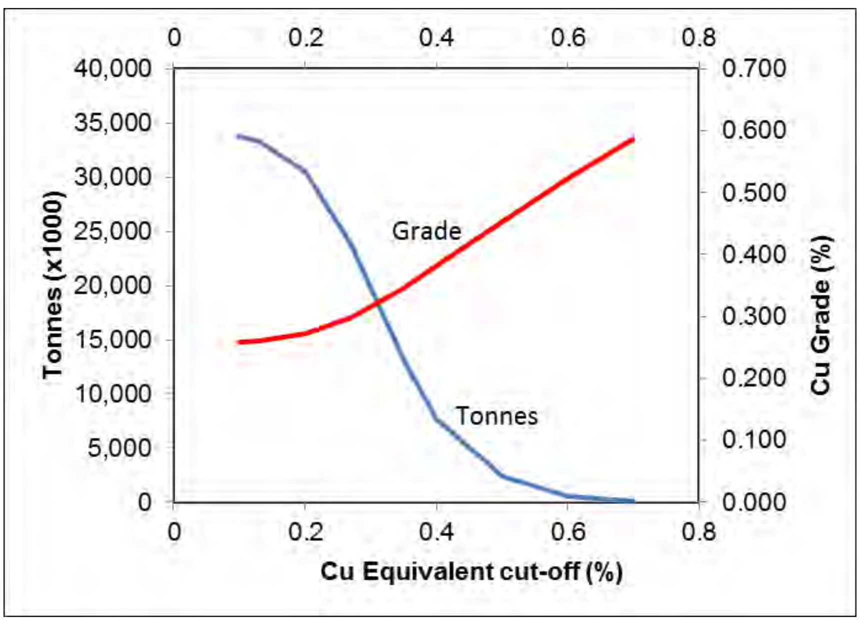
(Source: SRK, 2016)

Figure 14-21 Central Zone Indicated Category Grade Tonnage Curves



(Source: SRK, 2016)

Figure 14-22 Central Zone Inferred Category Grade Tonnage Curves



(Source: SRK, 2016)

Figure 14-23 South Zone Inferred Category Grade Tonnage Curves

14.16 Comparison with RPA 2011 Resource Estimates in the Central Zone

To compare the SRK resource estimates with the RPA 2011 resource estimates (Rennie, 2011), SRK tabulated the RPA estimates within the currently designed pit (Table 14-20). The RPA results can be compared with the SRK estimated tonnage and grade presented in Table 14-21.

As can be seen within the Whittle shell, the SRK estimate reports higher gold and copper grades and higher metal content for the Indicated category. In the Inferred category estimated copper grades are quite similar and SRK estimated gold grades are lower, with the overall metal contents similar for both copper and gold. For the material, potentially mineable by underground methods, the SRK estimate reports similar tonnage and grades.

Table 14-20 Central Zone RPA Estimates* within the SRK Whittle Shell

Category	Cut-off	Quantity	Grade			Contained Metal		
	CuEq (%)	(x1000 Tonnes)	Cu (%)	Au (g/t)	Ag (g/t)	Cu (000's lb)	Au (000's oz)	Ag (000's oz)
Indicated	0.70	14,284	0.655	0.715	1.892	206,182	329	869
	0.60	19,782	0.595	0.627	1.740	259,389	399	1,107
	0.50	27,781	0.531	0.542	1.554	325,070	484	1,388
	0.40	39,940	0.464	0.457	1.348	408,165	587	1,731
	0.35	48,049	0.429	0.416	1.238	454,959	642	1,913
	0.27	65,298	0.373	0.352	1.058	537,671	739	2,221
	0.20	84,588	0.326	0.302	0.916	607,746	821	2,492
	0.13	100,946	0.292	0.268	0.825	649,776	871	2,677
	0.10	105,717	0.282	0.259	0.800	658,211	882	2,719
Inferred	0.70	106	0.581	0.446	1.787	1,360	2	6
	0.60	232	0.501	0.421	1.587	2,565	3	12
	0.50	673	0.398	0.387	1.179	5,908	8	26
	0.40	2,207	0.317	0.330	0.843	15,435	23	60
	0.35	3,992	0.286	0.291	0.715	25,162	37	92
	0.27	9,648	0.240	0.231	0.575	50,985	72	178
	0.20	18,165	0.202	0.189	0.515	81,045	110	301
	0.13	27,860	0.172	0.155	0.458	105,782	139	411
	0.10	31,869	0.161	0.143	0.430	113,219	147	441

*The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

Table 14-21 Central Zone SRK Estimates* within the SRK Whittle Shell

Category	Cut-off CuEq (%)	Quantity (x1000 Tonnes)	Grade Cu (%)	Grade Au (g/t)	Grade Ag (g/t)	Contained Metal Cu (000's lb)	Contained Metal Au (000's oz)	Contained Metal Ag (000's oz)
Indicated	0.70	18,560	0.664	0.847	1.926	271,680	505	1,149
	0.60	24,217	0.608	0.749	1.783	324,509	583	1,388
	0.50	31,712	0.550	0.656	1.630	384,333	669	1,662
	0.40	43,463	0.484	0.551	1.439	463,310	769	2,011
	0.35	51,299	0.450	0.499	1.335	508,539	823	2,202
	0.27	69,431	0.389	0.413	1.156	595,886	921	2,580
	0.20	87,456	0.343	0.355	1.035	661,887	997	2,910
	0.13	101,498	0.312	0.319	0.955	697,238	1,043	3,116
	0.10	104,446	0.305	0.313	0.938	702,155	1,050	3,151
Inferred	0.70	10	0.483	0.555	1.400	111	0	0
	0.60	63	0.432	0.469	1.257	603	1	3
	0.50	220	0.382	0.383	1.149	1,857	3	8
	0.40	914	0.338	0.251	0.925	6,807	7	27
	0.35	2,095	0.306	0.203	0.829	14,158	14	56
	0.27	9,129	0.236	0.172	0.671	47,431	51	197
	0.20	22,621	0.190	0.154	0.618	94,575	112	449
	0.13	31,872	0.169	0.138	0.591	118,465	141	606
	0.10	33,162	0.165	0.135	0.585	120,745	144	624

*The reader is cautioned that the figures in this table should not be misconstrued with a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of cut-off grade.

14.17 Resources based on Open Pit and Underground Designs (MMTS)

Open pit and underground economic mining limits have been developed at a scoping level of detail, based on the resource model provided by SRK as described above. As a PEA, the economic mining limits have included revenues from Inferred resources. The full mining plan is described in Section 16 and details of the pit and stope designs are included in Appendices C &D respectively. The resultant design delineated Resources are present below in Table 14-22 and Table 14-23.

Table 14-22 Summarized Indicated and Inferred Pit Mill Feed by Phase

Pit Phase	Class	ktonnes	NSR ($)	Cu%	Au g/t	Ag g/t
C621	IND	4,697	$49.33	0.482	0.516	1.315
	INF	159	$25.20	0.287	0.184	0.795
C622i	IND	7,055	$30.02	0.299	0.301	0.918
	INF	50	$20.70	0.247	0.128	0.755
S621	INF	6,666	$23.00	0.307	0.048	1.918
S622i	INF	11,628	$20.38	0.249	0.082	1.479
S623i	INF	6,525	$21.05	0.251	0.094	1.607
UG Phase	**Class**	**ktonnes**	**NSR ($)**	**Cu%**	**Au g/t**	**Ag g/t**
West West	IND	4,248	$61.48	0.606	0.632	1.626
	INF	123	$29.49	0.288	0.305	0.924
West	IND	6,040	$46.47	0.437	0.515	1.339
	INF	126	$21.60	0.231	0.185	0.771
Tall	IND	29,033	$47.11	0.441	0.527	1.355
	INF	30	$23.80	0.300	0.118	0.840
East	IND	2,089	$34.47	0.396	0.245	1.042
	INF	386	$22.30	0.276	0.123	0.629

The combined pit and underground delineated resource by assurance of existence class is shown in Table 14-23.

Table 14-23 Pit and Underground Delineated Resource by Assurance of Existence Class

Class	*Mill Feed*	*NSR ($)*	*Cu%*	*Au g/t*	*Ag g/t*
Indicated	53,162	$45.62	0.437	0.492	1.301
Inferred	25,693	$21.35	0.265	0.079	1.600

For both the above Tables:

- In the open pits, the NSR cut-off used is \$11.30/tonne with a provision for mining loss of 5% and dilution of 2%
- For the underground, no loss and dilution quantities are added to the reported tonnes calculated from the 3D stope shapes. Block Cave bulk mining is planned and as such there is no possibility of selectively separating ore and waste in the stope. Therefore, the stope shape is considered as the boundary of all extracted material after dilution and all internal waste is included. It is considered to be the material that reports to the draw point and is then loaded and hauled to surface. It is therefore the underground mill feed tonnes with reported grades that include loss and dilution.
- All mineralized material classified as Indicated (67%) and Inferred (33%) Mineral Resources has been considered in the mine plan. The PEA is preliminary in nature and it includes Inferred mineral resources that are considered too speculative geologically to have the economic consideration applied to them that would enable them to be characterized as mineral reserves. Mineral resources that are not mineral reserves, do not have demonstrated economic viability and there is no certainty that the results of the PEA will be realized.

15 Mineral Reserve Estimates

The current study is at a PEA level and therefore there are currently no Mineral Reserves estimated for the Kwanika Project.

16 Mining Methods

16.1 Introduction

A production schedule based on a 15,000t/d mill feed rate has been developed for the Kwanika Project. The mine production is based on both Open Pit and underground Block Cave mining methods. The Life of Mine (LoM) plan starts mining two open pit phases in Central Zone while access and level development are established for the Block Cave beneath the open pit in the Central Zone. The final stage of mining is from 3 open pit phases in South Zone. See Figure 16-1. This General Arrangement shows the major project elements including:

- Project access road from the north
- The combined TSF and RSF to the west
- A water storage pond
- Central Pit with Central underground beneath
- South pit phase
- A plant site
- A camp

These major facilities are placed on reasonable locations. Future studies will further assess and optimize the layout. Other minor facilities will be located as required in future studies.

This is a contractor-operated Open Pit mining fleet and underground development mining fleet with an owner's fleet planned for underground mucking and hauling. Contractor mining operating costs have been included. Capital costs for the contractor fleets are included in the operating cost. The purchase cost of the underground loaders and trucks are included as a capital expense.

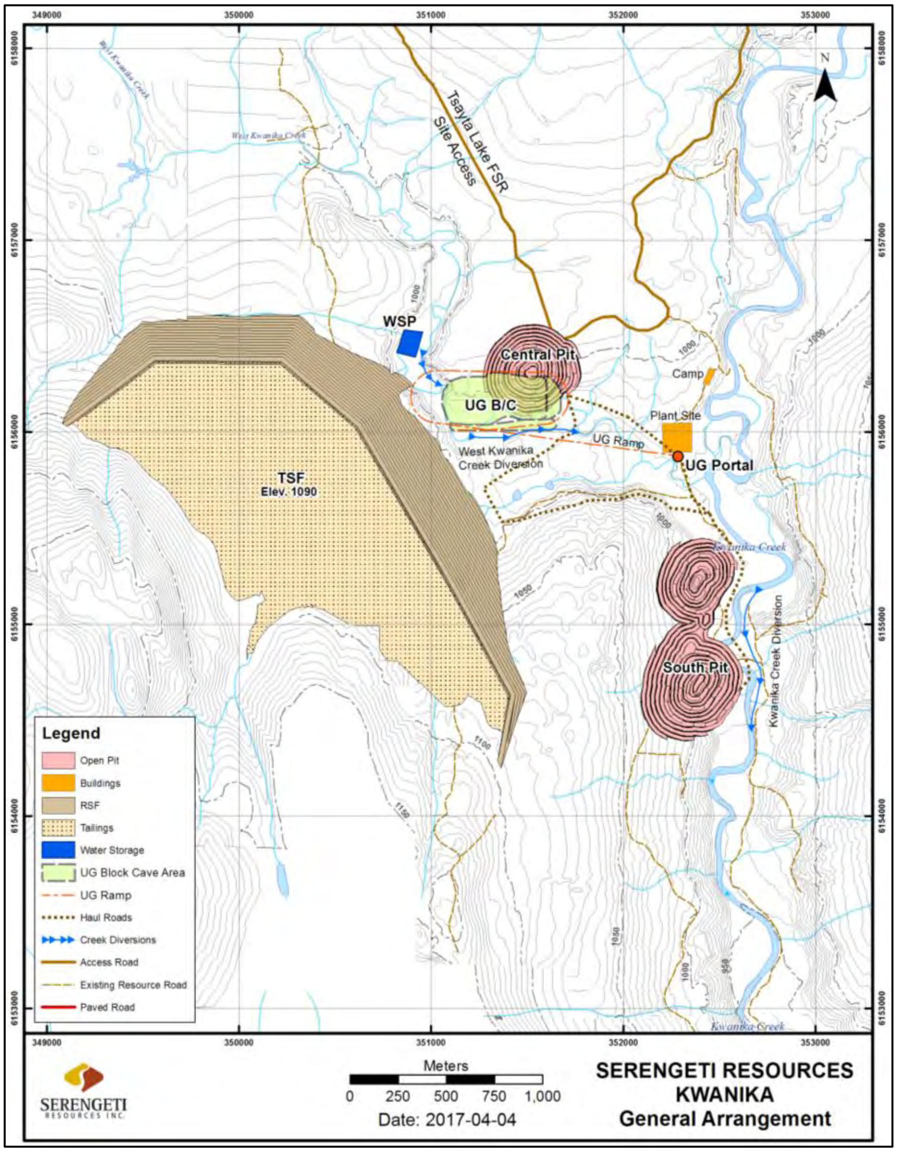


Figure 16-1 General Arrangement of Mine Area at End of Mine Life

The mine planning work for this study is based on the 3D block model (3DBM) created by SRK for the NI 43-101 Mineral Resource Estimate dated December 2016. The mine engineering uses MineSight® software well proven in the industry, and includes converting the SRK resource model to MineSight, pit optimization, detailed pit design, and optimized production scheduling.

In addition to the geological information used for the block model, other data used for mine planning includes the base economic parameters, mining cost data derived from local projects, MMTS' cost database and some of the basic project parameters from the 2013 PEA, such as throughput rate.

The following sections present some of the basic aspects common to the Underground and Open Pit mining engineering work, and the resultant mining components of the mine plan. These in turn are the basis of the mine costs. The background information of certain aspects of the mine plan, the pit phasing or extents of the underground design for example, are presented in more detail in Appendices C and D.

16.1.1 Mining Datum

The project design work is based on NAD83 coordinates. The historical drillhole information is based on various surveys with different sets of control that have been converted to NAD83. The resource model by SRK uses a DEM created from LiDAR data collected in September 2016. A larger area, low resolution topography surface has been used for mine planning.

16.1.2 Production Rate

Several factors have been considered to establish an appropriate mining and processing rate. These include:

- **Resource Size:** A mine life of 12.5 to 20 years to find an optimal the NPV for the Project, and a capital payback period of 3 to 5 years.
- **Operational Constraints:** Ensure power, water, critical supplies, and limited infrastructure don't limit production rate.
- **Construction Constraints:** Physical size and weight of equipment and shipping limits are possible at the location of the Project.
- **Project Financial Performance:** Economies of scale are realized at higher production rates, and lead to reduced unit operating costs. These are revised to meet above mentioned physical, capital, and operational constraints.

Throughput studies for the 2013 PEA indicated a throughput of 15,000t/day was appropriate for the size of the mineable resource at that time. The revised resource model in this study is similar in size, so a throughput of 15,000t/d is used in the 2017 study. If the mineable resource base is significantly increased in future studies, the NPV advantage of a higher throughput rate should be investigated.

16.2 Mine Planning – General

The mine planning aspects of the Project include both open pit and underground design and engineering. The following information is common to both.

16.2.1 3D Block Model

The resource models from SRK have been loaded into the MineSight mine planning model and engineering and financial items have been added. The items in the Central and South zone models are described in Table 16-1 below. It is also noted if the item is supplied by SRK or added by MMTS for mine planning purposes.

Table 16-1 Central 3DBM Items

Item	Item By	Description
ROCK	SRK	Rock Type Code
SG	SRK	(tonnes / m^3)
OPT	SRK	Ore Percentage (%)
CU	SRK	Copper Grade (%)
AU	SRK	Gold Grade (g/t)
AG	SRK	Silver Grade (g/t)
CUEQ	SRK	Calculated Copper Equivalent (%)
CLASS	SRK	Resource Category (1=Measured; 2=Indicated; 3=Inferred)
TOPO	MMTS	Percent of Block Below Surface (%)
NSR	MMTS	Net Smelter Return ($/t), based on initial 2016 values
CUEQ2	MMTS	Calculated Copper Equivalent (%), based on initial 2016 values
MO	SRK	Molybdenum Grade (%)
NSRO	MMTS	Net Smelter Revenue ($/t), based on 2013 PEA values
NET	MMTS	Block Value for Audit ($)
NSR16	MMTS	Net Smelter Revenue ($/t), based on secondary 2016 values
CUQ16	MMTS	Calculated Copper Equivalent (%), based on secondary 2016 values
NET16	MMTS	Block Value for Audit using NSR16 ($)
BCPCT	MMTS	Percent of Block Within Block Cave (%)
NSR17	MMTS	Net Smelter Revenue ($/t), based on PEA metal values

Table 16-2 South 3DBM Items

Item	Item By	Description
ROCK	SRK	Rock Type Code
SG	SRK	(tonnes / m^3)
ORPT	SRK	Ore Percentage (%)
CU	SRK	Copper Grade (%)
AU	SRK	Gold Grade (g/t)
AG	SRK	Silver Grade (g/t)
CUEQ	SRK	Calculated Copper Equivalent (%)
CLASS	SRK	Resource Category (1=Measured; 2=Indicated; 3=Inferred)
TOPO	MMTS	Percent of Block Below Surface (%)
NSR	MMTS	Net Smelter Return ($/t), based on initial 2016 values
CUEQ2	MMTS	Calculated Copper Equivalent (%), based on initial 2016 values
MO	SRK	Molybdenum Grade (%)
NSRO	MMTS	Net Smelter Revenue ($/t), based on 2013 PEA values
NET	MMTS	Block Value for Audit ($)
NSR17	MMTS	Net Smelter Revenue ($/t), based on PEA metal values

16.2.2 Net Smelter Revenue

The Net Smelter Revenue (NSR) is calculated for each block in the 3DBM. This accounts for offsite charges and process recoveries. The NSR is used as a cut-off item for break-even mill feed/mine rock selection and for the grade bins for cash flow optimization. The offsite charges are applied to each metal to generate a Net Smelter Price (NSP). The NSP is based on base case metal prices, US dollar exchange rate, and offsite transportation, smelting, and refining charges (see Appendix C for details).

Metallurgical recoveries used for the NSR calculations are based on test work done by SGS Metallurgical Services Ltd and estimates by MMTS, and are detailed in Section 13. The metal prices and resultant NSPs used are shown in Table 16-3.

Table 16-3 Metal Prices and Resultant NSPs for NSR17

Metal	Market Price	NSP	Recovery
Copper	$US 2.90/lb.	$ 3.29/lb.	89%
Gold	$US 1270/g	$ 48.71/g	70%
Silver	$US 19.00/oz	$ 0.67/g	75%

Note - These prices and recoveries were used for NSR17 which in turn is the basis of the NSR cutoff used for the Pit Resources calculations. Different recoveries were used in the Cash Flow model.

The NSR calculation is as follows:

NSR ($/t recovered) = dolval_Cu + dolval_Au + dolval_Ag

Where:

Dolval_Cu = Cu% / 100 x (NSPCu) x 2204.62 lb/tonne x (RecCu)

Dolval_Au = Au g/tonne x (NSPAu) x (RecAu)

Dolval_Ag = Ag g/tonne x (NSPAg) x (RecAg)

NSP = Net Smelter Price (as above)

Rec = Recovery %

16.2.3 Mining Parameters

Open Pit

Mining dilution for the open pit is based on whole block dilution from the grade interpolation with 2% additional dilution on ore/waste boundaries. In dilution material, the grade is conservatively set at zero. In future studies and during operations, the dilution will be assigned grades marginally below the cut-off grade at the cut-off boundary.

An allowance for a mining loss of 5% has been used to account for material lost at the ore/waste boundary, misdirected loads, and spillage during hauling.

The pit delineated resources are calculated from the 3DBM within the detailed pit designs using an NSR cut-off of $11.30/tonne. The mining recoveries and dilution described above, convert the in-situ resource material tonnages into a ROM mill feed used in production scheduling.

Underground

No loss and dilution adjustments are included in the mining resource for the underground. Block Cave bulk mining is planned and as a bulk mining method, there is no possibility of selectively separating ore and waste in the stope. Therefore, the stope shape is considered as the boundary of all extracted material after dilution and all internal waste is included. Therefore, by using all the material within the 3D stope shape it can be considered to be the material that reports to the draw point and equivalent to the mill feed tonnes with grades that include loss and dilution. No further loss and dilution parameters need to be applied.

16.3 Underground Mining

16.3.1 Selection of Mining Method

The selection of block cave mining as the preferred underground mining method was made in the 2013 preliminary economic assessment carried out by Moose Mountain Technical Services. This 2017 study has looked at different underground mining methods at various cut-off grades and based on the AMEC evaluation from 2013, that indicated that the North Zone has the potential to successfully cave, the conclusion remains that block caving is still the best approach to mining the Kwanika underground deposits. A preliminary optimization process has been carried out to determine the location of the extraction level, both laterally and vertically, with the goal of maximizing the ore grade of the rock mass to be mined. A more detailed geotechnical assessment and block cave optimization study is warranted at higher levels of study.

16.3.2 Block Cave Stope Design

A general mining outline for the block cave stope has been delineated. Note that the 3D shape used for the block cave is the outside limit after caving is completed. As mentioned above, the resultant tonnes and grade therefore are inclusive of mining loss and dilution.

At more advanced levels of study, a production schedule would be optimized by drawing ore from higher grade areas of the stope earlier in the schedule on a drawpoint by drawpoint sequence. At this level of study some allowance for this grade optimization has been accomplished by dividing the stope into four distinct mining domains so that production can be scheduled from higher to lower grades over the block cave life. Further Block Cave optimization is required in the future studies mentioned above. The domains or cave blocks are listed in Table 16-4 below.

Table 16-4 Block Cave Mining Domains

Property	W-West	West	Tall	East	Tot/Ave
Extraction Level (m):	470	470	470	520	
NSR ($/t):	60.57	45.96	47.08	32.57	47.47
Tonnage[1] (Million)	4.37	6.16	29.06	2.48	42.08
Footprint (m^2):	23,952	23,047	36,463	10,756	94,218
Average Height (m):	66	97	290	84	162
Average Width (m):	171	192	166	145	170
Average Length (m):	140	120	220	74	554

[1] Includes development ore, drawbell and undercut ore and cave ore.

Figure 16-2 below shows the general design for the Central Zone Block Cave. Ramp access starts from surface with the portal of the decline located close to the coarse ore stockpile adjacent to the mill.

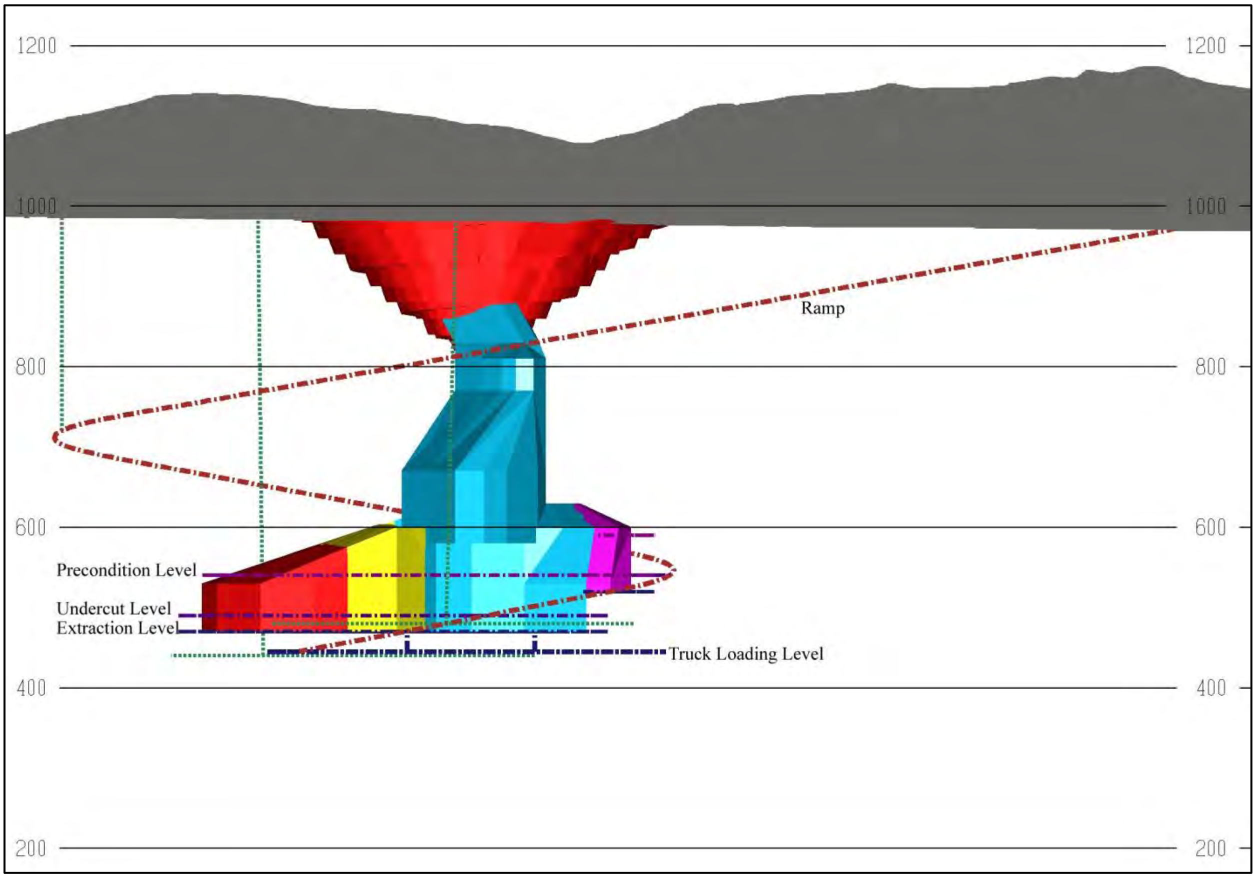


Figure 16-2 Section of Block Cave Configuration

16.3.3 Initial Access

Mining starts with the Central open pit phases while the underground development is in progress. The initial access ramp will decline at 17 % grade from approximately 1000m elevation to 540m, then to subsequent lower development levels to get the block cave ready for production. The ramp will be driven establishing remuck bays every 150 metres as well as passing bays every 300 metres. Passing bays allow for two-way traffic during both development and operations.

Sufficient level development must be completed to start underground ore production from the caving operation, before the open pit production is exhausted. Level development then is subsequently continued as the caving advances laterally across the stope foot print.

16.3.4 Development Levels

The Block Cave design is made up of the main stope where the caving generates muck for mill feed, plus the development levels under the stope that induces the cave and provides for the broken muck to be extracted and hauled to the mill. Mine Infrastructure for ventilation and dewatering are also required.

Figure 16-3 below shows more detail using a section through the 'Tall' Mining Domain.

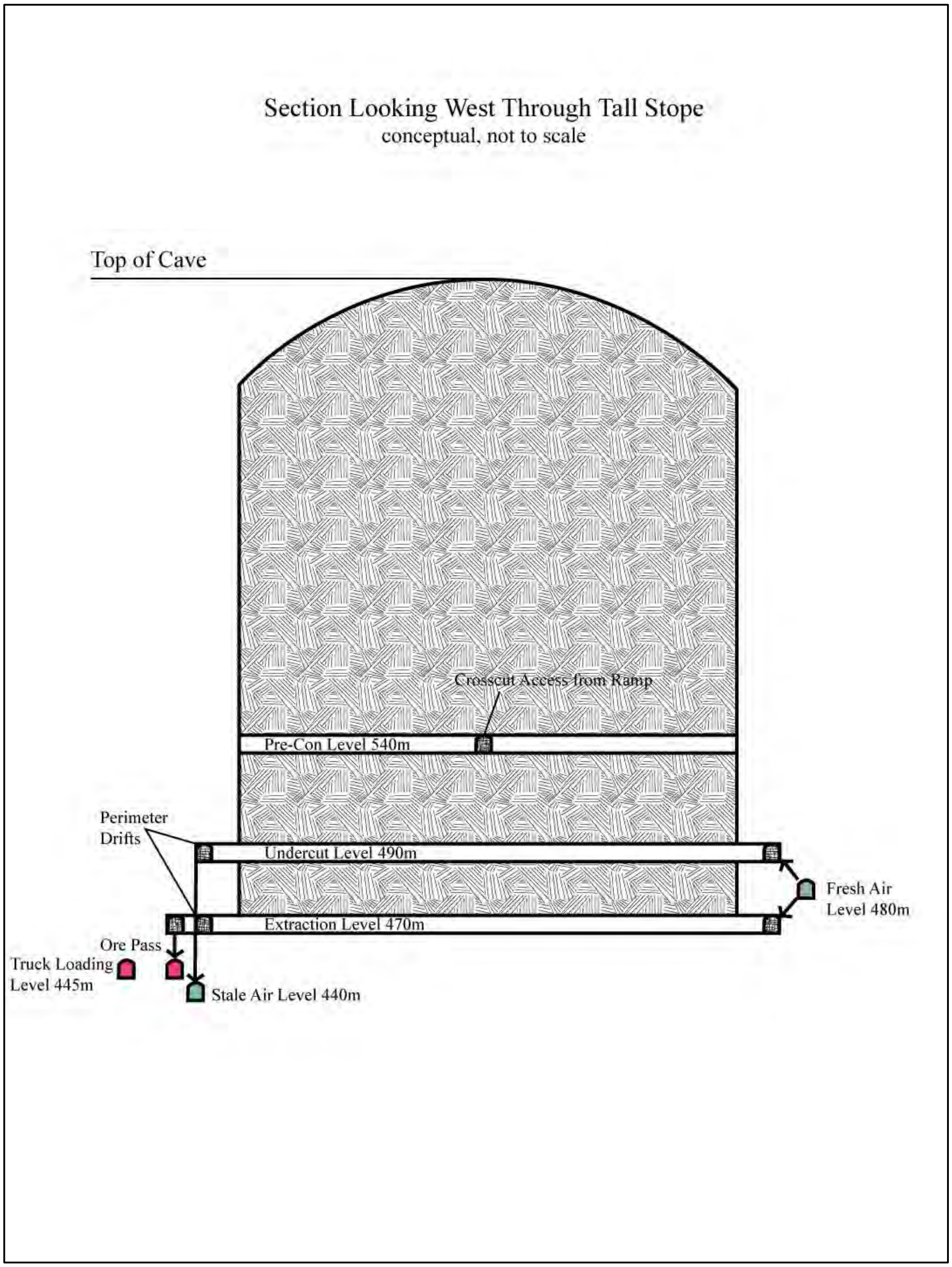


Figure 16-3 Block Cave Development Levels

A description of the components of the development levels follows. Appendix D has additional details on the underground mining design and parameters.

16.3.4.1 **Pre-Conditioning Level (540m)**

A pre-conditioning[1] level has been designed with the purpose of fracturing the rock mass within the cave to ensure successful caving. At this stage of design, the requirement for pre-conditioning has not been evaluated but has been included for conservatism.

16.3.4.2 **Undercut Level (490m)**

Development for block-caving applying conventional gravity flow requires an undercut, where the rock mass underneath the block is fractured by longhole drilling and blasting in preparation for caving.

When the undercut level is blasted, the ore body is subject to the stress of the now unsupported span thus initiating the cave for this material to be collected in the drawpoints on the extraction level below.

16.3.4.3 **Extraction Level (470m)**

LHDs will load from the drawbells on the extraction level and tram to the ore passes which each have a stationary rock breaker and grizzly. Ore then drops into an ore pocket and chute arrangement for truck loading on the level below.

16.3.4.4 **Fresh Air Level (480m)**

The fresh air level runs east-west and north of the cave footprint and will provide fresh air to the extraction and undercut levels through a series of raises. Additionally, a fresh air connection will be made to the truck loading level.

16.3.4.5 **Truck Loading Level (445m)**

The truck loading level is located directly under the two ore passes at the south end of the extraction level. This level comprises a loop from the main ramp whereby trucks will leave the ramp, travel clockwise through the loop, enter the truck loading level from the west, travel east, then will load from one of the two ore bins, before exiting the level back to the ramp and then to surface to dump on the coarse ore stockpile at the plant.

[1] Per the following document “Preliminary Caveability Assessment – Kwanika Deposit, Stephen Godden, Amec, 04 April, 2012”, MMTS has included pre-conditioning in the mine design.

16.3.4.6 **Return Air Level (440m)**

The return air level will exhaust air through a series of raises from the extraction and undercut levels as well from the truck loading level. Additionally, since this level is the low point of the mine, the sumps will also be located here.

16.3.5 Underground Infrastructure

Components of the underground infrastructure required for the development and production phases of the underground operations, will be installed before the access is started and as the development progresses as required. These general underground mine facilities are described below.

16.3.5.1 **Power Supply, Catenary, Heating and Ventilation Systems**

The main power supply will be located in the fresh air raise and will be distributed to various locations for auxiliary ventilation, pumping and lighting, the stationary rockbreakers at the grizzlies on the extraction level, the parking area, shop, and the dewatering pumps.

A catenary line on the decline will also be included to provide to power assist to the electric haul truck fleet. This will increase the travel speed, reducing the required number of trucks. It will also provide lower cost energy to the trucks considering the BC power grid services the mine. It also improves air quality underground which in turn reduces the ventilation requirements.

16.3.5.2 **Ventilation & Heating of Mine Air**

The primary ventilation circuit comprises two 5m diameter borehole raises from surface, of which one will be fresh air and the other, exhaust air. The fresh air raise will bring fresh air down to the fresh air level for distribution to the working sites. The air will then be directed through the undercut and extraction drifts to the return air drift.

This same circuit also provides ventilating air to the maintenance shop and parking area on the extraction level.

The total quantity of ventilating air is estimated to be 540,000CFMs supplied by two vane-axial fans located at the top of the fresh air raise. Only one fan will operate whilst the other will be on standby. A propane-fired heater will be in series with these fans. The heat required for the mine air is based on the expected temperature rise necessary to heat the ambient air to -2°C prior to its introduction underground.

16.3.6 Underground Operations

Unit operations for the Underground activities comprise the following:

16.3.6.1 Development of Undercuts and Drawbells

Undercutting and drawbell development will be carried out using longhole drilling and blasting techniques. Drawbells for production will be developed at the rate of 32 drawbells per year. All of the undercutting will be in ore while approximately 52% of the drawbell development will be in ore with the remainder in waste. Development of undercuts and drawbells is included in operating costs and will be carried out using owner's personnel and equipment.

16.3.6.2 Production Mucking

Between four and five 17-t load-haul-dump units will be required to tram from designated drawpoints on a shift by shift basis, to one of two ore passes. Each of the load-haul-dump units will work with a mobile rockbreaker and two blockholers to bring down hang-ups in the drawpoints. Secondary blasting will take place in the drawpoints if the oversize is too large for the rockbreakers to handle.

The two ore passes will be located outside the cave footprint on the south side of the cave. Each ore pass will be equipped with a grizzly. A stationary rockbreaker will work at each grizzly to further reduce oversize and ensure the production rate. The ore passes below the grizzlies will have a capacity of 1,000 tonnes each, enough to load eighteen 55-t trucks each. All activities associated with production mucking are an operating expense and will be carried out using the owner's resources.

16.3.6.3 Ore Haulage to Surface

The selected method for hauling ore up the access ramp is with 55-t electric haul trucks with trolley-assist. The catenary line providing power to the trucks will run from the truck loading level to surface. At surface, the trucks will exit the ramp and switch to battery power to dump their loads near the gyratory crusher then return to the ramp.

16.3.6.4 Mine Development Operations

All development, ramps, raises and direct level development will be carried out by contractors.

Vertical development will include the primary ventilation raises from surface, an intermediate ventilation raise from surface during ramp development, the ventilation raises from the fresh air level to the undercut and extraction levels and the return air raises from these levels to the return air level.

Table 16-5 below shows the summary of life of mine development.

Table 16-5 Life of Mine Development

Type	Length
Ramps & Misc Excavations	16,332m
Direct Level Development	12,737m
Raises	5,214m
Total:	**34,283m**

16.3.6.5 Mine Services

Mine services will comprise the following and will be carried out by the owner's fleet and personnel:

- Road maintenance
- Mine dewatering
- Maintenance of power supply, catenary line, heating, and ventilation systems
- Delivery of fuel, explosives and shop supplies
- Rehabilitation

16.3.7 Technical Services & Supervision for Underground Operations and Maintenance

Technical services will comprise a crew of professionals including geologists, mining engineers, surveyors, drawing technicians, environmental technicians, and samplers. This department will support the operation every day (single shift) including weekends. Samplers and beat geologists will carry out their work two shifts per day.

Operational supervision will include the manager of mining, mine general foreman, mine supervisors and trainers. These personnel will support the operation every day (single shift) with the mine supervisors providing coverage two shifts per day.

Maintenance supervision will include the manager of maintenance, maintenance planners and shop foreman to provide daily support for the mining operation.

Both mechanical and electrical maintenance will be performed and charged to specific pieces of equipment, mobile and stationary and will be performed on a two shift per day basis.

16.3.8 Underground Mine Safety

Mine safety will include ongoing training of personnel from the maintenance, operations, and technical services departments. Additionally, mine rescue teams will be trained from these pools of personnel. Two sets of mine rescue gear will be available with one located on surface and the second in one of the refuse stations. Mine rescue support will be arranged with other underground mines operating within the local area.

A mine warning system will be utilized to warn personnel underground of an incident. This can be done through a radio system sending signals to the cap lamps worn by underground personnel. Generally, a signal would send personnel to the nearest refuse station. The principal means of egress from the mine will be the access ramp, which will be in fresh air. Secondary egress will be from the fresh air raise, which is accessible from each mine level and will be equipped with a ladder way.

16.3.9 Underground Mine Equipment

A typical list of mine equipment follows comprising stationary and mobile units. The mobile equipment is divided into owner and contractor.

Table 16-6 Mine Equipment

Stationary Equipment	Number of Units
UG Substation	1
UG Power Distribution	1
UG Pumping & Drainage	2
Compressors	2
Trolley Assist Catenary	3,970m
Mine Rescue Gear	2
Refuge Stations	4
Pan Feeders	2
Primary Ventilation Fans	2
Mobile – Owner	
Haul Truck (55t) trolley assisted	8
LHD (6 m^3)	2
LHD (9 m^3)	6
Longhole Drill	1
Grader	1
Personnel Carrier	3
Mobile Rockbreaker	5
Blockholer	2
Stationary Rockbreaker	2
ANFO Loader	1
Shotcrete Sprayer	1
Concrete Mixer	1
Boom Trucks	2
Pickup Trucks	6
Scissor Trucks	2

Mobile – Contractor	
Jumbo Drill Rig (2-Boom electric-hydraulic	3
Development Haul Truck - 40t	5
Development LHD (9m^3	4
Rockbolting Jumbo	2
Emulsion Loader	2
Scissor Truck	3

16.3.10 Underground Schedule (Access, Level Development and Stope Sequence)

The following describes the overall mining sequence of the development and stoping. Appendix D has additional details on scheduling and development.

16.3.11 Access and Level Development

The access ramp from surface will be collared near the process facility at the 1,000m elevation. Using a single development crew with an average advance rate of 5.5m/day, the 17% gradient ramp will reach the pre-conditioning level, elevation 540m, as indicated in the schedule below. At this point, the crew can now work in multiple headings with productivity of 8.0m/day.

Table 16-7 Schedule Showing Access to Mine Levels

Level	El (m)	Ramp Distance (m) from Surface	Time Elapsed (Days)
Pre-Conditioning Level	540	2,706	492
Undercut Level	490	3,000	545
Fresh Air Level	480	3,059	556
Extraction Level	470	3,118	567
Truck Loading Level	445	3,265	594
Stale Air & Dewatering Level	440	3,294	599

16.3.12 Stope Sequence

Once the undercut and extraction levels have been reached and sufficiently developed, production mining crews are able to commence opening up the undercuts and developing the drawbells. When the level development is advanced enough to allow caving to commence at a combined ore production rate of 6,000tpd is achieved made up of development the start of caving. After approximately 1,460 days, full production is achieved at the rate of 15,000tpd.

The cave will be initiated with the W-West domain in order to provide the highest grade first. This will be followed by initiating a second cave under the Tall domain, followed by the West domain. Lastly, the East domain, whose extraction level is located at the 520m elevation, will be initiated to complete the caving sequence.

16.3.13 Production Rate

The production rate is dependent upon the rate at which caving can take place and is dependent on establishing drawbells to match the production rate. It has been assumed that caving can proceed at a rate of 0.20 vertical metres/day, which is consistent with other technical studies carried out on caving projects in northwest B.C. Thus, approximately 60 drawbells will be required to reach a caving rate of 15,000tpd. This is achieved by developing between two and three drawbells per month for the life of mine. During this period, approximately 209 drawbells will be developed.

16.3.14 Opportunities for South Area Underground

There is a contiguous region of mineralized material beneath the South Pit. An analysis has been performed to determine if the mineralized material is suitable for underground mining (See Figure 16-4). Examination of the resource modeling also indicates the mineralized zones have some potential of being expanded with denser drilling.

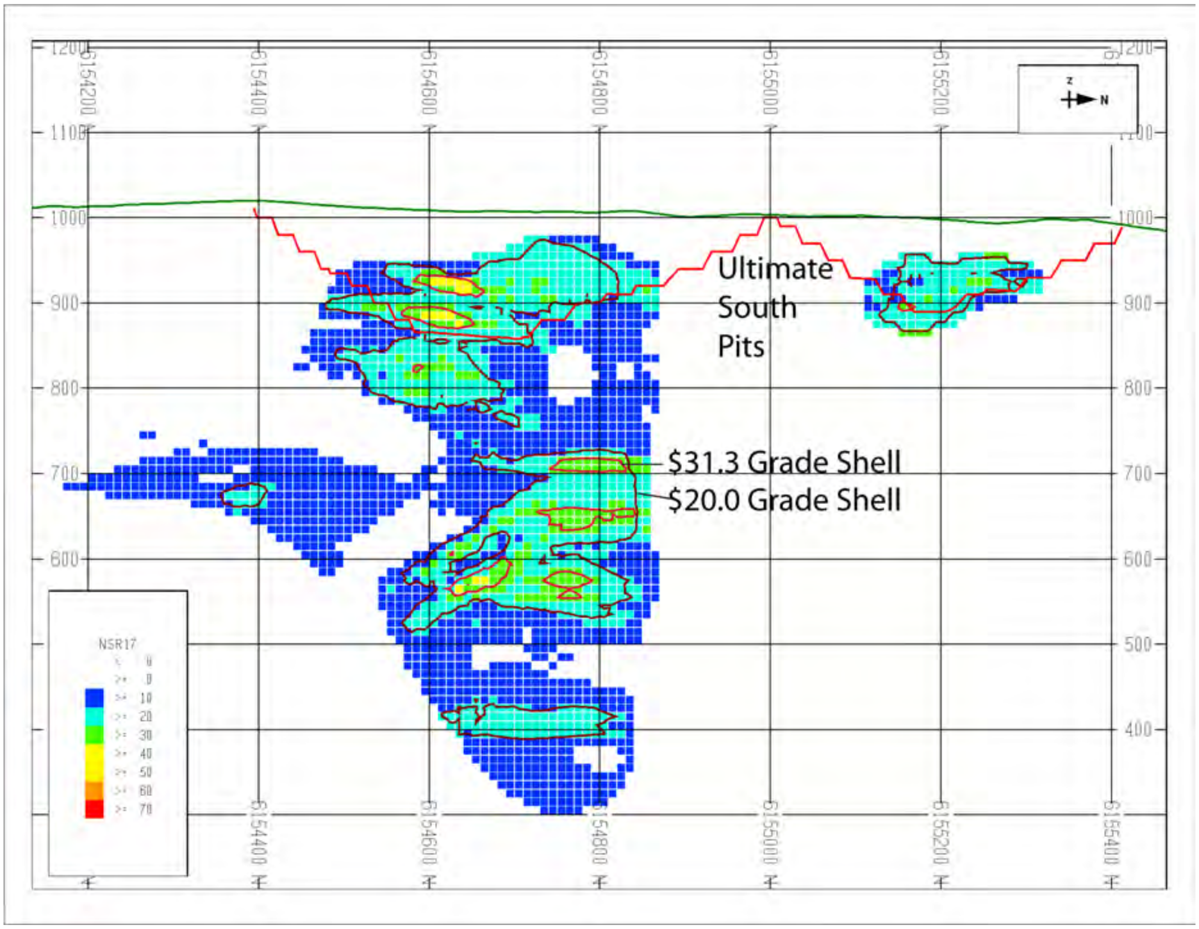


Figure 16-4 Section of South Area Including Pit and Grade Shell below Pit

Given the current level of drilling and the metal price assumptions in this report, the analysis determined that the South Underground is economically marginal and should not be included in the PEA. With additional drilling and/or higher metal prices there is potential to expand the Project resource beneath South Pit.

16.4 Open Pit Mining

16.4.1 LG Phase Selection

Lerchs-Grossman (LG) pits have been used to evaluate the economic pit limit and the optimal pushbacks or phases. A series of LG pits have been generated with varying metal prices. The lower LG price case pits provide higher margin (Revenues minus waste and ore mining) as early mining areas. In this study, two startup pit phases have been selected in the higher grade Central Zone to provide early revenues for both early capital payback and to cover costs while the development of the higher grade underground mining areas, are in progress. LG pit phases are selected using the following design constraints:

- large enough to accommodate the multiple unit mining operations of drilling, blasting, loading, and hauling
- have bench sizes large enough so the number of benches mined per year is reasonable (sinking rate)
- wide enough so the shovels can load the trucks efficiently

The Central LG cases are developed into detailed pit designs using the 45% and 65% metal price cases (See Figure 16-5 Section of Central Pit LG Phases 45%, 65%, and 100%Figure 16-5). The 45% price case is developed into Central Pit Phase 1 and the 65% price case is developed into Central Pit Phase 2. The mineralization beneath the 65% central pit was determined to be more economically mined during the Block Cave portion of the central region.

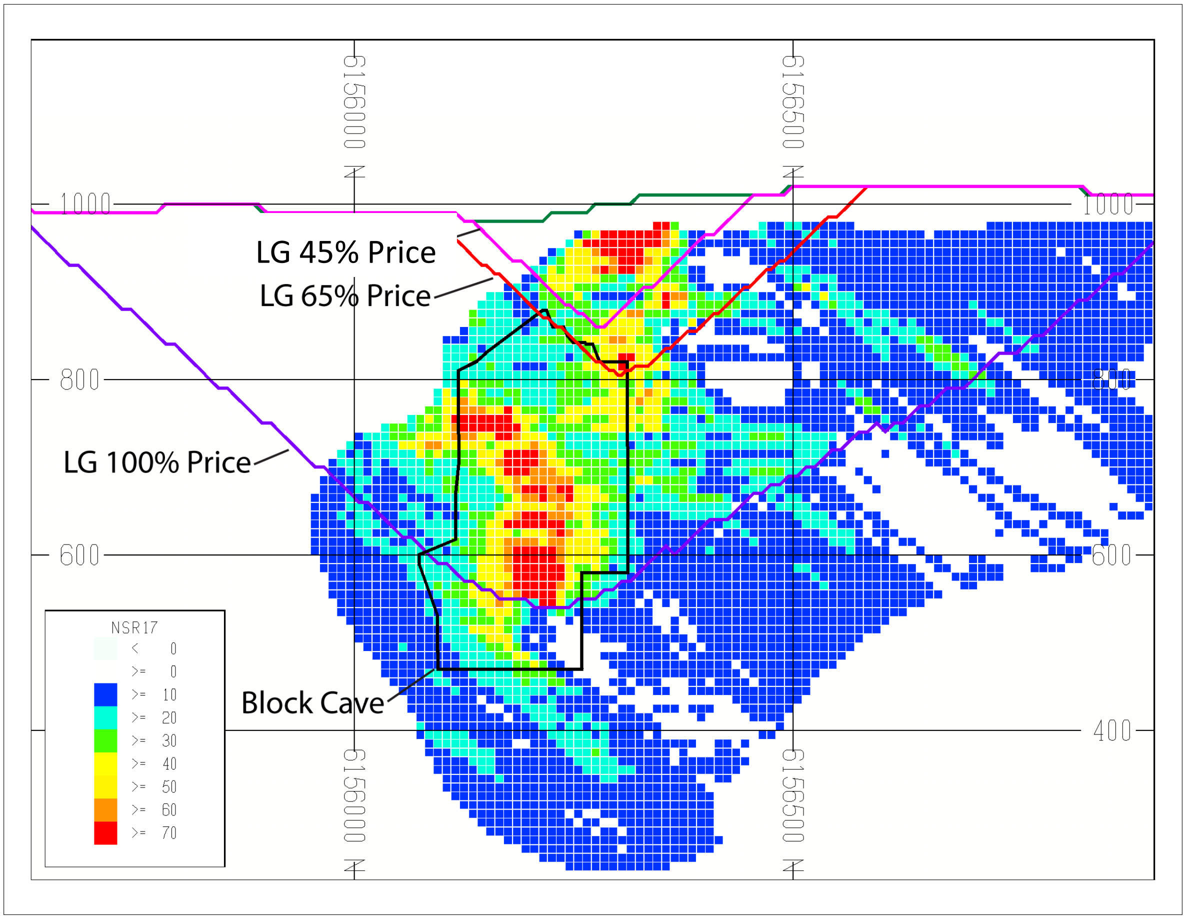


Figure 16-5 Section of Central Pit LG Phases 45%, 65%, and 100%

The South Pits will be mined after the Central pit phases and the Block Cave mining. Three pit phases are designed based on two LG cases. The South LG cases are developed into detailed pit designs using the 80% and 100% metal price cases (See Figure 16-6). The northern 100% price case is developed into South Phase 1, the southern 80% price case is developed into South Phase 2, and the southern 100% price case is developed into South Phase 3.

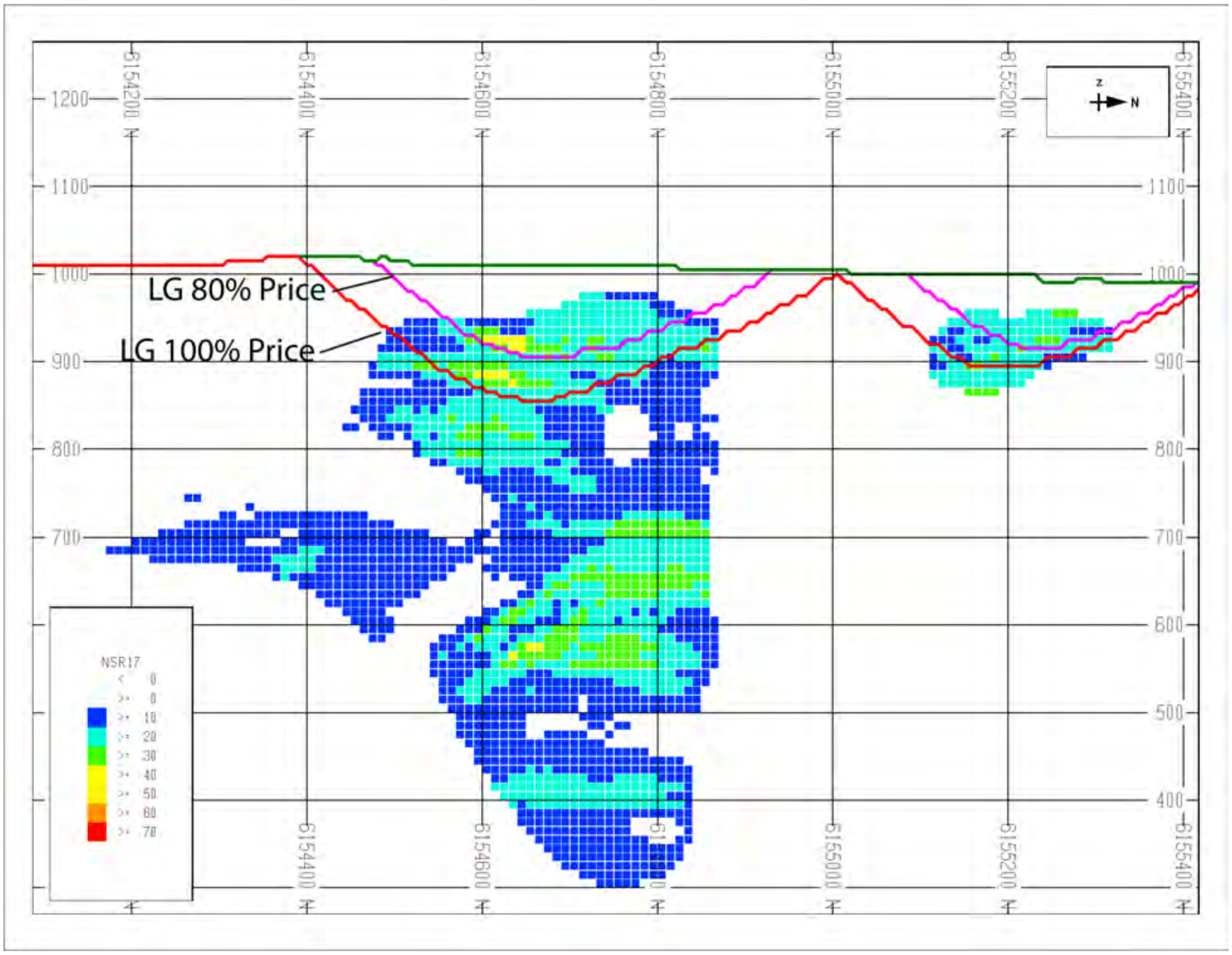


Figure 16-6 Section of South Pit LG Phases 80% and 100%

Further details regarding the LG pit cases selected as the economic pit limits for the Kwanika mine area are discussed in Appendix C.

16.4.2 Detailed Pit Designs

Scoping level pit designs demonstrate the viability of accessing and mining economical resources at the Kwanika site. Geotechnical assessment has not been started at this level of design therefore the pit phases are generated based on typical slope design parameters and using suitable road widths for the equipment size, and minimum mining widths based on efficient operation for the size of mining equipment chosen for the Project. The parameters are listed in the Table below.

Table 16-8 Pit Design Parameters

Parameter	Value
Overall Pit Slope	
Overall Pit Slope angle (PSA)	40°
Bench Detail	
Bench Face Angle	70°
Bench Height	10m
Minimum Safety Berm Width	8m
Safety Berm Vertical Spacing	20m
Additional Criteria	
Minimum Mining Width Between Phases	100m
Minimum Mining Width Operational (i.e., at pit bottoms)	30m
Ramp Grade	8%

Note at this level of study the bench level detail is not based on geotechnical evaluation but is only used as typical values to meet the typical overall pit slope parameters.

16.4.2.1 Kwanika Pit Phases

The Kwanika pit design includes two phases for Central Zone (C621 and C622i) and three phases for South Zone (S621, S622i, S623i). Access to each bench is provided by ramps built into the high walls. See Figure 16-7.

The description of the detailed pit design phases uses the following naming conventions:

- The prefix “C” indicates Central Zone,
- The prefix “S” indicates South Zone,
- The first digit signifies the original LG Pit Case used,
- The middle digit signifies the revision number,
- The last digit signifies the pit phase number,

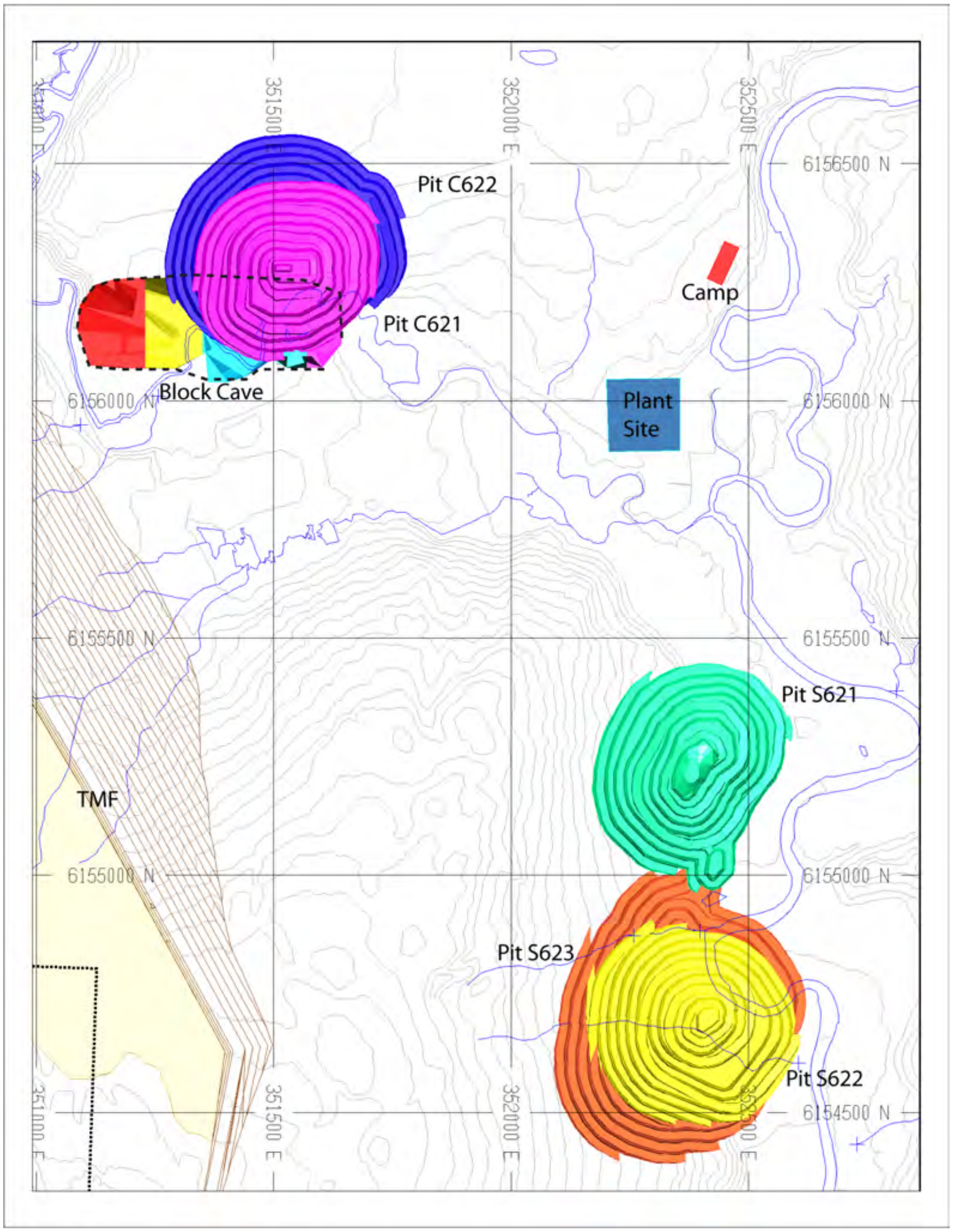


Figure 16-7 Pit Phases

Kwanika Central Phases C621, C622i

Mining of Phase C621 and C622i begins during pre-production. Any ore encountered during pre-production is stockpiled. The design intention of the pre-production phase is to expose mineralized material for the mill start-up with a minimum waste strip and then continue into higher margin mill feed during the payback period. The continuation of open pit mining into a third Central pit phase has been curtailed in this plan, in favor of starting up the higher-grade Block Cave as early as possible.

Kwanika South Phases S621, S622i, & S623i

The Kwanika South Pit is generally lower grade material and so the start of the pre-strip for the south pit is delayed until the last year of UG production in Year 9. A stockpiling strategy is still used to continue deferring low-grade material and advancing high-grade material. South Pit Phase 1 is a small northern pit in the southern resource. It is scheduled to be mined first to allow for scheduling flexibility. Mining South Phase 1 first allows the opportunity to backfill waste from the later phases. In this study, backfilling of South Phase 1 is not done.

A north-south section view of all the Kwanika Central Pit phases is shown in Figure 16-8. A north-south section view of all the Kwanika South Pit phases is shown in Figure 16-9.

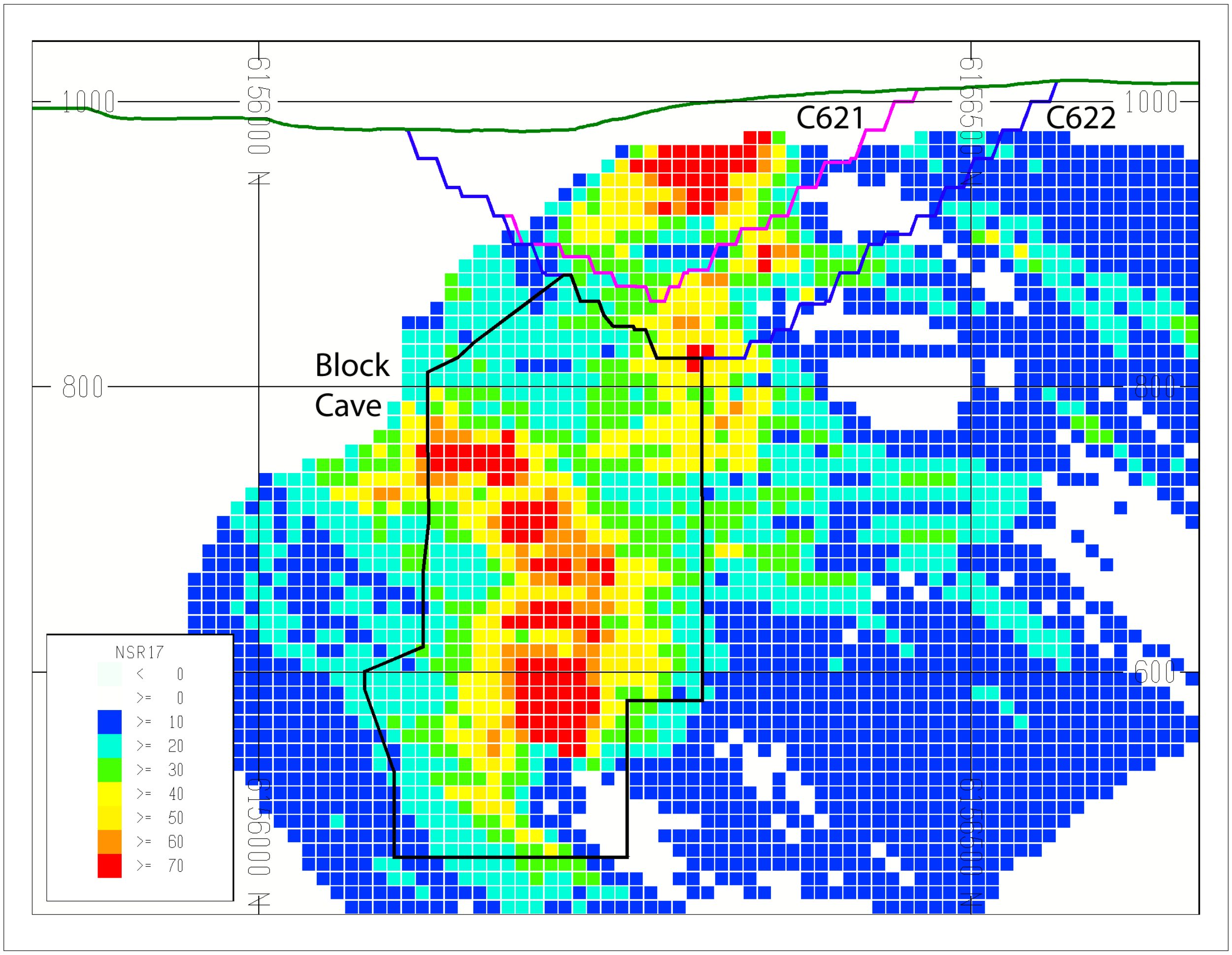


Figure 16-8 North-South Section View of all Central Pits at East 351520 – Looking West

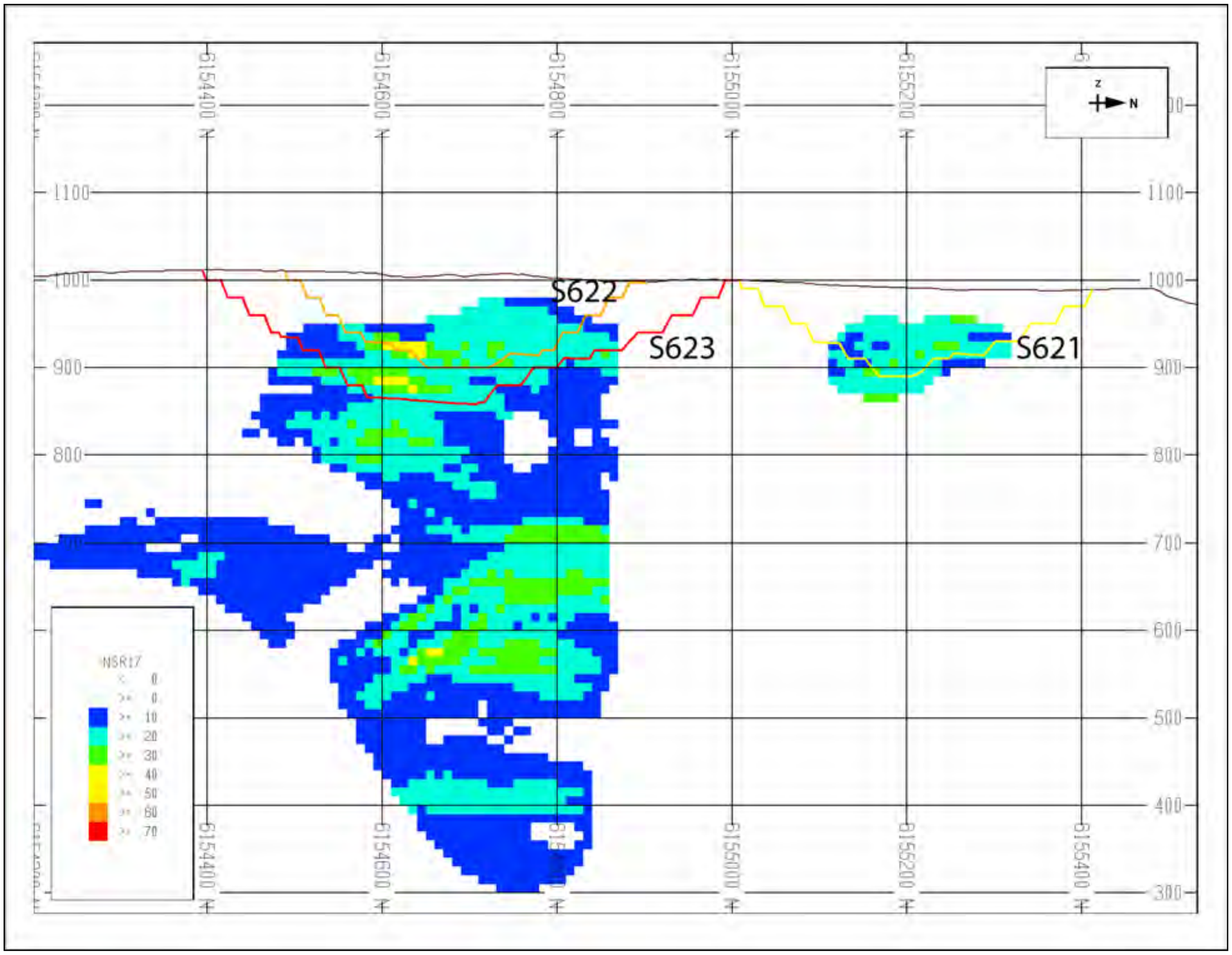


Figure 16-9 North-South Section View of all South Pits at East 352300 – Looking West

16.4.2.2 Rock Storage Facilities (RSF)

All RSFs are designed with a free dumped natural angle of repose of 37°, with the overall stepped slope at an overall slope 3H:1V to allow any final re-sloping at minimal cost. A 20% swell factor is applied to in bank volumes to determine placed volumes in the RSFs.

Mine rock placement is done using primarily bottom-up in 30-60m lifts. The RSF is used as a confining buttress for the Tailings Storage Facility (TSF). The RSF rock buttress is a similar distance as alternative RSFs. An opportunity exists to backfill South Phase 1 (S621) in the later part of the mine life.

Topsoil salvage and foundation preparation will be required. An estimate of the extent of foundation preparation and topsoil salvage will be performed in future studies.

Further details are available in Section 18.77.

16.4.2.3 ROM Stockpiles

There are stockpiles throughout the mining schedule. Long term stockpiles are placed close to the primary crusher to maximize the grade of the plant feed and smooth strip ratio and fleet requirements. In the mine production schedule, the stockpiles reach a maximum size of 7Mt.

16.4.2.4 Construction Fill

Site preparation allowance has been made for the mining equipment and pre-strip waste rock to be used for site works during the construction of the mine equipment assembly area, explosives manufacturing plant, ammonium nitrate prill storage, and explosives magazines. Construction of facilities such as the mine offices, maintenance, and fuel tanks will be completed before pre-production mining commences therefore the mine pre-production equipment can assist and reduce construction costs.

16.4.3 Open Pit Mine Water Management

Water management in the open pit mining area is concerned with ensuring efficient operating condition in the open pit operating areas. Overall site water management is covered in Section 18.5.

Where possible diversion ditches will be constructed around the open pit, RSF, TSF, and surface haul roads to keep clean water from becoming impacted. All impacted water will be channeled to sumps and settling ponds and if necessary pumped to the water storage pond for use in the operations process water or to be treated before discharge into the environment. Any drainage water from the active pit area will also be pumped to water storage pond.

16.4.4 Open Pit Mine Operations

The mining operations are typical of open-pit operations in northern British Columbia and employ accepted bulk mining methods and equipment. There is considerable operating and technical expertise, services, and support in northern British Columbia. A large capacity operation is designed and large-scale equipment is specified for the major operating functions in the mine to generate high productivities, which reduce unit mining costs. Large-scale equipment also reduces the on-site labour requirements, and dilutes the fixed overhead costs for mine operations.

The project is conceived as a contract mining operation. A unit mining cost is estimated for the Project based on previous studies and MMTS experience at typical operations, plus a contractor margin component that is on-par with current industry standards and practices. The contractor is responsible for all mining areas including direct mining and mine maintenance (see below for more details). Mine technical services, such as geology, engineering, and management will be the owner's responsibility and is captured in the general mine expense area (See below).

16.4.4.1 General Organization

The Kwanika operations are organized as illustrated in Figure 16-10. Mine operations is organized into three areas: direct mining, mine maintenance, and general mine expense (GME). Other areas of the organization are dealt with elsewhere in the report.

The direct mining area accounts for drilling, blasting, loading, hauling, and pit maintenance activities in the mine. Costs collected for this area include the mine operating labour, mine operating supplies, equipment operating hours and supplies, and distributed mine maintenance costs. The distributed mine maintenance costs include items such as maintenance labour, repair parts, and energy (fuel or electricity) which contribute to the operating cost of the equipment. The contractor is responsible for all direct mining costs and maintenance costs.

The GME area accounts for the owner's supervision, safety, and training of personnel required for the direct mining activities. GME also accounts for technical mine engineering, and geology functions. Costs collected for this area include the salaries of personnel and operating supplies for the various services provided by this function.

In this study, direct mining and mine maintenance are planned as a contractor operated fleet with the equipment ownership and labour being entirely contractor sourced. The viability and cost effectiveness of contracting can be determined in future detailed planning and commercial negotiations.

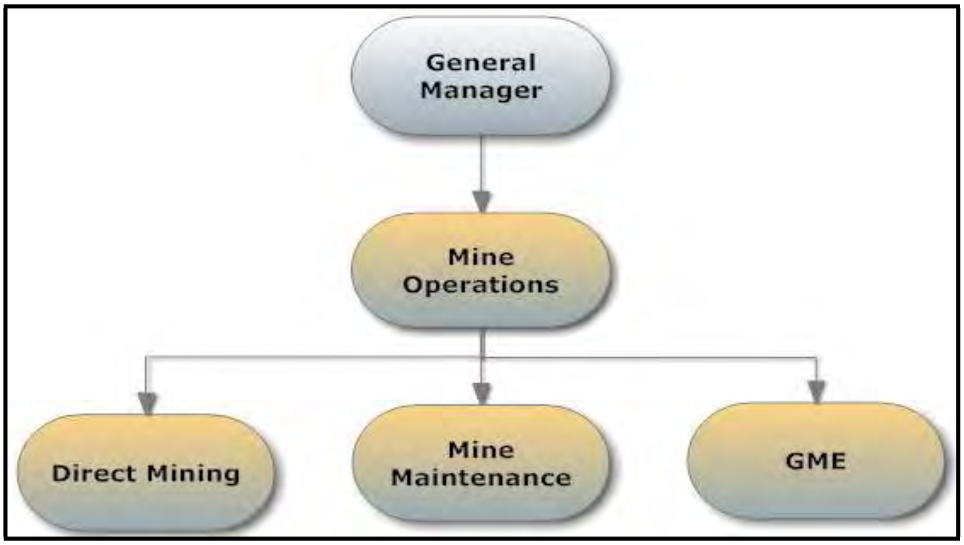


Figure 16-10 General Organization Chart

Details of the mine operation organization will be updated in future studies.

16.4.4.2 Mine Load and Haul Fleet Selection

The mine load and haul fleet has been selected in the 2013 PEA has been used for this study as well, which indicate the lowest cost-per-tonne fleet of shovels and haul trucks for this size of operation is the 15m^3 bucket capacity, diesel-hydraulic shovel matched with the 136t truck. This equipment is used to form the basis of the unit mining cost estimate. It is assumed the open pit mining will be by contractor and a contractor's uplift has been applied to the mining costs of this fleet.

16.4.4.3 Mine De-watering activities

No hydrology work has been performed to this point. At this stage of planning, an allowance for pit dewatering activities, responsibility of the mining contractor, will include the following:

- sloped pit floors as required
- in-pit sumps
- water collection system(s)

Pit water will be collected and treated to meet applicable regulations prior to discharge to the environment.

16.5 Mine Production Schedule

The mine production schedule is based starting with the first two open pit phases in Central Zone, transitioning into the Block Cave and then transitioning into open pit mining in South Zone at the end. The final mill feed is from processing the low grade stockpiles.

16.5.1.1 Schedule Inputs

The Open pit scheduling uses a cut-off grade strategy to improve cashflow at the beginning of the Project using the grade bins in the following Table.

Table 16-9 Grade bin for Cut-off Grade Optimization

	NSR Grade Bins ($/t)
Low Grade	11.30<=NSR<15
Mid-Low Grade	15<=NSR<20
Mid-Grade	20<=NSR<30
High Grade	30<=NSR

Typically, the mill feed grade can be increased by sending low and mid-grade classes to stockpiles in early periods of the production schedule. The mill feed grade is maximized and this effectively increases the revenue per tonne milled early in the schedule. The schedule optimizer develops a COG strategy to increase the Project NPV by stockpiling lower grade material for processing later in the LOM schedule, increasing mill head grades and, therefore, revenues early in the production schedule.

Note that during actual operations blast hole assays will be used for mill feed COGs and for a COG strategy. Using the five grade bins in Table 16-9 is to develop long term planning strategy. Actual mining operations will not use this many grade bins.

Further optimization of stockpile usage will be performed in future studies.

The schedule optimization utilizes the following criteria in each period to maximize the NPV for the open pit operation of the production schedule.

- Mining precedence (i.e. Phase 2 after Phase 1)
- the truck haul cycle time and resultant variable unit cost
- shovel productivity and loading costs
- estimated operating and capital costs, process recoveries, and metal prices
- 360 mine operating days scheduled per year and 24h/day
- annual mill feed of 5,400kt/a is targeted based on an average throughput of 15,000t/d

The underground production is not selective and is based on the average stope grade of the Block cave domains in the sequence they are mined.

16.5.1.2 Schedule Results

Scheduling results are presented by period as well as cumulatively and include:

- tonnes and grade mined by period broken down by material type, bench, and mining phase
- tonnes transported by period to different destinations (mill, stockpiles, and RSF)

The mine schedule considers Time 0 the time that the mill starts; the full capacity production of mill feed is expected in Year 1.

The Kwanika mine plan includes:

- UG Ramp development starting in Year-2
- Pre-stripping Central Zone pit phases C621 and C622i starting in Year-1
- OP production from Central zone pit phases C621 and C622i from Year 1-Year 2,
- UG production from Year 1-Year 9
- OP production from South Zone Pit phases S621, S622i & S623i from Year 9-Year 14
- A final year of production from stockpile in Year 15

The summarized production schedule results are shown in Table 16-10. Tonnes and grades are Run-of-Mine (ROM) from the resources reported in Figure 16-11. Full results are in Appendix C.

Table 16-10 Life of Mine Production Summary

Open Pit Production			MSSP Period		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
			YEAR	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
		Totals	Units																	
Total UG + OP Mill Feed		**78,855**	**kTonnes**		**-**	**5,401**	**5,400**	**5,400**	**5,400**	**5,400**	**5,400**	**5,400**	**5,400**	**5,401**	**5,401**	**5,401**	**5,401**	**5,401**	**5,401**	**3,248**
	Cu	0.381	%		-	0.558	0.529	0.434	0.441	0.441	0.441	0.440	0.441	0.408	0.283	0.307	0.268	0.259	0.209	0.173
	Au	0.357	g/tonne		-	0.599	0.577	0.510	0.526	0.525	0.526	0.522	0.526	0.376	0.130	0.059	0.080	0.097	0.106	0.095
	Ag	1.398	g/tonne		-	1.536	1.502	1.330	1.354	1.353	1.354	1.349	1.354	1.171	1.437	1.732	1.561	1.615	1.233	0.893
Mining Schedule by Phase																				
C621 Waste					**9,005**	**3,287**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
C621 Direct Mill Feed		**3,506**	**kTonnes**		-	**3,506**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu	**0.54**	**%**			0.545														
	Au	**0.57**	**g/tonne**			0.565														
	Ag	**1.48**	**g/tonne**			1.476														
C622i Waste					**4,235**	**5,471**	**1,441**	-	-	-	-	-	-	-	-	-	-	-	-	-
C622i Direct Mill Feed		**1,507**	**kTonnes**		-	**156**	**1,350**	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu	**0.45**	**%**			0.274	0.466													
	Au	**0.53**	**g/tonne**			0.495	0.536													
	Ag	**1.42**	**g/tonne**			1.418	1.425													
UG Development Mill Feed		**90**	**ktonnes**		-	**37**	-	**31**	-	**22**	-	-	-	-	-	-	-	-	-	-
	Cu	0.456	**%**			0.515		0.441		0.377										
	Au	0.469	**g/tonne**			0.565		0.526		0.226										
	Ag	1.308	**g/tonne**			1.467		1.354		0.978										
UG Level Development Mill Feed		**730**	**ktonnes**		-	**135**	**140**	**125**	**105**	**33**	**114**	**78**	-	-	-	-	-	-	-	-
	Cu	0.470	**%**			0.597	0.485	0.435	0.441	0.441	0.440	0.377								
	Au	0.512	**g/tonne**			0.622	0.545	0.512	0.526	0.526	0.520	0.226								
	Ag	1.368	**g/tonne**			1.606	1.416	1.333	1.354	1.354	1.346	0.978								
UG 470 Cave Production Mill Feed		**41,255**	**ktonnes**			**1,313**	**3,910**	**5,244**	**5,295**	**5,345**	**5,286**	**5,322**	**5,400**	**4,141**	-	-	-	-	-	-
	Cu	0.452	**%**			0.597	0.553	0.434	0.441	0.441	0.441	0.441	0.441	0.405						
	Au	0.516	**g/tonne**			0.622	0.592	0.510	0.526	0.526	0.526	0.526	0.526	0.354						
	Ag	1.354	**g/tonne**			1.606	1.532	1.329	1.354	1.354	1.354	1.354	1.354	1.139						
S621 Waste			**kTonnes**	-	-	-	-	**13**	-	-	-	-	-	**7,592**	**6,452**	**181**	-	-	-	-
S621 Direct Mill Feed		**5,777**	**kTonnes**	-	-	-	-	-	-	-	-	-	-	-	**2,949**	**2,828**	-	-	-	-
	Cu	**0.32**	**%**												0.301	0.343				
	Au	**0.05**	**g/tonne**												0.053	0.044				
	Ag	**2.00**	**g/tonne**												2.009	1.989				
S622 Waste					-	-	-	-	-	-	-	-	-	-	**2,484**	**7,093**	**1,124**	**92**	-	-
S622 Direct Mill Feed		**9,544**	**kTonnes**		-	-	-	-	-	-	-	-	-	-	-	**2,573**	**5,393**	**1,579**	-	-
	Cu	**0.26**	**%**													0.267	0.268	0.251		
	Au	**0.09**	**g/tonne**													0.075	0.080	0.120		
	Ag	**1.55**	**g/tonne**													1.450	1.561	1.704		
S623 Waste					-	-	-	-	-	-	-	-	-	-	-	-	**8,046**	**5,363**	**148**	-
S622 Direct Mill Feed		**5,272**	**kTonnes**		-	-	-	-	-	-	-	-	-	-	-	-	**8**	**3,795**	**1,469**	-
	Cu	**0.27**	**%**														0.230	0.263	0.302	
	Au	**0.10**	**g/tonne**														0.044	0.088	0.132	
	Ag	**1.74**	**g/tonne**														1.219	1.584	2.161	
Stk1-3	**Stockpile Mined**	**11,174**	**kTonnes**		**382**	**3,131**	**3,434**	-	-	-	-	-	-	-	**393**	**1,353**	**961**	**1,343**	**176**	-
	Stockpile Reclaimed	**11,174**	**kTonnes**		-	**253**	-	-	-	-	-	-	-	**1,260**	**2,452**	-	-	**28**	**3,932**	**3,248**
	NSR	**$18.46**	**$/Tonne**			$79.85								$12.26	$24.85			$15.74	$15.26	$15.16
	Cu	**0.232**	**%**			0.690								0.418	0.261			0.179	0.175	0.173
	Au	**0.184**	**g/tonne**			1.006								0.446	0.224			0.103	0.096	0.095
	Ag	**0.928**	**g/tonne**			2.048								1.278	0.748			0.851	0.886	0.893

Figure 16-11 shows the LOM mill feed production schedule:

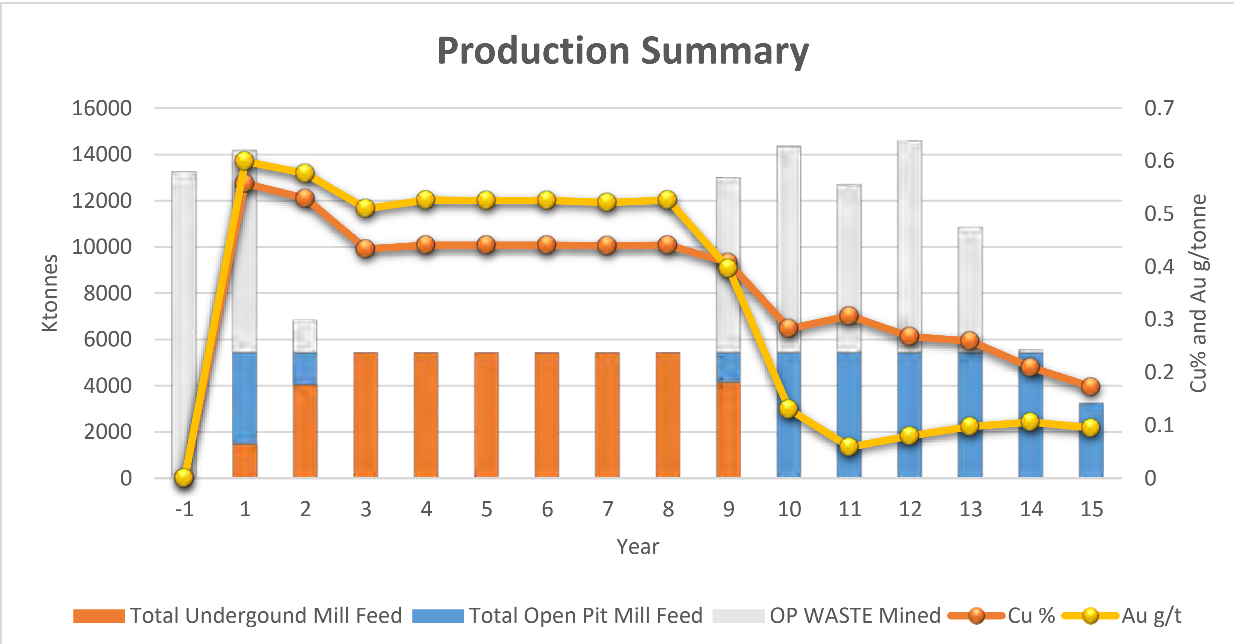


Figure 16-11 ROM Mill Feed Sources and Mill Head Grades for Feed Cu and Au

17 Recovery Methods

A conventional copper-gold flotation process is proposed for the Kwanika project including crushing, grinding, and multi-stage froth flotation to produce a copper concentrate with gold and silver credits. The process is supported by preliminary test work discussed in Section 13. For this scoping study, molybdenum is not included in evaluating the economics of the Project, but is an opportunity for future studies.

Typical process flowsheets can be applied to the Project at this scoping level study.

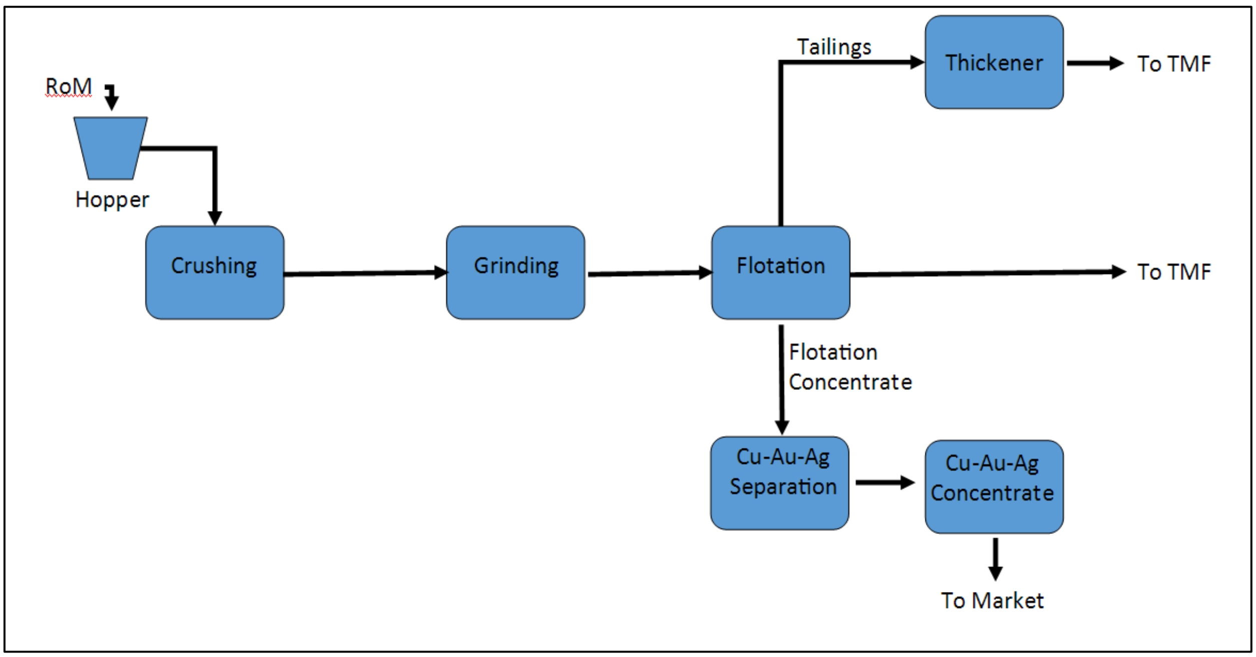


Figure 17-1 Conceptual Process Flowsheet

18 Project Infrastructure

The following section discusses the Project infrastructure including site access, power supply, on-site buildings and services, on-site electrical power supply and distribution and water management systems. These are all developed to account for the location of the Project site making effective use of services already in the local area, as well as accounting for specific onsite needs of the proposed operations within the geographic and environment setting of the site.

18.1 Site Access Road

The Kwanika Property is approximately 140km northwest (approximately 200km by road) of Fort St. James. Currently there is an existing forest service road (FSR) between Fort St. James and the Tsayta Lake Road providing surface access to the site (see Figure 5-1).

For the purposes of this study, the approximately 30km long Tsayta Lake Road will be upgraded to meet the needs of the operation to accommodate bulk freight delivery by tractor-trailer units

18.2 Permanent Electrical Power Supply

Permanent electrical power supply is by means of a transmission line to the site's substation from the Kemess Power Line approximately 75km from the Kwanika Project site. This assumes interaction with BC Hydro's power network, transmission line right of way and proposed design concept.

18.3 On-Site Infrastructure

On-site infrastructure includes:

- On-site roads
- Maintenance and truck shop
- Administration/dry building
- Camp facilities
- Cold storage warehouse
- Communication systems
- Fuel storage and dispensing (diesel)
- Assay laboratory
- Medical facilities
- Housing and permanent camp facility
- Site utilities
- Water and sewage systems
- Water supply
- Wastewater treatment systems
- Tailings Storage Facility
- Water Storage Pond

A high level layout of the on-site infrastructure is shown in the Figure below.

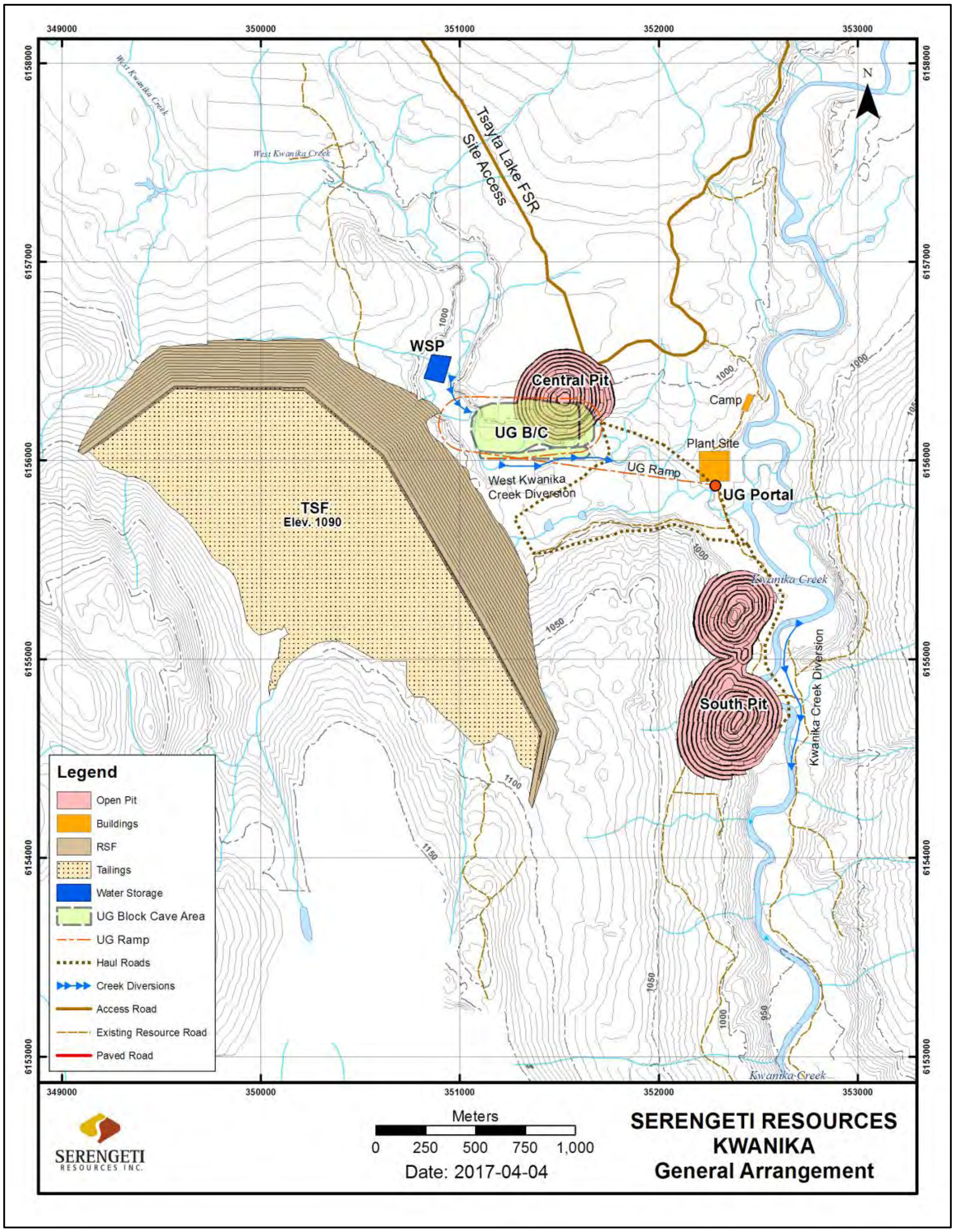


Figure 18-1 General Site Layout and Site Infrastructure

18.3.1 On Site Roads

Service Roads

These are differentiated from haul roads in that they are not wide enough for production activities by the off-highway mining equipment. Service roads are designed to access mine facilities including the TSF, and for maintenance traffic between the two mine pit locations. These roads are wide enough for light vehicle traffic, or, in some cases will be marginally wider to accommodate access to maintenance shops for the mine production equipment. Future studies will show these roads in more detail as the General Arrangement becomes finalized.

Haul Roads

Used for the large, off highway mining equipment, these roads are built between the pit, the primary crusher at the plant site, the RSF, and the TSF and will be constructed with run of mine rock as a sub-base with a top course of crushed mine rock.

18.3.2 Maintenance and Truck Shop

The truck shop and maintenance building will be provided by the open pit contract miners. Due to the short duration of the open pit mining in the production schedule, this will not be a permanent building.

18.3.3 Administration/Dry Building

The administration and dry building will be a modular building supported on concrete spread footings, complete with furniture and equipment.

18.3.4 Cold Storage Warehouse

The cold storage warehouse will be a pre-engineered sprung steel structure with an un-insulated fabric cover. The building will be supported on pre-cast concrete lock blocks on a prepared gravel surface

18.3.5 Communications System

A satellite-based system will be needed for external voice and data communications services. An on-site network will be established that will connect buildings and radio transceivers will be used for remote monitoring and control. An ultra-high frequency (UHF) radio system or something similar will be used for mobile communication.

18.3.6 Fuel Storage and Distribution

The primary project diesel fuel storage will be in two bulk storage tanks located near the truck shop complex. Typically, this facility is provided to the Project as part of the fuel supply contract. For inclement weather or other service disruptions, fuel storage should provide approximately two weeks of storage capacity. The fuel tanks will be located in a dedicated, lined containment area sized to hold 110% of the tank volume. Filling of the diesel storage tank will be accomplished by diesel tanker trucks using a truck unloading pump system located at the diesel storage facility.

18.3.7 Assay Laboratory

The assay laboratory will be a pre-fabricated modular structure located close to the mill building or within the mill building. The building will house all necessary equipment for metallurgical grade testing and control.

18.3.8 Medical Facilities

The Administration complex will include a first aid room that is fully equipped to handle standard first aid requirements, diagnostics, and emergency medical treatment. The facility will be staffed by a registered nurse who is fully trained for an assignment in a remote northern location. Physicians off-site will also provide remote support as required. Persons with serious medical conditions will be evacuated to an established medical facility. The Kwanika site is not considered remote since there is reasonable road access. If a medical emergency cannot be evacuated by the regional air ambulance service due to severe weather conditions, it is assumed that the on-site ambulance will provide evacuation by road to Fort St. James.

18.3.9 Permanent Camp

On-site personnel including management personnel, general labour, and visitors will be housed in camp accommodations on site, although, some local personnel may elect to travel to the site for work on a daily basis. The camp will be operated motel style; that is, personnel will check into their room when they arrive on-site and check out of their room when they leave site. Camp size will be finalized in the next phase resulting from the PFS execution plan.

The camp will be shipped to site in pre-fabricated modules for single storey assembly on site. The modules will be placed on wood crib block foundations. Construction of the camp will take 3 - 4 months, with an 8-person construction crew.

18.4 Site Utilities

18.4.1 On-site Electrical Substations and Power Distribution

Power is delivered from off-site via overhead lines to an electrical substation on-site. Site power will be distributed to various modular electrical rooms on site from the substation by means of an overhead line to the following areas:

- Primary crushing
- Tailings/water management
- Explosive manufacturing
- Permanent camp
- Maintenance/truck shop
- Underground facilities

The primary distribution switchgear will be located inside the main substation area. The total estimated running load for all site facilities is approximately 25 MW. Secondary system voltages utilized will include for major drives and secondary distribution, motor control centers, long-line piping heat tracing, and lower voltages for lighting, instrumentation, controls and general usage.

Electrical substations next to the plant will be fed by overhead lines and insulated cables via duct banks. Pipe racks will also be used where possible for major cable tray routes within the plant area. Cable trays that are at grade level and exposed will have high visibility covers for awareness and mechanical protection. The line will also service the primary crushing and mining facilities, as well as a line that will service water supply stations, tailing, the explosives plant and the waste management facility.

Pre-fabricated and pre-assembled electrical sheds will be utilized to house all electrical distribution equipment.

18.4.2 Building Services

All process areas will be heated to a minimum temperature of 5^{0}C during the cold season, by providing propane-fired heating units along perimeter walls and above doorways.

All staff-occupied areas will be heated to a minimum of 20^{0}C during the cold season, by supplying filtered and tempered outdoor air mixed with return air. The air will be distributed through ductwork.

Plumbing, fire protection and dust control will be provided as per national codes and accepted industry practices.

Separate heavy vehicle and light vehicle fueling stations will be provided at the diesel storage facility.

18.4.3 Sewage Systems

Wastewater and sewage generated on-site will be treated at the sewage treatment plant. Treated effluent from the sewage treatment plant will be compliant with provincial and federal regulations.

All potable water that is on-site for domestic use is expected to report to the sewage treatment plant for treatment prior to discharging to the environment. Therefore, it is assumed that the volume of sewage generated is equivalent in volume to the amount of potable water produced on site for domestic use.

18.4.4 Service Water Storage, Treatment and Distribution

Water will be required at the site to meet the following fresh water demands:

- Concentrator water requirements: plant area wash down, road dust suppression, gland seal water, reagent make-up water, and at the beginning of operations, the initial process water make-up.
- Fire water at the concentrator (site fire truck).

18.4.5 Potable Water Supply, Storage and Distribution

Potable water will be required to meet demands for drinking, food preparation, clean-up in kitchen and dining facilities, personal hygiene (toilets and/or urinals, sinks and showers), laundry, and for safety shower/eye wash stations.

Fresh water will be treated in the Potable Water Treatment Plant to meet the criteria of local and national water quality regulations and guidelines.

18.5 Water Management

Water management describes and compiles processes that enable the effective management of water to meet the needs of the operation and control any required discharge into the surrounding environment. This requires the review and interpretation of existing climatic data to estimate the rainfall, snowfall, evaporation, and sublimation expected at the site and the impact of the operations and disturbance from mining activities. Where possible, diversion ditching will be used to keep clean water clean. Figure 18-2 illustrates the site wide conceptual water management flowsheet for the Project. An early task for future studies will be to start monitoring to develop a site wide water balance.

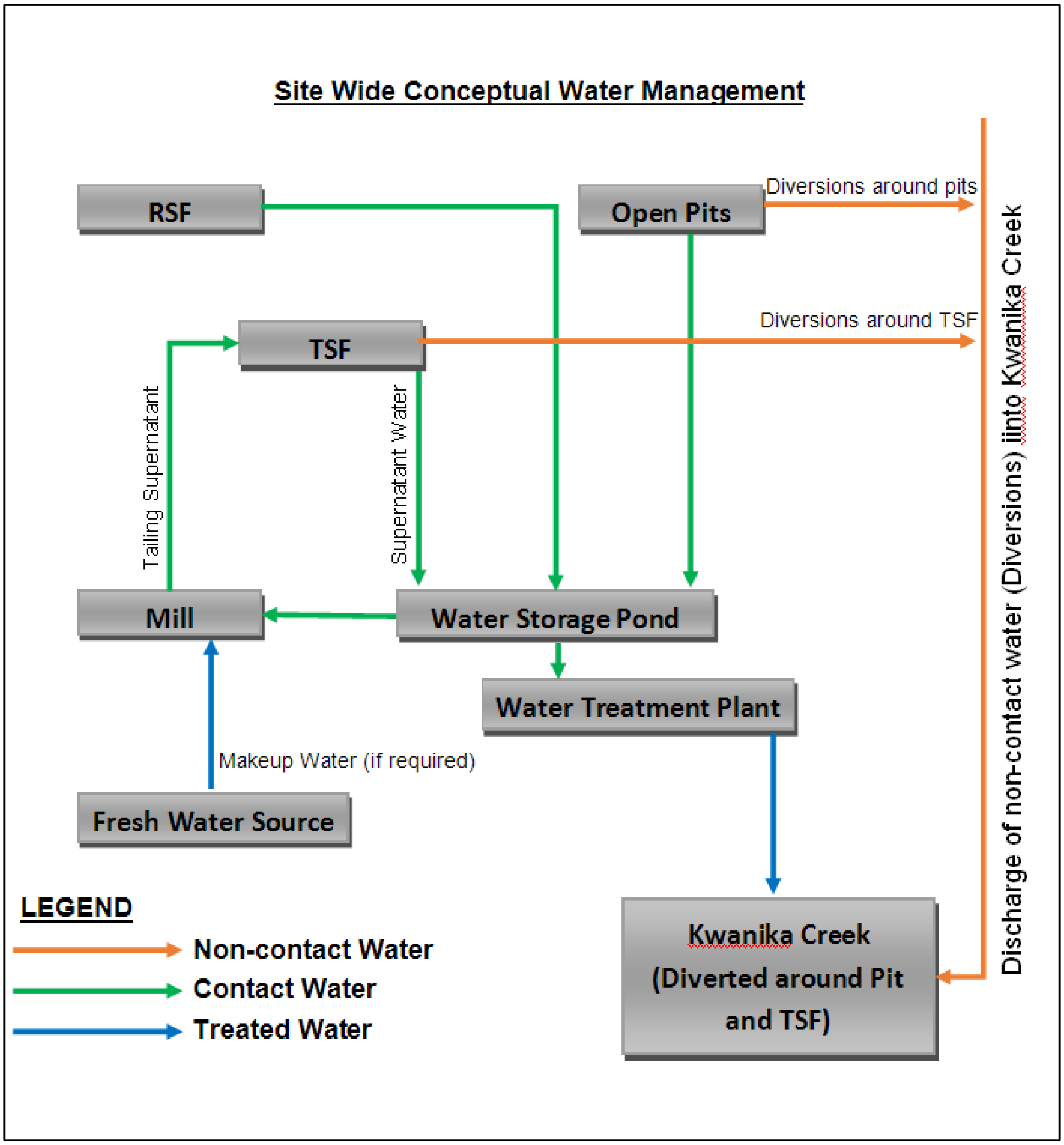


Figure 18-2 Site Wide Conceptual Water Management

To date no significant water management work has been completed for the Kwanika Project. In future studies these water management items should be considered:

- Preliminary estimated annual pit inflow quantities and quality to the proposed open pit should be performed once the open pit and underground stopes have been optimized during the next level of study,
- An estimate of the fresh water makeup for the process plant,
- A site water schematic water balance flow sheet,
- A life of mine site water schematic water balance flow sheet. This model will be used to size civil and mechanical infrastructure to support the water requirements of the Project, and
- Background hydrological and hydro-geological studies in order to develop an overall water management plan. This will also include details for the proposed diversion of Kwanika Creek.

18.5.1 Process Water Intake, Treatment, and Distribution

Process water will be used at the concentrator in grinding and flotation processes to produce a copper-gold concentrate. Plant water outputs are primarily in the tailings slurry reporting to the tailings storage facility (TSF). Water will be reclaimed from TSF to the Water Storage Pond (WSP).

Contact water will also be directed to the WSP. Water from the WSP will be pumped to the process water tank, and from there distributed to the concentrator process users.

A water balance has not been completed for this PEA.

Surplus from process or other contact water on the site will be accumulated in the WSP and will require treatment in the Water Treatment Plant (WTP) prior to discharging into the environment, and must meet federal and provincial discharge criteria and surface water quality objectives in the receiving water body.

18.5.2 Water Storage Pond (WSP)

The WSP is required during the pre-production construction stage to collect surface runoff water from the Tailings Starter Dam. This will later be used to contain decant water from the settled tailings in the TSF and seepage water from the TSF. The TSF will not be used to store mill make-up water. At closure the WSP will continue to be used until water quality meets discharge criteria and the whole system can be closed.

Construction fill materials for the WSP will be sourced from the pre-production open pit excavation and diversion channel excavations.

18.5.3 Water Treatment Plant (WTP)

In the absence of a water balance or waste rock geochemistry study a WTP has been included in the PEA design as a contingency to ensure that any surplus contact water can be treated before discharge to meet federal and provincial end-of-pipe discharge criteria (MMER) without negatively impacting the environment.

18.5.4 Kwanika Creek Diversion

Kwanika Creek is a fish-bearing stream that runs through the Kwanika leases. The current proposed TSF and Central Pit locations intersect West Kwanika Creek and require two diversions. The proposed South pit intersects Kwanika Creek and requires one diversion. Kwanika Creek Diversion Channels are shown in Figure 18-1. The diversion channel locations are chosen to maintain flow along as much of the existing Kwanika Creek and West Kwanika Creek alignments as possible and minimize the diversion length required. Future studies will need to provide an alternate assessment study for the site infrastructure and diversion requirements.

Stream diversions will need to include fisheries rehabilitation and mitigation work as required.

The consequence of exceeding the capacity of the southern diversion system will be the flooding of the South Zone Pit and the subsequent dewatering efforts in the pit. The current design has sized the diversion system with consideration of the 1 in 30 year flood, and the construction of a high soil/rock berm around the pit, local to prevent flooding of the pit bottom over the life of the mine. This assumption will require further study to ascertain the most appropriate design criteria of such an event. However, at scoping level MMTS have assumed the design measures adopted are fit-for-purpose.

18.6 Tailings Storage Facility (TSF)

18.6.1 Tailings Design Parameters

The following parameters have been used for the design of the TSF:

- A dry density of 1.3 tonnes/m3 is used to calculate tailings volume.
- Total LOM capacity of the tailings impoundment will accommodate 80Mt of mill feed.
- Tailings from the process facility will not require any special chemical suppression or management measures.
- Dam and Buttress slope 1V:3H
- Core Width: 10m
- Freeboard: 5m
- Max Height: 80m

The TSF is proposed to be located to the west of the Central Pit, adjacent to West Kwanika Creek (Figure 18-1). The key advantage of this location is the proximity of the TSF to the Central Pit and the process facility. The scoping level design for this PEA includes previous for Best Available Technology (BAT) and Best Available Practice (BAP) according to the revised BC regulations.

The project area is moderately seismic and the dam will be designed to meet CDA specification.

The TSF will be slurry tailings storage; however, water will not be stored in the TSF in order to minimize ponded water in the facility. The tailings will be pumped from the mill and spigotted into the TSF such that a beach develops between the dam and the pond. Thus, barge location is maintained and there is drainage towards the spillway on closure. Any excess water not returned to the plant will be pumped to the Water Storage Pond (WSP). As required, any excess water will be treated and discharged to the environment.

18.6.2 Tailings Starter Dam

The initial rockfill Tailings Starter Dam will be constructed with compacted rockfill and a low-permeability till core. The Starter Dam will hold six months of tailing production and have a capacity of approximately 11Mt of tailings, and be approximately 25m high (including a freeboard of 5m). The Starter Dam will be constructed in advance of mill start-up and the production of the first concentrate. A coffer dam will be constructed upstream of the TSF to facilitate the construction of the TSF from disruptive run-off in the tailing construction area and can be an alternative water storage facility to the WSP for mill start-up water.

18.6.3 TSF Construction

All mine rock from the Open Pits will be placed as a buttress for the TSF, downstream of any sand dam construction. No additional RSF locations are required. Rock fill slopes on the TSF buttress will be constructed using the bottom-up method in benches. The Ultimate Dam capacity is approximately 80Mt of mill tailings, resulting in an Ultimate Dam crest elevation of 80m (including 5m freeboard). The fill slopes will be configured close to the final re-sloped closure configuration to minimize closure costs. Dam raises are made with compacted cycloned sand cells, and a low permeability core, using the centerline method. The raises are constructed from April through October each year (see Figure 18-3).

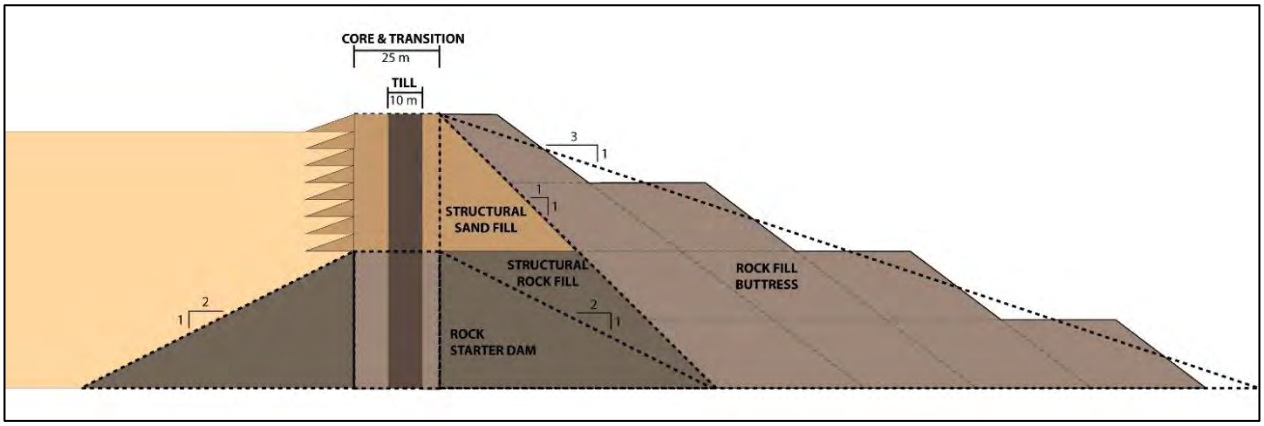


Figure 18-3 Conceptual Section of Tailings Storage Facility

The current TSF is positioned away from the Pinchi fault running between the TSF and the Central pit. The fault and the adjacent area will be subject to a thorough sub-surface geotechnical investigation in the next phase of the Project.

Water management structures such as upstream diversions, closure spillway, reclaim barge to the WSP, and seepage collection, are part of the TSF. The WSP is intended to receive excess water from the TSF, and ensure the TSF is not used for controlling excess water on site. The TSF will discharge the supernatant water into the WSP, and this water will be reclaimed back to the mill.

Critical objectives of the closure program for the TSF will be to ensure acceptable water quality into the receiving environment from the TSF/WSP/WTP system. It will also consider embankment stability, erosion control, reclamation, and closure.

18.7 Mine Rock Storage Facilities (RSF)

This section describes the Projected mine rock storage requirements for the Project:

During mining operations, mine rock production will range between 1.5Mt/a and 9Mt/a, averaging 7Mt/a. The following parameters have been used for designing the RSF:

- Direct haul from mine pit to RSF
- SG of in-situ mine rock: 2.7
- Swell factor of 20% applied for final RSF volumes
- Average embankment slopes of mine rock storage facility during mining operations: 37° with an overall reclaim slope of 26°
- Maximum mine rock bench height: 30-60m
- Total LOM mine rock to be accommodated by the RSF is 83Mt (dry basis) or 64M m^3.

It is assumed that the majority of mine rock will be non-acid generating (NAG). The remaining portion of potentially acid generating (PAG) mine rock, if present, will be strategically placed in the RSF where water infiltration can be limited. Future studies will detail the ARD/ML characteristics of the mine rock to support a final mine rock disposal plan. See Section 26 for full future study recommendations.

19 Market Studies and Contracts

There have been no market studies conducted and no contracts reached between Serengeti Resources Inc. and smelters at the time of this PEA.

Prices used in the financial model reflect recent comparable studies, the spot price is as of March 1st 2017, and the Alternative case is 10% above the base case prices. See Section 21.

Exchange

Concentrates will be sold either to North American, European, or Asian smelters and refineries. Typical TC/RC charges are assumed in the cash flow models. (See Appendix F)

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Regulatory Framework

Serengeti's Kwanika Project falls within the category of a "reviewable project" of the British Columbia Environmental Assessment Act (BCEAA), administered by the BC Environmental Assessment Office (BCEAO), and will also trigger the Canadian Environmental Assessment Act (CEAA).

The project is deemed to have a production capacity of 5.48Mtonnes/year; well over the threshold of 75,000 tonnes/year which triggers BCEA requirement. A proposed diversion of Kwanika Creek, a fish-bearing stream, will require a Federal Fisheries Act approval, and the Project will trigger the requirements of the CEAA. Other requirements of Provincial and Federal Acts and Regulations may also apply, depending upon final design components. Attempts will be made in future studies to parallel ongoing engineering work with the approval process to have less impact on the Project schedule. Additional costs of monitoring, mitigation and decommissioning may also be required as regulatory requirements change.

The current mine plan only requires limited sections of stream diversions. Future work should be diligent in considering mine plan alternatives and details that do not require federal Fisheries Act approvals.

20.2 Regional Land Use Processes

The project is located within lands that have been dedicated in the Fort St. James Land and Resource Management Plan, approved by government in 1999. The Project area is within the Multi-Value Resource Management Zone Land Use designation, where lands are managed to integrate a wide range of resource values, including mining.

20.3 Environmental and Corporate Social Responsibility Programs Already in Progress

Project related work completed to date on the environmental and corporate Social Responsibility/First Nations/Community Impact aspects of the Kwanika project is described by Serengeti below. This work has been undertaken mainly in support of the past exploration programs and will positively contribute to ongoing background studies and relationships with the other stakeholders in the area.

A project specific Valued Ecosystem Component (VEC) study was completed in collaboration with the Takla Lake First Nation (TLFN) in 2008. This study identified the principal issues that should be addressed in further environmental assessment studies on the Project.

A baseline water quality report was conducted over a ten-month period in 2008. Since that time selected drainages have been sampled and analyzed pre and post drilling activity on the property.

A project area, archaeological overview assessment (AOA) was completed in 2008. Since that time, proposed drilling plans are reviewed annually with TLFN within whose traditional territory the Project lays. Proposed drilling sites are evaluated via Preliminary Field Reconnaissance (PFR) conducted by a registered archaeologist, commonly accompanied by a TLFN elder to evaluate and minimize impact on archaeological values.

An Exploration Agreement covering activities up to and including mine construction on the Project was signed with TLFN in 2010. This agreement provided for communication protocols, project related employment and training opportunities, community capacity building and community improvements, principally around education, as well as providing for TLFN participation in any project environmental assessment process. Since that time, annual project impact mitigation strategies are implemented collaboratively with TLFN including the employment of a locally hired environmental monitor and monitoring program.

In terms of employment and contracting opportunities, local First Nations members have generally composed greater than 50% of the onsite project employment and significant contracting opportunities have been provided since 2007.

To date, no formal permitting activities have been initiated on the Project other than through the Ministry of Energy, Mines and Natural Gas, Notice of Work (NOW) process where a multiyear permit was used covering exploration activities.

20.4 Fisheries Resources and Permitting Issues

British Columbia government fisheries inventory data indicate that the following fish species are present in Kwanika Creek:

- Rainbow trout,
- Dolly Varden char,
- Burbot,
- Mountain whitefish, and
- Peamouth chub.

There are no fish obstructions listed in Kwanika Creek and therefore migratory fish species residing in the upper Nation River system including Tsayta Lake could potentially use Kwanika Creek for spawning.

Re-routing Kwanika Creek through a channel in order to develop an open pit in the Kwanika Valley would require a Section 35(2) approval from Fisheries and Oceans Canada (DFO) under the federal Fisheries authorizations required for both diversion of the watercourse and development of the open pit.

In order for DFO to issue a Fisheries Act approval the proponent would need to demonstrate the following:

- That the impact is reasonably unavoidable
- That the level of impact poses an acceptable risk to fish in consideration of DFO risk assessment criteria, and
- That the impact can be compensated for and achieve the “no net loss” policy of DFO.

The project has the potential to compromise fish habitat. Providing a diversion channel around the open pit would enable migratory fish access to upper Kwanika Creek and its tributaries, reducing the potential impact to fish. It is expected that DFO will review this stream diversion carefully and will likely require special provisions to mitigate potential impacts to fisheries resources.

At this scoping study level, the cost of the replacement of the lost fish habitat in Kwanika Creek through the construction of fish compensation measures in the diversion channel could be up to $3 Million. Detailed studies are recommended to fully identify the level of impact to fish and fish habitat, fish habitat compensation options and costs, and potential fisheries related permitting risks associated with the Project.

20.5 Closure and Reclamation

A conceptual framework for closure and reclamation is provided below. Future studies will add detail to the site-specific requirements for the Project.

20.5.1 Mine Area Closure and Reclamation

At the cessation of mining operations an approved closure plan will be implemented to return the operating area to a condition that will meet the end land use objectives.

The open pits will be allowed to fill through seepage and surface run-off. Stream run-off may be directed into the completed mining areas (open pit and Underground) to reduce the ARD and metal leaching potential as quickly as possible. Any remaining exposed highwalls will be seeded to provide a degree of revegetation. The Kwanika Creek diversion channels will either continue to operate or will be decommissioned, as required by approval conditions.

The TSF will be capped and the outer slopes of the RSF will be re-sloped to blend with the natural landscape and to enable access for wildlife. Natural seepage water collected within the WSP will be pumped and discharged as required to the open pit until the water quality meets discharge criteria.

21 Capital and Operating Cost Estimates

21.1 Capital Costs

The initial capital costs for the Project are based on capital cost estimates developed by MMTS, including mining (open pit pre-strip, and underground access development), process plant, site infrastructure, tailings storage facility and water management startup costs.

Initial capital has been designated as all capital expenditures required prior to mill start-up for producing copper concentrates for shipment to contract smelters. Sustaining capital includes replacement equipment purchases. A summary of major capital costs is shown in Table 21-1.

The estimate covers the direct field costs of executing this project, plus the indirect costs associated with design, procurement, and construction efforts.

Owner's Costs are estimated as 5% of the process and infrastructure direct capital costs.

All currencies in this section are expressed in Canadian dollars. US costs in this report have been converted using a fixed currency exchange rate of US$0.77 to CAD$1.00. The expected accuracy range of the capital cost estimate is +/- 40%.

This PEA estimate is prepared with a base date of Q1 2017 and does not include any escalation past this date.

Further details of the Basis of Estimate can be found in Appendix E.

Table 21-1 Capital Cost Summary

Direct Costs	Initial Capital Cost (KCAD)
Overall Site	$15,700
Open Pit Mining – Pre-Production	$28,600
Open Pit Mining - Equipment	$2,500
Underground Mining - Development	$39,500
Underground Mining – Direct Level Development	$4,800
Underground Mining – Equipment and Infrastructure	$35,100
Processing Plant (including Ore Handling)	$120,000
Tailing Storage Facility	$35,000
Water Management	$23,000
On-Site Infrastructure	$38,300
Off-Site Infrastructure	$18,800
Sub-Total	$361,300
Indirect Costs	
Project Indirects	$41,000
Owner's Costs	$13,000
Contingencies	$61,000
Sub-Total	$115,000
Total Initial Capital Cost	**$476,300**

Any work which is scheduled to begin after plant start-up is included in the sustaining capital costs.

Table 21-2 Sustaining and Closure Capital Cost Summary

Description	Capital Cost (KCAD)
Open Pit Mining – Sustaining	-
Underground Mining – Infrastructure and Equipment	$36,600
Closure	$46,300
Total Sustaining and Closure	**$82,900**

The detailed breakdown of this capital cost estimate is included in Appendix E.

21.2 Operating Cost Estimate

The operating costs for the Project, as shown in Table 21-3.

The operating cost estimates in this section are based upon typical costs for comparable projects in MMTS' database and from public sources. When required, costs in this report have been converted using an average currency exchange rate of US$0.77 to CAD$1.00. All costs are reflected in 2017 Canadian dollars. The expected accuracy range of the operating cost estimate is +/-35%.

Table 21-3 Operating Cost Summary

	$/tonne Mined	$/tonne Milled
Mine (OP)*	$2.97	$7.98
Mine (UG)**	$11.73	$11.73
UG Ore Rehandle		$1.09
Mining Total* **		**$10.20**
Mill		**$9.00**
G & A******		**$1.95**
Tailing Treatment		Included in G&A
Water Treatment		Included in G&A
Total		**$21.15**

*Note: OP mining cost, divided by total OP tonnes mined / milled.

**Note: UG mining cost divided by total UG tonnes mined / milled.

***Note: Total mining cost (OP + UG) divided by total tonnes milled.

****Note: G&A Costs are applied at $2.02/tonne milled until mining ramps down in Year 14 and Year 15.

The operating costs include mining, processing, tailings handling, water treatment, and G&A. Underground mining costs also include stope and level development.

21.2.1 Open Pit Mine Operating Costs

All Open Pit mining operating costs are shown in Canadian dollars. Open Pit mine operating costs are derived from historical data collected by MMTS and typical project comparisons in the Project area. Additional data is derived from recent mining contractor estimates or proposals for BC-based mining contracts.

The unit costs are based on the following data:

- The 2013 PEA update demonstrated a mining cost of $2.30/tonne using a detailed cost model. Additionally, the price is well supported from similar sized mining operations. For this report, a mining contractor performs 100% of the open-pit mining. Therefore, a markup of 20% is added to the previous $2.30/tonne estimate for basic mining cost of $2.76/tonne. The mining cost is varied based on a conceptual haul distance, where the $2.76/tonne average represents the average haul cycle. Based on the variance from the average haul cycle, the mining cost is factored up or down (for instance, the mining costs in Year 2 are $2.48/tonne based on a haul cycle that is approximately 10% shorter.
- All open pit mine equipment is assumed to be diesel-hydraulic.
- Blasting costs are included in the established mining unit costs.
- An additional rehandle of UG ore is included in OP mining costs during underground production years to move ore from the portal to the primary crusher using OP equipment.

21.2.2 Underground Operating Costs

Underground mine operating costs have been estimated by MMTS and are shown in Table 21-4, including rockmass conditioning, undercutting and drawbell development, and haulage of mill feed up the ramp to the portal. This cost also includes level and stope development and underground mine and maintenance management, technical services and all other mine services. See Appendix D for more details.

All underground mining operating costs are shown in Canadian dollars. The mine operating costs are derived from historical data collected by MMTS and typical project comparisons in the Project area. Life of mine operating costs is shown in the Table below:

Table 21-4 Life of Mine UG Operating Costs

Underground Area	$/t
UG Level and Stope Development	$4.64
UG Block Cave Production*	$7.09
Total:	**$11.73**

*Includes Caving, ore haulage to surface and mine services

21.2.3 Process Operating Costs

The operating cost estimate of $9/tonne milled for the Project is a factored estimate based operating costs from comparable local industrial-scale operations.

21.2.4 General and Administrative

G&A costs are the costs that do not relate directly to the mining or processing operating costs. The costs include:

- Personnel – general manager and staffing in accounting, purchasing, and environmental departments, and other G&A departments.
- G&A expenses – insurance, administrative supplies, medical services, legal services, human resources related expenses
- Travelling, accommodation and camp costs, air/bus crew transportation, and external assay/testing.

The G&A cost is estimated at $1.95/t milled. This cost is estimated assuming a fly-in-fly-out shift rotation from Prince George, and a full-service camp. Remote or head office costs, such as a Vancouver office, are not included in the G&A Estimate. G&A costs will reduce as the Project ramps down to stockpile reclaim only, and this is reflected in the estimated G&A costs for the final two years.

22 Economic Analysis

22.1 Economic Results Summary

An economic evaluation of the Project incorporating all the relevant capital, operating, working, sustaining costs, and royalties has been performed. For the 15 year mine life and 78.9Mt resources inventory, the following financial parameters have been calculated using the base case metal prices:

- 21.1% Pre-Tax internal rate of return (IRR)
- 3.7-year Pre-Tax payback on $476M capital
- CAD $324.4M Pre-Tax net present value (NPV) at 7% discounting.
- CAD $1.20/lb Cu Net Cash Cost of Production (C1) for LoM
- CAD $0.70/lb. Cu Net Cash Cost of Production (C1) for first 8 years
    - Net Cash cost is net of Au and Ag credits

Table 22-1 shows the metal prices and economic evaluation results for the multiple cases run for the Project.

Table 22-1 Metal Price Cases and Results Summary

Parameter	Unit	Base Case	Spot Price	Alternate
Copper	US$/lb	$2.90	$2.71	$3.19
Gold	US$/oz	$1,270	$1,258	$1,397
Silver	US$/oz	$19.00	$18.47	$20.90
Exchange Rate	US$/CAD$	0.77	0.75	0.77
Economic Results (Pre-Tax)				
Net Revenue	$ M	$710	$635	$1,041
NPV5%	$ M	$411	$362	$635
NPV7%	$ M	$324	$282	$519
NPV8%	$ M	$287	$247	$468
NPV10%	$ M	$220	$186	$380
IRR	%	21.1	19.6	27.8
Payback	years	3.70	3.90	3.00
Economic Results (After-Tax)				
Net Revenue	$ M	$475	$426	$692
NPV5%	$ M	$255	$222	$404
NPV7%	$ M	$191	$163	$321
NPV8%	$ M	$163	$137	$285
NPV10%	$ M	$114	$91	$222
IRR	%	16.6	15.3	22.1
Payback	years	4.00	4.20	3.30

The detailed financial model is provided in Appendix F.

It should be noted that the data in the financial analysis incorporates engineering and cost estimates at a scoping level of study which is not suitable for capital investment or production decisions. As well, Inferred Mineral Resources have been included in the production schedule and cash flow model. These are considered too geologically speculative to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. Therefore, there can be no certainty that the estimates or results contained in the PEA will be realized.

22.1.1 Financial Model Methodology

The production schedule has been incorporated into the 100% equity financial model to develop annual recovered metal production from the relationships of tonnage processed, head grades, and recoveries.

Metal revenues are calculated based on stated prices. Unit operating costs for open pit and underground mining, processing, site services, G&A, and off-site charges (smelting, transportation, and royalties) are applied to annual milled tonnages. All operating costs are then added together to determine the overall operating cost, which is deducted from the revenues to derive annual operating net cash-flow (Net Revenue).

Initial and sustaining capital costs have been incorporated on a year-by-year basis over the mine life and deducted from the net revenue to determine the net cash flow before taxes. Initial capital expenditures include all costs accumulated prior to first production of copper; sustaining capital includes underground infrastructure and equipment purchases, and environmental and closure costs.

An estimate is provided for the BC mineral tax regime and provincial and corporate income taxes as would be typical to an operating mine. A detailed tax model is not part of this study.

22.2 Results

22.2.1 Metal Production

The metal production values indicated in Table 22-2 are a summary of the results of the production schedule, which is used in the cash flow to determine projected revenues.

Table 22-2 Metal Production from Kwanika Project

	Years 1 to 5	LOM
Total Tonnes to Mill (000s)	27,001	78,855
Annual Tonnes to Mill (000s)	5,401	5,401
Average Grades		
Copper (%)	0.480	0.357
Gold (g/t)	0.547	0.381
Silver (g/t)	1.415	1.398
Total Production (after Recovery)		
Copper (000s lb)	260,260	600,635
Gold (000s oz)	356	676
Silver (000s oz)	921	2,659
Average Annual Production		
Copper (000s lb)	52,052	40,042
Gold (000s oz)	71	45
Silver (000s oz)	184	177

Table 22-3 Summary of the Base Case Economic Evaluation

Economic Evaluation Summary	Unit	Pre-Tax	Post-Tax
Initial Capital	$ M	$476	$476
Cum Net Cash Flow	$ M	$710	$475
LOM	Years	15	15
Payback	Year	3.73	4.03
NPV5	$ M	$411	$255
NPV7	$ M	$324	$191
NPV10	$ M	$220	$114
IRR	%	21.1%	16.6%

The undiscounted base case annual cash flows are illustrated in Figure 22-1.

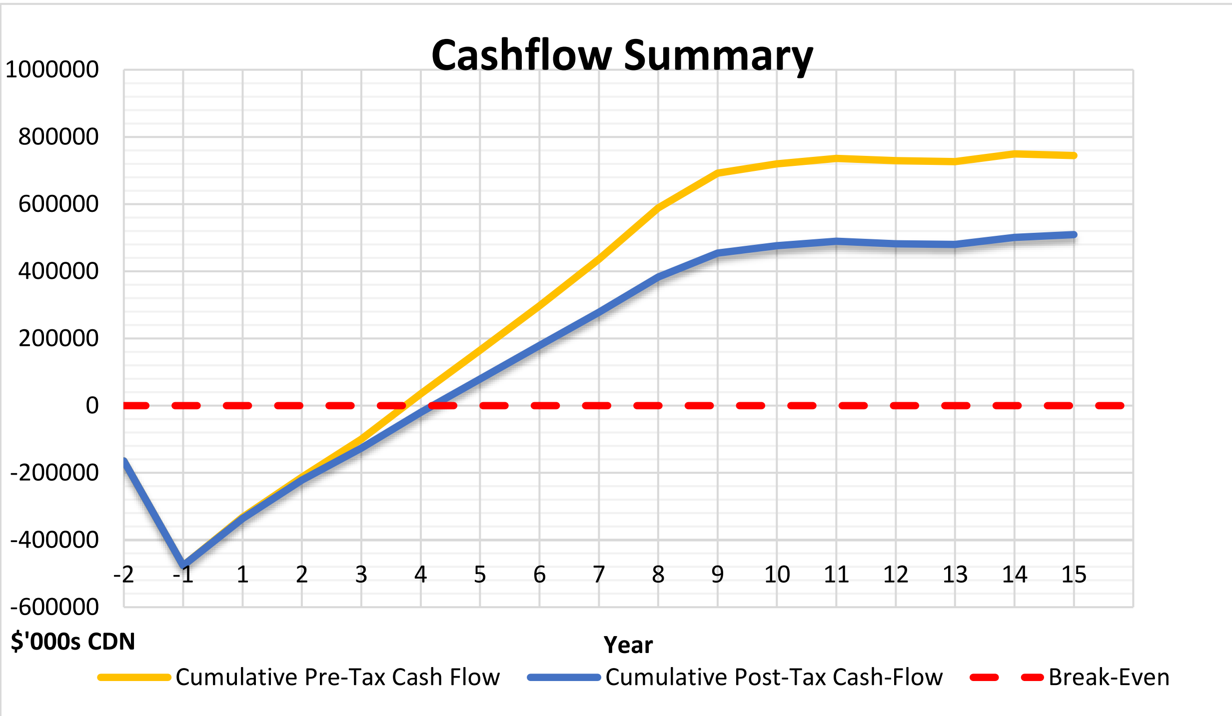


Figure 22-1 Undiscounted Cumulative Cash Flow

22.2.2 Sensitivity Analysis

Sensitivity analyses have been carried out on the following parameters:

- all metal prices
- gold only, with other metals fixed at base case values
- exchange rate
- initial capital expenditure
- on-site operating costs

The analyses are presented graphically as financial outcomes in terms of NPV and IRR. Both the Project NPV and IRR are most sensitive to Copper Price followed closely by Exchange Rate, with Capital Cost having the least impact. The NPV and IRR sensitivities can be seen in Figure 22-2 and Figure 22-3. These results are presented graphically only to show trends for future evaluation. At a scoping level of engineering and costing the absolute values are not deemed relevant for economic evaluation.

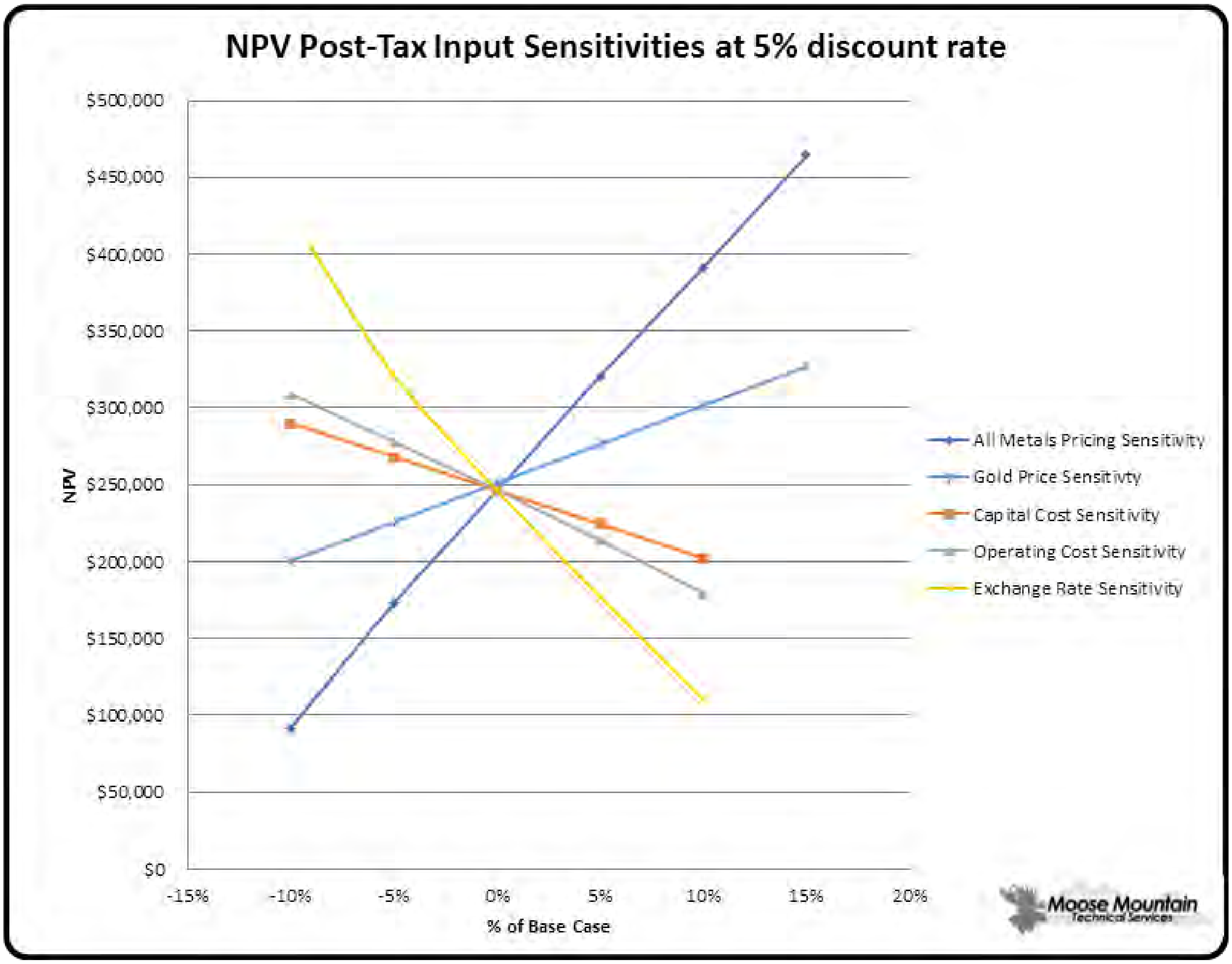


Figure 22-2 Base Case Sensitivity to Post-Tax NPV @ 5%

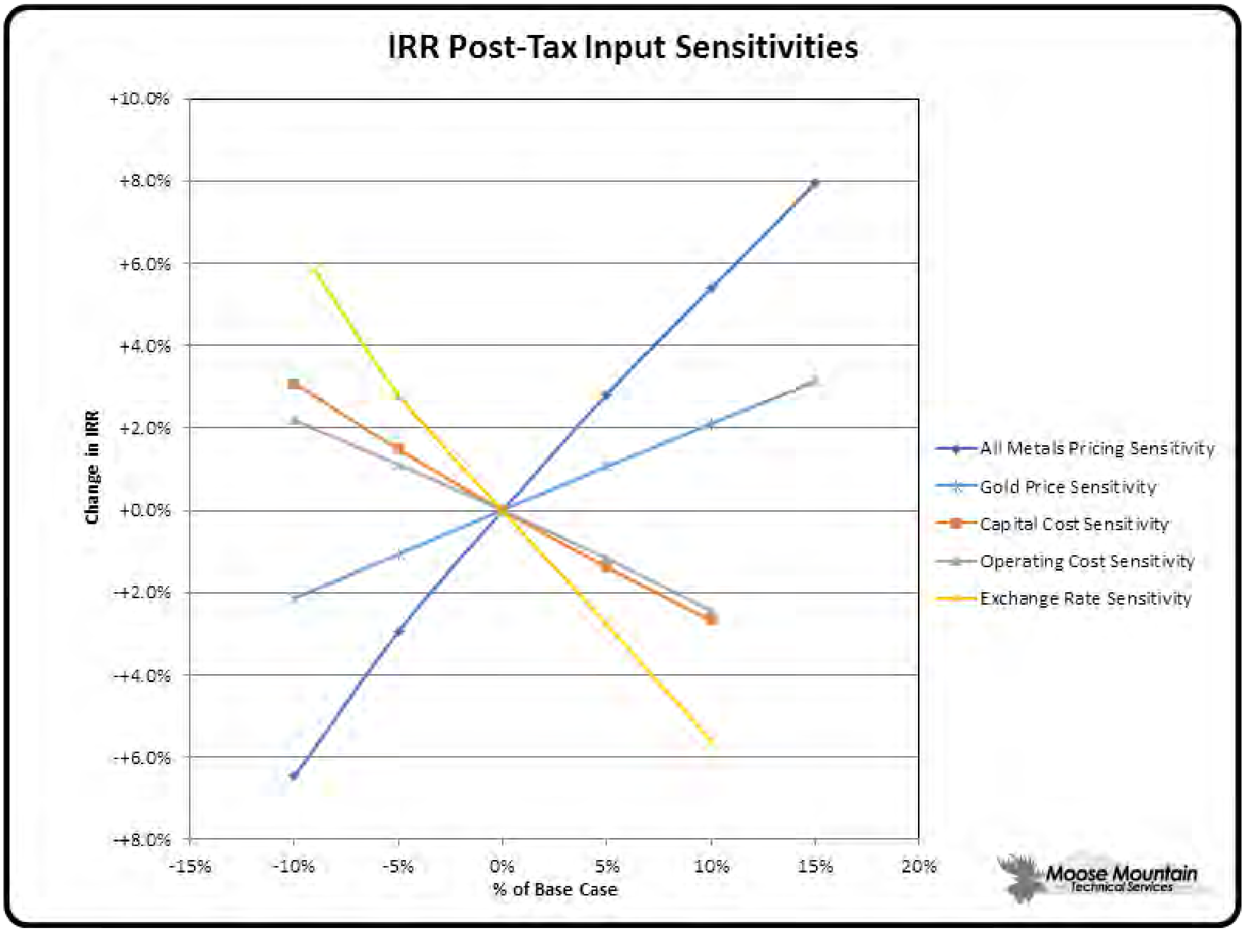


Figure 22-3 Base Case Sensitivity to Post-Tax IRR

22.2.3 Royalties

The Kwanika mining leases are un-encumbered by Royalty Agreements.

23 Adjacent Properties

23.1 Regional

The Quesnel Trough is the host to several other porphyry copper ± gold mines and significant deposits. These deposits include: the Mount Polley Mine, the former Kemess Mine and its related infrastructure located north of Kwanika, and the Mount Milligan Mine development project located approximately 85km south of Kwanika.

23.2 Local District

The adjacent Lustdust claims, previously owned by Alpha Gold Corporation, are located immediately to the north of the Kwanika property. The Lustdust property has been the subject of exploration for more than fifteen years on various precious and base metal vein and skarn occurrences and contains a small Indicated and Inferred copper-gold Mineral Resource known as the Canyon Creek Zone. The other significant prospect in the general vicinity of Kwanika is the Lorraine porphyry copper-gold property jointly controlled by Teck Corporation and Lorraine Copper Corp. which contains a modest, Indicated and Inferred Mineral Resource in two deposits. Lorraine Copper Corp. purchased the Lustdust property in June 2016.

24 Other Relevant Data and Information

MMTS has relied upon Serengeti to provide information regarding the existence and extent of any environmental, legal, regulatory or First Nations liabilities to which the Project is subject.

24.1 Project Execution

The execution plan is conceptual at this stage of the Project but provides an outline of the major steps to be undertaken to enter production. The timelines are based on typical projects of similar scope and scale and MMTS experience. Additional study and planning could advance the Project timeline. The regulatory and environmental timelines can vary greatly from project to project and can significantly alter a project timeline.

Table 24-1 Execution Schedule

Kwanika Project Execution Conceptual Schedule		2017				2018				2019				2020				2021				2022			
Project Activity	Duration	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4
Commence Baseline Studies	24mon																								
Develop Project Description	12mon																								
Submit Project Description	3mon																								
Initiate Environmental Assessment Process	24mon																								
Submit Environmental Assessment	3mon																								
Conduct Regulatory Review	12mon																								
Receive EA Approval																									
Complete PEA Study	3mon																								
Conduct Exploration and Geotech Drilling	6mon																								
Metallurgical Testwork	6mon																								
Develop PFS Study	15mon																								
Conduct Exploration and Geotech Drilling	18mon																								
Develop Feasibilty Study	18mon																								
Production Decision																									
EPCM	27mon																								
Complete Mine Permitting Process	27mon																								
Receive Construction & Mine Permits	24mon																								
Construction	24mon																								
Commissioning	3mon																								
Production Start																									
Project Activity	66 Mon																								

25 Interpretations and Conclusions

The Kwanika deposit represents a copper-gold-silver deposit that is amenable to open pit and underground block caving and conventional milling consisting of flotation concentration.

25.1 Geology and Resource Modeling

SRK updated the Mineral Resources for the Central Zone and South Zone in 2016. The current Mineral Resources estimates are summarized in Table 14-16 Mineral Resource Statement*Table 14-16 and Table 14-17 above.

SRK drew the following conclusions:

- Drilling, core handling and sampling and security protocols were appropriate and samples should be representative of the mineralization.
- Conventional assaying techniques were used, sample QA/QC protocols were adequate and checks at a secondary laboratory were consistent with the primary laboratory results.

In the Central Zone SRK modelled estimation domains as grade shells supported by lithology and alteration. In general, the high grade estimation domain correlates well to the monzonite, increased zone of veining, albitic alteration, and chalcopyrite mineralization. The final estimation domains were restricted by the Pinchi fault, the overlying unconformable sedimentary package and the overburden.

In the South Zone the grade appears to be structurally controlled and is not bound by lithology or alteration. Estimation domains represent grade shells limited by the West Fault.

Following geostatistical analysis and variography, SRK constructed resource block models with high grade restriction applied in the Central Zone on composite assay grades from high grade populations. In the South Zone a typical capping procedure was applied for resource estimation.

After validation and classification, SRK considers that the mineral resources for the Kwanika Project are appropriately reported. The Central Zone is reported at 0.13% copper equivalent cut-off grade for near surface mineralization and 0.27% copper equivalent for potential underground mining by block caving method. The South Zone is reported at 0.13% copper equivalent cut-off grade for open pit resources.

There are two major factors that may affect the quality and quantity of the current estimates, and thereby highlight opportunities for improvement:

- Uncertainty in the volume of the supergene zone and related recoveries.
- Uncertain influence of faulting and barren dykes.

Both of these risks could be greatly reduced by re-logging drillholes and addition of several oriented core drillholes.

SRK is not aware of any potential significant risks and uncertainties that could affect the reliability or confidence on the reported resource.

25.2 Metallurgy

The metallurgical test work carried out on samples from the Kwanika central deposit indicates mineralization responds well to a process consisting of conventional multi-stage flotation. A copper recovery of 91%, with gold and silver gold recovery of 75% has been estimated to a concentrate grading 24% copper.

Metallurgical test work on the central deposit has been preliminary in scope, and no metallurgical test work has been conducted on the south deposit.

A mill throughput in the order of 15,000 tonnes per day is proposed for the basis of cost estimates.

25.3 Underground Mine Plan

The AMEC Preliminary Caveability Assessment 2013 indicated the Kwanika Central zone at depth appears is amenable to the low-cost block caving mining method. The current work indicates that the revised Block Cave PEA design adds significant positive contribution to the financial results of the Project both due to higher mill feed grades and lower mining costs. Block Caving of the deep Central Zone should continue be a part of future studies. Significantly more data is required to a more advanced levels of study to assess the geotechnical characteristics for the proposed Block Cave and an optimized mine plan is needed.

25.4 Open Pit Mine Plan

The Central and South pits have been designed to their break-even economic limit and provide low cost mining for the near surface material. Below the Central zone ultimate economic pit, a high grade Block Cave also shows positive economic potential. A trade-off between the lower pit benches and extending the Block Cave stope upwards, indicates better economic results if the lower Central pit is mined instead as part of the Block Cave. The results of this trade-off will need to be confirmed in future studies.

The South Zone pit phases are marginally economic in the base case and there are near economic mineralized zones adjacent to the designed pits. With more drilling and resource modeling in the future, grade or continuity of these zones may improve, or if metal prices increase, these near economic zones may become economic and add to future resources and reserves.

The early and late term use of open pit mining in the schedule with all underground mining in between makes contractor mining economically more favourable. If future studies indicate running the open pit and underground concurrently, an owner operation should be evaluated for the open pit.

Using the open pit rock to buttress the TSF is at the same cost as hauling it to alternate sites in the vicinity of the pits. The geotechnical design of the TSF has not been done at this level of study but the use of the open pit rock as a buttress will improve the design factor of safety for the TSF and reduce risk which will be a benefit in the permitting process.

25.5 Infrastructure

25.5.1 Site Access and Power Supply

The project site has reasonable logistical advantage over many greenfield developments being considered for new development, due to its location. Local forestry roads can be upgraded and year round truck access is available for construction, and, freight and operating supplies during the life of the mine. The local roads connect into existing rail lines and ports. The area is also a mining district with established services and suppliers. This advantage of location reduces the initial capital for project development and reduces the overhead costs that relate to remote sites.

The regional power grid is only 75 km away. A connection to the existing line is relatively low capital cost into a low-cost supply of energy. This provides a distinct advantage over remote sites dependent on on-site generation. Maximizing the use of electrical equipment will not only reduce energy costs, but will also gain significant carbon credits and favour through the permitting process.

25.5.2 Site buildings

The short mine life and use of mining contractors promotes the use of more modular buildings. Some facilities should be permanent but predominantly modular construction with road access will realize lower building costs. Contractors with short term assignments, such as the early and late term open pit mining, can be expected to supply their own temporary shops and offices.

The distance to local town is too far for daily shift exchange so on-sight accommodations will be required. The site is close enough to regional facilities so that medical emergencies can be evacuated therefore an onsite first aid facility will suffice, and staff doctors and nurses will not be required.

25.5.3 Site Utilities

There is sufficient water onsite to meet industrial and personnel requirements. Water management will be more related to handling surplus water and will require water storage and diversion as well as treatment before discharge.

25.5.4 Tailings and Water Management

The recently revised BC Regulations are requiring a higher factor of safety and use of BAT and BAP in TSF design. A detailed geotechnical study has not been undertaken at this level of study but the design features of buttressing the dam with mine rock, and not storing surplus water in the TSF will provide the BAT/BAP provision which will give an advantage to the TSF design.

25.6 Regulatory, Environment, and Permitting

Based on the scoping level project defined by this PEA study, the knowledge of land use expectations, and the regulatory process in British Columbia, there is nothing that has come to light that is not a normal part of a proposed mining operation. At this time, MMTS is not aware of any constraints in this regard that may prevent the Mineral Resources at Kwanika from being exploited.

The Project lies within an area designated in 1999 for multiple land uses, including mining. Details of current land use plans require confirmation and updating.

Provincial and Federal Environmental Assessments and Certificates will be required due to the nature and scope of the Project.

Pending further work on ARD potential, the Project will need to demonstrate the ability to manage for ARD concerns during and following mining.

Significant environmental issues such as fish stream diversions; ARD potential and wildlife habitat are expected to be manageable.

Reclamation of all site disturbances is expected to be completed within industry norms.

25.7 Economic and Financial

The financial results from the base case plan used in this report are based on the parameters and assumptions as described which are preliminary in nature. The financial sensitivity to these parameters and assumptions has been indicated and as noted there are other areas in the mine plan that need evaluation that could increase or decrease the economic mining limits. To advance the Project a PFS is required. This will require significant site investigation work and testing, including exploration drilling, geotechnical drilling and testing, metallurgical sampling and testing, and base line environmental studies. With this new field data, specific test work and designs will be done in progressive levels to evaluate alternatives as required by BC Regulations as well as to determine the economic responses to the new information.

25.8 Opportunities

- This PEA study provides a scoping level basis for a viable operation with the opportunity to add more economic resources both on site and in the local area.
- There are other properties in the local area that have the potential of using the Kwanika facilities on a contract or joint venture basis.
- The expanded resource can use the facilities and infrastructure from this study. A significant mineralized resource in the Kwanika deposit surround the resources used in this mine plan. The potential exists for some of this marginal material to be brought into an economic resource base after this proposed operation has met capital payback, or if future expansions can provide a lower operating cost due to economies of scale.

26 Recommendations

It is recommended to advance the Project to a higher level of study to continue towards an eventual production decision. The recommended studies include field investigations to:

- Gather environmental background information
- Waste rock characterization for ARD and metal leaching
- Surface and groundwater observations and monitoring
- Geotechnical drilling and sampling
- Infill and step-out exploration drilling,

This information will be needed to advance the engineering and environmental studies to a stage that they can be used for the Project Description required in the permitting process. Future studies will be required to progress the Project to a PFS and Feasibility level of assessment.

26.1 Exploration Drilling

Recommendations for exploration requirements are based on the results of this PEA. The most current recommendations from Serengeti are more project specific than SRK's previous recommendations for resource modeling in its 2016 Technical Report.

With their knowledge of the Project geology, Serengeti proposes a two-phase drill program to upgrade the current Central and South zone resources enabling the Project to move to a PFS stage. This will include a 13,200m drill program on the Central and South zones. All costs related to the exploration program are included in the exploration cost estimate. This included, drilling, mobilization, camp, crew transport, logging, and assay charges estimated at $225/m all-in.

26.1.1 Phase I

26.1.1.1 Central Zone Resource Upgrade

Four holes will be completed on the Central Zone to upgrade Inferred to Indicated resources with 2,500m of drilling. Two additional holes will be drilled to explore and expand the high-grade gold zone and test for a potential gold upgrade to the resource. Deepening of one existing hole plus two wedge cuts closer to the deposit at roughly 400m each will be drilled to investigate the possibility of an expanded resource to the north of the current block model. Total drilling for these seven holes plus the wedging will be 5,100m. All drilling going forward on the Central Zone would include full geotechnical logging.

26.1.1.2 Central Zone North Deep Target Exploration

A 1000m hole is proposed to test for a deeply buried mineralized system with the potential to displace lower grade South Zone material currently scheduled for production in the final five years of the Project. This is the best opportunity for a near mine discovery of a new mineralized center.

26.1.1.3 Central Zone Geotechnical & Metallurgical Drilling

Three holes will be drilled along the proposed tailings facility where the tailings footprint is proximate to the Pinchi Fault.

Three other holes will be drilled east to west to test the underground block cave properties of the Central Zone. These three holes drilled into the Central Zone will be spread out into the three key metallurgical domains and will provide sampling for metallurgical studies (MET Domains). Total drilling will include 3,000m in six holes.

26.1.2 Phase II

26.1.2.1 South Zone Resource Upgrade

Currently all resources in the South Zone are classified as Inferred because drill spacing does not meet the indicated category. Serengeti proposes a 29-hole drill program to upgrade the resource from Inferred to Indicated. Most drillholes will be shallow but three deep holes will be included to test pit resource as well as a deep mineralized zone beneath the south pit.

26.2 Updated Resource Models

With the proposed drill programs the Resource models will need to be updated. The Central Zone model will be updated with the Phase1 drilling program, the South Zone with the Phase 2 program. The resource model cost includes evaluation of the QA/QC program.

26.3 Underground Block Cave Mining

The following recommendations for UG mining will advance the Project to PFS level:

- Undertake a geotechnical assessment of the proposed Central Zone Block Cave to confirm cavebility, fragmentation and caving rate for the mine design. Drill data for this is included in the proposed 2017 drilling program.
- Following the Geotech study and a new Resource model, optimize the footprint of a Block Cave design and an optimized schedule of the sequence of drawing point production.
- Redesign to a PFS level, the rest of the underground facilities, including access ramps, ventilation, dewatering, etc. This will be a combination of engineering design and mining contractor estimates.

26.4 Open Pit Mining

The following recommendations for OP mining will advance the Project to PFS level:

- Undertake a geotechnical investigation for pit slope design. Geotechnical investigations for the RSF will be included with the TSF work.
- Re-design the Central and South zone pit phases based on the updated resource model and other updated information

26.5 Metallurgical Testing

Significant metallurgical test work is required to provide suitable representative information for the Project. This will include drill core samples from fresh core. Metallurgical test holes have been included in the exploration program.

The existing test data is from a composited sample, future evaluation will need to be conducted to test the variability of the deposit. No metallurgical test work has yet been done on South Zone. An initial program for Central and South zones will be associated with PFS level planning. Following this, a second program can be anticipated to refine the results for a future Feasibility Study.

26.6 Infrastructure

26.6.1 General Site Studies

Additional planning and geotechnical studies for the surface facilities and structure on the site will be required to PFS level for the Project Description in the permit application. This will include:

- Location of the process plant area and other buildings and facilities. Site optimization and impact assessment will be included.
- Geotechnical investigation of the site conditions will be part of this work to properly design the mill and other major equipment foundations.
- The study would also identify any local borrow sources for fill and construction material such as gravel source for concrete and engineering fill.
- Water diversions. Design work incorporating Hydrology hydro-geology and environmental study results.

26.6.2 TSF and Water Management

Future PFSs for the TSF geotechnical and water management need to consider the following issues:

- Dam failure during construction, initial filling, operation, or closure.
- Geotechnical assessment of alternative sites for TSF, diversions and RSF as required by BC mines regulations.
- Waste rock characterization for ARD potential and appropriate methods to handle it.

- Assessment of the water balance due to local conditions and streamflow records and assessment of water requirements for the operation. This includes consideration of water treatment for discharge of surplus water.

26.7 Environmental Assessment

To advance the Project, the following actions are recommended for regulatory and permitting work. The timelines for environmental baseline studies and requisite permitting can be varied, and as such, nearly all items related to environmental studies are on the Project Execution Plan critical path.

- Required Environmental Baseline studies, many of which require two years of data, should be initiated immediately to adhere to project timing. Total project timing to Permit Approvals is estimated to be four years.
- Support for the Project from local communities and First Nations should be solicited and participation encouraged. This is most effectively done with a personal involvement from Project personnel.
- Detailed requirements for land use, Environmental Assessment processes, and detailed fisheries, wildlife, ARD and other issues should begin immediately to provide guidance to mine planning and capital cost requirements.

26.8 Cost of Recommended Work up to PFS*

To advance the Project to PFS level the following approximate costs will be incurred:

Table 26-1 Recommendations and Future Study Costs

Recommendations and Future Study Costs:		
Exploration Drilling	Phase 1	$2,000,000
Exploration Drilling	Phase 2	$1,000,000
Geology and Resource Model Updates	PH1 and PH2	$120,000
OP Mining Prefeasibility Study	Geotech	$150,000
OP Mining Prefeasibility Study	Pit Designs	$80,000
UG Mining Prefeasibility Study	Geotech	$180,000
UG Mining Prefeasibility Study	Block Cave Footprint Finder	$25,000
UG Mining Prefeasibility Study	Stope and Development Design	$80,000
Metallurgical Testwork Program		$350,000
General Site Infrastructure		$250,000
TSF and Water/Waste Management		$250,000
Environmental and Permitting*	Baseline Studies	$2,250,000
Environmental and Permitting*	Regulatory Coordination and Report	$250,000
Total		**$6,985,000**
*This includes permitting work up to Prefeasibility/Project Description. An additional $2,500,000 is estimated to complete the permitting process.		

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Certificates of Qualified Persons

Certificates are included in this section for the following Qualified Persons:

- Chad Yuhasz (P.Geo), SRK Consulting
- Marek Nowak (P.Eng), SRK Consulting
- James H. Gray (P.Eng.), Moose Mountain Technical Services
- Tracey D. Meintjes (P.Eng.), Moose Mountain Technical Services

CERTIFICATE OF QUALIFIED PERSON

To Accompany the report entitled: Independent Technical Report for the Kwanika Project, Preliminary Economic Assessment Update 2017, with an effective date of April 3, 2017.

I, Chad Yuhasz, residing at 202-535 Smithe St., Vancouver, BC V6B 0H2 do hereby certify that:

1) I was a Principal Consultant at the time of preparation of the resource estimate, with the firm of SRK Consulting (Canada) Inc. ("SRK") with an office at Suite 2200-1066 West Hastings Street, Vancouver, BC, Canada;

2) I am a graduate of the University of Regina, Saskatchewan, Canada (2003) with a BSc in Geology. I have practiced my profession continuously since 2001 and have been involved in NI 43-101 compliant estimation of copper, molybdenum, lead, zinc, nickel and gold. I have direct operational experience from large open pit porphyry copper, and small underground narrow vein gold mines in Canada, USA, Mexico, Panama, Peru, Chile, Argentina, Australia and China;

3) I am a Professional Geologist registered with the Association of Professional Engineers & Geoscientists of British Columbia (APEGBC #31779);

4) I have personally inspected the subject project August 7-9 2016;

5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;

6) As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;

7) I am the co-author of this report and responsible for Executive Summary, Sections 4, 5, 6, 7, 8, 9, 10, 11, and accept professional responsibility for those sections of this technical report;

8) I have had no prior involvement with the subject property.

9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;

10) SRK Consulting (Canada) Inc. was retained by Serengeti Resources Inc. to prepare a technical audit of the Kwanika project. In conducting our audit a gap analysis of project technical data was completed using CIM "Best practices" and Canadian Securities Administrators National Instrument 43-101 guidelines. The preceding report is based on a site visit, a review of project files and discussions with Serengeti Resources Inc. personnel;

11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Kwanika Project or securities of Serengeti Resources Inc.; and

12) That, at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Original document signed and sealed by Chad Yuhasz, P.Geo."

Vancouver, BC, Canada

April 19, 2017

Chad Yuhasz, P.Geo

Principal Resource Consultant, SRK Consulting (Canada) Inc.

CERTIFICATE OF QUALIFIED PERSON

To Accompany the report entitled: Independent Technical Report for the Kwanika Project, Preliminary Economic Assessment Update 2017, with an effective date of April 3, 2017.

I, Marek Nowak, residing in Port Coquitlam, BC do hereby certify that:

1) I am a Principal Geostatistician with the firm of SRK Consulting (Canada) Inc. ("SRK") with an office at Suite 2200-1066 West Hastings Street, Vancouver, BC, Canada;
2) I have a Master of Science degree from the University of Mining and Metallurgy, Cracow, Poland, and a Master of Science degree from the University of British Columbia, Vancouver, Canada. I have over 30 years of experience in the mining industry, as a mining engineer (in Poland), geologist and geostatistician (in Canada). I specialize in natural resource evaluation and risk assessment using a variety of geostatistical techniques. I have co-authored several independent technical reports on base and precious metals exploration and mining projects in Canada, and United States;
3) I am a Professional Engineer registered with the Association of Professional Engineers and Geoscientists of British Columbia [Member ID: 119958];
4) I have not visited the subject property.
5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101 and this technical report has been prepared in compliance with National Instrument 43-101 and Form 43-101F1;
6) As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I am the co-author of this report and responsible for Section 12, and 14, and accept professional responsibility for those sections of this technical report;
8) I have had no prior involvement with the subject property;
9) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10) SRK Consulting (Canada) Inc. was retained by Serengeti Resources Inc. to prepare a technical audit of the Kwanika project. In conducting our audit a gap analysis of project technical data was completed using CIM "Best practices" and Canadian Securities Administrators National Instrument 43-101 guidelines. The preceding report is based on a site visit, a review of project files and discussions with Serengeti Resources Inc. personnel;
11) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Kwanika project; or securities of Serengeti Resources Inc.; and
12) That, at the effective date of the technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

"Original document signed and sealed by Marek Nowak, PEng."

Vancouver, BC, Canada

April 19, 2017

Marek Nowak, PEng

Principal Geostatistician Consultant, SRK Consulting (Canada) Inc

CERTIFICATE OF QUALIFIED PERSON

JAMES H. GRAY

I, James H. Gray, of Calgary, Alberta, do hereby certify:

1. I am a Mining Engineer with Moose Mountain Technical Services with a business address at 210 1510 2nd Street Nth, Cranbrook, BC, V1C 3L2.
2. This certificate applies to the Technical Report entitled "**NI43-101 Technical Report for the Kwanika Project Preliminary Economic Assessment Update 2017**", dated April 3, 2017 (the "Technical Report").
3. I am a graduate of the University of British Columbia (Bachelor of Applied Science – Mineral Engineering, 1975).
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#11919), and the Association of Professional Engineers, and Geoscientists of Alberta (Member #M47177).
5. My relevant experience includes operation, supervision, and engineering in North America, South America, Australia, Eastern Europe, and Greenland.
6. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
7. My most recent personal inspection of the Property was on October 18, 2011.
8. I am responsible for Subsections of Sections 1, 21, 22, 25, and 26 that pertain to mining, and Sections 2, 3, 14.3, 15, 16, 18 through 24 of the Technical Report, as well as the general compilation of the report.
9. I am independent of Serengeti Resources Inc. as defined by Section 1.5 of the Instrument.
10. I have no prior involvement with the Property that is the subject of the Technical Report.
11. I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.
12. As of the date of this certificate, to the best of my knowledge, information, and belief, the Technical Report, within my sections of responsibility referred to above, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 19th day of April 2017 at Vancouver British Columbia

"Original document signed and sealed by James H. Gray, P.Eng."

James H. Gray, P.Eng.
Principal Mining Engineer
Moose Mountain Technical Services

CERTIFICATE OF QUALIFIED PERSON

TRACEY D. MEINTJES

I, Tracey Meintjes, of Vancouver, British Columbia, do hereby certify:

1. I am a Mining Engineer with Moose Mountain Technical Services with a business address at 210 1510 2nd Street Nth, Cranbrook, BC, V1C 3L2.
2. This certificate applies to the technical report entitled "**NI43-101 Technical Report for the Kwanika Project Preliminary Economic Assessment Update 2017**", dated April 3rd, 2017 (the "Technical Report").
3. I am a graduate of the Technikon Witwatersrand, (NHD Extraction Metallurgy – 1996).
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of British Columbia (#37018).
5. My relevant experience includes process engineering, operation, supervision, and mine engineering in South Africa and North America. I have been working in my profession continuously since 1996.
6. I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
7. I have not personally inspected the Property.
8. I am responsible for Subsections of Sections 1, 21, 22, 25, and 26 for issues pertaining to metallurgy and mineral processing as well as Section 13 and 17 of the Technical Report.
9. I am independent of Serengeti Resources Inc. as defined by Section 1.5 of the Instrument.
10. I have no prior involvement with the Property that is the subject of the Technical Report.
11. I have read the Instrument and the Technical Report has been prepared in compliance with the Instrument.
12. As of the date of this certificate, to the best of my knowledge, information, and belief, the Technical Report, within my sections of responsibility referred to above, contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 19th day of April 2017 at Vancouver British Columbia

"Original document signed and sealed by Tracey D. Meintjes, P.Eng."

Tracey D. Meintjes, P.Eng.
Principal Mining Engineer
Moose Mountain Technical Services

APPENDICES

Appendix A – Analytical Quality Control Data and Relative Precision Charts

This Appendix is available upon request from Serengeti Resources or can be found in the previously filed report:

Independent Technical Report for the
Kwanika Copper-Gold Project, Canada
Resource Statement
December 2016

Prepared by SRK Consulting (Canada) Inc.
2200–1066 West Hastings Street
Vancouver, BC V6E 3X2
Canada

Appendix B – AMEC Block Caveability Assessment

This Appendix is available upon request from Serengeti Resources or can be found in the previously filed report:

NI 43-101 TECHNICAL REPORT FOR THE KWANIKA PROPERTY
PRELIMINARY ECONOMIC ASSESSMENT 2013
March 4, 2013

Prepared by Moose Mountain Technical Services
1975 1st Ave. South
Cranbrook, B.C. V1C 6Y3 Canada
Tel: 250.489.1212

Appendix C – OP Mining

Open Pit Mining Methods

1 Introduction

A production schedule based on a 15,000t/d mill feed rate has been developed for the Kwanika Project. The mine production is based on both Open Pit and underground Block Cave mining methods. The Life of Mine plan starts mining two open pit phases in Central Zone while access and level development are established for the Block Cave beneath the open pit in Central. The final stage of mining is from three open pit phases in South Zone. This report focuses on the design criteria and techniques used to develop the open pit portion of the mine plan. The mine planning work for this study is based on the 3D block model (3DBM) created by SRK for the NI 43-101 published resource model, dated December 2016. The mine engineering uses MineSight® software well proven in the industry, and includes converting the SRK resource model to MineSight, pit optimization (MS-EP), detailed pit design, and optimized production scheduling (MineSight® Strategic Planner [MS-SP]).

In addition to the geological information used for the block model, other data used for mine planning includes the base economic parameters, mining cost data derived from local projects, MMTS' cost database and some of the basic project parameters from the 2013 PEA, such as throughput rate.

Currency used in this report is Canadian dollars unless otherwise stated.

2 Mining Datum

The project design work is based on NAD83 coordinates. The historical drillhole information is based on various surveys with different sets of control that have been converted to NAD83. The resource model by SRK uses a DEM created from LiDAR data collected in September 2016. A large area, low resolution topography surface was used for mine planning.

3 Production Rate

Several factors are considered when establishing an appropriate mining and processing rate. Key factors include:

- **Resource Size:** A typical mine life is 12.5 to 20 years; beyond this, time- value discounting shows an insignificant contribution to the NPV of the project, and capital investment typically is targeted at projects with a payback period of 3 to 5 years.
- **Operational Constraints:** Power, water, critical supplies, or limited infrastructure for operations support can limit production rate.
- **Construction Constraints:** Physical size and weight of equipment and shipping limits can determine the maximum size of available units.
- **Project Financial Performance:** Generally, economies of scale can be realized at higher production rates, and lead to reduced unit operating costs. These are tempered to the above mentioned physical and operational constraints and generally higher capital requirements for higher tonnage throughputs.

Higher production rates generally pay back fixed capital at a faster rate, thereby improving project NPV.

Throughput studies for the 2013 PEA indicated a throughput of 15,000t/day was appropriate for the size of the mineable resource at that time. The revised resource model in this study is similar in size, so a throughput of 15,000t/d is used in the 2017 study. If the mineable resource base is significantly increased in future studies, the NPV advantage of a higher throughput rate should be investigated.

4 Mine Planning 3D Block Model

Table 4-1 Central 3DBM Items

Item	Item By	Description
ROCK	SRK	Rock Type Code
SG	SRK	(tonnes / m^3)
OPT	SRK	Ore Percentage (%)
CU	SRK	Copper Grade (%)
AU	SRK	Gold Grade (g/t)
AG	SRK	Silver Grade (g/t)
CUEQ	SRK	Calculated Copper Equivalent (%)
CLASS	SRK	Resource Category (1=Measured; 2=Indicated; 3=Inferred)
TOPO	MMTS	Percent of Block Below Topography Surface (%)
NSR	MMTS	Net Smelter Return ($/t), based on initial 2016 values
CUEQ2	MMTS	Calculated Copper Equivalent (%), based on initial 2016 values
MO	SRK	Molybdenum Grade (%)
NSRO	MMTS	Net Smelter Revenue ($/t), based on 2013 PEA values
NET	MMTS	Block Value for Audit ($)
NSR16	MMTS	Net Smelter Revenue ($/t), based on secondary 2016 values
CUQ16	MMTS	Calculated Copper Equivalent (%), based on secondary 2016 values
NET16	MMTS	Block Value for Audit using NSR16 ($)
BCPCT	MMTS	Percent of Block Within Block Cave (%)
NSR17	MMTS	Net Smelter Revenue ($/t), based on PEA metal prices

Table 4-2 South 3DBM Items

Item	Item By	Description
ROCK	SRK	Rock Type Code
SG	SRK	(tonnes / m^3)
ORPT	SRK	Ore Percentage (%)
CU	SRK	Copper Grade (%)
AU	SRK	Gold Grade (g/t)
AG	SRK	Silver Grade (g/t)
CUEQ	SRK	Calculated Copper Equivalent (%)
CLASS	SRK	Resource Category (1=Measured; 2=Indicated; 3=Inferred)
TOPO	MMTS	Percent of Block Below Surface (%)
NSR	MMTS	Net Smelter Return ($/t), based on initial 2016 values
CUEQ2	MMTS	Calculated Copper Equivalent (%), based on initial 2016 values
MO	SRK	Molybdenum Grade (%)
NSRO	MMTS	Net Smelter Revenue ($/t), based on 2013 PEA values
NET	MMTS	Block Value for Audit ($)
NSR17	MMTS	Net Smelter Revenue ($/t), based on PEA metal prices

5 Net Smelter Revenue

Mill feed cut-offs are determined using the Net Smelter Revenue (NSR) (net of offsite charges and onsite mill recovery) NSR in $/t, which is calculated for each block in the 3DBM using the Net Smelter Price (NSP).

The NSP values for the various metals are calculated using the following formulae:

NSPCu = (NetRevenueCu – OffsiteFreightDeliveryCu) / NetCu [in Con]

NetRevenueCu = NetPaymentCuConc – Refining

NSPAu = (NetRevenueAu – OffsiteFreightDeliveryAu) / NetAu [in Con]

NetRevenueAu = NetPaymentAuConc – Refining

NSPAg = (NetRevenueAg – OffsiteFreightDeliveryAg) / NetAg [in Con]

NetRevenueAg = NetPaymentAgConc – Refining

The NSR17 is used as a cut-off item for break-even mill feed/mine rock selection and for the grade bins for cash flow optimization. The NSP is based on base case metal prices, US dollar exchange rate, and offsite transportation, smelting and refining charges. The metal prices and resultant NSPs used are shown in Table 5-1.

Table 5-1 Metal Prices and NSP for NSR17 calculation

Metal	Market Price	Unit	NSP for LGs	Unit
Copper	$2.90	$US/lb	$ 3.29	$/lb
Gold	$1270	$US/oz	$ 48.71	$/g
Silver	$19.00	$US/oz	$ 0.67	$/g

The method of calculating NSR is as follows:

NSR (CDN$/t recovered) = dolval_Cu + dolval_Au + dolval_Ag

Where:

Dolval_Cu = Cu% / 100 x (NSPCu) x 2204.62 lb/tonne x (RecCu)

Dolval_Au = Au g/tonne x (NSPAu) x (RecAu)

Dolval_Ag = Ag g/tonne x (NSPAg) x (RecAg)

NSP = Net Smelter Price (as above)

Rec = Recovery %

6 Mining Loss and Dilution

Mining dilution for the open pit assumes whole block dilution from the grade interpolation. Additional dilution has been added to account for material on ore/waste boundaries. In this study, the grade of the dilution is set at zero even though it will be close to the cut-off grade at the cut-off boundary.

Mining loss is an allowance for material lost at the ore/waste digging line in the pit and through misdirected loads, spillage, etc. during mining.

The mining reserves used for scheduling are estimated from grades in the 3DBM within the detailed pit designs with the appropriate mining loss and dilution applied as described above. The mining recoveries and dilution convert the in-situ resource material tonnages into a ROM mill feed.

In the open pits, the NSR17 cut-off used is CDN$11.30/tonne with a provision for mining loss of 5% and dilution of 2%

7 LG Phase Selection

Lerchs-Grossman (LG) pits have been used to evaluate the economic pit limit and the optimal pushbacks or phases. A series of LG pits are generated with varying metal prices. The lower LG price case pits provide higher margin (Revenues minus waste and ore mining) as early mining areas. In this study, two startup pit phases have been selected in the higher grade Central Zone to provide early revenues for both early capital payback and to cover costs while the development of the higher grade underground mining areas, are in progress. LG pit phases are selected using the following design constraints.

- large enough to accommodate the multiple unit mining operations of drilling, blasting, loading, and hauling
- have bench sizes large enough so the number of benches mined per year is reasonable (sinking rate)
- wide enough so the shovels can load the trucks efficiently

The metal prices used for the LG pit shapes are preliminary and are established early in the study. These are different than the metal prices used for the NSR17 calculation used later in the study.

Table 7-1 Metal Prices and NSP for LG calculation

Metal	Market Price	Unit	NSP for LGs	Unit
Copper	$ 2.75	$US/lb	$ 3.23	$/lb
Gold	$ 1230	$US/oz	$ 48.98	$/g
Silver	$ 17.75	$US/oz	$ 0.65	$/g

The LG pit resources are graphed and examined for inflection points in a Tonnage vs LG shell basis (see Figure 7-1) to determine any effects with respect to selection of pit phases or an ultimate economic pit.

7.1 Central LG Cases

Mill feed from the Central LG cases are graphed in Figure 7-1.

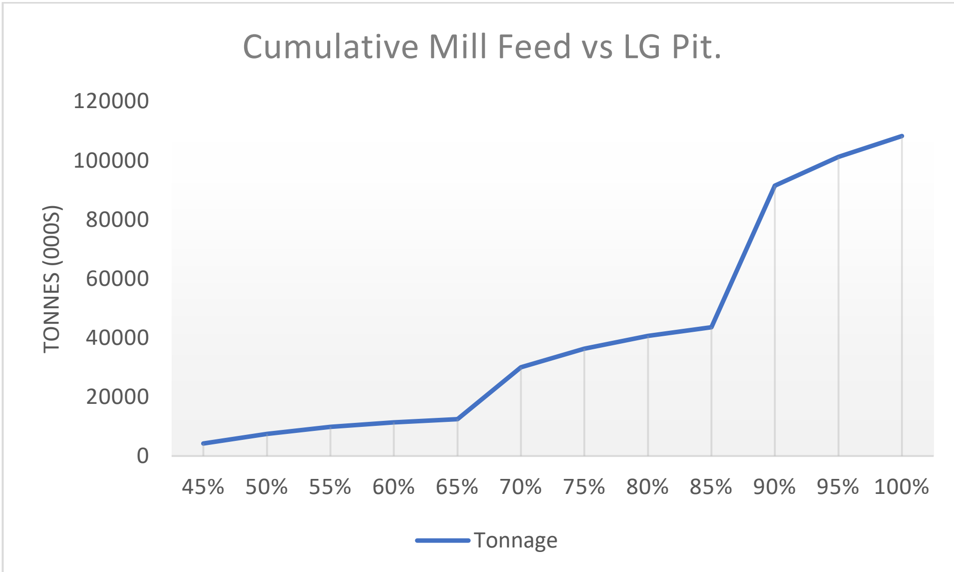


Figure 7-1 LG Pit Cases vs Tonnes Mined

The Central LG cases are developed into detailed pit designs using the 46% and 65% metal price cases (See Figure 7-2).

The 46% price case is developed into Central Pit Phase 1 and the 65% price case is developed into Central Pit Phase 2. These 2 phase have chosen to provide lower pre-stripping costs and significantly higher grade mill feed during pre-production and while the underground development is ongoing. The ultimate economic open pit is larger than these 2 phases however mineralization beneath the 65% central pit was determined to be more economically mined during the Block Cave portion of the central region. This is based on the incremental economics of the total open pit mining cost per tonne of mill feed (waste plus ore) for the ore grade material below the 65% shell vs extending the top of the block cave up into this material. The Block Cave development is justified as economic for material below the 100% pit shell. This material can be mined with no extra development cost by extending the cave upwards. Figure 7-2 illustrates where the Block Cave is extended up into the ultimate open pit material and mines at a lower cost per tonne of mill feed that the open pit for this increment.

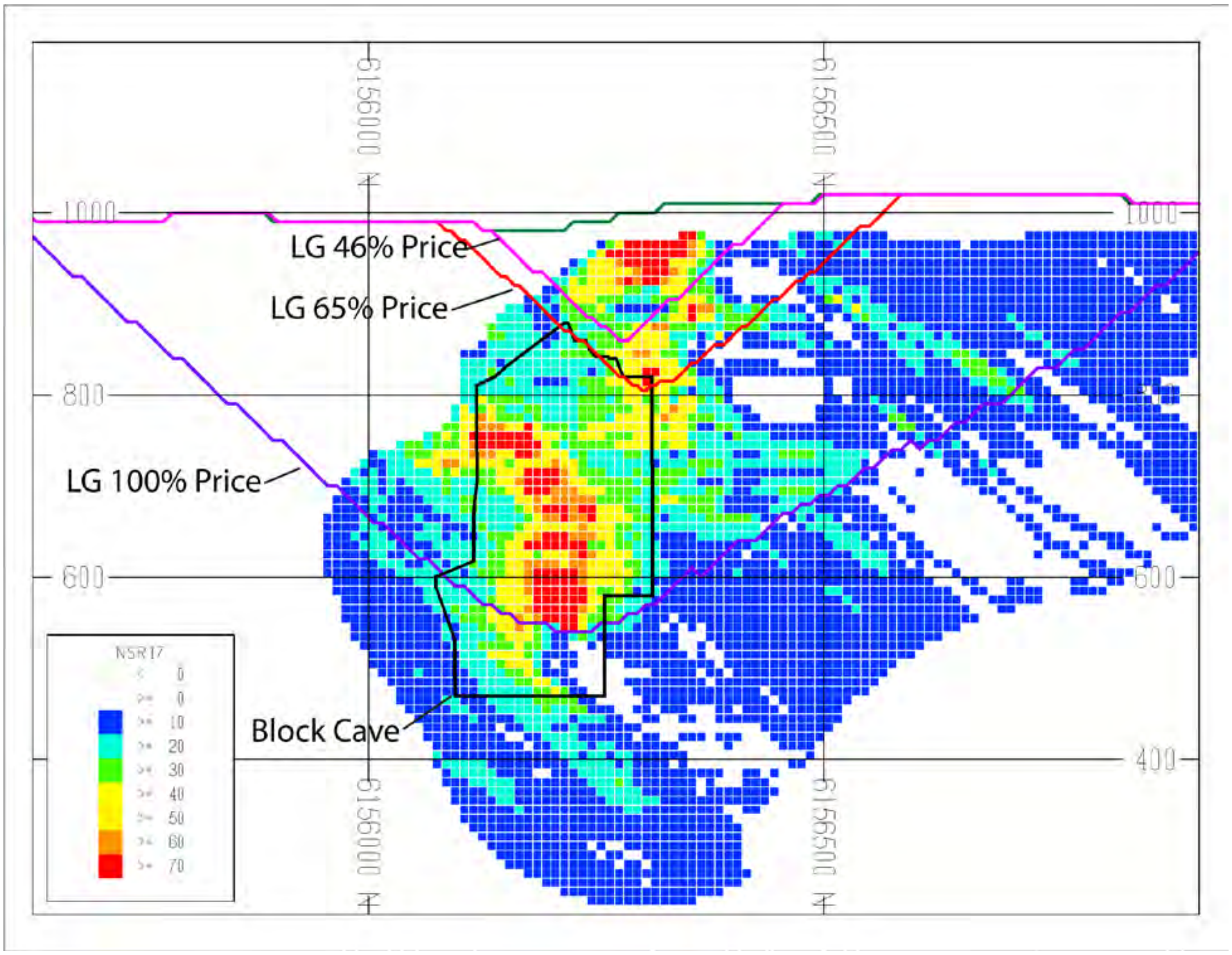


Figure 7-2 Section of Central Pit LG Phases 46%, 65% and 100%

7.2 South LG Cases

The South Pits will be mined after the Central pit phases and the Block Cave mining. Three pit phases are designed based on two LG cases.

The data is graphed (see Figure 7-3). There is no obvious inflection point so the 100% (pit06) price case is used as a guide for the ultimate pit phase.

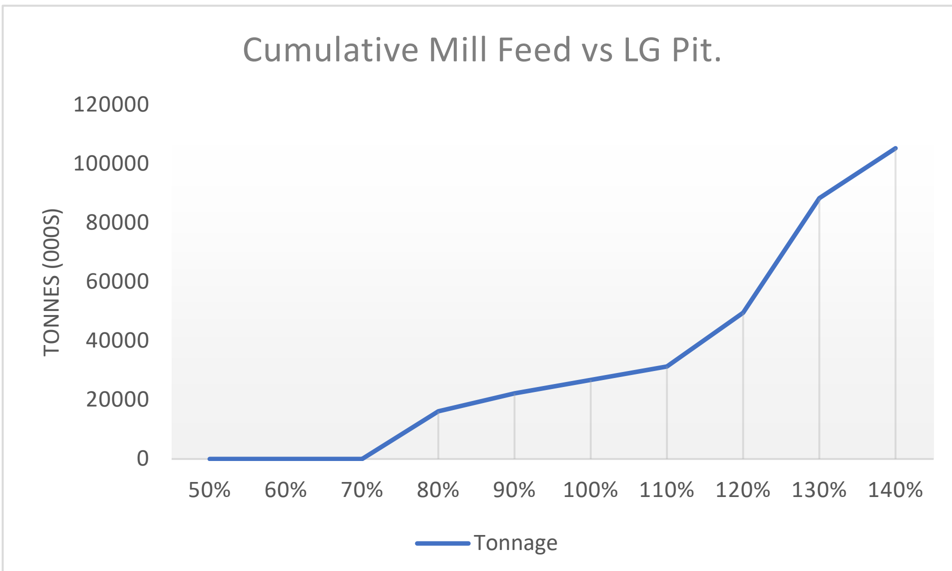


Figure 7-3 LG Pit Cases vs Tonnes Mined

The South LG cases are developed into detailed pit designs using the 80% and 100% metal price cases (See Figure 7-4). The northern 100% price case is developed into South Phase 1, the southern 80% price case is developed into South Phase 2, and the southern 100% price case is developed into South Phase 3

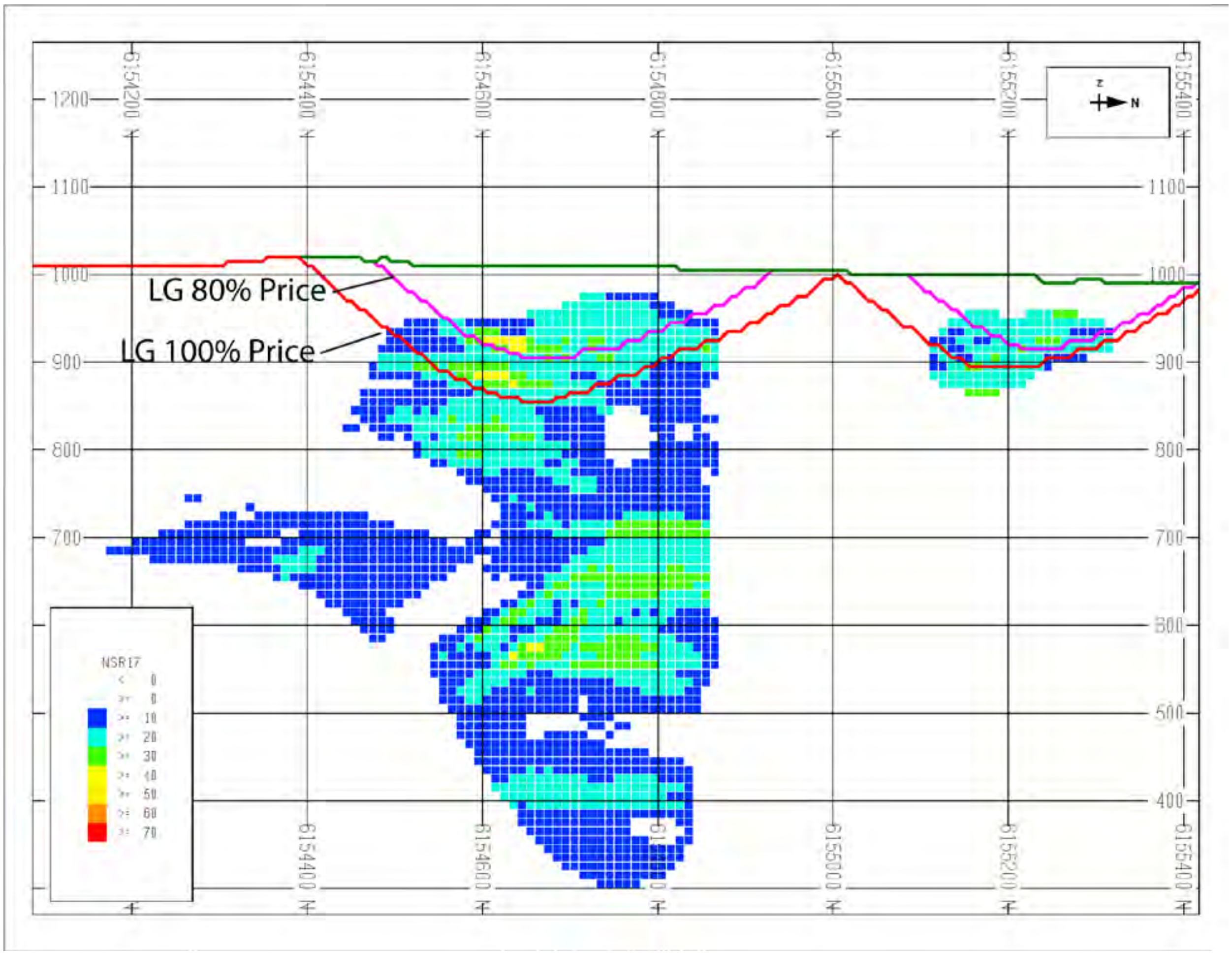


Figure 7-4 **South Pit LG Phases 80% and 100%**

8 Detailed Pit Designs

8.1 Design Standards

Design parameters for the pit phases are estimated based on similar projects in the project area.

Table 8-1 Pit Design Parameters

Parameter	Value	Unit
Bench height	10	m
Safety berm width	8	m
Safety berm vertical spacing	20	m
Minimum mining width between phases	100	m
Minimum mining width operational (i.e., at pit bottoms)	30	m
Ramp grade	8	%

8.2 Kwanika Pit Phases

The Kwanika pit design includes two phases for Central Zone (C621 and C622i) and three phases for South Zone (S621, S622i, S623i). Access to each bench is provided by ramps built into the high walls. See Figure 8-1.

The description of the detailed pit design phases uses the following naming conventions:

- The prefix "C" indicates Central Zone,
- The prefix "S" indicates South Zone,
- The first digit signifies the original LG Pit Case used,
- The middle digit signifies the revision number,
- The last digit signifies the pit phase number,

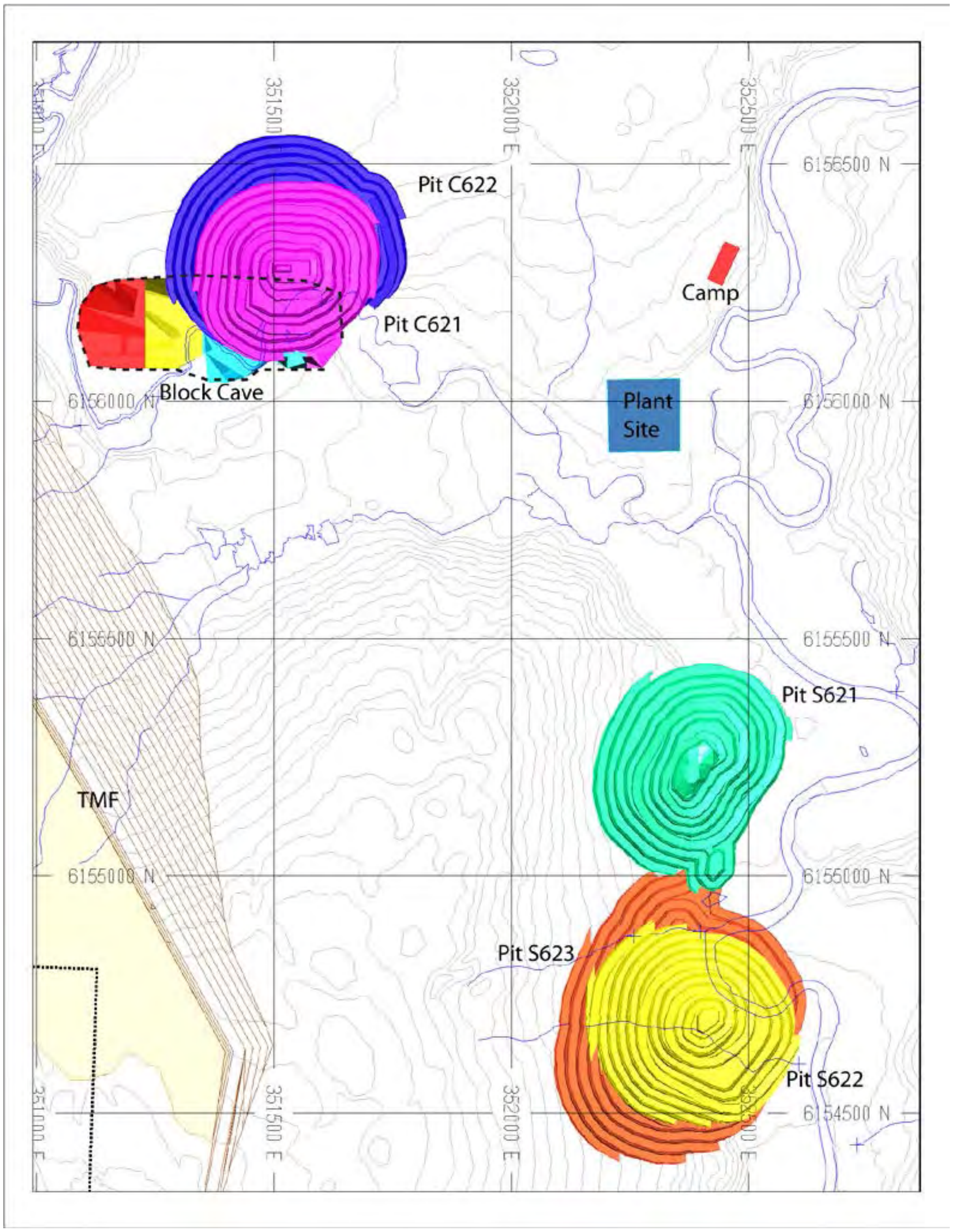


Figure 8-1 Kwanika Pit Phases

A north-south section view of the Kwanika Central Pit phases is shown in Figure 8-2.

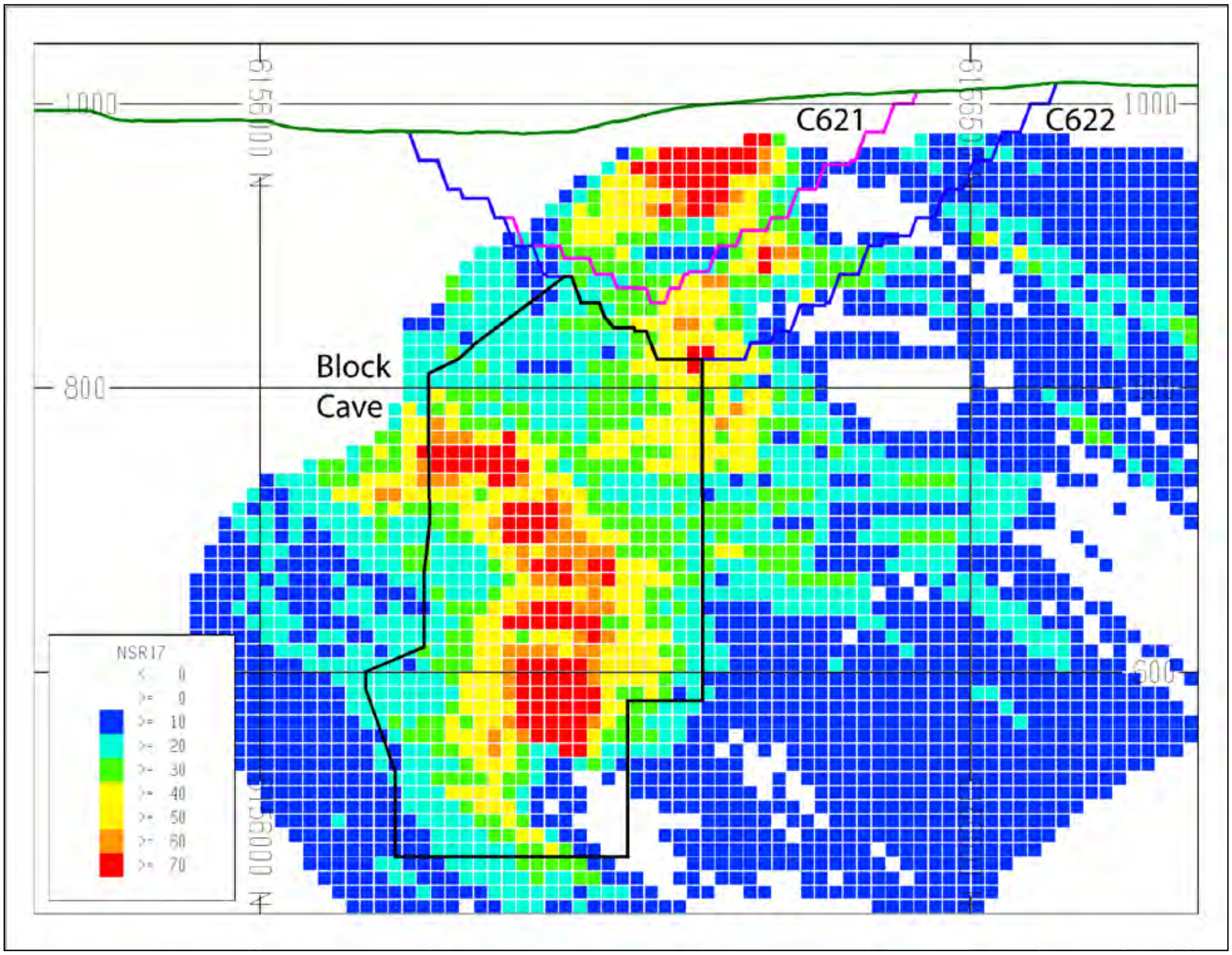


Figure 8-2 **North-South Section View of all Central Pits at East 351520 – Looking West**

A north-south section view of all the Kwanika South Pit phases is shown in Figure 8-3.

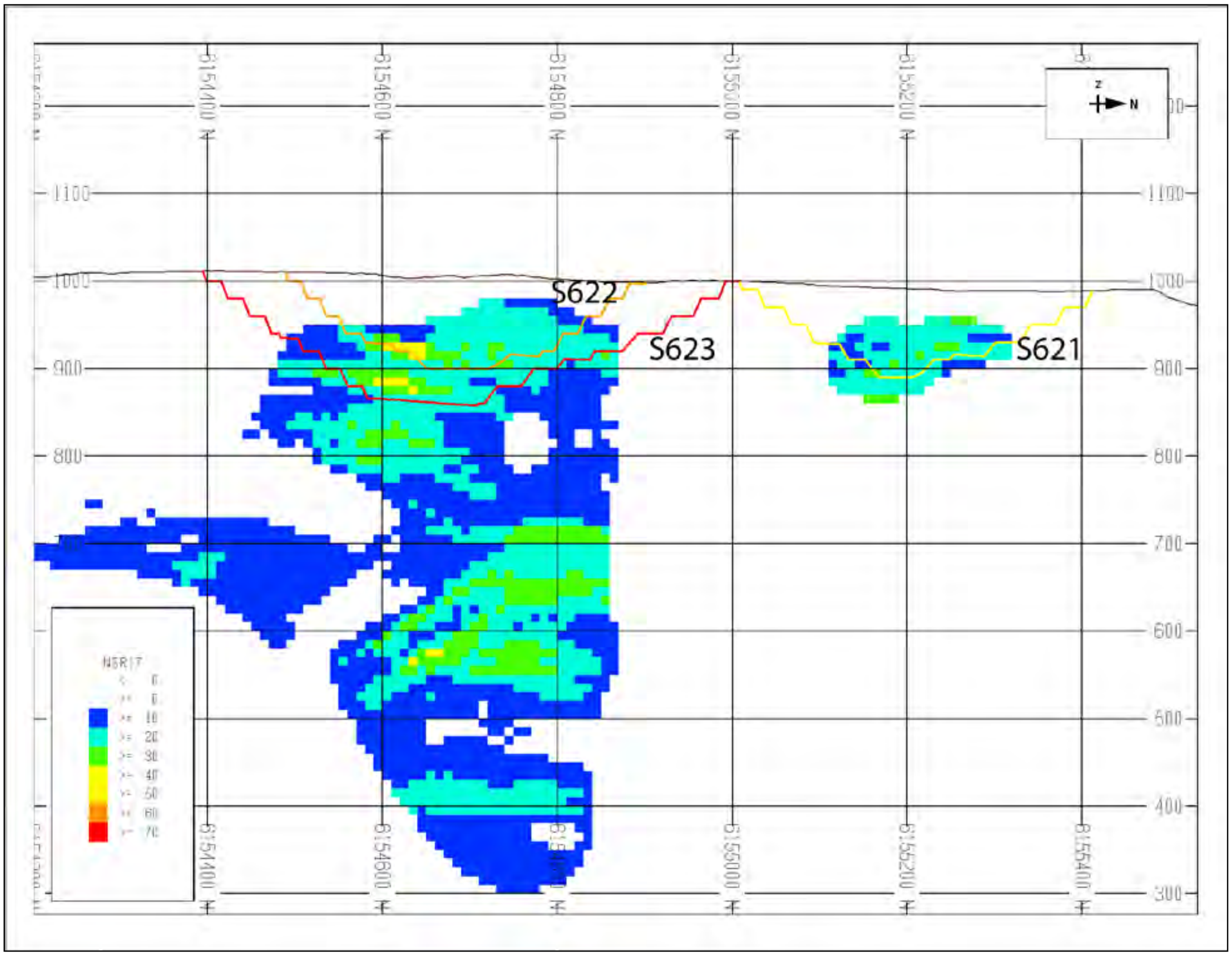


Figure 8-3 **North-South Section View of all South Pits at East 352300 – Looking West**

Further details regarding pit designs for scheduling are included in Section 10.1.

9 Rock Storage Facilities

9.1 Design Parameters

All Rock Storage Facilities (RSF)s are designed with a dump face angle at the natural angle of repose of 37°. Final RSFs are designed with overall 3:1 slope by terracing to minimize the cost of final re-sloping if required. A 20% swell factor is applied to in situ volumes to calculate the volumes that need to be placed. In the current design, the RSF is used as a buttress for the Tailings Storage Facility (TSF).

9.2 Construction Methods

Mine rock placement is done using primarily bottom-up construction methods. Bottom-up placement involves the truck placing the material in lifts 30-60m high and constructing the RSF to final limits from the bottom working upwards. Some wrap-arounds may be used as required.

9.3 RSF placement for Tailing Storage Facility Buttress

The mined waste rock will be used to buttress the TSF. This TSF must provide sufficient tailings storage volume within its footprint with a minimum 5m freeboard. Costing of the tailings dam is developed from a preliminary section (see Figure 9-1). The geometry of the TSF will be optimized in the next level of study.

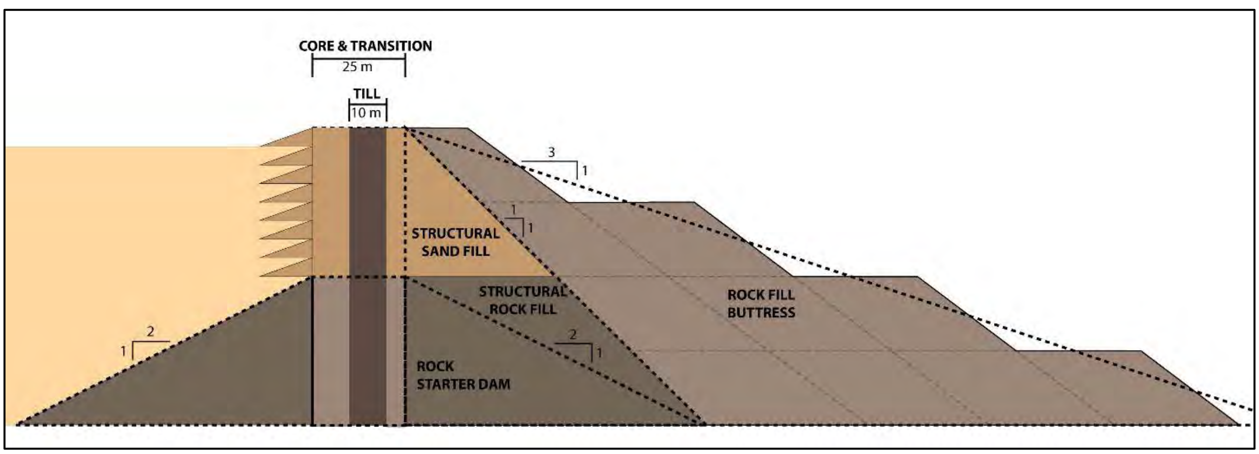


Figure 9-1 Preliminary Tailings Section

9.4 Foundation Preparation

Necessary foundation preparation requirements will be followed to allow for stable RSF and TSF construction.

9.5 ROM Stockpiles

There are stockpiles used throughout the mining schedule. Long term stockpiles are placed close to the primary crusher to maximize the grade of the plant feed and smooth strip ratio and fleet requirements. In the mine production schedule, the stockpiles reach a maximum size of 7Mt.

10 General Site Considerations

10.1 Topsoil Salvage

Topsoil salvage will be required. An estimate of the extent of topsoil salvage will be performed in future studies.

10.2 Mine Drainage

The primary purpose of the diversion ditch network is to prevent non-contact surface water from impacting certain disturbed areas. Diverting surface water will reduce water treatment requirements. The diversion ditches are located primarily around the perimeter of the pits, the TSF, the stockpile, the plant, and all mine haul roads.

Water diversion channels are also required for West Kwanika Creek which currently runs eastward through the future TSF. An additional diversion is required for Kwanika Creek southward around the South Pit mining area.

Any contact water that is impacted by disturbed areas, will be directed into a water storage pond (WSP) north of the TSF, where the water will be reclaimed to the process plant or treated before discharge.

11 Mine Operations

The mining operations are typical of open-pit operations in northern British Columbia and employ accepted bulk mining methods and equipment. There is considerable operating and technical expertise, services, and support in northwest British Columbia. Larger capacity equipment is specified for the major operating functions in the mine to generate higher productivities, which reduce unit mining costs. This also reduces the on-site labour requirements, and dilutes the fixed overhead costs for mine operations. Because of the short operating terms for the open pit operations, a mining contractor is specified. The In the end the, contractor will specify the size of equipment to achieve the lowest cost, successful bid.

A unit mining cost is estimated for the project based on previous studies and MMTS experience at typical operations, plus a contractor margin component that is on-par with current industry standards and practices. The contractor is responsible for all mining areas including direct mining and mine maintenance (see below for more details). Mine technical services, such as geology, engineering, and management will be the owner's responsibility and is captured in the general mine expense area (See below).

11.1 General Organization

The Kwanika operations are organized as illustrated in Figure 11-1. Mine operations is organized into three areas: direct mining, mine maintenance, and general mine expense (GME). Other areas of the organization are dealt with elsewhere in the report.

The direct mining area accounts for drilling, blasting, loading, hauling, and pit maintenance activities in the mine. Costs collected for this area include the mine operating labour, mine operating supplies, equipment operating hours and supplies, and distributed mine maintenance costs. The distributed mine maintenance costs include items such as maintenance labour, repair parts, and energy (fuel or electricity) which contribute to the operating cost of the equipment. The contractor is responsible for all direct mining and maintenance costs.

The GME area accounts for the supervision, safety, and training of all personnel required for the direct mining activities as well as technical support from mine engineering, environmental and geology functions. Costs collected for this area include the salaries of personnel and operating supplies for the various services provided by this function.

In this study, direct mining and mine maintenance are planned as a contractor operated fleet with the equipment ownership and labour being entirely contractor sourced. The viability and cost effectiveness of contracting can be determined in future detailed planning and commercial negotiations.

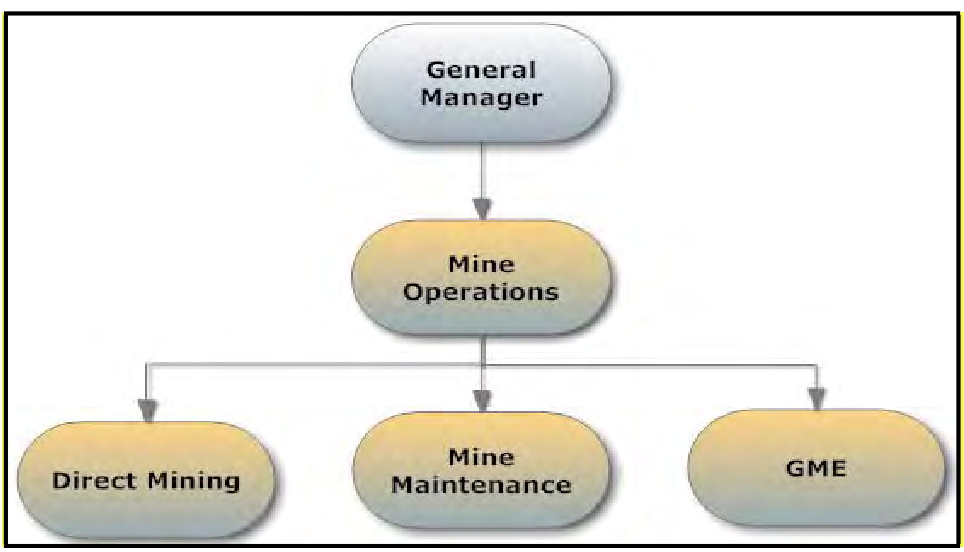


Figure 11-1 General Organization Chart

Details of the mine operation organization will be updated in future studies.

11.2 Direct Mining Area

The direct mining area accounts for the drilling, blasting, loading, hauling, and pit maintenance activities in the mine. Typical mining operations and some assumptions from the 2013 PEA are described below. The contractor chosen will determine the fleet and operating procedures.

In situ rock requires drilling and blasting to create suitable fragmentation for efficient loading and hauling of both mineralized and mine rock material. Mill feed material and mine-rock limits are defined in the blasted muck pile through blasthole assays and grade control technicians. A fleet management system assists in optimizing deployment and utilization of the loading and haulage fleet to meet the production plan. Support personnel and equipment are required to maintain the mining area, ensuring the operation runs safely and efficiently.

11.2.1 Drilling

Areas are prepared on the bench floor blast patterns in the in-situ rock. The blasthole drills are fitted with GPS navigation and drill control systems to optimize drilling. The GPS navigation enables stake-less drilling and is recommended for efficiency in aligning with hole collar locations and accuracy of set-up. The drills are also fitted with automatic samplers to provide grade control samples from the drill cutting in the mineralized material zones. These samples are used for blasthole kriging to define the mineralized material/mine rock boundaries on the bench as well as stockpile grade bins for the grade control system to the mill OCS.

Diesel hydraulic drills (150mm bit size) are used for production drilling; both in mill feed material and mine rock.

11.2.2 Blasting

Powder Factor

Typical pattern and spacing for drilling and blasting is assumed, and is based on the rock characteristics and production rates. Blasting parameters will be determined by the contractor. A detailed blasting study should be conducted in the next study phase.

11.2.3 Explosives

It is assumed an explosives supplier provides the blasting materials and technology for the mine. This may be directly to the mine owner or through the contract mining company. Because of the remote nature of the operation, an explosives plant may be built on site. The nature of the business relationship between the explosives supplier and the mining operator or mining contractor will determine who is responsible for obtaining the various manufacture, storage and transportation permits, as well as any necessary licenses for blasting operations. This will be established during commercial negotiations.

Specifications for blasting plant and explosives storage magazines and the locations of these facilities must adhere to the *Explosives Act* of Canada, regulations as published by the Explosives Regulatory Division of Natural Resources Canada, and regulations as published by the MEMPR in BC, in particular the Health, Safety and Reclamation Codes for Mines in BC.

11.2.4 Explosives Loading

Loading of the explosives is typically done with bulk explosives loading trucks provided by the explosives supplier. The trucks should be equipped with GPS guidance and be able to receive automatic loading instructions for each hole from the engineering office. This practice is common now in Canada and the explosives supplier's trucks have this capability already installed. The GPS guidance is a necessity for compatibility with stake-less drilling. The explosives product used is typically a mix of ANFO and emulsion.

11.2.5 Blasting Operations

The blasting crew comprises day-shift-only contractor employees. Based on existing mines of similar size and previous experience, the estimated crew size is two people. The blasting crew coordinates the drilling and blasting activities to ensure a minimum two weeks of broken material inventory is maintained for each shovel. Also, the blast patterns are not staked and therefore the blasting activities must also have GPS control. The blasters require handheld GPS units to identify the holes for the pattern tie-in. A detonation system is used consisting of electric cap initiation, detonating cord, surface delay connectors, non-electric single-delay caps and boosters.

The explosives contractor supplies and manufactures bulk explosives on site. The explosives contractor's employees deliver explosives to the blasthole using a bulk loading truck as is common in Canadian surface mines.

Blasthole sampling will be used to determine the mine rock/resource boundaries for identifying material designations on the pit bench for daily operations in the Ore Control System (OCS). Blasthole cuttings will be assayed across each bench giving a higher resolution of resource and mine rock than the 3DBM used in this study, which has been built from wider spaced exploration drillholes. From the blasthole assays, resource and mine rock boundaries will be defined for the production shovels and the OCS. This methodology is typical of ore control in operations in porphyry deposits.

11.2.6 Mine Load and Haul Fleet Selection

The mine load and haul fleet has been selected for the 2013 PEA is used for this study as well. Similar projects in the area have shown that the lowest cost-per-tonne fleet of shovels and haul trucks for large hard rock open pit mines that are currently being used at BC mines are the 15m^3 bucket, diesel-hydraulic shovel matched with the 136t truck. This equipment is used to form the basis of the unit mining cost estimate and production schedule. It is assumed that the open pit mining will be by contractor and a contractor's uplift has been applied to the mining costs of this fleet.

Productivities of the selected equipment are derived from truck/shovel matching studies, and include conceptual and average truck haul cycle estimates for each year in the open pit mining schedule.

11.2.7 Loading

Mill feed material and mine rock boundaries are defined in the blasted muck pile using the OCS.

Typically, one loader type is used simplify the maintenance function and reduce capital equipment and maintenance spares. This applies to other mine fleet equipment as well. Three 15.3m^3 diesel hydraulic shovels have been selected as the primary digging units based on production requirements.

There are years when there is a component of mill feed material being reclaimed from the stockpile to feed the mill. In these years, it is intended to relocate the necessary loading equipment to the stockpile area for the required length of time.

Bench widths are designed to ensure maximum operating widths to enable double-sided loading of trucks by the shovels. In some areas (such as pit bottoms or where ramps run across narrow pushback sections), single-side loading is necessary and the productivity for the shovels is reduced. For this study, it is assumed that this represents a small percentage of the total material mined.

Optimization of the shovel fleet will be conducted in future studies.

11.2.8 Hauling

Mill feed material and mine rock haulage is handled by haul trucks with a 136t payload. Haulage profiles have been estimated from pit centroids to designated dumping points considered as average haul cycles for PEA level of study.

11.2.9 Pit Maintenance

Pit maintenance services include haul road maintenance, mine dewatering, transporting operating supplies, relocating equipment, and snow removal.

A rock crusher for road grading material will be included to improve truck travel speeds, reduce mechanical fatigue to the haul trucks, increase traction (improving safety), and to enhance tire life, which is a major mine operating cost.

During winter, the snow fleet is operated by mine operations staff that will not be required for activities such as dust control and summer field programs. During severe winter storms, additional crew members are drawn from truck and shovel operations to operate the snow fleet. This ensures any fleets deemed to be priority will remain operating.

11.2.10 Mine Maintenance Area

Mine maintenance activities are directed under the mining contractor.

11.2.11 General Mine Expense Area

This section describes the mine General Mine Expense (GME) as estimated in the mine cost model.

The GME area accounts for owner management of mining operations.

A chief mine engineer directs the owner's technical support staff which coordinates the activities of the open pit and underground contractors with the process plant to meet the operation's production requirements.

The geology department includes technical personnel for local step-out and infill drill programs for onsite exploration activities and updates the long range mine mineralized material models. The geology department also provides grade control for mine operations, manages and executes the blasthole sampling and blasthole kriging of the short range blasthole models for operations planning and mill feed material grade definition as well as any underground grade control requirements.

Geotechnical considerations are covered by the mine engineering department.

The Environmental department is normally comprised of a department head and environmental professionals including biologists, environmental engineers, and technicians. Major responsibilities include regulatory administration, environmental permitting and monitoring, and operational waste management and reclamation.

11.3 Mine De-watering activities

No hydrology work has been performed to this point. At this stage of planning, an allowance for pit dewatering activities, responsibility of the mining contractor, will include the following:

- sloped pit floors as required
- in-pit sumps

- water collection system(s)

Pit water will be collected and treated prior to discharge to the environment.

12 Production Schedule

Scheduling results are presented by period as well as cumulatively and include:

- tonnes and grade mined by period broken down by material type, bench, and mining phase
- tonnes transported by period to different destinations (mill, stockpiles, and RSF)

The mine schedule considers Time 0 the time that the mill starts; the full capacity production of mill feed is expected in Year 1.

12.1 Schedule Results

The Kwanika mine plan includes:

- 2 years of pre-production
  - UG Ramp development starting in Y-2
  - Pit pre-stripping starting in Y-1.
- OP production from Central Zone pit phases C621 and C622i from Y1-Y2,
- UG production from Y1-Y9
- OP production from South Zone Pit phases S621, S622i & S623i from Y9-14
- a final year of production from stockpile Y15

The summarized production schedule results are shown in Table 12-2. Tonnes and grades are Run-of-Mine (ROM).

12.2 Kwanika Central Phases C621, C622i

Mining of Phase C621 and C622i begins during pre-production. Any ore encountered during pre-production is stockpiled. The design intention of the pre-production phase is to expose mineralized material for the mill start-up with a minimum waste strip and then continue into higher margin mill feed during the payback period. The continuation of open pit mining into a third Central pit phase has been curtailed in this plan, in favor of starting up the higher-grade Block Cave as early as possible.

Central Pit Phase 1: C621

Mining of Phase C621 begins in Year -1 at an elevation 1000m and is mined down to 860m by the end of Year 1 (see Figure 12-1)

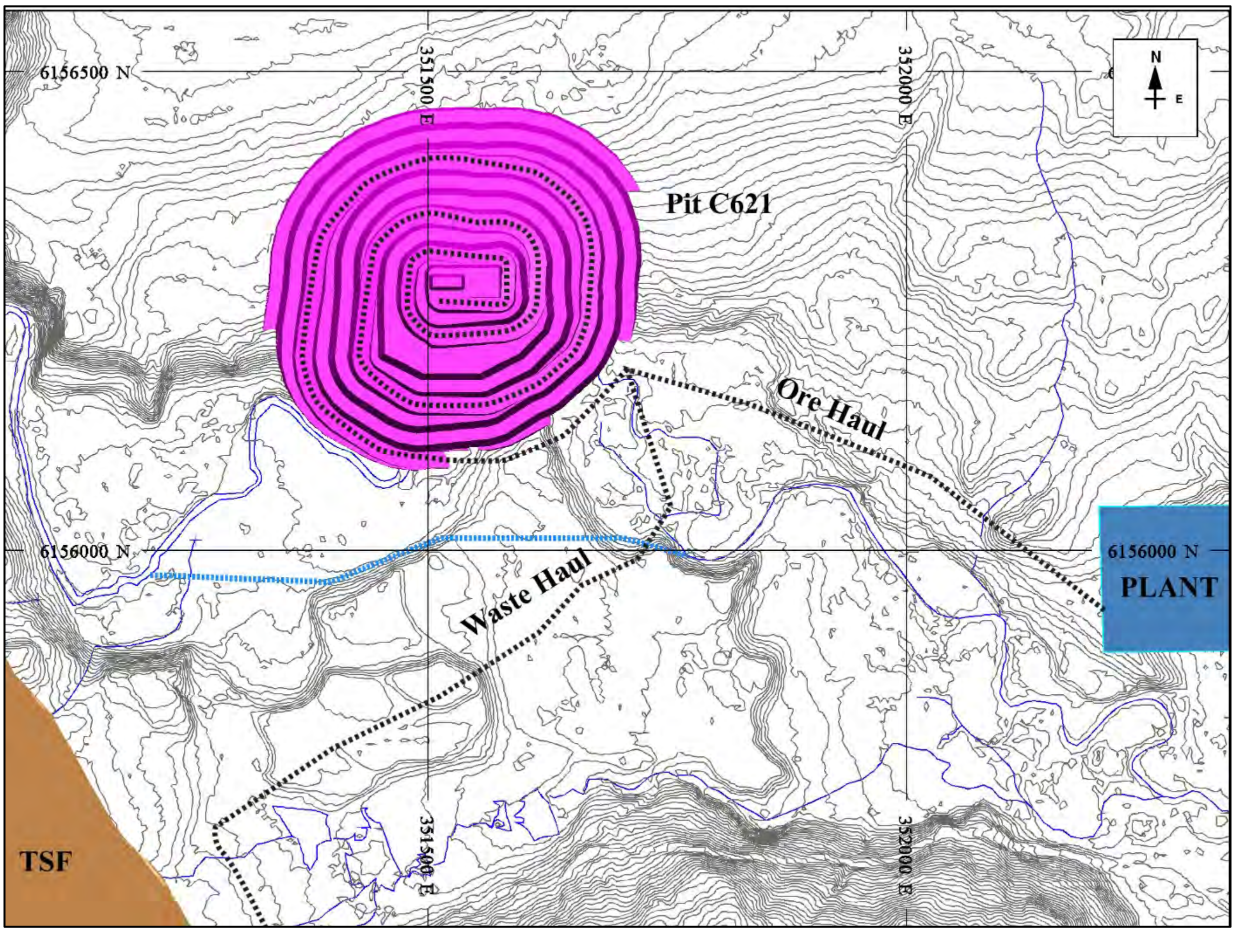


Figure 12-1 Central Pit Phase 1 (C621)

Central Pit Phase 2: C622

Mining of Phase C622 begins in Year -1 at an elevation 1014m and is mined down to 820m by the end of Year 2 (see Figure 12-2).

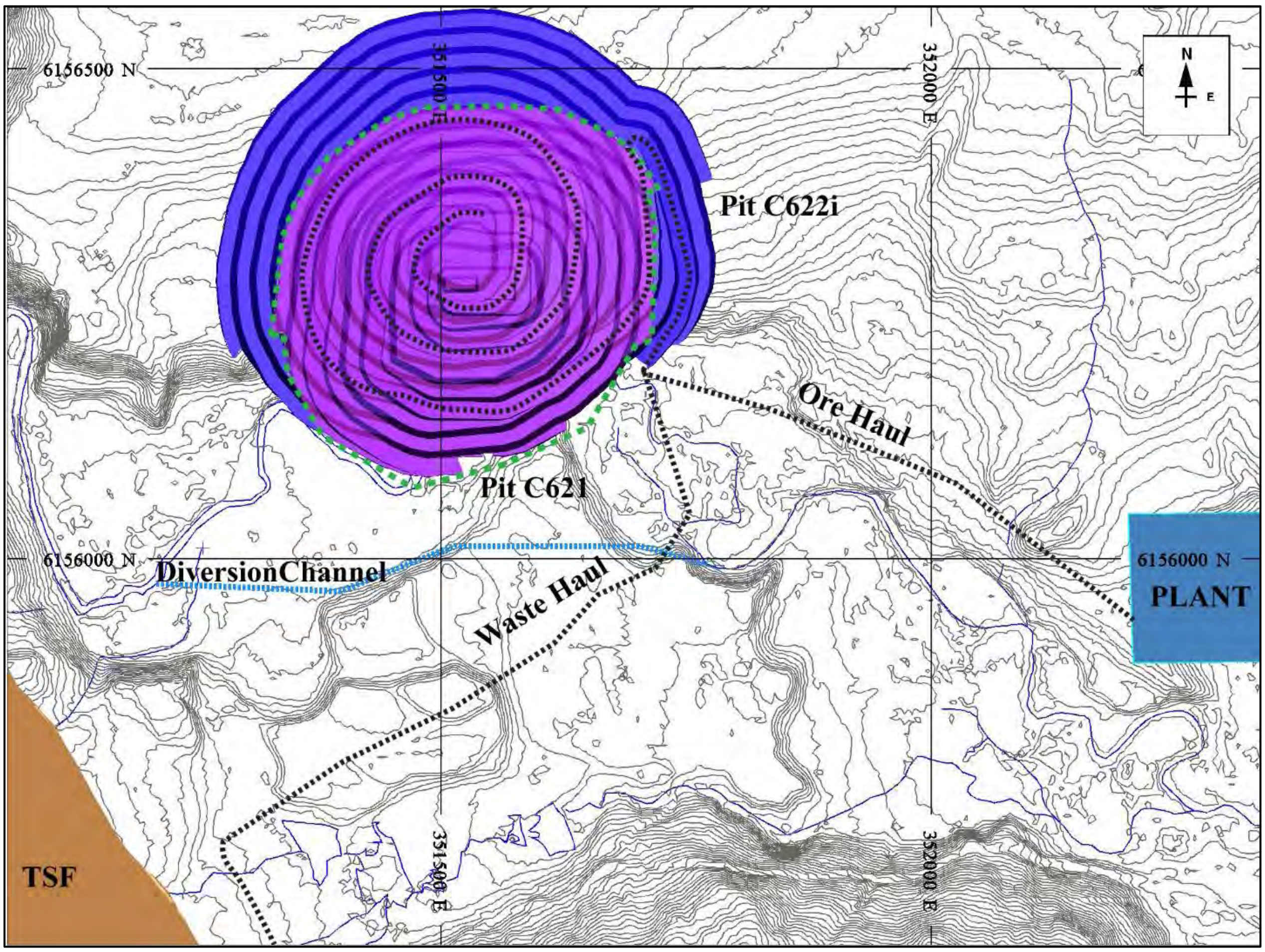


Figure 12-2 Central Phase 2 (C622)

Kwanika South Phases S621, S622i, & S623i

The Kwanika South Pit is generally lower grade material and so the start of the pre-strip for the south pit is delayed until the last year of UG production in Y9. A stockpiling strategy is still used to continue deferring low-grade material and advancing high-grade material. South Pit Phase 1 is a small northern pit in the south zone resource. It is scheduled to be mined first to allow for scheduling flexibility. Mining South Phase 1 first allows the opportunity to backfill waste from the later phases. In this study, backfilling of South Phase 1 is not included in the plan or costs.

South Pit Phase 1: S621

Pre-stripping of Phase S621 begins in Year 9 at an elevation 1014m and is mined down to 840m by the end of Year 11 (see Figure 12-3). S621 is pre-stripped early and the waste rock is used to build and buttress the TSF.

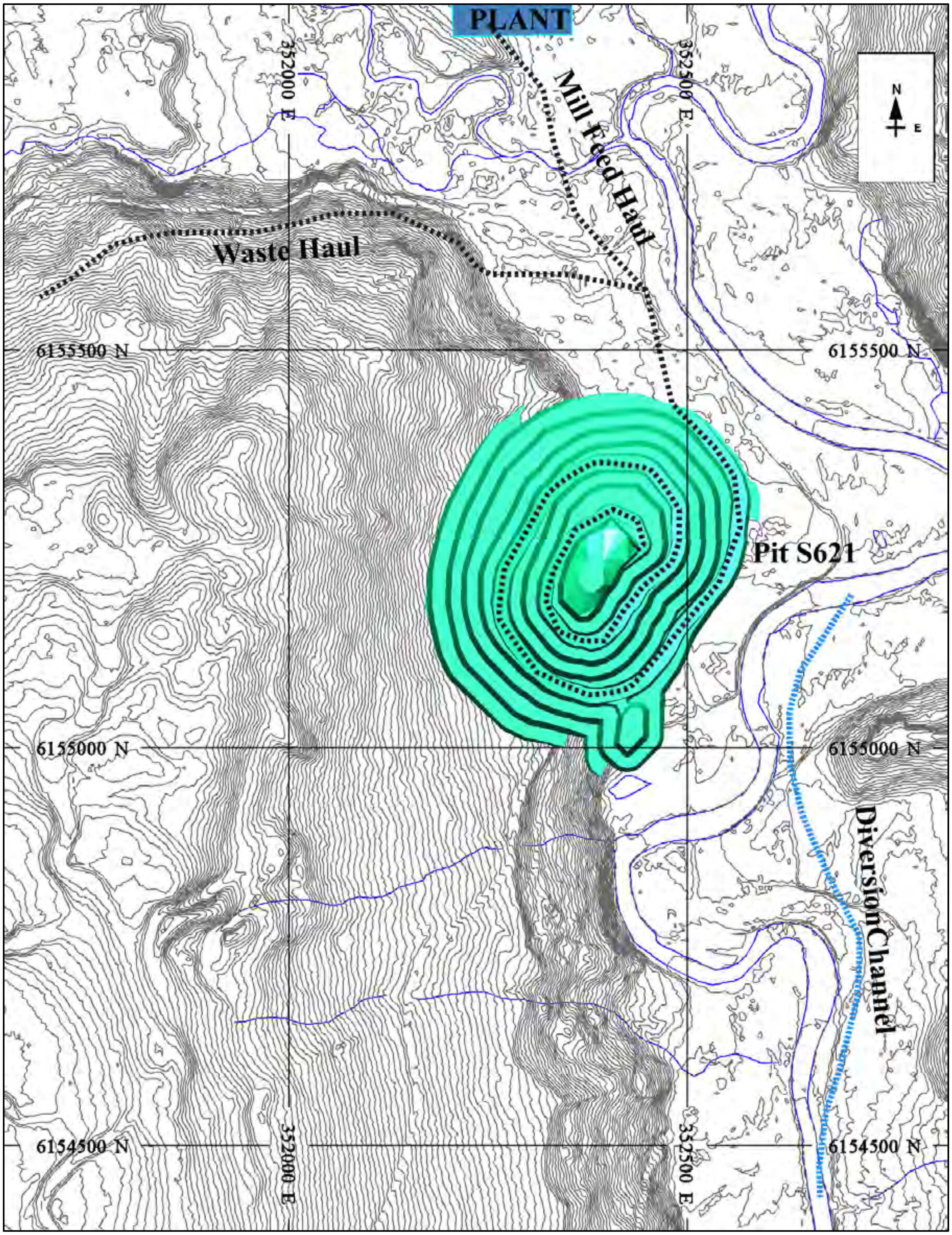


Figure 12-3 South Phase 1 (S621)

South Pit Phase 2: S622

Pre-stripping of Phase S622i begins in Year 10 at an elevation 1030m and is mined down to 830m by the end of Year 13 (see Figure 12-4).

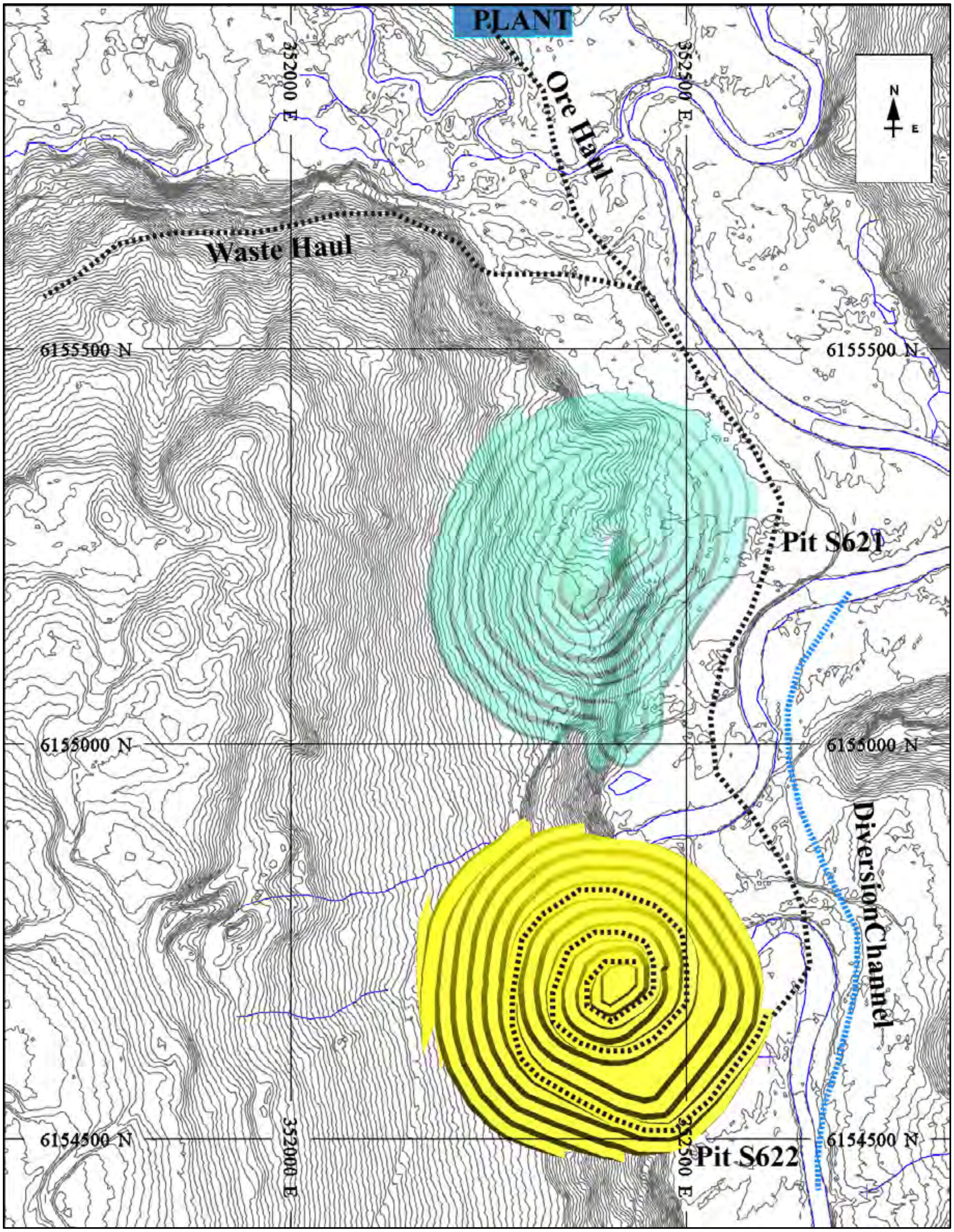


Figure 12-4 South Phase 2 (S622)

South Pit Phase 3: S623i

Pre-stripping of Phase S623 begins in Year 12 at an elevation 1040m and is mined down to 800m by the end of Year 14 (see Figure 12-5).

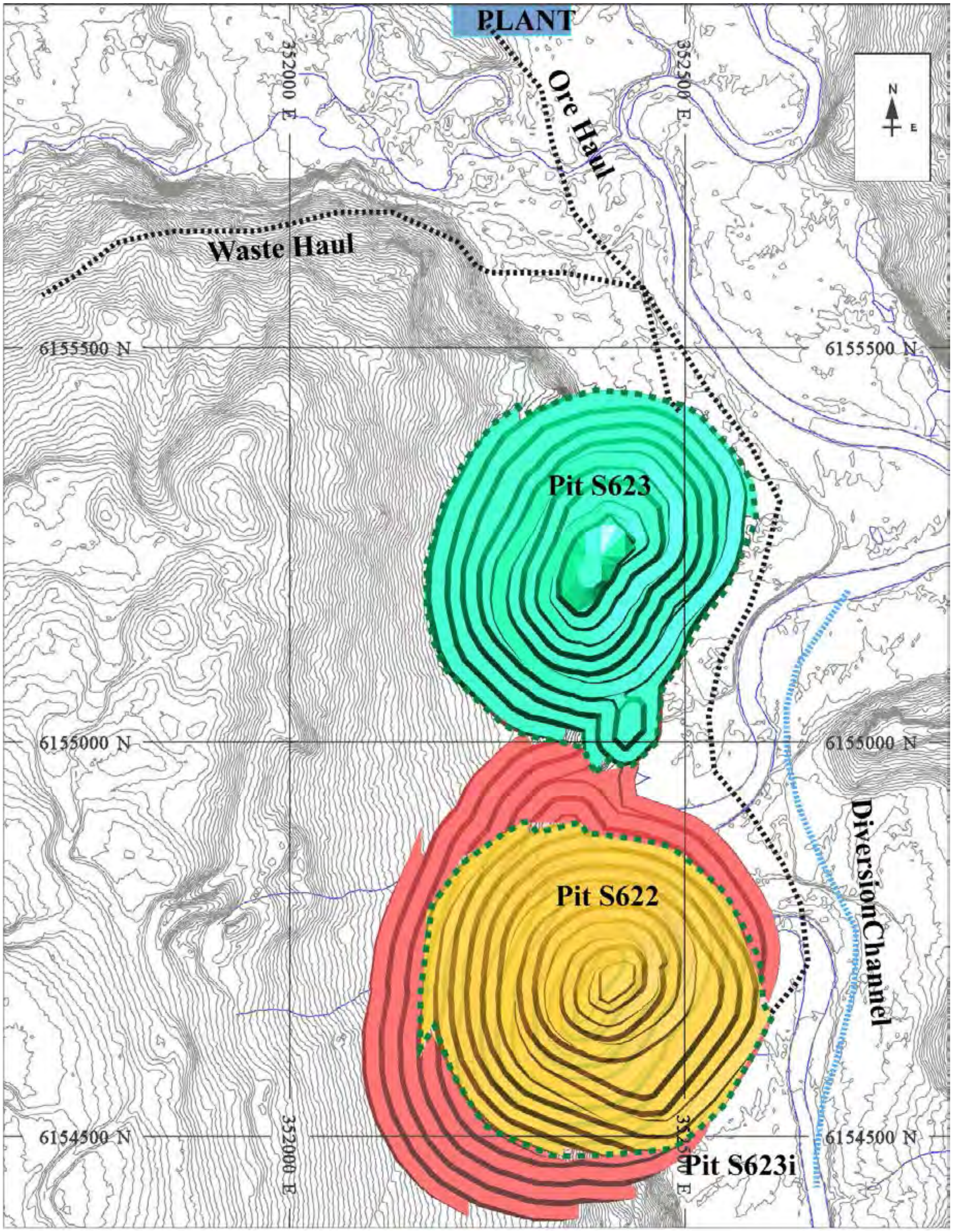


Figure 12-5 South Phase 3 (S623)

The MS-SP schedule utilizes these main criteria in each period to maximize the NPV.

- Mining precedence (Phase 2 after Phase 1)
- the haul cycle time (including 0.5 min dump and maneuver), and resultant variable unit cost
- shovel productivity (including 2.75 min load & exchange time per hauler, 70% overall efficiency)
- estimated operating and capital costs, process recoveries, and metal prices
- 360 mine operating days scheduled per year and 24h/day
- annual mill feed of 5,400kt/a is targeted based on an average throughput of 15,000t/d

Cut-off Grade Optimization

Typically, the mill feed grade can be increased by sending low and mid-grade classes to stockpiles in early periods of the production schedule. The mill feed grade is maximized and this effectively increases the revenue per tonne milled early in the schedule. Further optimization of stockpile usage will be performed in future studies.

To optimize the project NPV, grade bins have been specified based on NSR17 block values. The MS-SP optimizer develops a COG strategy to increase the project NPV by stockpiling lower grade material for processing later in the LOM schedule, increasing mill head grades and revenues early in the production schedule. The material types specified in Table 12-1 are COGs used for selectivity within the MS-SP optimized scheduler. Mining operations will not use this many grade bins in actual operations.

Table 12-1 Material Types Defined for MS-SP

	NSR Grade Bins
Low Grade	11.30<=NSR17<15
Mid-Low Grade	15<=NSR17<20
Mid Grade	20<=NSR17<30
High Grade	30<=NSR17

Table 12-2 Life of Mine Production Summary

Open Pit Production			MSSP Period		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
			YEAR	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
		Totals	Units																	
Total UG + OP Mill Feed		78,855	kTonnes		-	5,401	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,401	5,401	5,401	5,401	5,401	5,401	3,248
	Cu	0.381	%		-	0.558	0.529	0.434	0.441	0.441	0.441	0.440	0.441	0.408	0.283	0.307	0.268	0.259	0.209	0.173
	Au	0.357	g/tonne		-	0.599	0.577	0.510	0.526	0.525	0.526	0.522	0.526	0.376	0.130	0.059	0.080	0.097	0.106	0.095
	Ag	1.398	g/tonne		-	1.536	1.502	1.330	1.354	1.353	1.354	1.349	1.354	1.171	1.437	1.732	1.561	1.615	1.233	0.893
Mining Schedule by Phase																				
C621 Waste					9,005	3,287	-	-	-	-	-	-	-	-	-	-	-	-	-	-
C621 Direct Mill Feed		3,506	kTonnes		-	3,506	-	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu	0.54	%			0.545														
	Au	0.57	g/tonne			0.565														
	Ag	1.48	g/tonne			1.476														
C622i Waste					4,235	5,471	1,441	-	-	-	-	-	-	-	-	-	-	-	-	-
C622i Direct Mill Feed		1,507	kTonnes		-	156	1,350	-	-	-	-	-	-	-	-	-	-	-	-	-
	Cu	0.45	%			0.274	0.466													
	Au	0.53	g/tonne			0.495	0.536													
	Ag	1.42	g/tonne			1.418	1.425													
UG Development Mill Feed		90	ktonnes		-	37	-	31	-	22	-	-	-	-	-	-	-	-	-	-
	Cu	0.456	%			0.515		0.441		0.377										
	Au	0.469	g/tonne			0.565		0.526		0.226										
	Ag	1.308	g/tonne			1.467		1.354		0.978										
UG Level Development Mill Feed		730	ktonnes		-	135	140	125	105	33	114	78	-	-	-	-	-	-	-	-
	Cu	0.470	%			0.597	0.485	0.435	0.441	0.441	0.440	0.377								
	Au	0.512	g/tonne			0.622	0.545	0.512	0.526	0.526	0.520	0.226								
	Ag	1.368	g/tonne			1.606	1.416	1.333	1.354	1.354	1.346	0.978								
UG 470 Cave Production Mill Feed		41,255	ktonnes			1,313	3,910	5,244	5,295	5,345	5,286	5,322	5,400	4,141	-	-	-	-	-	-
	Cu	0.452	%			0.597	0.553	0.434	0.441	0.441	0.441	0.441	0.441	0.405						
	Au	0.516	g/tonne			0.622	0.592	0.510	0.526	0.526	0.526	0.526	0.526	0.354						
	Ag	1.354	g/tonne			1.606	1.532	1.329	1.354	1.354	1.354	1.354	1.354	1.139						
S621 Waste			kTonnes	-	-	-	-	13	-	-	-	-	-	7,592	6,452	181	-	-	-	-
S621 Direct Mill Feed		5,777	kTonnes	-	-	-	-	-	-	-	-	-	-	-	2,949	2,828	-	-	-	-
	Cu	0.32	%												0.301	0.343				
	Au	0.05	g/tonne												0.053	0.044				
	Ag	2.00	g/tonne												2.009	1.989				
S622 Waste					-	-	-	-	-	-	-	-	-	-	2,484	7,093	1,124	92	-	-
S622 Direct Mill Feed		9,544	kTonnes		-	-	-	-	-	-	-	-	-	-	-	2,573	5,393	1,579	-	-
	Cu	0.26	%													0.267	0.268	0.251		
	Au	0.09	g/tonne													0.075	0.080	0.120		
	Ag	1.55	g/tonne													1.450	1.561	1.704		
S623 Waste					-	-	-	-	-	-	-	-	-	-	-	-	8,046	5,363	148	-
S622 Direct Mill Feed		5,272	kTonnes		-	-	-	-	-	-	-	-	-	-	-	-	8	3,795	1,469	-
	Cu	0.27	%														0.230	0.263	0.302	
	Au	0.10	g/tonne														0.044	0.088	0.132	
	Ag	1.74	g/tonne														1.219	1.584	2.161	
Stk1-3	**Stockpile Mined**	11,174	kTonnes		382	3,131	3,434	-	-	-	-	-	-	-	393	1,353	961	1,343	176	-
	Stockpile Reclaimed	11,174	kTonnes		-	253	-	-	-	-	-	-	-	1,260	2,452	-	-	28	3,932	3,248
	NSR	$18.46	$/Tonne			$79.85								$12.26	$24.85			$15.74	$15.26	$15.16
	Cu	0.232	%			0.690								0.418	0.261			0.179	0.175	0.173
	Au	0.184	g/tonne			1.006								0.446	0.224			0.103	0.096	0.095
	Ag	0.928	g/tonne			2.048								1.278	0.748			0.851	0.886	0.893

Figure 12-6 shows the LOM mill feed production schedule:

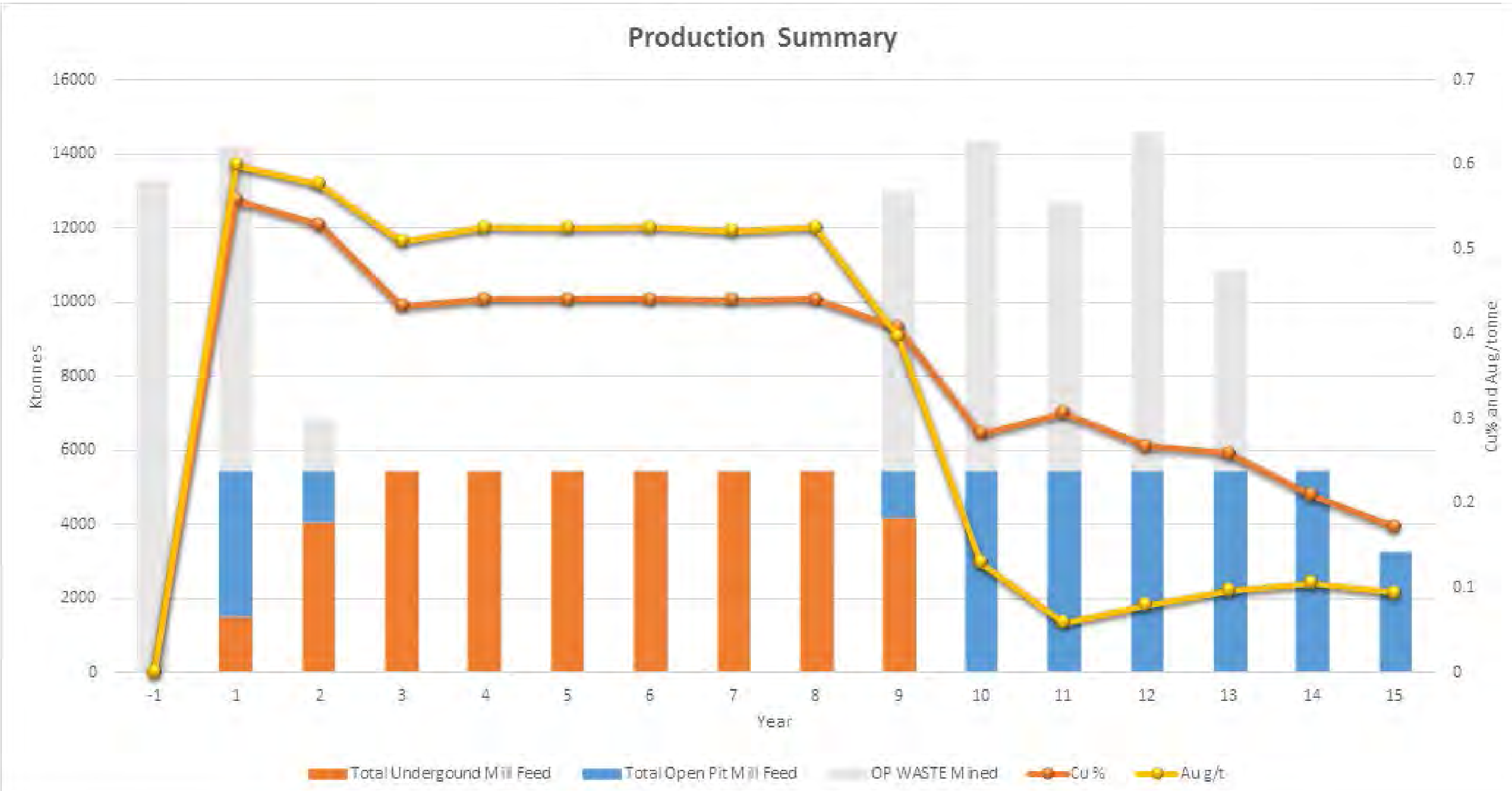


Figure 12-6 **ROM Mill Feed Sources and Mill Head Grades for Feed Cu, Au**

12.2.1 Open Pit Mine Operating Costs

All Open Pit mining operating costs are shown in Canadian dollars. Open Pit mine operating costs are derived from historical data collected by MMTS and typical project comparisons in the Project area. Additional data is derived from recent mining contractor estimates or proposals for BC-based mining contracts.

The unit costs are based on the following data:

- The 2013 PEA update demonstrated a mining cost of $2.30/tonne using a detailed cost model. Additionally, the price is well supported from similar sized mining operations. For this report, a mining contractor performs 100% of the open-pit mining. Therefore, a markup of 20% is added to the previous $2.30/tonne estimate for basic mining cost of $2.76/tonne. The mining cost is varied based on a conceptual haul distance, where the $2.76/tonne average represents the average haul cycle. Based on the variance from the average haul cycle, the mining cost is factored up or down (for instance, the mining costs in Year 2 are $2.48/tonne based on a haul cycle that is approximately 10% shorter.
- All open pit mine equipment is assumed to be diesel-hydraulic.
- Blasting costs are included in the established mining unit costs.

Appendix D – UG Mining

1.0 Underground Block Cave

1.1 Selection of Mining Method

The selection of block cave mining as the preferred underground mining method was made in the 2013 preliminary economic assessment carried out by Moose Mountain Technical Consultants. This 2017 study has looked at different underground mining methods at various cut-off grades and has concluded that block caving is still the best approach to mining the Kwanika underground deposits. A preliminary optimization process has been carried out to determine the location of the extraction level, both laterally and vertically, with the goal of maximizing ore grade of the rock mass. A more detailed optimization is warranted at a higher level of study.

Following the AMEC recommendation and a general block cave configurations, the resultant overall mining outline for the block cave stope has been delineated. Note that the 3D shape used for the block cave is the outside limit after caving is completed. The resultant tonnes and grade therefore are inclusive of mining loss and dilution. (AMEC Report, 2013)

At more advanced levels of study, a production schedule would be optimized by drawing ore from higher grade areas of the stope earlier in the schedule. At this level of study some allowance for this grade optimization has been accomplished by dividing the stope into four distinct mining domains so that production can be scheduled from higher to lower grades over the block cave life. The domains or cave blocks are listed in the Table below.

Table 1-1 Block Cave Mining Domains

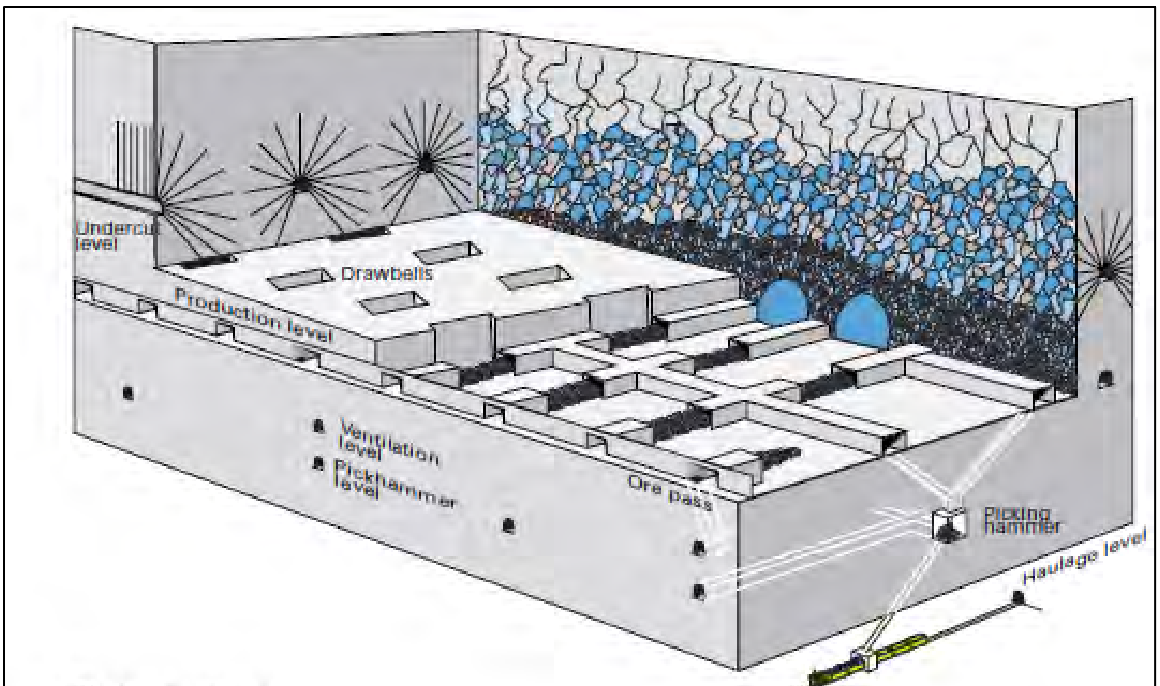
Stope Mining Domains	W-West	West	Tall	East	Total
Extraction Level (m):	470	470	470	520	
NSR ($/t):	60.57	45.96	47.08	32.57	47.47
Tonnage[1] (Million)	4.37	6.16	29.06	2.48	42.08
Footprint (m2):	23,952	23,047	36,463	10,756	94,218
Average Height (m):	66	97	290	84	162
Average Width (m):	171	192	166	145	170
Average Length (m):	140	120	220	74	554

1[1] Includes development ore, drawbell and undercut ore and cave ore.

1.2 General Description of Block Caving

Block Caving is a low-cost, large-scale production underground mining method applicable to:

- low-grade, massive orebodies with large dimensions both vertically and horizontally;
- a rock mass that behaves properly, breaking into blocks of manageable size;
- a ground surface which is allowed to subside.



Block caving layout.

Figure 1-1 Schematic Showing Typical Block Caving Arrangement (from Atlas Copco)

Block Caving is based on gravity combined with internal rock stresses, to fracture and break the rock mass. The drilling and blasting required for ore production is minimal and applies to only the undercuts and drawbells. Caving is induced by undercutting the block by blasting, sometimes with the assistance of pre-conditioning, thus rendering the rock mass unable to support the overlying rock.

Gravitational forces, in the order of millions of tonnes, act to fracture the block. Continued pressure, and secondary blasting to break the rock into smaller pieces to pass the drawpoints, where the ore is handled by load-haul-dump units (LHDs) which tram to ore passes that load trucks on the level below.

1.3 Level Development

Level development for block caving comprises the following general functions with the level elevation for this plan included in brackets:

- Pre-Conditioning Level (540m elevation)
- Undercut Level (490m elevation)
- Fresh Air Level (480m elevation)
- Extraction Level (470m elevation)
- Truck Loading Level (445m elevation)
- Exhaust Air/Dewatering Level (440m elevation)

Figure 1-2 shows the various levels. The dimensions and equipment specification are from the referenced document, configurations and equipment will be adapted for the Kwanika Central Zone block cave in advanced studies.

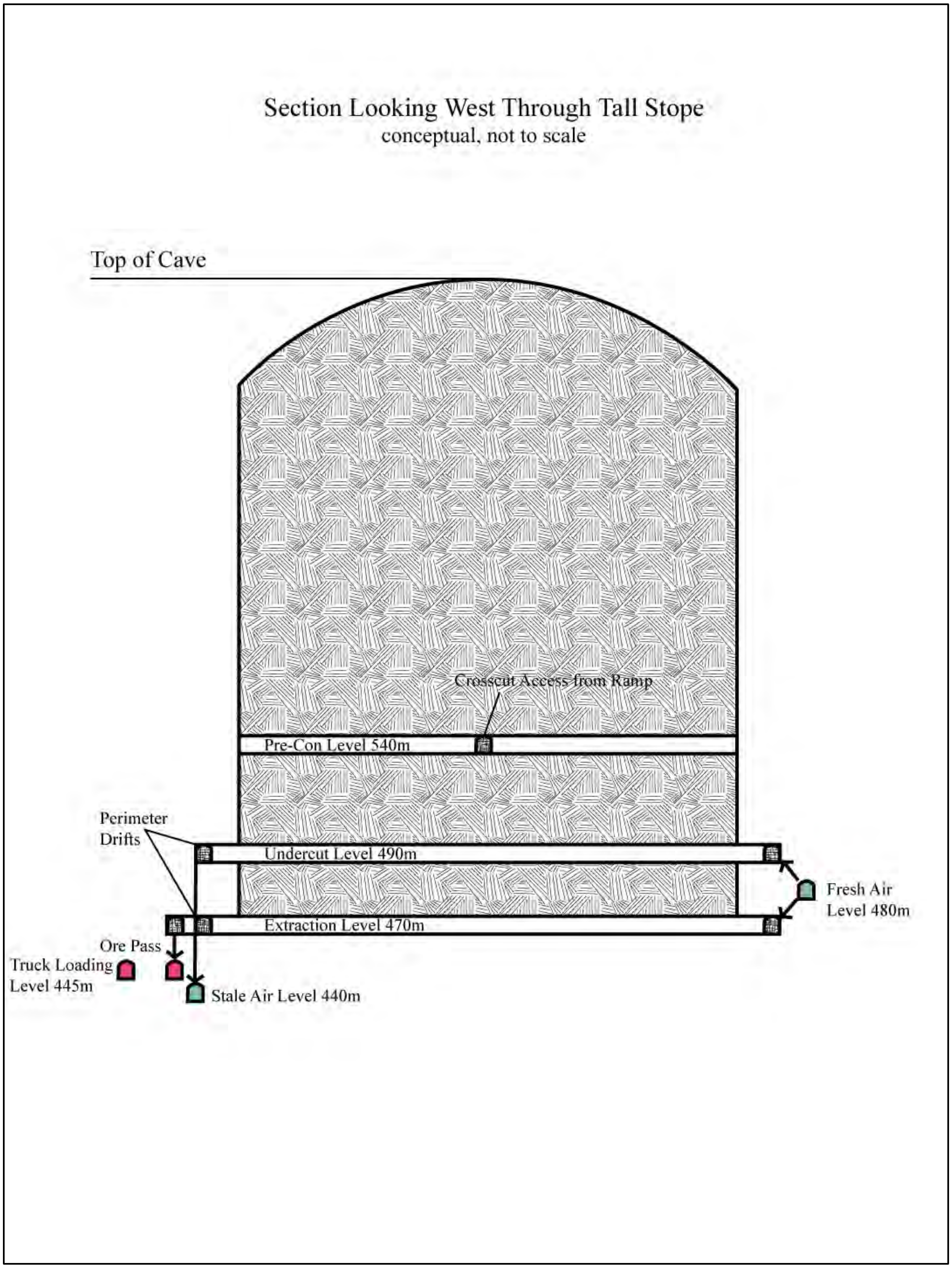


Figure 1-2 Conceptual Section of Block Cave Showing Development Levels

[Figure 1-3] below shows the general design for the Central Zone Block Cave. Ramp access starts from surface with the portal of the decline located close to the coarse ore stockpile adjacent to the mill.

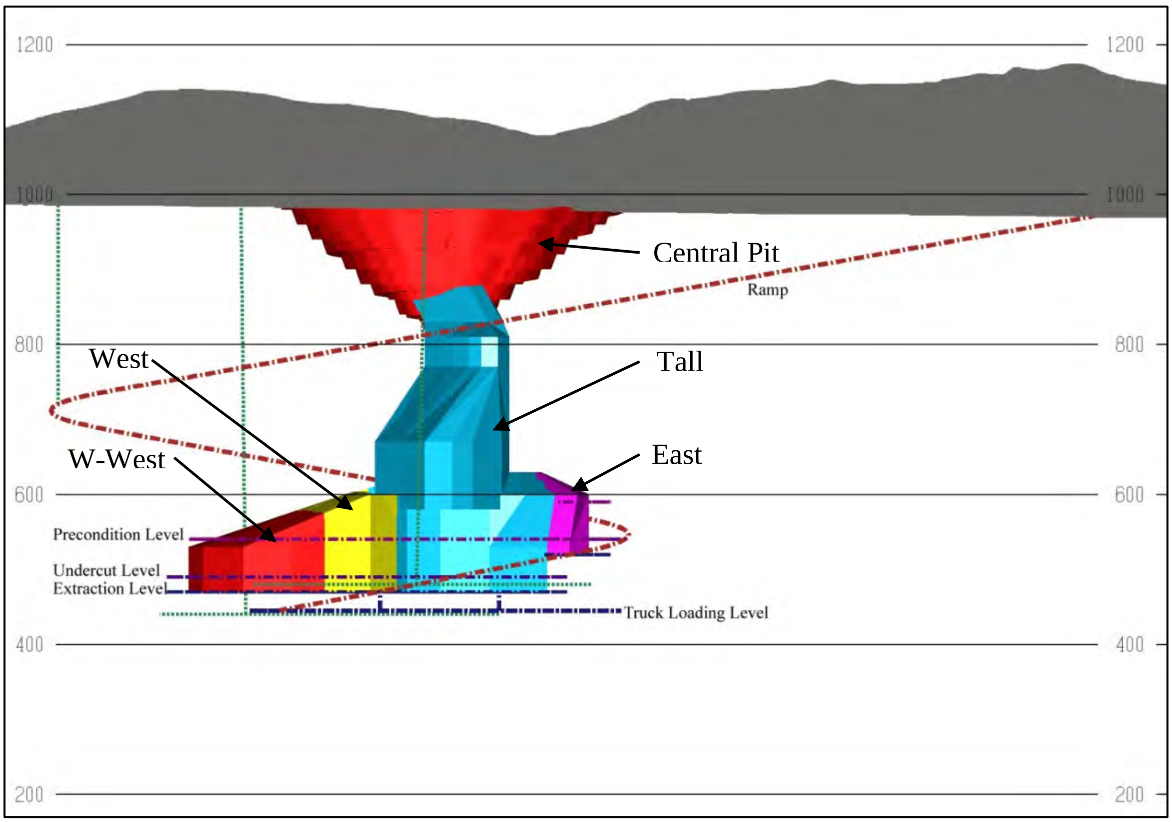


Figure 1-3 Central Zone Block Cave General Layout

Mining starts with the Central open pit phases while the underground development is in progress. The decline must advance to the development levels and sufficient level development must completed in order to start underground ore production from the caving operation, before the open pit production is exhausted. Level development then is subsequently continued as the caving advances laterally across the stope foot print.

A description of the components of the development levels follows.

1.3.1 Pre-Conditioning Level (540m)

A pre-conditioning[1] level has been included in the design at the 540m elevation with the purpose of providing a platform for carrying out the pre-conditioning activity[2] of the overlying rock mass to ensure successful caving (See **Figure 1-4**).

[1] Per the following document "Preliminary Caveability Assessment – Kwanika Deposit, Stephen Godden, Amec, 04 April, 2012", MMTS has included pre-conditioning in the mine design.

[2] Pre-Conditioning is a method of loosening the natural rock fractures to assist in natural caving. Common methods include hydro-fracturing and longhole drilling and blasting.

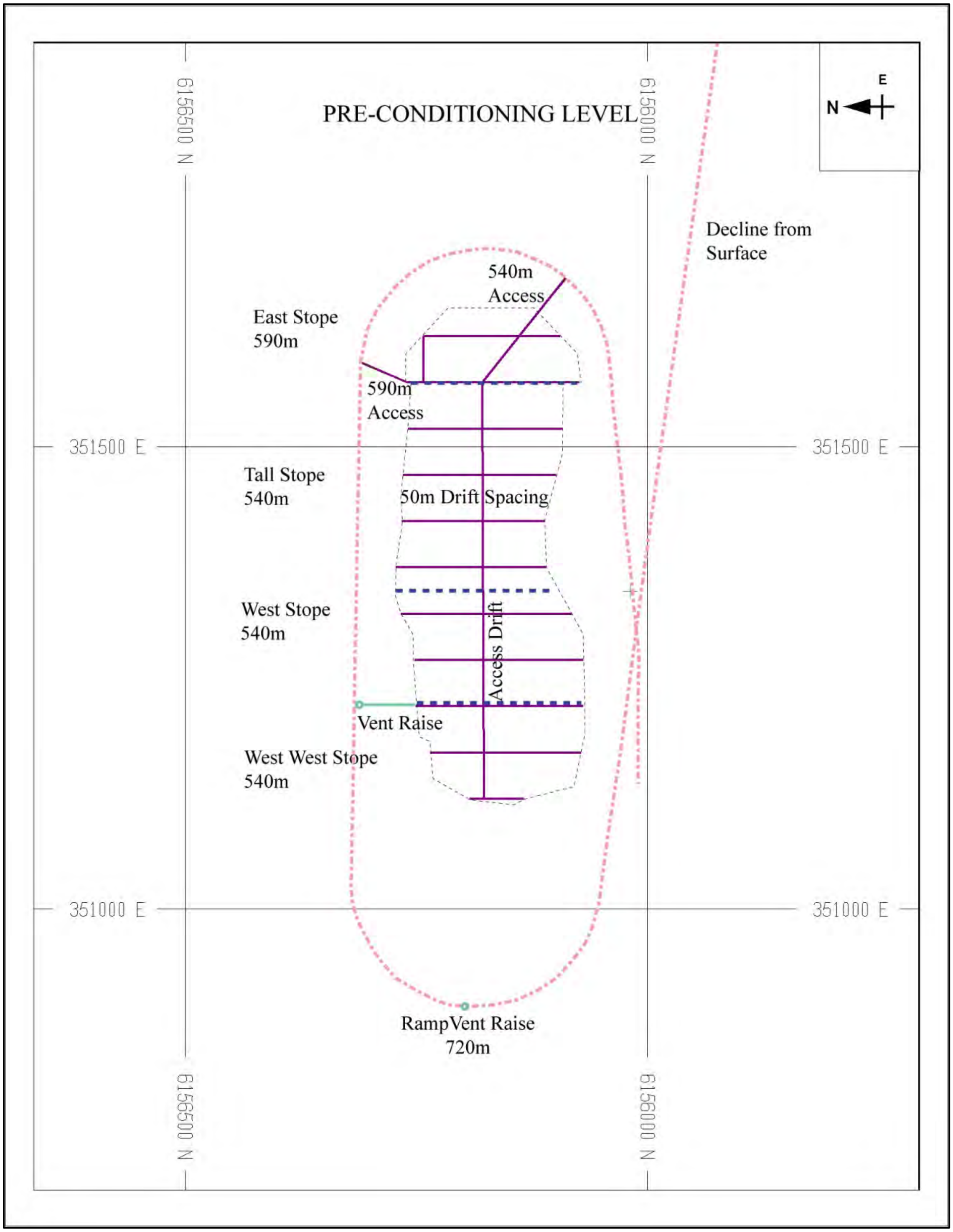


Figure 1-4 Preconditioning Level

At this stage of design, the requirement for pre-conditioning has not been evaluated but has been included for conservatism. The preconditioning level comprises 12 drifts running the width of the block, all connected by a cross-cut from the access ramp and is located approximately 50 metres above the undercut level. Fresh air will be provided from the ramp and exhaust air will leave through a vent raise. Note in the stope design that the Pre-conditioning is designed (and costed) all at one elevation which provides for the upside design quantities. More detailed design will be required in future studies to account for the variable stope height across the Mining domains and may reduce the preconditioning quantities.

1.3.2 Undercut Level (490m)

Development for block-caving applying conventional gravity flow requires an undercut, where the rock mass underneath the block is fractured by longhole drilling and blasting in preparation for caving.

The undercut level comprises 20 undercut drifts running north-south along to the edges of the cave footprint, and a perimeter drift located external to the cave footprint, which allows working crews 360˚ access to the undercut drifts and ensuing undercutting from those drifts.

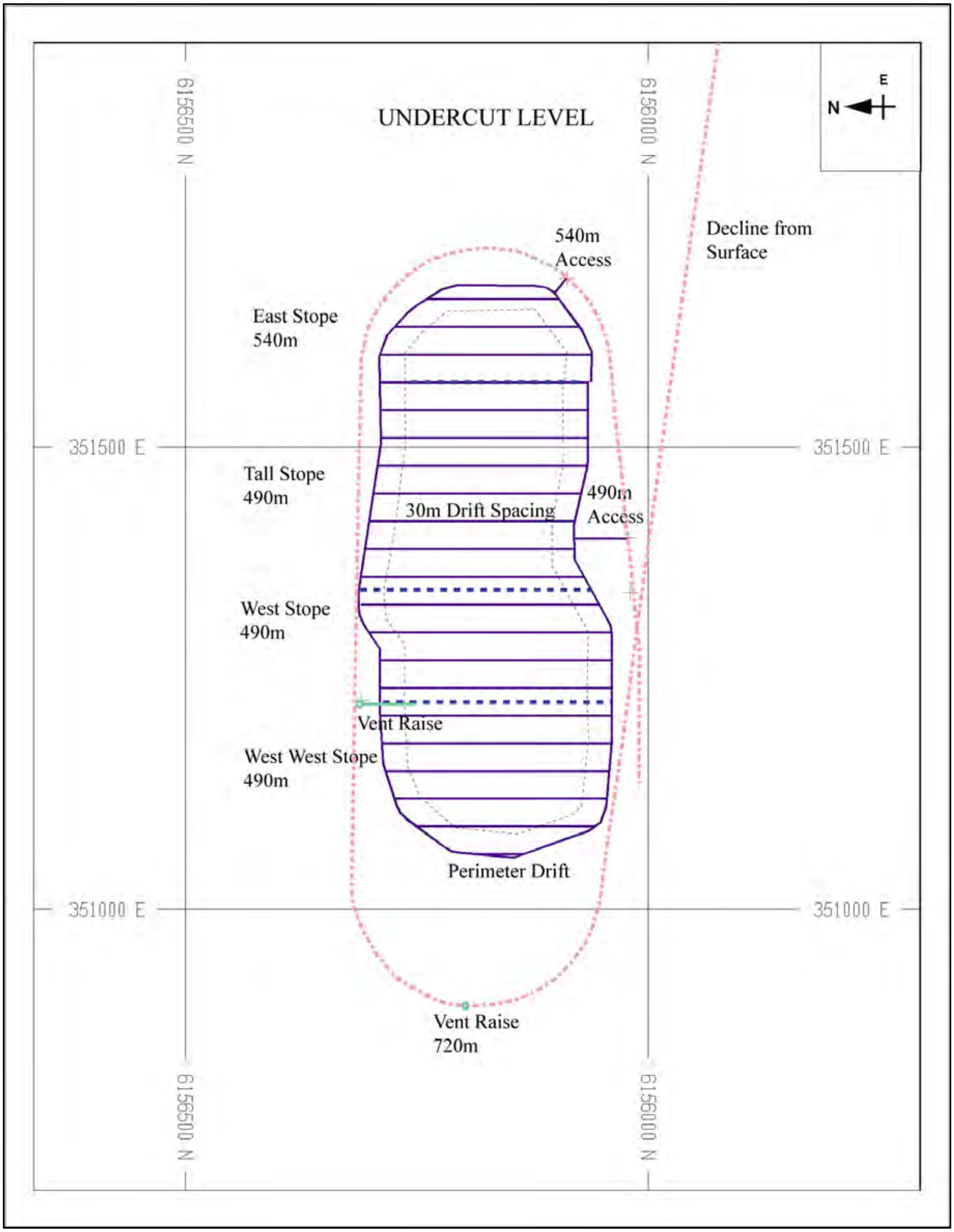


Figure 1-5 Undercut Level

The undercut drifts are spaced 30 metres apart on centre and allow for the excavation of long triangular troughs, 20 metres high which will connect at the 510m elevation. The trough below the undercut level is the top part of the drawbell, as shown in Figure 1-6.

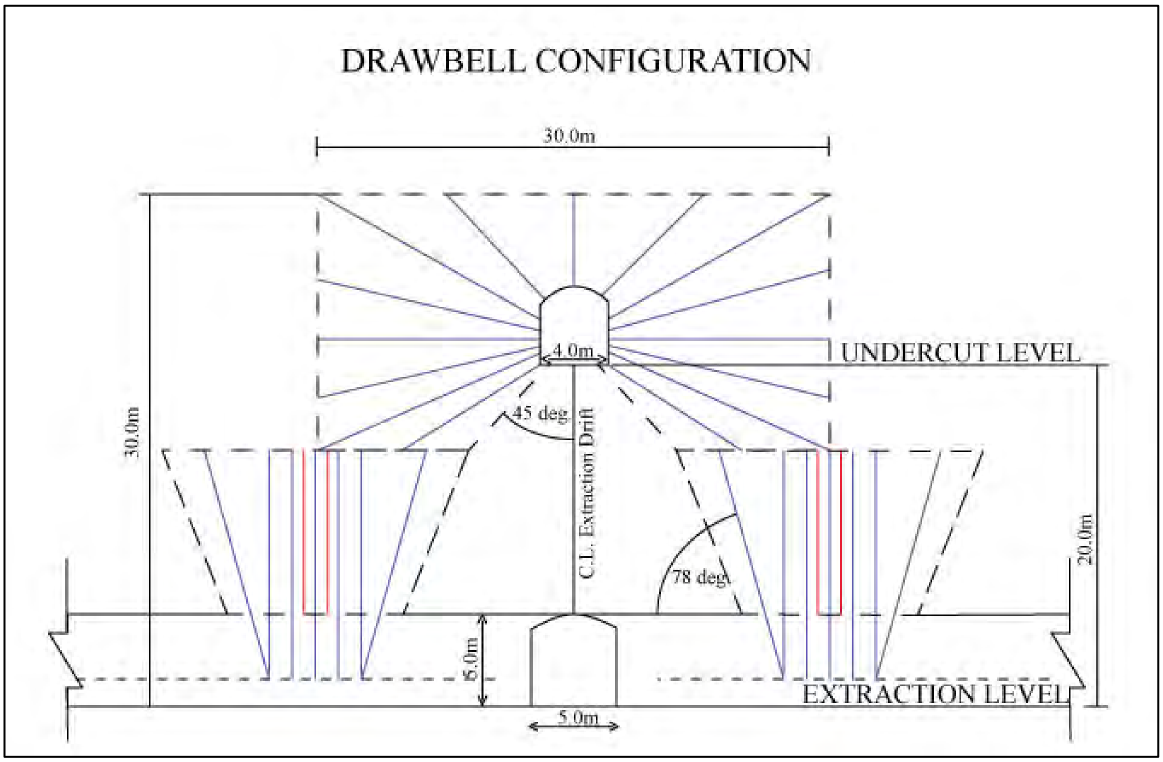


Figure 1-6 Drawbell Configuration

When the fanned drill pattern above the undercut level is blasted, the ore body is entirely undercut thus initiating the cave for this material to be collected in the drawbells on the extraction level below.

1.3.3 Extraction Level (470m)

The extraction level comprises 20 extraction drifts running north-south to the extremities of the cave footprint, along with a perimeter drift located outside the cave footprint, which allows working crews' 360˚ access to the work areas.

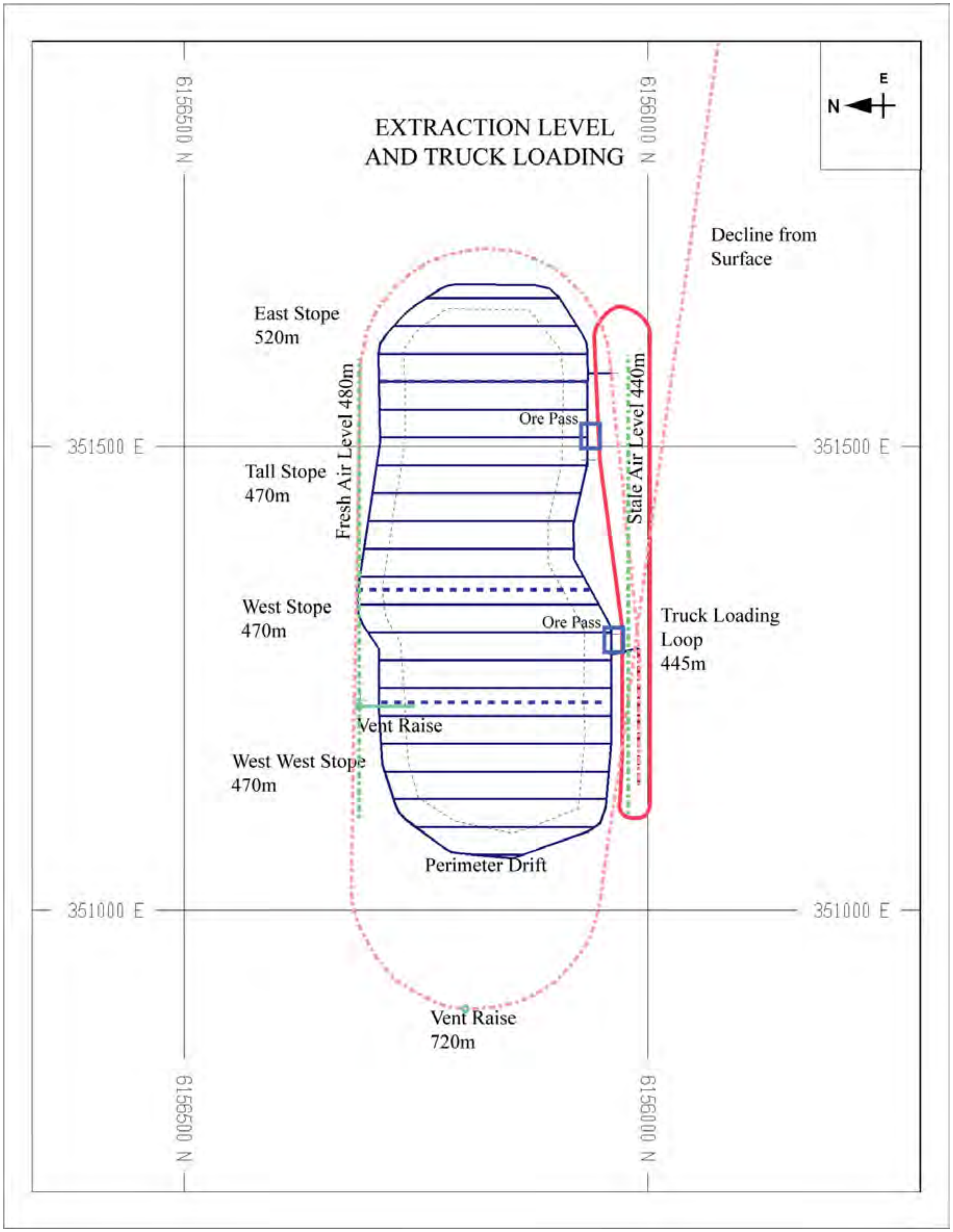


Figure 1-7 Extraction Level

The extraction drifts area spaced 30 metres apart on centre and along their length, drawbells are developed every 15 metres. These are shown in Figure 1-7 as the lower part of the cone. Each drawbell is served with two drawpoints, one from each of the adjacent extraction drifts. The purpose of the drawbells is to funnel the broken ore from the undercut into the two drawpoints associated with each drawbell.

LHDs will load from the drawbell and tram to the ore passes which each have as stationary rock breaker and grizzly. Ore then drops into an ore pocket and chute arrangement for truck loading on the level below.

1.3.4 Fresh Air Level (480m)

The fresh air level runs east-west and north of the cave footprint at the 480m elevation. It will provide fresh air to the extraction and undercut levels through a series of raises. Additionally, a fresh air connection will be made to the truck loading level. See Figure 1-7.

1.3.5 Truck Loading Level (445m)

The truck loading level is located directly under the two ore passes at the south end of the extraction level. This level comprises a loop from the main ramp whereby trucks will leave the ramp, travel clockwise through the loop, enter the truck loading level from the west, travel east, then will load from one of the two ore bins, before exiting the level back to the ramp. See Figure 1-7.

1.3.6 Return Air Level (440m)

The return air level runs east-west and south of the cave footprint at the 440m elevation. It will take exhaust air through a series of raises from the extraction and undercut levels as well from the truck loading level. Additionally, since this level is the low point of the mine, the sumps will also be located here. See Figure 1-7.

1.4 Underground Operations

Unit operations comprise the following:

- Development of undercuts and drawbells by owner
- Production mucking by owner
- Ore Haulage to Surface
- Mine development

1.4.1 Development of Undercuts and Drawbells

Opening up the bottom of the rock mass above the undercut level or "undercutting" and drawbell development, will be carried out using longhole drilling and blasting techniques. These activities will effectively prepare drawbells for production at the rate of 32 drawbells per year. All of the undercutting will be in ore whilst approximately 55% of the drawbell development will be in ore with the remainder in waste. Development of undercuts and drawbells is operating cost and will be carried out using owner's resources.

1.4.2 Production Mucking

Between four and five 17-t load-haul-dump units will be required to work on the extraction level every shift. These units will tram from drawpoints designated on a shift by shift basis, to one of two ore passes. Each of the load-haul-dump units will work with a mobile rockbreaker in their designated drawpoints. Additionally, there will be two blockholers to work along with this fleet, to bring down hang-ups in the drawpoints. Secondary blasting will take place in the drawpoints if the oversize is too large for the rockbreakers to handle[3].

The two ore passes will be located outside the cave footprint on the south side of the cave. Each ore pass will have three entrances to allow muck to be dumped from three different sides. Each ore pass will be equipped with a grizzly with spacing 1.0m x 1.0m. A stationary rockbreaker will work at each grizzly to further reduce oversize and ensure the production rate. The ore passes below the grizzlies will have a capacity of 1,000 tonnes each, enough to load eighteen 55-t trucks each. All activities associated with production mucking are an operating expense and will be carried out using the owner's resources.

1.4.3 Ore Haulage to Surface

The selected method for hauling ore up the access ramp is with 55-t electric haul trucks with trolley-assist. The catenary line providing power to the trucks will run from the truck loading level to surface. At surface, the trucks will exit the ramp, leave the catenary power and will use battery power whilst dumping their loads near the gyratory crusher then return to the ramp.

1.4.4 Mine Development

All development, ramps, raises and direct level development will be carried out by contractors.

[3] The intention is to maintain a steady draw from each block, and records are kept of volumes extracted from individual draw-points. It is often necessary to assist the rock mass fracturing, by longhole drilling and blasting in widely spaced patterns.

Vertical development will include the primary ventilation raises from surface, an intermediate ventilation raise from surface during ramp development, the ventilation raises from the fresh air level to the undercut and extraction levels and the return air raises from these levels to the return air level.

Table 1-2 below shows the summary of life of mine development.

Table 1-2 Life of Mine Development

Type	Metres
Ramps & Misc. Excavations	16,332
Direct Level Development	12,737
Raises	5,214
Total:	**34,283**

1.5 Technical Services & Supervision (Underground Operations and Maintenance)

Technical services will comprise a crew of professionals including geologists, mining engineers, surveyors, drawing technicians, environmental technicians and samplers. This department will support the operation on a daily basis (single shift) including weekends. Samplers and beat geologists will carry out their work two shifts per day.

Operational supervision will include the manager of mining, mine general foreman, mine supervisors and trainers. These personnel will support the operation on a daily basis (single shift) with the mine supervisors providing coverage two shifts per day.

Maintenance supervision will include the manager of maintenance, maintenance planners and shop foreman to provide daily support for the mining operation.

Both mechanical and electrical maintenance will be performed and charged to specific pieces of equipment, mobile and stationary and will be performed on a two shift per day basis.

1.6 Mine Safety

Mine safety will include ongoing training of personnel from the maintenance, operations and technical services departments. Additionally, mine rescue teams will be trained from these pools of personnel. Two sets of mine rescue gear will be available with one located on surface and the second in one of the refuse stations. Mine rescue support will be arranged with other underground mines operating within the local area.

A mine warning system will be utilized to warn personnel underground of an incident. This can be done through a radio system sending signals to the cap lamps worn by underground personnel. Generally, a signal would send personnel to the nearest refuse station. The principal means of egress from the mine will be the access ramp, which will be in fresh air. Secondary Egress will be from the fresh air raise, which is accessible from each mine level and will be equipped with a ladder way.

1.7 Mine Equipment

Mine equipment comprises stationary equipment as well as mobile equipment. Mobile equipment is further divided into that provided by the owner and that provided by the company as shown in Table 1-3 below.

Table 1-3 Mine Equipment

Stationary Equipment	Units
UG Substation	1
UG Power Distribution	1
UG Pumping & Drainage	2
Compressors	2
Trolley Assist Catenary	3,970m
Mine Rescue Gear	2
Refuge Stations	4
Pan Feeders	2
Primary Ventilation Fans	2

Mobile – Owner	
Haul Truck (55t) trolley assisted	8
LHD (6CM)	2
LHD (9CM)	6
Longhole Drill	1

Grader	1
Personnel Carrier	3
Mobile Rockbreaker	5
Blockholer	2
Stationary Rockbreaker	2
ANFO Loader	1
Shotcrete Sprayer	1
Concrete Mixer	1
Boom Trucks	2
Pickup Trucks	6
Scissor Trucks	2

Mobile - Contractor	
Jumbo Drill Rig (2-Boom e/h)	3
Development Haul Truck - 40t	5
Development LHD (9CM)	4
Rockbolting Jumbo	2
Emulsion Loader	2
Scissor Truck	3

1.8 Underground Access and Infrastructure

1.8.1 Access

Access to all levels of the block cave will be provided by a ramp from surface (1,000m elevation) down to the Exhaust Air/Dewatering level (440m elevation). The ramp will be driven establishing remuck bays[4] every 150 metres as well as passing bays every 300 metres. Passing bays allow for two-way traffic during both development and operations.

1.8.2 Power Supply, Catenary, Heating and Ventilation Systems

The main power supply will be located in the fresh air raise and will run to a substation located on the extraction level. From there, power will be distributed to all the other levels for auxiliary ventilation, pumping and lighting. The stationary rockbreakers at the grizzlies on the extraction level will also need power. Lastly, power will be provided to the parking area, shop and return air level where the dewatering pumps are located.

A catenary line on the decline will provide power assist to the electric haul truck fleet. This will increase the travel speed, reducing the required number of trucks. It will also provide lower cost energy to the trucks considering the mine is serviced by the BC power grid. It also improves air quality underground which in turn reduces the ventilation requirements.

1.8.3 Ventilation & Heating of Mine Air

The primary ventilation circuit comprises two 5m diameter borehole raises from surface, of which one will be fresh air and the other, exhaust air. The fresh air raise will bring fresh air down to the fresh air level from where air will flow through a series of raises to the perimeter drifts located on the north side of both the undercut and extraction levels. The air will then be directed south through the undercut and extraction drifts and will be collected on the south side of these levels and dispatched down to the return air drift, which runs underneath these two levels.

The return air drift brings stale air to the exhaust borehole back to surface. This same circuit also provides ventilating air to the maintenance shop and parking area on the extraction level. Only minor air flows are required in the ramp during production because the haulage fleet will be electric. There will however be diesel-operated service equipment that will need to use the ramp for access. Ventilation to the ramp will be provided by splitting off a small portion of air from the fresh air raise and directing it up the ramp.

The total quantity of ventilating air is estimated to be 540,000 CFMs supplied by two vane-axial fans located at the top of the fresh air raise. Only one fan will operate whilst the other will be upon standby. A propane-fired heater will be located in series with these fans. The heat

[4] The mucking activity in the tunneling cycle is the longest thus remuck bays are used to muck the advancing face as quickly as possible to allow the next unit operation at the advancing face, to take place. As that activity is taking place, the stockpiled waste in the remuck bays is loaded into haul trucks for removal from the ramp to surface.

required for the mine air is based on the expected temperature rise necessary to heat the ambient air to -2°C prior to its introduction underground.

1.9 Mine Services

Mine services will comprise the following and will be carried out by company forces and equipment:

- Road maintenance
- Mine dewatering
- Maintenance of power supply, catenary line, heating and ventilation systems
- Delivery of fuel, explosives and shop supplies
- Rehabilitation

1.9.1 Road Maintenance

Road maintenance will be carried out using a motor grader and 6-cm load-haul-dump unit. Generally speaking, road maintenance on the ramp will take place four hours a day during shift changes. The other areas of the mine requiring road maintenance will be the truck loading level and occasionally the extraction level.

1.9.2 Mine Dewatering

This facility comprises two sump chambers and a pump chamber. It will be located at the lowest elevation of the mine, being the 440m elevation, which is the low point on the ventilation-drainage level; allowing gravity drainage to the two sump chambers. Each sump will comprise settling and clean water compartments.

Only one pump will run at a time with the other acting as back-up. Settled water will be pumped through a pipe located in the fresh air raise and will be discharged into the tailing pond.

1.9.3 Maintenance of Power Supply, Catenary, Heating, and Ventilation

Mine electricians will maintain and extend the electrical network throughout the mine life.

The catenary line on the decline will require constant maintenance throughput the mine life. This will generally be done during shift changes to minimize impact on operating time.

The mine air heater and principal fans will be located at the top of the fresh air raise and will require maintenance on an ongoing basis. Additionally, as the mine is developed and both the undercuts and extraction levels are opened up, bulkheads will need to be constructed and/or removed.

1.9.4 Delivery of Fuel, Explosives and Shop Supplies

Fuel will be dispatched from surface to the parking bay through a lined borehole. This fuel will be used by both the contractor's equipment and the mine owner's equipment. Dedicated crews will be responsible for the daily delivery of explosives (emulsion) to magazines located on both the undercut and extraction levels. These explosives will be used by both the contractor's development crews as well as the company's operating crews. Shop supplies will be sent down to the shop on daily scheduled deliveries from either surface warehouses or direct drop off from the supplier at the portal.

1.9.5 Rehabilitation

Maintenance of all levels below the undercut level will be required throughout the mine life. Most of the maintenance will be in the rehabilitation of the drawpoints which are subject to rock stress and damage from secondary blasting of oversize from large blocks from the cave, passing through the drawbells. Rehabilitation of the extraction drifts is also required as well but to lesser extent than the drawbells. Only minor rehabilitation will be required in the other levels. As drawpoints wear or crumble from rock stresses, they'll need to be rebuilt which includes installation of ground support comprising steel, rebar bolts, screen and shotcrete. These costs have been included in the annual operating budget.

1.10 Schedule (Access, Level Development and Stope by Stope Sequence)

1.10.1Access and Level Development

The access ramp from surface will be collared near the process facility at the 1,000m elevation. Using a single development crew with an average advance rate of 5.5m/day, the 17% gradient ramp will reach the pre-conditioning level, elevation 540m, as indicated in the schedule below. At this point, the crew can now work in multiple headings with productivity of 8.0m/day.

Table 1-4 Schedule Showing Access to Mine Levels

Level	El (m)	Ramp Distance (m) from Surface	Time Elapsed (Days)
Pre-Conditioning Level	540	2,706	492
Undercut Level	490	3,000	545
Fresh Air Level	480	3,059	556
Extraction Level	470	3,118	567
Truck Loading Level	445	3,265	594
Stale Air & Dewatering Level	440	3,294	599

Referring to the above Table, approximately 545 days into the schedule, the undercut level is reached at which time a second development crew can be deployed with productivity of 8.0m/day. Two development crews are used until day 1,550 of the life of mine underground schedule, after which time only one crew is needed. Access to the remaining mine levels are also shown in the Table above.

1.10.2Stope Sequence

Once the undercut and extraction levels have been reached and sufficiently developed, production mining crews are able to commence opening up the undercuts and developing the drawbells. Approximately 1,000 days into the schedule, the undercut and drawbells are significantly enough developed to allow caving to commence at a combined[5] extraction rate of 6,000tpd. After approximately 1,460 days, full production is achieved at the rate of 15,000tpd.

The cave will be initiated with the W-West domain in order to provide the highest grade first. This will be followed by initiating a second cave under the Tall domain, followed by the West domain. Lastly, the East domain, whose extraction level is located at the 520m elevation, will be initiated to complete the caving sequence.

1.10.3Production Rate

The production rate is dependent upon the rate at which caving can take place and is dependent on establishing drawbells to match the production rate. It has been assumed that caving can proceed at a rate of 0.20 vertical metres/day, which is consistent with other technical studies

[5] Production comprises ore from undercutting, drawbell development and caving.

carried out on caving projects in northwest B.C. Thus, approximately 60 drawbells will be required to reach a caving rate of 15,000 tpd. This is achieved by developing between two and three drawbells per month for the life of mine. During this period, approximately 209 drawbells will be developed.

Appendix E – Capital Costs

1 Summary

Initial capital has been designated as all capital expenditures required producing copper concentrates for shipment to contract smelters. Sustaining capital covers expenditures after start-up and includes underground mining equipment and infrastructure and mine closure and reclamation. A summary of the major initial capital costs is shown in Table 1-1:

Table 1-1: Initial Capital Cost Summary

WBS Area	Direct Costs	Initial Capital Cost ($X1000)
10	Overall Site	15,700
20	Open Pit Mining – Pre- Production	28,600
20	Open Pit Mining - Equipment	2,500
20	Underground Mining - Development	39,500
20	Underground Mining – Direct Level Development	4,800
20	Underground Mining – Equipment and Infrastructure	35,100
40	Processing Plant (including Ore Handling)	120,000
50	Tailing Storage Facility	35,000
50	Water Management	23,000
70	On-Site Infrastructure	38,300
80	Off-Site Infrastructure	18,800
	Sub-Total Direct Costs	**361,300**
	Indirect Costs	
90	Project Indirect Costs	41,000
98	Owner's Costs	13,000
99	Contingencies	61,000
	Sub-Total Indirect Costs	**115,000**
	Total Initial Capital Cost	**476,300**

Sustaining and closure capital costs are summarized in Table 1-2:

Table 1-2: Sustaining Capital Cost Summary

Sustaining Capital Cost Description	Capital Cost ($X1000)
Open Pit Mining – Sustaining	-
Underground Mining – Equipment and Infrastructure	36,600
Closure	46,300
Total Sustaining and Closure Capital Cost	**82,900**

The following sections detail the basis of estimate, organization of, and different estimating methodologies as they apply to the various components of the estimate.

2 Basis of Estimate

The purpose of the Basis of Estimate is to describe the methodology in the development of the capital cost estimate. The accuracy of the estimate is scoping level (+/- 40%) unless otherwise noted.

All currencies in this section are expressed in Canadian dollars. Costs in this report have been converted using a fixed currency exchange rate of US$0.77 to CAD$1.00 if applicable.

The PEA establishes a plant throughput capacity that enables a competitive operating cost and minimized initial capital cost. The PEA is based on a throughput of 15ktpd.

The capital costs are compiled based on the following parameters:

- Benchmarking process ancillary buildings of similar characteristics, to 2017 costs
- Budgetary estimates for facilities not included in a similar plant
- Commodity rates/units
- Single blended all-in labour rates
- Productivity values
- Estimate base date of Q1, 2017
- Escalation excluded beyond Q1-2017

Table 2-1 presents the basis of estimate used in the PEA:

Table 2-1: Basis of Estimate

Kwanika Project PEA Estimate	Order of Magnitude
Purpose: to establish the plant throughput capacity that produces the optimal financial performance	*Methodology of the development of the initial capital and operating costs*
Accuracy - Indicative Range	+40% to -40%
Level of Engineering Definition	0-2%
CAPITAL COST ESTIMATE	
General Site Items	
Location	Preliminary
Maps and Surveys	Aerial Photo/Contour Plot
Soil Tests & Geotechnical	None
Site Visits	Not essential
Delivery Strategy	Assumed
Mining	
Site Visits	Yes (2011)
Maps and Surveys	Scoping level
Drilling	Exploration
Mining method	Designed (typical)
Mining schedule	Designed
Mine Geotechnical & Hydrology	Typical
Mine Geological Model	Scoping level
Equipment selection	Designed Typical
Process	
Plant Capacity	15 ktpd max
Metallurgical Samples	Scoping Level
Metallurgical Test work	Scoping Level
Energy & Material Bal.	None
Process Flow sheet	Typical

Kwanika Project PEA Estimate	Order of Magnitude
Infrastructure	
Scope of Estimate	Scoping level
Equipment Selection	Assumed - adjusted from typical
Mechanical G.A.'s	None
GA - Structural	None
Piping Drawings	None
Electrical Drawings	None
Detailed Design Drawings	None
Specifications	None
Infrastructure: Power, Water, & Roads	Assumed - adjusted from typical
General Cost Approach	Factored Block Costs
Major Equipment Costs	Data base/Factored
Minor Equipment Costs	Data base/Factored
Civil Work	Assumed
Equipment Foundations	Factored based on in-house experience
Building Foundations	Factored based on building footprint
Equipment Structural Steel	Factored % direct equipment costs
Building Structural Steel	Factored based on building area/volume
Piping & Instrumentation	% direct equipment costs
Electrical	% of total direct costs/benchmarked
Freight & Logistics	% of total direct costs
Indirect Costs	% of total direct costs
Spare Parts	% of total direct costs
Load out	Data bank/Factored
Commissioning & Start-up	Benchmarked from similar projects

Kwanika Project PEA Estimate	Order of Magnitude
Tailing Storage Facility	Benchmarked from similar facilities
Water Management	Benchmarked from similar facilities

Estimate Base Date and Validity Period

This PEA document has a base date of Q1-2017. No escalation has been applied to the estimate beyond Q1-2017.

2.1 Estimate Approach

All equipment and material costs are exclusive of spare parts, taxes, duties, freight and packaging. These costs, if appropriate, are covered in the indirect section of the estimate.

International System of Units (SI) measurement system is used in the estimate.

All capital cost estimates have been derived either from original data, or from historical costs. The source of any estimate being evaluated or reviewed has been examined to ensure that the development of the estimate is appropriate for the purpose of this study.

2.2 Estimate Organization

The Estimate was assembled and coded based on the approved project Work Breakdown Structure (WBS). The WBS is a hierarchical roll up structure of project areas and sub-areas.

2.2.1 WBS Coding

Coding Format:

X## - ## - ####

WBS Area Section Sequence

Numeric #

Table 2-2: WBS Structure

10 Overall site	Area	
10	1010	Site Preparation
	1020	Overall Site Electrical
	1030	Overall Site Controls and Communication
20 Open Pit and Underground Mining		
20	2010	Pre-Production
	2020	Mining Equipment
	2030	Mining Facilities
	2040	Mining Dewatering
	2050	Mining Electrical and Communication
30 Ore Handling		
40 Process		
50 Tailing Disposal		
50	5010	Tailing Disposal
	5020	Tailing Reclaim
	5030	Water Management
60 Environmental		
60	6010	Environmental
70 On-site Infrastructure		
70	7010	Ancillary Buildings
	7020	Site Services and Utilities
	7030	Plant Mobile Equipment
	7040	Temporary Services
80 Off Site Infrastructure		
80	8010	Off-site Infrastructure

90 Project Indirects		
90	9010	Project Indirects
98 Owners Cost's		
98	9810	Owner's Cost
99 Contingencies		
99	9910	Contingency

3 Infrastructure Items

The following WBS areas can be classified as "Infrastructure":

- 10 – General Site
- 50 – Tailings Disposal
- 70 – On-site Infrastructure
- 80 – Off-site Infrastructure

These areas are grouped together for the purpose of this document as they follow a similar estimating methodology and elements of costs. The sections below describe the major or typical items and the elements of costs that underlie each of these infrastructure WBS areas.

3.1 Area 10: General Site

General site items typically include general site preparation and earthworks, site roads, yard lighting, and site electrical distribution (overhead lines).

3.2 Area 50: Tailings and Water Management

The Tailings and Water Management area typically includes the tailings dam construction, tailings pipelines, reclaim barge, and creek diversions.

3.3 Area 70: On-site Infrastructure

On-site infrastructure includes all buildings and services on-site to support the mining and processing operations. Typically these items include an administration and dry building, warehousing, laboratory, sewer and potable water services, and fuel storage. The Kwanika project does not include a truckshop in this category as it is assumed the open-pit and underground mining contractors will supply their own facilities.

3.4 Area 80: Off-site Infrastructure

Off-site Infrastructure includes the external powerline connecting the Kwanika project to the BC power grid.

3.4.1 Area 90: Project Indirects and Contingency

These items are described below in Section 2.4.

3.5 Elements of Costs

The following methods, factors, and elements apply to the infrastructure items described above:

3.5.1 Direct Costs

For major equipment and materials please refer to Table 2-1: Basis of Estimate for the estimate methodology.

3.5.2 Labour Rates and Costs

A single blended labour rate of $100/hr was used for all construction labour in the estimate. These labour rates were developed based on typical construction labour elements.

The labour rates include the following:

- Vacation and statutory holiday pay
- Fringe benefits and payroll burdens
- Overtime and shift premiums
- Small tools
- Consumables
- Personal protection equipment
- Contractors overhead and profit

MMTS assumes the construction man-hours/workweek to be 10 hours a day with three (3) weeks on and one (1) week off rotation. It has been assumed that 60% of the workforce will be local and 40% will come from the Greater Vancouver Area however, the source and availability of labour should be verified in the next phase of the Project.

Travel and living allowances are included in the construction indirect section.

A productivity factor of 1.2 is applied to the labour portion of the estimate to allow for the inefficiency of long work hours, climatic conditions and due to the 3 week in 1 week out rotation. This is based on in-house data supplied by contractors on previous similar projects in Northern British Columbia.

3.5.3 Duties and Taxes
Duties and Taxes (GST) have not been included in the estimate.

3.5.4 Owner's Costs
MMTS has included 5% of the Direct Costs (Process and Infrastructure direct costs) to cover the Owner's costs. MMTS finds this to be a reasonable assumption.

3.6 Indirect Costs
MMTS included for contractors in-directs including:

- Winter and summer road maintenance
- Living out allowance
- Overheads and Profit as a percentage of the direct costs

Indirects are split in the following areas:

1. Mine Area Indirects (included in direct contract mining costs)
2. Engineering, Procurement, and Construction management (EPCM)
3. Water Treatment Indirects
4. Contingency

EPCM is calculated on a percentage basis to include the following:

- Detailed Engineering
- Procurement
- Construction Management

3.6.1 Vendor Representatives (during construction)
Costs for Vendor Representatives are calculated based on the number of vendors over 8 weeks @ $1,500 per day.

3.6.2 Temporary Construction Facilities and Equipment (Construction Indirects)
An allowance has been included.

3.6.3 Commissioning and Start-up
Commissioning and start-up is based on an assessment of the requirements based on historical information.

3.6.4 Spares (Commissioning and Strategic)

Commissioning, mining capital and start-up spares has been included as a percentage of process equipment and mining equipment costs. These costs were based on MMTS in-house data and experience.

3.6.5 Freight and Logistics

The freight and logistics allowance is calculated on a percentage of the equipment and bulk material costs basis, based on MMTS in-house experience.

3.7 Contingency

The estimated contingencies are for undefined items of work which are incurred within the defined scope of work covered by the estimate, which cannot be explicitly foreseen or described at the time the estimate was compiled, due to a lack of complete accurate and detailed information. Therefore, the contingency is an integral part of the estimate. The contingency is not to be considered as a compensating factor for estimating inaccuracy, nor is it intended to cover such items as any potential "changes in project scope", "Acts of God", prolonged labour strikes, labour disruptions beyond the control of the project manager, currency fluctuations or cost escalation beyond the estimated rates.

The estimated contingency allowance is assessed based on British Columbia Securities Commission (BCSC) guidelines.

It is considered that this estimate will adequately cover minor changes to the current scope, to be expected during the next phase of the Project.

No provision has been made, or contingency allowed, for major design amendments or changes to the scope, which may result from additional test work or pilot plant testing which would be carried out to verify the current design in the next phase of the Project. No provision has been made, or contingency allowed, for major design amendments or changes to the scope, which may result from additional geotechnical studies or further investigation of the site conditions.

4 Area 20: Open Pit and UG Mining

4.1 Open Pit Mining

Table 4-1: Open Pit Mining Capital Costs

Direct Costs	Initial Capital Cost (KCAD$)
Open Pit Mining – Pre-Production	$28,600
Open Pit Mining - Equipment	$2,.500

4.1.1 Open Pit Mining – Pre-production

Mining pre-production includes the pre-stripping of Central phase 1 and 2 during the pre-production period (year -1) and is estimated to take one year. This area accounts for all contractor mining operations including labour, fuel, and consumables for the operating open pit equipment.

4.1.2 Open Pit Mining – Equipment

The mining estimate is based on a contract-mining scenario common in the industry. This approach minimizes initial capital costs as no equipment purchases are required as equipment is supplied by the contractor. Additionally, the mining schedule calls for short-term campaigns of open pit mining in Central pit before shutting down during underground mining, and then restarting several years later for South pit mining. Therefore, purchasing and owning the equipment would be less economical due to the short duration of each open pit, and the long idle periods during underground operations.

A mobilization cost is incurred in the initial capital to cover the contractor's costs to move to and commission equipment at site. The mobilization cost is estimated at 6% of the capital purchase cost of the mining equipment fleet and is posted to the "Open Pit Mining – Mobile Fleet" code (above). The mobilization costs are based on 136-tonne trucks and 12m3 hydraulic shovels as were proposed in the 2013 PEA. The contractor will choose whatever best fits their quote, as such an equipment list or quantity is not estimated.

The open pit contractor is expected to provide their own field office and maintenance facility on-site and supply any capital replacements and spare parts.

4.2 Underground Mining

4.2.1 UG Development

Underground development capital costs are based on historical unit rates from contractors or other underground operations. The costs are based on the excavation dimensions and length in meters.

4.2.2 UG Mobile Equipment

UG Contractor Fleet

There is an underground contractor mobile fleet for the Kwanika underground operation that has a mobilization cost based on comparable contractor quotes or costs from other projects. The contractor fleet is assumed to be made up of the following, but the contractor chosen will be free to choose whatever best fits their proposal:

Table 4-2: UG Mining Contractor Equipment

Mobile - Contractor	
Jumbo Drill Rig (2-Boom electric-hydraulic	3
Development Haul Truck - 40t	5
Development LHD (9m^3	4
Rockbolting Jumbo	2
Emulsion Loader	2
Scissor Truck	3

Owner Fleet

Certain underground activities will be conducted by the Owner's personnel, including loading and hauling ore from the drawpoints and delivering to the coarse ore stockpile near the plant as well as general underground services. Contractor activities are specifically related to development and block caving. The owner's underground fleet is estimated using recent quotes from equipment suppliers or estimated costs based on similar projects or studies. Equipment costs are assumed to be all-in, inclusive of freight, assembly, and spare parts for the purpose of this study.

Table 4-3: UG Mining Owner's Fleer

Equipment	LoM Qty	Unit Cost ($000's)
Haul Truck (55t) trolley assisted	8	$500
LHD (6 m^3)	2	$1,200
LHD (9 m^3)	6	$1,550
Longhole Drill	1	$1,150
Grader	1	$240
Personnel Carrier	3	$300
Mobile Rockbreaker	5	$230
Blockholer	2	$580
Stationary Rockbreaker	2	$190
ANFO Loader	1	$405
Shotcrete Sprayer	1	$630
Concrete Mixer	1	$445
Boom Trucks	2	$330
Pickup Trucks	6	$50
Scissor Trucks	2	$380

Other underground capital costs include stationary equipment like compressors, refuge stations, conveyors, and pumps, as well as underground electrical substation and distribution. These costs are based on historical data for comparable projects.

5 Area 30 and 40: Ore handling and Process Costs

Process costs are factored and are based on similar process plants constructed in BC in the past 5 years using MMTS's database. Where necessary adjustments have been made for throughput by factoring.

6 Area 60: Environmental Costs

An allowance is included for fisheries and fish habitat rehabilitation work that may be required due to stream diversions (included in Water Management costs). Future environmental assessment studies will define actual requirements.

A typical closure and reclamation plan is assumed for this level of study. Costs are estimated for the resloping of the RSF/TSF buttress, reclamation of the open pit, closure of the underground, and decommissioning of all plant, equipment, and structures.

Table 6-1 Reclamation and Closure Costs

Reclamation Costs					
Pits		**Qty**	**Unit Cost**	**Unit**	**Total**
Seed/Fertilize		90	$1,000	ha	$90,000
Application		90	$1,000	ha	$90,000
					$180,000
Tailings					
Resloping		90	$8,300	ha	$747,000
Seed/Fertilize		90	$600	ha	$54,000
Application		90	$1,000	ha	$90,000
Seal and Reveg Pond					
Till Placement		210	$100,000	ha	$21,000,000
Seed/Fertilize		210	$600	ha	$126,000
Application		210	$1,000	ha	$210,000
					$22,227,000
Other					
Site Prep		80	$5,000	ha	$400,000
Seed/Fertilize		80	$600	ha	$48,000
Application		80	$1,000	ha	$80,000
					$528,000
Contingency (Scoping)		**40%**			**$9,174,000**
Additional tailings and mill closure costs					**$14,200,000**
Total Closure and Reclamations					**$46,309,000**

7 Exclusions

Owner's costs for past exploration, studies and future costs for future studies have not been included as well as certain royalties, taxes, and interests. At this stage of study, the future financial capacities of equity investors or partners are unknown. These details and options will need to be assessed in future more advanced studies.

There is no allowance for foreign exchange fluctuation. All exchange rates used, and all exposure to items sourced in any other currency is presented in the estimate.

In addition the following items are excluded from the initial capital estimate:

- Environmental costs/studies, Baseline, permitting, and EIA requirements mentioned within this report
- Future Exploration Programs
- Permits and Fees (Owner's Cost)
- Finance and Interest Charges (Owner's Cost)
- Working Capital (included in the Financial Analysis Model in subsequent years)
- All scoping, Pre-Feasibility, Trade-off, or Feasibility Study costs prior to construction
- Force majeure
- Taxes – except as included in Owner's Cost
- Overtime
- Cost outside battery limits
- Interest during construction
- Sunk costs
- Sustaining capital costs other than the major items listed in Table 2-1.

Appendix F – Financial Analysis

1 Summary

The Kwanika financial model combines inputs from the production schedule and open pit and underground mining, processing, and other general and administrative costs into an overall model of the potential financial performance of the Project. Revenues are derived from the recovered metal produced from the production schedule and the stated metal price assumptions. This Appendix describes the inputs and the complete parameters and assumptions that drive them.

All currency is Canadian dollars unless otherwise stated.

Financial Inputs are as follows:

- Revenue
- Open Pit Mining Operating Costs
- Underground Mining Operating Costs
- Processing Operating Costs
- G&A Costs
- Capital Costs
- Taxes

2 Revenue

Revenue is driven by the mill feed – that is, the tonnes and grade of material from either the Underground or Open Pit mining activities processed through the mill. See Appendix C and D for details on the mine scheduling. Assumptions for producing concentrate from the milled materials are listed here:

Table 1-1: Metal Prices

Gold Price		$1,270.00	$US/oz
Silver Price		$19.00	$US/oz
Copper Price		$2.90	$US/lb
US Exchange rate		0.770	US$/CAD$

Table 1-2: Process Recoveries

Process Recoveries - To Concentrate	Central	South
Gold	75%	70%
Copper	91%	89%
Silver	75%	75%

Table 1-3: Refinery Costs

Copper Concentrate		
Copper Concentrate Moisture	8	%
Copper Concentrate Grade	24	%
Payable AU	97.75	%
Payable AG	90	%
Losses	0.40	%
AU Refining	$4.50	US$/Oz
AG Refining	$0.60	US$/Oz
Cu deductions	1.0	%
Cu Refining	$0.090	US$/lb
price participation (cu floor)	$1.50	US$/lb
price participation % for amount above prpart	0.0	%
Price Participation Cap	$0.04	US$/lb
Smelting	$90	US$/DMT

Table 1-4: Transportation and Offsite Costs

Freight for trucking	30.00	$/WMT
Freight for rail	15.00	$/WMT
Freight for ships	70.00	$/WMT
Other Offsite Costs (Losses, Ins,Sell,supv,Assay)	16.00	$/WMT

The above parameters are used to calculate the amount of metal in concentrate, and then the metal is multiplied by the prices, less the concentrate costs and losses and less the offsite costs to find the net revenue for the payable metal produced or Net Smelter Revenue.

2.1 Open Pit Mine Operating Costs

Open Pit mine operating costs are applied as a $/tonne unit cost. The unit cost varies by period to account for changes in the haul cycle. The unit cost is applied to tonnes/year of mill feed, stockpile placement, and waste mining. Additional costs are applied for stockpile re-handling, as well as an ore hauls from UG. Details on the unit cost and build-up are found in Appendix C.

2.2 Underground Mining Operating Costs

Underground mining costs are applied as a $/tonne unit cost including level and stope development, and block cave production. Details on the unit cost and build-up are found in Appendix D.

2.3 Processing Costs

A unit processing cost is applied to the mill feed that accounts for power, materials, labour, and facilities operating costs. At a scoping level of study, a typical overall cost is applied as described in Section 21 of the PEA Technical Report.

2.4 General and Administrative Costs

G&A costs are applied per tonne of material processed based on the G&A Estimate in Section 21 of the PEA Technical Report.

2.5 Capital Costs

Initial and Sustaining capital costs are listed in Appendix E. Closure and reclamation costs are listed with the sustaining capital items. These capital items are subtracted from the cashflow by period.

2.6 Taxes

A simplified tax model is used to estimate the Canada Federal Income Tax, BC Income Tax, and BC Mineral Tax. Since at a scoping level of study the engineering for the processes and design are only at 2% of what will be the final design, unit costs and final designs are very preliminary and 'typical' costs have been used in many parts of the study. Also the annual revenue profile is preliminary and will change as the ore body grades, the mine plan, and production schedule are refined in future studies.

The application of taxes in the financial model should be considered as an allowance for taxation, rather than a prediction of what the future operation taxes will be. The following describes how Canadian tax law has been applied to the cash flow developed for this PEA.

Federal Taxes

Federal taxable income is subject to a corporate income tax rate of 15%. In general terms, taxable income is defined as gross revenue minus the following deductions:

- Operating costs
- Capital cost allowance (CCA)
- Cumulative Canadian exploration expense (CCEE)
- Cumulative Canadian development expense (CDEE)
- Provincial mining taxes and royalties

CCA class 41(a) allows accelerated CCA on capital acquisitions made before the commencement of commercial production or for the purpose of a major expansion. This accelerated CCA will be phased over the 2017 to 2020 calendar years. Kwanika will be allowed to claim a percentage of accelerated CCA during the phase-out period.

Provincial Taxes

Taxable income is subject to a corporate income tax rate of 11%. Taxable income for British Columbia is harmonized with the federal system.

Under the B.C. Mineral Tax Act, mines are subject to mining taxes in two stages.

Stage 1 is a 2% tax on "net current proceeds", which is defined as gross revenue minus operating costs and non-capital reclamation costs. Operating costs include all current operating expenses and post-production development costs, but exclude exploration, pre-development and capital costs). Stage 1 tax constitutes a form of minimum tax, which is fully creditable against Stage 2 tax in the current year or future years, with notional interest at 125% of the prevailing federal bank rate.

Stage 2 is a 13% tax on "net revenue", which is defined as net revenue (used for Stage 1 tax) minus capital costs, exploration costs, pre-development costs, and an investment allowance. If a mine has negative "net revenue", the result is added to its cumulative expenditure account (CEA) and can be carried forward indefinitely to reduce net revenue in future years.

3 Financial Model

		PERIOD	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
Description		YEAR	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Mill Feed					5,401	5,400	5,400	5,400	5,400	5,400	5,400	5,400	5,401	5,401	5,401	5,401	5,401	5,401	3,248	
Cu %					0.558	0.529	0.434	0.441	0.441	0.441	0.440	0.441	0.408	0.283	0.307	0.268	0.259	0.209	0.173	
Au g/t					0.599	0.577	0.510	0.526	0.525	0.526	0.522	0.526	0.376	0.130	0.059	0.080	0.097	0.106	0.095	
Ag g/t					1.536	1.502	1.330	1.354	1.353	1.354	1.349	1.354	1.171	1.437	1.732	1.561	1.615	1.233	0.893	
Cu Lb					60,409	57,364	46,979	47,768	47,740	47,765	47,668	47,768	44,171	30,652	33,272	28,380	27,472	22,201	11,026	
Au Oz	$31.10347				78.0	75.1	66.4	68.5	68.3	68.5	67.9	68.5	48.9	17.0	7.7	9.7	11.8	12.9	6.9	
Ag Oz	$31.10347				200	196	173	176	176	176	176	176	153	187	226	203	210	161	70	
Cu US$/lb	$ 2.90				167,214	158,785	130,041	132,225	132,147	132,217	131,949	132,225	122,267	84,846	92,098	78,557	76,043	61,452	30,521	
Au US$/oz	$ 1,270.00				96,506	92,848	82,081	84,707	84,510	84,685	84,006	84,707	60,488	21,010	9,488	12,050	14,635	15,900	8,587	
Ag US$/oz	$ 19.00				3,406	3,331	2,948	3,003	3,000	3,003	2,991	3,003	2,597	3,186	3,842	3,462	3,582	2,735	1,191	
GROSS REVENUE US$ ('000)					267,126	254,964	215,071	219,935	219,656	219,904	218,945	219,935	185,352	109,042	105,428	94,069	94,260	80,087	40,299	
Refinery cost $ ('000)					20,640	19,610	16,099	16,375	16,365	16,374	16,338	16,375	15,050	10,408	11,252	9,625	9,341	7,561	3,751	
Transportation costs $ ('000)					16,257	15,438	12,643	12,855	12,848	12,854	12,828	12,855	11,887	8,249	8,954	7,638	7,393	5,975	2,967	
NET SMELTER REVENUE ('000)	$1.30	FEX			310,020	296,075	250,571	256,399	256,056	256,362	255,178	256,399	213,780	122,956	116,713	104,905	105,682	90,474	45,619	
Open Pit Mining Unit Cost $ ('000)			-	$2.10	$2.48	$2.46	$2.50	$2.50	$2.50	$2.50	$2.50	$2.50	$2.50	$2.76	$2.96	$3.36	$4.00	$4.01	$4.01	
Open Pit Mining Cost $ ('000)			-	-	40,381	19,736	5,929	5,899	5,899	5,899	5,899	5,899	24,454	35,741	41,521	52,189	48,710	10,160	2,456	
UG Mining Unit Cost $ ('000)			-	-	$6.40	$6.99	$7.03	$7.10	$7.16	$7.09	$7.13	$7.24	$7.24	-	-	-	-	-	-	
UG Mining Costs $ ('000)			-	-	9,500	28,302	37,956	38,326	38,683	38,263	38,518	39,085	29,970	-	-	-	-	-	-	
Processing Costs $ ('000)			-	-	48,605	48,603	48,600	48,600	48,600	48,600	48,600	48,600	48,607	48,609	48,609	48,609	48,609	48,609	29,235	
G&A Costs $ ('000)			-	-	10,909	10,909	10,908	10,908	10,908	10,908	10,908	10,908	10,910	10,910	10,910	10,910	10,910	8,728	3,281	
UG Development $ ('000)					56,080	45,271	30,720	17,173	13,918	19,881	11,999	-	-	-	-	-	-	-	-	
TOTAL OPERATING COSTS $ ('000)			-	-	165,475	152,821	134,112	120,906	118,009	123,551	115,924	104,492	113,941	95,260	101,040	111,708	108,229	67,497	34,971	-
REVENUE BEFORE TAXES $ ('000)			-	-	144,545	143,254	116,459	135,493	138,046	132,811	139,254	151,907	99,839	27,696	15,673	-6,803	-2,548	22,977	10,648	-
Start-up Capital Requirements $ ('000)			146,284	221,926																
Sustaining Capital $ ('000)																				
UG Equip and Infrastructure $ ('000)			1,100	34,041	125	24,655	2,625	125	8,790	125	125	-	-	-	-	-	-	-	-	
UG Development $ ('000)			17,553	26,733																
Prestrip Costs $ ('000)				28,607																
Reclamation $ ('000)																				46,309
TOTAL CAPITAL COSTS $ ('000)			164,937	282,700	125	24,655	2,625	125	8,790	125	125	-	-	-	-	-	-	-	-	46,309

Description		PERIOD	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
		YEAR	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Pre-tax cash flow $ ('000)			- 164,937	- 311,306	144,420	118,599	113,834	135,368	129,256	132,686	139,129	151,907	99,839	27,696	15,673	- 6,803	- 2,548	22,977	10,648	- 46,309
Cumulative pre-tax cash flow $ ('000)			- 164,937	- 476,243	- 331,823	- 213,224	- 99,391	35,977	165,234	297,920	437,049	588,956	688,795	716,491	732,164	725,361	722,814	745,791	756,438	710,129
BC MINERAL TAX STAGE			-	-	2,891	2,865	2,329	2,710	2,761	2,656	15,857	19,748	12,979	3,601	2,038	-	-	1,757	1,378	-
INCOME TAXES (CANADA FEDERAL 15%, BC 11%) ('000)			-	-	-	-	4,576	25,090	27,809	28,027	27,715	31,081	18,355	3,757	2,161	-	-	1,120	1,632	-
TOTAL TAXES $ ('OOO)		$ -	-	-	2,891	2,865	6,906	27,799	30,570	30,683	43,572	50,829	31,334	7,358	4,199	-	-	2,877	3,010	-
Pre-tax cash flow $ ('000)			($164,937)	($311,306)	$144,420	$118,599	$113,834	$135,368	$129,256	$132,686	$139,129	$151,907	$99,839	$27,696	$15,673	($6,803)	($2,548)	$22,977	$10,648	($46,309)
Cumulative pre-tax cash flow $ ('000)			($164,937)	($476,243)	($331,823)	($213,224)	($99,391)	$35,977	$165,234	$297,920	$437,049	$588,956	$688,795	$716,491	$732,164	$725,361	$722,814	$745,791	$756,438	$710,129
After-tax cash flow $ ('000)		$0	($164,937)	($311,306)	$141,529	$115,734	$108,441	$107,524	$98,579	$101,871	$95,359	$100,951	$66,624	$19,579	$11,393	($4,734)	($1,665)	$16,536	$7,482	($33,818)
Cumulative After-tax cash flow $('000)			($164,937)	($476,243)	($334,714)	($218,980)	($110,539)	($3,015)	$95,563	$197,434	$292,793	$393,744	$460,368	$479,946	$491,339	$486,604	$484,940	$501,476	$508,957	$475,140
Discounted NCF 5% $ ('000)	Pre Tax	5%	($157,083)	($282,364)	$124,755	$97,572	$89,192	$101,014	$91,860	$89,807	$89,684	$93,258	$58,374	$15,422	$8,312	($3,436)	($1,225)	$10,526	$4,645	($19,242)
Discounted NCF 7% $ ('000)	Pre Tax	7%	($154,147)	($271,907)	$117,890	$90,479	$81,162	$90,201	$80,494	$77,225	$75,677	$77,222	$47,433	$12,297	$6,504	($2,638)	($923)	$7,783	$3,371	($13,701)
Discounted NCF 8% $ ('000)	Pre Tax	8%	($152,720)	($266,895)	$114,645	$87,174	$77,473	$85,305	$75,420	$71,686	$69,599	$70,362	$42,819	$10,999	$5,763	($2,316)	($803)	$6,707	$2,878	($11,589)
Discounted NCF 10% $ ('000)	Pre Tax	10%	($149,943)	($257,278)	$108,505	$81,005	$70,682	$76,412	$66,329	$61,899	$59,004	$58,567	$34,993	$8,825	$4,540	($1,791)	($610)	$5,000	$2,107	($8,329)
Discounted NCF 5% $ ('000)	After Tax	5%	($157,083)	($282,364)	$122,258	$95,215	$84,967	$80,236	$70,058	$68,950	$61,470	$61,975	$38,953	$10,902	$6,042	($2,391)	($801)	$7,575	$3,264	($14,052)
Discounted NCF 7% $ ('000)	After Tax	7%	($154,147)	($271,907)	$115,530	$88,293	$77,317	$71,648	$61,390	$59,290	$51,869	$51,318	$31,652	$8,693	$4,727	($1,836)	($603)	$5,601	$2,369	($10,005)
Discounted NCF 8% $ ('000)	After Tax	8%	($152,720)	($266,895)	$112,350	$85,068	$73,803	$67,758	$57,520	$55,038	$47,703	$46,760	$28,574	$7,775	$4,189	($1,612)	($525)	$4,827	$2,022	($8,463)
Discounted NCF 10% $ ('000)	After Tax	10%	($149,943)	($257,278)	$106,333	$79,048	$67,334	$60,694	$50,586	$47,523	$40,442	$38,921	$23,351	$6,238	$3,300	($1,247)	($398)	$3,599	$1,480	($6,082)

Pre-Tax Rate of Return	**21.1%**						
After Tax Rate of Return	**16.6%**		**NPV 0%**	**NPV5%**	**NPV 7%**	**NPV 8%**	**NPV 10%**
Pre-tax Initial Payback	**3.73**	years	$ 710,129	$ 411,070	$ 324,421	$ 286,507	$ 219,916
After Tax Initial Payback	**4.03**	years	$ 475,140	$ 255,174	$ 191,199	$ 163,173	$ 113,901
Operating Cost per Mill Feed Tonne	**$ 21.15**						

Appendix G – Design Basis

MINE DESIGN BASIS - Kwanika PEA February 2017

Revised: Feb 17, 2017 JHG

	Foreign Exchange Rate $US : $C		0.77 $US/$C		Updated Feb 21, 2017 - Tracey Meintjes	Estimate
FACTORS						
			31.10348 gm/oz		standard	
			2204.6 lb/tonne		standard	
METAL PRICES						
	Copper	$2.90 $US/lb.	3.766	$C/lb.	Updated Feb 21, 2017 - Tracey Meintjes	
	Gold	$1,270.00 $US/oz	53.028	$C/gm	Updated Feb 21, 2017 - Tracey Meintjes	
	Silver	$19.00 $US/oz	0.793	$C/gm	Updated Feb 21, 2017 - Tracey Meintjes	
	Molybdenum	$8.50 $US/lb.	11.039	$C/lb.	Updated Feb 21, 2017 - Tracey Meintjes	
TOPOGRAPHY						
MS-EP	SRK_Topography_Central, SRK_Topography_South					
OP and UG Mine Design	TOPO EXT (112011)					
Cutoff Grade						
MS-EP, Reserves	$11.30					
Cost Model	$11.02					

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MINE DESIGN BASIS - Kwanika PEA February 2017

PROCESS

	Central Zone	South Zone	Source	Approved
	91.0%	89.0%	Tracey Meintjes, Feb 21, 2017	
	75.0%	70.0%	Tracey Meintjes, Feb 21, 2017	
	75.0%	75.0%	Tracey Meintjes, Feb 21, 2017	

Process Costs ($/tonne Mill Feed)

Cost:	**$C/tonne Mill Feed** Case 1 - Small	
Process Cost	$9.00	Tracey Meintjes, Feb 21, 2017
G&A	$2.02	Graham Milne - Revised G&A workup, Feb 22, 2017
Water Treatment		Included in Process cost (TM, March 2017)
Tailing Construction		Included in Process cost (TM, March 2017)
Total	**$11.02**	

PRODUCTION TARGETS	Case 1 - Small	
AVG Metallurgical Process Daily thoughput	15,000	tpd
Metallurgical Process Production days	365	days
Annual Throughput	5,475,000	tpa
Available mine production days	360	days/yr

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Net Smelter Revenues and Prices for NSR17 — MINE DESIGN BASIS - Kwanika PEA February 2017

Description	Variable	Calculation	Values	Comments	Units	Source:	Approved:
Average Mill Feed Grades							
Gold	Au	Input	0.609		g/t		
Copper	Cu	Input	0.215		%		
Silver	Ag	Input	2.21		g/t		
Molybdenum	Mo	Input	51.9		ppm		
Process Recovery to Copper Concentrate							
Gold	CCAuRec	constant	70.0%		**%**	*uses south zone recoveries*	
Copper	CCCuRec	constant	89.0%		**%**	*uses south zone recoveries*	
Silver	CCAgRec	constant	75.0%		**%**	*uses south zone recoveries*	
Process Recovery to Gold Dore							
Gold	GDAuRec			**61.5**	**%**	*Daniel Sepulveda, Oct 7, 2011*	
Silver	GDAgRec			**18**	**%**	*Daniel Sepulveda, Oct 7, 2011*	
Concentrate Specs							
Copper grade	ConCu	constant (Head Grade 0.15 - 0.4% Cu)	25.0	24	%	*Daniel Sepulveda, Jan 21, 2013*	
Gold Grade	ConAu	=(CCAuRec / 100 x Au) / (Cu x (CCCuRec/100) / ConCu)	55.7		gpt		
Silver Grade	ConAg	=(CCAgRec / 100 x Ag) / (Cu x (CCCuRec/100) / ConCu	217		gpt		
Molybdenum	ConMo	constant	50.0		%		
Moisture Cu Con	cmoiCu	Input	9%		%		
Moisture Mo Con	cmoiMo	Input	5%		%		
Prices							
Gold Price	AUPRC	Input	$1,270		US$/oz	*email from Jim G on Feb 17*	
Copper Price	CUPRC	Input	$2.90		US$/lb	*email from Jim G on Feb 17*	
Silver Price	AGPRC	Input	$19.00		US$/oz	*email from Jim G on Feb 17*	
Molybdenum Price	MoPRC	Input	$8.49		US$/lb		
US Exchange rate	XRATE	Input	$0.77		US$/CDN$	*email from Jim G on Feb 17*	
Gold Price	AUCDN	=AUPRC/XRATE/gpoz	53.03		CDN$/g		
Copper Price	CUCDN	=CUPRC/XRATE	3.766		CDN$/lb		
Silver Price	AGCDN	=AGPRC/XRATE/gpoz	0.793		CDN$/g		
Molybdenum Price	MoCDN	=MoPRC/XRATE	11.03		CDN$/lb		

Conversions						
Pounds per tonne conversion	ppt	Constant	2204.62		lb/tonne	
Grams per ounce conversion	gpoz	Constant	31.10348		gr/oz	
MoS2 to Mo		Constant	0.599		Mo/MoS2	
Smelter Terms						
Copper Conc						
cu unit deductions	dedcu	Input	1.0%	1.0%	%	*Daniel Sepulveda, Oct 7, 2011*
au payable	payau	Input	98.0%	97.8%	%	*DS, Oct7 2011 / Tracey Meintjes, Feb 21, 2017*
ag payable	payag	Input	93%	90%	%	*Tracey Meintjes, Feb 21, 2018*
smelting	smelt	Input	100.0	**85.000**	US$/DMT	*Tracey 22 Nov 2016 - 2016 prices range from 98 to 105*
cu refining - base	refcua	Input	0.080	0.085	US$/lb	*Daniel Sepulveda, Oct 7, 2011*
price participation (cu floor)	prpart	Input	1.500	1.500	US$/lb	*Tracey 22 Nov 2016*
price participation % for amount above prpart	prpart%	Input	1.5%	1.5%	%	*Tracey 22 Nov 2016*
Price Participation Cap	prpartCAP	Input	4.0%	0.040	US$/lb	*Tracey 22 Nov 2016*
Price Participation	PP	MIN((CUPRC-prpart)*prpart%	0.021		US$/lb	
cu refining - with participation	refcu	refcua + PP	0.101		US$/lb	(copmpares well with 2016 Antofagasta/Jiangxi RC)
au refining	refau	Input	8.000	8.000	US$/oz	*Tracey 22 Nov 2016*
ag refining	refag	Input	0.600	0.600	US$/oz	*Tracey 22 Nov 2016*
Moly Conc						
Mo payable	paymo	Input	99.0%	99.5%	%	*Daniel Sepulveda, Oct 7, 2011*
Losses in handling and roasting unit deductions	dedMo	Input	0.50%	2.00%	%	*Tracey 22 Nov 2016*
Roasting	smelt	Input	2.00	1.50	US$/lb	*Tracey 22 Nov 2016*
Mo refining - base	refmoa	Input	0.000	0.000	US$/lb	*Daniel Sepulveda, Oct 7, 2011*
Dry Concentrate tonnes						
Copper Conc	DMTCu	= 1-cmoiCu	91%		%	
Moly Conc	DMTMo	= 1-cmoiMo	95%		%	
Net Copper Revenue per Tonne Copper Conc.						
Cu in Conc	NetCu	= DMTCu*ConCu*ppt	501.55		lb/WMT	
Net payable Cu in Concentrate	NPyCu	= DMTCu*(ConCu-dedcu)*ppt	481.49		lb/WMT	
Net payment Cu in Concentrate	PayCu	= NPyCu*CUCDN	$1,813.40		CDN$/WMT	
Refining Cu	CuRef	= NetCu*refcu/XRATE	$65.79		CDN$/WMT	
Net Revenue Copper	NRCu	= PayCu-CuRef	$1,747.61		CDN$/WMT	TCRC $142
Net Gold Revenue per Tonne Copper Conc.						
Au in Conc	NetAu	= DMTCu*ConAu	50.68		g/WMT	
Net payable Au in Concentrate	NPyAu	= payau*NetAu	49.67		g/WMT	
Net payment Au in Concentrate	PayAu	=NPyAu*AUCDN	$2,633.89		CDN$/WMT	
Refining Au	AuRef	= NetAu*refau/XRATE/gpoz	$16.59		CDN$/WMT	
Net Revenue Gold	NRAu	= PayAu-AuRef	$2,617.30		CDN$/WMT	
Net Silver Revenue per Tonne Copper Conc.						
Ag in Conc	NetAg	= DMT*ConAg	197.06		g/WMT	
Net payable Ag in Concentrate	NPyAg	= payag*NetAg	183.27		g/WMT	
Net payment Ag in Concentrate	PayAg	=NPyAg*AGCDN	$145.39		CDN$/WMT	
Refining Au	AgRef	= NetAg*refag/XRATE/gpoz	$4.59		CDN$/WMT	
Net Revenue Silver	NRAg	= PayAg-AgRef	$140.80		CDN$/WMT	
Net Revenue Total Copper Conc	TRev	= NRCu + NRAu + NRAg	$4,505.72		CDN$/WMT	
Proportion Copper	TRCu	= NRCu/TRev	38.79%		%	
Proportion Gold	TRAu	= NRAu/TRev	58.09%		%	
Proportion Silver	TRAg	= NRAg/TRev	3.12%		%	

Offsites, Freight, and Distribution Copper Conc								
Smelting	Smelt	=smelt*DMT/XRATE	$118.18		CDN$/WMT			
freight for trucking	ftruck	Input	$33.00	$32.94	CDN$/WMT	*Tracey 22 Nov 2016*		
freight for rail	frail	Input			CDN$/WMT	*Daniel Sepulveda, Oct 7, 2011*		
Stevedoring	fsteve	Input			US$/WMT	*Daniel Sepulveda, Oct 7, 2011*		
freight for ships	focean	Input	$68.00	$67.99	US$/WMT	*Tracey 22 Nov 2016*		
Other Offsite Costs (Losses, Ins,Sell,supv,Assay)	other	Input	$16.00	$0.00	CDN$/WMT	*Tracey 22 Nov 2016*		
Offsites, Frgt, Distr. Total	OFD	= Sum (Smelt : Other)	$255.49		CDN$/WMT			
Proportion Copper	OFDCu	=OFD*TRCu	$99.10		CDN$/WMT			
Proportion Gold	OFDAu	=OFD*TRAu	$148.41		CDN$/WMT			
Proportion Silver	OFDAg	=OFD*TRAg	$7.98		CDN$/WMT			
Net Moly Revenue per Tonne Moly Conc.								
Mo in Conc	NetMo	= DMT*ConMo*ppt	1047.19		lb/WMT			
Net payable Mo in Concentrate	NPyMo	= (paymo-dedMo)*NetMo	1031.49		lb/WMT			
Net payment Mo in Concentrate	paymo	=NPyMo*MoCDN	$11,373.14		CDN$/WMT			
Refining Mo	MoRef	= NetMo*refmo	$0.00		CDN$/WMT			
Net Revenue Moly	NRMo	= paymo-MoRef	$11,373.14		CDN$/WMT			
Offsites, Freight, and Distribution Moly Conc								
Roasting & Smelting	smelt	smelt*ppt/DMT/XRATE	$5,439.97		CDN$/WMT			
Trucking	ftruck	Input	$70.00	$66.96	CDN$/WMT	*Tracey 22 Nov 2016*		
Rail	frail	Input			CDN$/WMT	*Daniel Sepulveda, Oct 7, 2011*		
Ocean Freight	focean	Input	$93.00	$88.93	US$/WMT	*Tracey 22 Nov 2016*		
Other Offsite Costs (Losses, Ins,Sell,supv,Assay)	other	Input	$16.00	$50.00	CDN$/WMT	*Tracey 22 Nov 2016*		
Offsites, Frgt, Distr. Total	OFD	= Sum (Smelt : Other)	$5,646.75		CDN$/WMT			
Proportion Molybdenum	OFDMo	=OFD	$5,646.75		CDN$/WMT			
Gold Dore'								
Au Payable	auDpay	Input		99.6	%	*Daniel Sepulveda, Oct 7, 2011*		
au refining + transport	auDref	Input		2.00	US$/oz	*Daniel Sepulveda, Oct 7, 2011*		
	auDTr	Input			CDN$/Tonne			
Au Dore NSP	auDNsp	=AUCDN*AuDpay/100-AuDref/XRATE/gpoz-auDTr/1000	0.00		CDN$/g			
Ag Dore NSP	agDNsp		0.000		CDN$/g			
Au distribution to Dore	AuDR	Au in Dore / (Au in Dore + Au in Cu Conc.)	0%		%			
Ag Distribution to Dore	AgDR	Ag in Dore / (Ag in Dore + Ag in Cu Conc.)	0%		%			
Net Smelter Return per Tonne Cu Conc. (Wet)								
NSR Copper	NSRCu	= NRCu - OFDCu	$1,648.52		CDN$/WMT			
NSR Gold	NSRAu	= NRAu - OFDAu	$2,468.89		CDN$/WMT			
NSR Silver	NSRAg	= NRAg - OFDAg	$132.82		CDN$/WMT			
NSR Total	NSR	= NSRCu + NSRAu + NSRAg + NSGMo	$4,250.22		CDN$/WMT			
Net Smelter Return per Tonne Mo Conc. (Wet)								
NSR Moly	NSRMo	= NRMo - OFDMo	$5,726.39		CDN$/WMT			
Net Smelter Price (to Mine Gate) for NSR17								
Copper	NSPCu	= NSRCu/NetCu	**$3.29**		CDN$/lb		3.77	0.4794
Gold	NSPAu	= NSRAu/NetAu x AuCR/(AuDR + AuCR) + auDNsp x AuD	**$48.71**		CDN$/g		53.03	4.3161
Silver	NSPAg	= NSRAg/NetAg	**$0.67**		CDN$/g		0.79	0.1193
Moly	NSPMo	= NSRMo/NetMo	**$5.47**		CDN$/lb		11.04	5.5706

MINE DESIGN BASIS - Kwanika PEA February 2017

ECONOMIC PIT LIMITS FOR LG

Process, G&A , Tailings Treatment, Site Services and Water Treatment Costs

Mining Cost	**2.76**	$C/t total material
Processing and GA Cost	**11.30**	$C/t mill feed

LG CASES

LG Net Smelter Prices (At the Mine Gate)

CASE	Net Price for Mine, Plant, & O/H			
	Copper	**Gold**	**Silver**	**Moly**
	$C/lb	**$C/g**	**$C/g**	**$C/lb**
100.0%	$3.233	$48.979	$0.652	$5.693
Recovery used for LG	89%	70%	75%	

Base Case Metal Prices (for NSP Calcs)

Copper	Gold	Silver	Moly
$US/lb	**$US/oz**	**$US/oz**	**$US/lb**
2.75	1230	17.75	8.49

MINE DESIGN BASIS - Kwanika PEA February 2017

PIT SLOPE ANGLES			Source	Approved
Bench Face Angle (degrees)	70		PEA 2013	
Overal Angle (degrees)	40	Overall design angle from PEA 2013 pits (includes ramps		
Final Bench Height (m)	20m		PEA 2013	
Minimum Catch Bench Width (m)	8m		BC Mines Regs	

SERENGETI RESOURCES INC. posco DAEWOO Moose Mountain Technical Services

MINE DESIGN BASIS - Kwanika PEA February 2017

RESERVE/RESOURCE ESTIMATION				Source	Approved
Pit Delineated Reserve Calculation - Using MineSight PITRES Routine					
Mining Recovery (Open Pit)		95%		assumed	by client
Whole Block Dilution (Open Pit)		2%		assumed	by client
Mining Recovery (UG)		100%		assumed	
Whole Block Dilution (UG)		0%		assumed	
Default SG (Ore):					
Central		2.75	t/m^3	from 3DBM model	
South		2.75	t/m^3	from 3DBM model	
Default SG (Waste):					
Central		2.70	t/m^3	from 3DBM model	
South		2.70	t/m^3	from 3DBM model	
Ore Cut off					
Break-Even COG (Covers Milling plus G&A)	$	11.30	$C/t ore	see Process tab	
Low Grade	$	15.00	$C/t ore	arbitrary grade bins	
Mid-Grade	$	20.00	$C/t ore	"	
High Grade	$	30.00	$C/t ore	"	

NSR (NSR17)

NSR (CDN$/t recovered)=dolval Cu + dolval Au + dolval Ag

Dolval Cu = Cu % / 100 x (NSPCu) x 2204.62 lb/tonne x (RecCu)

Dolval Au = Au g / tonne x (NSPAu) x (RecAu)

Dolval Ag = Ag g / tonne x (NSPAg) x (RecAg)

Cu Equivalent (CUQ17)

CUQ17 = (MetVal Cu + MetVal Au + MetVal Ag) / (CuPrice) / 2204.62 *100

MetVal Cu = Cu % / 100 x (CuPrice) x 2204.62 lb/tonne

MetVal Au = Au g / tonne x (AuPrice)

MetVal Ag = Ag g / tonne x (AgPrice)

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MINE DESIGN BASIS - Kwanika PEA February 2017

all from PEA 2013

PIT DESIGN

Equipment Fleet
Major Mining Fleet

Waste & Ore Shovels PC2500/EX3000 (15ktpd)
Waste & Ore Trucks Cat785 (15ktpd)
Drills

Equipment Selection
Bunching Maximum (dependant on fleet match)
Shift eff for tkph calc
TKPH limit
Fleet Matching factor
Minimum 3 passes to load a truck.

Max pit ramp slope	8%		For winter conditions
Min Inside Switchback Radius	5	m	as per JHG
Waste Dump Angle of Repose	37	degrees	Blasted rock
Largest Vehicle Overall Width	3.5	m	CAT 785 Spec
Maximum Tire Height (33.00-R51)	3.1	m	
Minimum Haul road outside berm height	2.3	m	Mines Act based on tire height
Minimum Shoulder / Berm Width	6.8	m	
External road ditch			
Double lane highwall haul road allowance	17.3	m	BC Mines Act
Double lane external haul road allowance	24.1	m	BC Mines Act
Single lane highwall haul road allowance	13.8	m	BC Mines Act
Single lane external haul road allowance	20.6	m	BC Mines Act
Runaway lanes or retardation barriers where conditions/risk warrant on roadways where the grade exceeds 5%			BC Mines Act
Speed Limits Around Corners		kph	CAT Handbook

Last bench will not need haul road. Temporary internal ramps will be used
Last 2 haul benches have single lane haul roads
Minimum pit base width is shovel operating width.

For Scoping Level pit designs...	# of lanes	Width	Grade
Main Ramp	2	17.3	8%
Last 2 benches of Ramp	1	13.8	8%
Last 2 benches of Pit Phase	None		

Assume Minimum Mining Width 30m *based on approx. minimum turning radius of truck*

Road Design (External to pit, Non-Haul Road) *added March 3, 2010*

	Lanes	Width	Max Grade	Description
Type I	2	20	10%	Well traveled, main access roads: all vehicles, can accomodate PRE-PRODUCTION haul truck access
Type II	1	10	10%	Light traveled, main access roads: light trucks, tractor trailers, and service vehicles
Type III	1	12	15%	Infrastructure service roads: pick-up trucks and service/parts vehicles, tractor trailers may require chains
Type IV	1	6	15%	Remote facility access road: pick-up trucks
Type V	1	10	15%	Pioneering road: limited use, construction equipment

Berms

Waste Dump Engineering

Natural angle of repose	37^{o}
Maximum free dumping height	300m
Dumped waste swell factor	1.2
Maximum waste rock crest elevation	
Height of waste rock pile	

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MINE DESIGN BASIS - Kwanika PEA February 2017

full-time OP 5 years, plus 5 years half UG, half OP

5475 kt/year throughput

Updated Jan 16, 2017 by GM

SOURCE: KSM 2016 DBM Labour Rates

General and Administrative Costs										
G&A	**# Positions**	**Base Salary**	**Site Premium**	**Prod/Safety Bonus**	**Surface OT Modifier**	**burden**		**total compensation**	**annual cost**	**$/tonne milled**
Mine Manager	1	107000	10%	35%	5%	30%		$ 192,600	$ 192,600	
Administrative Assistant	1	49000	10%	35%	5%	30%		$ 88,200	$ 88,200	
Finance and Accounting (site)	1	89000	10%	35%	5%	30%		$ 160,200	$ 160,200	
Human Resources	1	89000	10%	35%	5%	30%		$ 160,200	$ 160,200	
Information Technology	1	89000	10%	35%	5%	30%		$ 160,200	$ 160,200	
Purchasing	2	68000	10%	35%	5%	30%		$ 122,400	$ 244,800	
Warehouse	2	49000	10%	35%	5%	30%		$ 88,200	$ 176,400	
Environmental and Sustainability Offi	1	89000	10%	35%	5%	30%		$ 160,200	$ 160,200	
Environmental Technician	1	68000	10%	35%	5%	30%		$ 122,400	$ 122,400	
Security-First Aid	4	49000	10%	35%	5%	30%		$ 88,200	$ 352,800	
									$ 1,818,000	**$ 0.33**
Site Costs										
Camp and Catering Costs	**164**	$100 per man day						$5,986,000	$ 5,986,000	**$ 1.09**
Open Pit**	0	** Most of the mine schedule is either OP or UG, not both concurrently. Therefore, larger UG number used for camp cost calculations								
UG	121									
GME	11									
Process	32									
Employee Transport	**164**	$8,667 per person per year						$8,667	$ 1,421,388	**$ 0.26**
Air Strip Op and Maintenance		100,000 per year							$ 100,000	**$ 0.02**
Site Office Costs		200,000 per year							$ 200,000	**$ 0.04**
Site Maintenance Costs		200,000 per year							$ 200,000	**$ 0.04**
									$ 7,707,388	**$ 1.41**
Contingency								20%	$ 1,541,478	**$ 0.28**
								Total G&A	**$ 11,066,866**	**$ 2.02**

	Open Pit	Processing	UG Mine	UG Contractor	GME (Above)
TOTAL PERSONNEL ON-SITE AS PER TABLES BELOW:	68	32	121	70	11

costs below are included in the Mining and Processing unit costs

SALARIED STAFF - Open Pit PER SHIFT	
Mining Operations	
Operations Superintendant	1
Shift Foreman	1
Trainer / Safety Officer	1
Administrative Assistant	1
Mine Planning	
Senior Mining Enginer	1
Mining Engineer	1
Surveyor	1
Drill and Blast Engineer	1
Senior Geologist	1
Pit Geologist/Samplers	2
Mine Maintenance	
Maintenance Superintendant	1
Maintenance Shift Foreman	1
Mechanical Foreman	1
No. Open Pit Positions	**14**

PROCESSING Per Shift	# Positions
Plant Superintendant	1
Plant Shift Foreman	2
Senior Metallurgist	1
Metallurgist	1
Assayer	2
Lab Technicians	4
Plant Maintenance Foremai	1
Operations Labour	10
Maintenance Labour	10
# Processing Positions	**32**

Open Pit Hourly Per Shift	# Positions
Drill Operator	4
Blasters	2
Shovel Operator	4
Haul Truck Driver	14
Grader Operator	2
Excavator Operator	2
Loader Operator	4
Track Dozer Operator	6
Scraper Operator	0
Crusher Operator	0
Water Truck Operator	2
Fuel Truck Operator	2
Electrician	1
HD Mechanic	6
LD Mechanic	1
Machinist	0
Crane Operator	0
Welder	2
Tireman	0
Labourer	2
# Open Pit Hourly	**54**

OPEN PIT FLEET LIST (etimated, GM	Equip Qty	# Shifts	# Operators
Drills	2	4	8
Blasters	2		8
Shovels	2		8
Trucks	7		28
Dozers	3		12
FEL	2		8
Excavator	1		4
Grader	1		4
Water Truck	1		4
Low Boy	1		4
Pickups	4		4
Crew Van	1		4
SHIFT QTY	**27**		**96**
Mechanics	3		12
Welders	1		4
Electricians	1		4
Labourer	2		8
	7		**28**
	34		**124**

ON-SITE - INCL DAYSHIFT/NIGHTSHIFT

UNDERGROUND MANPOWER - Owner
(from "Kwanika UG Block Cave Mark2_07Feb2017_MP"

	On Staff	On Site
Mine Operations		
Manager of Mining	1	1
Mine Superintendent	2	1
Mine Supervisor	4	2
Mine Trainer	2	1
Clerk	2	1
Maintenance Superintendent	2	1
Maintenance Supervisors	2	1
Maintenance Planner	1	1
Mechanics (Mobile)	40	20
Mechanics (Stationary)	8	4
Mechanics Helpers	12	6
Shop Labourers	4	2
Fuel/Lube Guys	4	2
Chief Electrician	1	1
Electricians	8	4
Subtotal:	**93**	**48**

Operators (Mine)

In House Mine Opei	On Staff	On Site
LHD Operators	10	5
Stationary Rockbrea	4	2
Mobile Rockerbreak	10	5
Blockholer Operatoi	8	4
Truck Drivers	24	12
LH Drillers	8	4
LH Blasters	8	4
Delivery Crew	8	4
Grader Operator	4	2
Scotcrete Crew	12	6
Labours	12	6
Subtotal:	**108**	**54**

Technical Services (Mine)

	On Staff	On Site
Superintendent	1	1
Chief Engineer	1	1
Long Range Planr	1	1
Short Range Plan	2	1
Services	1	1
Surveyors	8	4
Cad People		1
Chief Geologist	1	1
Senior Geologist	1	1
Beat Geologist	4	2
Samplers	8	4
Cad People	2	1
Subtotal:	**30**	**19**

UG MANPOWER (Contractor)				Hourly Personnel (Contractor)		
	On Staff	On Site			On Staff	On Site
Project Manager	1	1		Underground Forem	4	2
General Superintendent	1	1		Underground Mine I	2	2
Assistant Project Manager	1	1		Underground Miner	32	16
Senior Engineer	2	1		Underground Miner II		
Planning Enginner	4	2		Truck Drivers	20	10
Draftsman	2	1		Electrician	4	2
Quality Manager	2	1				
Safety Advisor	2	1				
Environmental Manager	1	1				
Field Technician	4	2				
Office Manager	2	1				
Payroll Clerk	2	1				
Secretary	2	1				
Janitor	4	2				
Mechanical Superintendent	1	1				
Mechanical Foreman	2	1				
Mechanics	18	9				
Fuel Man	4	2				
Buyer	2	1				
Chief Surveyor	2	1				
Surveyor	12	6				
Subtotal:	**71**	**38**		**Subtotal:**	**62**	**32**
TOTAL ON-SITE UG CONTRACTOR			**70**			

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MINE DESIGN BASIS - Kwanika PEA February 2017

Underground Basic Design Parmeters

			Source	Approved
Preliminary Operating Design Parameters				
Block caving	$/tonne		Kwanika Deposit - Preliminary Caveability Assessment	
Caving 7.00	$7.00		AMEC 2013	
Preconditioning	$1.00			
Undergound loss and dilution	0	%		
Sublevel Open Stoping with backfill	$25.00		MMTS estimate from data base (MikeP)	

Capital Cost design Perameters

Drawbell and Development Cost

Undercut Drifts 4.5m W x 4.5m H	$	6,612	$/m
Precon Drifts 4.0m W x 4.0m H	$	5,695	$/m
Ext. Drives 5.0m W x 4.5m H	$	7,097	$/m
DPs 4.5m W x 4.5m H	$	6,612	$/m
Fresh Air	$	8,197	$/m
Return Air	$	8,197	$/m
Truck Loading	$	8,768	$/m
Workshops/Sump	$	9,276	$/m

Raise Bores

Dia. (m)	$/m
2.5	3,500
3.0	4,000
3.5	4,500
4.5	6,500

Ramp

Grade		17	%
Access Ramp (single heading)	$	9,697	$/m
Access Ramp (double heading)	$	8,242	$/m

Block Cave Parameters

Block Cave Grade Shell	22	$ NSR
Bottom 3 benches (30m) are derated due to development		
Material Recovery within bottom 30ı	52	%

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MINE DESIGN BASIS - Kwanika PEA February 2017

Dam Details		Starter Dam Details	
Dam + Buttress Slope	3H:1V	Dam	2H:1V
Core Width	10 m	Core Width	10 m
Dam Capacity	full	Dam Capacity	6 months
Dam Freeboard	5 m	Dam Freeboard	5 m
Maximum height	80 m	Maximum height	25 m
Dam Earthworks	83 Mt	Dam Earthworks	9 Mt
Dam Core	5 Mt	Dam Core	1 Mt
Dam Length	3,800 m		

Dam Earthworks (Buttress) bulk density	2.25 t/m3
Tailings dry density	1.3 t/m3

Basic template used for costing	Rock fill buttress (Rock Storage Facility)