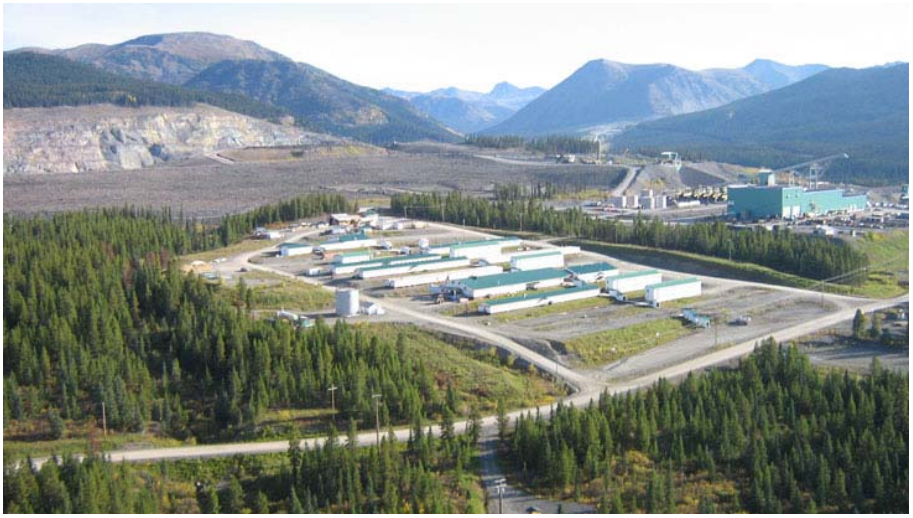
Technical Report for the Kemess Underground Project and Kemess East Resource Estimate, British Columbia, Canada

Prepared for

AuRico Metals Inc.



Prepared by

SRK Consulting (Canada) Inc.
2CA046-004
May 2016



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Project No: 2CA046-004

Effective date: 29 February, 2016
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Executive Summary

Introduction

The Kemess Underground Project is 100% owned by AuRico Metals Inc., with AuRico holding the surface rights to the property through its mineral claims and mining leases.

The Project is located in a mountainous area of north-central British Columbia, Canada approximately 250 km north of Smithers and 430 km northwest of Prince George. ES Figure 1 shows the location of the Kemess Underground Project.

The property is host to the former Kemess South (KS) Mine, the Kemess Underground (KUG) deposit, and the Kemess East (KE) deposit. The KUG and KE deposits are the subject of this Technical Report update. For the KUG deposit, the previous (2013) Feasibility Study has been updated, and for the KE deposit, the previous (2015) resource estimate has been updated.

The KS Mine comprised a large open pit mine feeding gold-copper ore to a 52,000 t/d processing plant. During the life of the KS mine, approximately 3.0 million ounces of gold and 750 million pounds of copper were recovered from 218 Mt of ore. Open pit mining and processing ceased in March 2011, on depletion of the mineral reserves. The processing plant and other facilities and equipment that are required to support an underground mining operation at the KUG deposit are currently under care and maintenance by AuRico site personnel. This plant, facilities and equipment are located at an elevation of approximately 1,200 m above mean sea-level (AMSL) and is currently accessible year-round by aircraft and during the summer via an access road from Mackenzie, BC.

The KUG deposit lies approximately 6.5 km to the north of the existing KS processing plant and other infrastructure, while the KE deposit lies approximately 1.0 km east of the KUG deposit. The KUG deposit is located beneath two north facing alpine cirques with ground surface elevations ranging from 1,500 m to 2,000 m AMSL, all above the tree line. Road access to the area overlying the deposit is currently limited to summer months only.

Below the tree line, broad, open, drift, and moraine covered valleys characterize the area. These areas are moderately vegetated with spruce-willow-birch forest, while poorly drained areas form peat bogs populated by alder brush, willow, and stunted spruce trees.

The climate is generally moderate, although snow can occur during any month. Temperatures range from -35°C to 30°C and average annual precipitation amounts to 890 mm. Extreme weather conditions are possible at the higher elevations.

Existing on-site infrastructure includes offices, warehouse, laydowns, maintenance facilities, a 300-person accommodation camp footprint, ore processing plant, crushed ore stockpile and reclaim, access and service roads, airstrip, explosives magazines, tailings storage facilities, and electrical sub-station.

The Kemess South Mine operated on a fly-in fly-out (FIFO) basis with the majority of employees working a cycle of two-weeks-on and two-weeks-off, commuting via charter aircraft from regional

centres. The airstrip is approximately 1,600 m in length and can accommodate most short take-off and landing (STOL) aircraft types. Hercules aircraft have occasionally used the airstrip for heavy lifts.

A company-owned, 380 km power line originating in Mackenzie, provides power to the mine site via the BC Hydro grid.

Waste rock and tailings from the mining and processing of KUG ore is planned for deposition in the KS open pit (as the KUG tailings storage facility), with a 25 m high dam required at the east end of the KUG TSF to facilitate all storage requirements. Process water will be available from the KS open pit for processing of KUG ore.

Potable water is currently sourced from permitted wells and treated through an on-site water treatment plant.

AuRico currently maintains all site facilities and infrastructure on a care and maintenance basis, pending decisions regarding environmental certification and permitting of the KUG project.

The mineralized zone is approximately 200 m to 550 m below surface. The lateral extents (or footprint) of the orebody, as outlined by this study, is approximately 570 m east-west and 90 m to 300 m north-south.



ES Figure 1: Kemess Property Location 1

(Source: AuRico, 2016)

Responsibility

This report has been prepared by SRK Consulting (Canada) Inc. under the direction of AuRico as the study manager, with primary contributions from AuRico, AMEC Foster Wheeler, BioteQ Environmental Technologies, Conveyor Dynamics, ERM Consultants Canada, Exen Consulting Services, KWM Consulting, Mine Ventilation Services, and Struthers Technical Solutions. A complete list of contributors is included in Section 1.

SRK does not accept liability for the statements, findings, and opinions expressed in the portions of the reports authored by other contributors.

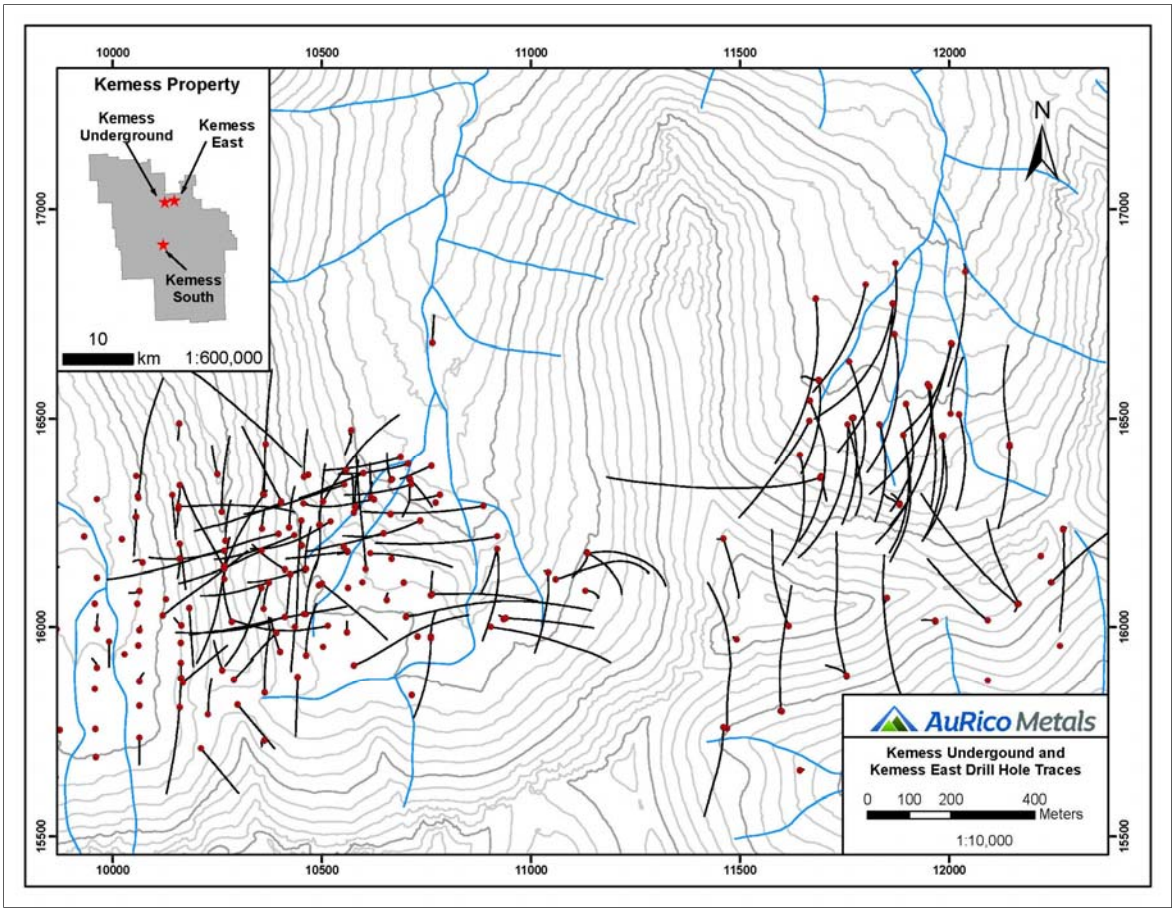
A list of units and abbreviations is presented in Section 26 at the end of this report.

Geology and Mineralization

The KUG deposit is a copper-gold-silver porphyry deposit and is typical of calc-alkaline porphyry copper-gold deposits in the western cordillera. The deposit has a low-grade ore zone at a depth of 150 m below the surface on its western flank and a higher grade zone 300 m to 550 m below surface on the eastern side, which forms the KUG project. KUG is hosted by potassic altered Takla Group volcanic rocks and Black Lake plutonic rocks. The deposit is centered on a mineralized porphyritic monzodiorite/diorite pluton and associated WSW trending dykes, which extend to the southwest. Higher grade copper-gold mineralization is characterized by secondary biotite alteration in the volcanic and the eastern plutonic host rocks.

The KE deposit is a copper-gold-silver-molybdenum porphyry deposit and is typical of calc-alkaline porphyry copper-gold deposits in the western cordillera. The deposit is deeply buried and mineralization starts at an average depth of 900 m below surface and extends to 1,500 m below surface. Unlike KUG, there is no siginificant low grade mineralization associated with KE. At KE, there is reasonable continuity of mineralization within the deposit. KE is mostly hosted by potassic altered Black Lake plutonic rocks. In the eastern portion of the deposit, weak mineralization is hosted within potassic altered Takla Volcanics, but still largely within the Black Lake plutonic rocks

ES Figure 2 below displays the drillholes for both the KUG (west) and KE (east) deposits.



ES Figure 2: Kemess Underground and Kemess East Drillhole locations

(Source: AuRico, 2016)

Mineral Resource Estimates

In 2015–2016, SRK re-visited the resource estimates for the KUG and KE deposits. Using updated and audited drillhole databases; the geology models for both deposits were rebuilt, all statistical analyses were updated, grades were re-estimated based on the updated geological domain interpretations, and the resources were reclassified. This work has resulted in two updated mineral resource statements for the Kemess property.

The Mineral Resources are reported in accordance with CSA, NI 43-101, and have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. The resource estimates was carried out by Chad Yuhasz, PGeo, APEGBC, Senior Consultant (Resource Geology), SRK Consulting. Mr. Yuhasz has over 13 years of operational and consulting experience in the minerals industry, specifically in mineral resource estimation, production geology, feasibility studies, and economic evaluations. Mr. Yuhasz is a Certified Professional Geologist and a Qualified Person as defined by international reporting codes.

Kemess Underground

The effective date of this Mineral Resource estimate is 29 February, 2016 and is based on all data available. The Mineral Resource statement for KUG is presented in ES Table 1. Mineral Resources are reported inclusive of Mineral Reserves, and are constrained to a reasonable prospects for economic extraction solid, with all Leach Zone and Hazelton material excluded from this statement.

ES Table 1: Kemess Underground Mineral Resource Statement, KUG Gold-Copper-Silver Deposit, Northwest British Columbia, Canada, 29 February, 2016

Resource Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Contained Metal		
					Copper (000's lbs)	Gold (000's oz)	Silver (000's oz)
Measured							
Indicated	246,400	0.22	0.42	1.75	1,195,300	3,328	13,866
Measured + Indicated	**246,400**	**0.22**	**0.42**	**1.75**	**1,195,300**	**3,328**	**13,866**
Inferred	21,600	0.22	0.40	1.70	104,700	277	1,179

Notes:

1. Mineral Resources are inclusive of Mineral Reserves. Due to the complexity of cave mine scheduling, SRK is of the opinion that reporting resources inclusive of the reserves is a more accurate approach for reporting.
2. Resources stated are contained within a zone representing “reasonable prospects for economic extraction” located above C$15.00/t NSR cut-off. A variable specific gravity value was assigned by domains for all model blocks.
3. NSR calculation is based on assumed copper, gold and silver prices of US$3.20/lb, US$1,275/oz and US$21.00/oz, respectively.
4. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.
5. Contained metals are in situ and undiluted, and do not include metallurgical recovery losses.

The “reasonable prospects for eventual economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off value, taking into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considers that portions of the KUG resource are amenable for underground extraction via panel cave mining development.

The 2016 Mineral Resource statement has been prepared by identifying the portion of the resource model that could potentially be mined using panel cave mining methods, treating indicated and inferred blocks as potentially economic. The study identified a volume potentially suitable for block/panel cave mining at a Net Smelter Return (NSR) cut-off value of C$15.00/t.

The cost parameters were selected based on experience and benchmarking against similar projects. The reader is cautioned that the results from the cave optimization are used solely for the purpose of testing the "reasonable prospects for eventual economic extraction" by a panel cave and do not alone represent an attempt to estimate Mineral Reserves. The commodity prices (ES Table 2) are used to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off NSR value.

ES Table 2: Kemess Underground Commodity Prices, and Recovery Assumptions, used to Calculate NSR Values and Reasonable Cut-off

Commodity	Value	Unit
Copper	3.20	US$ per pound
Gold	1,275	US$ per ounce
Silver	21.00	US$ per ounce
Copper Recovery	91%	Assumed Recovery
Gold Recovery	72%	Assumed Recovery
Silver Recovery	65%	Assumed Recovery

Initial estimates for underground panel cave mining costs, combined with processing and general and administrative (G&A) costs consistent with the size of operation envisioned for the KUG project, indicate that a mine could be economically viable with a NSR cut-off in the range of C$14.00/t to C$16.00/t ore.

SRK considers that the blocks located within the conceptual panel cave show "reasonable prospects for eventual economic extraction" and can be reported as a Mineral Resource.

Kemess East

The effective date of this Mineral Resource estimate is 29 February, 2016 and is based on all data available. The Mineral Resource statement for KE is presented in ES Table 3. Mineral Resources are reported constrained to a reasonable prospects for economic extraction solid.

ES Table 3: Kemess East Mineral Resource Statement, KE Copper-Gold-Silver-Molybdenum Deposit, Northwest British Columbia, Canada, 29 February, 2016

Resource Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Moly Grade (% Mo)	Contained Metal			
						Copper (000's lbs)	Gold (000's oz)	Silver (000's oz)	Moly (000's lbs)
Measured									
Indicated	39,270	0.40	0.50	1.99	0.008	347,550	630	2,510	7,050
Measured + Indicated	**39,270**	**0.40**	**0.50**	**1.99**	**0.008**	**347,550**	**630**	**2,510**	**7,050**
Inferred	109,670	0.37	0.38	1.99	0.009	890,540	1,340	7,010	20,860

Notes:

1. Resources stated are contained within a reasonable prospects for economic extraction solid above C$17.00/t NSR cut-off. A variable specific gravity value was assigned by domains for all model blocks.
2. NSR calculation is based on assumed copper, gold and silver prices of US$3.20/lb, US$1,275/oz and US$21.00/oz, respectively. Molybdenum was not assigned any value in the NSR calculation.
3. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.
4. Contained metals are in situ and undiluted, and do not include metallurgical recovery losses.
5. 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 Resources estimated will be converted into Mineral Reserves.

The “reasonable prospects for eventual economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off value, taking into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considers that portions of the KE resource are amenable for underground extraction via panel cave mining development.

The 2016 Mineral Resource statement has been prepared by identifying the portion of the resource model that could potentially be mined using panel cave mining methods, treating indicated and inferred blocks as potentially economic. The study identified a volume potentially suitable for block/panel cave mining at an NSR cut-off value of C$17.00/t ore.

The cost parameters were selected based on experience and benchmarking against the KUG Project. The reader is cautioned that the results from the cave optimization are used solely for the purpose of testing the “reasonable prospects for economic extraction” by a panel cave and do not represent an attempt to estimate Mineral Reserves. The commodity prices (ES Table 4) are used to assist in the preparation of a Mineral Resource estimate and to select an appropriate resource reporting cut-off NSR value. Although molybdenum was modeled, it was not assigned value in the NSR calculation due to lack of metallurgical testing.

ES Table 4: Kemess East Commodity Prices, and Recovery Assumptions, used to Calculate NSR Values and Reasonable Cut-off

Commodity	Value	Unit
Copper	3.20	US$ per pound
Gold	1,275	US$ per ounce
Silver	21.00	US$ per ounce
Molybdenum	8.36	US$ per pound (Not used)
Copper Recovery	91%	Assumed Recovery
Gold Recovery	72%	Assumed Recovery
Silver Recovery	65%	Assumed Recovery
Molybdenum Recovery	NA	Not Applicable

SRK considers that the blocks located within the conceptual panel cave show "reasonable prospects for eventual economic extraction" and can be reported as a Mineral Resource.

Mineral Reserve Estimate

The Mineral Reserve estimate has been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM Standards on Mineral Resources and Mineral Reserves and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

The Mineral Reserve estimate was developed from the resource model "SRK_KUG_FS2015_ENG.bmf" which was prepared by SRK Consulting in September 2015. The Mineral Reserve statement for the KUG project is presented in ES Table 5.

ES Table 5: Mineral Reserve Statement, Kemess Underground Gold-Copper-Silver Deposit, Northwest British Columbia, Canada, 29 February, 2016

Reserve Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Contained Metal		
					Copper (000's lb)	Gold (000's oz)	Silver (000's oz)
Proven							
Probable	107,381	0.27	0.54	1.99	629,595	1,868	6,878
Proven and Probable	**107,381**	**0.27**	**0.54**	**1.99**	**629,595**	**1,868**	**6,878**

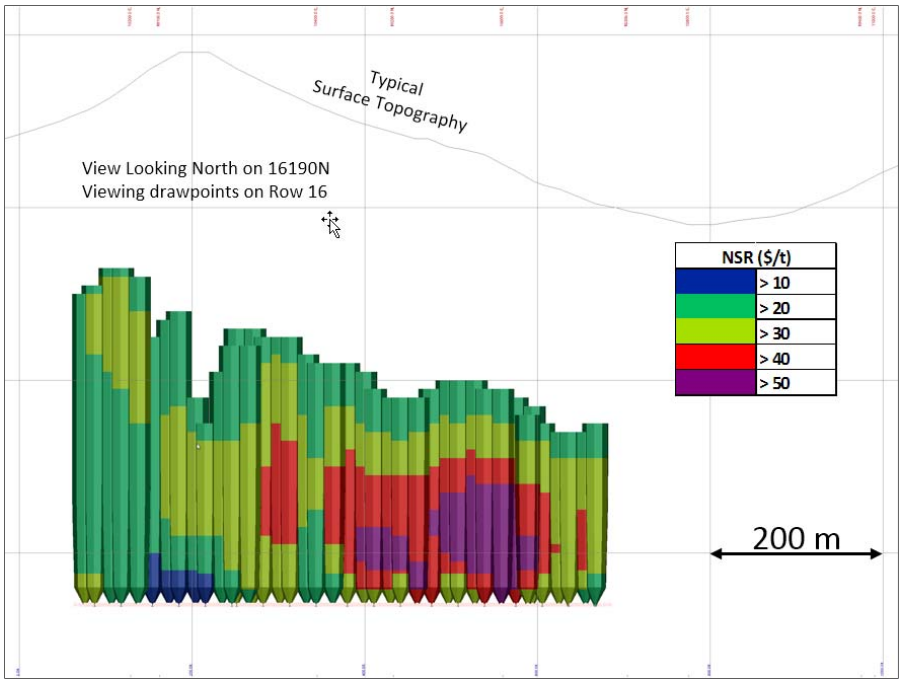
Notes:

1. Estimated at US$2.50/lb Cu, US$1,200/oz Au and US$16.00./oz Ag, using an exchange rate of 0.75 US$:C$, cut-off NSR Value of C$15.30/t and a shut-off NSR Value of C$17.30/t of ore. Metallurgical recoveries and other parameters are shown in Table 14.2.
2. Mineral Reserve tonnage and mined metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

Mining

The opportunity to exploit the KUG gold-copper-silver deposit by underground mining methods was evaluated by a preliminary economic analysis (PEA) completed by AMC Mining Consultants in July 2011 (AMC, 2011). This PEA considered longhole open stoping, sub-level caving, and block cave mining methods, with the block cave approach recommended due to superior economics. As a result, AuRico (then Northgate Minerals) elected to conduct a Feasibility Study (FS) for the KUG deposit based on block caving. The initial FS was completed in 2013 (SRK, 2013).

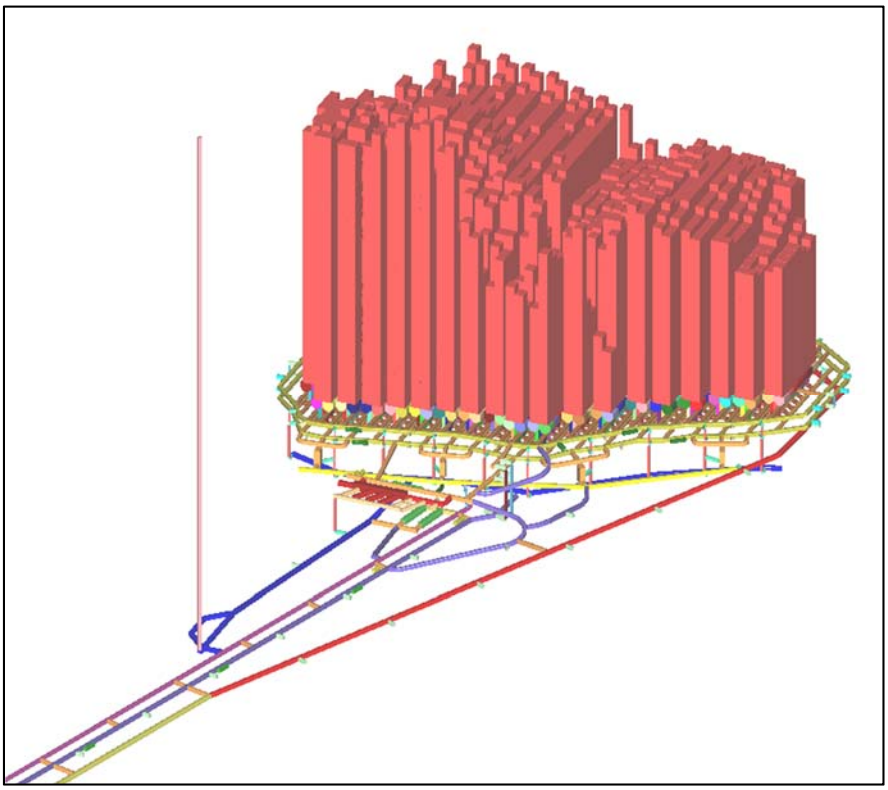
The KUG mine will be located approximately 6.5 km north of the existing KS site. This gold-copper-silver porphyry deposit hosts a probable Mineral Reserve of 107.3 Mt within a selected PCBC™ cave volume that has footprint dimensions of approximately 570 m east-west and 90 m to 300 m north-south. Mineralization extends from approximately 200 m to 550 m below surface. The typical range of column heights and NSR values (in C$ per tonne) is shown in ES Figure 3.



ES Figure 3: Typical range of panel cave column heights

(Source: SRK, 2016)

ES Figure 4 presents an overall view of the KUG mine layout.



ES Figure 4: Isometric view of Panel Cave (Looking northwest)

(Source: SRK, 2016)

The KUG panel cave mine was designed using Gemcom's PCBC™ software. The planned production schedule calls for 107.3 Mt of ore mined at a head grade of 0.27% Cu and 0.54 g/t Au and 1.99 g/t Ag.

Geotechnical information was reviewed; cavability and fragmentation parameters established based on empirical observations and Itasca Consulting Group Inc.'s (Itasca) numerical modeling, with this work carried out as part of the 2013 KUG FS (SRK, 2013).

Triple declines will be developed comprising access, intake air and conveyor declines. The access decline will provide access for personnel, equipment, and materials/consumables.

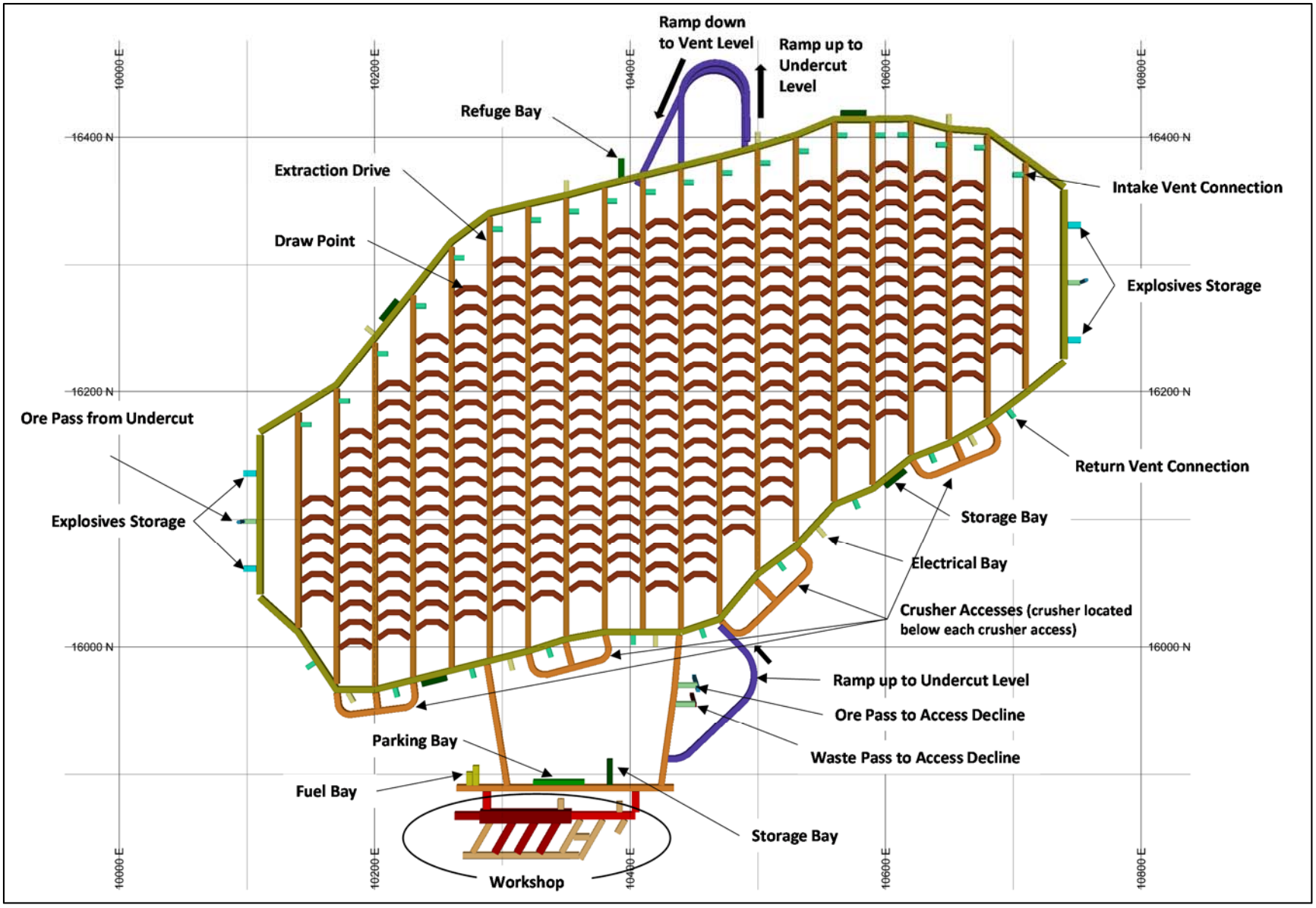
The final design establishes a single extraction level (ES Figure 5) that includes 582 drawpoints (291 drawbells). The cave will be initiated in the highest grade ore in the northeast of the orebody and progress to the southwest over the life of the mine.

Ore will be recovered on the 1,140 mRL (elevation) extraction level using 8.6 m^3 manually-operated diesel load-haul-dump (LHD) machines. They will deliver ore to four single-toggle, jaw crushers located on the south side of the caving footprint. Following crushing, ore will be transferred to either one of two transfer conveyors for subsequent transfer to a 42 inch (1,067 mm) wide conveyor belt that will transport ore to surface.

The 3.2 km long underground conveyor will rise 305 m vertically and transfer ore to another 4.9 km surface conveyor that will drop 98 m vertically. A total of 9 Mt per year (Mt/y) of ore will be discharged onto a stockpile located at the existing Kemess South process plant site.

Infrastructure required for the mine will include a 6.0 m wide x 5.0 m high intake air decline and 5 m diameter exhaust air raise and associated fan pairs, with each pair capable of delivering approximately 400 m^3/sec. The access and conveyor declines will have a dedicated ventilation system separate from the main mine ventilation system for safety reasons.

Other mine infrastructure will include two dewatering pumps, each capable of pumping 88 L/s. Only one pump will be used at any one time with the other as back-up.



ES Figure 5: Extraction level layout

(Source: AuRico, 2016)

There will be an underground maintenance facility to service and maintain the underground mobile equipment. Major equipment rebuilds will be done at the surface workshop that was previously used for the open pit. ES Table 6 presents a complete list of underground infrastructure.

ES Table 6: Underground Infrastructure

Item	Description
Primary Ventilation	Two main intake fans and two main exhaust fans, each capable of approx. 200 m³/s at 2.4 kPa to 2.7 kPa.
Conveyor	3,200 m long x 1,067 mm wide, speed of 2.9 m/s.
Crushers	Four 55 inch x 42 inch jaw crushers delivering a nominal 80% minus 165 mm product.
Maintenance	Workshop capable of accommodating ten pieces of equipment c/w 20 t crane, wash bay, drill shop, offices, stores and refuge station.
U/G Storage and Warehousing	Eight storage bays, one fuel bay, and four sets of explosive magazines.
Mine Dewatering	Two pumps, each capable of 88 L/s.
Electrical Power Supply and Distribution	Two main 25.0 kV power cables down declines to main sub-station.
Lunchroom and Refuge Stations	Three each.

ES Table 7 shows the annual lateral development and raise requirements to fully develop the KUG mine, totaling 47,751 m lateral development and 2,196 m raise development.

ES Table 8 shows the annual KUG mine production with first ore produced in Year -2 and full production achieved in Year 3, with full production occurring over ten years. Total ore mined is 107.3 Mt at 0.27% Cu and 0.54 g/t Au and 1.99 g/t Ag for 285.7 kt Cu, 1,868 koz Au and 6,878 koz Ag.

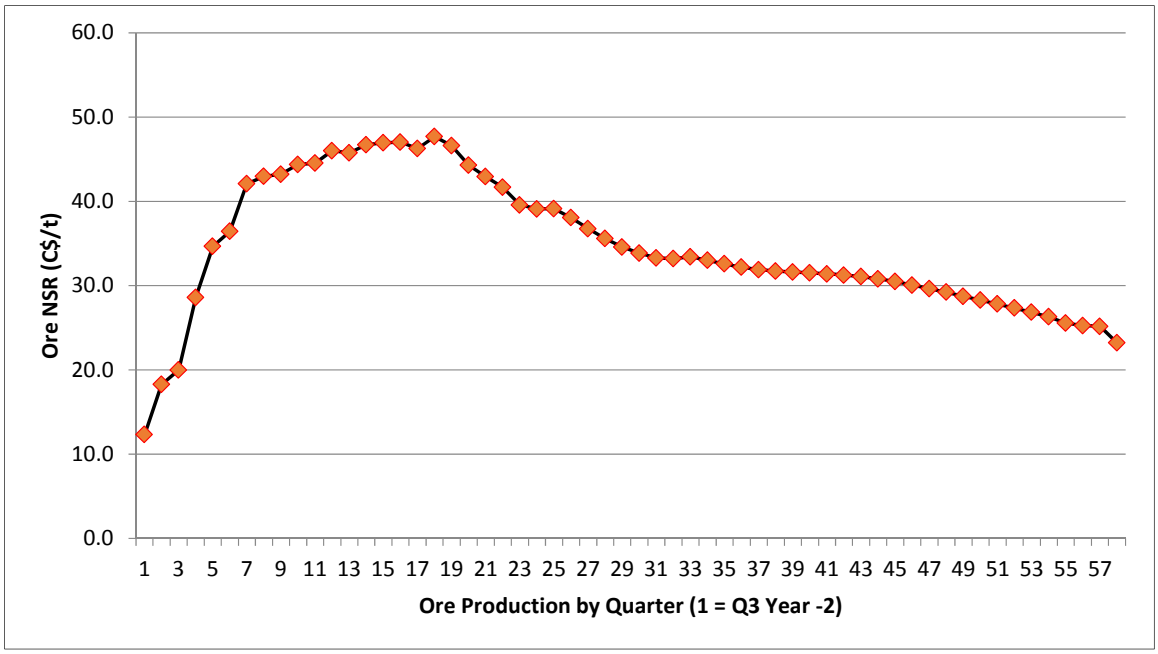
ES Figure 6 shows ore NSR (C$/t terms) by quarter. Lower values in early quarters are associated with initial development to the south of the footprint. However, as development and related undercutting and drawbell opening pushes to the east of the footprint, average ore NSR values increase rapidly to peak at almost C$48/t in Q18 of ore production.

ES Table 7: Lateral Development Schedule

Item	Units	Total	Year -4	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Total Lateral Development	m	47,751	1,347	5,510	7,331	8,503	7,214	6,343	4,582	4,420	2,501	0
Total Raise Development	m	2,196	0	0	890	469	228	202	126	142	139	0

ES Table 8: Production Schedule

Name	Units	Total	Year -4	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
Undercut Ore	t	3,324,864	0	0	0	398,694	773,027	657,363	501,346	530,017	464,417	0	0	0	0	0	0	0	0
Drawbell Ore	t	1,722,946	0	0	0	192,598	410,350	334,652	262,075	279,986	243,286	0	0	0	0	0	0	0	0
Development Ore	t	1,550,802	0	0	223,802	377,953	244,737	237,634	163,053	165,457	138,166	0	0	0	0	0	0	0	0
Cave Ore	t	100,782,885	0	0	0	66,169	2,009,623	4,610,044	8,093,200	8,032,121	8,155,313	9,015,093	9,011,443	9,011,075	9,003,991	9,014,949	8,999,986	9,021,130	6,738,749
TOTAL Ore	t	**107,381,498**	**0**	**0**	**223,802**	**1,035,413**	**3,437,736**	**5,839,693**	**9,019,675**	**9,007,581**	**9,001,181**	**9,015,093**	**9,011,443**	**9,011,075**	**9,003,991**	**9,014,949**	**8,999,986**	**9,021,130**	**6,738,749**
TOTAL Waste	t	**2,975,734**	**91,018**	**371,232**	**324,822**	**336,812**	**592,749**	**582,021**	**309,779**	**352,595**	**14,705**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
TOTAL Mine	t	**110,357,232**	**91,018**	**371,232**	**548,625**	**1,372,226**	**4,030,485**	**6,421,714**	**9,329,455**	**9,360,176**	**9,015,886**	**9,015,093**	**9,011,443**	**9,011,075**	**9,003,991**	**9,014,949**	**8,999,986**	**9,021,130**	**6,738,749**
Cu Grade	% Cu	0.27	0.00	0.00	0.14	0.26	0.34	0.34	0.34	0.32	0.29	0.26	0.25	0.25	0.25	0.24	0.23	0.21	0.20
Cu	t Cu	285,660	0	0	319	2,701	11,632	19,842	30,634	29,021	26,250	23,823	22,703	22,387	22,112	21,449	20,300	19,126	13,363
Au Grade	g/t Au	0.54	0.00	0.00	0.22	0.46	0.66	0.73	0.76	0.70	0.61	0.55	0.51	0.48	0.47	0.46	0.43	0.40	0.38
Au	oz Au	1,867,859	0	0	1,568	15,312	72,879	137,109	220,437	202,271	176,785	158,444	148,229	140,436	136,271	133,306	125,684	117,394	81,732
Ag Grade	g/t Ag	1.99	0.00	0.00	1.26	1.77	2.10	2.25	2.30	2.30	2.15	2.05	1.99	1.96	1.94	1.88	1.79	1.69	1.59
Ag	oz Ag	6,878,658	0	0	9,055	58,937	231,930	422,864	666,481	667,084	622,465	595,442	575,998	569,043	560,905	544,918	517,633	490,921	344,982
Ore NSR	C$/t	35.1	0.0	0.0	16.4	31.9	43.4	46.0	47.0	43.9	39.0	35.2	33.2	32.1	31.4	30.6	28.9	27.1	25.3



ES Figure 6: Ore NSR by quarter

(Source: SRK, 2016)

Mineral Processing

The Kemess Underground (KUG) project intends to process 9 Mt/y (24,600 t/d equivalent) through one of the two original Kemess South (KS) grinding circuits that processed KS ore. The “A side” grinding circuit has been removed. The existing Kemess “B side” grinding circuit will be used to process the KUG ore following primary crushing underground and stockpiling ahead of the process plant. The original flotation, thickening, and concentrate handling facilities remain, while the tailings will be pumped to the KS open pit (as the KUG Tailings Storage Facility, TSF). Near the end of the KS operation in 2009 to 2011, tailings were being deposited in the KS open pit.

A comparison of the KS and KUG ores has been made to determine the suitability of the KS flowsheet and equipment. From the last two years of operating data, the KS process plant achieved the equivalent of more than 9 Mt/y of throughput on one side of the parallel circuits with a KS ore type similar to that of KUG ore. Subsequent SMC and Bond Work Index (BWI) tests conducted on a range of KUG material suggest that, on average, 9 Mt/y can be achieved with this ore with some variability due to ore hardness.

The KS process plant circuit generated a concentrate grade of 20% Cu to 23% Cu at 70% to 85% Cu recovery and 55% to 70% Au recovery. For the last few years of KS operation, grades were 0.1% Cu to 0.2% Cu and 0.2 g/t Au to 0.4 g/t Au compared to KUG estimated averages of 0.27% Cu and 0.54 g/t Au.

For KUG ore, the process plant will produce a single concentrate at an estimated grade of 22% Cu, and is expected to achieve recoveries of 91% Cu, 72% Au and 65% Ag.

A significant mineralogical difference in the KUG ore is its higher pyrite to chalcopyrite ratio. This will result in greater mass flow of rougher concentrate to the regrind circuit than similar KS hypogene ore. In addition, the finely disseminated nature of the KUG copper and gold particles necessitates a finer regrind size than the KS regrind circuit had historically achieved. Testwork indicates that metallurgical performance deteriorates as this regrind stream coarsens from an 80% passing size (P_{80}) of 18 µm to over 40 µm. This affects pyrite rejection in the cleaning circuit as well as gold losses – which can be as high as 20% with the pyrite to cleaner tails.

To achieve target recoveries and a reasonable concentrate grade, the primary grind is expected to be approximately 150 µm while the regrind size is to be a P_{80} of 20 µm. A 22% Cu concentrate is being targeted, as higher grade of concentrate results in significant gold losses from pyrite cleaning.

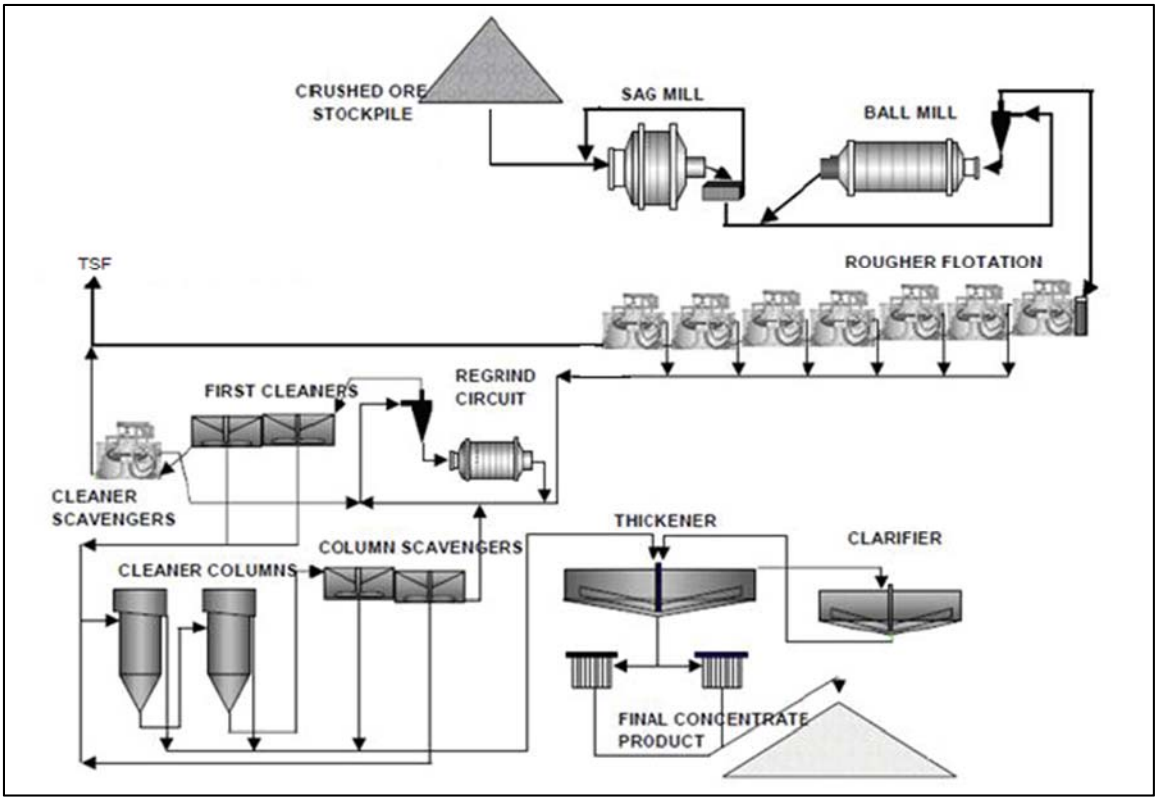
The existing regrind circuit capacity will not be sufficient to handle the greater rougher concentrate mass pull (increasing from 14% to 18% of the plant feed) combined with the finer grind size requirements of the KUG ore. A series of fine grinding tests including batch Eliason, Levin, and IsaMill Signature Plot were completed to estimate the energy required for the regrind circuit. A stirred regrind mill circuit has been included in the flowsheet following the existing regrind ball mill and capital has been allocated for this equipment.

Process Plant

Four single-toggle jaw crushers will be installed underground in the KUG mine. Ore will be conveyed from the crushers to the conveyor decline portal where it will be transferred to an overland conveyor for delivery to the existing process plant ore feed stockpile.

Based on the metallurgical testwork that has been completed for the KUG project, the existing Kemess South process plant flowsheet has the unit operations required to process the ore and achieve the predicted results summarised in Section 16. There are some additional equipment requirements, primarily for fine regrinding, but the main preparation efforts will focus on reconditioning the existing equipment.

One SAG mill/ball mill grinding line was removed from the KS grinding circuit. The targeted process plant throughput for the KUG project is estimated to be 24,600 tpd on average. The simplified flowsheet in ES Figure 7 illustrates the revised flowsheet. A copper flotation concentrate containing gold and silver values will be produced and transported to smelters for processing. Tailings will be discharged to the KS open pit which will be the KUG tailings storage facility (TSF) which has sufficient volume capacity for the duration of the KUG operations. Process reclaim water will be pumped to the KS process plant facility from the KUG TSF.



ES Figure 7: KUG simplified flowsheet

(Source: SRK, 2016)

Surface Infrastructure

The Kemess South Mine was brought into production as a 52,000 t/d open pit mine in 1998 and produced continuously until 2011.

It was a fully integrated complex that also included:

- Primary crusher, stacker conveyor, and ore stockpile;
- 52,000 t/d process plant (or concentrator) producing Cu-Au concentrate;
- Administration, workshop/service, and warehouse complex;
- Camp with accommodation for in excess of 300 on-site personnel;
- Tailings storage facility (TSF) with associated tailings and reclaim water pipeline systems;
- 380 km, 230 kV electrical power line from BC Hydro's Kennedy siding sub-station;
- 1.6 km airstrip capable of handling most STOL (short take-off & landing) aircraft and Hercules transport planes when heavier loads were required;

- 380 km all-season access road from Mackenzie; and
- Rail load-out facility at Mackenzie.

One half of the grinding circuit has been removed, effectively reducing grinding capacity to approximately 25,000 t/d. However, the remaining facilities are intact, having been under care and maintenance since 2011 and are available to support the KUG project. Where refurbishment, upgrade, or addition to existing facilities is required, capex has been estimated and provided for under processing, G&A, and site services cost estimates.

For KUG tailings storage, it is planned to use the Kemess South open pit, with the open pit having capacity for approximately 107 Mt tailings and 3.0 Mt waste rock from KUG development. To achieve this storage capacity, a 25 m high dam (the “east dam”) is required to be constructed at the east end of the open pit. A spillway will be constructed in the south-west area of the pit to allow discharge of excess water once KUG operations have ceased while ensuring adequate water cover of potentially acid-generating (PAG) tailings.

Access to the KUG mine workings (via triple declines) will be provided by an access corridor comprising an access road west of the Kemess South open pit, access tunnel through the hill to the north of the open pit, and access road across the Kemess Lake Valley. Electrical reticulation will be extended along this corridor to provide grid power to the KUG mine. Mining contactor facilities will be established in the Kemess Lake Valley following development of the access tunnel. The conveyor used to transport crushed ore from the KUG mine will be installed along the access corridor with discharge to the existing Kemess South primary crusher surge hopper (with crusher removed).

Environmental, Permitting, and Closure General Setting

The environmental assessment and permitting framework for metal mining in Canada is well established. The federal and provincial environmental assessment processes provide a mechanism for reviewing major projects to assess their potential impacts. AuRico examined and scoped the permitting and environmental requirements of the KUG project in the context of the ongoing regulatory requirements of the Kemess South mine.

The KUG project exceeds production rate thresholds for a gold mine under the Canadian Environmental Assessment Act 2012; therefore, a federal environmental assessment (EA) was triggered. The project does not automatically trigger the need for a provincial review under the BC Environmental Assessment Act. However, as a federal EA is required and the potential exists to de-risk project approvals through the provincial EA process with prescribed timelines, AuRico requested the KUG project be designated provincially as a reviewable project. The project is currently in the application phase of a substituted EA process. Favourable decisions are expected to be reached prior to the end of 2016.

Following a successful EA process, the KUG project will be required to obtain a number of new provincial and federal licenses/permits. A number of existing permits for the Kemess South Mine have been in place since 1996 that remain in good standing, but may require amendment or renewal before construction begins.

As part of the EA process, AuRico has considered potential effects on several valued components of the natural and human environment including, among other components, aquatic and terrestrial ecosystems, current use of lands and resources for traditional purpose. The most substantive potential impacts of the project are associated with the long-term management of waste rock, tailings, mine water and process water and their downstream effects on high quality fish habitat. With the application of appropriate engineering design, project planning, and implementation of mine and environmental management plans, it is anticipated that the project will avoid any significant environmental effects.

Aboriginal and Public Consultations

AuRico has engaged substantively with the surrounding communities to inform them of plans and gain their support for the KUG project. Since 2010, AuRico has continued to meet regularly and consult with the Tse Keh Nay prior to and during the EA process. These meetings helped to identify Tse Keh Nay members' Aboriginal rights and interests that may be affected by the KUG project, provided updated information about the KUG project, and built relationships. Consultations to date have been positive to support the EA review and permitting process.

AuRico undertook communications and consultations with the public between 2011 and 2015. This includes residents of northwest BC communities including Mackenzie, Smithers, Terrace, Prince George, as well as the local governments representing these communities; interest groups and non-governmental organizations; commercial interests, license and tenure holders; and recreational land-users.

Closure and Post Closure Water Management

At closure, mine dewatering will cease and water-retaining bulkheads will be installed within the declines to flood the underground workings. Once groundwater levels rebound to stable levels, contact water from the underground will discharge via natural groundwater flow paths to East Cirque Creek. Groundwater quality in the area of the underground mine is not expected to change materially from present conditions and long-term water treatment is not anticipated.

During the initial closure phase, excess water within the KUG TSF will require ongoing treatment and controlled discharge to Attichika Creek. Water quality within the KUG TSF will improve over time with ongoing treatment and natural recharge from precipitation. Once water quality within the KUG TSF reaches concentrations that allow for untreated discharge to the receiving environment, active water treatment will cease and the project will transition to post closure. It is estimated that on-going water treatment will be required for six years following the operations phase. In post closure, the KUG TSF will have a free overflow to Waste Rock Creek (ahead of Attichika Creek).

Capital Cost Estimate

A total of C$950M in capital is estimated (CapEx) to be spent over the life of the project as summarized in ES Table 9.

Full pre-production expenditure including capitalised pre-production operating costs is estimated to be approximately C$524M to first ore production.

The total estimated expenditure to commercial production including capitalised pre-production operating expenditure (OpEx) is C$687M. This is partially offset by pre-commercial revenue such that the net cash requirement for the project development is C$604M (after adjustments for working capital and taxes).

Ongoing sustaining capital is approximately C$263M, including underground mobile equipment replacements and ongoing underground development.

ES Table 9: Summary Capital Cost Estimates

Item	To First Production (C$M)	Pre-Commercial [1] (C$M)	Sustaining (C$M)	Life of Mine (C$M)
Project Owners Costs	7	7	0	7
Mine	205	266	207	473
Mill	31	38	13	51
Site Service	13	13	9	22
Conveyor	40	40	0	40
UG Electrical	18	19	0	19
UG Ventilation	10	10	1	11
Access Corridor	36	36	0	36
Water Treatment Build	13	13	0	13
TSF and Pipeline Build	7	8	32	39
Reclamation Cost	0	0	1	1
Total Capital	**380**	**449**	**263**	**712**
Capitalised Pre-production OpEx(COC)	144	238	0	238
Total Capital (Incl. COC)	**524**	**687**	**263**	**950**
Pre-commercial Net Revenue	N/A	(83)	N/A	(83)
Net CapEx (Incl pre-commercial OpEx and Net Revenue)	**524**	**604**	**263**	**867**

Note: [1] Pre-Commercial values inclusive of those “To First Production”

Operating Cost Estimate

The cost of labour is the primary driver of the operating costs (OpEx). The scheduling to minimize labour costs and maximize productivity is best met with rotating shifts. All of the hourly and many of the staff will rotate on a schedule of two weeks on-site and two weeks off-site. Where possible, some senior staff will rotate four days on-site and three days off-site. This will minimize the need for on-site senior staff. Weekend coverage will be maintained by rotating a minimum team to ensure coverage of all essential areas.

Summaries of the KUG life-of-mine and unit OpEx estimates from the commencement of commercial production are shown in ES Table 10.

ES Table 10: OpEx estimate - Summary of Unit Costs

Item	Life of Mine (C$/t ore)	Commercial Production (C$/t ore)
Mining Production	5.54	4.89
Primary Ventilation	0.40	0.33
Conveyor	0.17	0.18
Subtotal UG Mining	**6.11**	**5.39**
Processing plant	5.71	5.69
Water Treatment	0.43	0.26
Site Services	1.22	0.82
G & A	3.33	2.11
Total KUG Cost	**16.80**	**14.27**

Economic Analysis

The economic analysis was undertaken using an Excel-based, discounted cashflow (DCF) model that modelled cashflows by quarter (three-month periods). The nominal discount rate of 5% was used with an exchange rate (US$ per C$) of 0.75.

Long term commodity pricing used was US$2.50/lb for copper, US$1,200/oz for gold, and US$16.00/oz for silver. ES Table 11 summarises the base case assumptions.

The cashflow model generally uses 2016 real Canadian dollars.

SRK Note: on instruction from AuRico, an assumption of 0% inflation was used throughout the model.

The payback periods are shown in ES Table 12. The discounted payback period is 5.6 years from the start of commercial production and 4.1 years for a non-discounted basis from commercial production.

Net Present Value (NPV) of the project cashflow after tax is C$258M at a discount rate of 5%. The undiscounted cashflow is C$695M after tax. The after-tax IRR is 11.9%.

ES Table 11: Base Case Assumptions

Base Case Pricing	Units	Long-Term
Copper	US$/lb	$2.50
Gold	US$/oz	$1,200
Silver	US$/oz	$16.00
Exchange rate	US$ per C$	0.75
Modeled Recovery to Concentrate		
Copper Recovery to Concentrate	%	91
Gold Recovery to Concentrate	%	72
Silver Recovery to Concentrate	%	65

ES Table 12: Payback Periods

Payback Periods	From 1st production (years)	From commercial production (years)
Discounted Payback Period	6.3	5.6
Non-discounted payback period (real)	4.8	4.1
Nominal payback period	4.8	4.1

Project Execution

This project will be managed by AuRico personnel with support from engineering consultants as required. Contractors will carry out all construction activities including construction of the access corridor to the triple decline portal area in the Kemess Lake Valley including driving the access tunnel; extending electrical reticulation; establishing the ventilation raises; developing and installing the underground fixed infrastructure; installing water treatment plants; installing conveyor system; installing process plant equipment; and constructing the Kemess Underground (KUG) TSF east dam and spillway.

Construction of the KUG access corridor is planned to start in Q4 Year -5 along with installation of the electrical power reticulation system, with Year -5 being the first year of KUG project development. Development of the triple declines will occur from Q4 Year -4. Year -4 also sees construction of the selenium treatment plant, discharge pumping-pipeline system from the KUG TSF to Attichika Creek, initial camp upgrades and communications system installation.

Installation of the conveyor system occurs in Year -2 and Year -1, ahead of commissioning of the first two primary jaw crushers and conveyor system from Q1 Year 1. Mill reconditioning also commences in Year -2 ahead of process plant start-up in Q4 Year -1.

A metals treatment plant is installed in Year -1, required for removing various metals from KUG TSF water. As with the selenium treatment plant, this plant will be located in the existing process plant building. Two fine grind mills are also installed in the process plant this year.

Commercial production is achieved from Q4 Year 1 (defined as the process plant achieving >60% of the targeted 9 Mt/a throughput). Cave production and ore processing continues until Year 13.

Year 6 and Year 7 see construction of the initial KUG TSF east dam to 1260 mRL. The first dam raise follows in Year 9 to the 1265 mRL, followed by the second dam raise to the 1270 mRL in Year 10, and the final dam raise to the 1275 mRL in Year 11. The final KUG TSF-related construction occurs in Year 12 when the spillway is constructed.

The peak workforce complement of 573 people, with a maximum 321 people on-site, occurs in Q3 Year -1 as a result of development-construction activities both underground and in the process plant, as well as operational readiness activities; comprising both contractor and owner personnel.

Risks & Opportunities

SRK conducted a risk and opportunities assessment workshop for the Kemess Underground project. Risks and opportunities were quantified for both likelihood (probability of occurrence) and consequence (impact of occurrence). Risks and opportunities were provided by SRK, AuRico, and the other external consultants listed in this report.

The consequences were assessed over six categories; safety impact, revenue impact, revenue deferral/advance, capital cost, operating cost, and schedule delay/advance. The opinion of the likelihood of occurrence was based on professional opinion given on the current understanding of the project. The responsible party for each risk and opportunity ranked is denoted in the risk and opportunity register. Risks and opportunities are ranked under the assumption that mitigation methods mentioned in this study are put in place, and that industry standard practices are generally implemented.

The risks and opportunities outlined in this section were considered at the asset level only. Effects at the corporate level, which may impact cost of capital or corporate reputation, were not considered as part of this assessment and therefore are not reflected in the risk and opportunity rankings.

A total of 59 items were assessed, and each was considered as both a potential risk and opportunity. The number of items assessed should not be taken as an indication of the risk or opportunity around the project, but of the diligence of the assessment.

No risks or opportunities scored above a moderate risk ranking. A moderate risk represents a threat to the project and effective additional mitigation measures are recommended. The highest risk scored 10/100 and the highest opportunity scored 8/100. The low values of the risk and opportunity

scoring are typical for a project which is in the feasibility study stage. No technical risks were identified that were significant enough to recommend that the project does not proceed. The Kemess Underground Project is considered to be of low technical risk in the context of other block/panel caving projects.

ES Table 13 summarises key risks and opportunities for the project.

ES Table 13: Summary of Key Risks and Opportunities

ID	Description	Likelihood	Safety Impact	Revenue Impact	Revenue Deferral/Advance	Capital Costs	Operating Costs	Schedule Delay	Ranking
Risks									
GTC01r	Drawpoint Failure	15%–50%	-	10% < X <20% over LOM	-	-	X < 5%	-	10
GTC02r	Draw Cone Interaction	15%–50%	-	5% < X <10% over LOM	-	-	-	-	6
GTC04r	Mudrush Risk	5%–15%	Single fatality	-	X < 10% per year	X < 10%	X < 5%	-	9
UGM01r	Loader Automation	15%–50%	-	-	X < 10% per year	X < 10%	X <5%	-	6
UGM04r	Mining Dilution	15%–50%	-	5% < X <10% over LOM	-	-	-	-	6
UGM08r	Underground Development Schedule	15%–50%	-	-	-	10% < X < 20%	-	X <1 year	6
PRO06r	Recovery and Concentrate Grade	50%–90%	-	2% < X <5% over LOM	-	- X < 10%	-	-	8
EXM01r	Commodity Price	15%–50%	-	10% < X < 20% over LOM	-	-	-	-	8
EXM04r	Labour Market	15%–50%	-	-	-	10% < X < 20%	5% < X < 10%	-	8
EXM06r	Electricity Cost	15%–50%	-	-	-	X < 10%	5% < X < 10%	-	6
EXM05r	Fuel Cost	15%–50%	-	-	-	X < 10%	5% < X < 10%	-	6
Opportunities									
UGM04o	Mining Dilution	15%–50%	-	5% < X <10% over LOM	-	-	-	-	6
UGM11o	Draw Rate	> 90%				X < 10%	X < 5%		7
TMG06o	Mineral Reserve Increase	50%–90%		5% < X <10% over LOM					8
EXM01o	Commodity Price	15%–50%	-	10% < X < 20% over LOM	-	-	-	-	8
EXM04o	Labour Market	15%–50%	-	-	-	10% < X < 20%	5% < X < 10%	-	8
EXM05o	Fuel Cost	15%–50%	-	-	-	X < 10%	5% < X < 10%	-	6

Discussion and Conclusions

This section summarises the main conclusions and updates to the KUG project since the previous feasibility study, completed in 2013, as well as the updated KE resource estimate.

On the basis of the revenue and costs outlined in Section 21, the project is estimated to have a post-tax NPV of C$258M at a 5% discount rate and a post-tax internal rate of return of 11.9%. The cumulative project cash flow becomes positive in real terms 4.1 years after commencement of commercial production. On a discounted basis at 5%, the project has returned the investment 5.6 years after commencement of commercial production.

Mineral Resource Estimate

The Mineral Resource estimate presented in Section 13 represents an update to both the Kemess Underground Mineral Resource and the Kemess East Mineral Resource, and includes new 3D models of geological structure, lithology and alteration. In addition, a new interpretation of grade estimation domains, a change in grade estimation parameters and a new block size were incorporated in the models.

The Kemess Underground Mineral Resource model was prepared by SRK, considering 149 drillholes drilled by AuRico Metals and predecessor companies during the period of 1976 to 2011. The structural and lithological model was prepared by AuRico Metals, while the alteration model and grade estimation was prepared by SRK.

The Kemess East Mineral Resource model was prepared by SRK, considering 66 holes drilled by AuRico Metals and predecessor companies during the period of 2005 to 2015. The structural and lithological models were prepared by AuRico Metals, while the alteration model and grade estimation was prepared by SRK.

SRK is of the opinion that the current drillhole information is sufficiently reliable and accurate to confidently interpret the boundaries of the porphyry mineralization, and the assay data are sufficiently reliable to support the Mineral Resource estimation for both deposits. The audit showed no significant errors from the resource area regarding the recording of tabulated analytical data. The Kemess Underground and Kemess East databases for the 2016 resource were verified and can be relied upon for resource estimation.

It was concluded that the resource models reasonably match the data, while reviewing the results on section. Locally, the estimations were reviewed by graphing the mean grade of the variable against the easting, northing and elevation for each of the domains (Swath Plots). The block averages in general bisect the sample plot indicating no bias between the model estimates and the samples.

To assess the sensitivity of the resource statements to changes in NSR cut-off value, SRK summarized tonnage and grade above cut-off at a series of increasing NSR cut-offs by resource category. It can be observed that the Kemess Underground resource is reasonably insensitive to cut-off values in the increment from C$14.00/t NSR to C$16.00/t ore NSR, which is likely the value

range of interest; while the Kemess East resource is reasonably insensitive to cut-off values in the increment from C$16.00/t NSR to C$18.00/t ore NSR, which is likely the value range of interest.

Mineral Reserve Estimate

SRK is of the opinion the Mineral Reserve estimate for the KUG deposit has been completed to a level appropriate for feasibility studies.

To enable a single cut-off parameter to be used when selecting the economic parts of the resource to be mined, SRK estimated the NSR value received for concentrate produced from each tonne of resource, net of all costs and losses incurred in the off-site transportation and processing of concentrate.

The Mineral Reserve Estimate stated in this report is constrained by the size of the facility that can receive the tailings with 107 Mt. The Mineral Resource statement includes 243 Mt of indicated material of which a significant proportion could be converted into Mineral Reserves, providing alternative storage location(s) could be identified.

Mining

The selection of cave mining as the preferred underground mining method was made in 2011 as part of a PEA completed by AMC Mining Consultants (AMC, 2011). The 2013 FS (SRK, 2013) and this 2016 update continued with this decision. SRK`s scope included reviewing existing geotechnical reports and data, collecting and reviewing additional geotechnical data, providing geomechanical mine design parameters, cavability assessment, fragmentation assessment, and ground support recommendations.

Numerical modelling of KUG was based on rock mass properties provided by SRK. Itasca's modelling results show the KUG deposit will cave given the planned dimensions and the cave will propagate to surface ~1.5 years after cave initiation.

The severity of drawpoint hang-ups varies between geotechnical domains, with the Black Lake Intrusive expected to generate the largest frequency and largest average size. The Hazelton and Takla Potassic domains are expected to generate relatively few and small hang-ups and the Takla Phyllic domain is expected to be relatively insignificant in terms of hang-ups.

The KUG workings are accessed by a triple decline system, with declines for access (of personnel, equipment and materials), conveying and intake air ventilation. This approach is adopted due to challenging topography in the project area and to minimize the environmental impact of the KUG project. The primary ventilation circuit is completed by a dedicated return air raise to surface from south of the cave footprint.

The mine layout includes undercut and extraction levels. The undercut level is located at the 1,158 mRL and is accessed via ramps from the extraction level. The extraction level is located at the 1,140 mRL and is accessed via the access decline at the 1,099 mRL.

Caving zone ore will be mucked from drawpoints by 8.6 m^3 (17 t) semi-autonomous LHDs and dumped into jaw crushers located to the south of the footprint. Transfer conveyors will then transport ore to the main underground conveyor.

An underground conveyor, 3.2 km in length, will transport ore from the transfer conveyors to a transfer located adjacent to the triple decline portals. An overland conveyor, 4.9 km in length, will then transport ore to the existing crushed ore stockpile ahead of the process plant.

Total ore tonnes associated with the KUG Feasibility Study is 107.3 Mt, with a target production rate of 9.0 Mt/y or 25 kt/d based on 360-day per year operation. Potential exists to increase production beyond 9.0 Mt/y based on the number of available drawpoints. Production is currently constrained by underground ventilation and tramming limitations, and potentially by processing plant throughput.

The ingress of water into the underground workings is likely to increase when the cave propagates into the “broken zone”. Extreme inflows resulting from snowmelt and rainfalls will be mitigated by using the lower ventilation drifts for emergency water storage. Extreme inflow events that flood the ventilation level may result in a short-term losses of production.

Processing/Process Plant

The KUG project intends to process 9 Mt/y (24,600 t/d equivalent) through one of the two original Kemess grinding circuits that processed Kemess South material. The “A side” grinding circuit has been removed. The existing Kemess “B side” grinding circuit will be used to process the ore following primary crushing underground and stockpiling ahead of the concentrator. The original flotation, thickening, and concentrate handling facilities remain, while the tailings will be pumped to the Kemess South pit where tailings were being deposited near the end of the Kemess South operation from 2009 to 2011.

A comparison of the Kemess South and KUG ores has been made to determine the suitability of the Kemess South flowsheet and equipment. From the last two years of KS operating data, the process plant achieved the equivalent of more than 9 Mt/y of throughput on one side of a parallel circuit with a similar Kemess South ore type.

A significant mineralogical difference in the KUG ore is its higher pyrite to chalcopyrite ratio. This will result in greater mass flow of rougher concentrate to the regrind circuit than similar Kemess South Hypogene ore. In addition, the finely disseminated nature of the copper and gold particles necessitates a finer regrind size than the Kemess regrind circuit had historically achieved.

Existing regrind circuit capacity will not be sufficient to handle the greater rougher concentrate mass pull combined with the finer grind size requirements of the underground ore. A series of fine grinding tests were completed to estimate the energy required for the regrind circuit. A stirred mill regrind circuit has been included in the flowsheet following the existing regrind ball mill, and capital has been allocated in the study.

The predicted metallurgical results have been based on a comparison of the locked cycle test results from a 2003 program and more recent testwork in 2011. It is expected that the KUG

concentrator will generate a copper concentrate of 22% Cu at 91% copper recovery, 72% gold recovery, and 65% silver recovery.

The copper concentrate produced from the KUG orebody will contain both gold and silver credits and is expected to be free of deleterious elements and not subject to any penalties. As such, KUG concentrate is expected to be marketable to both smelters and traders.

Surface Infrastructure

A dedicated access corridor is to be developed to connect the existing Kemess South facilities to the KUG workings, with access via the west side of the Kemess South open pit and through an access tunnel (865 m length) to the Kemess Lake Valley. In addition to providing access for personnel, equipment and materials, this access corridor will contain the electrical powerline for the underground mine, ore conveyor and mine dewatering pipeline.

For KUG tailings storage, it is planned to use the Kemess South open pit, with a capacity for approximately 107 Mt tailings and 3.0 Mt waste rock from underground development. To achieve this storage capacity, a 25 m high dam (East Dam) is required to be constructed at the east end of the open pit. A spillway will be constructed in the south-west area of the pit to allow discharge of excess water once KUG operations have ceased while ensuring adequate water cover of potentially acid generating (PAG) tailings.

Additional tailings storage capacity is potentially available in the Kemess South open pit, but this would require further raising the East Dam, constructing an additional dam in the South saddle area and reconfiguring the closure spillway.

For water management, a more conservative scenario has been selected for this update compared with the earlier FS (SRK, 2013). Based on extensive modeling, the scenario assumes discharge from the Kemess South pit will begin prior to process plant start-up and will require treatment for selenium and dissolved metals. The water treatment plants were designed by BioteQ Environmental Technologies Inc. to remove contaminants from impacted underground and surface water streams prior to discharge into Attichika Creek.

The selenium treatment plant will be installed in Year -4 and located on the ground floor of the existing process plant building. The metals treatment plant will be installed in Year -1 on a newly constructed second floor of the same building. Both plants will be predominantly supplied as pre-fabricated modules to minimize on-site construction.

Existing Kemess South infrastructure that can be utilized for the KUG project includes the grid-connected electrical powerline and access road (both originating at Mackenzie, approximately 400 km south-east of Kemess), admin-workshop-warehouse complex, camp (including kitchen, sewage, potable water supply and recreation facilities), airstrip, site roads and fuel tank farm.

Environmental and Social

The KUG project is in essence the addition of an undeveloped ore body to an existing mining project. As such, it has the advantage of being able to make use of significant environmental data gathered throughout two previously completed Environmental Assessments (EA) as well as 15 years of data gathered during development and operations.

AuRico has engaged substantively with the surrounding communities to inform them of plans and gain their support for the project. Since 2010, AuRico has continued to meet regularly and consult with the Tse Keh Nay (TKN) prior to and during the EA process. Consultations to-date have been positive to support the EA review and permitting process. TKN and AuRico signed an Interim Measures Agreement in May 2012 and are currently negotiating an Impacts and Benefits Agreement.

At KUG closure, mine dewatering will cease and water-retaining bulkheads will be installed within the declines to flood the underground workings. Once groundwater levels have rebound to stable levels, contact water from the underground will discharge via natural groundwater flow paths to East Cirque Creek. Groundwater quality in the area of the underground mine is not expected to change materially from present conditions and long-term water treatment is not anticipated.

KUG tailings and waste rock will be stored in the Kemess South open pit. The PAG waste materials will be stored under a water cover to prevent metal leaching or acid rock drainage. As the mine approaches the end of its operational life, a large above-water NAG tailings beach will be constructed at the east end of the TSF. At closure, this NAG tailings beach will extend from the East Dam to the supernatant pond at the west end of the TSF.

Economic Evaluation

A total of C$950M is estimated to be spent in capital over the life of the project. Preproduction expenditure is estimated to be approximately C$524M to first ore production, and an additional C$163M to achieve commercial production which is anticipated to be offset by C$83M in pre-commercial production revenue to give a total C$604M net cash required for project development to achieve commercial production. Ongoing sustaining capital is approximately C$263M, including underground mobile equipment replacement and ongoing underground development.

The economic analysis was undertaken using an Excel-based DCF model that modelled cash flows by quarter (three-month periods). A nominal discount rate of 5% was used with an exchange rate (US$ per C$) of 0.75. Long term commodity pricing used was US$2.50/lb for copper, US$1,200/oz for gold and US$16.00/oz for silver. SRK believes the metal prices that have been chosen are realistic price scenarios reflecting industry norms.

The cost of labour is the primary driver of the operating costs. The total unit operating cost after the commencement of full commercial production is estimated at C$14.27/t. A majority of this cost is made up of a mining cost of C$4.89/t, a processing cost of C$5.69/t, and a G&A cost of C$2.11/t (refer to ES Table 7 for further details).

The major assumptions used for the cost estimation are C$0.91/litre for diesel fuel, C$0.50/litre for propane and C$0.058/kWhr for electrical power. The labour rates were provided by AuRico and include base wage/salary, production bonus, and 30% burden of the base for all employee categories. Shift rotations are four days on and three days off for senior staff and two weeks on and two weeks off for all other employees.

Recommendations

This section identifies opportunities that can increase confidence in the success of the KUG project and add economic value, as well as opportunities that can increase the confidence in the KE resource estimate. Additional details are found in the body of this report.

Mineral Resources

The mineral resources for both the KUG and KE deposits are considered quite well defined. However, confidence in both resource models could be increased, by replacing key drillholes that are currently based on unreliable downhole survey resuts, with a drillhole that is accurately surveyed. Drilling to better define the post mineralization structures, as well as drilling to better define the potassic alteration zones, could also lead to a higher confidence resource model as well as potentially increasing the resources.

Mineral Reserves

There is an opportunity to significantly increase the KUG reserves beyond those stated in this report, depending on available tailings storage capacity. The tailings facility can be optimized to determine the ultimate capacity within the KE pit or alternative storage location(s) identified.

With additional tailings storage capacity, a significant proportion of the 243 Mt of indicated material in the KUG Mineral Resources could be converted into Mineral Reserves.

Mine Design

Caveability and fragmentation will vary from the base case because there is always a degree of uncertainty scaling up from empirical and numerical modelling. Additional rockmass data would reduce that uncertainty and enhance confidence in the representativeness of the given information.

Development is the largest component of mine capital costs. Internal ramps, portions of the perimeter drifts, and even the maintenance facility are examples of potential cost reductions without compromising the project.

A fully integrated mud rush risk assessment is required prior to commencement of caving. It should be noted that the presence of a mud rush risk does not make mining infeasible.

Processing

Future metallurgical testwork could provide further optimization of the target primary and regrind sizes for the various ore zones in the deposit.

The addition of a secondary crusher ahead of the existing process plant has the potential to increase throughput by 10% or higher to a nominal 9.9 Mt/y depending on ore hardness.

Environmental and Social

It is recommended the KUG project be advanced through the BC environmental assessment process with a high quality EA Application, thereby ensuring optimal review and approval timelines. In addition, AuRico should continue consultation with the TKN First Nations, other Aboriginal groups and the public in order to build support for the project.

Contingency plans should be developed for water management during operations and post-closure to ensure discharge water quality guidelines are achieved. End land use objectives for the KUG project are recommended to be consistent with the former Kemess South Mine.

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1 Introduction and Terms of Reference

This technical report was prepared for AuRico Metals Inc. (AuRico) to summarize the results of a feasibility study (FS) on the Kemess Underground project (KUG) and an updated resource estimate for the Kemess East (KE) deposit, both located in north-central British Columbia, Canada, and approximately 6.5 km north of the past-producing Kemess South Mine.

The contract with SRK Consulting permits AuRico to file this report as a technical report with the Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under provincial securities law, any other uses of this report by any third party is at that party's sole risk. The responsibility for this disclosure remains with AuRico. The user of this document should ensure that this is the most recent technical report for the property, as it is not valid if a new technical report has been issued.

Table 1.1: Areas of Responsibilities

Company	Area of Responsibility
SRK	Mineral Resource Estimation, Quality Assurance/Quality Control (QA/QC), Geology, Geotechnical Engineering, Mineral Reserve Estimate, Underground Mine Design, Schdule and Infrastructure, Mining and G&A Costs, Economics, Project Execution Plan, Construction and Development
Mine Ventilation Services	Mine Ventilation
Conveyor Dynamics	Conveyor Design - Surface & Underground
KWM Consulting	Metallurgy and Mineral Processing
AMEC Foster Wheeler	Water Management, Surface Infrastructure and Waste Management Engineeering
BioteQ Environmental Technologies	Waste water treatment
ERM Consultants Canada	Environmental and Permitting
Struthers Technical Solutions	Electrical reticulation

1.1 Qualification of SRK

The SRK Group comprises of more than 1,000 professionals, offering expertise in a wide range of resource engineering disciplines. The independence of the SRK Group is ensured by the fact that it holds no equity in any project it investigates and that its ownership rests solely with its staff. These facts permit SRK to provide its clients with conflict-free and objective recommendations. SRK has a proven track record in undertaking independent assessments of mineral resources and mineral reserves, project evaluations and audits, technical reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies, and financial institutions worldwide. Through its work with a large number of major international mining companies, the SRK Group has established a reputation for providing valuable consultancy services to the global mining industry.

1.2 Responsibility

This report has been prepared by SRK under the direction of AuRico as the study manager, with primary contributions from AuRico, AMEC Foster Wheeler, BioteQ Environmental Technologies, Conveyor Dynamics, ERM Consultants Canada, Exen Consulting Services, KWM Consulting, Mine Ventilation Services, and Struthers Technical Solutions.

SRK does not accept liability for the statements, findings and opinions expressed in the portions of the reports authored by other contributors. This technical report was written by the authors shown in Table 1.2.

Table 1.2: List of Authors and Responsibilities

Author	Company	Area of Responsibility
Chad Yuhasz, P.Geo	SRK Consulting	Mineral Resource Estimation, Quality Assurance/Quality Control (QA/QC), Geology
Jarek Jakubec, C.Eng, MIMMM	SRK Consulting	Geotechnical Engineering, Mineral Reserve Estimate
Iouri Iakovlev, P.Eng	SRK Consulting	Mine Design & Schedule, Underground Infrastructure, Construction and Development, Mining and G&A Costs, Project Economics and Project Execution Plan
Dan Stinnette, P.Eng	Mine Ventilation Services	Mine Ventilation
Andrew Jennings, P.Eng	Conveyor Dynamics	Conveyor Design - Surface & Underground
Chris Struthers, P.Eng	Struthers Technical Solutions	Electrical Reticulation
Ken Major, P.Eng	KWM Consulting	Metallurgy and Mineral Processing
David Kratochvil, P.Eng	BioteQ Environmental Technologies	Waste Water Treatment
Rolf Schmitt, P.Geo	ERM Consultants Canada	Permitting, Environment, Social Impact Assessment, and Stakeholder Consultation
Serge Chevrier, P.Eng	AMEC FW	Access Corridor
Steve Rice, P.Eng	AMEC FW	Tailings Storage Facility
Marianne Rosted, P.Eng	AMEC FW	Access Corridor
Don Kidd, P.Eng	AMEC FW	Water Management

Any previous technical reports or literature used in the compilation of this report are referenced in the relevant text as necessary.

All units in this report are based on the International System of Units (SI), except industry standard units, such as troy ounces for the mass of precious metals.

This report uses abbreviations and acronyms common to the mineral industry. Definitions are provided in Section 26.

1.3 Declaration

SRK's opinion contained herein and effective, is based on information collected by SRK throughout the course of SRK's investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of AuRico, and neither SRK nor any affiliate has acted as advisor to AuRico, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

2 Reliance on Other Experts

SRK has not performed an independent verification of land title and tenure information as summarized in Section 3 of this report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, but have relied on AuRico Metals. AuRico has provided a letter dated July 2, 2015 confirming that all land title information was registered with the British Columbia Mineral Titles Online in the name of AuRico Metals Inc. at the time of the creations of AuRico Metals on July 2, 2015. The reliance applies solely to the legal status of the rights disclosed in Sections 3.1 and 3.2.

SRK was informed by AuRico that there are no known litigations potentially affecting the Kemess Project.

SRK has no expertise in Concentrate Marketing Studies and relied on Exen Consulting Services (Exen) who was contracted by AuRico Metals to provide the marketing and logistics of copper concentrates from the Kemess underground project. Their review is summarized in Section 18.

3 Property Description and Location

The Kemess Property is located in the mountainous area east of the Spatsizi Plateau and west of the Swannell Ranges near Thutade Lake approximately 250 km north of Smithers and 430 km northwest of Prince George at 57°02' north longitude and 126°47' west latitude. Figure 3.1 displays the project location. The property spans the boundary between the 94E and 94D NTS sheets and is within the Omenica Mining Division.

Broad, open, drift, and moraine covered valleys characterize the area, which yield to sub-alpine plateaus and rugged incised peaks and cirques. Elevations range from 1,200 m to 2,000 m, with the tree line occurring at approximately 1,500 m. All the work completed during the 2002 to 2010 drill programs occurred above the tree line in three cirques that open to the north forming a common southern headwall. Lower elevations on the property are moderately vegetated with spruce-willow-birch forest, while poorly drained areas form peat bogs populated by alder brush, willow, and stunted spruce trees.



Figure 3.1: Project Location

(Source: SRK, 2013)

3.1 Mineral Tenure

AuRico holds mineral title to 58 claims totaling 29,302 ha. AuRico also has leasehold on an additional four claims totaling 3,483 ha. A plan map showing claim boundaries is provided in Figure 3.2, and a listing of all claims held by AuRico and current claim status is provided in (Source: SRK, 2013)

Table 3.1.

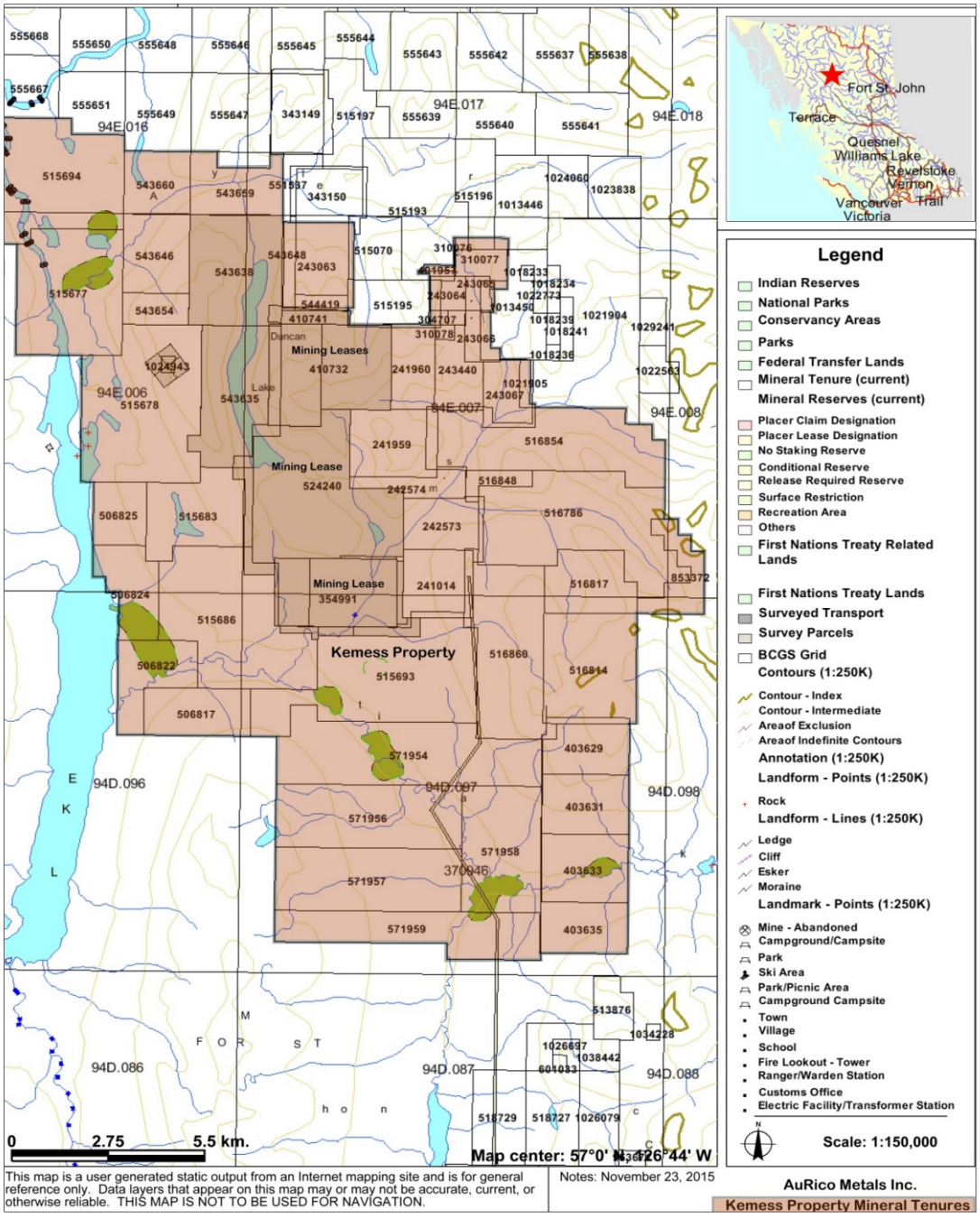


Figure 3.2: Mineral Tenure map showing AuRico claim boundaries (shaded) and claim numbers

(Source: SRK, 2013)

Table 3.1: Mineral Tenure information as recorded with BC mineral titles online

AuRico Metals Inc. - Kemess Property, BC

Summary	No.	Hectares	Acres
A. Mining Leases	4	3,483.330	8,607.308
B. Mineral Claims	58	29,302.692	72,406.951
	62	32,786.022	81,014.260

AuRico Metals Inc. - Client No. 281385

Mining Leases

Tenure No.	Name/District Lot	Owner	Map No.	Lease Term Date	Good To Date	Area (ha)
354991	Kemess South Lease: DL's 7198, 7199, 7200, 7201,7204, 7207	281385 (100%)	094E007	2027/sep/15	2016/sep/15	862.330
410732	Kemess North Lease: DL's 7032, 7328	281385 (100%)	094E007	2034/sep/29	2016/sep/29	950.000
410741	Kemess North Lease: DL.7329	281385 (100%)	094E007	2034/sep/29	2016/sep/29	106.000
524240	Midway Lease: DL.7342	281385 (100%)	094E	2035/dec/22	2016/dec/22	1565.000
4 Mining Leases					**Hectares:**	**3483.330**

Mineral Claims

Tenure No.	Claim Name	Owner	Map No.	Issue Date	Good To Date	Area (ha)
241014	SEM #1	281385 (100%)	094E007	1989/jul/18	2025/dec/14	400.000
241959	NEK 3	281385 (100%)	094E007	1990/may/03	2025/dec/14	500.000
241960	NEW KEMESS 3	281385 (100%)	094E007	1990/may/03	2025/dec/14	375.000
242573	DU 2	281385 (100%)	094E007	1990/aug/02	2025/dec/14	500.000
242574	NEK 4	281385 (100%)	094E007	1990/aug/01	2025/dec/14	350.000
243063	CAN 1	281385 (100%)	094E007	1991/feb/02	2025/dec/14	500.000
243064	DUNC 1	281385 (100%)	094E007	1991/feb/01	2025/dec/14	100.000
243065	DUNC 2	281385 (100%)	094E007	1991/feb/01	2025/dec/14	100.000
243066	DUNC 3	281385 (100%)	094E007	1991/feb/01	2025/dec/14	150.000
243067	CREEK	281385 (100%)	094E007	1991/feb/02	2025/dec/14	300.000
243440	ALISON 1	281385 (100%)	094E007	1991/may/14	2025/dec/14	500.000
304706	GOZ 1	281385 (100%)	094E007	1991/sep/21	2025/dec/14	25.000
304707	GOZ 2	281385 (100%)	094E007	1991/sep/21	2025/dec/14	25.000
310076	DUN 1	281385 (100%)	094E007	1992/may/31	2025/dec/14	225.000
310077	DUN 2	281385 (100%)	094E007	1992/may/31	2025/dec/14	225.000
310078	DUN 3	281385 (100%)	094E007	1992/may/31	2025/dec/14	225.000
355408	MILL CREEK 4	281385 (100%)	094E007	1997/apr/23	2025/dec/14	25.000
401957	UN 1	281385 (100%)	094E007	2003/apr/29	2025/dec/14	50.000
403629	BEAR 6	281385 (100%)	094E007	2003/jul/09	2025/dec/14	500.000
403631	BEAR 8	281385 (100%)	094E007	2003/jul/09	2025/dec/14	500.000
403633	BEAR 12	281385 (100%)	094E007	2003/jul/09	2025/dec/14	500.000

Tenure No.	Claim Name	Owner	Map No.	Issue Date	Good To Date	Area (ha)
403635	BEAR 16	281385 (100%)	094E007	2003/jul/09	2025/dec/14	375.000
405949	LAT 1	281385 (100%)	094E007	2003/oct/05	2025/dec/14	25.000
414229	DUNC 4	281385 (100%)	094E007	2004/sep/15	2025/dec/14	25.000
414230	DUNC 5	281385 (100%)	094E007	2004/sep/15	2025/dec/14	25.000
414231	UN 2	281385 (100%)	094E007	2004/sep/16	2025/dec/14	25.000
414232	UN 3	281385 (100%)	094E007	2004/sep/16	2025/dec/14	25.000
506817	TLK 1	281385 (100%)	094D	2005/feb/11	2025/dec/14	423.346
506822	TLK 2	281385 (100%)	094D	2005/feb/11	2025/dec/14	423.244
506824	TLK 3	281385 (100%)	094D	2005/feb/11	2025/dec/14	387.724
506825	TLK 4	281385 (100%)	094E	2005/feb/11	2025/dec/14	281.830
515677		281385 (100%)	094E	2005/jun/30	2025/dec/14	1108.035
515678		281385 (100%)	094E	2005/jun/30	2025/dec/14	1443.310
515683		281385 (100%)	094E	2005/jun/30	2025/dec/14	669.335
515686		281385 (100%)	094D	2005/jun/30	2025/dec/14	1427.856
515693		281385 (100%)	094D	2005/jun/30	2025/dec/14	1534.100
515694		281385 (100%)	094E	2005/jun/30	2025/dec/14	1353.164
516786		281385 (100%)	094E	2005/jul/11	2025/dec/14	1391.637
516814		281385 (100%)	094D	2005/jul/11	2025/dec/14	863.906
516817		281385 (100%)	094D	2005/jul/11	2025/dec/14	440.555
516848		281385 (100%)	094E	2005/jul/11	2025/dec/14	105.661
516854		281385 (100%)	094E	2005/jul/11	2025/dec/14	1197.161
516860		281385 (100%)	094D	2005/jul/11	2025/dec/14	1075.379
543635		281385 (100%)	094E	2006/oct/19	2025/dec/14	897.552
543638		281385 (100%)	094E	2006/oct/19	2025/dec/14	861.624
543646		281385 (100%)	094E	2006/oct/19	2025/dec/14	439.571
543648		281385 (100%)	094E	2006/oct/19	2025/dec/14	105.501
543654		281385 (100%)	094E	2006/oct/19	2025/dec/14	175.891
543659		281385 (100%)	094E	2006/oct/19	2025/dec/14	421.784
543660		281385 (100%)	094E	2006/oct/19	2025/dec/14	351.483
544419		281385 (100%)	094E	2006/oct/25	2025/dec/14	70.351
571954		281385 (100%)	094D	2007/dec/14	2025/dec/14	1340.848
571956		281385 (100%)	094D	2007/dec/14	2025/dec/14	988.435
571957		281385 (100%)	094D	2007/dec/14	2025/dec/14	988.868
571958		281385 (100%)	094D	2007/dec/14	2025/dec/14	847.371
571959		281385 (100%)	094D	2007/dec/14	2025/dec/14	830.261
853372	Kemess Creek 1	281385 (100%)	093D & 094E	2011/may/03	2025/dec/14	264.315
1024943	CAIRNX	281385 (100%)	094E	2014/jan/09	2026/jan/09	17.596
58 Mineral Claims					**Hectares:**	**29302.692**

3.2 Underlying Agreements

There are no underlying agreements relevant to KUG project.

3.3 Permits and Authorization

A number of existing permits for the Kemess South Mine have been in place since 1996 that remain in good standing, authorizing the use of roads, the power line, explosives magazines, camp and process plant related facilities/functions (e.g., nuclear gauge and boiler use).

The list of provincial and federal authorizations, licences, and permits that are anticipated to be required for the construction and/or operation of the KUG project are discussed in Section 19 of this report. The list is not intended to be exhaustive and does not suggest the permitting methodology which may include amendment of existing permits or the requirement for new applications. AuRico has been engaged with the MMPO since August 2015 and will continue to meet with the appropriate governmental agencies to discuss the permitting requirements and approach to the KUG project.

3.4 Environmental Considerations

The KUG project will require approval from provincial and federal governments prior to receipt of permits, licenses and authorizations to construct and operate the proposed KUG mine.

The KE deposit is currently at the exploration stage, so only permits related to exploration drilling are currently required. This work is subject to an exploration agreement with the TKN First Nations.

Reclamation and closure preparation of Kemess South has been undertaken annually via the progressive reclamation strategy in the approved Mine Reclamation and Closure Plan. AuRico has been recognized for being pro-active in reclamation and closure at the mine site, and has been the recipient of a number of reclamation awards from the British Columbia Technical and Research Committee on Reclamation and the Mining Association of British Columbia.

The reader is directed to section 19 for details on environmental considerations. Alternatively, the reader can refer to the Kemess Underground Environmental Assessment Application that is has been submitted to the B.C. EAO and the Canadian Environmental Assessment Agency.

3.5 Mining Rights in British Columbia, Canada

Mining leases are the tenure form that is required for mining activities and these are granted for a specific term, typically 21 years. Mining leases may be renewed in the last year of their term. No specific annual work commitment is required but the lease will only be renewed if it is required for mining activity. Leases are maintained by the annual payment of $20 per hectare which extends the “good to date” as shown in (Source: SRK, 2013)

Table 3.1.

Exploration claims allow for the exploration and sampling of a property. These are maintained by filing Assessment Reports and based on the overall expenditures the claim earns credits that maintain the tenure for a prescribed time as shown in (Source: SRK, 2013)

Table 3.1.

The surface rights on the Kemess property are public lands owned by the Province of British Columbia. All areas of proposed activities fall either on AuRico mining leases or on unpatented mining claims controlled by the Province. Proposed actions will be subject to approval by the Province of British Columbia of a Plan of Operations and qualified by the terms of the Record of Decision for that document. SRK is of the opinion that surface rights are sufficient for all planned construction and operational activities.

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

4.1 Accessibility

All-season road access to the Kemess property is via the Omineca resource access road from the town of Mackenzie (Figure 4.1). Operations at Kemess South were serviced by scheduled year round flights from various locations in British Columbia.

The Kemess Underground (KUG) deposit lies approximately 6.5 km to the north of the existing Kemess South (KS) processing plant and other infrastructure. The deposit is located beneath two north facing alpine cirques with ground surface elevations ranging from 1,500 to 2,000 m, all above the tree line. Vehicular access to the area overlying the deposit is currently limited to summer months only. Helicopter access is possible during winter. The KS processing plant and other infrastructure are located at an elevation of approximately 1,200 m and are accessible all year round.

The Kemess East (KE) deposit is located approximately 1 km east from the KUG deposit.

Below the tree line, broad, open, drift, and moraine covered valleys characterize the area. These areas are moderately vegetated with spruce-willow-birch forest, while poorly drained areas form peat bogs populated by alder brush, willow, and stunted spruce trees.

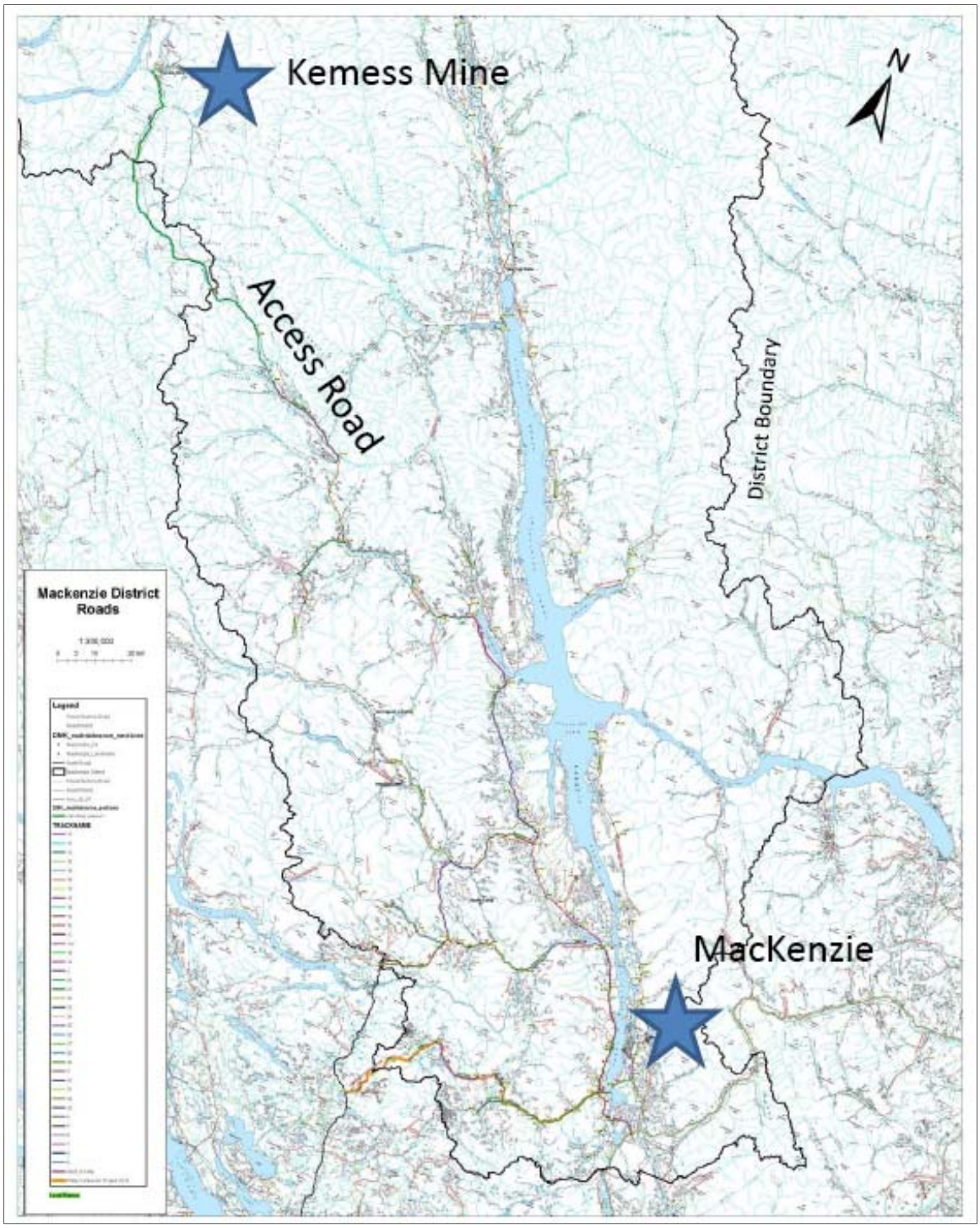


Figure 4.1: Road Access to Mine Site

(Source: SRK, 2013)

4.2 Local Resources and Infrastructure

Existing on-site infrastructure includes offices, warehouse, laydowns, maintenance facilities, a 300-person accommodation camp footprint, ore processing plant, crushed ore stockpile and reclaim, access and service roads, airstrip, explosives magazines, tailings storage facilities, and electrical sub-station.

The Kemess South Mine operated on a fly-in fly-out (FIFO) basis with the majority of employees working a cycle of two-weeks-on and two-weeks-off, commuting via charter aircraft from regional centres. The airstrip is approximately 1,600 m in length and can accommodate most short take-off and landing (STOL) aircraft types. Hercules aircraft have occasionally used the airstrip for heavy lifts.

A Company owned, 380 km power line originating in Mackenzie, provides power to the mine area via the BC Hydro grid.

Waste rock and tailings from the mining and processing of KUG ore is planned for deposition in the KS open pit (as the KUG tailings storage facility (TSF)), with a 25 m high dam required at the east end of the KUG TSF to facilitate all storage requirements. Process water will be available from the KS open pit for processing of KUG ore.

Potable water is sourced from permitted wells and treated through an onsite water treatment plant.

AuRico currently maintains all site facilities and infrastructure on a care and maintenance basis, pending decisions regarding environmental certification and permitting of the KUG project.

4.3 Climate

The climate is generally moderate, although snow can occur during any month. Temperatures range from –35°C to 30°C and average annual precipitation amounts to 890 mm. Extreme weather conditions are possible at the higher elevations.

During the Kemess South operations, access to site was 365 days a year and the mine operated continuously from 1997 through 2011. Site access for the current care and maintenance activities during winter months is only by fixed wing aircraft, although road access is still maintained during the non-winter months.

4.4 Physiography

The Kemess property is located in the northern and central plateaus and mountains physiographic region of British Columbia. This region is dominated by flat to rolling topography with mature erosional surfaces. The mountain chains tend to exhibit lower relief than the coastal and south eastern mountains and glacial drift can be quite thick (Figure 4.2).

Broad, open, drift, and moraine covered valleys characterize the area, yielding to sub-alpine plateaus and rugged incised peaks and cirques. Elevations range from 1,200 m to 1,800 m, with the tree line occurring at 1,500 m. Snow does typicallynot leave the higher elevations until late June.

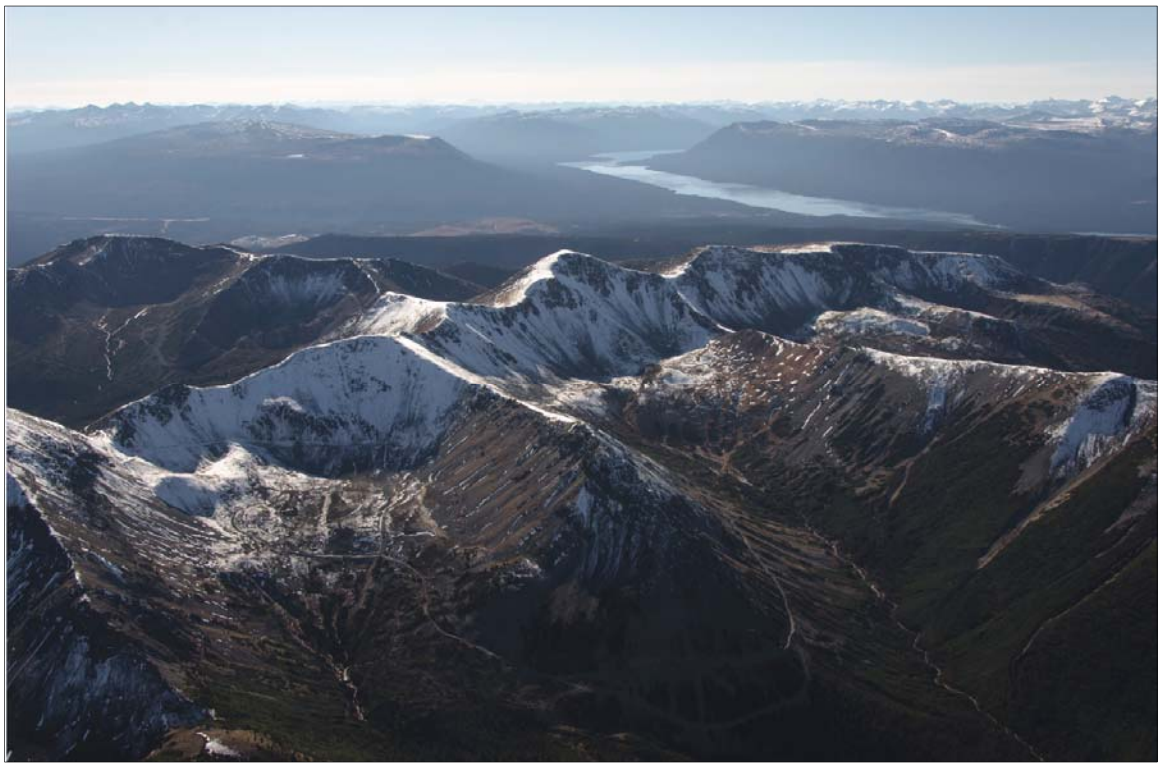


Figure 4.2: View of Kemess East (left foreground) Kemess Underground (centre foreground) and Kemess infrastructure (left background) Thutade Lake (upper right)

(Source: AuRico, 2016)

5 History

5.1 Prior Ownership and Ownership Changes

Kennco initially staked the Kemess North ground in 1966 following up on regional silt geochemical Surveys. From 1968 to 1971 they conducted soil, silt, and bedrock geochemical sampling, geological mapping, and 232 m of diamond drilling. From 1975 to 1976 Getty Mines optioned the property and did some additional geochemical work as well as 2,065 m of diamond drilling. In 1986 El Condor Resources optioned the property from Kennco and from 1986 to 1992 completed electromagnetic (EM) surveys including magnetometer and induced polarization (IP), and an additional 14,328 m of diamond drilling. In 1996 Royal Oak Mines acquired the property and went on to develop the Kemess South operation. Northgate Minerals Exploration acquired the Kemess South deposit and associated Kemess North property in 2000 from Royal Oak Mines. Northgate was subsequently acquired by AuRico Gold in 2011. In 2015 AuRico Gold and Alamos Gold merged and AuRico Metals spun off from that merger with Kemess being part of AuRico Metals.

5.2 Previous Development and Production

There is no previous production from the Kemess Underground deposit, or the Kemess East deposit. The history of the Kemess property records earlier discoveries in the Kemess North area.

5.3 Kemess South History (KS)

The first exploitable open pit deposit was outlined at the Kemess South deposit. Royal Oak Mines Inc. (Royal Oak) then owners of the property, commenced operations from the Kemess South deposit in 1998. The operation went into receivership in 1999, and in 2000 Northgate Exploration a predecessor company to AuRico Metals bought the property out of receivership.

Historical production from Kemess South since the original start-up is shown in Table 5.1. Production ceased in 2011 and work is now focused on reclamation and site rehabilitation.

Table 5.1: Kemess South production history

Operator	Year	WasteMined (t)	Ore Milled (t)	Grades		Metal produced	
				Mill Head			
				%Cu	g/t Au	Cu (t)	Au (oz)
Royal Oak	1997	6,014,000	0	0.000	0.000		
Royal Oak	1998	24,838,324	7,482,909	0.220	0.557	9,687	69,804
Royal Oak	1999	8,668,980	14,113,460	0.212	0.644	21,389	213,793
Northgate	2000	19,911,880	14,089,000	0.222	0.779	23,151	225,998
Northgate	2001	17,246,162	15,366,500	0.251	0.855	30,076	277,106
Northgate	2002	27,123,742	17,308,300	0.236	0.724	33,051	282,255
Northgate	2003	34,617,235	18,633,000	0.225	0.702	34,554	294,117
Northgate	2004	36,647,429	18,589,000	0.231	0.735	35,513	303,475
Northgate	2005	31,718,631	19,168,000	0.219	0.641	33,440	279,962
Northgate	2006	25,502,552	18,233,978	0.244	0.763	36,837	310,298
Northgate	2007	24,959,000	17,802,317	0.213	0.627	30,904	245,631
Northgate	2008	14,408,998	16,924,271	0.175	0.506	23,549	185,180
Northgate	2009	10,259,364	18,352,153	0.160	0.440	23,812	173,040
Northgate	2010	2,299,998	19,457,000	0.138	0.282	21,598	103,582
Northgate	2011	0	3,040,086	0.129	0.255	2,962	14,671
Total		**284,216,295**	**218,559,974**	**0.209**	**0.626**	**360,524**	**2,978,911**

5.4 Kemess Underground History (KUG)

Northgate consolidated the exploration information on the Kemess Underground deposit from previous owners and participants, which includes Kennco Exploration Ltd. from 1966 to 1971, Getty Mines Ltd. and Shell Oil from 1975 to 1976, and El Condor Resources Ltd. from 1986 to 1992.

Since 2000, Northgate has conducted field exploration on the project including drilling programs in 2000, 2001, 2002, 2003, 2004, 2010, and 2011. Details of the exploration history at Kemess Underground are summarized in Table 5.2.

Table 5.2: Kemess Underground exploration history

Period	Company	Work completed
1966–1971	Kennco Explorations Ltd.	Regional stream and soil geochemistry, staked 100 2-post mineral claims, mapping at 1:9600 scale and completed 232 m of x-ray core drilling in eight holes.
1975–1976	Getty Mines Ltd. and Shell Oil	Optioned property from Kennco and completed 1:4800 scale mapping, orthomapping, re-staking, geochemical surveying, and 2,065 m of diamond drilling in 13 holes (75–18 to 75–30). Option dropped in 1977.
1986–1992	El Condor Resources Ltd.	In 1986 El Condor optioned the property from Kennco and commenced sustained exploration that resulted in the discovery at Kemess South. Over a six year period at Kemess North, El Condor collected 1,025 rock samples and 5,402 soil samples, completed 76.85 km of induced polarization work, and drilled 14,328 m of core in 69 holes. Additional work included 167 km of line cutting, 54.5 km of roads, and 475 m of

Period	Company	Work completed
		trenching. A resource of 157 Mt at 0.37 g/t Au and 0.18% Cu resulted at Kemess North.
2000	Northgate Exploration Ltd.	Completed 4,104 m of diamond drilling in 12 holes identified a new higher-grade porphyry zone located east of El Condor's discovery. This work increased the resource at Kemess North to 360 Mt at 0.299 g/t Au and 0.154% Cu.
2001	Northgate Exploration Ltd.	Completed 8,220 m of diamond drilling in 16 holes, which increased resources to 442 Mt at 0.40 g/t Au and 0.23% Cu.
2002	Northgate Exploration Ltd.	Completed 33,686 m of diamond drilling in 58 holes (41 holes on Kemess North, 5 holes on Kemess East, and 12 holes at Nugget).
2003	Northgate Exploration Ltd.	Completed 21,851 m of diamond drilling in 61 holes (24 holes for exploration, 24 holes for condemnation, 7 holes for geotech, and 6 holes for water monitoring).
2004	Northgate Minerals Corp.	Completed 9,970 m of diamond drilling in 32 holes (16 holes on Kemess North and Nugget, 5 condemnation holes, 4 holes on Duncan Ridge, 6 holes at Hilda, and one hole at Kemess Centre).
2005	Northgate Minerals Corp.	Completed 16,158 m of diamond drilling in 40 holes (3 holes on Kemess North, 4 holes on NOR1, 3 holes on Kemess East, 3 holes on the Kemess North Offset Zone, 18 holes on the Bear Claims, 1 hole on the Nugget Zone, 2 holes on the Orion Zone and 6 holes at Duncan Ridge). Hole KN-05-24 discovers Kemess North Offset Zone.
2006	Northgate Minerals Corp.	Completed 8,689 m of diamond drilling in 18 holes (9 holes on Kemess North, and 9 holes on Kemess East). Completed Titan 24 IP survey along length of KN trend. Hole KH-06-03 discovers Altus zone at Kemess East.
2007	Northgate Minerals Corp.	Completed 18,132 m of diamond drilling in 28 holes (3 holes on Kemess North, 24 holes on Kemess East and 1 hole on NOR1). Completed Titan 24 IP survey grid at Kemess East. Hole KH-07-04 discovers Ora zone at Kemess East.
2008	Northgate Minerals Corp.	Completed 27 line kms of IP on the north dam grid.
2010	Northgate Minerals Corp.	Completed 16,439 m of diamond drilling in 30 holes on Kemess Underground deposit.
2011	Au Rico Gold Inc.	Completed 2,207 m of diamond drilling in 3 holes on Kemess Underground deposit and 3,962 m of diamond drilling in 16 holes for geotechnical and hydro-geological purposes in the vicinity of proposed underground infrastructure.

Note: Mineral resources prior to 2002 are historic, have not been verified by a Qualified Person, are not being treated as current by AuRico Metals, and should not be relied upon.

In February 2002, Northgate filed a technical report to the Canadian Securities Regulatory Authorities that included a mineral resource statement for the contemplated open pit deposit.

In June 2004, following further drilling, Northgate published a revised mineral resource estimate, which formed the basis of a pre-feasibility study.

In January 2005, Northgate completed the Kemess North Feasibility Study together with updated mineral reserve and resource estimates. The feasibility study envisaged a large open pit mining operation with tailings from an expanded milling operation deposited in the nearby Amazay Lake (Duncan Lake). The project involved mining and milling approximately 424 Mt of ore and stripping approximately 318 Mt of waste.

In May 2005 Northgate filed a NI 43-101 technical report titled “Revised Mineral Reserve and Resource Kemess North Project”. However, Northgate subsequently was unable to obtain regulatory approval to develop the deposit in the manner envisaged in the feasibility study and the deposit remains undeveloped.

In May 2010, Associated Mining Consultants Inc. (AMC) completed a review of the deposit. The review concluded that despite the need for more reliable base data, there is clearly a high-grade zone in the eastern part of the Kemess North deposit (Kemess Underground) that has the potential to be mined by block caving methods. The study considered that an annual production rate of approximately 8 Mtpa should be possible. Ore would be conveyed out of the mine via an inclined drift and then transferred to the existing Kemess concentrator by either truck haulage or conveyor.

Following the review, Northgate completed an infill drilling program during the summer of 2010 targeting the potential mining area. The program included a number of holes drilled specifically to gather geotechnical data and all drill core recovered was subjected to detailed geotechnical logging. The 2010 drilling program also provided core samples for a metallurgical test work program. Results from the 2010 and earlier drilling programs, provided input to a mineral resource estimate released by Northgate in February 2011.

In August 2011, Northgate announced the results of a preliminary economic assessment (PEA) for the Kemess Underground project which outlined a cumulative production of 1.1 Moz of gold and 490 Mlb of copper over a mine life of approximately 12 years. The current feasibility study is based on the February 2011 mineral resource estimate.

The 2011 drilling program conducted by AuRico served several purposes: to collect more baseline geotechnical information, to confirm geologic modelling, and to investigate the ground water and hydrogeology surrounding the deposit.

In 2013, AuRico and SRK compiled the feasibility study and technical report on the KUG project with then current long-term metal prices and costs.

This 2016 technical report provides an update to the 2013 KUG feasibility study and technical reports based on an updated mineral resource estimate, mine design, infrastructure design and cost estimates.

5.5 Kemess East History (KE)

In January 2015, AuRico issued a press release disclosing an initial mineral resource estimate for the Kemess East deposit.

In March 2016, AuRico Metals issued a press release disclosing an updated mineral resource estimate for the KE deposit, prepared by SRK Consulting Canada. This update is the subject of related parts of this technical report, with other parts of this report relating to the updated KUG mineral resource estimate and associated feasibility study.

Details of the exploration history at Kemess East are summarized in Table 5.3.

Table 5.3: Kemess East exploration history

Period	Company	Work completed
2005	Northgate Minerals Corp.	Completed three holes as part of the Kemess North drilling were in the Kemess East area, 779.7m.
2006	Northgate Minerals Corp.	Completed 8,689 m of diamond drilling in 18 holes (8 holes on Kemess North, and ten holes on Kemess East). Completed Titan 24 IP survey along length of KN trend. Hole KH-06-03 discovers Altus zone at Kemess East.
2007	Northgate Minerals Corp.	Completed 18,132 m of diamond drilling in 28 holes (three holes on Kemess North, 24 holes on Kemess East and one hole on NOR1). Completed Titan 24 IP survey grid at Kemess East. Hole KH-07-04 discovers Ora zone at Kemess East.
2013	AuRico Gold Corp.	Completed 13,330m of diamond drilling in 9 holes (7 holes on Kemess East and two holes on the Kemess Offset Zone). Completed regional soil sampling.
2014	AuRico Gold Corp.	Completed 16,872.5m of diamond drilling in 12 holes on Kemess East. Completed a ZTEM property wide geophysical survey. Conducted regional soil sampling.
2015	AuRico Metals Corp.	Completed 29,111m of diamond drilling in 34 holes (12 holes on Kemess East, 3 holes on the Kemess Offset Zone, 9 holes testing regional targets and 10 holes conducting geotechnical drilling for the Kemess Underground Feasibility Study). Completed a property wide Lidar Survey. Conducted regional soil and sampling and mapping.

6 Geological Setting and Mineralization

6.1 Regional Geology

Kemess occurs at the southern end of the Toodoggone mining camp, which describes a collection of occurrences and deposits found in Mesozoic volcanic rocks. The area is known for its copper-gold porphyry deposits and low sulphidation epithermal gold-silver vein deposits.

The oldest rocks in the belt are Permian marine and volcanic rocks, which are disconformably overlain by basalt dominated volcanic rocks of the middle Triassic Takla Group, which are in turn unconformably overlain by lower-middle Jurassic Hazelton Group volcanic rocks.

Intrusive rocks are prevalent in the area and have been categorized as late Triassic Alaskan type ultramafics such as pyroxene diorite, hornblende gabbro, and pyroxenite. Economically more significant are the early Jurassic intrusives of the Black Lake suite, which are granodiorite, hornblende diorite, pyroxene quartz diorite, quartz monzonite, and quartz monzodiorite.

The Mesozoic volcanic assemblages form upright, shallowly dipping to flat-lying sequences crosscut by high angle north to northwest trending faults. Significant dextral strike-slip features bound the eastern margin of the belt.

More local to Kemess Underground and Kemess East are north-northwest normal block fault structures. Thrust faulting is present in the district and is interpreted as Eocene or younger.

The district represents the results of three superimposed volcanic arc building stages that began in the upper Paleozoic. Marine volcanic and sedimentary successions dominated until the lower-middle Jurassic, when continental, quartz normative volcanism began with the deposition of the Hazelton Group-Toodoggone Formation sequences. The plutonic rocks of the Black Lake suite are coeval with the Toodoggone sequence and are likely co-magmatic. Block faulting has juxtaposed panels of varying depth into the magmatic and volcanic systems.

The Kemess Underground area is underlain by upper Triassic (Takla Group) andesite/basaltic volcanics and to a lesser extent lower Jurassic (Toodoggone Formation) dacitic fragmental volcanics.

Stocks, dykes, and possible sills of quartz monzonite/quartz diorite composition have intruded the Takla succession and are also lower Jurassic in age. The deposit area is transected by steeply dipping north to northwest trending normal faults. A laterally extensive, shallow dipping to flat lying, highly fractured, and altered broken zone occurs at or close to the surface in the area of the KUG deposit. Figure 6.1 shows the district geology, major intrusive masses, and location of the district's deposits.

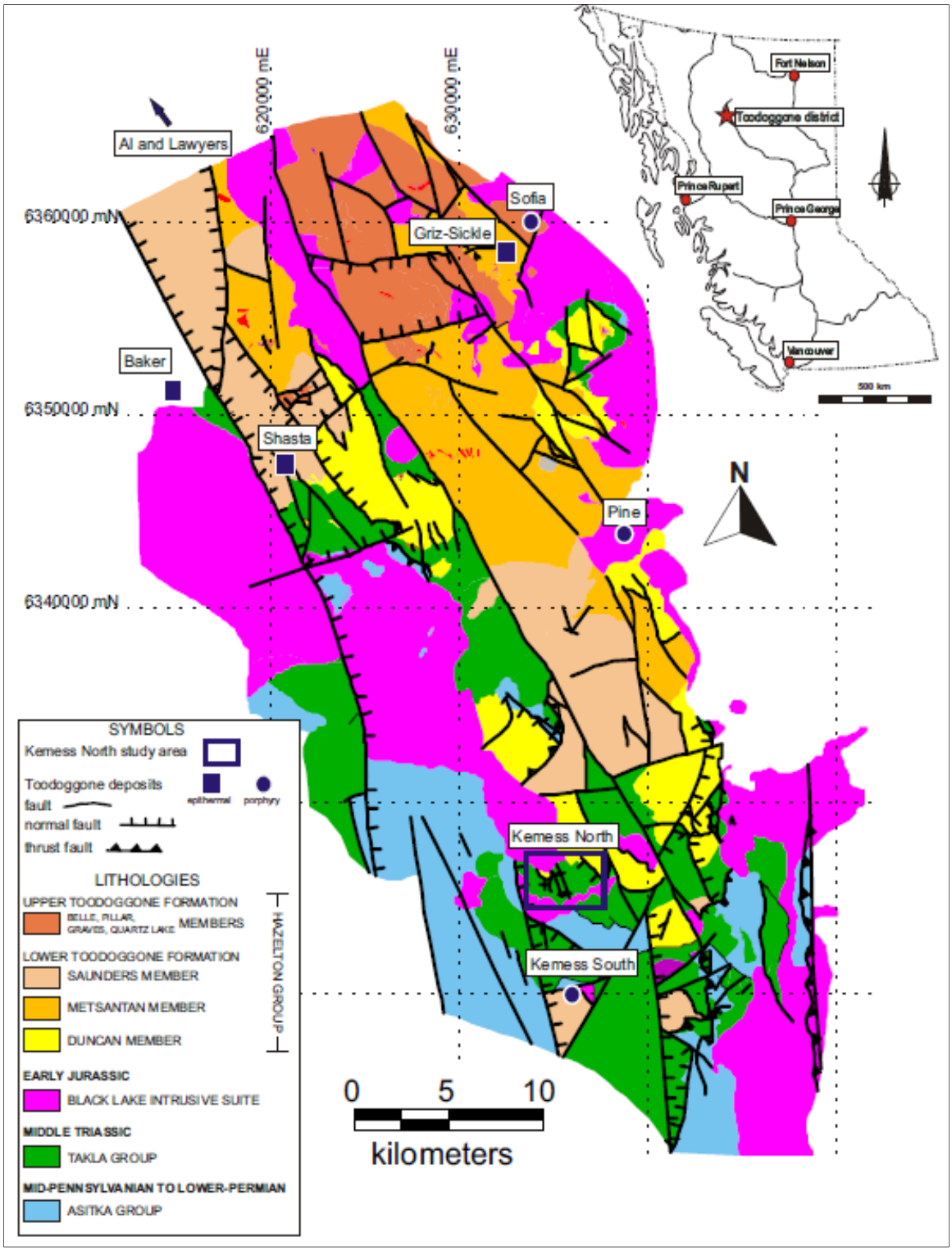


Figure 6.1: Toodoggone district geology

(Source: McKinley, 2006)

6.2 Property Geology

The Kemess Property is host to the Kemess South (KS), Kemess Underground (KUG) and Kemess East (KE) Deposits as well as the Kemess Offset Zone (KOZ) prospect. The property is underlain by the Hazelton Group-Toodoggone formation. The Hazelton Group is host to dacitic fragmental volcanic rocks. The Takla Group volcanics are unconformably below the Hazelton Group and the oldest units on the Kemess property are the Asitka Group volcanics and sediments. The units have been intruded by the Black Lake Suite of Intrusives. Some of these intrusives are the mineralized sources for the Kemess Underground and Kemess East deposits. Later Post Mineral varieties of intrusives cross cut these units and the mineralized intrusives. The KUG deposit is a near surface prophyry deposit with the Kemess East deposit 1km to the east. The Kemess Offset Zone is due east of the KUG deposit and could be the fault offset of the KUG deposit. The area is dominated by horst and graben style normal faulting, south dipping thrust faulting and southwest dipping dip slip faulting. The KUG deposit goes to a depth of 1,000m and going east the Kemess Offset Zone is offset to a greater depth and continues to Kemess East. At the eastern edge of the Kemess East Deposit the mineralized zone shifts up in elevation showing the horst and graben trend. The Kemess East deposit is cutoff to the east by the Kemess East Offset Fault.

6.2.1 Kemess Underground Geology (KUG)

The property is predominantly underlain by a thick (>1,000 m) succession of andesitic flows (Takla Group). The Takla volcanic rocks host a significant portion of the copper-gold mineralization and display phyllic alteration at Kemess North. On the surface is mostly a distinctive feldspar porphyritic unit (bladed feldpsar porphyry – BFP) with some sections of a distinctive augite phyric unit (augite andesite porphyry – AAP). Mantling the northern and eastern limits of the Kemess Underground area is the Hazelton Group-Toodoggone formation. The structurally controlled phyllic sections of this polylithic fragmental dacite in the southeastern area of the deposit can carry anomalous gold concentrations. The evidence suggests that basement structures and conduits that allowed extrusion of the local Toodoggone volcanic assemblage underlie the Kemess Underground area.

6.2.2 Kemess Underground Lithology

The Kemess North pluton beneath the East Cirque hosts the bulk of the copper-gold mineralization for the KUG project. To remain consistent with earlier work, the field term monzodiorite and quartz monzonite has been retained; however, petrographical work shows that the mineralized granitoid underlying the East Cirque is more correctly classified as a quartz diorite due to the paucity (<5% to absent) of alkali feldspars. The Kemess North pluton is an east trending and south dipping intrusion that seems to have occurred along the similar oriented fault structures.

In the 2010, drilling another phase of the Black Lake intrusive suite was identified in the Kemess Underground area. Pre-mineral sills and dykes of hornblende porphyritic diorite are prevalent in the altered Takla volcanic rocks that overlie the Kemess North pluton (not modeled as part of the resource).

Post-mineral dykes, including feldspar porphyry and minor mafic varieties, cross-cut the Takla volcanics and outcrop locally in cirque highwalls and along ridges. The feldspar porphyry dykes

also cross-cut the Jurassic-Toodoggone fragmental unit. Due to the pink colour of the feldspars, these dykes take the field term syenite and are generally barren and unaltered. The Kemess property geology is shown in Figure 6.2.

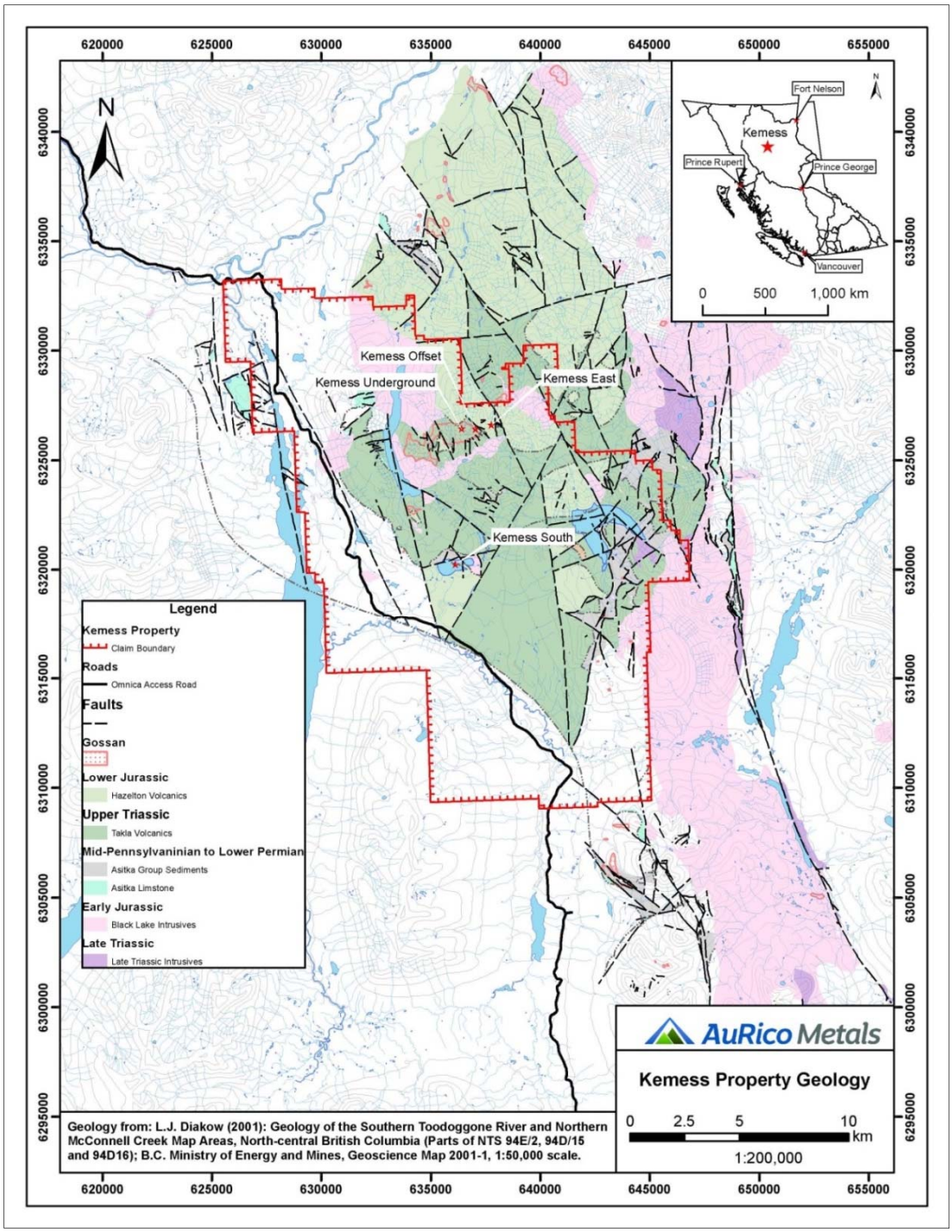


Figure 6.2: Kemess Property Geology

(Source: AuRico, 2016)

6.2.3 Kemess Underground Structure

The most prominent structure traversing the Kemess Underground area is the Kemess North (KN) Thrust Fault, an east-west trending south dipping reverse fault that truncates the Kemess North pluton and associated mineralization at depth. The KN Fault and the Kemess North pluton seem to be related with the KN Fault creating the pathway for the KUG pluton to intrude. Further movement along the KN Fault subsequently truncated the Kemess North Pluton. The KN Fault is complex in that there is a related north dipping secondary structure that demonstrates apparent normal offset with at least 1,000 m of southside up-apparent displacement. To the east of the Kemess Underground area the deposit is cut off by a northwest trending normal fault and north trending steeply dipping normal faults causing eastside down-apparent displacement. These north trending faults are interpreted as the youngest in the Kemess Underground area. To the south of the Kemess Underground area is another west trending fault forming the headwall in East Cirque, as drilling did not reaching adequate depths it is unknown the type of movement and displacement. Fault spacing ranges from 500 to 1,500 m.

6.2.4 Kemess Underground Alteration

A near surface flat-lying zone of intensely broken rock and rubble, historically referred to as the broken zone, occurs above the deposit. This zone is better described as the sulphate leach zone, which more accurately describes the mineral transitions that have occurred, resulting in poor drilling conditions. The zone averages a thickness of about 80 m from the surface to competent bedrock and is comprised of clay, multiple gouge zones, and a pyritic-argillic (clay) alteration component.

The key difference between this alteration unit and those beneath is that there is a distinct ICP detectable calcium depletion which is directly related to the complete absence of any sulphate minerals (gypsum) above this interface. The interface between the sulphate leach zone and the underlying competent phyllic alteration is generally sharp. The post-mineral porphyritic feldspar dykes remain unaltered and competent within the Sulphate Leach zone, indicating that it is likely a feature related to a Jurassic weathering process as opposed to present day weathering.

The phyllic zone underlies the sulphate leach zone and is dominated by very fine grained quartz-sericite-chlorite-pyrite alteration, which is usually pervasive in the upper sector and accompanied by abundant gypsum-pyrite veins. Pyrite content rise with depth, peaking in the 12–15% range and then reducing to 3–4%. The base of the phyllic zone is easily defined as the area where quartz replaces sulphate as the main vein component in the porphyry mineralization.

The potassic zone encompasses the deeper volume of rock mantling the Kemess North pluton. The potassic zone is marked by a decrease in pyrite percentage with an increase in chalcopyrite percentage. The potassic zone consists of increased quartz veining and magnetite enrichment within the quartz veins and / or stringers. The potassic zone was split into potassic weak and potassic strong for the 2016 KUG resource model. The potassic weak zone contained strong chlorite, sericite and weak biotite and quartz alteration with 5-8% sheeted quartz veins. The potassic strong zone contains intense biotite, chlorite and quartz alteration with strong to intense quartz stockwork veining.

6.2.5 Kemess Underground Mineralization

Gold-copper mineralization forms an inclined tabular zone that is centred on the East Cirque (note: not the cirque in which the Kemess East deposit is located) porphyritic monzodiorite, which form structural contours, strikes east-west and dips 20° to the south. The quartz diorite/quartz monzonite intrusive exhibits an irregular upper contact with various peaks and troughs. The general east west strike and shallow south dip geometry is consistent for over 400 m (10,660E to 10,180E). Between 10,260E and 10,160E the tabular morphology disappears and the monzonite occurs as wide dykes (10–100 m) within the Takla volcanics. The change in geometry for the monzonite could be due to the effects of cross faulting which have down dropped the tabular upper contact present in the East Cirque, or the rheologic conditions during intrusion changed going towards the west whereby steep fracture infilling was preferred over stopping.

Alteration and mineralization is associated with and zoned both vertically and laterally from the quartz diorite/quartz monzonite intrusive and its associated dykes intersected at depth beneath the Central and East Cirques. Figure 6.3 shows the mapping of surface alteration in the deposit area.

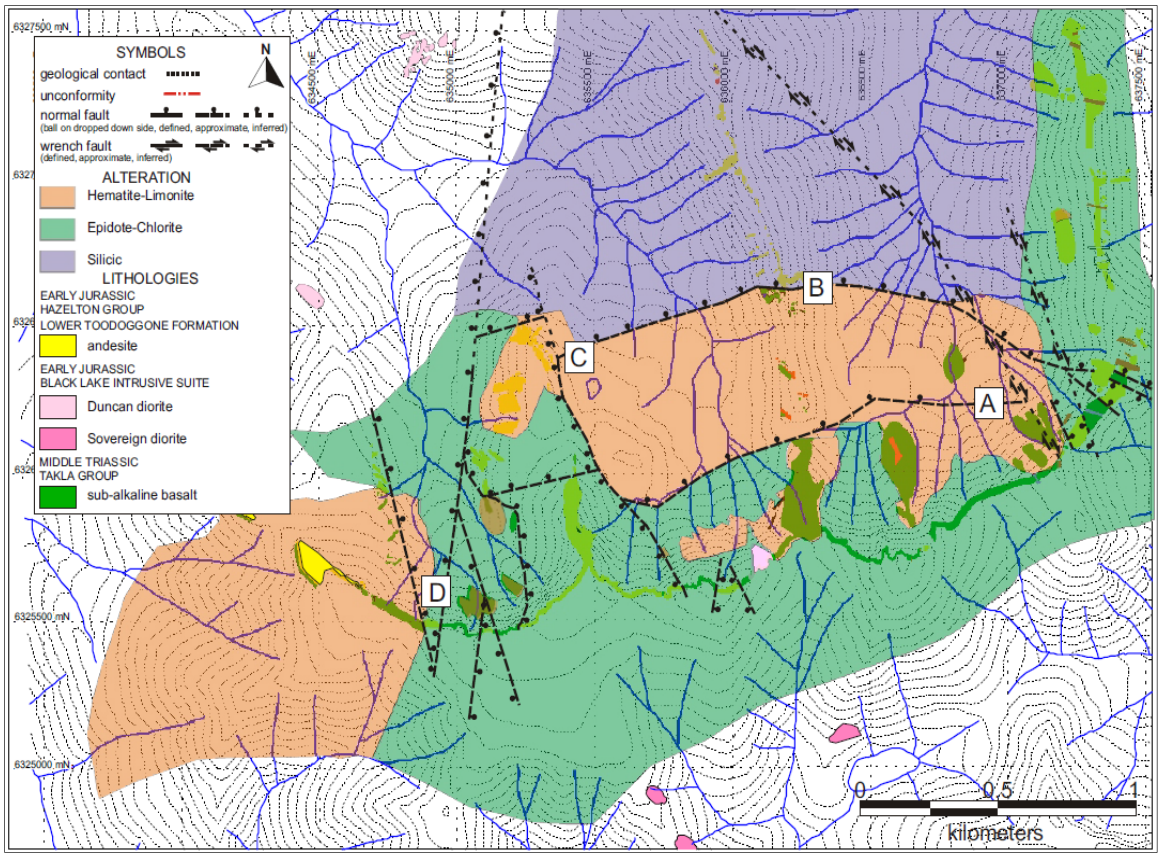


Figure 6.3: Surface alteration mapping at Kemess North

(Source: McKinley, 2006)

The sulphate leach zone, which consists mostly of iron oxides, sericite-chlorite-quartz-pyrite forms an extensive broken zone beneath bright orange-red outcrops at surface (hematite-limonite; McKinley 2006). Pyrite is common throughout (5–7%) as both disseminated and within vuggy quartz veining. This alteration zone is mostly barren of any significant copper and will often show a slight increase in gold with depth. The zone is a result of dissolution of the sulphate and carbonate minerals by highly acidic ground waters present currently and probably during the Jurassic.

Present over the entire area in all rock units except the late mafic dykes are barren pinkish zeolite-carbonate veins, which post-date and crosscut the above vein types and rock units. The zeolite-carbonate veinlets are a low temperature phenomenon. McKinley in 2006 completed a useful paragenetic study of the vein-types which is summarized in Figure 6.4. This vein classification scheme was adopted by exploration and has been in use since 2010.

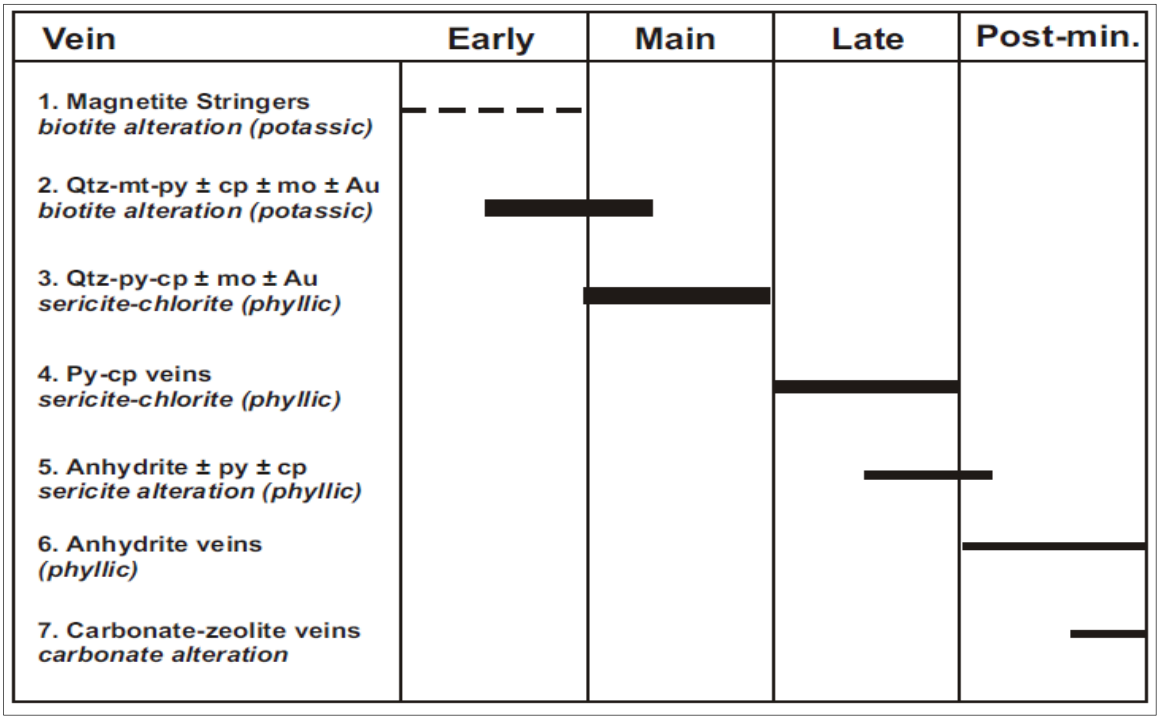


Figure 6.4: Kemess North Vein Paragenesis

(Source: McKinley, 2006)

Overall, sulphide mineralization throughout the deposit consists of 2–3% pyrite, with lesser amounts of chalcopyrite and traces of molybdenum. Pyrite occurs as disseminations, fracture fillings, and veins up to a few centimetres wide generally associated with quartz-gypsum-magnetite veins and zones of quartz-magnetite replacement. The mode of occurrence of chalcopyrite is similar except that veinlets are rare and significant disseminations occur in zones of stronger quartz-magnetite stockwork and quartz-magnetite replacements. Gold and copper grades variably diminish outward into the hangingwall and footwall. Total sulphide content in the core of the deposit averages 3–5%, rising to 10–12% in the phyllic halo.

Petrography shows a varying degree of accessory minerals throughout all rock types and alteration zones including: rutile, leucoxene, sphene, anhydrite, gypsum, epidote, zeolite, alunite, molybdenite, phlogopite, prehnite, and apatite.

6.2.6 Kemess East Geology (KE)

The Kemess East area of interest is underlain by the Hazelton Group-Toodoggone formation. The Hazelton Group-Toodoggone formation extends from surface to 600m. The Hazelton Group is unmineralized and displays weak propylitic alteration. The Hazelton Group is host to dacitic fragmental volcanic rocks. The Hazelton Group displays the same chemical characteristics as the Black Lake plutonic rocks indicating that they are related. In closer contact with the lower Takla Group, fragments of Takla within the Hazelton Group increase towards the unconformity. The structurally controlled phyllic sections of this polylithic fragmental dacite and pyritic veins and silcificed dacitic sections with disseminated pyrite in the southern area of the deposit can carry anomalous gold concentrations. The evidence suggests that basement structures and conduits that allowed extrusion of the local Toodoggone volcanic assemblage underlie the Kemess East area.

Below the Hazelton Group-Toodoggone formation is a succession of andesitic flows that are part of the Takla Group volcanics. The Takla volcanic rocks host a minor portion of the copper-gold mineralization and display propylitic to phyllic to potassic alteration at Kemess East. The weakly to unmineralized portion of the Takla volcanic rocks display propylitic and phyllic alteration. The Takla Group sequences include a feldspar porphyritic unit (bladed feldspar porphyry-BFP), augite porphyry unit (andesite augite porphyry-AAP) and basalts and andesites. The sequence in contact with the overlying Hazelton is not consistent.

6.2.7 Kemess East Lithology

The Kemess East pluton beneath the East Squared Cirque hosts the bulk of the copper-gold mineralization for the Kemess East deposit. To remain consistent with earlier work, the field terms monzodiorite and quartz monzonite have been retained. Early whole rock analysis points to the pluton being diorite in composition. The Kemess East pluton displays strong potassic alteration and transition to phyllic alteration towards the bottom with decreasing copper-gold mineralization.

Below the Kemess East pluton is another section of the Takla Group basalt with strong potassic alteration hosting copper-gold mineralization within quartz veins and stringers. Below this is another Black Lake intrusive unit that has only been intersected in three drill holes. This intrusive unit is below the mineralized Kemess East pluton, but displays similar lithological characteristics. The lower intrusive unit is moderately phyllic to potassic altered with lower grade copper-gold mineralization. The evidence suggests that the Black Lake intrusive plutons are stacked sills and are coeval to each other.

In the 2013-2014 drilling, another phase of the Black Lake intrusive suite was identified in the Kemess East area. This intrusive suite is weakly mineralized and displays strong phyllic alteration. Pre-mineral sills range from 100-200 meters in thickness and their relation to the Kemess East pluton and Kemess Underground pluton are unknown at this time. This phyllic intrusive could represent the eastern portion of the Kemess Underground pluton pinching out.

Post-mineral dykes, including feldspar porphyry and minor mafic varieties, cross cut the Takla volcanics and outcrop locally in cirque highwalls and along ridges. The feldspar porphyry dykes and mafic dykes also crosscut the Jurassic-Toodoggone fragmental unit and the Black Lake pluton in Kemess East. Due to the pink colour of the feldspars, these dykes take the field term syenite and are generally barren and unaltered.

6.2.8 Kemess East Structure

The most prominent structure traversing the Kemess East project area is the Kemess East Offset Fault (KEO), a southeast trending southwest dipping reverse fault that truncates the Kemess East pluton and associated mineralization at depth. To the east of the KEO Fault is a thick sequence of the Jurassic-Toodoggone Hazelton Group predominantly dacitic in composition. The amount of downdrop offset and any strike slip component is unknown at this time because of the lack of drilling at depth. It is thought that the strike slip component could be left lateral as similar Takla Group volcanic rocks are 1,000-1,500 meters north-west of the Kemess East area. Within the Kemess East area there are a few northwest trending faults dipping to the northeast into the KEO fault. These faults are thought to be related to the horst and graben style faulting and the KEO fault. These faults look to be compressional faults uplifting the lithology towards the KEO fault. At the moment two main north-west faults are evident and display 100 meter offsets.

To the west of the Kemess East deposit, the interpreted geology becomes more complex, mainly due to the lack of drilling. The ridge between Kemess Underground area and Kemess East area seems to be a down drop fault block with the Hazelton Group units overlying Takla Group units all the way down, so this could be several normal faults causing the units to be continuously down dropped. This is seen within the alteration as there seems to be stacking events of moderate to strong phyllic and weak potassic alteration.

West of the Kemess East area, is known as the Kemess Offset Zone. This is the area between Kemess Underground and Kemess East and lies beneath the east-west headwall to the south of Kemess East and Kemess Underground. Structural scenarios in the Kemess Offset Zone are vague due to the lack of drilling. Drill holes KN-05-27 and KN-05-28 intersected deep mineralized Black Lake Pluton as seen in Kemess East and at the same depths / elevation.

To the south of the Kemess East area is a steep normal fault that is marked by a steep headwall. This fault has a displacement of 150-200 meters shifting the units up. This uplift is possibly caused by the Black Lake Sovereign unit to the south that post-dates the mineralized Black Lake pluton in Kemess East.

6.2.9 Kemess East Alteration

The alteration within the Kemess East deposit displays broad thick sequences with gradual transitions between alteration zones. The propylitic alteration zone extends from the Hazelton Toodoggone Group into the Takla Volcanic Group. The propylitic alteration displays the typical alteration minerals such as epidote and hematite alteration with carbonate veining.

The phyllic zone within Kemess East is vague but extends over thick intervals. The intensity varies from moderate to strong with increased chlorite and sericite alteration with up to 3-5% pyrite. Sills

and dykes of Black Lake Intrusives display intense phyllic alteration with sericite, chlorite and quartz and 10-15% pyrite, destroying the host rock mineralogy. This is a significant difference to Kemess Underground where there is an upper leached zone with intense sericite, quartz and pyrite alteration and a lower chlorite, sericite alteration zone with abundant gypsum veining.

The potassic alteration zone within Kemess East is primarily within the Kemess East mineralized pluton, and weakly within the Takla Group volcanics on the east portion of the Kemess East area. The potassic alteration is dominated by biotite, chlorite and quartz secondary minerals with minor secondary sericite. Higher grades of copper-gold mineralization are associated with increased biotite and quartz secondary alteration. Quartz veining with mineralization is present in the Kemess East potassic zone, but it is not as dominant as at Kemess Underground which has intense quartz stockwork veining with mineralization.

6.2.10 Kemess East Mineralization

Copper-Gold mineralization forms a relatively flat-lying zone within the Kemess East deposit. Not enough drilling data is present to determine the true orientation of the mineralization and Kemess East pluton. Horst and graben faulting towards the Kemess East Offset Fault has resulted in up lift fault blocks in the copper-gold mineralization. The Kemess East pluton exhibits an irregular upper contact with various peaks and troughs, but has a rough easterly strike and slight shallow south dip. More drilling is needed to define the orientation of the Kemess East pluton.

Alteration and mineralization is associated with, and zoned both vertically and laterally from the Kemess East pluton at depth beneath the East Squared Cirque. At this time a surface alteration map has not been completed. Based on the collar lithologies and alteration it is evident the Kemess East area surface alteration is mostly propylitic.

The phyllic zone consists of pyrite associated with sericite-chlorite alteration. The phyllic zone is barren of significant copper and gold but spikes do occur due to remobilization of mineralization from fluids at depth. Pyrite within the phyllic zone ranges from 3-5% and is dissemined and veined. Minor quartz veining is present within the phyllic zone with pyrite +/- chalcopyrite. This is different to Kemess Underground where there is a sulphate leach zone that forms a broken zone near surface and lower phyllic zone high in gypsum veining. The phyllic zone in Kemess Underground contains significantly more pyrite mineralization than Kemess East.

The potassic zone contains a high percentage of the copper-gold mineralization with an upper zone of molybdenum mineralization. Molybdenum is present in the transition zone from phyllic to potassic alteration and is present with chalcopyrite and pyrite within quartz veins, late stage zeolite and carbonate veins, and within joints. The main copper-gold mineralization is present within the Kemess East pluton and is associated with chalcopyrite and pyrite finely disseminated within the host pluton. To a lesser degree, copper-gold mineralization is associated with quartz veining with chalcopyrite and pyrite. This is different than Kemess Underground where the bulk of the copper-gold mineralization is hosted within the overlying Takla Group and the quartz diorite / quartz monzonite intrusive. The pyrite to chalcopyrite ratio within Kemess East is roughly 1:1 whereas in Kemess Underground it is 3:1. There is also much less magnetite mineralization / alteration within the Kemess East deposit compared with the Kemess Underground deposit.

Present over the entire area in all rock units, except the late mafic dykes, are barren pinkish zeolite-carbonate veins, which post-date and crosscut the above vein types and rock units. The zeolite-carbonate veinlets are a low temperature phenomenon. McKinley in 2006 completed a useful paragenetic study of the vein-types which is summarized in Figure 6.5. This vein classification scheme was adopted by exploration and has been in use since 2010. Further studies are required to see if this applies to Kemess East.

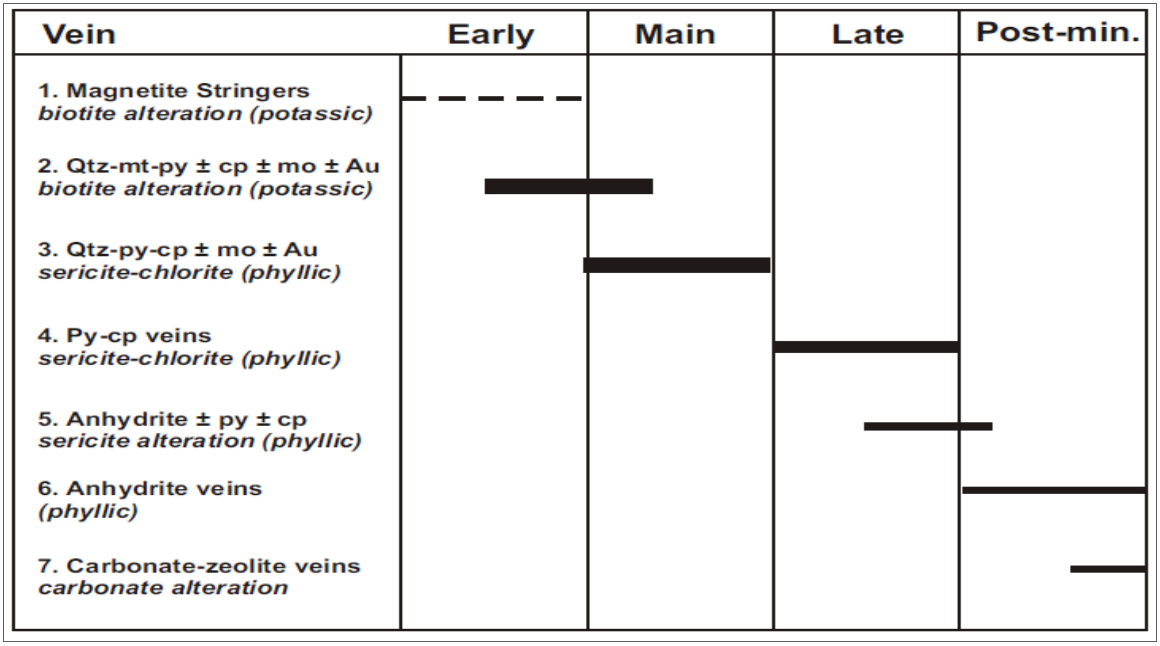


Figure 6.5: Kemess Property Vein Paragenesis

(Source: McKinley, 2006)

6.2.11 Kemess Offset Zone Geology (KOZ)

At surface are dacitic fragmental units of the Hazelton Group which unconformably overlay the Takla Volcanics units. The Takla Volcanics have been intruded by Black Lake Monzonite to Quartz Monzonite. Late Post Mineral Dykes are present that are steeply dipping to the northeast. These post mineral dykes cross-cut all the units and range from <1m to 10m in thickness. Alteration in the Kemess Offset Zone (KOZ) is similar to KUG except a pronounced propylitic zone is evident within the Hazelton group of rocks and minorly within the upper Takla Volcanics. The phyllic zone contains strong to intense quartz, sericite and pyrite alteration with the lack of a calcium leach zone and gypsum zone. These zones appear to be present, but with much weaker intensity compared with KUG. The potassic zone shows an increase in chlorite, biotite and silica alteration with an increase in quartz and magnetite veining. The potassic zone hosts most of the gold, copper, silver and molybdenum mineralization. Similar to KUG, the mineralization in the KOZ is hosted within the Takla Volcanics and the Black Lake Intrusive. Mineralization is ether disseminated, within stringers and veins with mostly pyrite, chalcopyrite and molybdenum. Gold is associated with the chalcopyrite.

7 Deposit Types

7.1 Kemess Underground Deposit Type

Kemess North is a copper-gold porphyry deposit and is typical of calc-alkaline porphyry copper-gold deposits in the western cordillera. The deposit has a low-grade ore zone at a depth of 150 m below the surface on its western flank and a higher grade zone 300–550 m below surface on the eastern side, which forms the Kemess Underground project. Kemess Underground is hosted by potassic altered Takla Group volcanic rocks and Black Lake plutonic rocks. The deposit is centered on a mineralized porphyritic monzodiorite/diorite pluton and associated WSW trending dykes, which extend to the southwest. Higher-grade copper-gold mineralization is characterized by secondary biotite alteration in volcanic and the eastern plutonic host rocks.

Porphyry style copper-gold mineralization occurs within the Takla volcanic rocks and intermediate intrusive rocks associated with weak to pervasive propyllitic, phyllic, and potassic (biotitic) alteration assemblages. The latter is associated with higher copper and gold grades. Alteration of Toodoggone assemblages ranges from fresh to weak propyllitic and is generally barren of significant sulphides and ore grade mineralization.

7.2 Kemess East Deposit Type

Kemess East is a copper-gold-silver-molybdenum porphyry deposit and is typical of calc-alkaline porphyry copper-gold deposits in the western cordillera. The deposit is deeply buried and mineralization starts at an average depth of 900 m below surface and extends to 1500 m below surface. Unlike Kemess Underground, there is no significant low grade mineralization associated with Kemess East. At Kemess East there is reasonable continuity of mineralization within the deposit. Kemess East is mostly hosted by potassic altered Black Lake plutonic rocks. In the eastern portion of the deposit, weak mineralization is hosted within potassic altered Takla Volcanics, but still largely within the Black Lake plutonic rocks. From whole rock analysis, it is evident that the deposit is centered on a mineralized porphyritic diorite pluton. This is in contrast to the Kemess Underground deposit which is centered on a mineralized porphyritic monzondiorite/diorite pluton. Due to the present state of drilling, it is difficult to gauge the orientation of the host diorite, but presently it appears to be nearly flat-lying and slightly dipping to the south. The alteration within the mineralized zone is characterized by secondary chlorite with lesser secondary biotite and quartz within the plutonic host rocks. Higher grade copper-gold mineralization is characterized by strong secondary biotite and quartz alteration and lesser chlorite alteration in the plutonic rocks. Higher grade copper-gold mineralization is also characterized by a slight increase in quartz veining and an increase in chalcopyrite to pyrite ratios.

Porphyry style copper-gold mineralization occurs within the Takla volcanic rocks and intermediate intrusive rocks associated with weak to pervasive propyllitic, phyllic, and potassic (biotitic) alteration assemblages. The latter is associated with higher copper and gold grades. Alteration of Toodoggone assemblages range from fresh to weak propyllitic and are generally barren of significant sulphides and ore grade mineralization.

8 Exploration

8.1 Kemess North/Kemess Underground Exploration

The Kemess North property represents an advanced project that has previously been through feasibility studies during its consideration as an open pit project. It was only in 2010 that a detailed review as an underground project began.

The early exploration work in the area identified a porphyry target, but it wasn't until deep drilling in 2001 that significant gold and copper grades were located. Since 2001, exploration has been directed at expanding the resource base in what was historically viewed as an open pit project. Table 8.1 outlines the number of holes drilled in the greater Kemess Underground area since 1976.

Table 8.1: Kemess North / Underground Drill Campaign Summary

Year	No. of Holes	Total Metres
1976	2	507.8
1989	5	731.8
1990	7	1,185.0
1991	4	1,111.9
1992	9	2,034.8
2000	7	3,345.2
2001	16	8,385.5
2002	33	18,694.2
2003	17	6,414.1
2004	8	3,027.5
2005	3	2,074.3
2006	6	3,251.3
2010	29	16,394.1
2011	3	2,206.8
Total	**149**	**69,364.3**

Because the target is deep, surface geological mapping and surface geochemical techniques add little value. Likewise, surface and airborne geophysical exploration have contributed little. Since the last work by El Condor in 1992, there has been no surface soil or rock sampling or trenching at Kemess Underground. Surface work has been confined to access roads and drill site construction. A regional airborne geophysical program completed in 2003 added to the understanding of the volcanic and intrusive events in the area. The program had little impact on the resource estimate for the Kemess Underground project.

The procedures followed in the field and through the interpretation stage of exploration have been supervised by professional geologists. Various crews under the supervision of professional geologists carried out the exploration work (Table 8.2). From 2001 to the present day, there has been continuity in personnel both in the field, in the laboratories, and with the data interpretation.

Table 8.2: Kemess North / Underground Exploration Employee / Contractors

Job function	Supervisor	Contractors
A. Geology		
2001	Hibbitts (NGX)	A. Bray
2002	Hibbitts (NGX)	B. La Peare B. Mercer E. Ramsay J. Mazvihwa C. Edmunds
2003	C. Edmunds (NGX)	M. Russer B. Kay R. Konst A. Tsaloumas L. Lindinger
2004	C. Edmunds (NGX)	M. Russer B. Kay R. Konst R. Brown B. McKinley
2005	C. Edmunds (NGX)	M. Russer B. Kay R. Konst B. McKinley
2006	C. Edmunds (NGX)	K. Lucas B. Kay R. Konst W. Barnes
2007	C. Edmunds (NGX)	B. Kay R. Konst W. Barnes
2010	C. Edmunds (NGX)	R. Konst A. Anwar W. Barnes S. Heard
2011	W. Barnes (AUQ)	K. MacWilliam J. Quan M. Toudeh-Kharman
B. Laboratory		
2001		Bondar-Clegg
2002–2011		ALS Chemex
C. Drilling		
2001–2005		Britton Brothers
2006		Suisse/Hy-Tech
2007		Hy-Tech
2010		Driftwood
2011		Hy-Tech

8.2 Kemess East Exploration

The Kemess East discovery is an early stage exploration project located one kilometer east of the Kemess Underground deposit and 6.5 km north of the Kemess mill facility.

Prior to 2005, regional exploration was restricted to shallow targets on the Kemess Property. In 2005 the discovery of deep mineralization immediately east of the Kemess Underground deposit, the Kemess Offset Zone (KOZ), led to subsequent geophysical surveys and drilling programs targeting deep mineralization. Table 8.3 presents the number of holes drilled in the greater Kemess East area from 2002 through 2014. Drilling prior to 2005, and some holes in 2006 and 2007 were too shallow to intersect any significant mineralization

Table 8.3: Kemess East Drill Campaign Summary

Year	No. of Holes	Total Metres
2005	3	779.7
2006	10	4,756.8
2007	27	17,712.6
2013	7	10,341.0
2014	12	16,872.5
2015	12	17,872.4
Total	**71**	**67,881.0**

Because the target is deep, surface geological mapping and surface geochemical techniques add little value. Likewise, surface and airborne geophysical exploration have contributed little. Since the last work by El Condor in 1992, there were no surface soils or rock sampling or trenching programs at Kemess East. Surface work has been confined to access roads and drill site construction. A regional airborne geophysical program, completed in 2003, added to the understanding of the volcanic, intrusive and structural events in the area. In 2006, Northgate Minerals contracted Quantec Geoscience to do several line kilometers of geophysical surveys over the Kemess Underground and Kemess East areas. The results from this geophysical work conducted by Quantec, identified a deep mineralized zone in Kemess East hosting copper-gold-silver-molybdenum and outlined the Kemess East Deposit (Figure 8.1). During the 2014 exploration program AuRico contracted Geotech Airborne to conduct a ZTEM survey over most of the Kemess Property. The results from the survey confirmed the results from the Quantec geophysical survey and identified more detail to the Kemess East and Kemess Underground deposits.

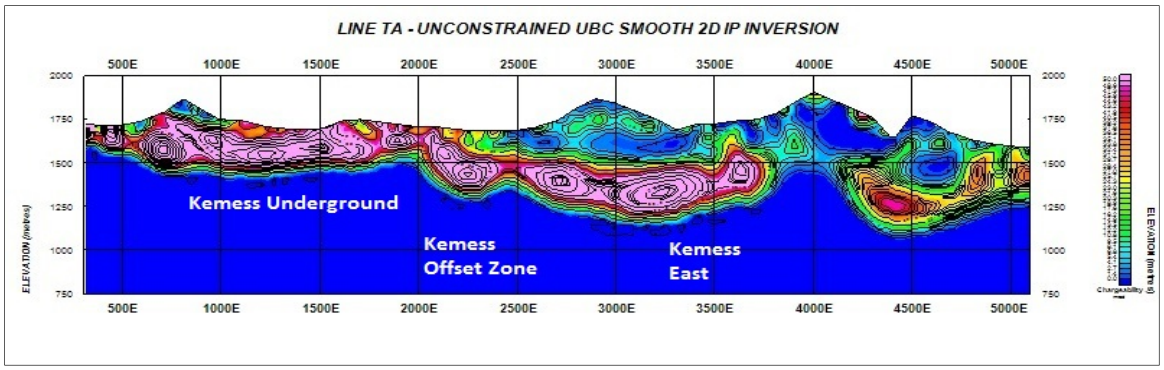


Figure 8.1: Kemess Underground to Kemess East Quantec Survey

(Source: SRK, 2013)

Drilling at Kemess was terminated in 2007 after the Kemess North Joint Review Panel negative recommendation for the permitting of the Kemess North Open Pit Mine. In 2010, the Kemess North Deposit was revisited as an underground operation. After a positive feasibility study on the Kemess Underground Deposit in 2013, AuRico re-started exploration and drilling programs at Kemess. This led to the discovery and definition of the copper-gold-silver-molybdenum Kemess East porphyry.

The procedures followed in the field and through the interpretation stage of exploration have been to industry standards. Various crews under the supervision of professional geologists carried out the exploration work (Table 8.4). From 2001 to the present day, there has been continuity in personnel in the field, in the laboratories, and with the data interpretation.

Table 8.4: Kemess East Exploration Employee / Contractors

Program	Supervisor	Contractors
A. Geology		
2005	C. Edmunds (NGX)	M. Russer B. Kay R. Konst B. McKinley
2006	C. Edmunds (NGX)	K. Lucas B. Kay R. Konst W. Barnes
2007	C. Edmunds (NGX)	B. Kay R. Konst W. Barnes
2013	W. Barnes (AUQ)	R. Konst B. Kay Q. Harper N. Skibinski
2014	W. Barnes (AUQ)	R. Konst Q. Harper S. Dyck N. Skibinski T. Stridsland

Program	Supervisor	Contractors
2015	W. Barnes (AUQ/AMI)	Q. Harper B. Kay S. Dyck H. Mills E. Miller
B. Laboratory		
2006		Assayers Canada
2005–2015		ALS Chemex
C. Drilling		
2005		Britton Brothers
2006		Suisse/Hy-Tech
2007		Hy-Tech
2013-2015		Driftwood

8.3 Kemess Offset Zone Exploration

The Kemess Offset Zone (KOZ) is located directly east of the KUG deposit. The KOZ is assumed to be the fault offset of the KUG deposit or could possibly be the extension of the KE deposit. At the eastern edge of the KUG deposit is a steeply dipping N-S normal fault causing the unit to be faulted down in a graben like structure. This is seen going from west to east from the KUG deposit to the KE deposit. Not much drilling has been done to test the KOZ, with most of the historic drilling too shallow to intersect the KOZ mineralization. In 2005 three drill holes intersected the KOZ but the grades were not significant enough to follow up with, due to low metal prices. Deeper drilling was conducted again in 2013 and 2015 in the KOZ due to geophysical evidence from the Quantec Titan Geophysical survey and the ZTEM Airborne survey displaying evidence of an extension of KUG and/or KE at depth. In the 2015 drilling campaign four drill holes specifically targeted the KOZ with one hole abandoned near surface due to poor drilling conditions. The best drill result was from KH-15-06A which intersected 817.5m grading 0.273g/t Au, 0.216% Cu, 1.43 g/t Ag and 0.006% Mo from 459.0m depth. This interval included 81.5m grading 0.437 g/t Au, 0.292% Cu, 1.44 g/t Ag and 0.010% Mo. The KOZ mineralization is hosted in similar host rocks as KUG and KE. Table 8.5 outlines the number of holes drilled in the greater Kemess Offset area since 2002.

Table 8.5: Kemess Offset Zone Drill Campaign Summary

Year	No. of Holes	Total Metres
2002	1	429.0
2003	1	397.1
2005	3	3,346.1
2006	4	2,063.0
2013	3	4,692.8
2015	4	4,433.2
Total	**16**	**15,361.1**

9 Drilling

The following section describes the drilling programs that have been conducted on the Kemess property. Figure 9.1 displays the drillhole traces for the Kemess Underground, Kemess Offset Zone, and the Kemess East zones of mineralization.

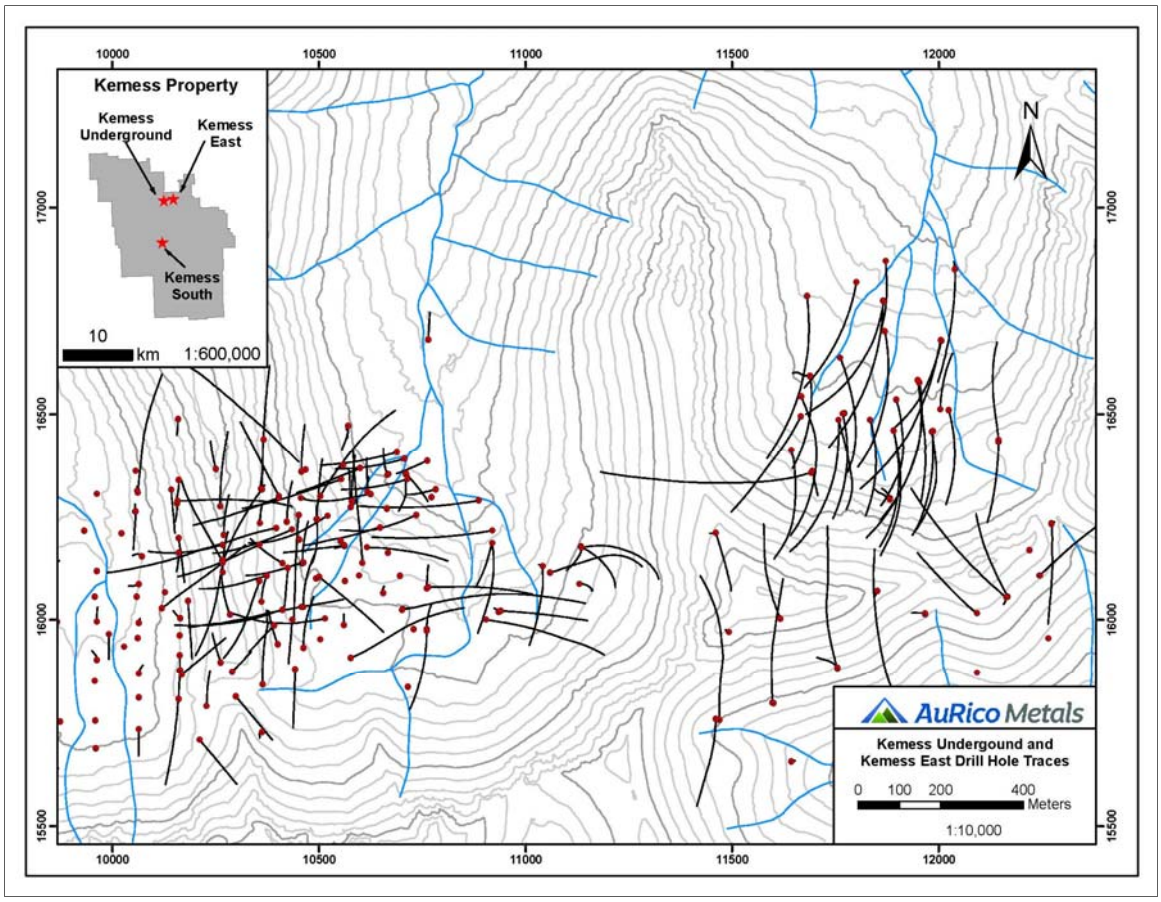


Figure 9.1: Kemess Underground and Kemess East Drillhole locations

(Source: AuRico, 2016)

9.1 Kemess North/Underground Drilling

Since May 2000, there have been ten summer drill programs completed in the Kemess North/Underground area. Various diamond drilling contractors based out of Smithers, BC completed this work.

At peak activity, up to four drill rigs were used on the property, three on skids and one helicopter portable. Three Hy-Tech diamond drills completed the most recent program in 2011, which accounted for 19 drillholes and 6,169 m. The drilling statistics and summaries that are listed above in Table 8.1 are based on drilling programs completed up to and including the 2010 program.

The KUG project area is approximately 1,050 m in an east-west direction and 610 m north-south and over 600 m vertical. For the most part, the drillhole spacing is less than 100 m and became quite well covered with the additional 2010 and 2011 holes.

The current KUG resource database contains 149 drillholes for a total of 69,364.31 m and an average length of 465.53 m, with the majority in the 200–600 m range. There are a few short holes less than 100 m, while the deepest hole is 1,206 m.

The broken zone, which presents challenging drilling conditions, covers much of the project. Historically, drilling an HQ diameter hole (64 mm core) to act as a casing for NQ (48 mm core), was used to complete the hole, solving the problem. In rare instances reduction to BQ (37 mm core) was necessary to reach target depth. The core recovery is very high with an average of ~70% in the broken zone and approximately 100% in the remainder.

In 2004, a test was conducted to compare assay results from holes with steep angles to holes with shallow angles. At that time, 29 holes were drilled at shallow angles (less than –60º) so that oriented core could be obtained to assist with the geotechnical program. It was found that there is no significant grade variation between the two data sets. Because the shallow angle holes tested various different directions, it appears likely that there is no preferred vein orientation in the deposit that could be missed with steep drilling.

9.1.1 Kemess Underground Procedures

The procedures used to locate exploration drillholes are as follows: the proposed drill site is located in the field by a geologist using a hand-held GPS unit, then the site is built using an excavator and the drill rig is pulled onto the site by a bulldozer and a sump for cuttings is constructed. The orientation of the drillhole is set by the geologist with a set of GPS placed pickets to provide the azimuth and specify the inclination of the hole. Because of the depth of the mineralization, most of the drillholes have been drilled at steep angles, approaching vertical. Exceptions to this rule are holes drilled for geotechnical studies. The majority of the 2010 program was drilled on a 255° azimuth inclined at 65°. There are 123 holes drilled at dips between 45° and 75°, with various azimuth orientations. The remaining 207 holes have been drilled at steep angles, greater than 75°.

All of the drilling on the project has been continuous core diamond drilling. Because of the broken zone at the surface, drilling procedures include setting surface casing then drilling with large diameter HQ core. Once through the broken zone and into more solid rock, the drillers generally reduced to NQ core to complete the hole. Upon completion of the hole, all of the drill pipe is removed from the hole, though the surface casing is left to mark the hole location.

During the 2010 campaign most of the holes were either cemented with coaxial cable along their entire length for a cavity monitoring system, or they were cased with plastic pipe to facilitate future televiewing systems once underground mining becomes operational. To facilitate the installation of both these systems, the HQ drill pipe was left in the ground.

9.1.2 Kemess Underground Sample Length / True Thickness

Sample length was determined by the geology of the deposit and an attempt was made not to allow samples to cross lithological boundaries. With NQ size drill core, sample lengths were generally 2 m, while with HQ core, sample lengths were reduced to 1.5 m. The majority (66%) of the assay intervals are 2 m in length, with the remainder mostly less than 2 m. The average assay interval is 1.92 m.

The term “true thickness” is not generally applicable to porphyry deposits as the entire rock mass is potentially mineralized material and there is no readily apparent preferred orientation to the mineralization. Because of the potential for mineralized material through the entire length of the hole, sampling was generally continuous from top to bottom of drillholes. The mineralization is generally confined to three main lithologies, Black Lake pre-mineral monzonite, Takla BFP (bladed feldspar porphyry), and Takla volcanics. These lithologies form large massive bodies underlying the central and eastern cirques. The mineralization is generally flat lying within the various lithologies. The mineralized Takla volcanic and western Black Lake intrusive rocks show generally vertical contacts. The higher-grade Kemess North pluton in the East Cirque appears to be an inclined tabular body, dipping 20° to the south and truncated to the north by a south dipping fault.

9.1.3 Kemess Underground Collar Survey

Survey control for the drillhole collars was by GPS using a base station that provided real-time correction, such that sub-centimeter accuracy was achieved. Precise collar locations for some of the 2011 drilling have yet to be confirmed and presently exist in a database with collars known to +/-10m of actual. The three drillholes completed to confirm the geotechnical properties of Kemess North were verified later on by GPS using a base station with sub-centimeter accuracy. A plan showing the drillhole collars and traces in the Kemess Underground area is shown in Figure 9.2.

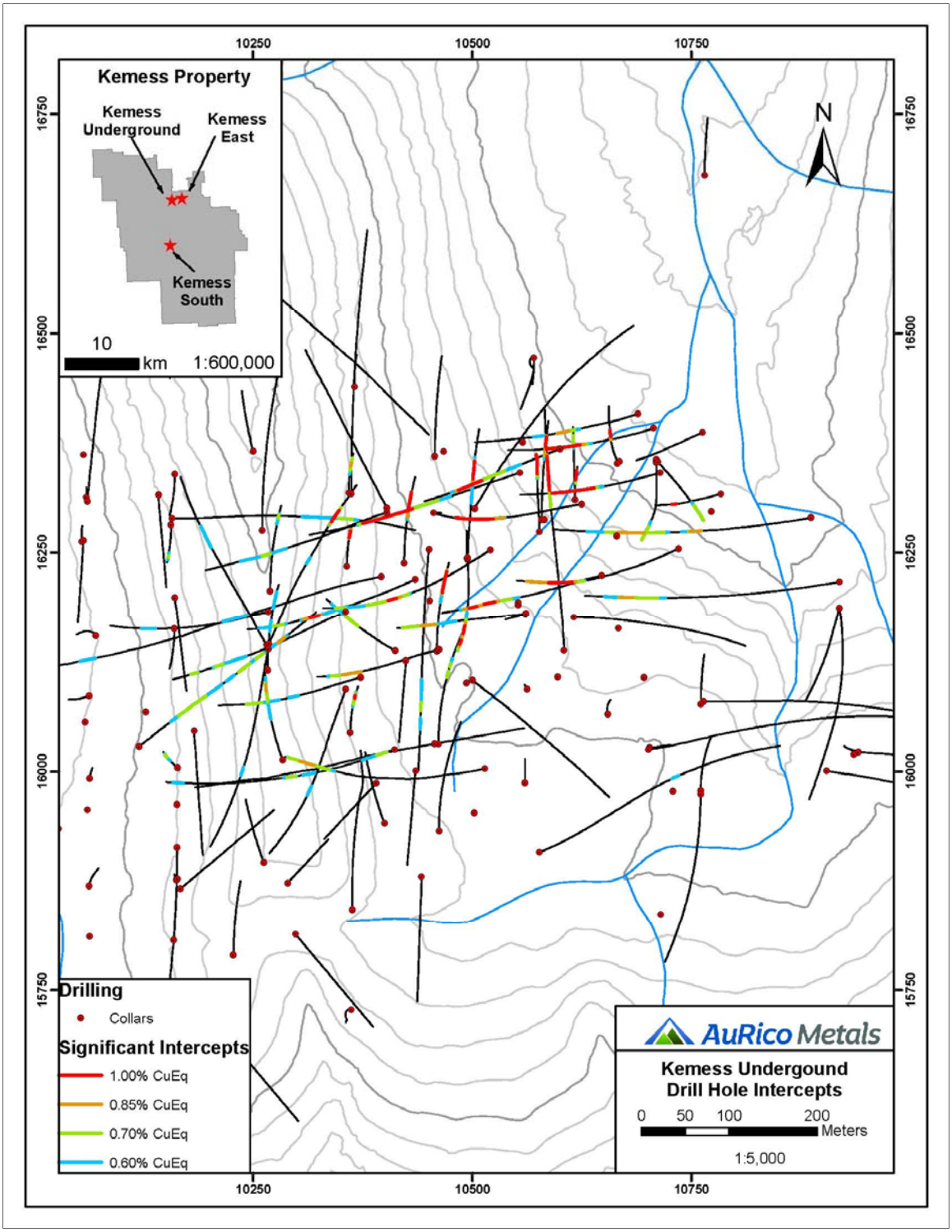


Figure 9.2: Kemess Underground Drillhole locations

(Source: AuRico, 2016)

9.1.4 Kemess Underground Downhole Survey

Differentially corrected GPS provided survey control for all collars, while a variety of down-hole survey techniques have been used to estimate the sub-surface trajectory of the drillholes.

The 2010 program relied upon non-magnetic methods such as Flex-It electronic gyro and Deviflex to survey the down-hole traces. All 2010 drillholes showed a consistent deviation to the right with minor steepening with depth. This result confirmed observations made from high resolution corrected magnetic instruments, such as the Flex-It, which was in use from 2003 through to 2008. Pre-2003 programs were surveyed using Sperry-Sun magnetic based instruments, and many of these holes showed anomalous hole traces sometimes increasingly deviating to the left with depth. A total of 34 holes have been identified with anomalous hole traces, including seven from the post-2002 period, which are listed in Table 9.1.

It is considered highly unlikely that the pre-2003 holes had subterranean trajectories that differed significantly from the 2010 experience, as people, machinery, and equipment specifications are very similar to those in use in 2000.

Table 9.1: Kemess Underground Drillholes with adjusted downhole survey data

Hole ID					
KN-00-08	KN-01-03	KN-02-03	KN-02-09	KN-02-25	KN-04-07
KN-00-10	KN-01-04	KN-02-04	KN-02-11	KN-02-54	KN-05-22
KN-00-11	KN-01-06	KN-02-05	KN-02-13	KN-02-57	KN-06-09
KN-00-12	KN-01-07	KN-02-06	KN-02-19	KN-03-16	KN-06-10
KN-01-01	KN-01-12	KN-02-07	KN-02-20	KN-03-20	
KN-01-02	KN-02-01	KN-02-08	KN-02-22	KN-04-01	

Using the 2010 down-hole survey data as a benchmark, it was decided to adjust the surveys for the 34 holes listed above using an average of best-fit 2010 trajectories. All surface and solid modelling was done using the database with adjusted surveys for these 34 holes.

9.2 Kemess East Drilling

Kemess East wasn't a primary focus of exploration during delineation of Kemess Underground, with the periods 2006-2007 and 2013-2015 being the main years for drilling programs at Kemess East. Prior to 2006, minor regional based drilling programs took place with select drilling in the vicinity of Kemess East. For example, in 2005 drilling intersected the Kemess Offset zone at depth adjacent to Kemess Underground. Various diamond drilling contractors based out of Smithers, British Columbia completed this work. Three Driftwood diamond drills completed the most recent program in 2015, which accounted for 12 drillholes and 17,418 m with all holes reaching target depth. The Kemess East exploration area, one kilometer east of the Kemess Underground Deposit, is approximately 1 km in an east-west direction, 1 km north-south, and over 1 km vertical. Drillhole spacing is 75 m to 200 m. The entire Kemess East database contains 71 drillholes for a total of 67,881 m with an average length of 956.1 m and the deepest hole reaching 1,703 m. However, almost half of the holes are less than 700 m in depth while the vast majority of significant intercepts

start below 700 m. Of the 71 drillholes in Kemess East, 35 drillholes were drilled deep enough to test the mineralized zone.

The broken zone, which covers much of the Kemess Underground area and presents challenging drilling conditions, is not present at Kemess East. Core recovery is very high with an average of 95%.

9.2.1 Kemess East Procedures

Proposed exploration drillhole sites were located in the field by a geologist using a hand-held GPS unit, then the site was built using an excavator and the drill rig pulled onto the site by a bulldozer and a sump for cuttings was dug. Helicopter portable drill rigs were setup up on timber pads and cuttings were filtered down the scree off the mountain sides. Up until 2013, the orientation of the drillhole was aligned by the geologist with a set of GPS placed pickets to provide the azimuth and specify the inclination of the hole. In 2013 a Reflex North Finder APS® (Azimuth Pointing System), a GPS based compass, was used to layout drillhole azimuths. In 2014, drillhole collar locations were positioned by a Trimble GPS with an RTK base station and azimuths were laid out using a Downhole Surveys Azi Aligner®, a gyro based compass, in order to acquire true north azimuth independent of the local magnetic field. In 2015, drillhole collar locations were positioned by a Trimble GPS with an RTK base station, azimuths were laid out using a Reflex TN14 Gyrocompass, in order to acquire true north azimuth independent of local magnetic field. Cross-checks were made with hand-held GPS for location and with a compass for azimuth. Final drill setups were checked by a geologist and dips confirmed with an inclinometer in 2013. In 2014 the drill setups were checked with the Downhole Surveys Azi Aligner® and in 2015 the drill setups were checked with the Reflex TN14 Gyrocompass.

Approximately a third of the holes were drilled vertical or near vertical while the remainder ranged from -45 to -85 degrees and averaged -70 degree dip. Inclined holes were drilled to allow for the use of drill core orientation tools and to target in difficult terrain. Oriented core drilling has been completed on the 2013-2015 drill programs. A variety of azimuths were also drilled due to the depth of the mineralized zone, the terrain, and to target the position of major structures.

All of the drilling on the property has been continuous core diamond drilling. Because of the depth of holes, drilling procedures included setting HW surface casing then drilling large diameter HQ core from 2013 onwards. Once to a depth of approximately 450 m, the drillers reduced to NQ core to complete the holes. Pre 2013 drilling procedures included setting HW surface casing then drilling with NQ diameter core. Upon completion of a hole, all of the drill pipe was removed from the hole, while the HW surface casing was left in to mark the hole location. The majority of casing has been capped and marked with the hole ID.

9.2.2 Kemess East Sample Length / True Thickness

Sample length was determined by core size and by the geology of the deposit, as an attempt was made not to allow samples to cross lithological boundaries. With NQ size drill core, sample lengths were generally 2 m, while with HQ core sample lengths were reduced to 1.5 m. The majority (70%)

of the assay intervals are 2 m in length, with the remainder mostly less than 2 m. The average assay interval is 1.84 m.

The term ‘true thickness’ is not generally applicable to porphyry deposits because the entire rock mass is potentially mineralized material. Normally there is no readily apparent preferred orientation to the mineralization. Because of the potential for mineralized material over the entire length of the hole, sampling was generally continuous from top to bottom of drillholes except for large barren sections of overlying Hazelton volcanics. The mineralization is generally confined to two main lithologies; hypogene Black Lake pre-mineral monzonite and Takla volcanics. These lithologies form large massive bodies underlying Kemess East and the mineralization is generally very thick and interpreted as flat lying. Due to minimal drilling of the mineralized zone in Kemess East, there is limited understanding of the shape of the orebody and the influence of faulting leave room for error in this ‘flat lying’ interpretation. In a worst case scenario, the true thickness of the mineralized intervals could be as low as 80% of some of the reported intercept lengths.

9.2.3 Kemess East Collar Survey

Survey control for the 2006-2015 drillhole collars was conducted utilizing a Trimble GPS with an RTK base station that provided real-time correction, such that sub-centimeter accuracy was achieved. Collar coordinates are recorded in both UTM WGS84/Nad83 Datum and a local true north coordinate system in the database. A plan showing the drillhole collars and traces in the Kemess East area is shown in Figure 9.3.

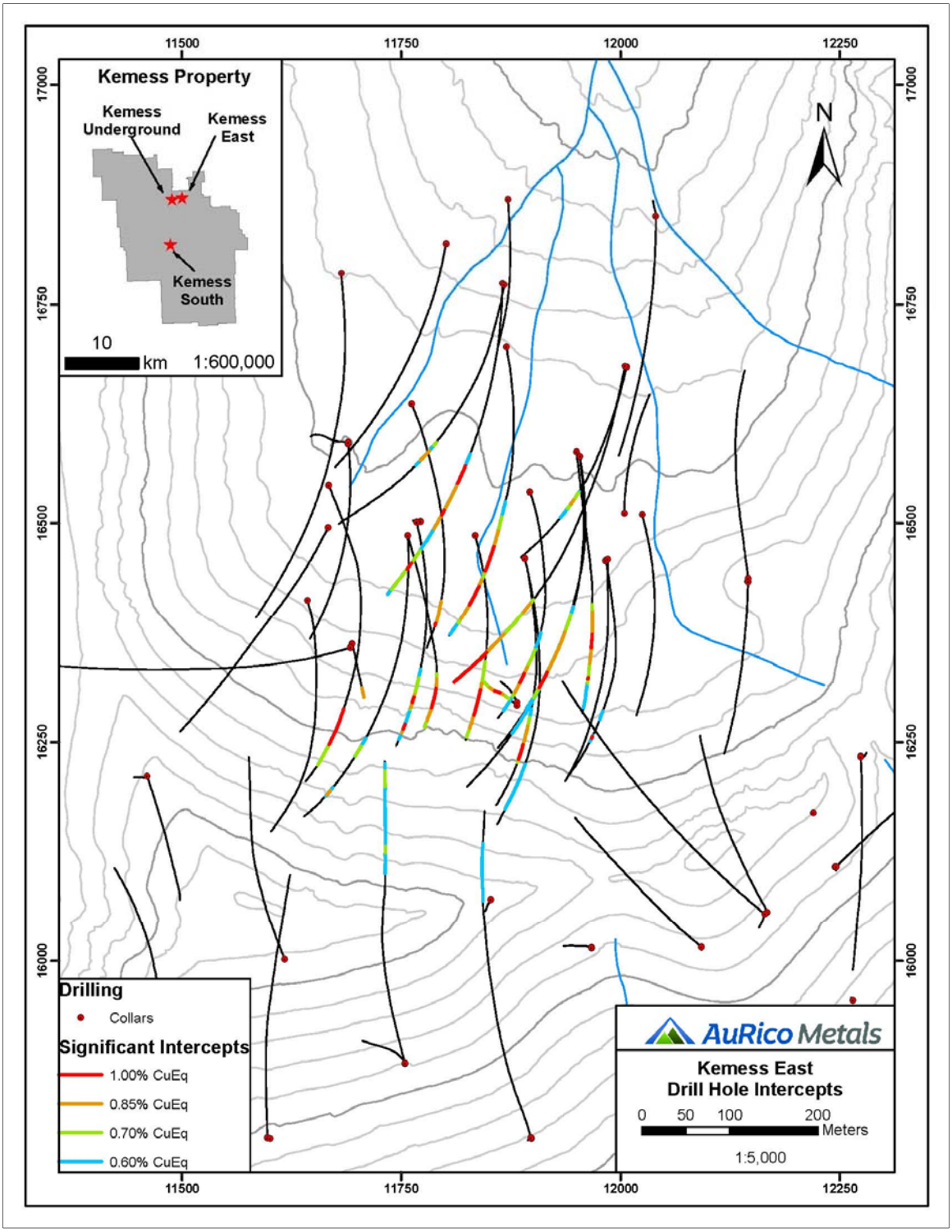


Figure 9.3: Kemess East Drillhole locations

(Source: AuRico, 2016)

9.2.4 Kemess East Downhole Survey

Differentially corrected GPS and GPS or gyro based compasses provided survey control for all collars, which was integrated with a variety of multi-shot down-hole survey instruments used to estimate the sub-surface trajectory of the drillholes. All holes drilled during the 2013-2015 programs were surveyed with a non-magnetic down-hole survey instrument to model changes in azimuth and dip. The surveys were performed by a Devico DeviFlex™ or a Reflex® Gyro and, in some cases, both. DeviFlex™ survey points were taken every 4 m and Gyro surveys were completed at 5 m or 10 m intervals in each hole at 50% and 100% hole completion. This non-magnetic data was cross-checked with a DeviTool™ PeeWee or Flex-IT™ magnetic multi-shot down-hole survey instrument with sample stations at various increments. The validity of magnetic survey data was determined based on local magnetic variation measurements provided by the tool.

All Reflex® Gyro non-magnetic 2013-2015 drillhole surveys show a consistent deviation to the right with very minor steepening with depth. This result confirms observations made from a high resolution corrected magnetic instrument, the Flex-It™, which was in use exclusively from 2005 through 2007. Pre-2005 programs were surveyed using Sperry-Sun™ magnetic based instruments.

Some Devico DeviFlex™ non-magnetic and Flex-IT™ magnetic multi-shot down-hole surveys showed anomalous hole traces. The comparison was made using the consistent down hole survey deviations observed with the Reflex® Gyro non-magnetic instrument. Errors in the Devico DeviFlex™ surveys are thought to be manmade and/or instrument based. The errors in the Flex-IT™ surveys are believed to be due to the magnetic nature of the country rock. A list of drill holes with anomalous hole traces is presented in Table 9.2.

Table 9.2: Kemess East Drillholes with adjusted downhole survey data

Hole ID	Survey Method	Reason For Removal
KH-07-04	FlexIT	Questionable results from magnetic FlexIT Survey
KH-07-07	FlexIT	Questionable results from magnetic FlexIT Survey
KH-07-11	FlexIT	Questionable results from magnetic FlexIT Survey
KH-07-16	FlexIT	Questionable results from magnetic FlexIT Survey
KH-07-17	FlexIT	Questionable results from magnetic FlexIT Survey
KH-07-18	FlexIT	Questionable results from magnetic FlexIT Survey
KH-07-23	FlexIT	Questionable results from magnetic FlexIT Survey
KH-07-25	FlexIT	Questionable results from magnetic FlexIT Survey
KH-14-01	Deviflex	Questionable Results from Deviflex Tool. Poor consistency between different surveys
KH-14-02	Deviflex	Questionable Results from Deviflex Tool. Poor consistency between different surveys
KH-14-03	Deviflex	Questionable Results from Deviflex Tool. Poor consistency between different surveys
KH-14-04	Deviflex	Questionable Results from Deviflex Tool. Poor consistency between different surveys
KH-14-05	Deviflex	Questionable Results from Deviflex Tool. Poor consistency between different surveys
KH-14-06	Deviflex	Questionable Results from Deviflex Tool. Poor consistency between different surveys

Using the Reflex® Gyro non-magnetic surveys from 2013 to 2015 as a benchmark, it was decided to adjust the surveys for the 14 holes listed above using an average deviation down hole based on the collar orientations. All surface and solid modelling was done using the database with adjusted surveys for these 14 holes.

The majority of the 2013-2015 drill core was oriented where possible, using various core-orientation tools. In 2013 the Reflex ACT II™ was used. In 2014, for all core from holes up to and including KH-14-04, the Fordia© CoreintR was used to orient core. For all remaining 2014 holes, excluding KH-14-07, the 2ic EzyMark™ was used to orient core. Core from KH-14-07 was not oriented. In 2015 the Reflex Ez-Mark tool was used for core orientation. The Reflex Ez-Mark tool is the same as the 2ic EzyMark™.

9.3 SRK Comments

SRK has reviewed all the drilling data and is of the opinion that it is sufficient for use in resource estimation. For both the KUG and KE deposits, the drillholes with erroneous downhole survey data were reviewed in detail, and it was decided to use the drillholes with the manually corrected survey data. In each deposit the effect of these holes is believed to be non-material on the final resource estimate. The majority of these drillholes are located internally to the mineralized bodies, and are surrounded by higher confidence holes, indicating that these drillholes are not causing extrapolation to the mineralized volumes, resulting in a volumetric bias. As mining methods for each deposit are expected to be underground bulk extraction methods, the local impact these drillholes may have on the final block estimate is also considered to be non-material. As the KE deposit is an earlier stage project with less drilling, SRK accounted for some of the impact of these drillholes in the final resource classification as is documented in 13.4.11.

10 Sample Preparation, Analyses, and Security

10.1 Kemess North/Underground Sample Preparation and Analyses

Pre-2001 drilling was not deep enough to test the higher-grade zones under consideration here. Consequently, drilling since 2001 is the most important exploration activity carried out at Kemess North/Underground and forms the basis of the mineral resource. More than 75% of the assays have been completed since 2001.

Samples from the Kemess Underground project are totally drill core based; there are no trench or grab samples in the database.

Pincock Allan Holt's Vancouver based consulting group developed a sampling program for the project prior to the 2002 exploration season. The same program was carried forward to the 2003 and 2004 seasons, and continued into the 2010 program. The QA/QC program established in 2002 and continued in subsequent seasons included onsite insertion of blind duplicates, blanks, and certified standard samples.

Sample intervals were determined by a staff geologist according to lithology, and ranged from 0.1 to 2.0 m, with the average length of samples being 1.95 m. Because of the low-grade nature of the mineralization and difficulty determining potential ore from non-ore material, the entire drillhole is sampled. Once in a uniform rock type, sample spacing was generally 2.0 m. The maximum 2.0 m sample length was selected so that more detail could be gained concerning the local variability of grade. As well, the 2.0 m core length provides a representative sample weight for NQ core. For HQ core, a maximum sample length of 1.5 m was applied.

Drill core was logged by a small team of geologists and split using a rock saw or hydraulic splitter. Samples were collected by staff technicians and then passed through a primary crusher. During the 2002 program, a portable sample preparation lab was leased from ALS Minerals, formerly ALS Chemex (ALS). For the 2003 program, a sample-bucking facility was built near the camp. Core samples were dried, and then crushed to 80% passing 10 mesh at the mine site. Each sample was riffled twice with one split being retained at the mine, and a 250 g sample sent by air and courier to ALS analytical laboratory in North Vancouver. The remainder of the sample was discarded.

At ALS, the –10 mesh samples (pre-2008) were pulverized to 85% passing 75 µm (PUL-22) prior to assay. In 2008 a pulverizer was added to the preparation process at Kemess and pulps were submitted for analysis for both 2008, 2010 and 2011 programs. Some samples were only crushed post 2008 and the sample was sent to ALS to be pulverized to save time. Both Kemess and ALS prepared pulps were assayed for a suite of 35 elements including iron using an aqua regia digestion and inductively coupled plasma atomic emission spectroscopy (ICP-AES; ME-ICP61, ME-ICP41) on a one-gram sub-sample. Copper analysis was completed by atomic absorption spectrometry (AA), following a triple acid digestion. Gold analysis was completed by standard one assay ton fire assay with AA finish.

In total, excluding quality control samples, 33,717 samples comprise the entire Kemess underground database. Since 2000, 31,059 samples have been submitted to ALS for copper and

gold analyses which represent more than 92% of all the analyses. Since 2003, silver has been analysed by multi-element ICP package provided by ALS. There are 20,022 silver assays in the Kemess underground database.

The remaining 12% of the assay work was carried out by various labs for the earlier exploration companies including Kennco, Getty, Shell Oil, and El Condor. Historical records of the sampling, analysis, and security of this earlier work are not available. Most of this work is for shallow drilling and is not particularly relevant to the Kemess underground project.

ALS Minerals laboratories are accredited ISO 9001-2008 by QMI and the North Vancouver Laboratory is accredited ISO 17025-2005 by the Standards Council of Canada for a number of specific test procedures, including the method used to assay samples submitted by AuRico. ALS also participate in a number of international proficiency tests, such as those managed by CANMET and Geostats.

10.2 Kemess North/Underground Sample Security

The portion of sample retained at the mine site was kept in a plastic bag with a sample tag and stored in a plastic pail. The portion of the sample sent to the laboratory was placed in a plastic bag with a sample tag, shipped in a plastic pail with two security tags and the pail top was sealed and taped. A submission sheet was sent along with each pail of samples that included the name of the sample preparation person, the date, the sample numbers, the number of samples, and the numbers of the security tags.

There is a core storage site near Kemess Lake that serves to archive all the drilling on the project. The remaining half cores are still in core boxes and are available for geology reviews as well as check assays. In addition, all coarse rejects are stored at the same site in plastic pails while pulps are stored in sea containers near the exploration office above camp.

Work completed by Kemess employees included core logging, sample layout, sample splitting, and preliminary sample preparation. A professional geologist oversaw all of the work from core logging to sample splitting, while the Chief Assayer at the mine oversaw the preliminary sample preparation and shipping.

10.3 Kemess East Sample Preparation and Analyses

Pre-2007 drilling in Kemess East was either not deep enough or did not test the deposit area under consideration here. Consequently, drilling from the 2007 and follow-up programs from 2013-2015 are the only relevant exploration activities carried out in Kemess East and form the basis of its mineral resource estimate. All of the assays used in the resource have been completed since May 2007. Samples from the Kemess East area are totally drill core based; there is no trench or grab samples in the Kemess East database.

Sample intervals were determined by a staff geologist according to lithology, and ranged from 0.3 m to 2.0 m, with the average length of 1.80 m. Because of the low-grade nature of the mineralization and difficulty separating potential ore from non-ore material, the entire drillhole was sampled or

sampling began once the hole entered favourable host rocks. The maximum 2.0 m sample length was selected so that sufficient detail could be gained concerning the local variability of grade. As well, the 2.0 m core length provides a representative sample weight for NQ core. For HQ core, a maximum sample length of 1.5 m was applied.

During the 2002 Kemess North drilling program, a portable sample preparation laboratory was leased from ALS Limited's Minerals Division (ALS), formerly ALS Chemex. For the 2003 program, a sample preparation facility was built near the camp area. Design of this on-site sample preparation laboratory and the project's quality control procedures were established by Pincock, Allen, and Holt consulting group of Vancouver and implemented under the supervision of Bill Smith, Chief Assayer at Kemess Mine.

Drill core was logged by a small team of geologists and split using a rock saw or hydraulic splitter. The logging geologist designated quality control (QC) samples (blanks, duplicates, and certified reference materials) to be inserted into the sample stream at regular intervals. Pre-2015 drill programs the QC samples were inserted such that 1 in 25 (> 4%) samples were submitted for quality control purposes. In the 2015 drill program QC samples were inserted such that 2 in 25 (8%) samples were submitted for quality control purposes. Core samples were dried, and then crushed to 80% passing 10-mesh at the mine site. Each sample was riffled twice with one split being retained at the mine, and a 200-250 g sample sent by air and courier to the analytical laboratory. The remainder of the sample was discarded. In 2008 and 2010, pulverisers were added to the preparation process at Kemess and pulps, 85% passing 150-mesh (75µm), were submitted for analysis for the 2013-2015 programs. The majority of prepared samples, 97%, were shipped to ALS in North Vancouver for analysis, with 3% of samples submitted to SGS Canada Inc. (SGS), formerly Assayers Canada Laboratories, in Vancouver during 2007.

Samples were analyzed for a suite of 35 elements, including iron, molybdenum, and silver, using 2-acid digestion and ICP atomic emission spectroscopy on a one-gram sub-sample in 2007 and using a 4-acid digestion thereafter. Pre-2015 copper analyses were completed by AA spectrometry, following a 3-acid digestion. Gold analyses were completed by standard 1-assay-ton fire assay with an AA finish. Samples from significantly mineralized zones were also submitted for molybdenum analyses, which included 4-acid digestion with an AA finish. In 2015 mineralized samples were additionally analyzed by an extra ore grade analysis for copper and molybdenum by ICP atomic emission spectroscopy, following a 4-acid digestion.

In total, excluding quality control samples, 35,714 samples comprise the greater 2002-2015 Kemess East area database. Since May 2007, 26,148 samples have been submitted for copper and gold assay and ICP analyses which represent more than 81% of all analyses. The other 19% are from shallow drilling and holes outside the Kemess East deposit vicinity, which are not particularly relevant to the project. A total of 10,066 samples were also submitted for molybdenum analyses by 4-acid digestion with an AA OR ICP-OES finish.

ALS Minerals Division laboratories are accredited ISO 9001-2008 by QMI and the North Vancouver Laboratory is accredited ISO 17025-2005 by the Standards Council of Canada for a number of specific test procedures, including the method used to assay samples submitted by AuRico. ALS also participate in a number of international proficiency tests, such as those managed by CANMET

and Geostats. SGS Canada Inc. laboratories are accredited ISO 17025 by the Standards Council of Canada.

10.4 Kemess East Sample Security

The portion of crushed sample retained at the mine site is kept in a plastic bag with a sample tag and stored in a plastic pail. The portion of the sample sent to the laboratory was placed in a plastic bag with a sample tag, shipped in a plastic pail with two numbered zap-strap style security seal tags attaching the lid to the pail. A submission sheet was sent inside each pail of samples that included the name of the sample preparation person, date, sample numbers, the number of samples, and the identification numbers on the security tags. An electronic copy was also emailed to the laboratory for verification of security seal tags.

There is a core storage area on site near Kemess Lake that serves to archive all the drilling on the project. The remaining half cores are stored there in core boxes and are available for geological reviews as well as check assays and additional test work. In addition, all coarse rejects are stored at the same site in plastic pails while pulps are stored in sea containers near the exploration office above camp. Work completed by Kemess employees included geological and geotechnical core logging, sample layout, sample splitting, and sample preparation. Professional geologists oversaw all of the work from core logging to sample processing, while the Chief Assayer at the mine provided additional oversight of sample preparation and shipping up until Kemess South mine closure in March, 2011.

10.5 Kemess North/Underground Specific Gravity Data

A great deal of effort has gone into the determination of the bulk density (BD) for the deposit. In total, more than 9,886 measurements have been performed on drilling within the Kemess Underground deposit. Four different sets of data are available, pre-1999 samples, as well as samples from 2000, 2002, and 2003. Sample material ranged from whole core samples 15 cm to 20 cm long for the 2003 work, quartered core for the 2000 and 2002 samples, and crushed coarse reject material for the pre-1999 samples. The 2003 lab work was by Lakefield Research using their wax immersion method, which compared favourably to determinations by measuring differential mass in air and water. After 2004, all drill samples had SG determined by air and water methods performed while logging.

Specific gravity (SG) data was collected by analysis of a 10 cm to 15 cm length of core representing every assay interval. The recorded value was then assigned to the corresponding sample interval in the assay table.

The specific gravity measurements are performed by onsite logging geologists, utilizing the standard water immersion (no wax coating) methodology. This methodology is applicable to samples with minor permeability, little to no surface pore spaces, and no fracturing. In circumstances of highly porous / fractured rock this methodology can lead to potentially erroneous results as the volume is not correctly calculated. In situations like this, the wax coating methodology is preferred. For this reason, in the near surface rock mass in the KUG deposit, mostly constrained to the highly friable leached cap unit, there is both a decrease in the number of available specific

gravity measurements as well as in the quality of the data. This is not believed to be material to the KUG resource as this particular unit is generally weakly mineralized and uneconomic. The decrease in number of data and confidence of the data is addressed in the Mineral Resource classification, and subsequent reporting of Mineral Resources, as this unit is excluded from the Mineral Resource totals.

10.6 Kemess East Specific Gravity Data

A great deal of effort has gone into the determination of the bulk density (BD) for the deposit. In total, more than 30,242 measurements have been performed on drilling within the Kemess East deposit. After 2004, all drill samples had SG determined by air and water methods performed while logging.

Specific gravity (SG) data was collected by analysis of a 10 cm to 15 cm length of core representing every assay interval. The recorded value was then assigned to the corresponding sample interval in the assay table.

The specific gravity measurements are performed by onsite logging geologists, utilizing the standard water immersion (no wax coating) methodology. This methodology is applicable to samples with minor permeability, little to no surface pore spaces, and no fracturing. In circumstances of highly porous / fractured rock this methodology can lead to potentially erroneous results as the volume is not correctly calculated. In situations like this, the wax coating methodology is preferred. The methodology utilized for SG determination within the Kemess East deposit is deemed sufficient.

10.7 Quality Assurance and Quality Control Programs

Drilling on the Kemess property has incorporated an industry standard QA/QC protocol, which includes the submission of regular field blank samples, field duplicate samples, and certified reference material samples, into the sample stream sent for analytical review.

10.7.1 Kemess North/Underground Quality Assurance and Quality Control Programs

Blank samples, material with very low concentrations of copper and gold, were used to test for contamination of samples. Samples were also compared against the laboratory certificates. Rocklabs Certified Reference Material was used as quality control standard samples to monitor accuracy. Duplicate samples were used to monitor and measure preparation and analytical precision. In total, 981 samples were submitted for quality control purposes as blind blanks, standards, or duplicates. This represents approximately one in every 34 samples, or 2.9% of the samples collected from 2002 through 2011 from drilling on the Kemess North/Underground deposit. Quality control information was recorded by geologists, core samplers, and sample preparation staff. This triple-redundancy data capture was used to identify and eliminate data entry errors.

10.7.2 Kemess North/Underground Blanks

Evaluation of gold and copper analyses of quality control blanks, of barren looking Hazelton rocks, indicates that no significant or systematic contamination or laboratory error occurred during the course of the 2002–2011 programs (Figure 10.1 to Figure 10.3).

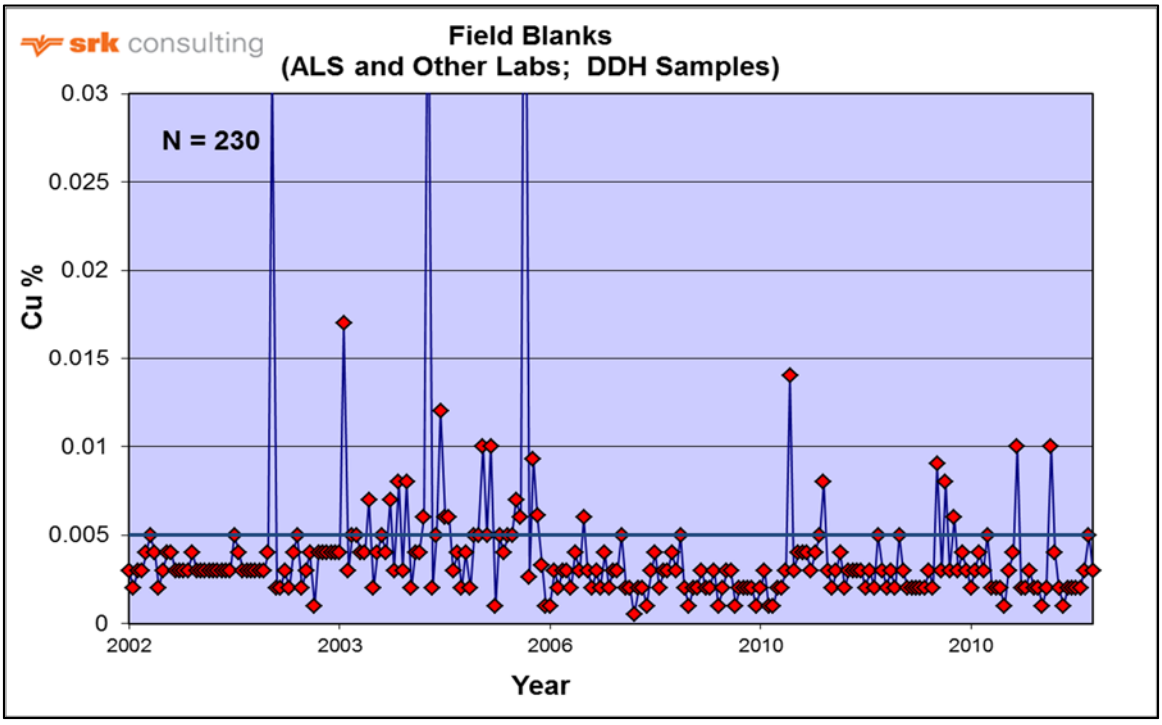


Figure 10.1: Kemess North/Underground Database Blanks over time – Copper

(Source: SRK, 2016)

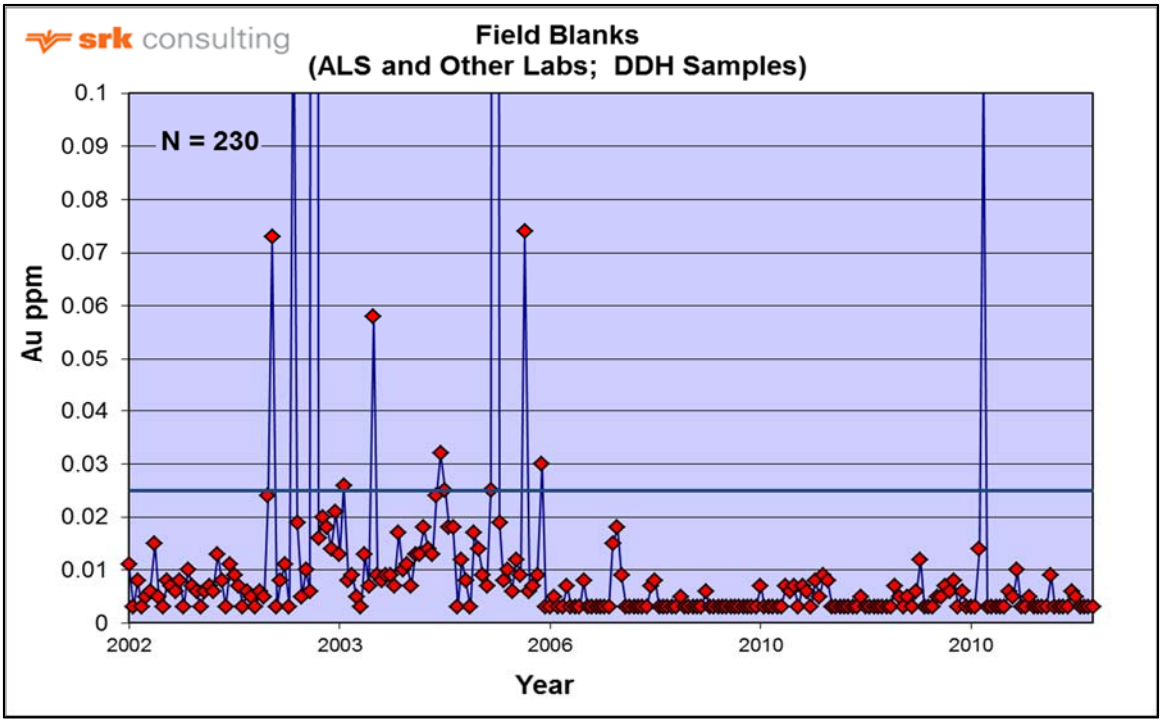


Figure 10.2: Kemess North/Underground Database Blanks over time – Gold

(Source: SRK, 2016)

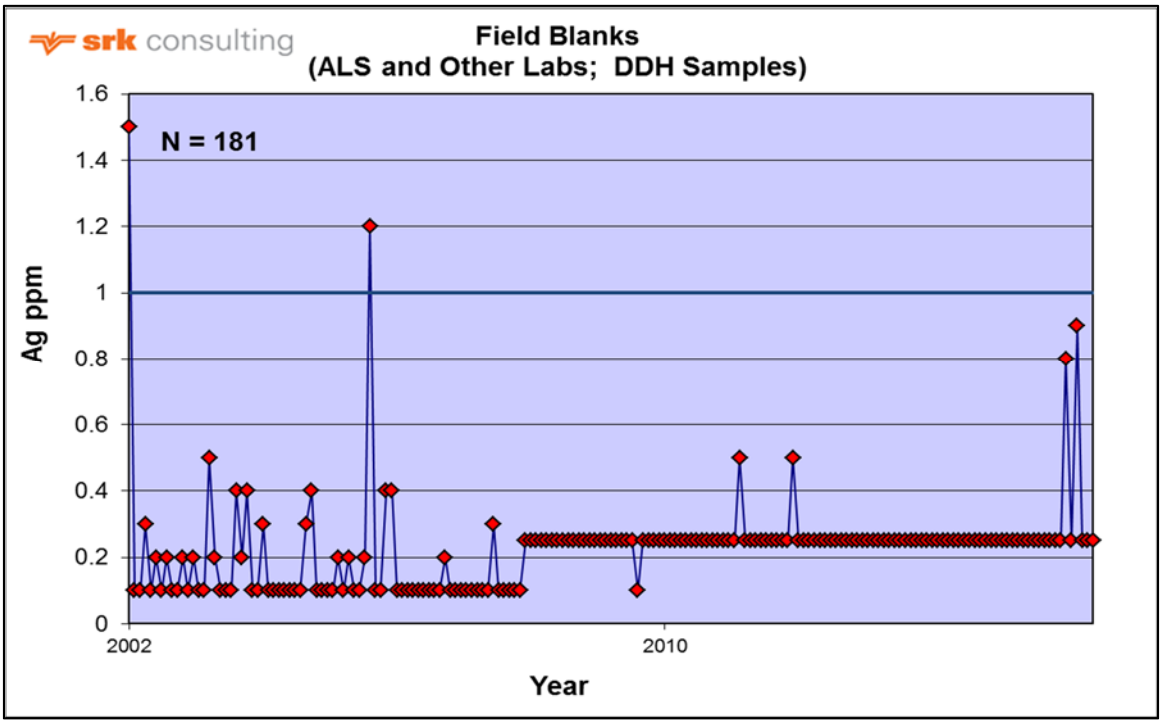


Figure 10.3: Kemess North/Underground Database Blanks over time – Silver

(Source: SRK, 2016)

11% of the copper blanks, 4% of the gold blanks, and 1% of the silver blanks fail and exceed 5 times the detection limit. During each drilling campaign blank gold outliers were investigated and in the vast majority of cases were found to display ICP signatures that are typical for Hazelton Group rocks with elevated silver and copper contents. These outliers are typically associated with rare gold mineralization associated with narrow quartz veins hosted in Hazelton rocks. From this it can be concluded that these outliers indicate natural, insitu, geologic variation of the blank material rather than contamination from adjacent samples or other sources during sample preparation. The blank copper outliers are typically coincident with these gold outliers, which support the conclusion that this is a natural, insitu, geologic variation. The few outliers, not exhibiting these characteristics, do not represent any significant or systematic contamination.

10.7.3 Kemess North/Underground Standards

Drilling at Kemess North/Underground has utilized various different certified standards for QA/QC monitoring over the course of the drilling campaigns. A notable deficiency in the QA/QC protocols for the Kemess North drilling, was the lack of standards certified for copper. All standards used, were certified for gold but not for copper. SRK recommended a re-assay program in 2015 to account for this lack of data, as is described in 10.7.4. The results for gold and silver are considered acceptable.

10.7.4 Kemess North/Underground Standards – Copper

No copper standards had been included in the 2002-2011 underground drilling data analysis. Copper performance was tracked and monitored along with the gold standards which gave confidence in the data quality.

SRK determined that it would be beneficial to have additional support for the copper assays. SRK selected assay samples across the modeled block cave area of influence (AOI) to conduct additional QA/QC procedures for copper. Kemess sent 200 reject samples to ALS labs along with 25 standards and blanks. Approximately 12% of the samples are standards and blanks with an insertion rate of 1 in every 8 samples.

Re-assaying the rejects along with standards and blanks, provides more confidence in the original assay data. All of the 11 blanks were below detection limit. All but one standard result was within 2 standard deviations. This is considered acceptable.

SRK also compared the original assays to the re-assayed rejects. The samples show excellent correlation for copper and good correlation for gold (Figure 10.4 and Figure 10.5). 96.5% of the copper values and 87.6% of gold values are within 10% between the original and reject resample data.

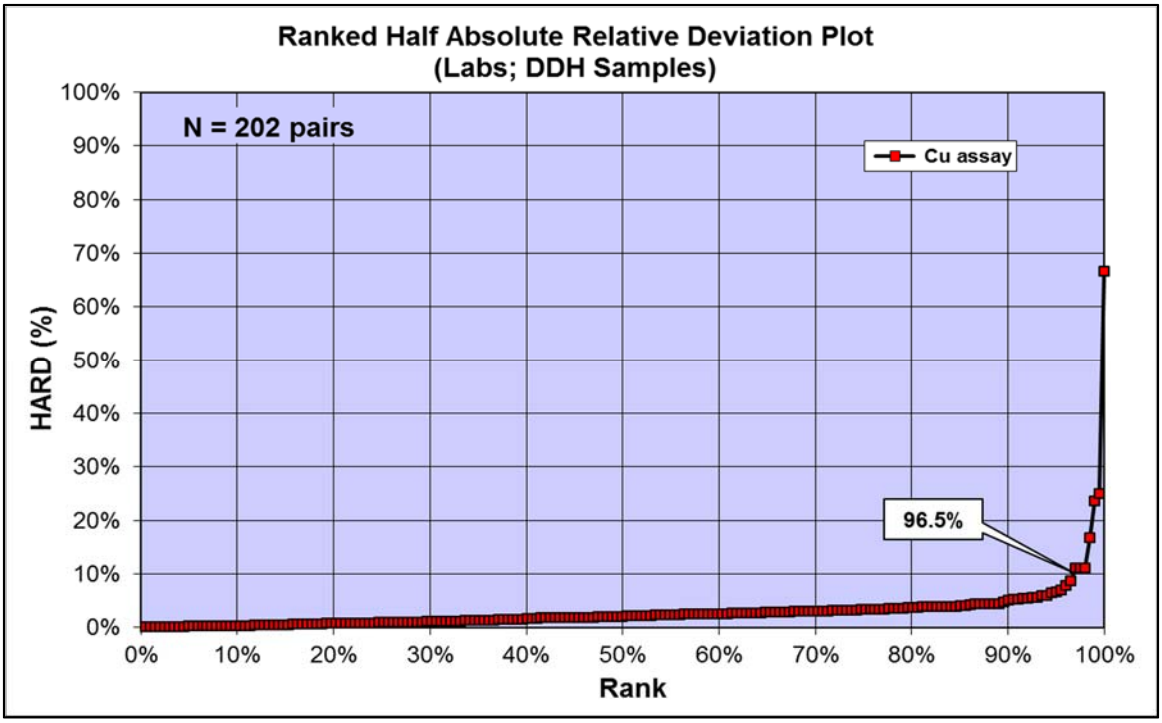


Figure 10.4: Kemess North/Underground Re-Sample and Original paired relative deviation – Copper

(Source: SRK, 2016)

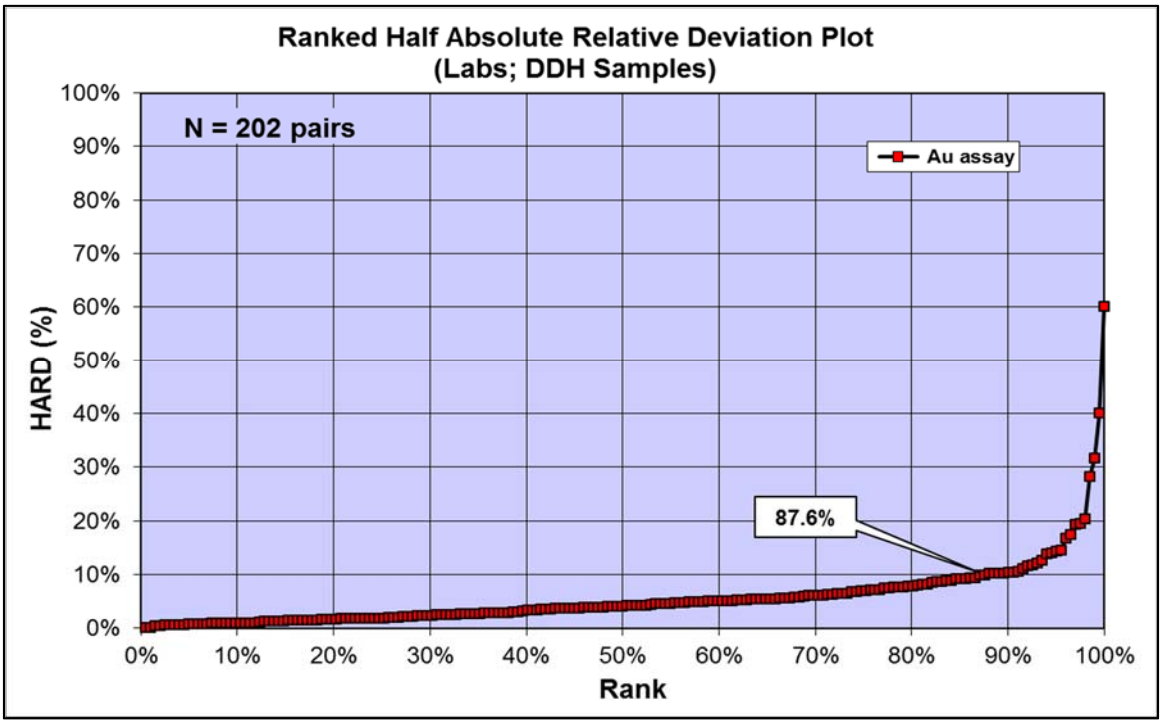


Figure 10.5: Kemess North/Underground Re-Sample and Original paired relative deviation – Gold

(Source: SRK, 2016)

10.7.5 Kemess North/Underground Standards – Gold

Results for the quality control standards (certified reference material) were reviewed throughout the course of each program. Standards that did not report within industry accepted ±3 standard deviation error limits (Table 10.1) were investigated and data entry errors corrected or affected samples rerun if necessary, with the exception of a few unresolved outliers encountered in 2002, 2003, and 2011. If failed standards performed acceptably on the second run then the original assays were corrected and new certificates were issued for the batches of associated samples. Standard failures reproduced on the second run were deemed to be due to normal variation in the certified reference material and therefore the original results were accepted as accurate. Final results for several standards are presented in Figure 10.6 to Figure 10.8. Overall the standards performed well and are considered acceptable.

Table 10.1: Kemess North/Underground 2002-2011 certified RockLabs standards for Gold

Standard	Average Au ppm	1 Standard Deviation Au ppm	- 3 Standard Deviation	+ 3 Standard Deviation
GTS-2	0.263	0.003	0.256	0.271
Ox5	0.968	0.043	0.839	1.097
OxD27	0.416	0.013	0.379	0.454
OxE21	0.651	0.026	0.573	0.729
OxH29	1.298	0.033	1.199	1.397
S2	1.530	0.054	1.370	1.691
SE44	0.606	0.017	0.555	0.657
SE58	0.607	0.019	0.550	0.664
SF12	0.819	0.028	0.735	0.903
SH35	1.323	0.044	1.191	1.455
SH55	1.375	0.045	1.240	1.510

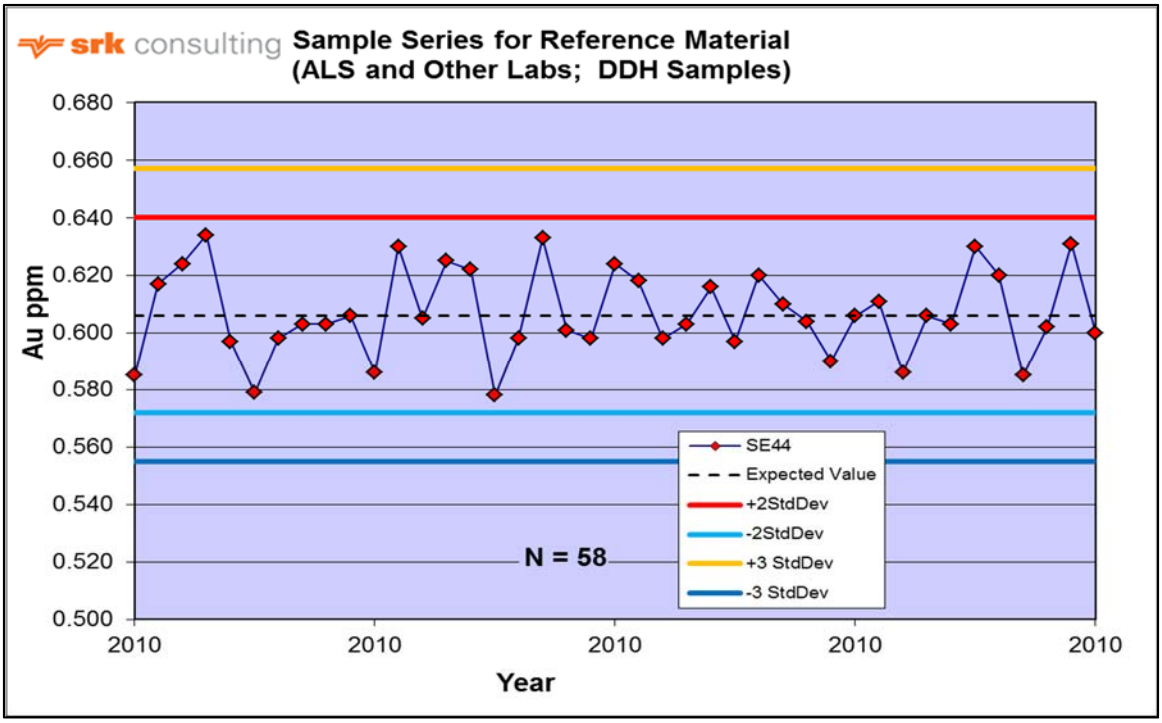


Figure 10.6 Kemess North/Underground Standard SE44 overtime – Gold

(Source: SRK, 2016)

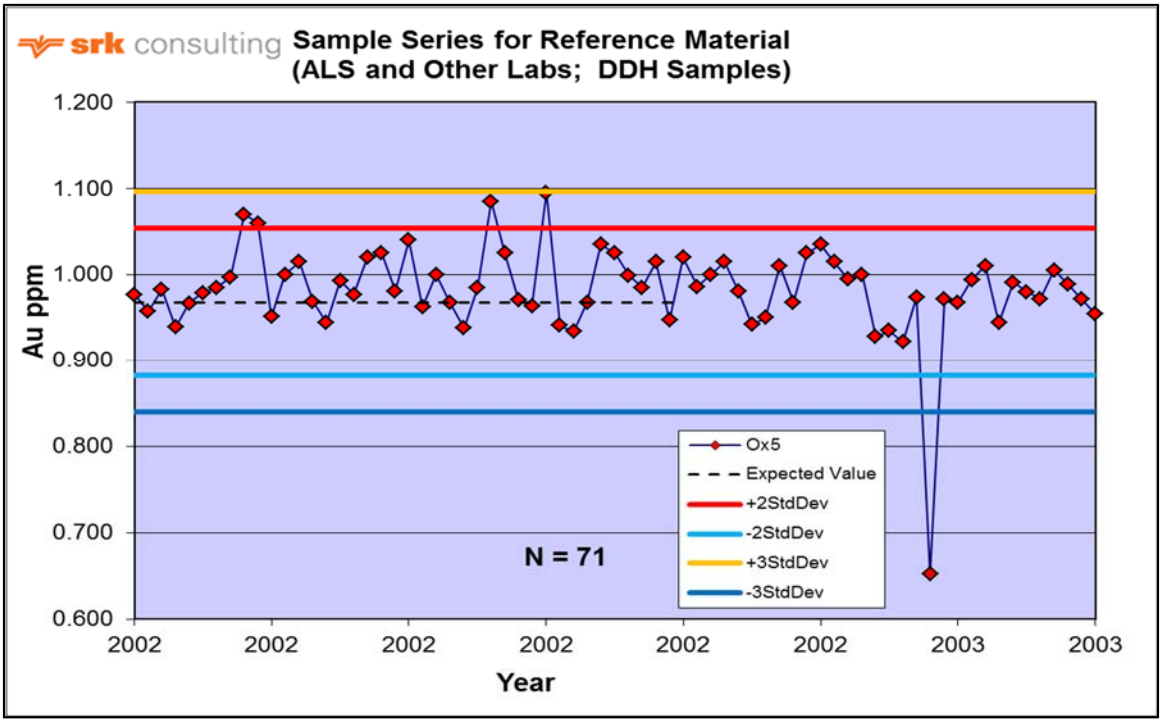


Figure 10.7 Kemess North/Underground Standard OX5 over time – Gold

(Source: SRK, 2016)

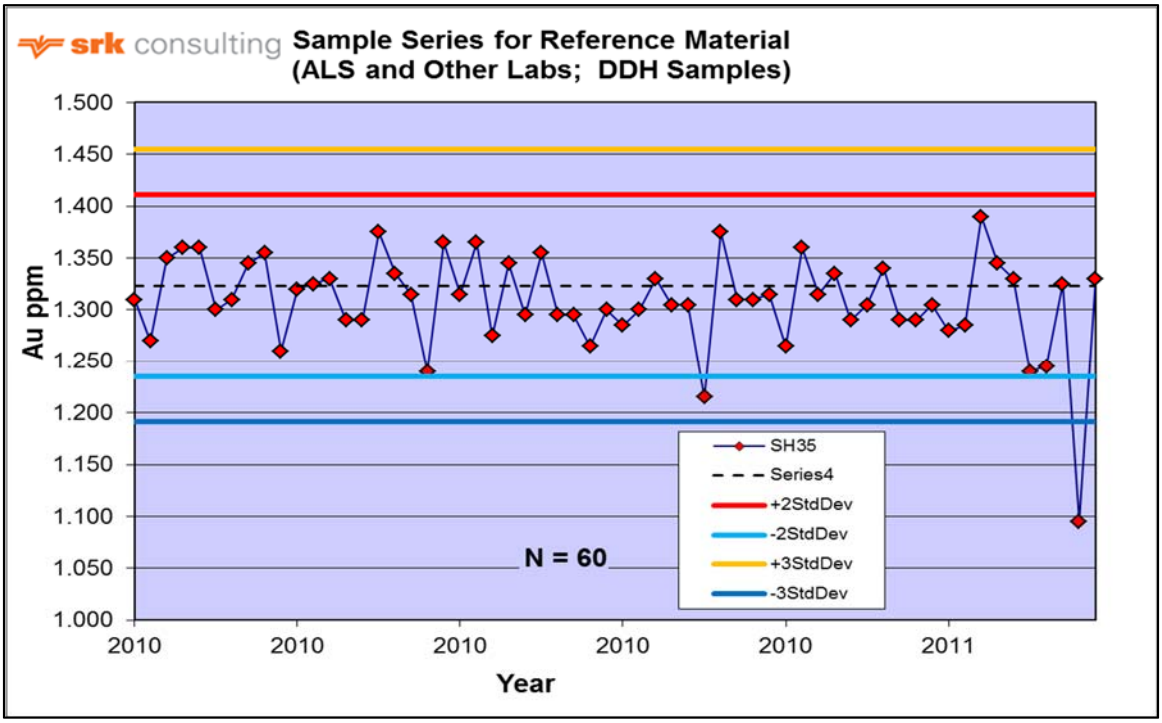


Figure 10.8 Kemess North/Underground Standard SH35 over time – Gold

(Source: SRK, 2016)

10.7.6 Kemess North/Underground Standards – Silver

There are no standards for silver. Silver is not a material component of the resource and therefore has not had detailed analysis completed.

10.7.7 Kemess North/Underground Standards – Conclusion

Based on the performance of the gold standards, over the entire course of the 2002–2011 programs, laboratory investigations of outliers, and copper reject re-assaying, all mainstream gold and copper assay results are considered acceptable.

10.7.8 Kemess North/Underground Duplicates

Combined preparation and analytical precision was examined using matched pairs created by taking a second 250 g riffle-split, referred to as a “reject-duplicate”, from randomly selected original mainstream crushed sample rejects. These matched pairs of duplicates and originals were used to measure precision and how it varies with grade. Scatter plots and Relative Deviation Plots of matched-pair assays are presented in Figure 10.9 to Figure 10.12. Both gold and copper correlate well. Silver (not shown) also correlates well.

10.7.9 Kemess North/Underground Duplicates - Copper

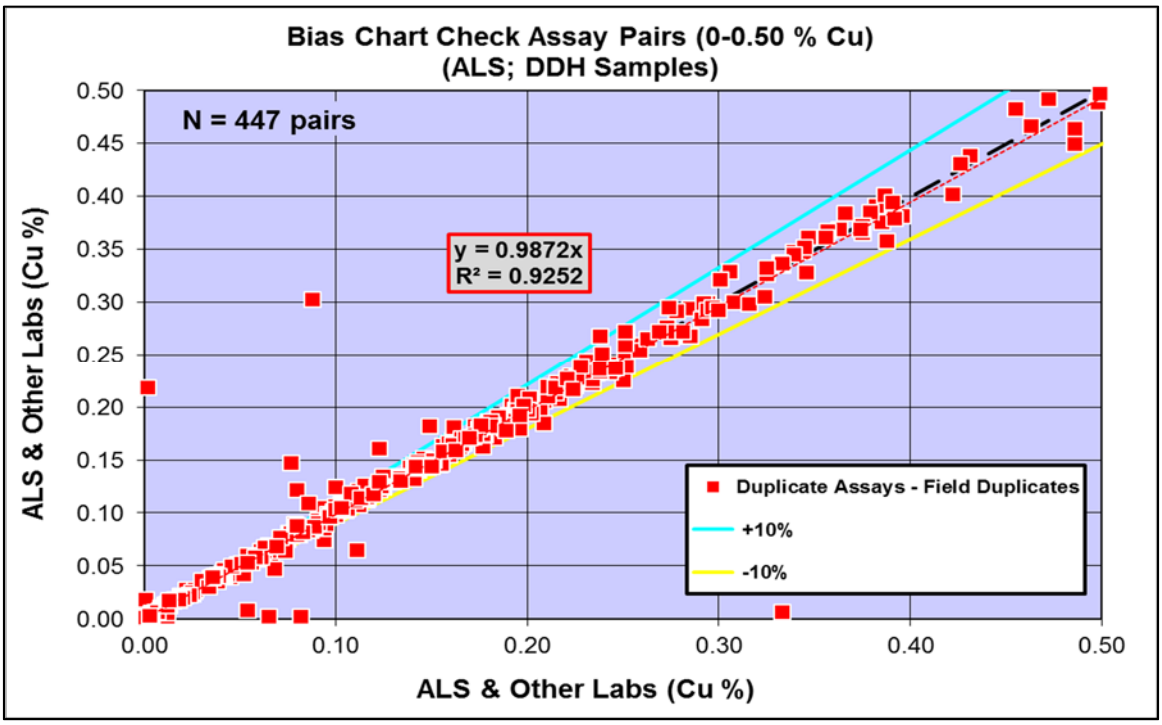


Figure 10.9: Kemess North/Underground Matched Pair Results for Reject Duplicates – Copper

(Source: SRK, 2016)

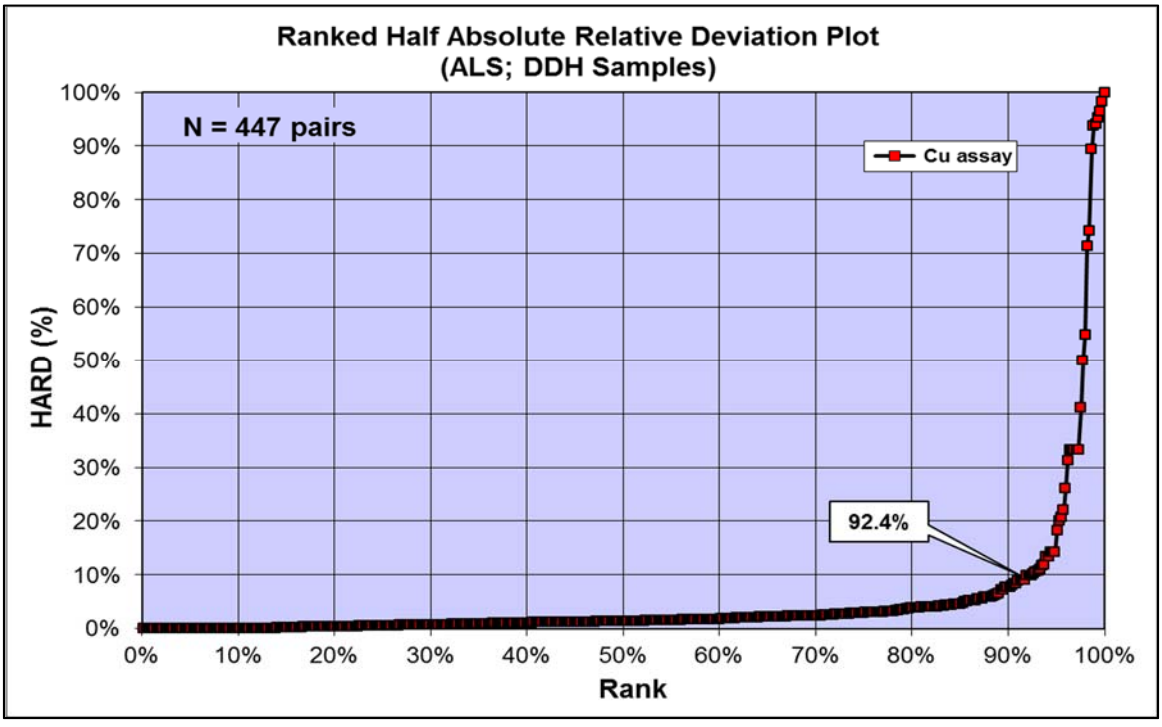


Figure 10.10: Kemess North/Underground Assay Paired Relative Deviation – Copper
(Source: SRK, 2016)

10.7.10 Kemess North/Underground Duplicates - Gold

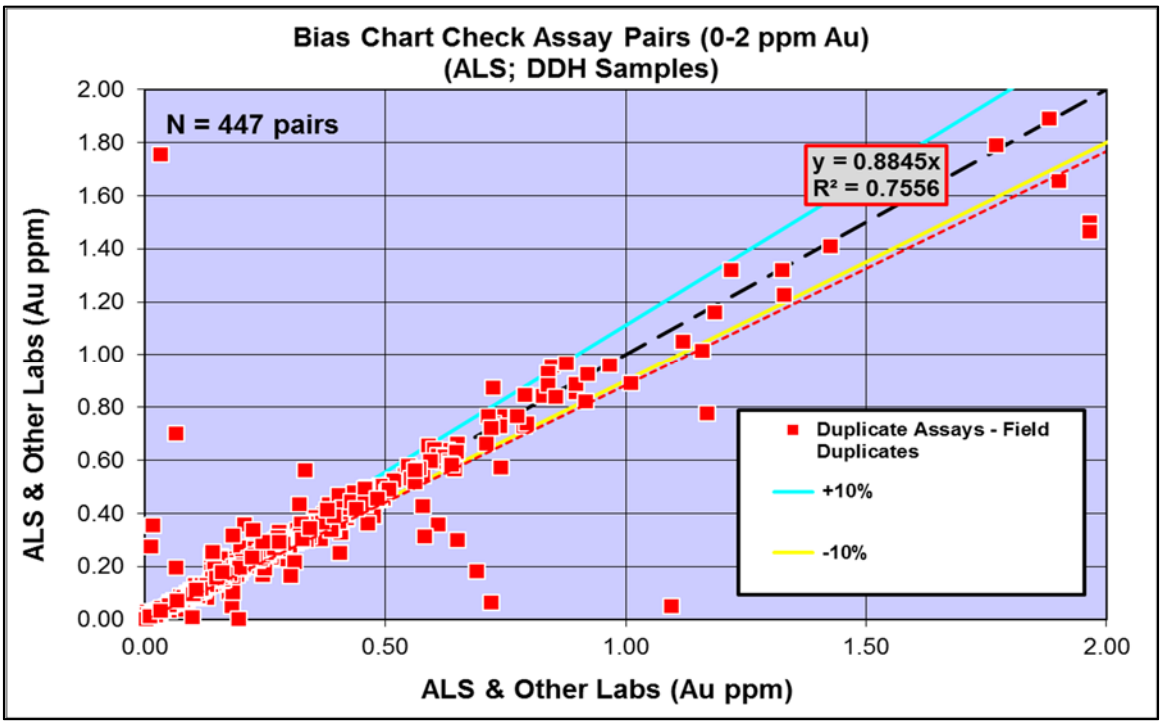


Figure 10.11: Kemess North/Underground Matched Pair Results for Reject Duplicates – Gold

(Source: SRK, 2016)

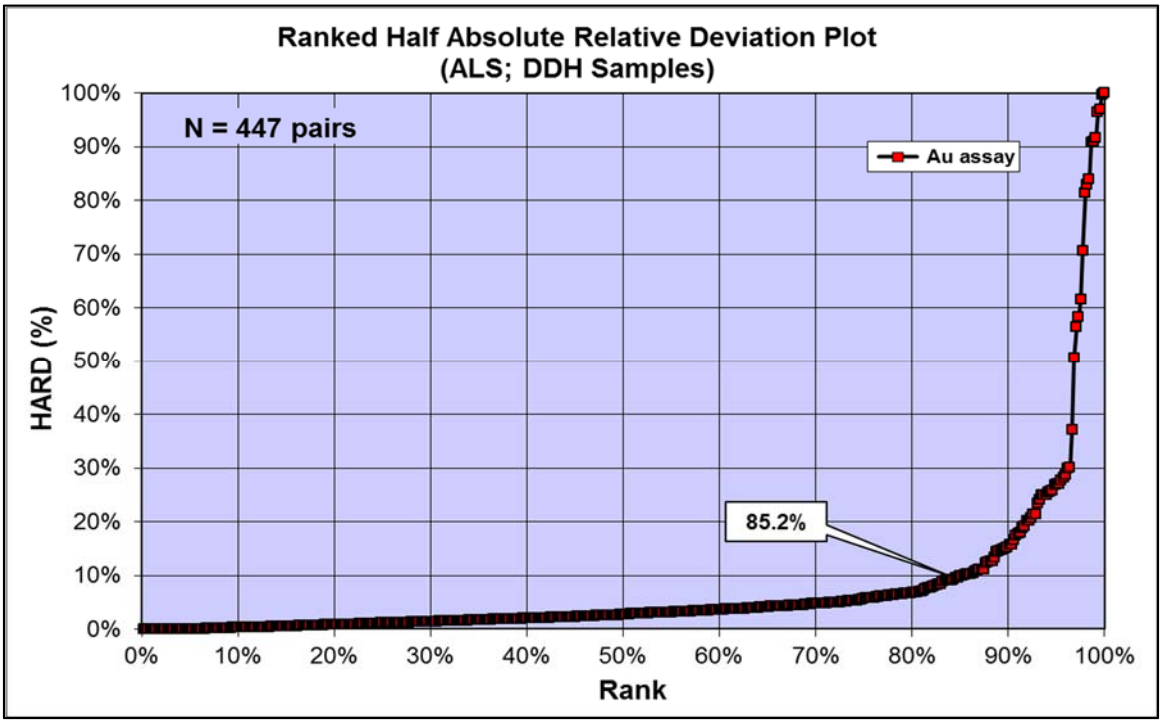


Figure 10.12: Kemess North/Underground Assay Paired Relative Deviation – Gold
(Source: SRK, 2016)

10.7.11 Kemess East Quality Assurance and Quality Control Programs

Blank samples, material with very low concentrations of copper and gold, were used to test for contamination of samples. Rocklabs and CDN Resource Labs Certified Reference Material were used as quality control standard samples to monitor accuracy. Duplicate samples were used to monitor and measure preparation and analytical precision. Duplicate samples were also sent to AGAT Laboratories as umpire samples. In total, 2,140 samples were submitted for quality control purposes as blind blanks, standards, or duplicates. This represents approximately one in every 16 samples, or 6.1%, of the samples collected from 2005 through 2015 from the drilling used in the Kemess East resource update. Quality control information was recorded by geologists, core samplers, and sample preparation staff. This triple-redundancy data capture was used to identify and eliminate data entry errors.

10.7.12 Kemess East Blanks

Evaluation of gold and copper analyses of quality control blanks, of barren looking Hazelton rocks, indicates that no significant or systematic contamination or laboratory error occurred during the course of the 2005–2015 programs (Figure 10.13 to Figure 10.16).

A blank is considered a failure when the assay exceeds 5 times the detection limit. 10% of the copper blanks, 1.5% of the gold blanks, <1% of the silver blanks, and <1.5% molybdenum blanks fail. During each drilling campaign blank gold and copper outliers were investigated and in the vast majority of cases were found to display ICP signatures that are typical for Hazelton Group rocks. These outliers are typically associated with rare mineralization associated with narrow quartz veins hosted in Hazelton rocks. From this it can be concluded that these outliers indicate natural, in-situ, geologic variation of the blank material rather than contamination from adjacent samples or other sources during sample preparation. The few outliers, not exhibiting these characteristics, do not represent any significant or systematic contamination.

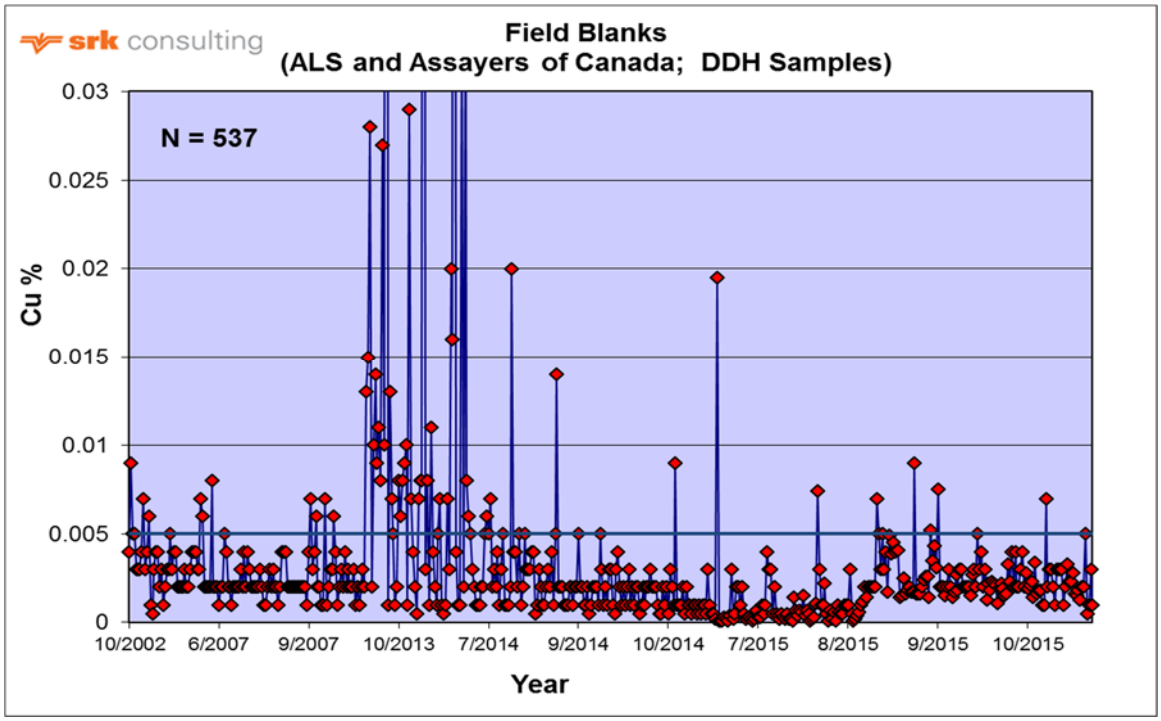


Figure 10.13: Kemess East Blanks over time – Copper

(Source: SRK, 2016)

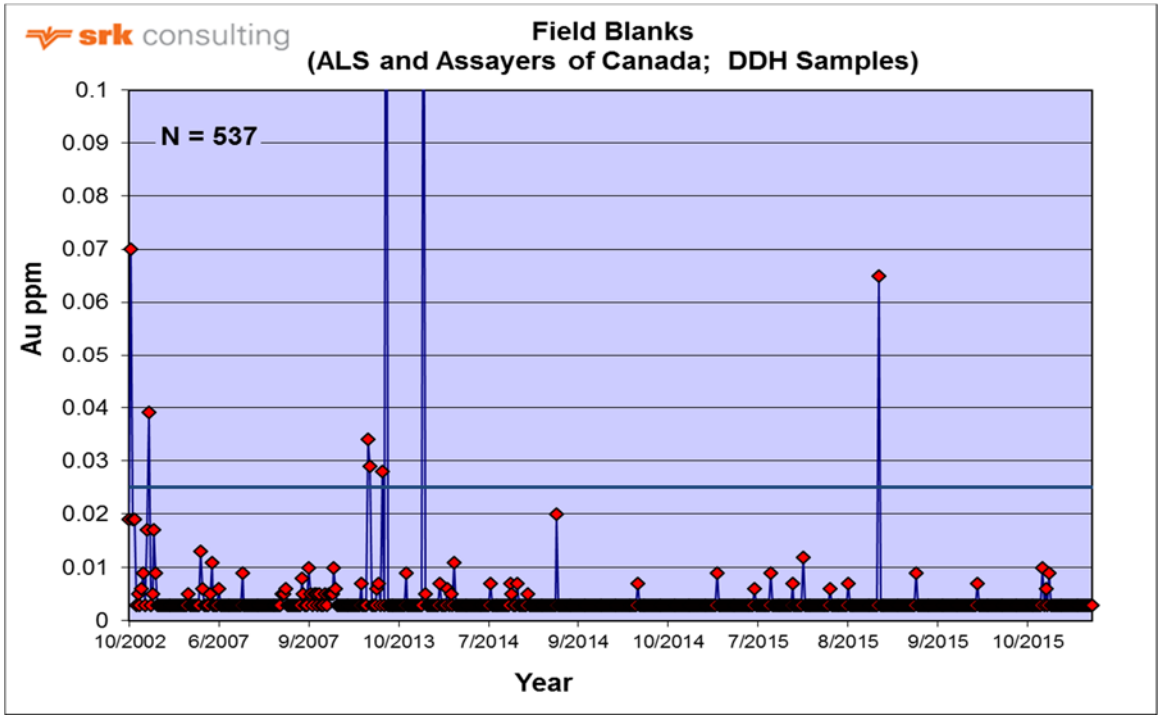


Figure 10.14: Kemess East Blanks over time – Gold

(Source: SRK, 2016)

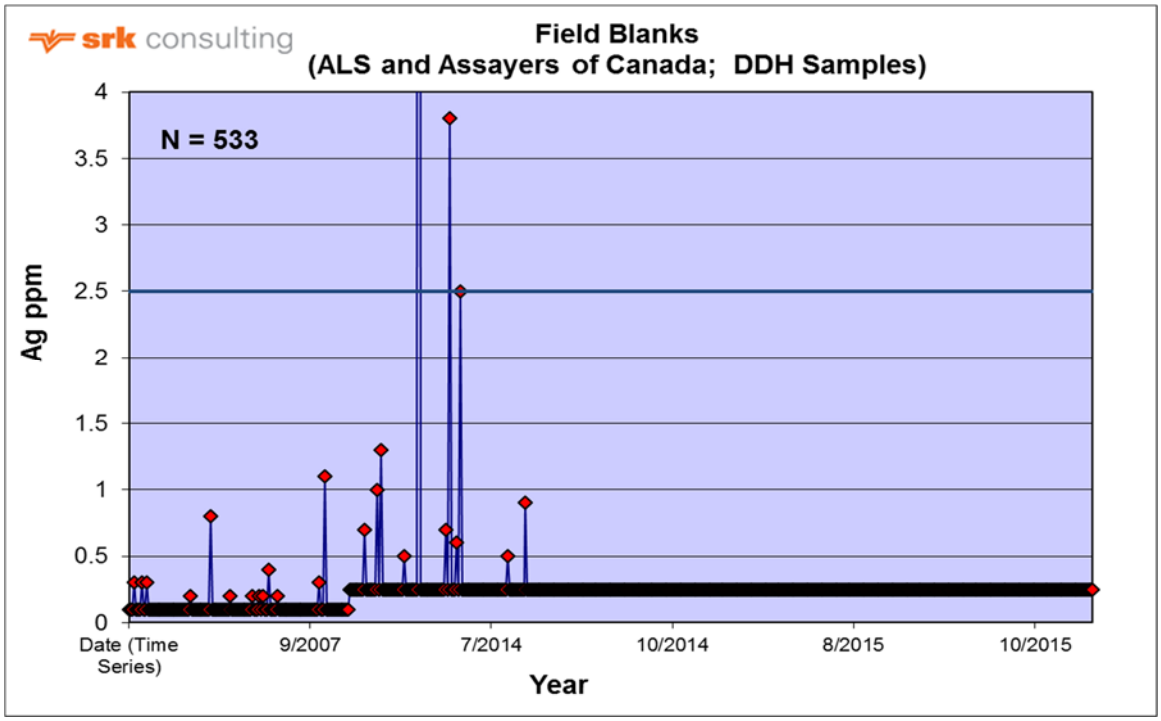


Figure 10.15: Kemess East Blanks over time – Silver

(Source: SRK, 2016)

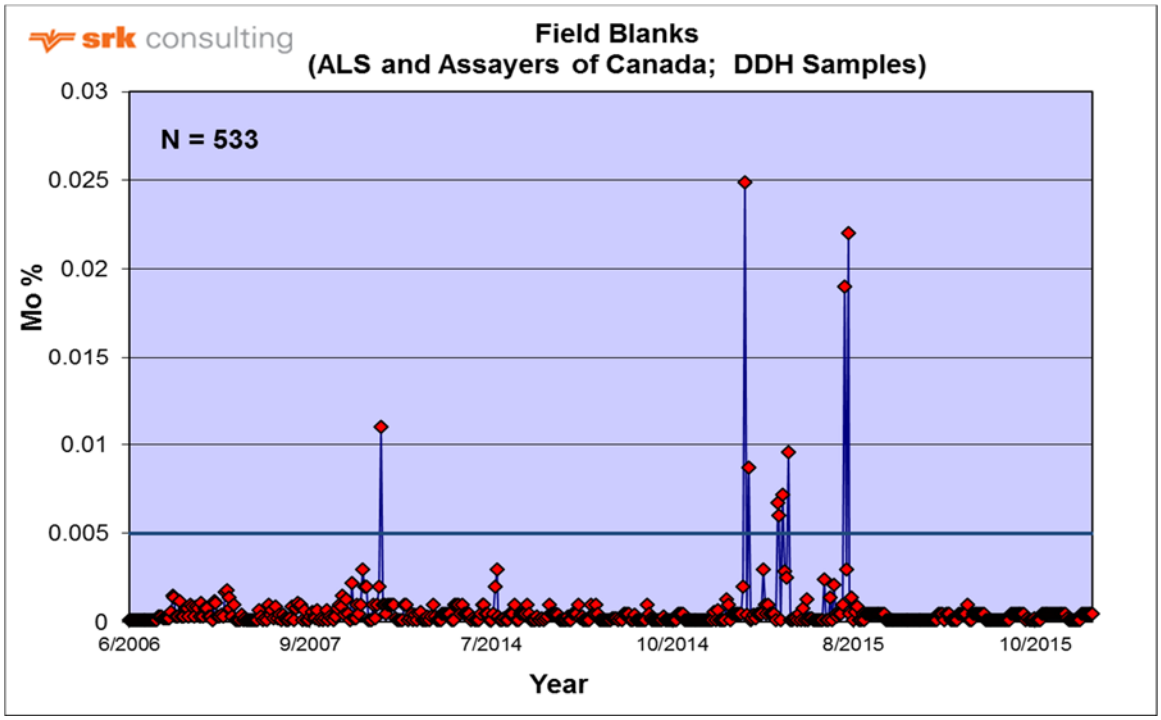


Figure 10.16: Kemess East Blanks over time – Molybdenum

(Source: SRK, 2016)

10.7.13 Kemess East Standards

Quality control standards (certified reference material) were included in the 2005 to 2015 drill campaigns for Kemess East. Drilling prior to 2013 only included gold standards. Drilling from 2013 to 2015 utilizes standards for gold, copper, silver, and molybdenum.

Standards that did not report within industry accepted ±2 standard deviation error limits for copper or ± 3 standard deviation error limits for gold, silver, and molybdenum (Table 10.2) were investigated and data entry errors corrected or affected samples rerun if necessary, with the exception of a few unresolved outliers encountered in, 2007, 2013, 2014 and 2015. If failed standards performed acceptably on the second run then the original assays were corrected and new certificates were issued for the batches of associated samples. Standard failures reproduced on the second run were deemed to be due to normal variation in the certified reference material and therefore the original results were accepted as accurate.

Overall the standards are considered acceptable.

Table 10.2: 2002-2015 Certified Standards

Metal	Standard	Average	2 StdDev	-2StdDev	+2StdDev	-3StdDev	+3StdDev
Cu %	CDN-CM-26	0.246	0.016	0.230	0.262		
	CDN-CM-30	0.73	0.034	0.696	0.764		
	CDN-CM-35	0.243	0.012	0.231	0.255		
	CDN-CM-39	0.538	0.024	0.514	0.562		
Au ppm	CDN-CM-26	0.372	0.048	0.324	0.420	0.300	0.444
	CDN-CM-30	1.300	0.120	1.180	1.420	1.120	1.480
	CDN-CM-35	0.324	0.032	0.292	0.356	0.276	0.372
	CDN-CM-39	0.687	0.064	0.623	0.751	0.591	0.783
	OX-5	0.968	0.086	0.882	1.054	0.839	1.097
	SE-19	0.583	0.052	0.531	0.635	0.505	0.661
	SE-29	0.597	0.032	0.565	0.629	0.549	0.645
	SE58	0.607	0.038	0.569	0.645	0.550	0.664
	SF12	0.819	0.056	0.763	0.875	0.735	0.903
	SH-24	1.326	0.086	1.240	1.412	1.197	1.455
	SH55	1.375	0.090	1.285	1.465	1.240	1.510
Ag ppm	CDN-CM-30	15.9	1.3	14.600	17.200	13.950	17.850
	CDN-CM-39	5.3	0.5	4.800	5.800	4.550	6.050
Mo %	CDN-CM-30	0.07	0.004	0.066	0.074	0.064	0.076
	CDN-CM-35	0.029	0.002	0.027	0.031	0.026	0.032
	CDN-CM-39	0.0137	0.0013	0.012	0.015	0.012	0.016

10.7.14 Kemess East Standards – Copper

Results for the quality control standards (certified reference material) were reviewed throughout the course of each program between 2013 and 2015.

From 2002 to 2007 copper performance was tracked and monitored along with the gold standards which gave confidence in the data quality. SRK determined that it would be beneficial to have additional support for the copper assays and a reject sampling program was conducted for the Kemess Underground deposit. The samples show excellent correlation for copper and good correlation for gold. Further information is available in 10.7.4. While this additional check did not include samples from the KE deposit, SRK is of the opinion that this validation confirms the copper results for the 2002 to 2007 drill results on the Kemess property.

The 2013 to 2015 drilling included four copper standards: CM-26, CM-30, CM-35, and CM-39.

CM-26 had no failures. For CM-30, 4.9% of the samples fall outside of two standard deviations and are considered failures. For CM-35, 7.1% are failures, and for CM-39, 17.9% of the samples are considered failures. Final results for CM-30 and CM-39 are presented in Figure 10.17 and Figure 10.18.

Though the failure of CM-39 is very high, all failed values were below two standard deviations. SRK recommends the removal of this standard. Overall the standards performed well and are considered acceptable.

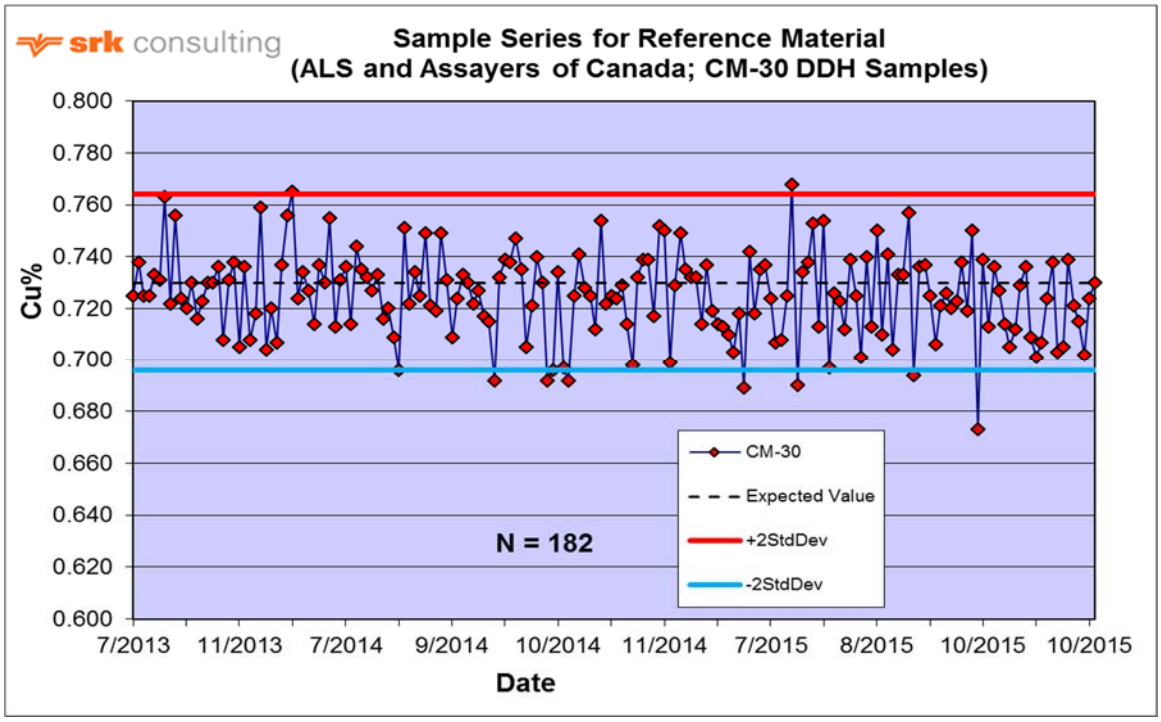


Figure 10.17: Kemess East Standard CM-30 over time – Copper

(Source: SRK, 2016)

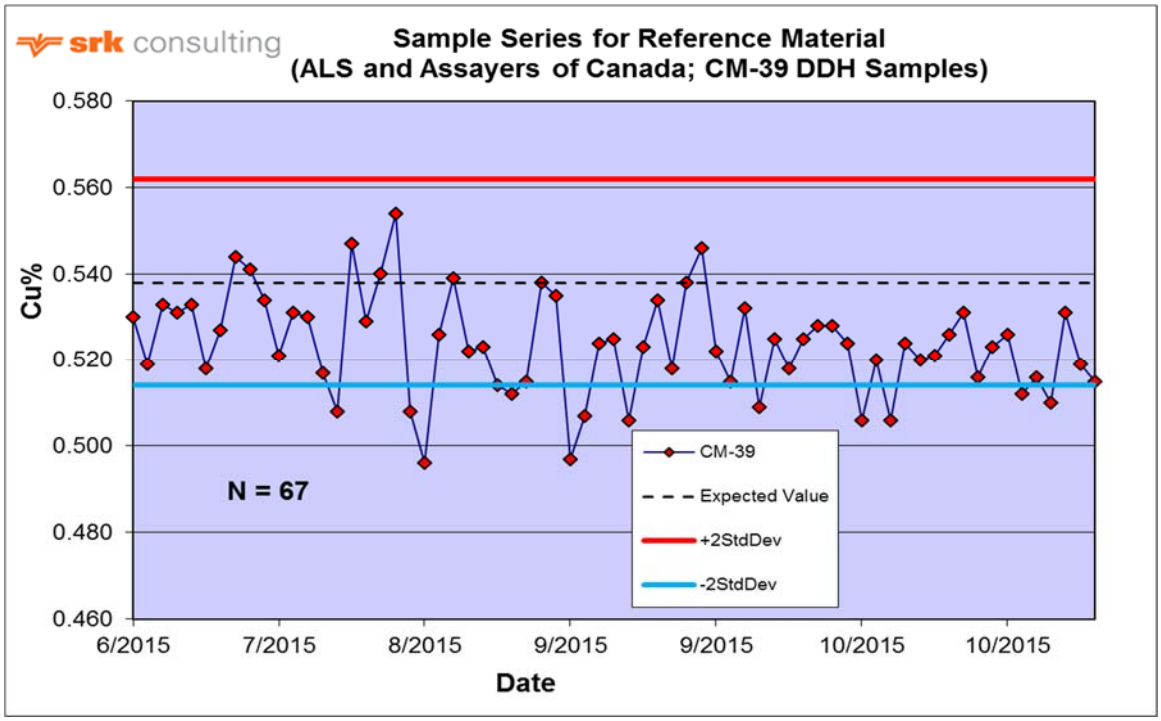


Figure 10.18: Kemess East Standard CM-39 over time – Copper

(Source: SRK, 2016)

10.7.15 Kemess East Standards – Gold

Results for the quality control standards (certified reference material) were reviewed throughout the course of each program. Gold standards were used in all drilling programs and twelve standards were used in total. Nine standards have more than ten samples and are detailed below. Not included are standards OX-5, OXD-27, SE-19, and SE-58 which were reviewed but are not statistically significant.

Standards CM-26, CM-30, CM-35, CM-39, SH-24, and SE-29 have less than a 2% failure outside of 3 standard deviations.

SH-55 and SF-12 have a 7.7% and 8.3% failure respectively. Both of these standards have less than 15 samples and are not considered statistically significant. If these standards continue to perform poorly, it is recommended to remove these standards from the QAQC procedures.

Overall the standards performed well and are considered acceptable.

Final results for CM-30 and SH-24 are presented in Figure 10.19 and Figure 10.20.

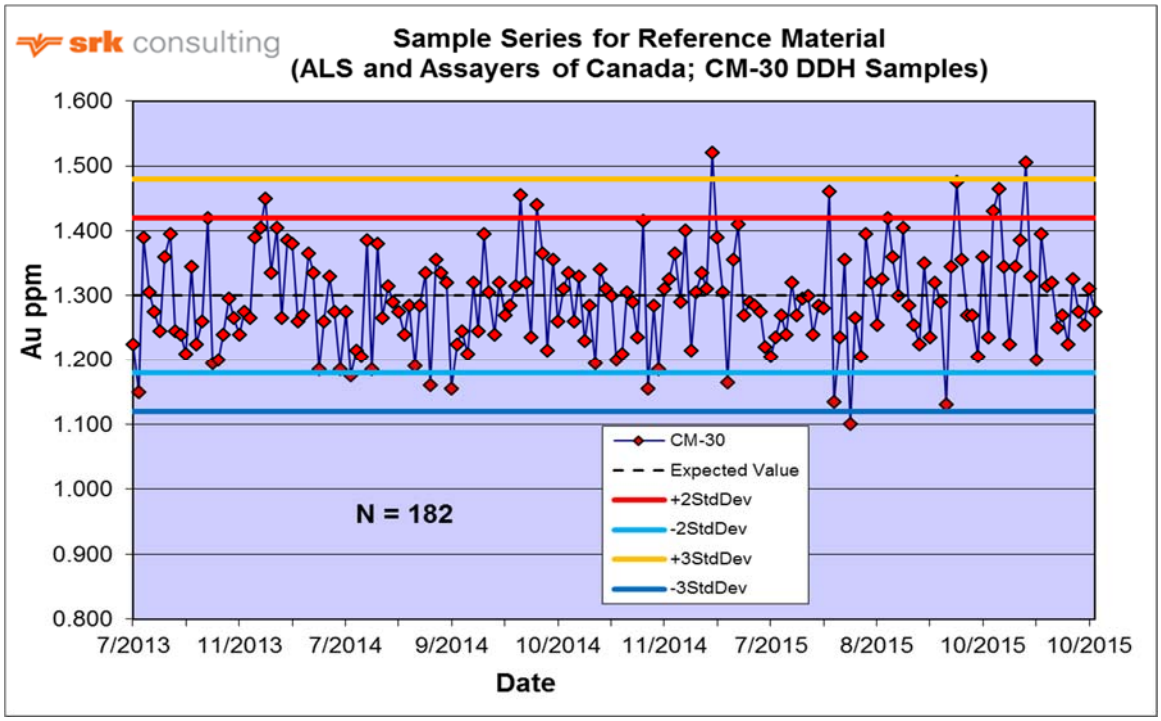


Figure 10.19: Kemess East Standard CM-30 over time – Gold
(Source: SRK, 2016)

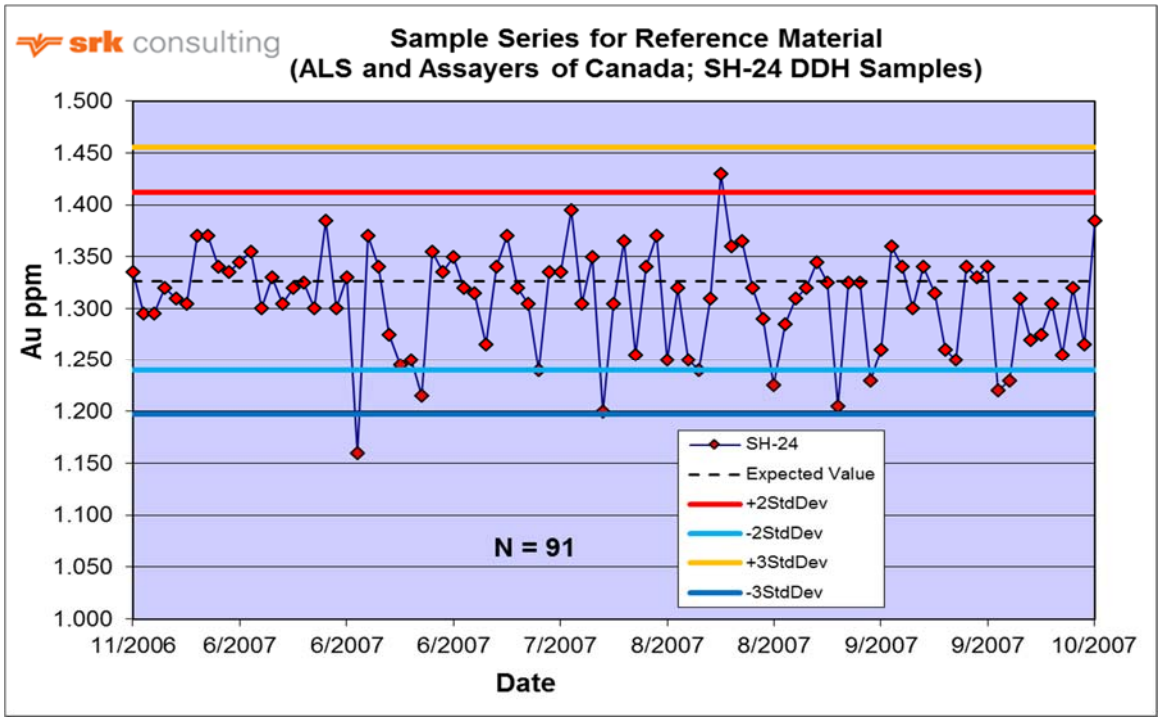


Figure 10.20: Kemess East Standard SH-24 over time – Gold

(Source: SRK, 2016)

10.7.16 Kemess East Standards – Silver

Silver standards were inserted for the 2013 to 2015 drill programs. There are no standards for silver before 2013. Silver is not a material component to the resource and the lack of standards prior to 2013 does not represent a material risk to the resource.

Four silver certified standards were used: CM-26, CM-30, CM-35, and CM-39. Standards CM-26 and CM-35 are not certified.

Standards CM-30 and CM-39 have failures of less than 2% and are considered acceptable. CM-30 is shown in Figure 10.21.

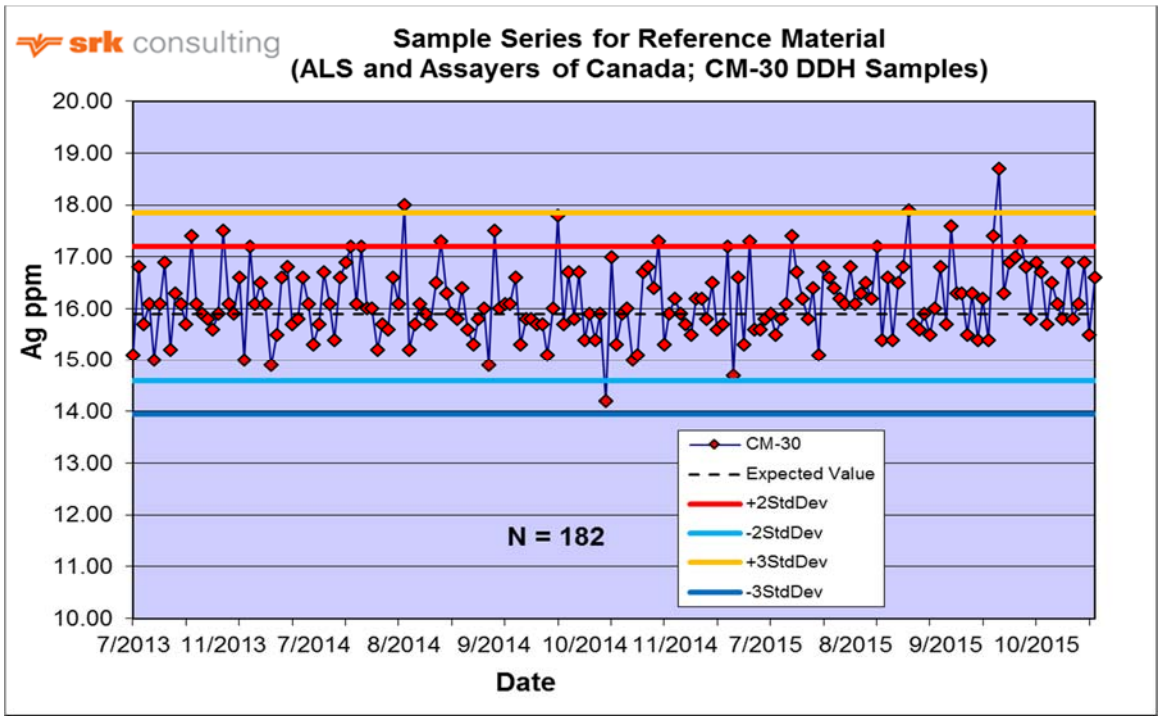


Figure 10.21: Kemess East Standard CM-30 over time – Silver

(Source: SRK, 2016)

10.7.17 Kemess East Standards – Molybdenum

Molybdenum standards were inserted for the 2013 to 2015 drill programs. There were no standards for molybdenum before 2013. Molybdenum is not a material component to the resource, and the lack of standards prior to 2013 does not represent a significant risk.

Three molybdenum standards were used: CM-30, CM-35, and CM-39.

Standard CM-30 has a 6.0% failure outside of 3 standard deviations and CM-35 has a 1.3% failure. CM-39 does not have any failures outside of 2 or 3 standard deviations. The standard results are considered acceptable.

Standard CM-35 is shown in Figure 10.22.

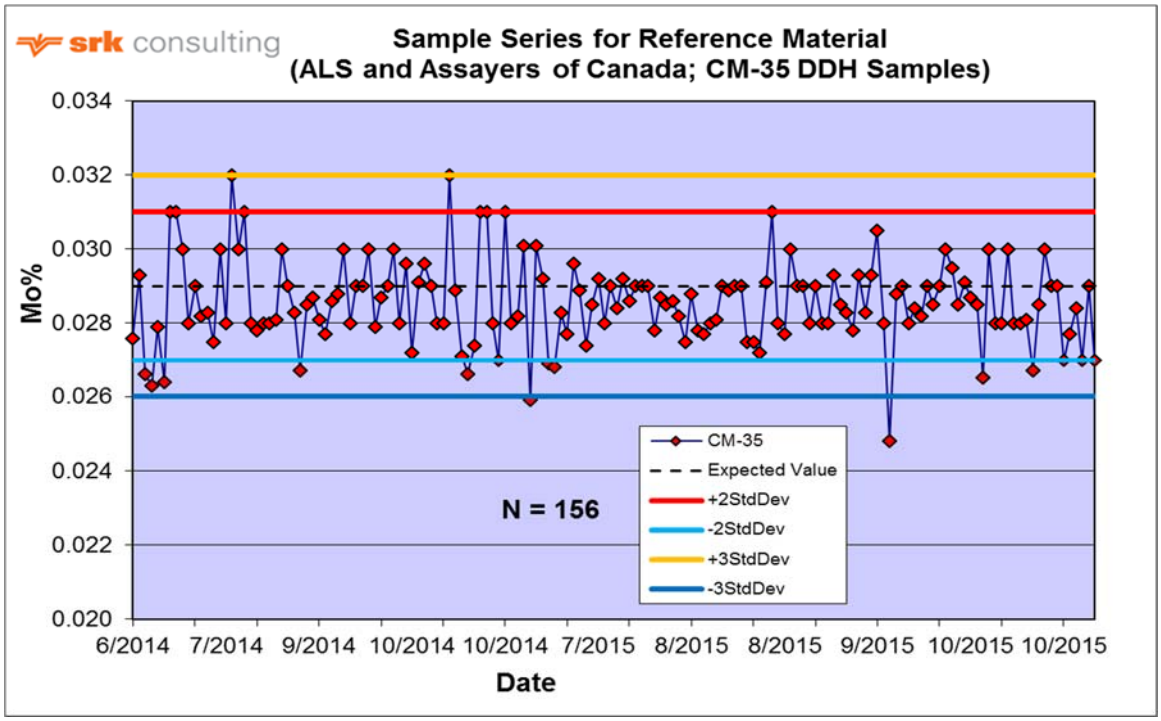


Figure 10.22: Kemess East Standard CM-35 over time – Molybdenum

(Source: SRK, 2016)

10.7.18 Kemess East Duplicates

Combined preparation and analytical precision was examined using matched pairs created by taking a second 250 g riffle-split, referred to as a “reject-duplicate”, from randomly selected original mainstream crushed sample rejects. These matched pairs of duplicates and originals were used to measure precision and how it varies with grade. Scatter plots and Relative Deviation Plots of matched-pair assays are presented in Figure 10.23 to Figure 10.26. Both copper and gold correlate well. Silver and molybdenum (not shown) also correlate well. Duplicates for all metals have greater than 85% of the pairs within 10% of each other.

10.7.19 Kemess East Duplicates - Copper

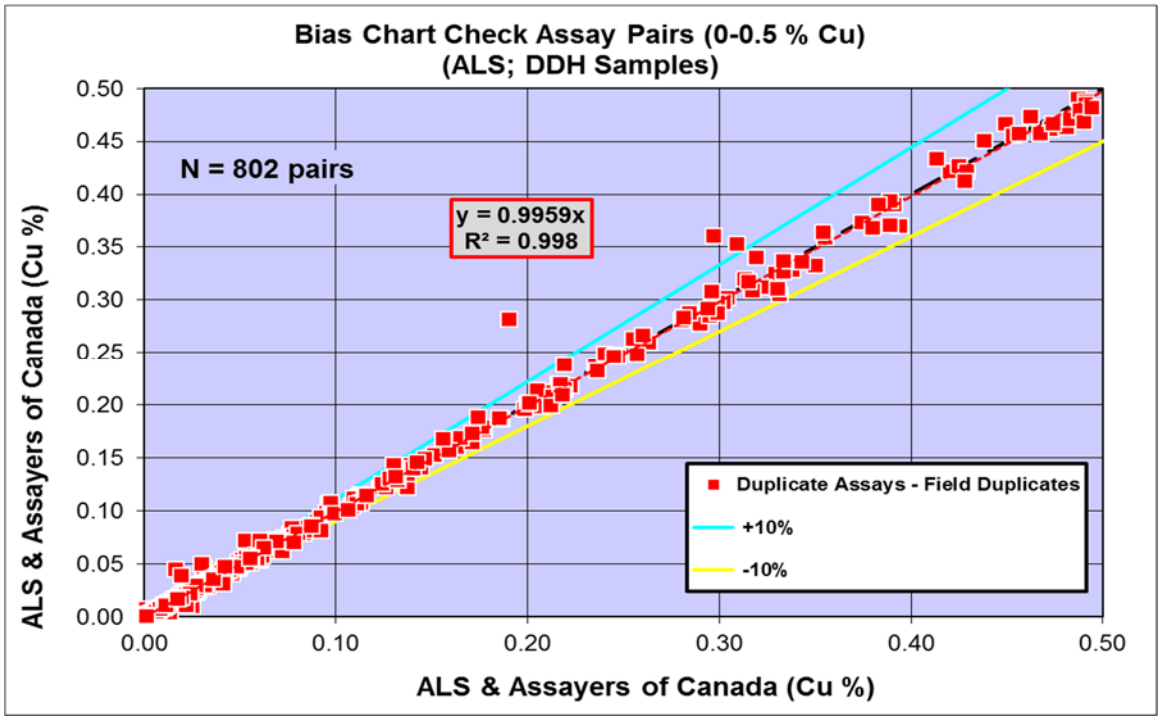


Figure 10.23: Kemess East matched pair results for reject duplicates – Copper

(Source: SRK, 2016)

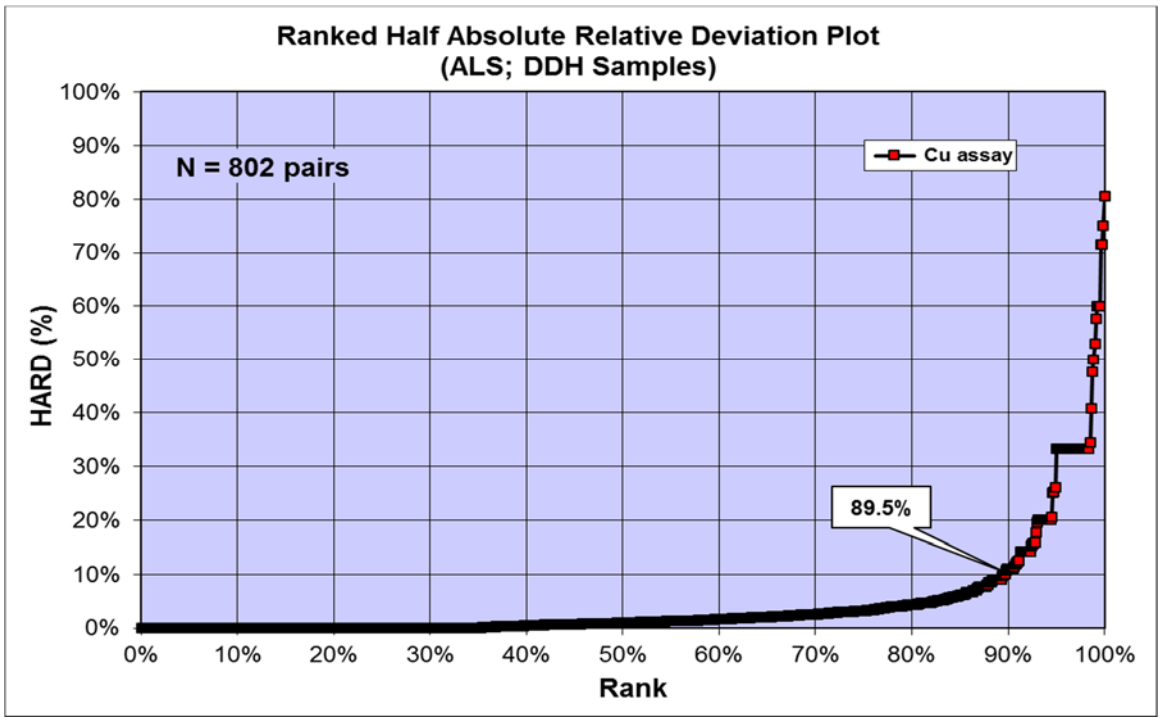


Figure 10.24 Kemess East assay paired relative deviation – Copper

(Source: SRK, 2016)

10.7.20 Kemess East Duplicates - Gold

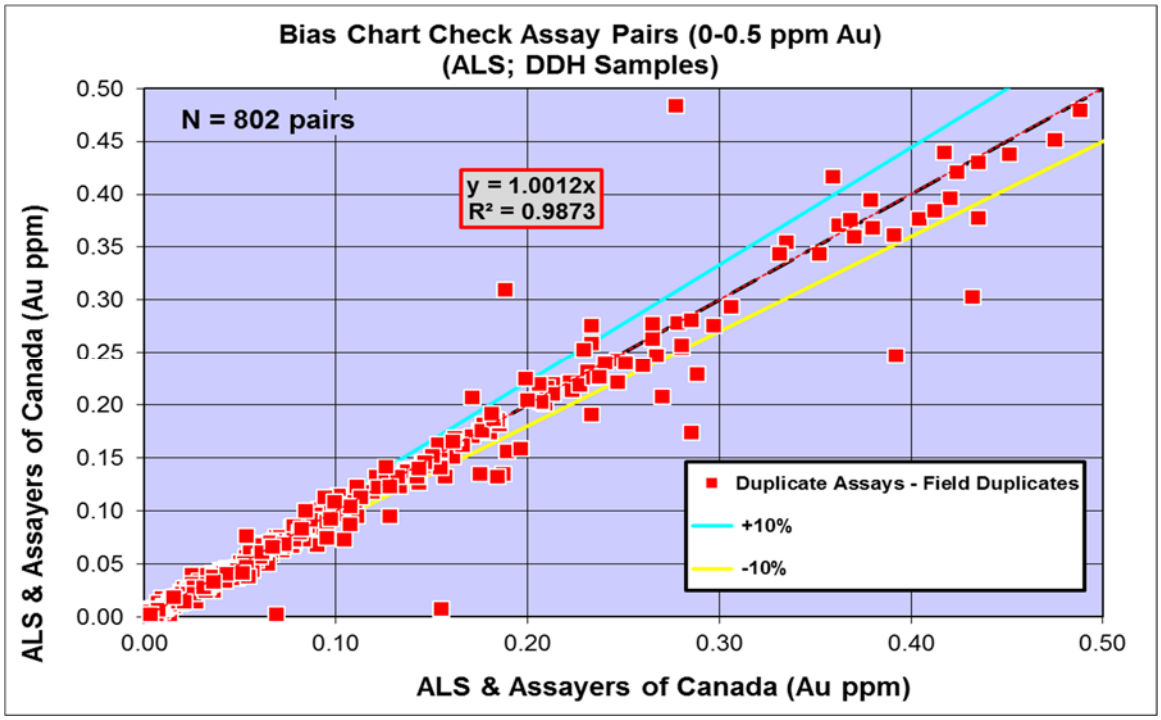


Figure 10.25: Kemess East matched pair results for reject duplicates – Gold

(Source: SRK, 2016)

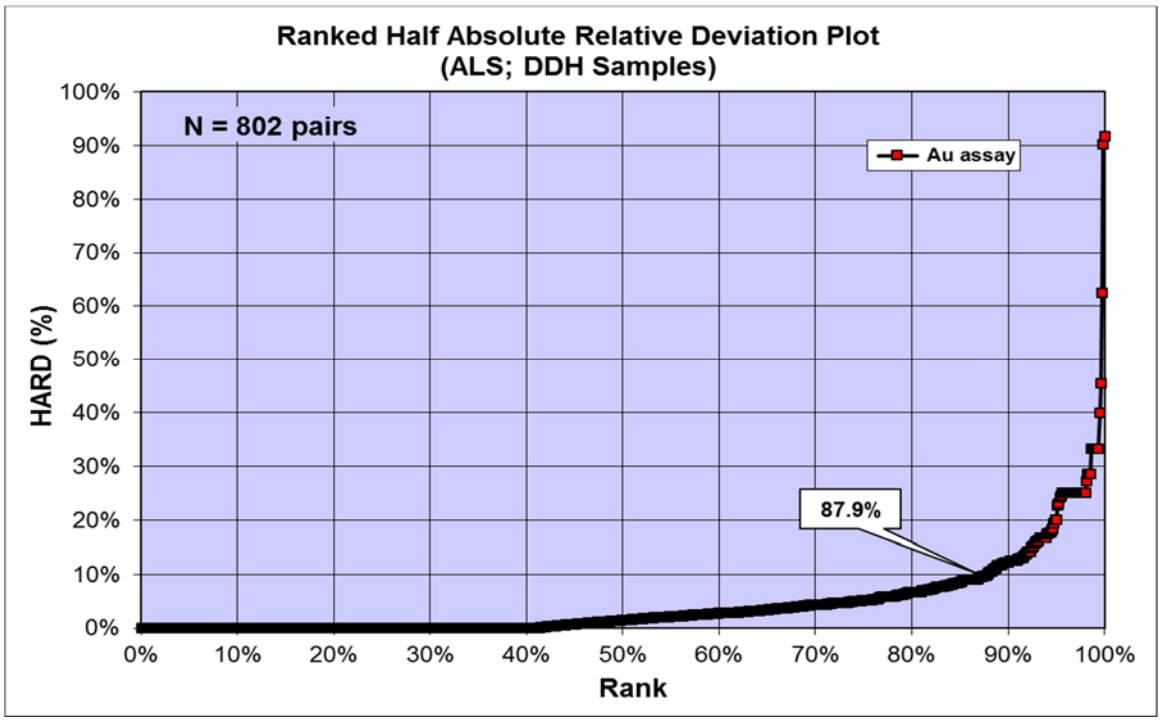


Figure 10.26: Kemess East assay paired relative deviation – Gold

(Source: SRK, 2016)

10.7.21 Kemess East Umpire Lab Duplicates

Umpire duplicate samples were sent to AGAT Laboratories to compare the results from ALS against an outside lab for the 2013 and 2014 assay data. The samples were prepared from a second split of the pulp (post pulverization). The original samples were sent to ALS and the duplicates were sent to AGAT. The umpire duplicates are considered acceptable for copper, gold, and silver assays with paired data showing 60% to 67% within 10%. Molybdenum duplicates did not perform well at only 16% of pairs within 10%. Molybdenum is not a material component to the resource, and is not considered a risk by SRK. Due to the relatively poor performance of the Umpire Lab duplicates, SRK reviewed the data in more detail to determine the respective grade ranges for the duplicate pulp samples. This review indicated that the majority of the Umpire Lab duplicates are at very low metal concentrations and well below potentially economically viable cut-offs. The number of available data points within mineralized zones is a statistically insufficient sample size to draw any conclusions. Figure 10.27 shows the copper relative bias of the original and umpire duplicate samples.

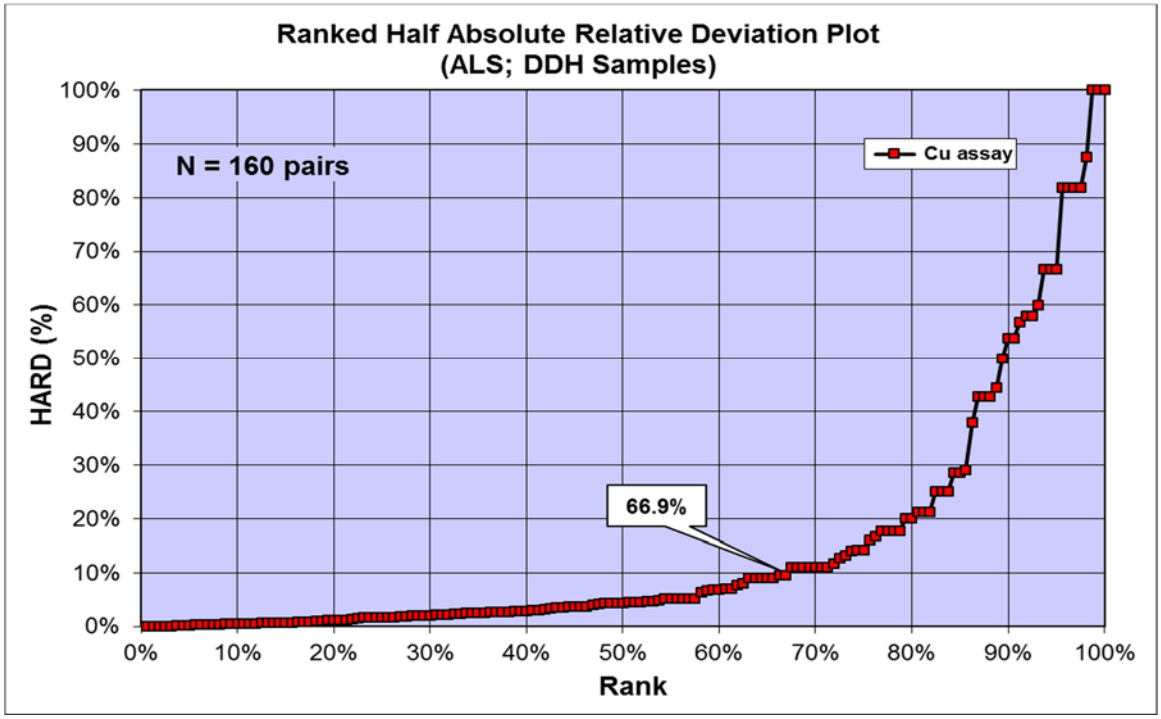


Figure 10.27 Kemess East assay paired relative deviation for Umpire duplicates – Copper

(Source: SRK, 2016)

10.8 SRK Comments

10.8.1 Kemess Underground

Evaluation of the 2002 through 2011 quality control samples indicates that the copper, gold and silver assay results for the Kemess Underground drilling programs are accurate and precise, and are therefore suitable for use in resource estimations.

10.8.2 Kemess East

Evaluation of the 2005 through 2015 quality control samples indicates that the copper, gold, silver and molybdenum assay results for the Kemess East drilling programs are accurate and precise, and are therefore suitable for use in resource estimations. SRK recommends that AuRico select pulp samples within the Kemess East resource mineralized area, and / or above potential cut-off values to be sent to the Umpire Lab. This will validate assays within the range of interest.

11 Data Verification

11.1 Kemess North/Underground Verifications by AuRico Metals Inc.

In 2011 three NQ diamond drill holes were completed in the block cave resource area. These holes were drilled primarily to confirm the previous geotechnical and geological data gathered in 2010. Subsidiary to that objective, the three 2011 holes serve as a means of verifying the geologic and resource models developed from the 2010 and earlier drilling.

Table 11.1 shows where key modelled geologic intercepts such as the KN Fault, top of Monzonite and base of Sulphate Leach were actually encountered in the 2011 verification drilling. It can be seen that the model for the top of the monzonite and base of the Sulphate Leach is confirmed to within 10 m by the three holes, however the actual position of the KN Fault is modelled less precisely to within 20 m to 30 m of the actual down-hole distance range. All three drill holes received post survey pick-ups and the collar elevations are within 1 m of the most recent topographic model.

Table 11.1: Distance (m) to Key Geological Features in 2011 drilling

Hole ID	Hole ID	Hole ID	Comments
KN-11-01	KN-11-05	KN-11-11	Geological contact
+4.6	+25	+22	Base of sulphate leach
–10	+8	+6	Top of monzonite
n/a	+26	+22	KN Fault/Hazelton

Table 11.2 compares the metal values encountered by the verification drilling with those predicted by the resource model using a 10 m x 10 m prism centred on the drillhole trace as it traverses the 2010 Block Cave resource volume. In general, the block model shows positive variance of 4.5, 7.3, and 8.1% for gold, copper, and silver respectively versus the verification drilling. It should be noted that two of the three holes encountered post-mineral Hazelton Group waste up to 26 m earlier than predicted, but also show a similar length of mineralized material above the 2010 resource volume, further underscoring the need for survey confirmation of the hole collar locations.

Table 11.2: Kemess North/Underground Analytical results from Verification Drilling Compared to Resource Estimates

Hole ID	Source	Cu (%)	Au (g/t)	Ag (g/t)	Length (m)
KN-11-01	BM	0.273	0.592	1.987	318.03
	DDH	0.211	0.464	1.792	318.03
KN-11-05	BM	0.248	0.423	1.895	497.80
	DDH	0.252	0.472	1.836	497.80
KN-11-11	BM	0.226	0.405	1.893	378.96
	DDH	0.216	0.383	1.646	378.96
	BM WW mean	0.248	0.462	1.919	
	DDH WW mean	0.230	0.442	1.764	
	BM-DDH diff %	7.3%	4.5%	8.1%	

In summary, the 2011 program confirms the results and geology of the deposit as described in the 2010 resource model. In two of the three holes, footwall waste material represented by the Hazelton Group was encountered 20–25m higher in elevation than predicted by the model, and two of the three holes yield analytical results that show negative variance when compared against the resource model. Some of this geological and analytical variance may be explained by collar survey errors noted by the departure of all the 2011 collars from topographic models.

11.2 Kemess East Verifications by AuRico Metals Inc.

The 2015 drill program was designed to infill the previously defined Kemess East mineral resource, providing additional information on grade, as well as key geological interpretations, such as lithology and alteration contacts, and major fault locations. In general, the 2015 drilling validated the previously defined geological interpretation for the Kemess East deposit, while providing additional information to increase the overall geological interpretation resolution.

11.3 Kemess North/Underground Verifications by SRK

11.3.1 Database Verification

There have been numerous data verification programs on the Kemess North/ Underground project, and the reader is referred to Section 13 of both the June 2004 and May 2005 technical report, and Section 11 of the April 2013 technical report for background detail. Both reports are available at sedar.com.

In September 2015, SRK completed an 85% audit of the Kemess Underground resource database to verify that analytical results have been entered correctly into the drillhole database used to prepare the 2015 mineral resource estimate. The audit process and results are summarized in the following sections.

All samples were checked against their assay certificates via sample number. Approximately 30,400 analyses were verified against the assay certificates. In total 85% of the Kemess Underground database was checked against the lab certificates. Overall the assay data is acceptable and any errors were minor and not of consequence.

3,960 samples could not be checked as they did not have lab certificates available. The primary reason for this is that some assays were analyzed at a lab on site and do not have assay certificates. It is possible that SRK did not receive a few certificates though SRK has checked a very large portion of the database and found it to be acceptable.

The 85% audit showed no significant errors from the resource area regarding the recording of tabulated analytical data. The Kemess Underground database for the 2016 resource was verified and can be relied upon for resource estimation.

11.4 Kemess East Verifications by SRK

11.4.1 Database Verification

In March 2016, SRK completed a 100% audit of the Kemess East resource database to verify that analytical results have been entered correctly into the drillhole database used to prepare the February 2016 mineral resource estimate.

All samples were checked against their assay certificates via sample number. Approximately 34,800 analyses were verified against the assay certificates for the 66 holes used in the Kemess East resource. In total 100% of the resource database was checked against the lab certificates. Overall the assay data is acceptable and any errors were minor and not of consequence.

The Kemess East database for the 2016 resource was verified and can be relied upon for resource estimation.

11.5 Site Visit

Chad Yuhasz, P.Geo of SRK visited the Kemess site on the dates of July 6, 2015 to July 8, 2015. During this site visit, he reviewed all drillhole logging, sampling, and QA/QC protocols. He is of the opinion that the implemented procedures and protocols, combined with the results and the additional SRK requested check re-assay program for the Kemess Underground deposit, make the results suitable for use in a resource estimation.

11.6 SRK Comment

SRK is of the opinion that the drill hole assay data are of adequate quality for the incusion in the resource estimates.

12 Mineral Processing and Metallurgical Testing

12.1 Introduction

In January 2005, Northgate Exploration Ltd. (a predecessor of AuRico) completed a Feasibility Study for their Kemess North Project. The Kemess North (KN) deposit was to be mined as an open pit with primary crushing facilities located near the pit and the ore conveyed to the existing process plant facilities that were to be expanded with the addition of a new grinding line. The metallurgical characteristics of the KN ore showed through test programs to be very similar to the KS hypogene ore such that minimal modifications were expected to the KS process plant flowsheet.

An initial metallurgy report for the KN deposit was prepared for Northgate Exploration by Lakefield in 2003. In 2011, KUG metallurgical work (report KM2911) was initiated by Northgate Minerals at G&T Metallurgical Services Ltd.'s Kamloops Canada lab to support the KUG project and verify the previous observations. Additional drill core sampling was completed in 2012 with samples delivered to G&T for determination of grinding parameters (report KM3442). As a result of 2013 KUG FS recommendations, additional grindability testwork was carried out in 2015 by ALS Metallurgy (formerly G&T, reports KM4730 and KM4864), with results therefrom used to define KUG process plant flowsheet requirements.

The KUG ores share a number of characteristics with the hypogene ores of the KS deposit.

- Both deposits show limited surface oxidation of the sulphide surfaces.
- Impurity elements in the final concentrate will occur in low concentrations.
- The sulphides are coarse grained and are adequately liberated for rougher flotation at a grind P_{80} of 150 µm.
- Average ball mill work indices are lower than the majority of porphyry deposits.
- The samples of KN ore used in the 2003 Lakefield program averaged 13.8 kWh/t while ball mill work indices for KS ranged from 13.8 kWh/t to 15.0 kWh/t. In the recent 2012 G&T program (ALS 2012, KM3442), the average work index for the monzonite ores was 14.1 kWh/t and for the Takla ores was 15.3 kWh/t. However, waste from the Toodoggone and Hazelton zones (that could infiltrate panel cave production) was hard, averaging 20.3 kWh/t for the two samples tested. SMC analysis in 2012 categorised the monzonite sample hardness as medium to moderate with the Takla samples being moderate to hard. SMC tests and Bond ball mill work index tests completed on the five ore domains by ALS in 2015 (ALS 2015a, KM4730) indicated an average Bond work index for the tests of 13.4 kWh/t (12.3 kWh/t to 14.7 kWh/t) and average SMC (A*b) of 50.5 (39.8 to 65.5), confirming previous testwork.
- The KUG ore is expected to have a higher average Cu grade of 0.27% Cu compared to 0.22% Cu for KS hypogene. Average Au grades are expected to be lower at 0.54 g/t Au versus 0.75 g/t Au for KS hypogene ore.

- The pyrite content of KUG ore will average 4% compared to 1% for the KS hypogene ore. Since pyrite contains finely disseminated gold, pyrite rejection in flotation has the potential to carry a larger portion of gold to tailings.
- Rougher concentrate mass pull is estimated to be 18%.
- Rougher flotation with KUG ores will produce a bulk sulphide float. Rougher concentrates have to be reground to a P_{80} of ≤25 µm for cleaner flotation to produce quality concentrates. Rougher concentrate regrind testwork results reported by ALS in 2015 (ALS 2015, KM4730 and KM4864) indicate that additional milling/grinding capacity needs to be added the plant flowsheet.
- Existing flotation cell capacity and dewatering equipment will be adequate for treatment of KUG ores.
- The ultimate settling density of KUG tailings will be at least 66% solids at a pulp density of 1.74 g/cm^3.
- The Net Neutralization Potential for all KUG composites is negative, averaging –50 t $CaCO_3$ per 1,000 t of tailings. The tailings will be acid generating unless stored subaqeously. Note that this feasibility study contemplates KUG tailings being stored underwater within the existing and permitted KS open pit tailings facility (KUG TSF).

Results of the 2003 Lakefield cycle tests with seven composites of different KN ore zones led to the projection of plant performance for the KUG ore as shown in Table 12.1.

Testwork completed at G&T Metallurgical Services Ltd (G&T 2011, KM2911) located in Kamloops, Canada, resulted in average rougher recoveries of 93.8% Cu and 90.2% Au for the 34 flotation tests completed (Table 12.2). These results are similar to the rougher results observed in the 2003 Lakefield test program. The cleaner results were dependent on feed grade and ore zone.

Table 12.3 compares the locked cycle test results from the 2003 Lakefield metallurgical program and the 2011 G&T test program. The 2003 samples represent the deep level KN open pit ore. The sample in the 2011 testwork has been blended from ore samples representing the first eight years of operation of the KUG panel cave project.

Table 12.4 provides a summary of the locked cycle tests completed in 2011 on a range of sample composites. This testwork provided an opportunity to look at the results of finer regrinding of the rougher concentrate.

Table 12.1: Projected Plant Metallurgy for KN Open Pit Ore, Lakefield 2003

Ore type	Concentrate Grade (% Cu)	Copper Recovery (%)	Gold Recovery (%)	Silver Recovery (%)	Ag/Au (in Concentrate)
Middle zone ores	22 to 24	86	50 to 60	40 to 45	3.0
Lower zone ores	24 to 26	92	60 to 70	50 to 55	3.0

Table 12.2: KUG Test Results From the 2011 Metallurgical Test Work Program (G&T 2011, KM2911 open circuit tests)

Comp	# of Samples	# of Tests	Material	Average Grades		Ratio	Rougher Recovery			Cleaner Recovery (22% Cu conc.)		
		Rougher/Cleaner		Au (ppm)	Cu (%)	Au/Cu	Mass (%)	Cu (%)	Au (%)	Mass (%)	Cu (%)	Au (%)
CP1	288	16/6	HG monzonite	1.84	0.65	2.84	23.2	95.5	90.8	2.89	89.64	66.33
CP2	186	1/1	MG monzonite	0.82	0.38	2.15		93.3	87.6	1.40	86.20	74.40
CP3	373	1/1	LG monzonite	0.4	0.24	1.69		90.9	86.4	0.87	78.68	63.04
CP4	402	1/1	Mixed lithology	0.46	0.23	1.98		91.4	91.7	0.91	84.01	71.73
CP5	548	6/1	Upper volcanics	0.45	0.24	1.89	16	91.9	89.3	1.01	82.65	48.66

Table 12.3: KN 2003 and KUG 2011 Locked Cycle Test Comparison

Sample		Head Grades		Cu Concentrate Grades		Concentrate Recoveries	
Description	Location	Cu (% Cu)	Au (g/t Au)	Cu (% Cu)	Au (g/t Au)	Cu (%)	Au (%)
KVK 2003 report							
SC2	Deep zone	0.36	0.77	25.7	43.7	94.6	75.3
SC3	Deep zone	0.24	0.44	24.7	30.7	90.3	60.9
G&T composite (blend rep years 1 to 8)							
LCT 1		0.36	0.78	23.5	40.5	91.3	73.2

Table 12.4: G&T Locked Cycle Tests, 2011 KUG Metallurgical Test Work (G&T 2011, KM2911)

Composite	Zone	Primary Grind P80 (µm)	Regrind P80 (µm)	Head Grades		Cu concentrate Grades		Concentrate Recoveries (%)	
				Cu (%)	Au (g/t)	Cu (%)	Au (g/t)	Cu	Au
CP1	High grade monzonite	144	21	0.67	1.82	26.7	55	93.9	71.4
		144	20	0.66	1.81	23.3	50	94.5	73.5
CP2	Medium grade monzonite	134	18	0.39	0.78	20.7	38	89.8	83.3
		134	14	0.37	0.81	23.9	44	95.2	81.4
CP3	Low grade monzonite	116	21	0.23	0.35	18.9	23	86.4	72.7
		116	14	0.23	0.35	25.0	31	89.4	71.9
CP4	Mixed lithology	154	19	0.23	0.43	21.2	31	90.6	72.3
		154	16	0.23	0.42	24.8	36	89.7	75.9
CP5	Upper volcanics	127	21	0.25	0.44	21.9	25	87.1	57.2
		127	17	0.25	0.48	21.8	30	91.3	66.4
Blend of all zones		123	16	0.36	0.78	23.5	41	91.3	73.2

The observations by G&T from this set of locked cycle tests were:

- 91% of the feed copper and 73% of the feed gold in the blend composite were recovered to a final concentrate containing 23.5% Cu and 41 g/t Au;
- The final copper recovery for cycles IV and V in the first series of locked cycle tests for each composite ranged from 86 to 94% and the recovery for gold ranged from 57 to 83%. The final copper concentrate grade ranged from 19% to 27% Cu;
- In the second set of locked cycle tests (after adjustments to reagents or finer regrind size), the copper recovery ranged from 89% to 95% and the gold recovery ranged from 66% to 81%. In this set of tests, the copper concentrate grade ranged from 21% to 25% copper; and
- The majority of gold losses were shown to occur in the cleaner scavenger tailings and attributed to gold particles interlocked with rejected pyrite or finely disseminated gold particles.

For the purpose of the preliminary economic analysis (PEA) completed in 2011, it was recommended to base the metallurgy on:

- Copper concentrate grade of 22% Cu,
- Copper recovery of 91%, and
- Gold recovery of 72%.

The results of the locked cycle tests completed in 2011 on domain composites supported this conclusion. An estimate of the project metallurgy based on the distribution of ore indicates recoveries of 91.3% and 71.9% for copper and gold respectively (Table 12.5) – which further supports the initial recommendations.

Table 12.5: KUG Domain Locked Cycle Results (G&T 2011, KM2911)

Domain Composite	Copper Concentrate			Distribution
	Grade (% Cu)	Cu Recovery (%)	Au Recovery (%)	(%)
CP1	22.8	94.4	73.5	8.7
CP2	23.9	95.2	81.4	12.4
CP3	25.0	89.4	71.9	13.5
CP4	24.8	89.7	75.9	21.1
CP5	20.9	91.0	66.4	39.9
Average	22.9	91.3	71.9	95.6*

* Distribution does not add to 100% as Hazelton waste (CP6) is 4.4% of feed and not tested

12.2 Sample Description

For the 2003 Lakefield metallurgical program, company exploration geologists selected 1,737 samples from diamond drill cores representative of the KN deposit. These were classified by depth and future mining area, and the samples were grouped into four domains. The samples of each domain were then grouped into seven super composites. The samples were received at Lakefield Research in December 2002.

For the 2011 G&T program (G&T 2011, KM2911), five mineral types were identified and samples collected and composited as per the following description (Table 12.6).

Table 12.6: 2011 KUG Composite Descriptions

Composite	# of Samples	Material	Average Grades		Ratio
			Au (ppm)	Cu (%)	Au/Cu
CP1	288	HG monzonite	1.84	0.65	2.84
CP2	186	MG monzonite	0.82	0.38	2.15
CP3	373	LG monzonite	0.40	0.24	1.69
CP4	402	Mixed lithology	0.46	0.23	1.98
CP5	548	Upper volcanics	0.45	0.24	1.89

A blended sample used for the locked cycle test program was made up of 25% CP5, 25% CP3, 35% CP2 and 15% CP1, representing the early production period.

The drill core samples for the 2012 supplemental hardness tests were grouped into the same five categories with additional samples representing CP6. This additional composite (CP6) represents the upper part of the caving zone and Hazelton (Toodoggone) waste rock that may be drawn in with the panel cave during the early years of cave mining.

12.3 Mineralogical Characteristics of KUG Ore

The host rock of the KUG deposit shows some clay and anhydrite formation in the near surface zone followed by chlorite and sericite alteration, with silicification and the presence of quartz dominating with increasing depth. Hematite/magnetite has also been encountered.

The pyrite content of samples ranges from 1.4% to 8.1%. The pyrite is coarse grained and essentially liberated at a primary grind P80 of 150 µm. The pyrite will be clean (i.e. not activated) and can be rejected by conventional flotation techniques. Chalcopyrite ranges from 0.5% to 1.2% or 0.16% Cu to 0.44% Cu equivalent.

The principal copper mineral is chalcopyrite. It occurs in intimate association with pyrite. Fine grinding will be required (P80 of 20 µm) to produce acceptable liberation of chalcopyrite from pyrite.

Secondary copper minerals, bornite and chalcocite occur as trace minerals and are not of economic significance.

Gold will average 0.54 g/t Au, ranging 0.18 g/t Au to 0.96 g/t Au. Gold occurs as native gold and electrum. The silver content of electrum ranges from 10% to 20%.

After fine grinding, ±30% of the gold will be liberated, with particle size ranging from 3 µm to 25 µm. Gold in chalcopyrite presents another fraction of ±30%. The remaining gold will occur as fine inclusions in pyrite with a minor fraction (10%) embedded in the host rock.

12.4 Specific Gravity and Ball Mill Work Indices

Specific gravity determinations were undertaken during the 2003 Lakefield metallurgical program with representative samples pycnometer tested, with the specific gravity averaging 2.78.

The Bond ball mill work indices were determined for all the composites in the 2003 program. The average BWI for the KN ore determined to be 13.8 kWh/t. In almost all cases, BWI testing was done at a closing screen size of 106 µm.

A specific program looking at work indices for the various ore regimes was completed in 2012 (ALS 2012, KM3442) and reported the average BWI for the two main ore classifications:

- Monzonite 14.1 kWh/t (range 12.3 kWh/t to 18.3 kWh/t), and
- Takla 15.3 kWh/t (range 14.0 kWh/t to 17.4 kWh/t).

The process plant throughput will be influenced by ore hardness and the blended draw of ore through the panel cave. In the early years of cave production there is a possibility of drawing Hazelton waste with the ore. Two Hazelton drill core samples were tested as part of the 2012 program and reported work indexes of 18.3 kWh/t and 22.2 kWh/t.

A cave production profile was provided by AuRico, based on expected quarterly blends and SRK derived, PCBC™ production schedule. The average BWI for the different ore domains was applied to annual blends (derived by combining each group of four quarters) and the blended BWI is presented in Table 12.7. The table indicates that the average annual blended BWI will vary from 15.2 kWh/t in the early years (due to the dilution by Hazelton waste) to 13.2 kWh/t in the later years when production will be primarily a blend of CP4 and CP5 domains. SMC test results also vary over the mine production plan – showing higher SAG mill resistance (lower A*b values) for Year 4 through Year 8. The variation in A*b values is more significant than the variation in BWI values, suggesting the SAG mill could be limited under some blends of ore domains. As such, development of a cave production plan that minimizes this impact should be developed as well as investigating means to increase mill throughput.

Table 12.7: Blended KUG Ball Work Index and SMC (A*b)

Domain	BWI (kWh/t)	SMC A*b	1	2	3	4	5	6	7	8	9	10	11	12	13
CP1	13.4	61.5	62.9%	51.8%	36.3%	26.0%	17.0%	7.7%	1.8%	0.1%	0.0%	0.0%	0.0%	0.0%	0.0%
CP2	13.3	51.9	0.1%	4.9%	13.8%	16.7%	15.4%	17.5%	19.6%	20.8%	18.5%	13.8%	7.3%	3.0%	1.1%
CP3	14.7	39.8	4.1%	12.2%	16.2%	15.7%	14.2%	17.3%	24.7%	38.9%	30.7%	16.1%	6.0%	2.1%	0.9%
CP4	12.4	65.5	0.0%	0.3%	0.9%	1.5%	1.6%	2.3%	4.5%	13.8%	29.4%	47.1%	56.2%	56.1%	51.8%
CP5	14.2	42.2	8.5%	14.1%	22.0%	34.2%	48.5%	52.3%	46.0%	22.7%	19.8%	22.7%	30.4%	38.8%	46.2%
Hazelton	20.3	24.0	24.4%	16.7%	10.8%	5.9%	3.3%	2.9%	3.4%	3.8%	1.6%	0.3%	0.1%	0.0%	0.0%
Blended BWI			15.2	14.8	14.5	14.3	14.2	14.2	14.3	14.2	13.8	13.3	13.2	13.2	13.3
Blended SMC (A*b)			49.8	49.4	48.4	47.7	46.4	45.0	44.3	45.8	49.8	54.1	55.9	55.5	54.4

BWI testing was done at a closing screen size of 106 µm

12.5 Fineness of Grind in Primary and Regrind Circuits

The target fineness of grind for the primary grinding circuit will remain at the historic KS P80 of 150 µm. Testwork has shown that adequate liberation of the sulphides for rougher flotation will be achieved. Copper recovery in the rougher circuits will range from 90% to 96%. Gold recoveries in the rougher flotation will range from 85% to 92%, but approximately 15% to 20% will be finely disseminated in pyrite and therefore rejected in the cleaner flotation, with the various stages of upgrade.

Regrinding of the rougher concentrate will have a significant impact on copper concentrate quality and gold recovery. Batch tests were conducted for the 2003 Lakefield test program (Lakefield 2003) in which the regrind fineness was progressively increased.

Concentrate grades rise as fineness increases while metal losses in cleaner flotation worsen only slightly within the range of regrinds tested. Typical results from the 2003 Lakefield test program are shown in Table 12.8. Table 12.9 is a comparison of the locked cycle flotation tests completed on the lower depth ores in 2003 and the recent locked cycle test completed at G&T in 2011 (G&T 2011, KM2911).

Table 12.8: Determination Of Regrind Product (2003)

Sample test No.	Fineness Regrind	Feed		Concentrate Analysis		Recoveries (%)	
Super comp 1	**P80**	**Cu (%)**	**Au (g/t)**	**Cu (%)**	**Au (g/t)**	**Cu**	**Au**
SC1-LCT2	34 µm	0.20	0.35	17.5	21.7	89.4	62.4
SC1-LCT3	12 µm	0.19	0.36	25.3	33.8	89.6	61.8

Table 12.9: Locked Cycle Test Comparison (SC2 and SC3 from 2003 vs. 2011 blend)

Sample	Head Grades		Cu Concentrate Grades		Concentrate Recoveries	
	Copper (%Cu)	Gold (g/t Au)	Copper (%Cu)	Gold (g/t Au)	Copper (%)	Gold (%)
SC2 (lower ore zone)	0.36	0.77	25.7	43.7	94.6	75.3
SC3 (lower ore zone)	0.24	0.44	24.7	30.7	90.3	60.9
2011 G&T blended composite:	0.36	0.78	23.5	40.5	91.3	73.2

Copper concentrate grades increased in similar proportion for all samples tested with increasingly finer grinding of rougher concentrates. The target fineness for the regrinding of rougher concentrates will be a P80 of 20 µm.

The results from the flotation work completed during the 2011 G&T program also determined that there was an advantage to finer regrind size (Table 12.4) showing improved metallurgical performance with decreasing regrind size.

In 2010, the Kemess South operators completed an evaluation of millpebs (i.e. irregularly shaped steel grinding media) for grinding media in the regrind ball mill circuit when production switched to the higher pyrite ratio, east KS pit ore. This mill feed required a finer liberation mesh to achieve acceptable recoveries compared to the previous hypogene ores. Observations using the millpebs media included increased recoveries of copper and gold and a reduction of metal lost from overgrinding or sliming in the grinding circuit. It is planned to continue with the use of millpebs media in the ball mill for the KUG ore regrind.

Preliminary regrind energy requirements were based on Bond BWI calculations that were adjusted for expected energy efficiencies attributed to stirred mill technology. In 2015, domain samples were delivered to ALS Kamloops where a suite of regrind energy tests were completed on lab-generated rougher concentrate to better assess the regrind energy requirements (ALS 2015, KM4730 and KM4864). The tests included the following:

- Eliason Tests – small scale, stirred mill test completed to determine the specific energy of regrinding.
- Levin Tests – small scale ball mill tests used to assess regrind ball mill energy requirements. For KUG ore, the Levin results are used to project the P80 product size based on the existing ball mill that will feed the new stirred mill(s).
- Specific Energy Tests – Stirred mill vendor-approved test (“signature plot”) for estimating power requirements down to P80 of 20 µm or finer.

For the Eliason tests, concentrates for the five ore domains were generated by standard rougher flotation tests. The Eliason test procedure was used to determine the energy requirements for each domain at a range of product sizes. This was also done for two blended samples. The projected energy requirements determined by the Eliason tests are provided in Table 12.10. The Levin test results are summarized in Table 12.11.

Table 12.10: Eliason Test Results, KUG Ore (ALS 2015a, KM4730)

Domain	Cumulative kWh/t @ Prod P_{80} of 20µm	Cumulative kWh/t @ Prod P_{80} of 15µm
CP1	34.5	59.5
CP2	23.3	45.2
CP3	27.3	67.3
CP4	44.0	107.2
CP5	34.0	71.1
Blend 11	30.6	64.0
Blend 22	36.8	85.5

Note: 1. Blend 1: 10% CP1, 20% CP2, 20% CP3, 0% CP4, 50% CP5
2. Blend 2: 0% CP1, 12% CP2, 13% CP3, 50% CP4, 25% CP5

The Specific Energy results from the signature plot tests were analysed by the equipment vendor who generated the following energy projections:

- Blend 1: Cum specific energy (kWh/t) = 17,499 P80 (-2.074)
- Blend 2: Cum specific energy (kWh/t) = 233,341 P80 (-2.961)

It was noted the large difference in cumulative specific energy between the two blends, particularly for a product P80 size finer than 20 µm.

Table 12.11: Levin Test Results, KUG Ore (ALS 2015b, KM4864)

Domain	Blend 1		Blend 2	
Size	Size P_{80}, µm	**Cumulative kWh/t**	Size P_{80}, µm	**Cumulative kWh/t**
Feed	150	0	133	0
Prod 1	54	5	44	5
Prod 2	46	7.5	36	10
Prod 3	38	15	31	20
Prod 4	29	30	25	40
Projected	**20**	**87.8**	**20**	**91.0**

The Levin test results showed that 88 kWh/t to 91 kWh/t would be required for a ball mill to achieve the target product P80 size of 20 µm. The signature plot results indicated a stirred mill could achieve this using 21 kWh/t to 37 kWh/t. The specific energy requirements increased significantly for a product P80 size of 15 µm.

12.6 Grinding Mill Capacity

The KUG project is based on a primary processing capacity of 9 Mt/y. This will be accomplished with only one of the original KS grinding lines to process KUG ore.

Analysis of grinding operating data from 2001 to 2010 indicates that the grinding specific energy generally increased as the KS open pit deepened and again when the East Pit ore was processed (Table 12.12).

Table 12.12: Historical Kemess South Grinding Energy Requirements

Year	Tonnes	Availability (%)	Grinding Energy (kWh)	Grinding SE (kWh/t)	Power Utilization (%)
2001	15,360,640	83.5	178,845,717	11.6	67.9
2002	17,308,131	88.7	223,799,692	12.9	80.0
2003	18,701,060	91.4	236,377,660	12.6	82.0
2007	17,801,734	90.8	253,861,504	14.3	80.9
2008	16,948,339	85.4	237,877,024	14.0	75.6
2009	18,352,556	91.1	252,676,982	13.8	80.6
2010	18,748,465	90.8	284,767,434	15.2	90.8

Data from the 2003 Lakefield test report was used to develop a preliminary design work index for the KUG Project (Table 12.13). The average BWI was determined to be 13.8 kWh/t.

Table 12.13: Kemess North Bond Ball Mill Work Index Tests

Description	Deposit Location	Identity	# of Samples	Rod Mill Work Index (kWh/t)	Ball Mill Work Index (kWh/t)
Domain 1					
Strong silicification and quartz flooding with minor to moderate chlorite and sericite alteration	Lower zones SC2	1A-1	111	14.4	14.3
		1A-2	54		14.1
		1A-3	76	14.3	13.0
		1B-1	82	15.5	14.2
Domain 2					
Strong chloritic, weak to moderate sericite, moderate silicification	Lower zones SC3	2A-1	70	18.7	15.9
		2A-2	66		14.8
		2A-3	54		16.1
		2A-4	54	17.3	13.1
	Middles zones SC4	2B-1	60		15.8
		2B-2	71	16.7	14.4
		2B-3	64		14.2
		2B-4	64	14.3	13.1
Domain 3					
Chlorite and sericite alteration with minor silicification	Middles zones SC5	3A-1	48		14.2
		3A-2	54	15.4	14.2
		3A-3	61		11.4
		3A-4	N/A		
		3A-5	52	16.9	16.5
		3A-6	50		14.6
		3A-7	51		13.7
	Upper zones SC1	3B-1	59		11.8

Description	Deposit Location	Identity	# of Samples	Rod Mill Work Index (kWh/t)	Ball Mill Work Index (kWh/t)
	Upper zones SC6	3B-2	50	14.4	12.0
		3B-3	5		12.3
		3B-4	45		10.2
	Upper zones SC1	3B-5	50		11.0
	Upper zones SC6	3B-6	43	16.7	11.1
		3B-7	54		14.0
		3B-8	54	19.3	13.4
	Upper zones SC1	3B-9	50		14.3
		3C-1	59		17.0
Domain 4					
Clay and anhydrite alteration	Near surface SC7	4-1	40		14.7
		4-2	38		15.4
Average				**16.2**	**13.8**

Source: Klaus Konigsmann KVK Consulting, 2003

For the preliminary design of the KUG process flowsheet, the BWI was based on the average work index measurements for the lower and middle ore zones, at 14.2 kWh/t. This is very similar to the KS average BWI of 14.6 kWh/t (Table 12.14).

Table 12.14: Kemess South Bond Ball Mill Work Index vs Ore Domain

Domain	Alterations	Bond Work Index (kWh/t)
CP5	Weak sericite	16.5
CP4	Weak potassium feldspar-weak sericite	15.4
CP4	Sericite-weak potassium feldspar	15.4
CP3	Potassium feldspar-sericite-weak chlorite	13.9
CP2	Chlorite-potassium feldspar-sericite	14.2
CP1	Sericite-potassium feldspar	13.0
CP1	Chlorite-sericite-weak potassium feldspar	13.6
Average		**14.6**

The results generated by the Bond BWI and SMC tests completed at ALS in October 2015 (ALS 2015a, KM4730), shown on Table 12.15, confirm the previous analysis. In fact, the average ore work index from the 2015 test program is slightly lower at an average of 13.4 kWh/t.

Table 12.15: KUG Ore, 2015 ALS Bond Ball Mill and SMC test results

Domain	BWI (kWh/t)	SMC Test A*b
CP1	13.4	61.5
CP2	13.3	51.9
CP3	14.7	39.8
CP4	12.4	65.5
CP5	14.2	42.2
Hazelton	20.3	24.0
Average ($_{Ore}$)	**13.4**	**50.5**
Average ($_{Ore + Hazelton}$)	**14.6**	**46.1**

With the grinding energy testwork results from ALS, the existing grinding circuit capabilities were re-evaluated based on a comparison to the historic KS mill operating data. Table 12.16 evaluates the Kemess South hypogene ore operating data. Plant data indicated that the mill power utilization

was between 80% and 85% and that plant availability was 91.2%. Assuming a product grind P80 of 150 µm, the SAG mill efficiency for power based calculations was estimated at 1.40.

In 2009, the KS east pit zone became the source of ore for the processing operations. This ore was reported to be harder and to have a higher ratio of pyrite to copper than the previous Kemess South ores. Samples of this ore were tested at SGS and the indication was that the BWI was 14.9 kWh/t to 15.3 kWh/t. In addition, knowing that the copper feed grades were dropping, the operations team completed some grinding circuit optimizations to try to use the installed mill capacity more effectively. Mill availability, still influenced significantly by the tailings pumping system, remained about the same, but the power utilization in the grinding mills increased to over 90%. An evaluation of the grinding circuit for the east zone ore (Table 12.17), similar to that for the hypogene ore, indicated that the SAG efficiency factor for 2010 was 1.50. For this evaluation, the BWI was assumed to be the average of the two samples, which is 15.1 kWh/t. Production through the mill was 18.75 Mt.

Various ore types for the KUG deposit were modelled using the parameters that have been established from previous calculations and recent testwork. A mill availability of 92% was assumed for the models. With KUG tailings being deposited in the KS open pit, the tailings pumping system would be simplified and would not be expected to have the same impact on continuous operations.

The monzonite ore grinding model (Table 12.18) indicates that the grinding mills would use about 80% of the installed power to achieve the target throughput.

Similarly, the Takla ore grinding model (Table 12.19) indicates that the grinding mills would use about 87% of the installed mill power to achieve target throughput.

Table 12.7 provides a summary of the annual ore blends from the panel cave based on the KUG cave production schedule derived by SRK using PCBCTM software. During the early years of operation, the component of feed from the harder Hazelton waste that is drawn into the cave zone is higher. This influences the grinding model. In Year 1, Hazelton waste is projected to supply a significant portion of the mill feed, resulting in the average weighted BWI for the year being 15.2 kWh/t. Given production ramp-up over the initial years of cave production, the process plant operating schedule will provide flexibility to accommodate the harder ore during the early production period.

The process plant throughput will fluctuate somewhat as the work index varies. Based on the testwork results, there are some areas of hard ore that may also influence the build-up of critical size ore within the mill charge. Critical size was not an identified problem during the operation of Kemess South. There were occasions when the ability to reduce the critical size may have been beneficial but no long-term economic benefit was identified.

Table 12.16: Kemess South 2003/2004 Hypogene Ore Grinding Circuit Evaluation

Primary SAG Mill				Secondary Ball Mill			
Mill operating parameters and power required				**Mill power required**			
Daily feed tonnage	49,610	t/d		Daily feed tonnage	49,610	t/d	
Mill availability	91.2	%		Mill availability	91.2	%	
Mill feed Rate	2,267	tph		Mill feed rate	2,267	tph	
Feed size, F80	150,000	µm		Feed size, F80	925	µm	
Product size, P80	925	µm		Discharge size, P80	150	µm	
SAG mill work index	14.82	kWh/t		Ball mill work index	14.82	kWh/t	
SAG efficiency factor	1.4			Diameter efficiency factor, EF3	0.91		
Transmission loss factor	1.0			Feed size efficiency factor, EF4	1.00		
Unit power consumption	6.59	kWh/t	Avg 2004	Fineness of grind factor EF5	1.00		
Mill power required	14,248	kW	15,253	Low ratio of reduction, EF7	1.03		
Mill power required	19,107	hp		Transmission loss factor	1.00		
SAG mill power installed	24,000	hp	79.6%	Unit power consumption	6.78	kWh/t	Avg 2004
				Mill power required	15,377	kW	15,402
				Mill power required	20,621	hp	
				Mill power installed	24,000	hp	85.9%

Table 12.17: 2010 Kemess South East Zone Ore Grinding Circuit Evaluation

East zone BWI, kWh/t	14.9						
Southeast zone BWI, kWh/t	15.3						
Primary SAG Mill				**Secondary Ball Mill**			
Mill operating parameters and power required				**Mill power required**			
Annual throughput	18,748,465	tpa		Daily feed tonnage	51,366	t/d	
Daily feed tonnage	51,366	t/d		Mill availability	90.8	%	
Mill availability	90.8	%		Mill feed rate	2,357	tph	
Mill feed rate	2,357	tph		Feed size, F80	918	µm	
Feed size, F80	150,000	µm		Discharge size, P80	150	µm	
Product size P80	918	µm		Ball mill work index	15.1	kWh/t	
SAG mill work index	15.1	kWh/t		Diameter efficiency factor, EF3	0.91		
SAG efficiency factor	1.5			Feed size efficiency factor, EF4	1.00		
Transmission loss factor	1.00			Fineness of grind factor, EF5	1.00		
Unit power consumption	6.89	kWh/t		Low ratio of reduction, EF7	1.03		
Mill power required	16,242	kW		Transmission loss factor	1.00		
Mill power required	21,781	hp		Unit power consumption	6.9	kWh/t	
SAG mill power installed	24,000	hp	90.7%	Mill power required	16,256	kW	
				Mill power required	21,800	hp	
				Mill power installed	24,000	hp	90.8%

Table 12.18: Monzonite Ore Grinding Model

Primary SAG Mill				**Secondary Ball Mill**			
Mill operating parameters and power required				**Mill power required**			
Annual feed tonnage	9	Mt/y		Daily feed tonnage	24,558	t/d	
Daily feed tonnage	24,558	t/d		Mill availability	92	%	
Mill availability	92	%		Mill feed rate	1,117	tph	
Mill feed rate	1,117	tph		Feed size, F80	918	µm	
Feed size, F80	150,000	µm		Discharge size, P80	150	µm	
Product size, P80	918	µm		Ball mill work index	14.1	kWh/t	
SAG mill work index	14.1	kWh/t		Diameter efficiency factor, EF3	0.91		
SAG efficiency factor	1.5			Feed size efficiency factor, EF4	1.00		
Transmission loss factor	1.00			Fineness of grind factor EF5	1.00		
Unit power consumption	6.43	kWh/t		Low ratio of reduction, EF7	1.03		
Mill power required	7,186	kW		Transmission loss factor	1.00		
Mill power required	9,535	hp		Unit power consumption	6.44	kWh/t	
SAG mill power installed	12,000	hp	80.3%	Mill power required	7,192	kW	
				Mill power required	9,644	hp	
				Mill power installed	12,000	hp	80.4%

Table 12.19: Takla Ore Grinding Model

Primary SAG Mill				**Secondary Ball Mill**			
Mill operating parameters and power required:				**Mill power required:**			
Annual feed tonnage	9	Mt/y		Daily feed tonnage	24,658	t/d	
Daily feed tonnage	24,658	t/d		Mill availability	92	%	
Mill availability	92	%		Mill feed rate	1,117	tph	
Mill feed rate	1,117	t/h		Feed size, F80	918	µm	
Feed size, F80	15,000	µm		Discharge size, P80	150	µm	
Product size, P80	918	µm		Ball mill work index	15.3	kWh/t	
SAG mill work index	15.3	kWh/t		Diameter efficiency factor, EF3	0.91		
SAG efficiency factor	1.5			Feed size efficiency factor, EF4	1.00		
Transmission loss factor	1.00			Fineness of grind factor, EF5	1.00		
Unit power consumption	5.98	kWh/t		Low ratio of reduction, EF7	1.03		
Mill power required	7,797	kW		Transmission loss factor	1.00		
Mill power required	10,456	hp		Unit power consumption	6.99	kWh/t	
SAG mill power installed	12,000	hp	87.1%	Mill power required	7,804	kW	
				Mill power required	10,465	hp	
				Mill power installed	12,000	hp	87.2%

12.7 Flotation Characteristics

Table 12.20 highlights the differences in the ore and processing characteristics of the Kemess South (hypogene) and the KUG deposits.

The rougher flotation for KUG ore will essentially be a bulk sulphide flotation. The bulk rougher concentrate will be re-ground. In three cleaner flotation steps, pyrite will be depressed at increasing lime pH levels to recover a marketable copper/gold concentrate.

The higher pyrite content of the KUG ore, as compared to the Kemess South hypogene ores, alters conditions in rougher and cleaner flotation.

Table 12.21 compares laboratory operating practices when treating the ores of Kemess South and KUG based on the 2003 Lakefield test program.

Table 12.20: Flotation Criteria

Ore Characteristics	Metal content (% or g/t)			Pyrite Content		Target Fineness of Grind	Hardness BM Work Index	Alkalinity in Cleaners
	Cu	Au	Ag	Pyrite	Activated	P80	kWh/t	pH
KS hypogene	0.22	0.65–1.0	3.0	1%	No	Primary 145 µm Regrind 35–50 µm	13.8–15.0 12.5 in sample tested	pH 9.5
KUG average: range:	0.22 0.16–0.44	0.56 0.18–0.96	1.7 0.1–2.7	4 % 1.5–8%	No	Primary 150 µm Regrind ≤20 µm	±13.8	First pH 10 Second pH 11 Third pH 11.5

Table 12.21: Comparison of flotation test conditions

Ore type	Rougher Flotation			Cleaner Flotation, Laboratory					
				First cleaner flotation			Second/third Cleaner Flotation		
	Time (min)	Collector Quantity	Rgh Conc. kg/t of Feed	Re-grind Minutes Laboratory	Flotation (min)	pH	Flotation (min)	pH	Product Fineness
KS hypogene	11	5 g/t	≈55	6 pebble mill	4	9.5	2	9.5	P80 ≈ 35 µm
KUG	11	15 g/t	≈130	15 SS mill	6	10.5	4.5	11/11.5	≤ 15 µm

Source: Klaus Konigsmann, KVK Consulting 2003

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Plant operations and equipment utilization will require some adjustments for KUG ore, as follows:

- As a result of the higher pyrite in the feed, the volumes of rougher concentrates are expected to double when treating KUG ores. The existing KS cleaner capacity will be sufficient for the KUG ore (i.e. at half the feed rate).
- Regrind mill unit power input will be greater than KS levels because of the finer grinding requirements. Steel consumption and power demand will increase proportionally. At 9 Mt/y and the increased mass pull, the existing regrind mill will not meet the requirements of the KUG project.
- Alkalinity (pH) in cleaner flotation will rise for pyrite depression without impeding gold flotation.
- Sufficient concentrate capacity exists in the existing thickeners and pressure filters.

Gold recoveries to the copper concentrate will be approximately ten percentage points lower when treating KUG ores compared to those obtained for KS hypogene. The pyrite of both deposits carry finely disseminated gold at comparable concentration. The higher pyrite content of the KUG ore will result in a correspondingly higher loss to the cleaner scavenger tailings. Detailed mineralogical analysis (Amtel, 2003) has shown that ultrafine inclusions of gold particles, primarily in pyrite but also in rock minerals, will generate almost 90% of this loss.

Mineralogical investigations indicate that for the KS hypogene ores, the pyrite content was ±1%. For the KUG deposit, the pyrite content is expected to be ±4%.

Table 12.2 summarizes the open cycle flotation results of the 2011 G&T test program (G&T 2011, KM2911). Additional cleaner flotation and locked cycle flotation tests were completed (Table 12.4). The observations and results from these tests generally support the conclusions from the 2003 Lakefield test program.

Table 12.22 provides an estimate of the additional requirements for the rougher flotation concentrate regrind circuit for the KUG ore. The existing regrind mill is a 4,476 kW ball mill with an estimated maximum power draw of 4,028 kW. The mill media charge was changed from balls to millpebs during the last couple of years of KS operation in an effort to reduce losses due to sliming observed when the target regrind size was set finer for the higher pyrite KS east zone ore.

For the KUG ore, testwork has shown that better metallurgical results are achieved if the regrind product P80 size is 20 µm (or finer). With an expected rougher mass pull of 18%, the estimated regrind power required from the existing ball mill regrind would not be sufficient. As such, additional stirred, ultrafine grinding mill capacity will be required to effectively treat KUG ore using the existing KS process plant.

Based on the combination of Levin (ball mill) and signature plot (stirred mill) test results, an additional 4,552 kW of regrind power is required to achieve a P80 size of 20 µm (Table 12.22). This will be provided with two 3 MW units added in open circuit after the existing regrind ball mill.

Table 12.22: Estimate of Rougher Concentrate Regrind Power (P_{80} size of 20 µm)

Description	Estimated Value
Existing Ball Mill Regrind	4,476 kW
Plant feed	24,658 t/d
Rougher concentrate **	4,438 t/d
Plant availability	92 %
Regrind circuit feed rate	201 tph
Regrind circuit, F80	150 µm
Regrind circuit, P80	20 µm
Ball mill available power	4,028 kW
Ball mill specific energy (SE)	20.0 kWh/t
Projected K80 (Table 12.11, Levin Test Results)	33 µm
Stirred mill projected SE (33µm)	12.4 kWh/t
Stirred mill projected SE (20µm)	35.0 kWh/t
Stirred mill SE required	22.6 kWh/t
Stirred mill power requirement	4,552 kW
Stirred mill selection	2 x 3,000 kW

** Based on test results for Year 2–Year 7 blends

12.8 Predicted Results

The predicted metallurgical results are based on a comparison of the locked cycle test results from the 2003 Lakefield program on KN samples and more recent supporting work by G&T on KUG samples (G&T 2011, KM2911). For the PEA completed by AMC Mining Consultants (Canada) Ltd in 2011, it was recommended to base the metallurgical performance on:

- Copper concentrate grade of 22% Cu,
- Copper recovery of 91%, and
- Gold recovery of 72%.

The results of the locked cycle tests completed in 2011 by G&T (KM2911) on composites of individual ore domains (Table 12.4) support these metallurgical parameters. This is further supported by an analysis of the project metallurgy based on the distribution of ore (Table 12.5) which indicates recoveries of 91.3% and 71.9% for copper and gold respectively.

12.9 Concentrate Quality

Historically, the copper concentrates produced from the KS operation were good quality with low penalty elements. This is expected to continue with the KUG ore based on 2011 G&T testwork results (KM2911). Copper concentrate will grade in the range of 22% to 23% Cu, 30 g/t to 50 g/t Au, and 75 g/t to 100 g/t Ag.

All impurities are expected to be below penalty limits; for example: arsenic ≤50 ppm, bismuth ≤80 ppm, calcium ≈0.7%, selenium ≤100 ppm, mercury ≤1 ppm, and lead and zinc at trace levels.

12.10 Tailings Characteristics

The ultimate settling density of KUG tailings was determined for three tailings samples comprising different composites at 66%, 69%, and 72% solids (w/w, by weight). Corresponding pulp densities were 1.737, 1.797, and 1.825 g/cc respectively.

Based on testwork on seven samples (KVK Consulting 2003), the net neutralization potential for KUG tailings will be negative at –50 t CaCO3/1000 t of tailings. Tailings will be stored subaqeously to avoid acidification of the pond.

12.11 Future Testwork

It is recommended that future testwork is conducted to investigate the effect of regrind size on gold recovery and concentrate quality. This is considered necessary to optimize regrind size by ore blend.

13 Mineral Resource Estimates

In 2015–2016 SRK revisited the resource estimates for the Kemess Underground (KUG) deposit, and the Kemess East (KE) deposit. Using updated and audited drillhole databases, the geology models for both deposits were rebuilt, all statistical analyses were updated, grades were re-estimated based on the updated geological domaining interpretations, and the resources were reclassified resulting in two new resource statements for the Kemess property.

The following section documents the procedures implemented for the KUG and KE resource model updates.

13.1 Kemess Topography

A topography Lidar point data file was provided to SRK in DXF format. The Lidar survey was conducted over the dates of June 26–28, 2015. The Lidar data file covers the entire Kemess Underground and Kemess East resource areas of interest. Maptek Vulcan software was used to triangulate the point data, and construct a 3D topography wireframe surface. SRK reviewed the surface in cross section comparing the elevation between drillhole collars and the topography surface. SRK found close agreement, and is of the opinion that the 3D topography model is suitable for use in resource estimation for the Kemess Underground and Kemess East deposits.

13.2 Kemess Coordinate System

All data was provided in a local coordinate system based on a transformation from the original UTM WGS84 / Nad83 Datum. The local coordinate system forms the basis for all resource estimation work.

13.3 Kemess Underground (KUG)

The Mineral Resource estimate presented in this section represents an update to the Kemess Underground Mineral Resource based on the following updates:

- New Structural 3D Model,
- New Lithological 3D Model,
- New Alteration 3D Model,
- New Grade Estimation Domain Interpretation,
- Change in block size, and
- Change in Grade Estimation Parameters.

The Mineral Resource model was prepared by SRK considering 149 drillholes drilled by AuRico Metals and predecessor companies during the period of 1976 to 2011. All onsite geological supervision, and the lithological and structural models were prepared by Wade Barnes, P.Geo of AuRico Metals, a “Qualified Person” as defined in National Instrument NI 43-101, while the

alteration model and grade estimation work was prepared by Chad Yuhasz, P.Geo of SRK Vancouver, a “Qualified Person” as defined in National Instrument NI 43-101.

This section describes the resource estimation methodology and summarizes the key assumptions utilized in the estimation of copper, gold, and silver grades. It is the opinion of SRK, that the resource evaluation and processes reported herein are a reasonable representation of the Mineral Resource for the KUG project, and is suitable as a basis for a feasibility level study. The Mineral Resources have been estimated and classified in conformity with generally accepted Canadian Institute of Mining (CIM) “Estimation of Mineral Resources and Mineral Reserves Best Practices” guidelines (2003), and are reported in accordance with the Canadian Securities Administrators’ National Instrument 43-101 (2014). Mineral Resources are not Mineral Reserves and do not demonstrate economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserves.

The database used to estimate the KUG project Mineral Resource was audited by SRK. SRK is of the opinion that the current drillhole information is sufficiently reliable and accurate to interpret with confidence the boundaries of the copper-gold porphyry mineralization, and the assay data are sufficiently reliable to support the Mineral Resource estimation.

Leapfrog Geo 3.0 software was used to construct the 3D geological solids, Snowden Supervisor 8.4 software was used to explore the statistical and geostatistical controls, Maptek Vulcan 9.1.5 software was used to construct the block model, prepare the assay data for estimation, complete the grade estimation, and tabulate the Mineral Resources.

13.3.1 Resource Database

Exploration work conducted in the early to mid-1970’s identified a mineralized porphyry target, but it was not until deep drilling in 2001 that significant copper and gold grades were located. Since 2001, drilling concepts were adjusted to focus on the higher grade underground potential mineralization, while moving away from lower grade near-surface open pit mineralization. Table 13.1 highlights the Kemess North/Underground drill programs by year.

The 2016 resource estimate is based on all drilling completed within the KUG deposit, comprising 33,057 copper assays, 33,043 gold assays, and 18,744 silver assays in 149 holes. Silver was not routinely assayed prior to 2003, when it was decided to include silver in the economics of the KUG deposit. Any existing core or reject material from pre-2003 drilling located within the resource area of interest was analyzed for silver in 2010. Some gaps in silver assays exist due to loss of core or sample material that was consumed in previous metallurgical sampling programs. Summary statistics for the KUG database are presented in Table 13.2.

Table 13.1 Kemess North/Underground Drillhole Campaign

Year	No. of Holes	Total Metres
1976	2	507.8
1989	5	731.8
1990	7	1,185.0
1991	4	1,111.9
1992	9	2,034.8
2000	7	3,345.2
2001	16	8,385.5
2002	33	18,694.2
2003	17	6,414.1
2004	8	3,027.5
2005	3	2,074.3
2006	6	3,251.3
2010	29	16,394.1
2011	3	2,206.8
Total	**149**	**69,364.3**

Table 13.2: Kemess North/Underground summary statistics for all drillhole data

Category	No. of Holes	No. of Records	Total Length (m)	Min Length (m)	Max Length (m)	Av Length (m)
Collar Table	149	-	69,364.3	16.75	1,206.4	-
Cu Assays	-	33,057	64,249.8	0.08	45.72	1.94
Au Assays	-	33,043	64,224.4	0.08	45.72	1.94
Ag Assays	-	18,744	36,937.8	0.12	22.9	1.97
Alteration	-	-	69,364.1	-	-	-
Lithology	-	-	69,364.1	-	-	-
Density	38	9,886	19,529.2	-	-	-

AuRico Metals implemented the acQuire database management system for the Kemess property in 2015. This process resulted in a comprehensive rebuild of the KUG database, as data was transferred into the software. As part of the rebuild, all assays were imported into acQuire directly from the digital lab assay certificate, eliminating any data transcription errors. As reported in the QA/QC section of this report, all appropriate validation and checks were performed.

AuRico Metals designed an acQuire to MS Access export script within the acQuire software, and provided the database to SRK in MS Access format on 5 August 2015. This database was loaded in Maptek Vulcan software and a standard database validation routine was performed to check for the following errors:

- Overlapping intervals,
- Intervals deeper than collar table hole length,

- Out of sequence intervals,
- Missing data, and
- Erroneous downhole deviation.

No material errors were reported.

13.3.2 Solid Body Modelling

The KUG deposit is classified as a porphyry style copper-gold-silver deposit. Previous models of the deposit focused on a combination geology / grade compartmentalization to control the mineralization. The 2016 model update re-interpreted the geology, and utilized experimental data analyses (EDA) of the data to characterize the copper, gold, and silver mineralization as a relationship to the deposit porphyry style lithology and alteration. The data was re-interpreted to fit a porphyry style model, and 3D lithology and alteration models were generated to be used to define the mineralization, and form grade estimation domains.

Structural Model

AuRico Metals geologists utilized Leapfrog Geo to model the KUG fault structures, as per the structural understanding of the deposit. In general the mineralization within the KUG deposit is well-defined and constrained within a known network of bounding faults. The KUG mineralization is cut-off in the north by a North-Bounding normal fault; striking east-west and dipping to the north. At depth the KUG mineralization is cut-off by the Kemess North Thrust reverse fault; striking east-west and dipping to the south. To the east of the KUG mineralization lies the East-Bounding normal fault; striking north-south and steeply dipping to the east. Further to the south there is an east-west striking, steeply dipping normal fault that lies just outside of the deposit area.

These faults were modelled in 3D, and used to create fault domains in Leapfrog, which were then utilized during construction of the 3D lithological and alteration models. The fault domaining process ensures that only the drillhole data that exists within each fault block is used in the geological modelling of the solids. For the lithology, the fault domaining was used to generate offset of the lithological units at the fault surfaces. For the alteration, the fault domaining was used to limit the extrapolation of the solids.

This process ensures that the geological interpretations match the structural model of the deposit, while making sure that the final block model doesn't extrapolate mineralization beyond the structures that limit the mineralized system.

Figure 13.1 shows the KUG structural model, as interpreted by AuRico Metals geologists.

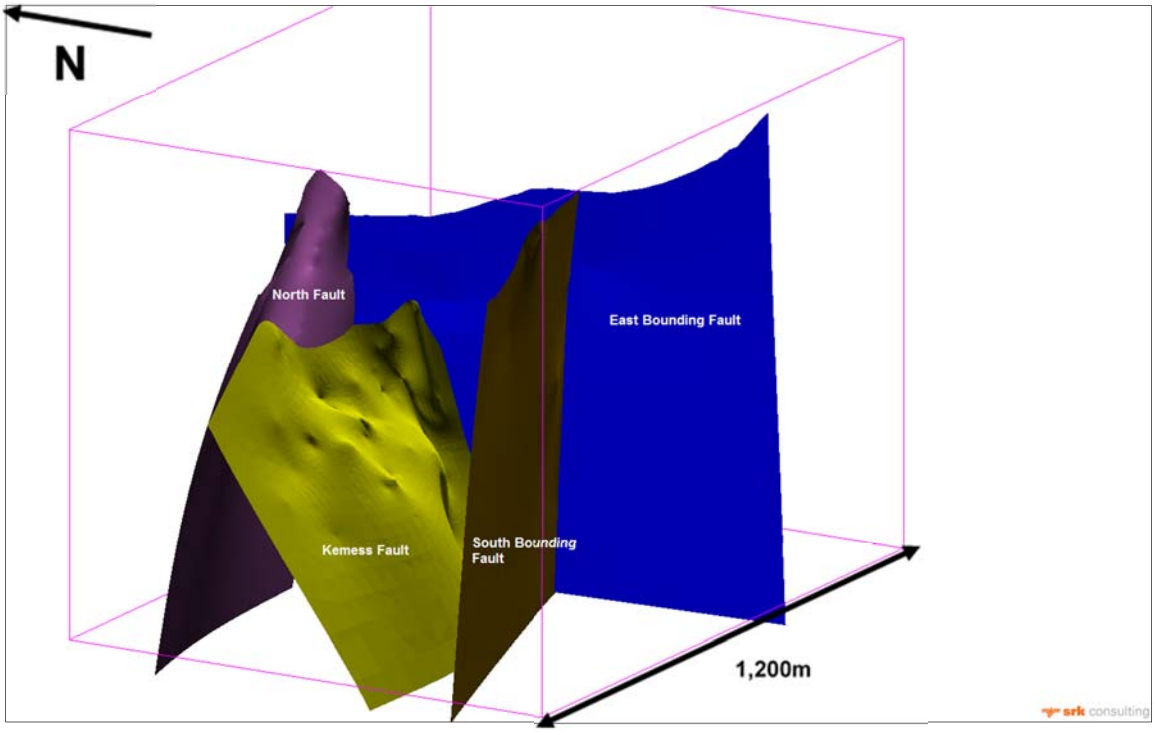


Figure 13.1: Kemess Underground Structural Model
(Source: SRK, 2016)

Lithology Model

AuRico Metals utilized Leapfrog Geo to model the KUG deposit lithology, as a function of the fault domaining highlighted in the structural model sub-section. Four primary lithological units have been identified within the KUG deposit AOI.

- Black Lake Intrusive (BKu) (Integer Code 10)
- Takla Volcanics (Takla) (Integer Code 11)
- Hazelton (Haz) (Integer Code 12)
- Post Mineralization Dykes (PDykes) (Integer Code 13)

Using Leapfrog Geo software, the lithological data was reviewed, and 3D fault domained lithological solids were constructed. While Leapfrog software was used to construct the initial lithological wireframes, the final interpretation was the result of detailed adjustment by AuRico geology staff. Figure 13.2 shows a 3D view of the final lithological wireframes.

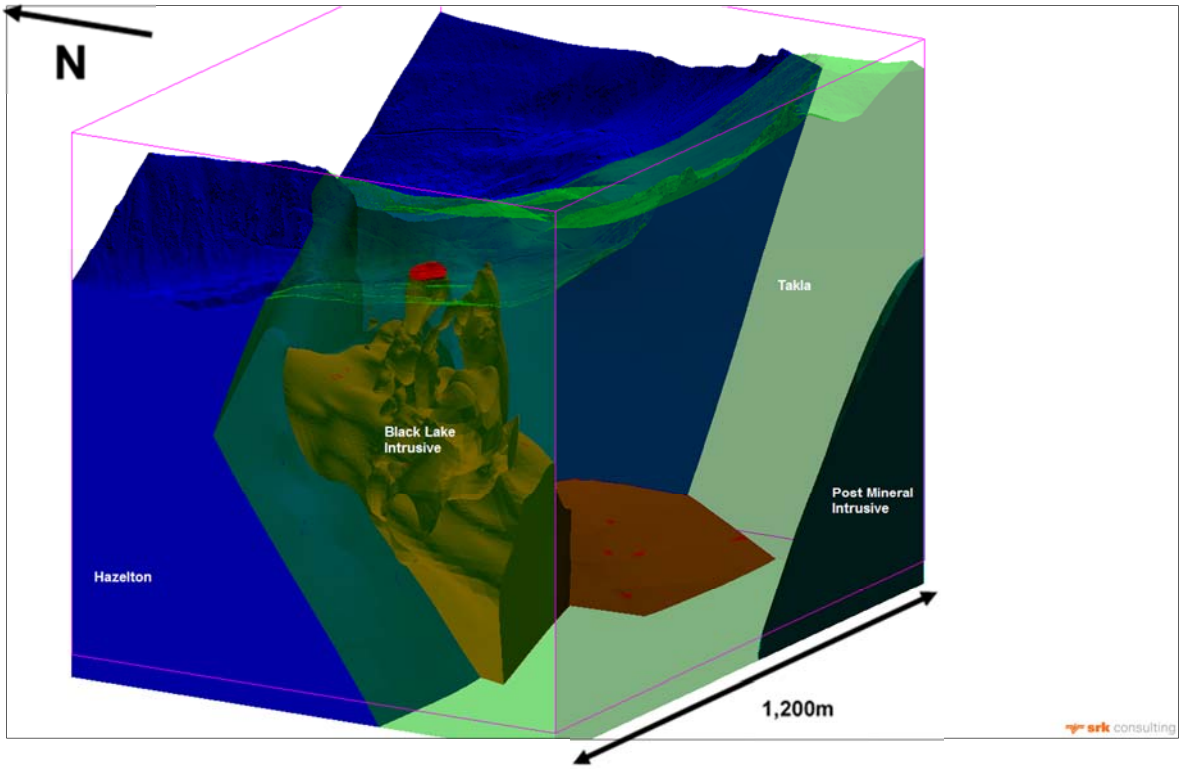


Figure 13.2: Kemess Underground Lithological Model
(Source: SRK, 2016)

Alteration Model

SRK geologists utilized Leapfrog Geo to model the KUG deposit alteration, as a function of the fault domaining highlighted in the structural model sub-section. Three primary alteration units have been identified within the KUG deposit AOI.

- Propylitic (Prop) (Integer Code 22)
- Phyllic (Phy) (Integer Codes 20 and 21)
- Potassic (Pot) (Integer Codes 23 and 24)

The phyllic zone was further subdivided into an upper sulphate-leach zone, and a lower phyllic-gypsum zone. This subdivision of the phyllic zone was deemed necessary based on general physical rock qualities, as well as a definable difference in copper and gold grade populations. The leach zone is in general mineralized at levels below economic cut-offs, due to the secondary alteration processes removing metal. The leach unit is also distinctly characterized as a broken rock and rubble zone, consistently represented by poor core recovery and RQD results. The phyllic-gypsum zone is more competent and contains a moderate to high degree of gypsum veining.

The potassic zone was subdivided into two domains as a function of the visual potassic alteration intensity. The weak potassic zone is characterized by weak quartz veining and chlorite-sericite-quartz alteration assemblages. The strong potassic zone is characterized by intense quartz veining

and stockworks, and strong chlorite-quartz-magnetite-biotite alteration assemblages. All experimental data analyses (EDA) analyses indicated distinct copper, gold, and silver populations as a function of potassic alteration intensity, with elevated grades distinctly correlated to increase in alteration intensity.

Propylitic alteration is only seen across fault boundaries in the surrounding younger Tooddogone Formation (Hazelton).

Based on this analysis, five alteration domains were modeled. The resultant wireframes were reviewed on section to check for a geological match to the drilling, as well as reviewed to ensure the wireframes were geologically plausible. Manual edits, and the Leapfrog interval selection tool were used in some cases to generate a more acceptable wireframe. The final alteration model is shown in Figure 13.3.

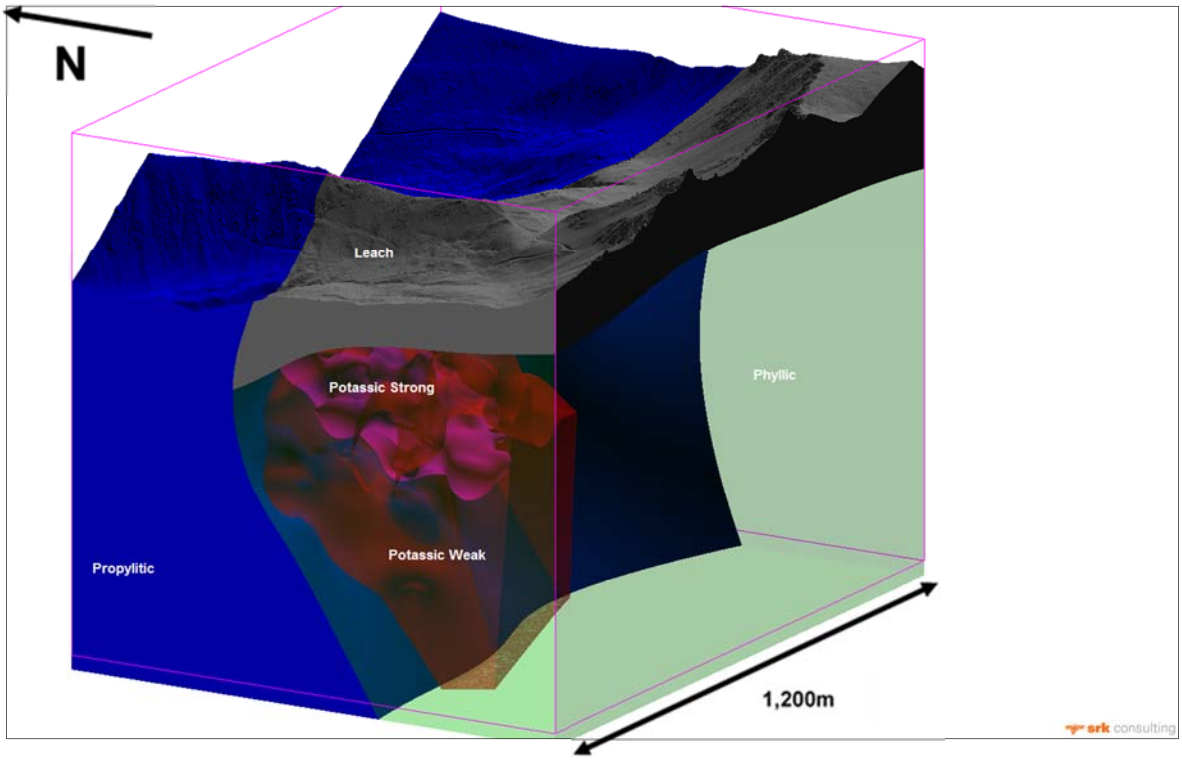


Figure 13.3 Kemess Underground Leapfrog Alteration Model
(Source: SRK, 2016)

13.3.3 Compositing

Logging protocols utilized by AuRico geologists in general applied a standard 2 m assay length for NQ core and 1.5 m assay length for HQ core, but do allow for an increase or decrease of sample length to allow for preservation of geological contacts. A standard assay length in porphyry copper systems is considered to be standard practice, as porphyry deposits are generally characterized by large volumes of continuous mineralization of large distances.

A statistical review of the KUG assay database identified variable assay lengths, with mean sample interval lengths of approximately 2.0 m. Vulcan software was used to generate downhole composites of 6.0 m lengths with a tolerance of 3.0 m merge, by estimation domain. This option allowed any end-of-hole, or end-of-domain sample less than 3.0 m in length to be added to the previous interval. Any missing intervals were treated as missing during the compositing. The total length of available raw assay data for copper, gold, and silver was stored for each composite interval. Any 6.0 m composite interval represented by less than 3.0 m of assay data, was excluded during the estimation routines. For the KUG deposit, this resulted in the removal of 221 (from 10,467) copper composites, 222 (from 10,461) gold composites, and 303 (from 6,281) silver composites from the final estimation database. This filter removes statistically underrepresented composites from use in the estimation of grades.

The compositing routine effectively grouped three raw assay intervals, and required a minimum of 50% of the composite interval to be represented by raw data. The final composites were statistically compared to the raw assays, to ensure no bias was introduced from the compositing.

Specific gravity was composited into 3.0 m standard lengths, to preserve more sample variability

13.3.4 Statistical Analysis and Estimation Domains

A full suite of experimental data analyses (EDA) was conducted for each of the copper, gold, and silver variables. The grade populations were evaluated by lithology, alteration, and combination domains. Snowden Supervisor software, and Vulcan software were utilized to analyze the statistics of each variable, as well as visual the data trends in 3D.

Grade populations were found to be correlated and defined by the alteration domains. Each alteration domain supported unique sample populations, and unique mineralization trends. As a result the five alteration domains were further evaluated independently for outliers, boundary analyses, and variography.

Kemess Underground Copper

Copper EDA indicated a strong correlation with the modeled alteration domains, as can be quite common in porphyry systems. The potassic zone contains the bulk of the potentially economic mineralization, with a marked change in mean values between the weak and strong potassic alteration zones.

In general, the copper populations were found to be quite stable, with low variability; indicating the alteration domaining was doing a good job of supporting population stationarity. Figure 13.4 and Figure 13.5 show the copper log histograms and log probability plots by alteration domains.

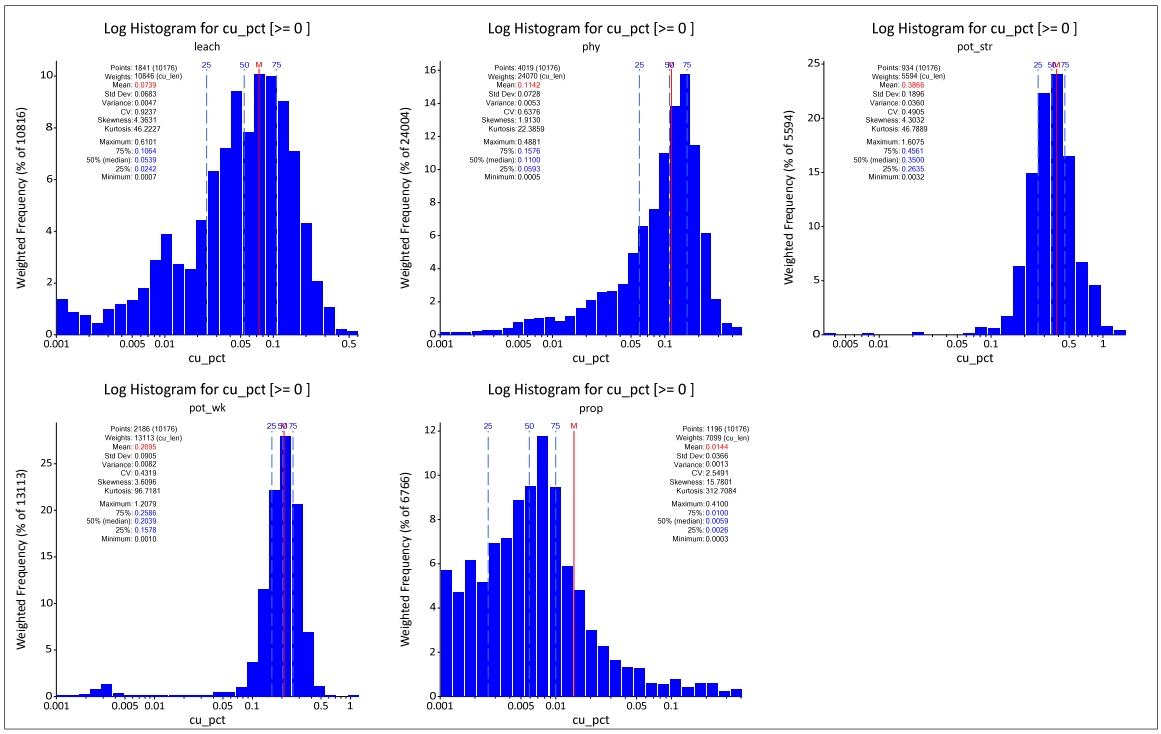


Figure 13.4: Kemess Underground Log Histograms - Copper (6 m composites)

(Source: SRK, 2016)

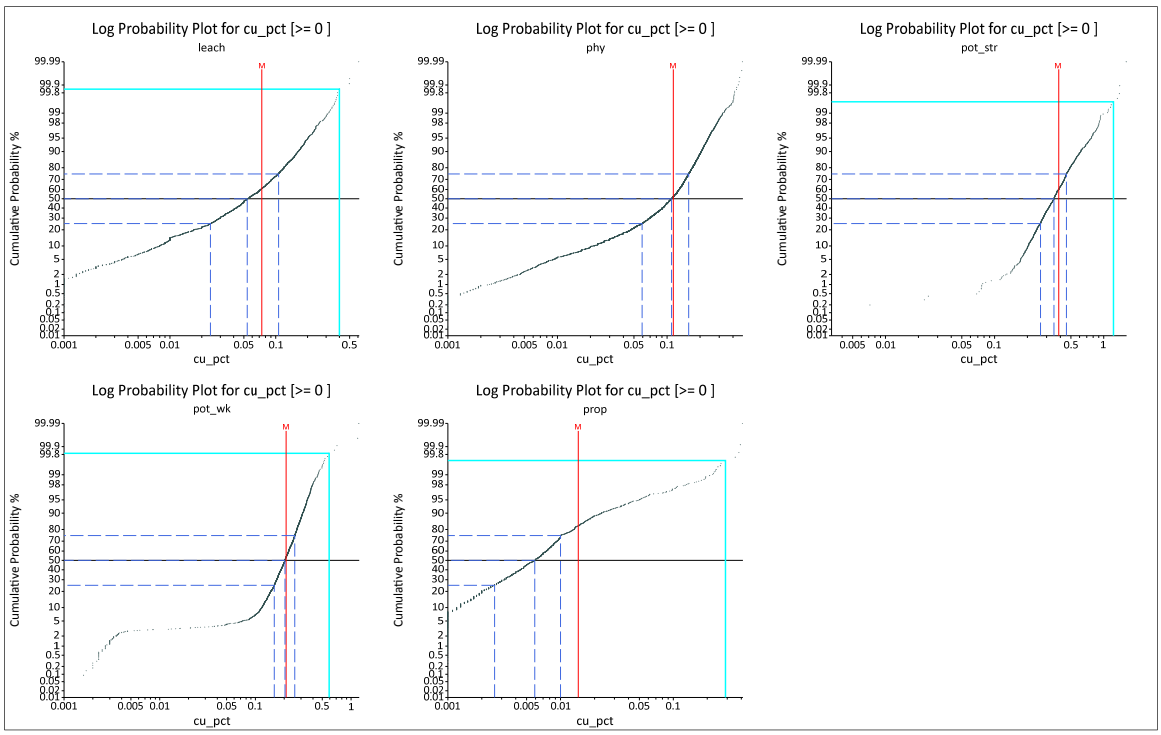


Figure 13.5: Kemess Underground Log Probability Plots - Copper (6 m composites)

(Source: SRK, 2016)

A full set of contact analyses plots were run for all contact possibilities, to identify the correct sample sharing to be utilized during grade estimation. Figure 13.6 is an example of how the copper grade population behaves at the potassic weak / potassic strong alteration domain contact. Due to the step in copper mean values at short distances from the contact, the final estimation routine treats this contact as a hard boundary. Thus no sample sharing is allowed across this contact.

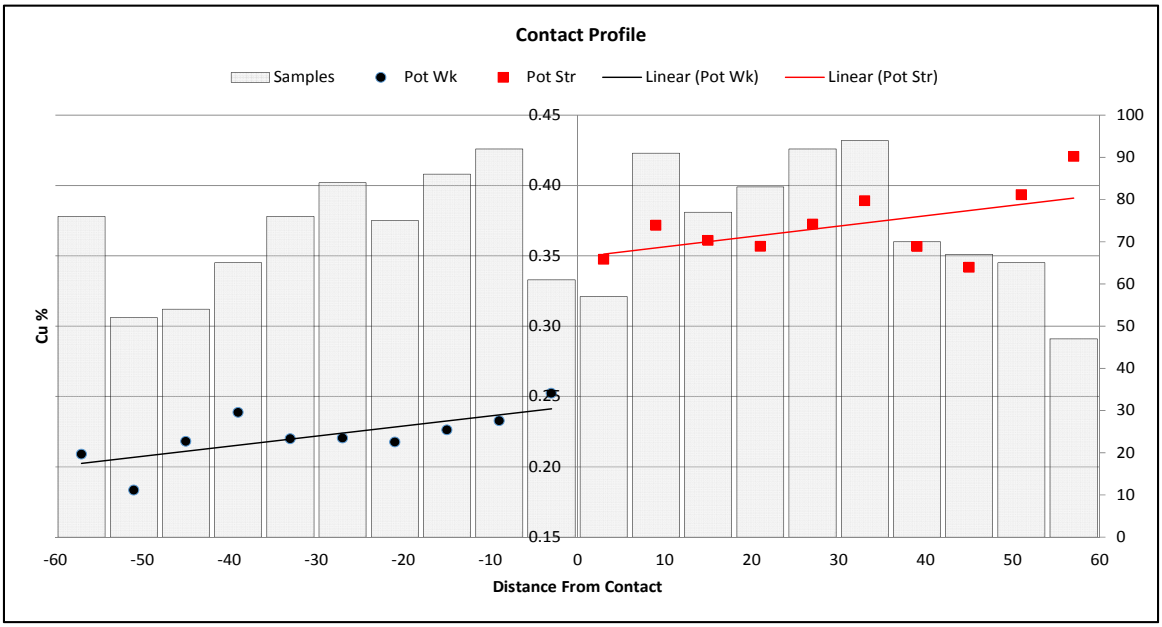


Figure 13.6: Kemess Underground Potassic Weak vs Potassic Strong Contact Analysis - Copper (6 m composites)

(Source: SRK, 2016)

Table 13.3Table 13.3 indicates the full matrix of contact analysis used in the copper resource estimation.

Table 13.3: Kemess Underground Contact Analysis – Copper

Copper	Leach	Phy	Prop	Pot Wk	Pot Str
Leach		Soft	Hard	Hard	Hard
Phy	Soft		Hard	Hard	Hard
Prop	Hard	Hard		Hard	Hard
Pot Wk	Hard	Hard	Hard		Hard
Pot Str	Hard	Hard	Hard	Hard	

Kemess Underground Gold

Gold EDA indicated a strong correlation with the modeled alteration domain, as can be quite common in porphyry systems. The potassic zone contains the bulk of the potentially economic mineralization, with a marked change in mean values between the weak and strong potassic alteration zones.

In general the gold populations were found to be quite stable, with low variability; indicating the alteration domaining was doing a good job of supporting population stationarity. Figure 13.7 and Figure 13.8 show the gold log histograms and log probability plots by alteration domains.

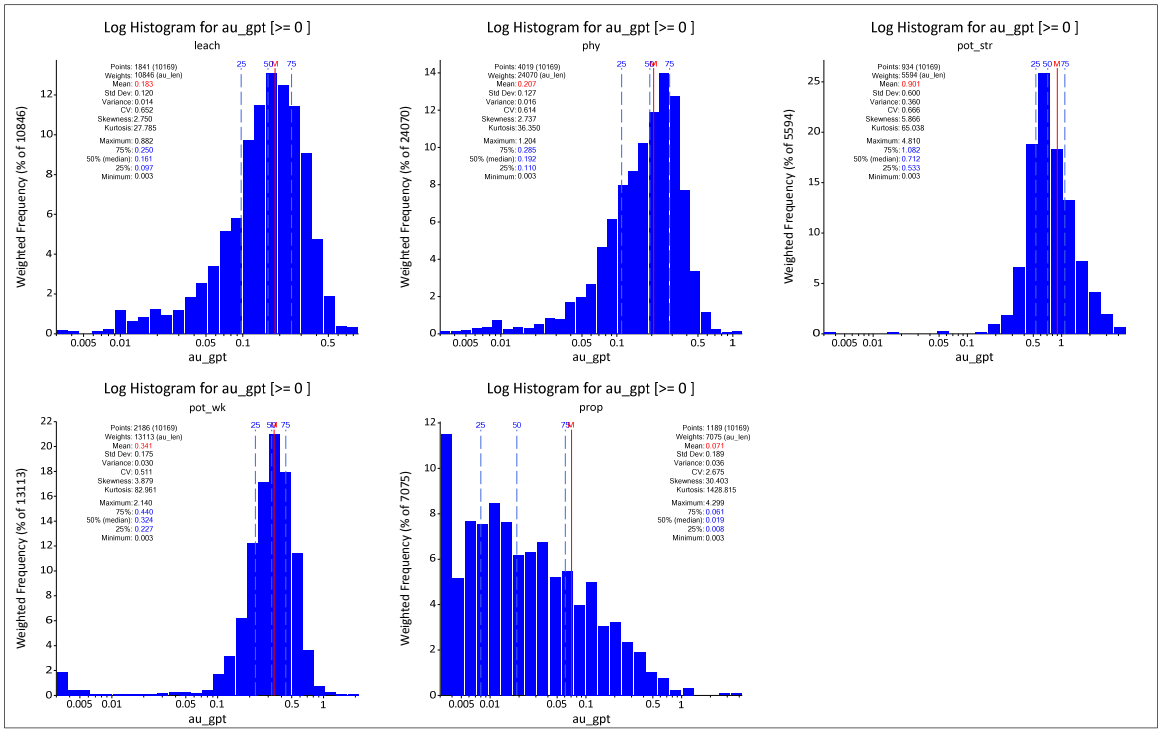


Figure 13.7: Kemess Underground Log Histograms - Gold (6 m composites)

(Source: SRK, 2016)

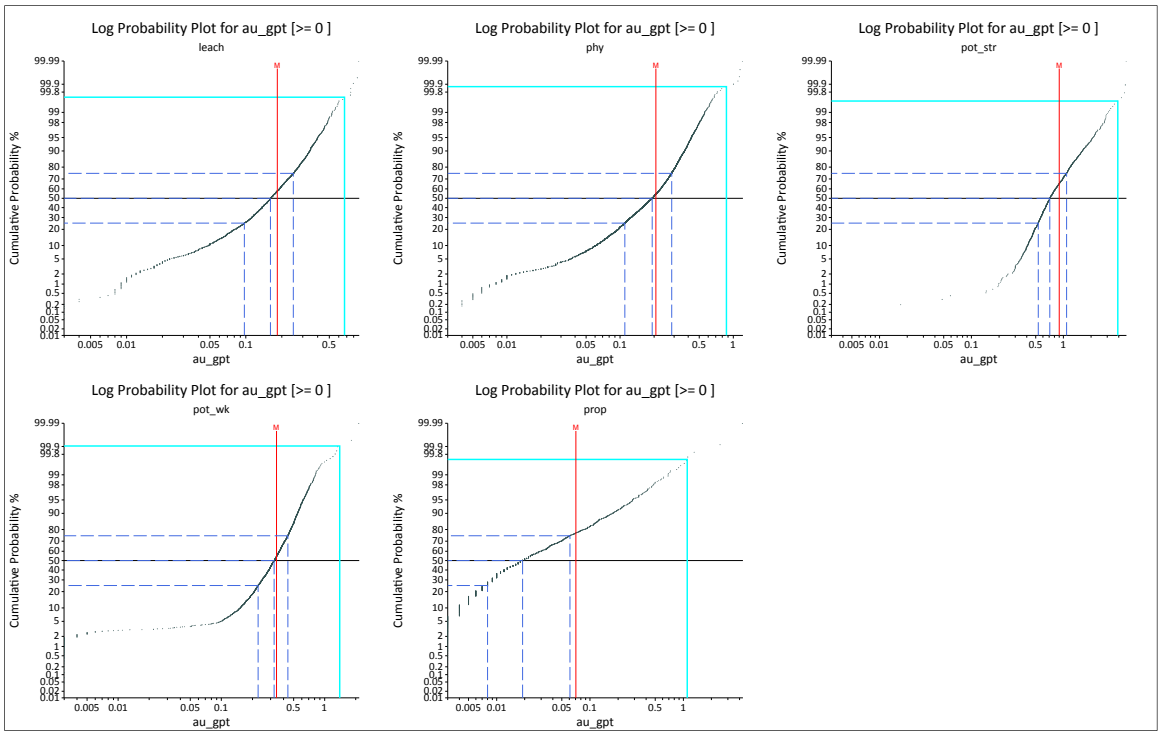


Figure 13.8: Kemess Underground Log Probability Plots - Gold (6 m composites)

(Source: SRK, 2016)

A full set of contact analyses plots were run for all contact possibilities, to identify the correct sample sharing to be utilized during grade estimation. Figure 13.9 is an example of how the gold grade population behaves at the potassic weak / potassic strong alteration domain contact. Due to the step in gold mean values at short distances from the contact, the final estimation routine treats this contact as a hard boundary. Thus no sample sharing is allowed across this contact.

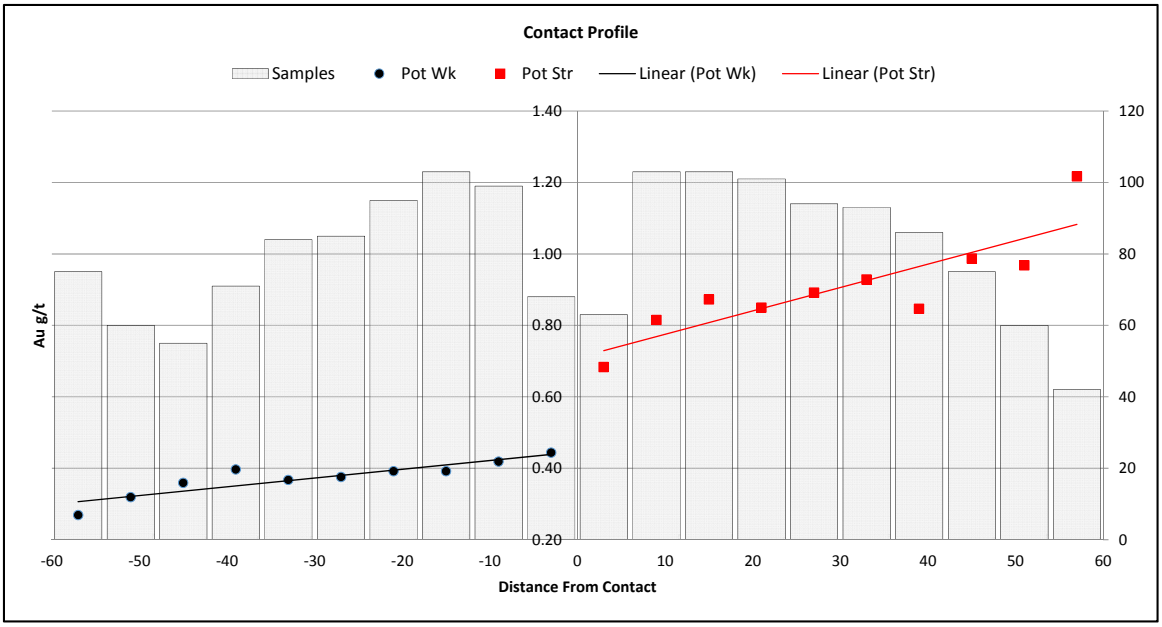


Figure 13.9: Kemess Underground Potassic Weak vs Potassic Strong Contact Analysis - Gold (6 m composites)

(Source: SRK, 2016)

Table 13.4 indicates the full matrix of contact analysis used in the gold resource estimation.

Table 13.4: Kemess Underground Contact Analysis – Gold

Gold	Leach	Phy	Prop	Pot Wk	Pot Str
Leach		Soft	Hard	Hard	Hard
Phy	Soft		Hard	Hard	Hard
Prop	Hard	Hard		Hard	Hard
Pot Wk	Hard	Hard	Hard		Hard
Pot Str	Hard	Hard	Hard	Hard	

Kemess Underground Silver

Silver EDA indicated a strong correlation with the modeled alteration domain, as can be quite common in porphyry systems. The potassic zone contains the bulk of the potentially economic mineralization, with a marked change in mean values between the weak and strong potassic alteration zones.

In general the silver populations were found to be quite stable, with low variability; indicating the alteration domaining was doing a good job of supporting population stationarity. Figure 13.10 and Figure 13.11 show the silver log histograms and log probability plots by alteration domains.

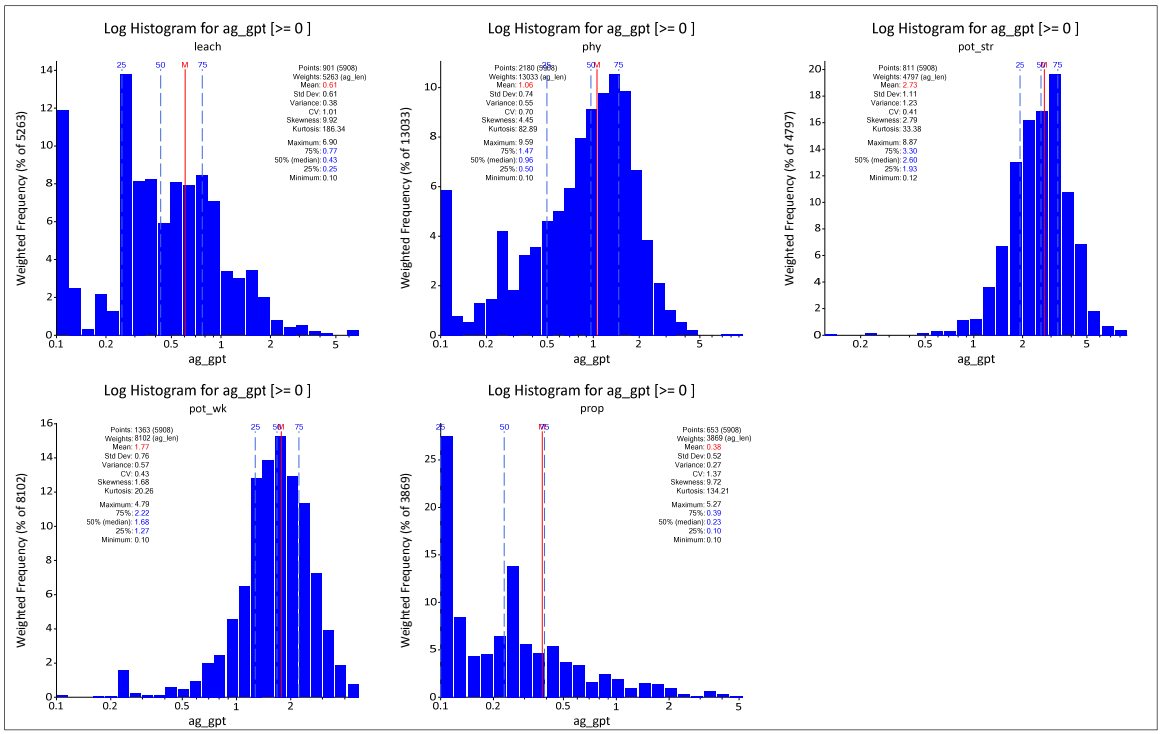


Figure 13.10: Kemess Underground Log Histograms - Silver (6 m composites)

(Source: SRK, 2016)

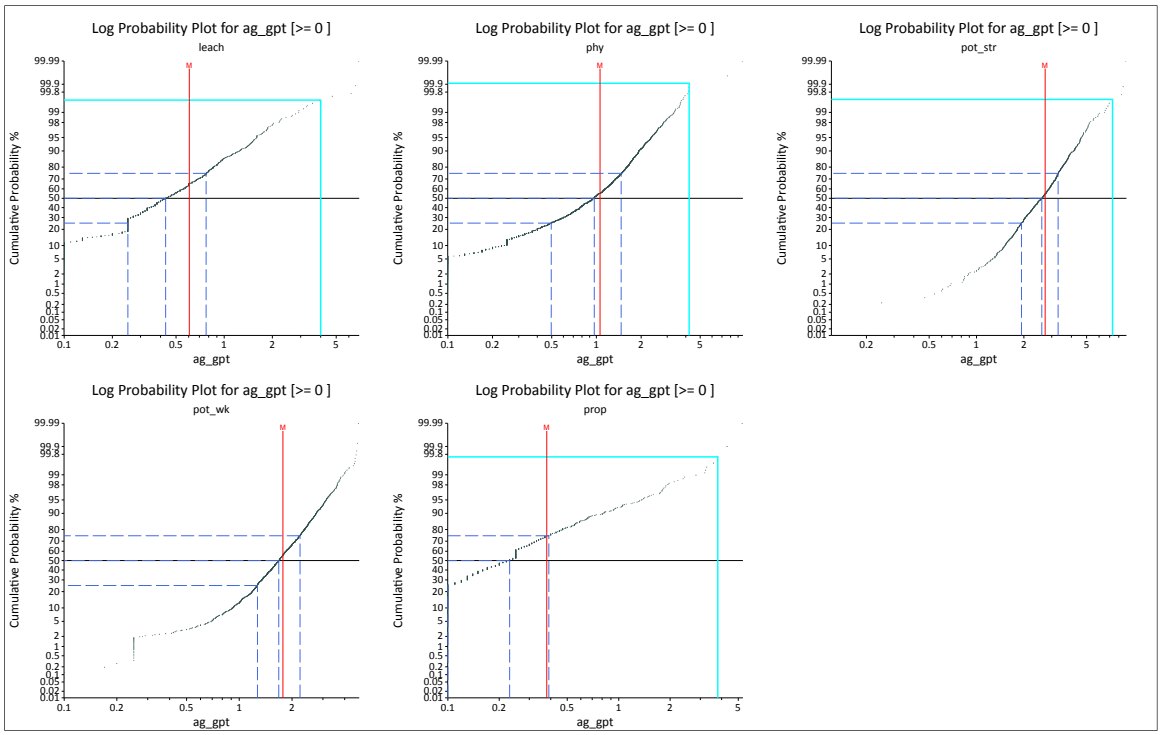


Figure 13.11: Kemess Underground Log Probability Plots - Silver (6 m composites)

(Source: SRK, 2016)

A full set of contact analyses plots were run for all contact possibilities, to identify the correct sample sharing to be utilized during grade estimation. Figure 13.12 is an example of how the silver grade population behaves at the potassic weak / potassic strong alteration domain contact. Due to the step in silver mean values at short distances from the contact, the final estimation routine treats this contact as a hard boundary. Thus no sample sharing is allowed across this contact.

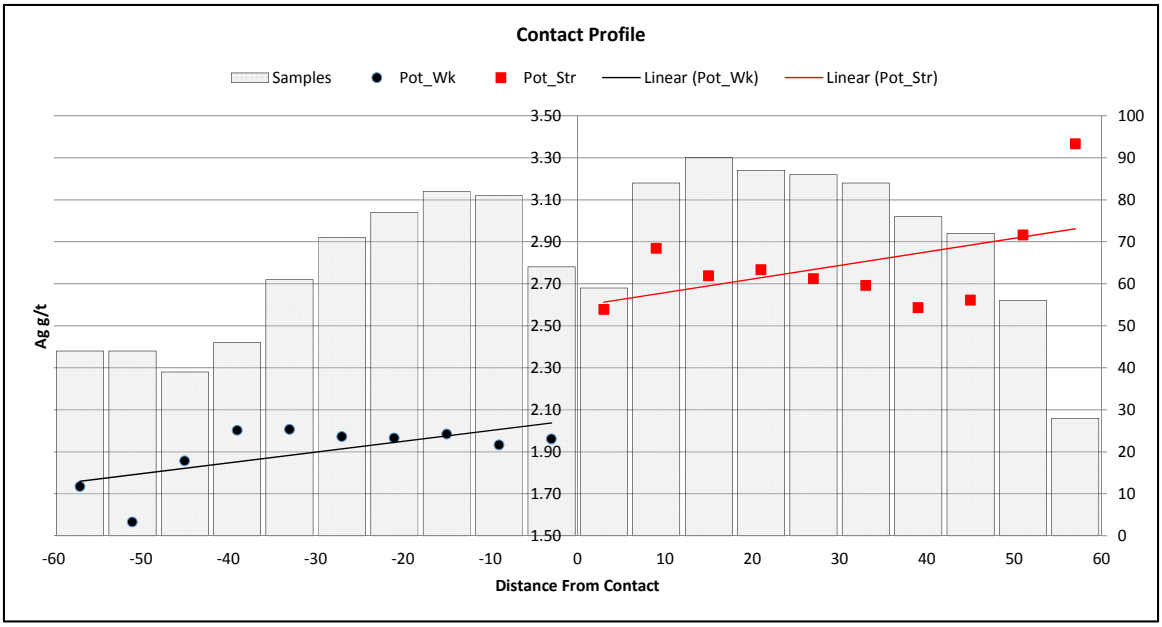


Figure 13.12: Kemess Underground Potassic Weak vs Potassic Strong Contact Analysis - Silver (6 m composites)

(Source: SRK, 2016)

Table 13.5 indicates the full matrix of contact analysis used in the silver resource estimation.

Table 13.5: Kemess Underground Contact Analysis – Silver

Silver	Leach	Phy	Prop	Pot Wk	Pot Str
Leach		Soft	Hard	Soft	Hard
Phy	Soft		Hard	Soft	Hard
Prop	Hard	Hard		Hard	Hard
Pot Wk	Soft	Soft	Hard		Hard
Pot Str	Hard	Hard	Hard	Hard	

Kemess Undergournd Specific Gravity

The KUG resource database contained 9,886 specific gravity composites, from 38 holes, within the KUG resource area of interest (AOI). These values were utilized to perform a block estimate of specific gravity for the KUG resource model. The descriptive statistics for the data are presented in Table 13.6.

Table 13.6: Kemess Underground descriptive statistics of SG data - (3m Composites)

Domain	# Samples	Min Density (g/cm^3)	Max Density (g/cm^3)	Mean Density	Cv
Black Lake Intrusive	2,552	2.38	3.48	2.77	0.04
Hazelton	1,110	2.35	3.11	2.69	0.03
Post Mineral Dykes	213	2.40	3.02	2.65	0.03
Takla High Grade	2,292	2.54	3.53	2.88	0.03
Takla Low Grade	3,719	2.03	3.73	2.69	0.05

Statistical EDA was performed on the 3.0 m specific gravity composites. Populations were found to be more strongly correlated to the lithology domains, as opposed to the alteration domains.

The statistical review highlighted a bimodal data population within the Takla formation, with a threshold of 2.83 g/cm3. The data was spatially reviewed in 3D and it was noted that the samples with specific gravity measurements in excess of 2.83 g/cm3 are spatially correlated, further supporting the decision to subdivide the Takla formation into a low density and high density domain for estimation purposes. The bimodal population within the Takla formation was further reviewed to determine if there was a correlation to the overlapping alterations. The results of this review indicated the bimodal distribution was not a function of alteration.

Snowden Supervisor software was utilized to run indicator variography on the Takla density composites at a 2.83 g/cm3 threshold. The Supervisor variogram spheroidal model option was used, to achieve compatibility with Leapfrog software. Table 13.7 presents the input parameters that were used in Leapfrog to generate a 3D indicator wireframe. This wireframe was exported to Vulcan software, clipped to the Takla lithology solid, and subsequently used to subdivide the Takla lithology into the low and high density domains for specific gravity estimation.

Table 13.7: Kemess Underground Leapfrog Model Parameters - For Takla High SG Indicator

Silver	Trend Ratio	Nugget Effect	Rotation		Spheroidal Model 1	
					Sill	Range
		Gamma (h)	(Leapfrog ZXZ)		Gamma (h)	(m)
Takla	1:1:0.5 AM	0.35	Z	240	0.65	175
			X	20		75
			Z	140		100

The log probability plots in Figure 13.13 show the specific gravity bimodal population within the Takla lithology.

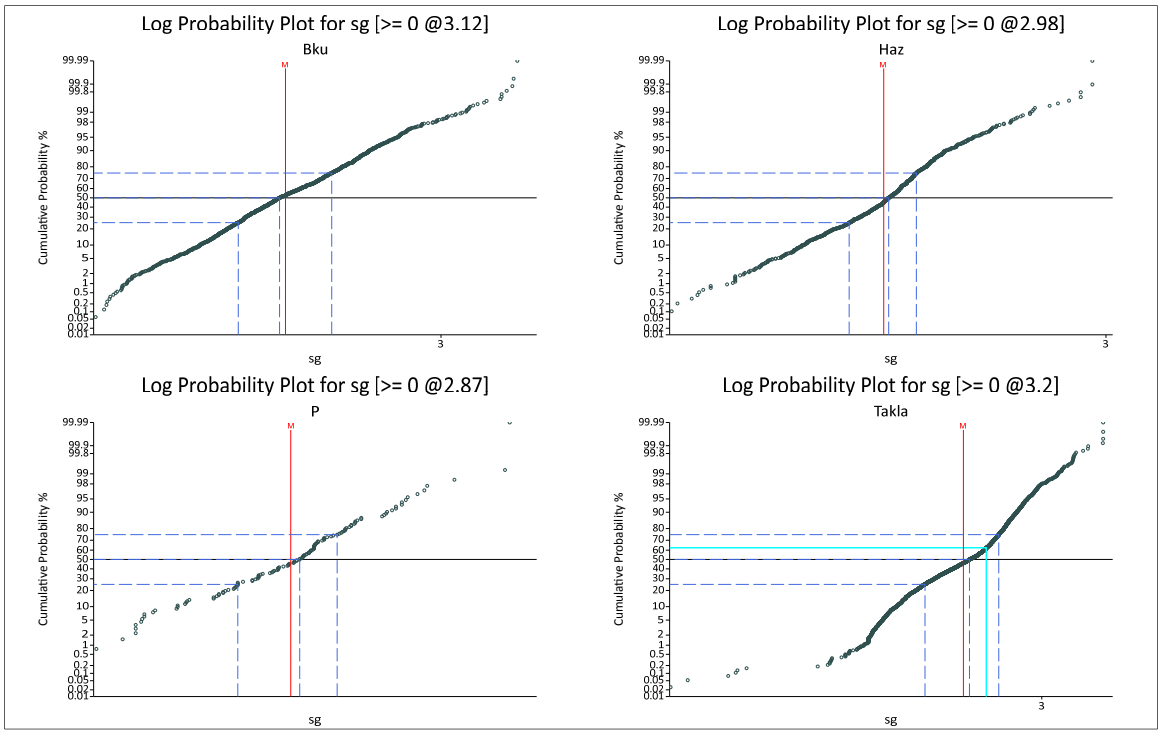


Figure 13.13: Kemess Underground Specific Gravity Log Probability Plots (3m Composites)

(Source: SRK, 2016)

13.3.5 Evaluation of Outliers

Post-compositing and post estimation domain generation, the database was evaluated for potential sample population outliers. Histograms and log probability plots were utilized to define breaks in the sample populations. The outlier evaluation was completed by estimation domain.

As the statistics indicated low variability sample populations, limited outliers were identified. Grade capping was applied to the outliers, and any value that exceeded the grade cap was systematically reduced to the grade cap value. A script was constructed in Vulcan software to populate a capped variable, and this variable was subsequently utilized in grade estimation.

Table 13.8, Table 13.9, and Table 13.10 identify the composite capping descriptive statistics for copper, gold and silver respectively.

Table 13.8: Kemess Underground Capping Statistics - Copper (6 m composites)

Copper (%)	Leach	Prop	Phy	Pot Wk	Pot Str
Pre Cap Mean	0.074	0.014	0.114	0.210	0.387
Pre Cap CV	0.924	2.549	0.638	0.432	0.491
Cap Value	0.400	0.290	N/A	0.590	1.230
Number Capped	4	4	N/A	4	4
Post Cap Mean	0.074	0.014	N/A	0.209	0.386
Post Cap CV	0.910	2.397	N/A	0.411	0.480

Table 13.9: Kemess Underground Capping Statistics - Gold (6 m composites)

Gold (g/t)	Leach	Prop	Phy	Pot Wk	Pot Str
Pre Cap Mean	0.183	0.071	0.207	0.341	0.901
Pre Cap CV	0.652	2.675	0.614	0.511	0.666
Cap Value	0.670	1.100	0.870	1.400	3.900
Number Capped	6	4	5	2	4
Post Cap Mean	0.183	0.067	0.207	0.341	0.898
Post Cap CV	0.643	1.967	0.605	0.498	0.649

Table 13.10: Kemess Underground Capping Statistics - Silver (6 m composites)

Silver (g/t)	Leach	Prop	Phy	Pot Wk	Pot Str
Pre Cap Mean	0.610	0.380	1.060	1.770	2.730
Pre Cap CV	1.010	1.370	0.700	0.430	0.410
Cap Value	4.000	3.800	4.210	N/A	7.300
Number Capped	4	2	2	N/A	3
Post Cap Mean	0.600	0.380	1.060	N/A	2.720
Post Cap CV	0.920	1.330	0.670	N/A	0.400

13.3.6 Geostatistical Analysis and Variography

Snowden Supervisor software was utilized to run experimental variography for each variable within each of the five estimation domains. Once a direction of maximum continuity was defined, the directional variograms were fit with variogram models. Normal Score Transformation was utilized to best fit the experimental variograms with spherical models, and the models were then back transformed.

Upon completion of the variogram model fitting, the models were exported directly from Supervisor software in Vulcan format and the models were subsequently imported directly into the Vulcan estimation parameter files.

The variogram models utilized for estimation are reported in Table 13.11, Table 13.12, and Table 13.13 for copper, gold, and silver respectively.

Kemess Underground Copper

The Variogram models used in the Ordinary Kriging estimation of copper are listed below in Table 13.11

Table 13.11: Kemess Underground Variogram Models – Copper

Domain	Nugget Effect	Direction		Spherical Model 1		Spherical Model 2	
				Sill	Range	Sill	Range
	Gamma (h)	(Vulcan BPD Angles)		Gamma (h)	(m)	Gamma (h)	(m)
Leach	0.06	Bearing Plunge Dip	310 0 0	0.26	53 23 62	0.68	498 282 163
Propylitic	0.39	Bearing Plunge Dip	275.3 -1.7 19.9	0.37	56 64 32	0.24	200 90 155
Phyllic	0.09	Bearing Plunge Dip	355 -1.7 -19.9	0.28	76 77 49	0.63	482 581 227
Pot Wk	0.23	Bearing Plunge Dip	330 0 20	0.32	73 42 16	0.45	245 183 147
Pot Str	0.14	Bearing Plunge Dip	306 3 160	0.36	48 24 28	0.5	142 149 75

Kemess Underground Gold

The Variogram models used in the Ordinary Kriging estimation of gold are listed below in Table 13.12.

Table 13.12: Kemess Underground Variogram Models – Gold

Domain	Nugget Effect	Direction		Spherical Model 1		Spherical Model 2	
				Sill	Range	Sill	Range
	Gamma (h)	(Vulcan BPD Angles)		Gamma (h)	(m)	Gamma (h)	(m)
Leach	0.08	Bearing Plunge Dip	354 -3.2 140	0.4	79 27 22	0.52	350 649 166
Propylitic	0.25	Bearing Plunge Dip	0 0 0	0.47	26 26 17	0.29	161 161 124
Phyllic	0.13	Bearing Plunge Dip	341 3.2 140	0.25	46 87 35	0.62	455 663 126

Domain	Nugget Effect	Direction		Spherical Model 1		Spherical Model 2	
				Sill	Range	Sill	Range
	Gamma (h)	(Vulcan BPD Angles)		Gamma (h)	(m)	Gamma (h)	(m)
Pot Wk	0.16	Bearing Plunge Dip	345 0 170	0.37	114 120 27	0.47	206 389 195
Pot Str	0.17	Bearing Plunge Dip	307 12 158	0.45	59 58 47	0.38	140 150 70

Kemess North Silver

The Variogram models used in the Ordinary Kriging estimation of silver are listed below in Table 13.13.

Table 13.13: Kemess Underground Variogram Models – Silver

Domain	Nugget Effect	Direction		Spherical Model 1		Spherical Model 2	
				Sill	Range	Sill	Range
	Gamma (h)	(Vulcan BPD Angles)		Gamma (h)	(m)	Gamma (h)	(m)
Leach	0.09	Bearing Plunge Dip	324.9 -1.7 170.1	0.39	96 123 68	0.52	339 539 212
Propylitic	0.26	Bearing Plunge Dip	284.5 2.1 24.9	0.50	19 25 16	0.24	51 144 63
Phyllic	0.10	Bearing Plunge Dip	334.9 2.1 4.5	0.18	73 110 29	0.73	383 341 140
Pot Wk	0.21	Bearing Plunge Dip	281.5 3.5 135.1	0.40	80 36 23	0.39	270 181 177
Pot Str	0.27	Bearing Plunge Dip	339.1 5.1 19.4	0.31	43 65 30	0.42	94 179 66

13.3.7 Block Model

A block model was constructed in Vulcan software as a standard block model. Two percent variables were coded to the model; a percent below topography and a percent inside the post mineralization dykes. Due to the narrow nature of the modeled post mineralization dykes, a percent model was generated to represent the dyke volumes, and a script was run to calculate the final block estimate diluted by the post mineralization dyke volumes. The block model was not rotated. The model parameters are shown in Table 13.14.

Table 13.14: Kemess Underground Block Model Parameters - Local Mine Grid

Local Mine Grid	Min	Max	Cell Size	No. Cells
East	9,800	11,350	10	155
North	15,500	16,700	10	120
Elevation	800	1,900	10	116

13.3.8 Specific Gravity Estimation

The specific gravity estimation was constrained by the previously defined domains (refer to Section 13.3.4). The inverse distance squared interpolation algorithm was applied to estimate block specific gravity values, as shown in Table 13.15.

An average domain density value was applied to any blocks where the below estimation parameters were not satisfied.

Table 13.15: Kemess Underground Specific Gravity Estimation Parameters

Domain	Default Mean	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Brg	Plunge	Dip	Major	Semi	Minor
BKu	2.77	5	18	4	280	-1.0	15	250	150	100
HAZ	2.69	5	18	4	280	-1.0	15	250	150	100
Takla Low	2.69	5	18	4	280	-1.0	15	250	150	100
Takla High	2.88	5	18	4	280	-1.0	15	250	150	100
Post Dyke	2.65	No estimation – All Post Mineral Rocks assigned the default average								

13.3.9 Grade Estimation

For estimation of copper, gold, and silver values, a multi-pass estimation routine was used. An anisotropic search routine was used for the estimation, based on the results of the variography. The rotations used for the search ellipses are the same as the respective variogram model rotation for each variable in each domain.

The estimation was generated using Maptek Vulcan software. Ordinary Kriging (OK) was used for copper, gold, and silver for the five domains (Leach, Propylitic, Phyllic, Potassic Weak, and Potassic Strong).

The grade of the post mineral intrusive dykes was set to a default detection limit grade. Due to the limited data set for silver, fewer blocks were estimated. These blocks were left as a default silver value of 0 g/t Ag. Discretization was set to 4x, 4y, and 2z for all estimations.

The estimation parameters are shown in Table 13.16 for copper, in Table 13.17 for gold, and in Table 13.18 for silver.

Table 13.16: Kemess Underground Estimation Parameters – Copper

Domain	Pass	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Bearing	Plunge	Dip	Major	Semi	Minor
Leach	1	4	12	3	310	0	0	225	125	75
	2	3	10	3				325	200	100
Propylitic	1	4	12	3	275	-2.0	20	80	80	50
	2	3	10	3				150	150	100
Phyllic	1	4	12	3	355	-2.0	20	225	225	60
	2	3	10	3				325	325	100
Pot Wk	1	4	12	3	330	0	20	75	60	30
	2	3	10	3				150	150	100
Pot Str	1	4	12	3	306	3	160	60	60	30
	2	3	10	3				150	150	75

Table 13.17: Kemess Underground Estimation Parameters – Gold

Domain	Pass	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Bearing	Plunge	Dip	Major	Semi	Minor
Leach	1	4	12	3	354	-3.2	140	125	75	50
	2	3	10	3				200	200	100
Propylitic	1	4	12	3	0	0	0	75	75	50
	2	3	10	3				150	150	100
Phyllic	1	4	12	3	341	3.2	140	200	250	50
	2	3	10	3				325	400	100

Domain	Pass	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Bearing	Plunge	Dip	Major	Semi	Minor
Pot Wk	1	4	12	3	345	0	170	75	60	50
	2	3	10	3				200	250	150
Pot Str	1	4	12	3	307	12	158	60	60	30
	2	3	10	3				150	150	75

Table 13.18: Kemess Underground Estimation Parameters – Silver

Domain	Pass	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Bearing	Plunge	Dip	Major	Semi	Minor
Leach	1	4	12	3	325	-2.0	170	150	200	100
	2	3	10	3				325	200	100
Propylitic	1	4	12	3	275	-2.0	20	150	150	100
	2	3	10	3				200	200	150
Phyllic	1	4	12	3	341	3.0	140	200	250	50
	2	3	10	3				325	325	150
Pot Wk	1	4	12	3	281	4.0	135	200	100	150
	2	3	10	3				250	150	200
Pot Str	1	4	12	3	339	5.0	19.0	100	150	50
	2	3	10	3				350	350	200

13.3.10 Model Validation

Various measures have been implemented to validate the resultant resource block model. These measures include the following:

- Visual comparison of drill hole composites with resource block grade estimates by zone, both in plan and section;
- Statistical comparisons between block and composite data; and
- Swath plot analysis (drift analysis) comparing the OK model with the underlying composite data, and declustered composite data.

The block model was initially validated visually by comparing block grades against drill hole composite data in cross-section and plan view for the individual domains. Figure 13.14 and Figure 13.15 are two representative sections showing the visual comparison of block estimates with the composite data.

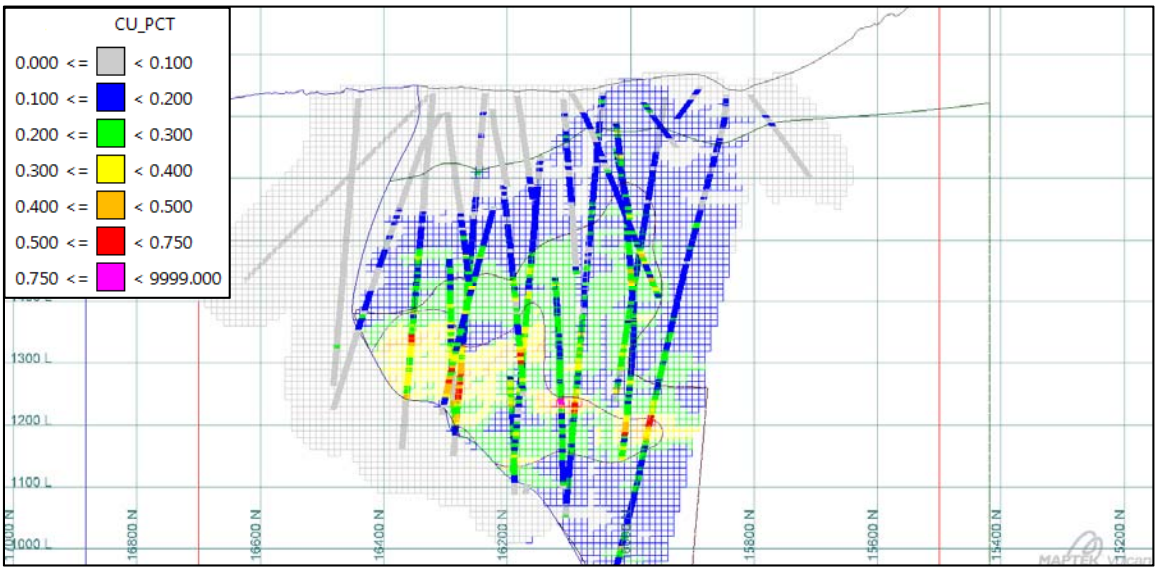


Figure 13.14: Kemess Underground Section 10,350E – Copper

(Source: SRK, 2016) Note – Grid spacing on section is 100m Vertical and 200m Horizontal

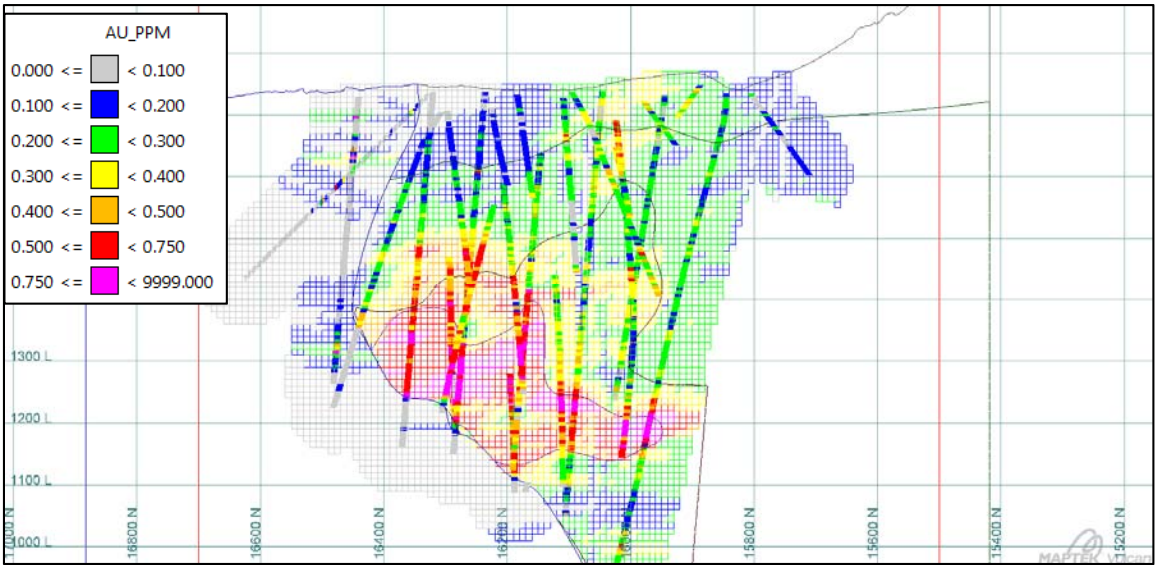


Figure 13.15: Kemess Underground Section 10,350E – Gold

(Source: SRK, 2016) Note – Grid spacing on section is 100m Vertical and 200m Horizontal

Global comparisons of the mean grade of the blocks against the mean grade of the un-declustered and declustered drillhole composites are outlined in Table 13.19.

Table 13.19: Kemess Underground Model Statistical Comparison

Domain	Variable	6.0 m Comps	NN Model	OK Model
Leach	Cu	0.074	0.077	0.078
	Au	0.181	0.174	0.177
	Ag	0.600	0.800	0.740
Prop	Cu	0.014	0.012	0.014
	Au	0.070	0.058	0.058
	Ag	0.380	0.390	0.400
Phy	Cu	0.113	0.100	0.102
	Au	0.205	0.169	0.172
	Ag	1.040	1.040	1.090
Pot Wk	Cu	0.200	0.185	0.185
	Au	0.325	0.295	0.294
	Ag	1.710	1.410	1.470
Pot Str	Cu	0.386	0.365	0.363
	Au	0.900	0.831	0.825
	Ag	2.730	2.580	2.560

The model was imported into Snowden Supervisor software in order to statistically validate the model globally and by estimation domain. Swath plots were generated to validate the spatial trends within the model, as shown in Figure 13.16, Figure 13.17 and Figure 13.18 for copper, gold and silver respectively. As part of the validation process, histogram comparisons, log probability plot comparisons, grade-tonnage comparisons, and QQ plot comparisons were reviewed.

It was concluded that the model does an adequate job of matching the data, and that no global bias is present, as the OK model generates the same tonnes and grade as the NN model at a zero cut-off, with no shift in the mean value on the histogram comparisons. Locally the estimations were reviewed by graphing the mean grade of the variable against the easting, northing and elevation for each of the domains (swath plots). As expected the mean grade of the composited samples is more variable than the mean grade of the blocks which are smoothed by kriging. The block averages in general bisect the sample plot indicating no bias between the model estimate and the samples. Larger variation between the means can be correlated with a reduction in the number of samples.

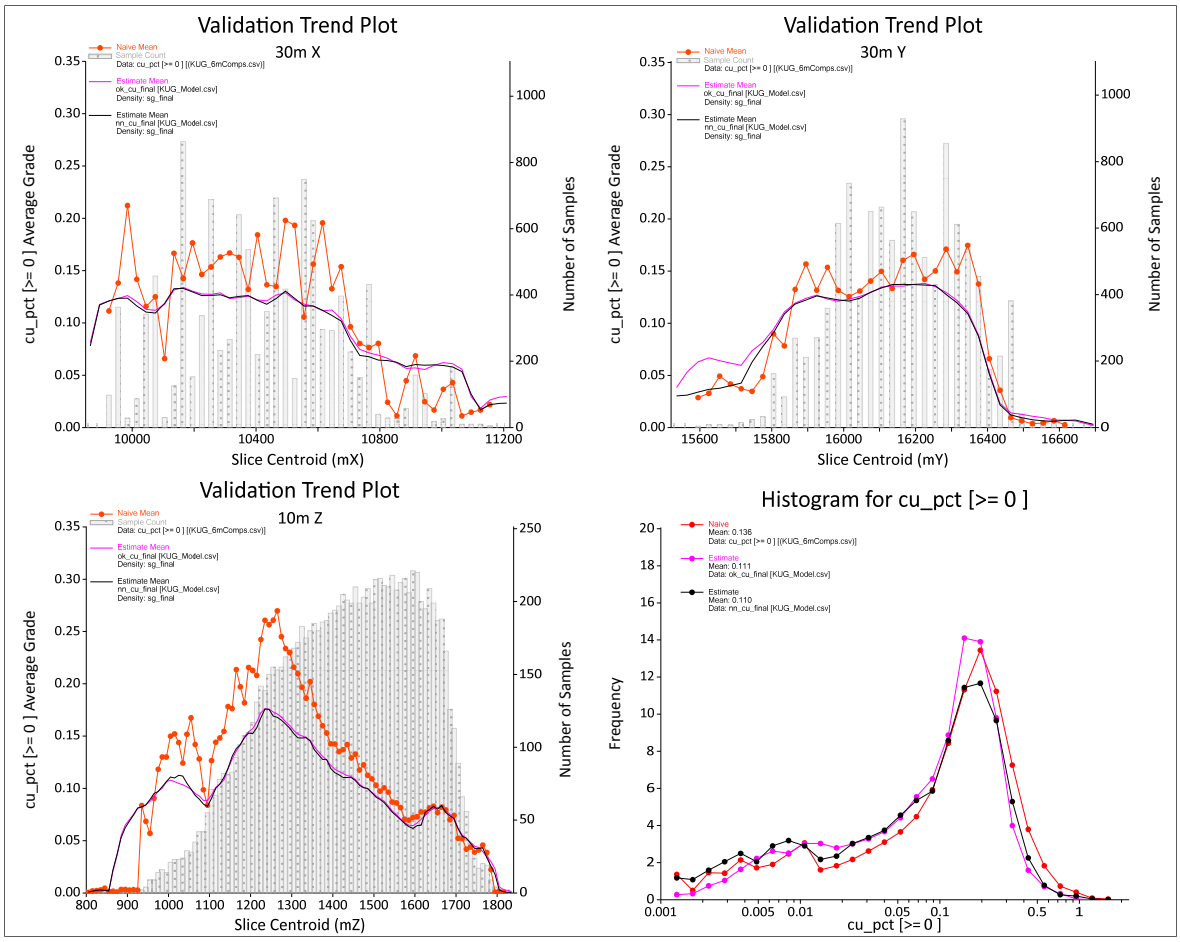


Figure 13.16: Kemess Underground Model Validation – Copper

(Source: SRK, 2016)

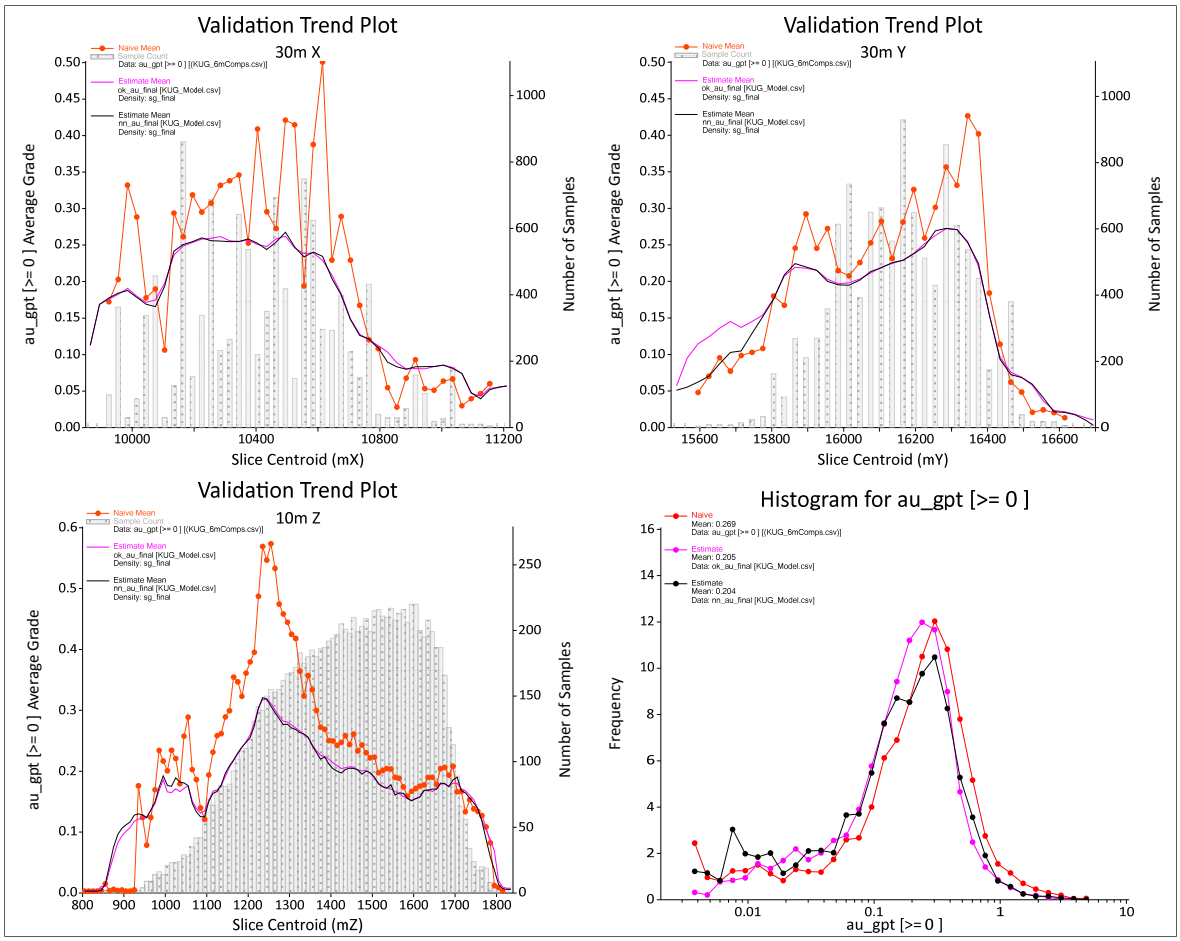


Figure 13.17 Kemess Underground Model Validation – Gold

(Source: SRK, 2016)

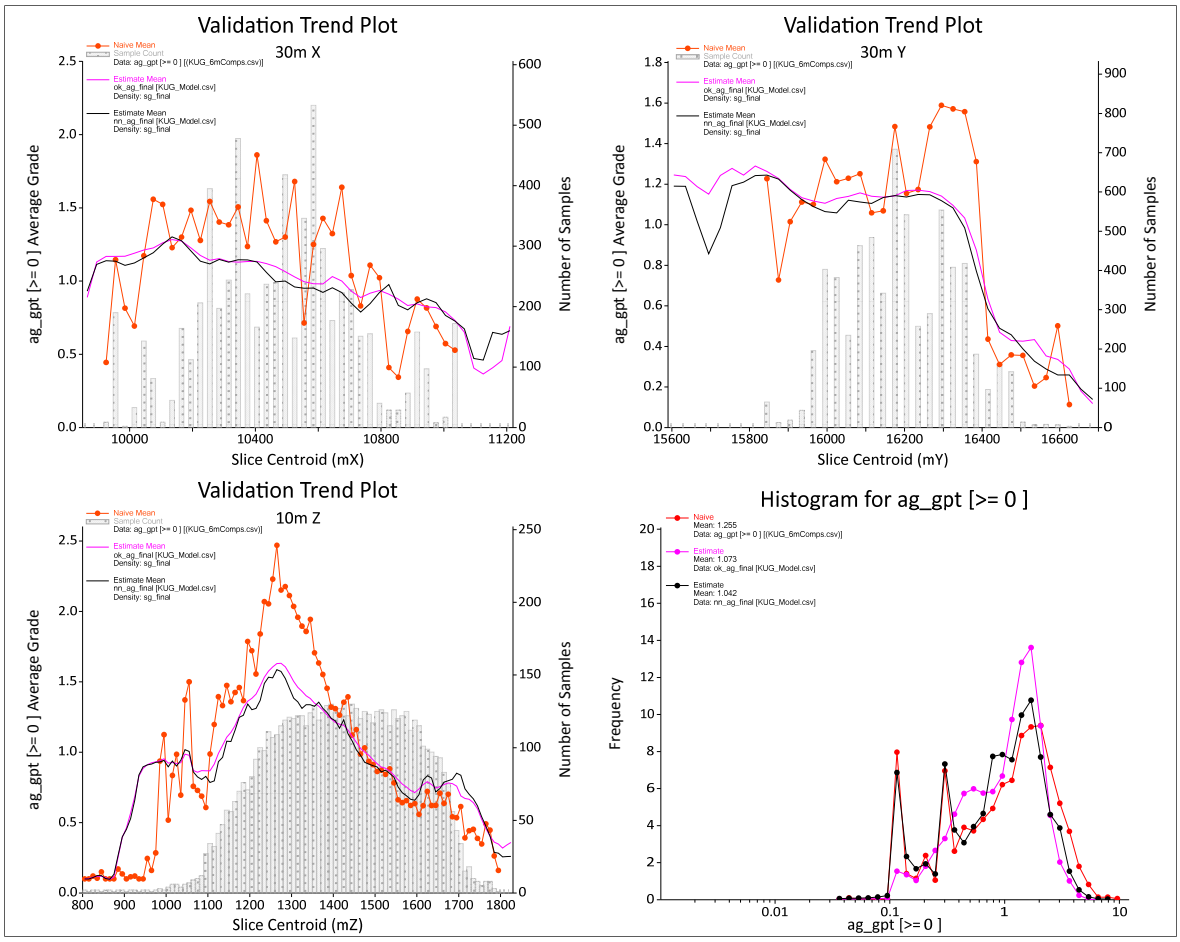


Figure 13.18: Kemess Underground Model Validation – Silver

(Source: SRK, 2016)

13.3.11 Mineral Resource Classification

The Resources at the Project are classified under the categories of Measured, Indicated, and Inferred according to standards as defined by the “CIM Definition Standards - For Mineral Resources and Mineral Reserves”, prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014.

Classification of the resources reflects the relative confidence of the grade estimates. This is based on several factors including; sample spacing relative to geological and geostatistical observations regarding the continuity of mineralization, mining history, specific gravity determinations, accuracy of drill collar locations, quality of the assay data, and many other factors which influence the confidence of the mineral estimation.

The classification parameters are defined in relation to the number of drillholes used to estimate the block grades and the block-composite separation distance. These classification criteria are

intended to encompass zones of reasonably continuous mineralization. At this stage no blocks have been classified as Measured.

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 primarily by core drilling on sections spaced at approximately 50 m to 80 m.

The classification of Indicated and Inferred Mineral Resources is based on geological confidence, drill hole spacing, and quality of the estimates. Porphyry copper deposits are characterised by large volumes of continuous grade over large distances radially distributed around and within the source intrusion. Post-mineral modification has offset the mineralization, i.e. Kemess North Fault and dykes have removed mineralised volumes. Confidence in the estimation domains supports an Indicated classification. This classification has been applied to the estimation of copper, gold, and silver within the best-defined and highest grade domains. Lower confidence domains, such as the Hazelton lithological unit and the Leach Zone were downgraded to inferred if the applicable indicated parameters were achieved. The Leached zone of the Talka Formation has 83 intervals of lost core and has been classified as completely Inferred.

The drill hole spacing of the Indicated resource is predominantly 80 m x 80 m between drill holes, with a small area to the north-east of approximately 50 m x 50 m spacing. Occasionally there is 100 m x 120 m spacing between drill holes at lower elevations due to down-hole deviation. The spatial continuity is reasonable for all domains except the Hazelton. The Inferred Mineral Resource classification has drill hole spacing ranging from 50 m x 100 m to 110 m x 130 m.

A post-process cleaning was conducted on the model classification to remove isolated indicated blocks surrounded by inferred, and isolated inferred blocks surrounded by indicated. A manual edit was performed by generating an indicated polygon for each level of the block model, and any inferred blocks within the polygon were converted to indicated. This cleaning generates a contiguous volume of indicated blocks that can then be used for reserve estimates

13.3.12 Mineral Resource Sensitivity

In order to assess the sensitivity of the KUG resource to changes in NSR cut-off grade, SRK summarized tonnage and grade above cut-off at a series of increasing NSR cut-offs by resource category.

The sensitivity analysis for Indicated blocks within the KUG resource model is provided in Table 13.20. It can be observed that the resource is reasonably insensitive to NSR cut-off values in the increment from C$14.00/t NSR to C$16.00/t NSR, which is likely the NSR range of interest. The resource base cut-offs are shown in bold.

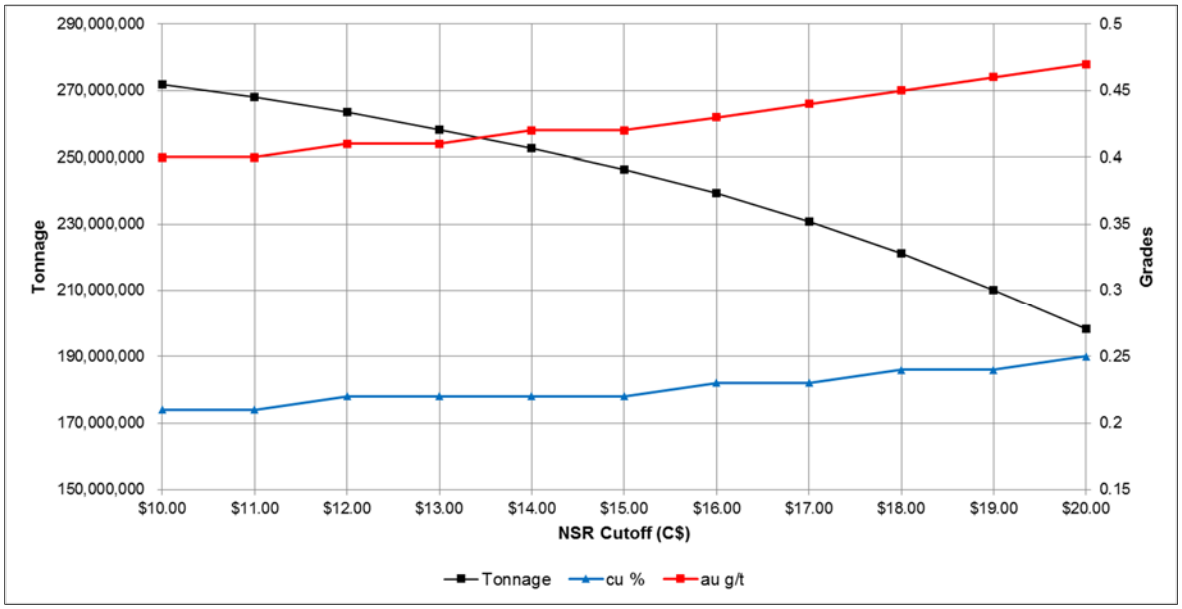
The sensitivity analysis for Inferred blocks within the KUG resource model is provided in Table 13.21. It shows the resource is reasonably insensitive to NSR cut-off values in the increment between C$14.00/t NSR to C$16.00/t NSR, which is likely the NSR range of interest. The resource base cut-offs are shown in bold.

Grade tonnage curves are illustrated in Figure 13.19 and Figure 13.20 for Indicated and Inferred resources respectively. These tonnages may not all be classified as Indicated and Inferred resource as this is reliant on the economic cut-offs chosen and geometry considerations. At higher cut-off the uncertainty increases.

Table 13.20: Kemess Underground Indicated Mineral Resource Sensitivity

Cut-off Value (C$/t NSR)	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Contained Metal	
				Copper (000's lbs)	Gold (000's oz)
10.0	271,900	0.21	0.40	1,258,900	3,497
11.0	268,100	0.21	0.40	1,241,200	3,448
12.0	263,400	0.22	0.41	1,277,800	3,473
13.0	258,300	0.22	0.41	1,252,700	3,405
14.0	252,600	0.22	0.42	1,225,100	3,411
15.0	**246,400**	**0.22**	**0.42**	**1,195,300**	**3,328**
16.0	239,400	0.23	0.43	1,213,800	3,309
17.0	230,900	0.23	0.44	1,170,600	3,266
18.0	221,100	0.24	0.45	1,170,100	3,199
19.0	210,100	0.24	0.46	1,111,700	3,107
20.0	198,300	0.25	0.47	1,093,100	2,997

Note: (Excluding Leach Zone and Hazelton) Inclusive of Reserves



Note: (Excluding Leach Zone and Hazelton) Inclusive of Reserves

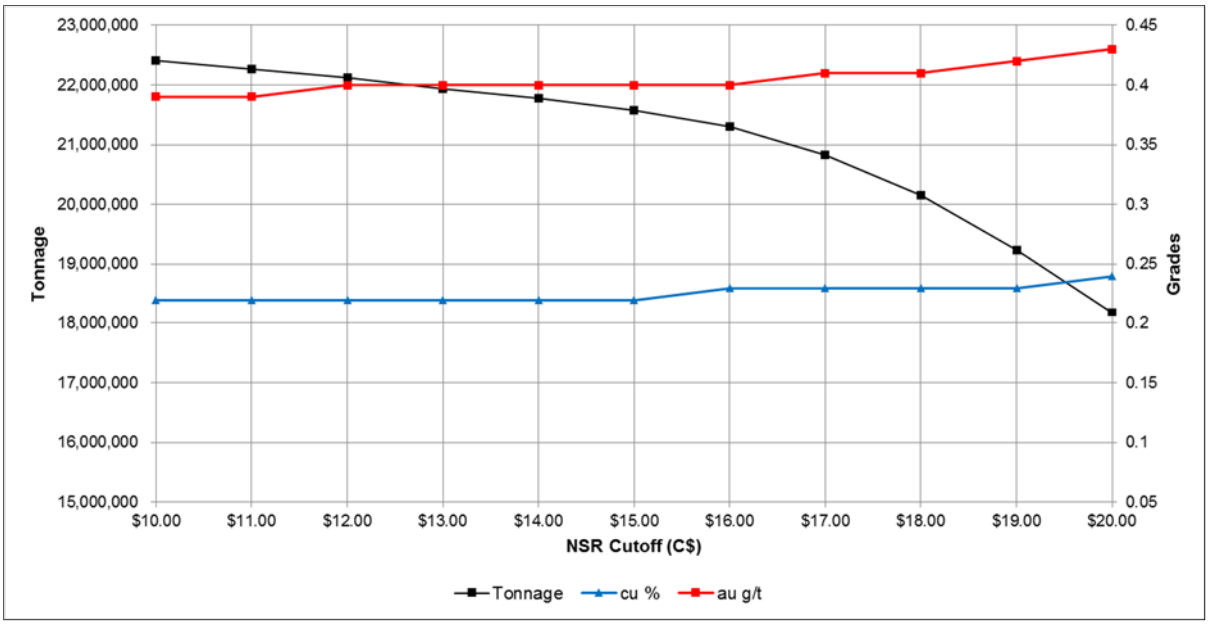
Figure 13.19: Kemess Underground Indicated Mineral Resource Sensitivity Plot

(Source: SRK, 2016)

Table 13.21 Kemess Underground Inferred Mineral Resource Sensitivity

Cut-off Value (C$/t NSR)	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Contained Metal	
				Copper (000's lbs)	Gold (000's oz)
10.0	22,400	0.22	0.39	108,700	281
11.0	22,300	0.22	0.39	108,000	279
12.0	22,100	0.22	0.40	107,300	285
13.0	21,900	0.22	0.40	106,400	282
14.0	21,800	0.22	0.40	105,600	280
15.0	**21,600**	**0.22**	**0.40**	**104,700**	**277**
16.0	21,300	0.23	0.40	108,000	274
17.0	20,800	0.23	0.41	105,600	274
18.0	20,100	0.23	0.41	102,200	266
19.0	19,200	0.24	0.42	97,600	260
20.0	18,200	0.24	0.43	96,200	251

Note: (Excluding Leach Zone and Hazelton) Inclusive of Reserves



Note: (Excluding Leach Zone and Hazelton) Inclusive of Reserves

Figure 13.20 Kemess Underground Inferred Mineral Resource Sensitivity Plot

(Source: SRK, 2016)

13.3.13 Mineral Resource Statement

The Mineral Resources (inclusive of Mineral Reserves) for the Kemess Underground (KUG) gold-copper-silver project, located in north-central British Columbia, have been estimated by SRK at 246,400kt at an average grade of 0.42 g/t Au, 0.22% Cu, and 1.75 g/t Ag, classified as Indicated Mineral Resources; with an additional 21,600kt at an average grade of 0.40 g/t Au, 0.22% Cu, and 1.70 g/t Ag, classified as Inferred Mineral Resources. The Mineral Resources are stated above a C$15.00/t NSR cut-off.

The Mineral Resources are reported in accordance with CSA, NI 43-101, and have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. The resource estimate was carried out by Chad Yuhasz, PGeo, APEGBC, Senior Consultant (Resource Geology), SRK Consulting. Mr. Yuhasz has over 13 years of operational and consulting experience in the minerals industry, specifically in Mineral Resource estimation, production geology, feasibility studies, and economic evaluations. Mr. Yuhasz is a Certified Professional Geologist and a Qualified Person as defined by international reporting codes.

The effective date of this Mineral Resource estimate is 29 February, 2016 and is based on all data available. The Mineral Resource statement for the Project is presented in Table 13.22. The resources for the Project are derived from the Vulcan block model using density values assigned as described in Section 13.3.8. Mineral Resources are reported inclusive of Mineral Reserves, and are constrained to a reasonable prospects for economic extraction solid, with all Leach Zone and Hazelton material excluded from this statement.

Table 13.22 Kemess Underground Mineral Resource Statement, KUG Gold-Copper-Silver Deposit, Northwest British Columbia, Canada, 29 February, 2016

Resource Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Contained Metal		
					Copper (000's lbs)	Gold (000's oz)	Silver (000's oz)
Measured							
Indicated	246,400	0.22	0.42	1.75	1,195,300	3,328	13,866
Measured + Indicated	**246,400**	**0.22**	**0.42**	**1.75**	**1,195,300**	**3,328**	**13,866**
Inferred	21,600	0.22	0.40	1.70	104,700	277	1,179

Notes:

1. Mineral Resources are inclusive of Mineral Reserves. Due to the complexity of cave mine scheduling, SRK is of the opinion that reporting resources inclusive of the reserves is a more accurate approach for reporting.
2. Resources stated are contained within a reasonable prospects for economic extraction solid above C$15.00/t NSR cut-off. A variable specific gravity value was assigned by domains for all model blocks.
3. NSR calculation is based on assumed copper, gold and silver prices of US$3.20/lb, US$1,275/oz and US$21.00/oz, respectively.
4. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.
5. Contained metals are in situ and undiluted, and do not include metallurgical recovery losses.

The “reasonable prospects for eventual economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off value, taking into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considers that portions of the KUG resource are amenable for underground extraction via panel cave mining development.

The 2016 Mineral Resource estimate has been prepared by identifying the portion of the resource model that could potentially be mined using panel cave mining methods, treating indicated and inferred blocks as potentially economic. The study identified a volume potentially suitable for block/panel cave mining at a Net Smelter Return (NSR) cut-off value of C$15/t.

The cost parameters were selected based on experience and benchmarking against similar projects. The reader is cautioned that the results from the cave optimization are used solely for the purpose of testing the “reasonable prospects for eventual economic extraction” by a panel cave and do not alone represent an attempt to estimate Mineral Reserves. The commodity prices (Table 13.23) are used to assist in the preparation of a Mineral Resource estimate and to select an appropriate resource reporting cut-off NSR value.

Table 13.23 Kemess Underground Commodity Prices, and Recovery Assumptions, used to Calculate NSR Values and Reasonable Cut-off

Commodity	Value	Unit
Copper	3.20	US$ per pound
Gold	1,275	US$ per ounce
Silver	21.00	US$ per ounce
Copper Recovery	91%	Assumed Recovery
Gold Recovery	72%	Assumed Recovery
Silver Recovery	65%	Assumed Recovery

Initial estimates for underground cave mining costs, combined with processing and general and administrative (G&A) costs consistent with the size of operation envisioned for the KUG project, indicate that a mine could be economically viable with a NSR cut-off in the range of C$14/t to C$16/t ore.

SRK considers that the blocks located within the conceptual panel cave show “reasonable prospects for economic extraction” and can be reported as a Mineral Resource.

13.3.14 Previous Mineral Resource Estimates

The previous mineral resource disclosure on the Kemess Underground deposit is shown below in Table 13.24. The previous mineral resource is stated at a C$13.00/t NSR cut-off, inside a potentially economically viable cave solid. In this table the mineral resources are reported exclusive of the previously disclosed probable mineral reserve of 100.4 Mt at 0.28% Cu and 0.56 g/t Au.

Table 13.24 Kemess Underground Mineral Resource Statement, Copper-Gold-Silver Deposit, Northwest British Columbia, Canada, December 31, 2012*

Resource Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Contained Metal		
					Copper (000's lbs)	Gold (000's oz)	Silver (000's oz)
Measured							
Indicated	65,432	0.24	0.41	1.81	346,546	854	3,811
Measured + Indicated	**65,432**	**0.24**	**0.41**	**1.81**	**346,546**	**854**	**3,811**
Inferred	9,969	0.21	0.39	1.57	46,101	125	503

Notes:

1. 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 resources estimated will be converted into mineral reserves.
2. Resources stated as contained within a potentially economically mineable solid above 13.00/t NSR cut-off. A variable specific gravity value was assigned by lithology domains for all model blocks.
3. NSR calculation is based on assumed copper, gold and silver prices of US$2.80/lb, US$1,100/oz and US$20.00/oz
4. Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, numbers may not add due to rounding.
5. Mineral resources are exclusive of mineral reserves.
6. Contained metals are in situ and undiluted, and do not include metallurgical recovery losses.

13.4 Kemess East (KE)

The Mineral Resource estimate presented in this section, represents an update to the Kemess East (KE) Mineral Resource based on the following updates:

- New Structural 3D Model,
- New Lithological 3D Model,
- New Alteration 3D Model,
- New Grade Estimation Domain Interpretation,
- Change in Grade Estimation Parameters, and
- Inclusion of additional drilling.

The Mineral Resource model was prepared by SRK considering 71 drillholes drilled by AuRico Metals and predecessor companies during the period of 2005 to 2015, of which 66 holes are located within the KE block model AOI. All onsite geological supervision, and the lithological and structural models were prepared by Wade Barnes, PGeo of AuRico Metals, a “Qualified Person” as defined in National Instrument NI 43-101, while the alteration model and grade estimation work was prepared by Chad Yuhasz, PGeo of SRK Vancouver, a “Qualified Person” as defined in National Instrument NI 43-101.

This section describes the resource estimation methodology and summarizes the key assumptions utilized in the estimation of copper, gold, silver, and molybdenum grades. It is the opinion of SRK, that the resource evaluation and processes reported herein are a reasonable representation of the Mineral Resource for the KE project. The Mineral Resources have been estimated and classified in conformity with generally accepted Canadian Institute of Mining (CIM) "Estimation of Mineral Resources and Mineral Reserves Best Practices" guidelines (2003), and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101 (2014). Mineral Resources are not Mineral Reserves and do not demonstrate economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserves.

The database used to estimate the KE project Mineral Resource was audited by SRK. SRK is of the opinion that the current drillhole information is sufficiently reliable and accurate to interpret with confidence the boundaries of the copper-gold porphyry mineralization, and the assay data are sufficiently reliable to support the Mineral Resource estimation.

Leapfrog Geo 3.0 software was used to construct the 3D geological solids, Snowden Supervisor 8.4 software was used to explore the statistical and geostatistical controls, Maptek Vulcan 9.1.5 software was used to construct the block model, prepare the assay data for estimation, complete the grade estimation, and tabulate the Mineral Resources.

13.4.1 Resource Database

Kemess East wasn't a primary focus of exploration during delineation of Kemess Underground, with the periods 2006-2007 and 2013-2015 being the main years for drilling programs at Kemess East. Prior to 2006, minor regional based drilling programs took place with select drilling in the vicinity of Kemess East. For example, in 2005 drilling intersected the Kemess Offset zone at depth adjacent to Kemess Underground. Various diamond drilling contractors based out of Smithers, British Columbia completed this work. Three Driftwood diamond drills completed the most recent program in 2015, which accounted for 12 drill holes and 17,872 m with all holes reaching target depth. The Kemess East exploration area, one kilometer east of the Kemess Underground Deposit, is approximately 1 km in an east-west direction, 1 km north-south, and over 1 km vertical. Drillhole spacing is 50 m to 200 m. The current Kemess East database contains 66 drill holes for a total of 66,403 m with an average length of 888 m and the deepest hole reaching 1,206 m. However, almost half of the holes are less than 700 m in depth while the vast majority of significant intercepts start below 700m. Of the 66 drill holes in the Kemess East AOI, 35 drill holes were drilled deep enough to test the mineralized zone. Table 13.25 highlights the Kemess East drill programs by year (within the KE block model AOI). Summary statistics for the KE database are presented in Table 13.26.

The broken zone, which covers much of the Kemess North area and presents challenging drilling conditions, is not present at Kemess East. Core recovery is very high with an average of 95%.

Table 13.25 Kemess East Drillhole Campaign

Year	No. of Holes	Total Metres
2002	1	490.7
2006	7	3,567.3
2007	27	17,712.6
2013	7	10,341
2014	12	16,872.5
2015	12	17,872.4
Total	**66**	**66,402.5**

Table 13.26 Kemess East Summary Statistics for drillhole data

Category	No. of Holes	No. of Records	Total Length (m)	Min Length (m)	Max Length	Av Length (m)
Collar Table	66	-	66,402.5	16.75	1,206.4	-
Cu Assays	-	34,832	62,758.8	0.2	12.0	1.80
Au Assays	-	34,832	62,758.8	0.2	12.0	1.80
Ag Assays	-	34,574	62,273.8	0.2	12.0	1.80
Mo Assays	-	34,574	62,273.8	0.2	12.0	1.80
Alteration	-	-	64,822.6	-	-	-
Lithology	-	-	65,939.2	-	-	-
Density	66	29,700	53,187.3	-	-	-

AuRico Metals implemented the acQuire database management system for the Kemess property in 2015. This process resulted in a comprehensive rebuild of the KE database, as data was transferred into the software. As part of the rebuild, all assays were imported into acQuire directly from the digital lab assay certificate, eliminating any data transcription errors. As reported in the QA/QC section of this report, all appropriate validation and checks were performed.

AuRico Metals designed an acQuire to MS Access export script within the acQuire software, and provided the database to SRK in MS Access format on 10 January 2016. This database was loaded in Maptek Vulcan software and a standard database validation routine was performed to check for the following errors:

- Overlapping intervals,
- Intervals deeper than collar table hole length,
- Out of sequence intervals,
- Missing data, and

- Erroneous downhole deviation.

No material errors were reported

13.4.2 Solid Body Modelling

The Kemess East deposit is classified as a porphyry style copper-gold-silver-molybdenum deposit. Previous models of the deposit focused on a combination geology / grade compartmentalization to control the mineralization. The 2016 model update re-interpreted the geology, and utilized experimental data analyses (EDA) of the data to characterize the copper, gold, silver, and molybdenum mineralization as a relationship to the deposit porphyry style lithology and alteration. The data was re-interpreted to fit a porphyry style model, and a 3D lithology and alteration models were generated to be used to define the mineralization, and form grade estimation domains.

Structural Model

AuRico Metals geologists utilized Leapfrog Geo to model the KE fault structures, as per the structural understanding of the deposit. In general, the mineralization within the KE deposit is well-defined and constrained within an interpreted network of bounding faults. The KE mineralization is cut-off in the east by a NW/SE striking fault that dips steeply to the SW, known as the Kemess East Offset Fault (KEOF). To the west, the current model limits the mineralization to the N/S striking, vertical dip fault, known as the West Bounding Fault. An E/W striking, vertical dipping fault, known as the Headwall Fault was used in the model to incorporate lithological offset. The mineralization continues on the south side of this fault, but in an uplifted block. Other faults were modelled to complete the structural interpretation. At this time, they were unused in terms of grade estimation controls.

These faults were modelled in 3D, and used to create fault domains in Leapfrog, which were then utilized during construction of the 3D lithological and alteration models. The fault domaining process ensures that only the drillhole data that exists within each fault block is used in the geological modelling of the solids. For the lithology, the fault domaining was used to generate offset of the lithological units at the fault surfaces. For the alteration, the fault domaining was used to limit the extrapolation of the solids.

This process ensures that the geological interpretations match the structural model of the deposit, while making sure that the final block model doesn’t extrapolate mineralization beyond the structures that limit the mineralized system.

Figure 13.21 shows the KE structural model, as interpreted by AuRico Metals geologists.

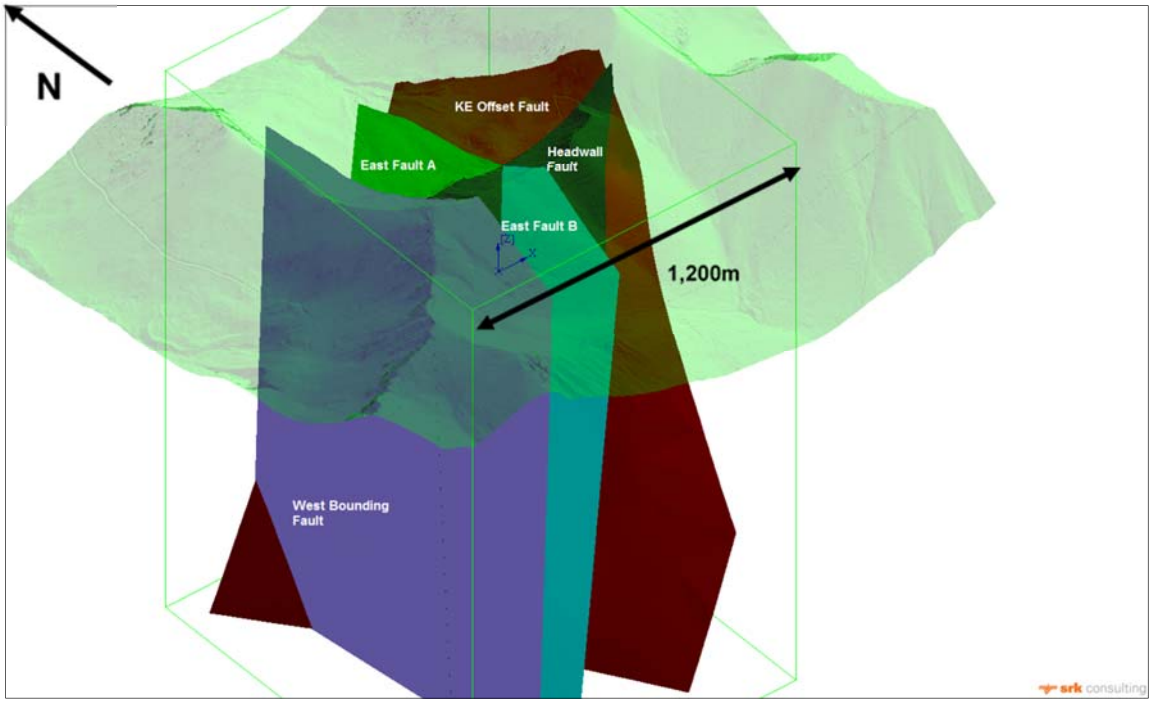


Figure 13.21: Kemess East Structural Model

(Source: SRK, 2016)

Lithology Model

AuRico Metals utilized Leapfrog Geo to model the KUG deposit lithology, as a function of the fault domaining highlighted in the structural model sub-section. Four primary lithological units have been identified within the KE deposit area of interest (AOI).

- Black Lake Intrusive (BKe) (Integer Code 11)
- Takla Volcanics (Takla) (Integer Code 10)
- Hazelton (Haz) (Integer Code 12)
- Post Mineralization Dykes (PDykes) (Integer Code 13 and 15)

Using Leapfrog Geo software, the lithological data was reviewed, and 3D fault domained lithological solids were constructed. While Leapfrog software was used to construct the initial lithological wireframes, the final interpretation was the result of detailed adjustment by AuRico geology staff. Figure 13.22 shows a 3D view of the final lithological wireframes.

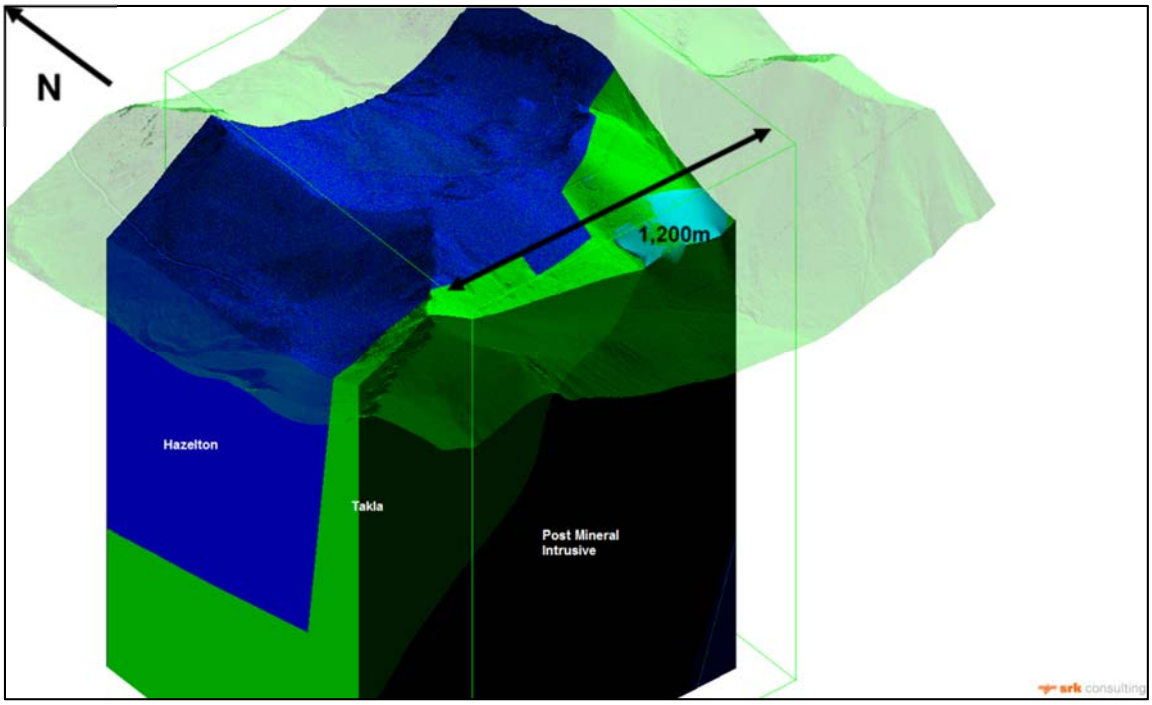


Figure 13.22: Kemess East Lithological Model

(Source: SRK, 2016)

Alteration Model

SRK geologists utilized Leapfrog Geo to model the KE deposit alteration, as a function of the fault domaining highlighted in the structural model sub-section. Three primary alteration units have been identified within the KE deposit area of interest (AOI).

- Propylitic (Prop) (Integer Code 22)
- Phyllic (Phy) (Integer Codes 21)
- Potassic (Pot) (Integer Codes 23, 24, and 25)

The leach unit present in the Kemess North deposit, is not present in the Kemess East deposit. The potassic zone was subdivided into three domains as a function of the visual potassic alteration intensity. The weak potassic zone is characterized by weak quartz veining and chlorite-sericite-quartz alteration assemblages. The moderate potassic zone is characterized by an increase in quartz veining. The strong potassic zone is characterized by intense quartz veining and stockworks, and strong chlorite-quartz-magnetite-biotite alteration assemblages. All experimental data analyses (EDA) analyses indicated distinct copper, gold, silver, and molybdenum populations as a function of potassic alteration intensity, with elevated grades distinctly correlated to increase in alteration intensity. Moly displayed a decrease in grade within the potassic strong core of the deposit.

Propylitic alteration is only seen across fault boundaries beyond the limits of economic mineralization, and as a result was not modeled.

Based on this analysis, four alteration domains were modeled. The resultant wireframes were reviewed on section to check for a geological match to the drilling, as well as reviewed to ensure the wireframes were geologically plausible. Manual edits, and the Leapfrog interval selection tool were used in some cases to generate a more acceptable wireframe. The final alteration model is shown in Figure 13.23.

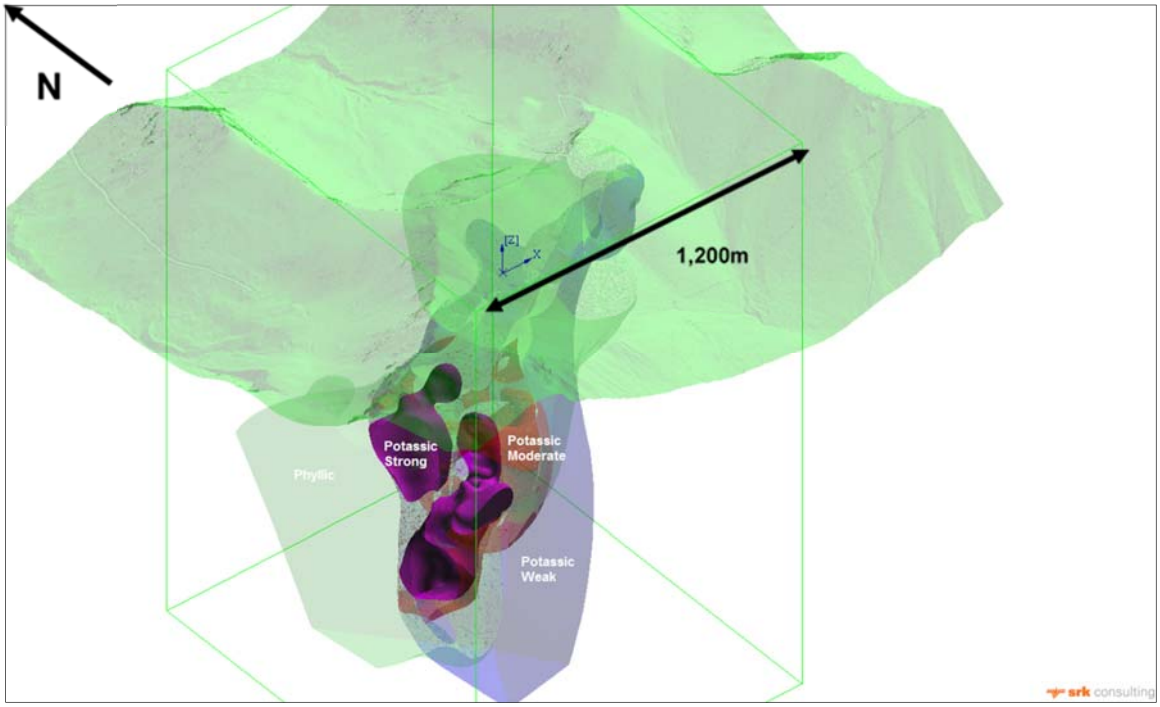


Figure 13.23: Kemess East Alteration Model

(Source: SRK, 2016)

13.4.3 Compositing

Logging protocols utilized by AuRico geologists in general applied a standard 2 m assay length for NQ core and 1.5 m assay length for HQ core, but do allow for an increase or decrease of sample length to allow for preservation of geological contacts. A standard assay length in porphyry copper systems is considered to be standard practice, as porphyry deposits are generally characterized by large volumes of continuous mineralization of large distances.

A statistical review of the KE assay database identified variable assay lengths, with mean sample interval lengths of approximately 2.0 m. Vulcan software was used to generate downhole composites of 6.0 m lengths with a tolerance of 3.0 m merge, by estimation domain. This option allowed any end-of-hole, or end-of-domain sample less than 3.0 m in length to be added to the previous interval. Any missing intervals were treated as missing during the compositing. The total length of available raw assay data for copper, gold, and silver was stored for each composite

interval. Any 6.0 m composite interval represented by less than 3.0 m of assay data, was excluded during the estimation routines. For the KE deposit, this resulted in the removal of 42 (from 10,215) copper composites, 42 (from 10,215) gold composites, 42 (from 10,134) silver composites, and 42 (from 10,134) molybdenum composites, from the final estimation database. This filter removes statistically underrepresented composites from use in the estimation of grades.

The compositing routine effectively grouped three raw assay intervals, and required a minimum of 50% of the composite interval to be represented by raw data. The final composites were statistically compared to the raw assays, to ensure no bias was introduced from the compositing.

In general, there were very few drillhole intervals without assay data. For the purpose of this model update all missing or unsampled intervals were treated as missing.

Specific gravity was composited into 3.0 m standard lengths, to preserve more sample variability.

13.4.4 Statistical Analysis and Estimation Domains

A full suite of experimental data analyses (EDA) was conducted for each of the copper, gold, silver, and moly variables. The grade populations were evaluated by lithology, alteration, and combination domains. Snowden Supervisor software, and Vulcan software were utilized to analyze the statistics of each variable, as well as to visualize the data trends in 3D.

Grade populations were found to be correlated and defined by the alteration domains. Each alteration domain supported unique sample populations, and unique mineralization trends. As a result, the four alteration domains were further evaluated independently for outliers, boundary analyses, and variography.

Copper

Copper EDA indicated a strong correlation with the modeled alteration domains, as can be quite common in porphyry systems. The potassic zone contains the bulk of the potentially economic mineralization, with a marked change in mean values between the weak, moderate and strong potassic alteration zones.

In general, the copper populations were found to be quite stable, with low variability; indicating the alteration domaining was doing a good job of supporting population stationarity. Figure 13.24 and Figure 13.25 show the copper log histograms and log probability plots by alteration domains.

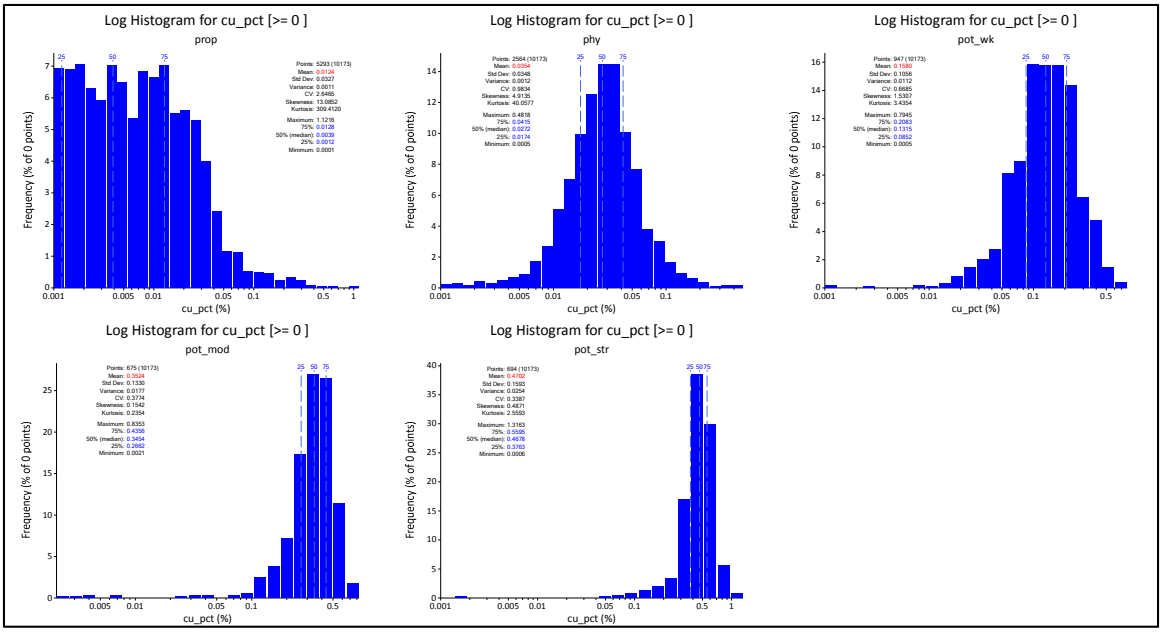


Figure 13.24: Kemess East Log Histograms - Copper (6 m composites)

(Source: SRK, 2016)

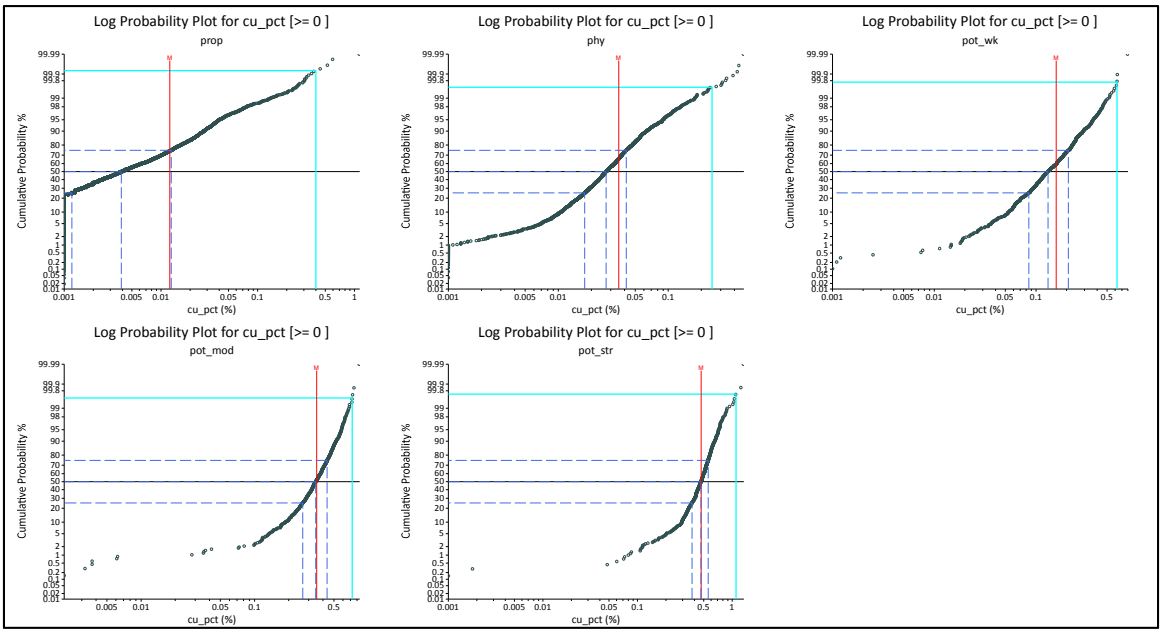


Figure 13.25: Kemess East Log Probability Plots - Copper (6 m composites)

(Source: SRK, 2016)

A full set of contact analyses plots were run for all contact possibilities, to identify the correct sample sharing to be utilized during grade estimation. Figure 13.26 is an example of how the copper grade population behaves at the potassic moderate / potassic strong alteration domain contact. Due to

the step in copper mean values at short distances from the contact, the final estimation routine treats this contact as a hard boundary. Thus no sample sharing is allowed across this contact.

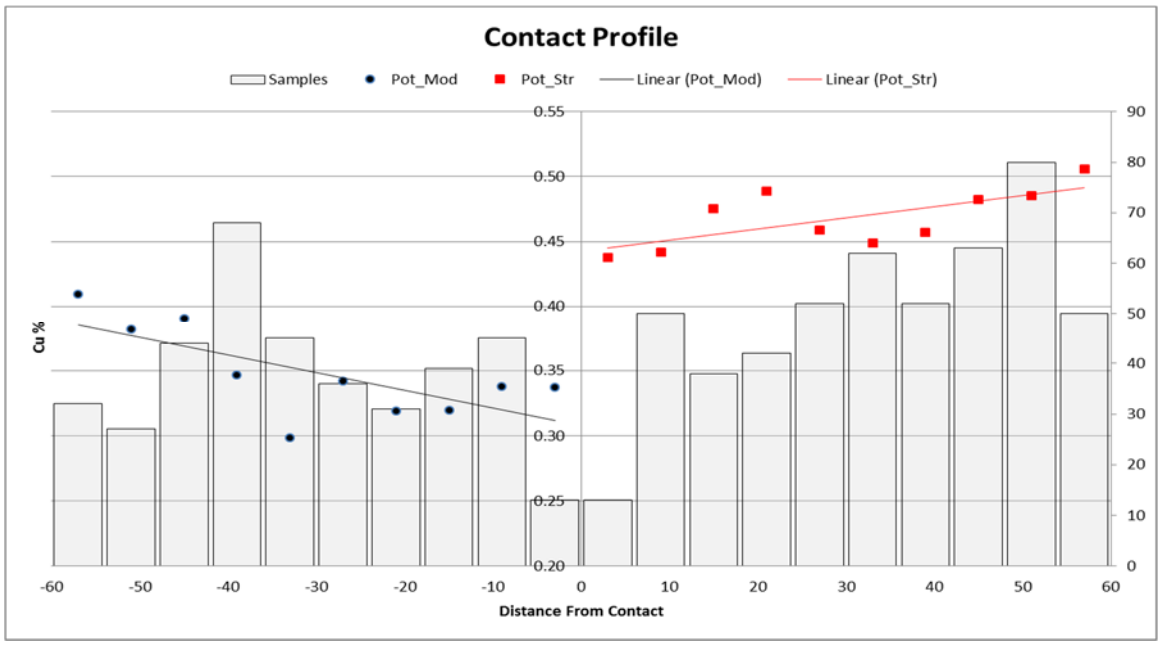


Figure 13.26: Kemess East Potassic Moderate vs Potassic Strong Contact Analysis - Copper (6 m composites)

(Source: SRK, 2016)

Table 13.27 indicates the full matrix of contact analysis used in the copper resource estimation.

Table 13.27: Kemess East Contact Analysis Matrix – Copper

Copper	Prop	Phy	Pot Wk	Pot Mod	Pot Str
Propylitic		Soft	Hard	Hard	Hard
Phyllic	Soft		Hard	Hard	Hard
Potassic Weak	Hard	Hard		Hard	Hard
Potassic Mod	Hard	Hard	Hard		Hard
Potassic Strong	Hard	Hard	Hard	Hard	

Gold

Gold EDA indicated a strong correlation with the modeled alteration domain, as can be quite common in porphyry systems. The potassic zone contains the bulk of the potentially economic mineralization, with a marked change in mean values between the weak, moderate and strong potassic alteration zones. In general gold showed a strong correlation to copper within each domain.

In general the gold populations were found to be quite stable, with low variability; indicating the alteration domaining was doing a good job of supporting population stationarity. Figure 13.27 and Figure 13.28 show the gold log histograms and log probability plots by alteration domains.

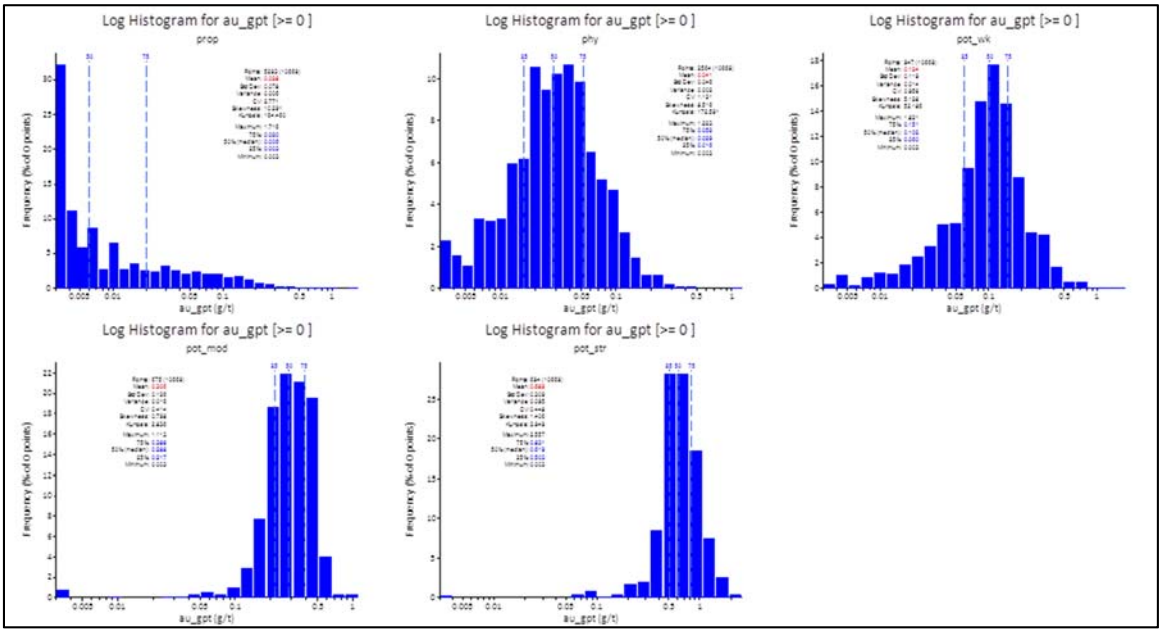


Figure 13.27: Kemess East Log Histograms - Gold (6 m composites)
(Source: SRK, 2016)

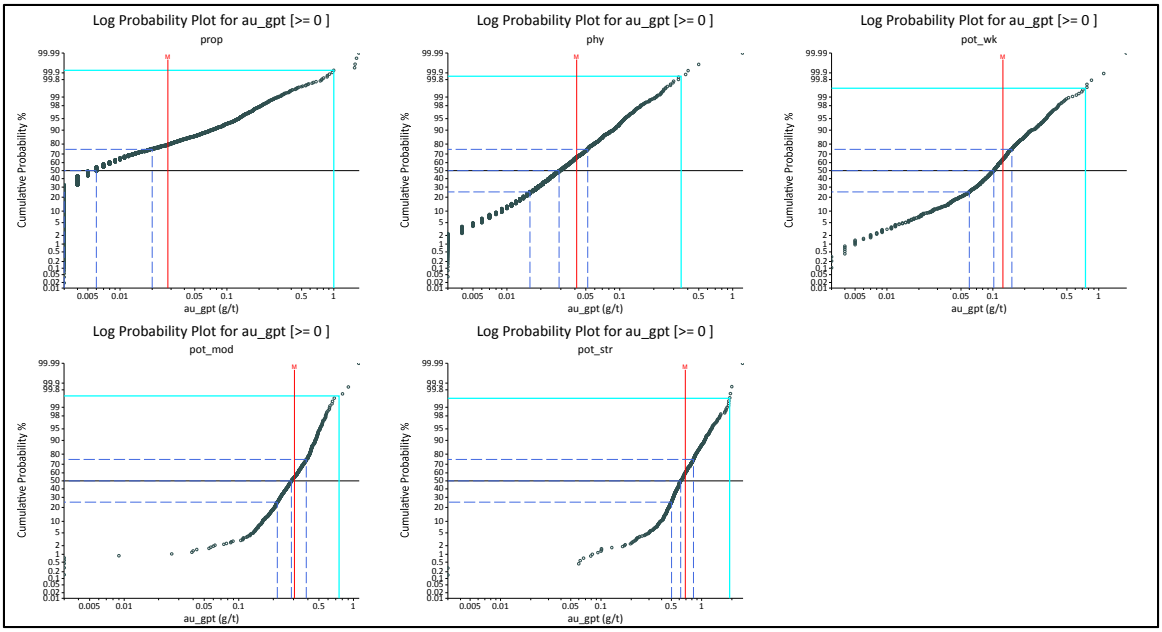


Figure 13.28: Kemess East Log Probability Plots - Gold (6 m composites)

(Source: SRK, 2016)

A full set of contact analyses plots were run for all contact possibilities, to identify the correct sample sharing to be utilized during grade estimation. Figure 13.29 is an example of how the gold grade population behaves at the potassic moderate / potassic strong alteration domain contact. Due to the step in gold mean values at short distances from the contact, the final estimation routine treats this contact as a hard boundary. Thus no sample sharing is allowed across this contact.

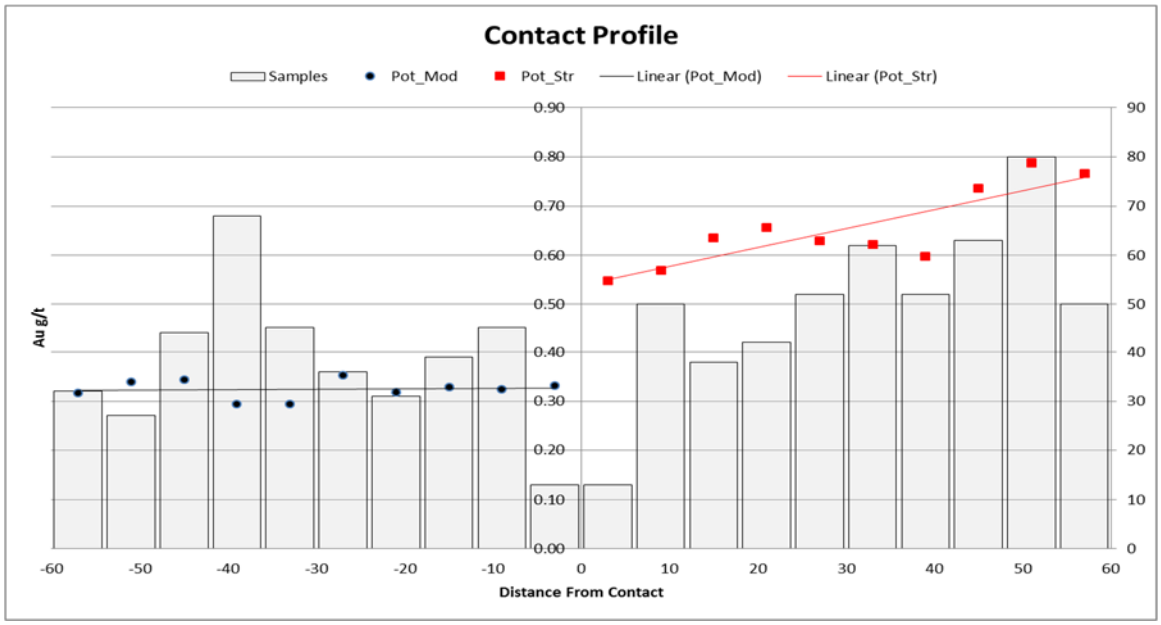


Figure 13.29: Kemess East Potassic Moderate vs Potassic Strong Contact Analysis - Gold (6 m composites)

(Source: SRK, 2016)

Table 13.28 indicates the full matrix of contact analysis used in the gold resource estimation.

Table 13.28 Kemess East Contact Analysis Matrix – Gold

Copper	Prop	Phy	Pot Wk	Pot Mod	Pot Str
Propylitic		Soft	Hard	Hard	Hard
Phyllic	Soft		Hard	Hard	Hard
Potassic Weak	Hard	Hard		Hard	Hard
Potassic Mod	Hard	Hard	Hard		Hard
Potassic Strong	Hard	Hard	Hard	Hard	

Silver

Silver EDA indicated a strong correlation with the modeled alteration domain, as can be quite common in porphyry systems. The potassic zone contains the bulk of the potentially economic mineralization, with a marked change in mean values between the weak, moderate and strong potassic alteration zones.

In general, the silver populations were found to be quite stable, with low variability; indicating the alteration domaining was doing a good job of supporting population stationarity. Figure 13.30 and Figure 13.31 show the silver log histograms and log probability plots by alteration domains.

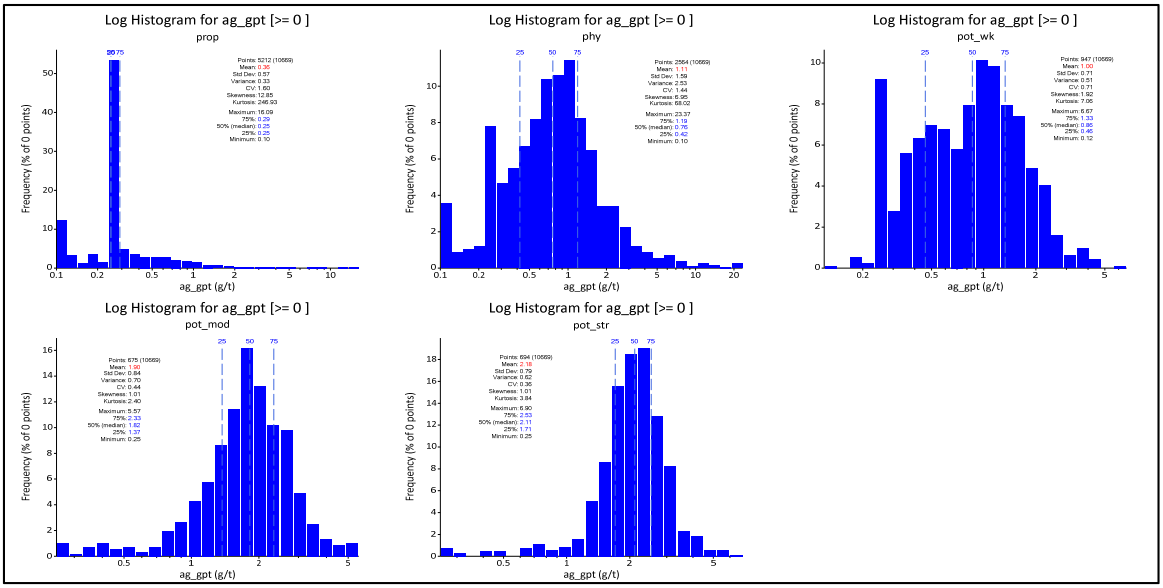


Figure 13.30: Kemess East Log Histograms - Silver (6 m composites)
(Source: SRK, 2016)

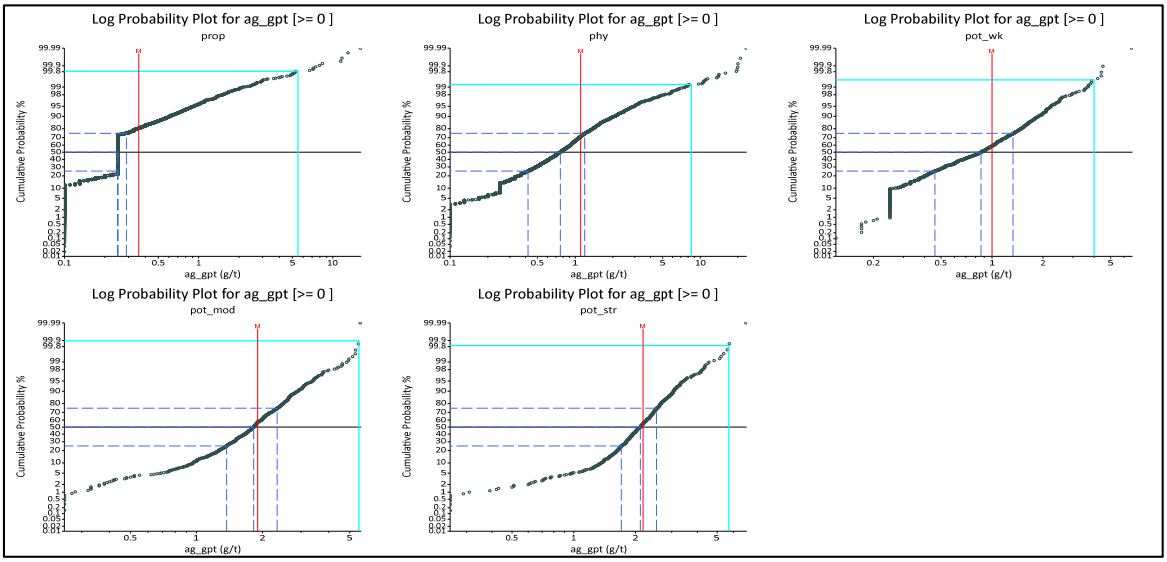


Figure 13.31: Kemess East Log Probability Plots - Silver (6 m composites)
(Source: SRK, 2016)

A full set of contact analyses plots were run for all contact possibilities, to identify the correct sample sharing to be utilized during grade estimation. Figure 13.32 is an example of how the silver grade

population behaves at the potassic moderate / potassic strong alteration domain contact. Due to the small step in silver mean values at short distances from the contact, the final estimation routine treats this contact as a soft boundary. Thus sample sharing is allowed across this contact.

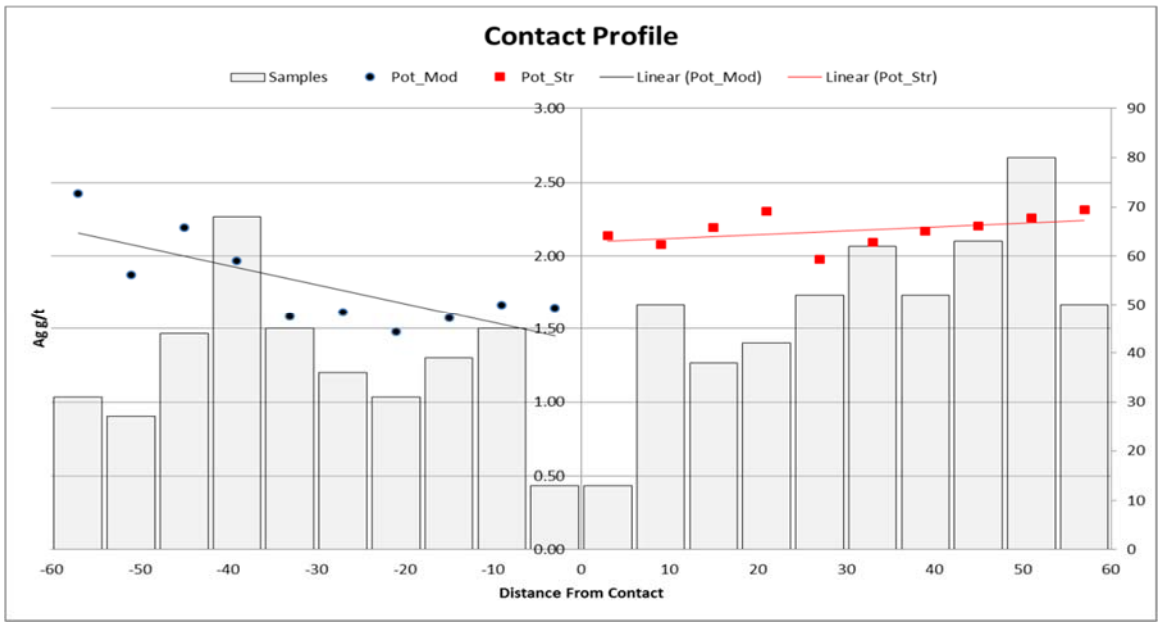


Figure 13.32 Kemess East Potassic Moderate vs Potassic Strong Contact Analysis - Silver (6 m composites)

(Source: SRK, 2016)

Table 13.29 indicates the full matrix of contact analysis used in the silver resource estimation.

Table 13.29 Kemess East Contact Analysis Matrix - Silver

Copper	Prop	Phy	Pot Wk	Pot Mod	Pot Str
Propylitic		Hard	Hard	Hard	Hard
Phyllic	Hard		Soft	Hard	Hard
Potassic Weak	Hard	Soft		Hard	Hard
Potassic Mod	Hard	Hard	Hard		Soft
Potassic Strong	Hard	Hard	Hard	Soft	

Molybdenum

Molybdenum EDA indicated a correlation with the modeled alteration domain, albeit with differing trends than the other variables of interest. The potassic zone contains the bulk of the potentially economic mineralization, with a marked change in mean values between the weak, moderate and

strong potassic alteration zones. The potassic strong zone shows a decrease in average molybdenum grades, indicating the core of the deposit with the highest grade copper and gold mineralization contains low molybdenum values.

In general, the molybdenum populations were found to be quite stable, with low variability; indicating the alteration domaining was doing a good job of supporting population stationarity. Figure 13.33 and Figure 13.34 show the molybdenum log histograms and log probability plots by alteration domains.

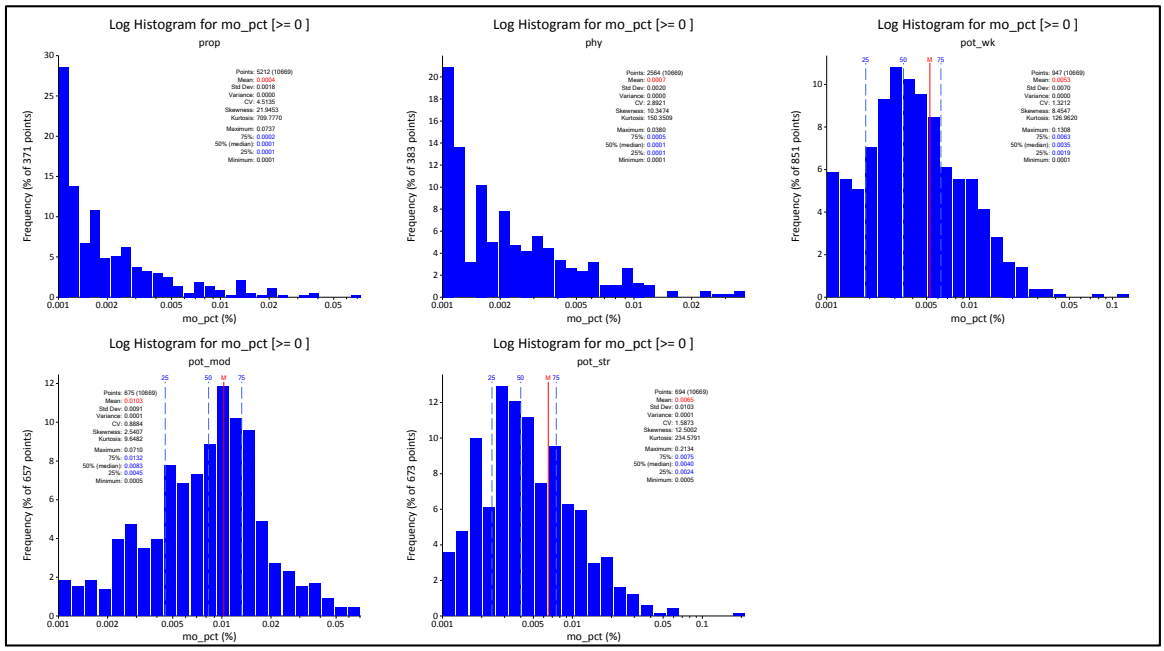


Figure 13.33: Kemess East Log Histograms - Molybdenum (6 m composites)
(Source: SRK, 2016)

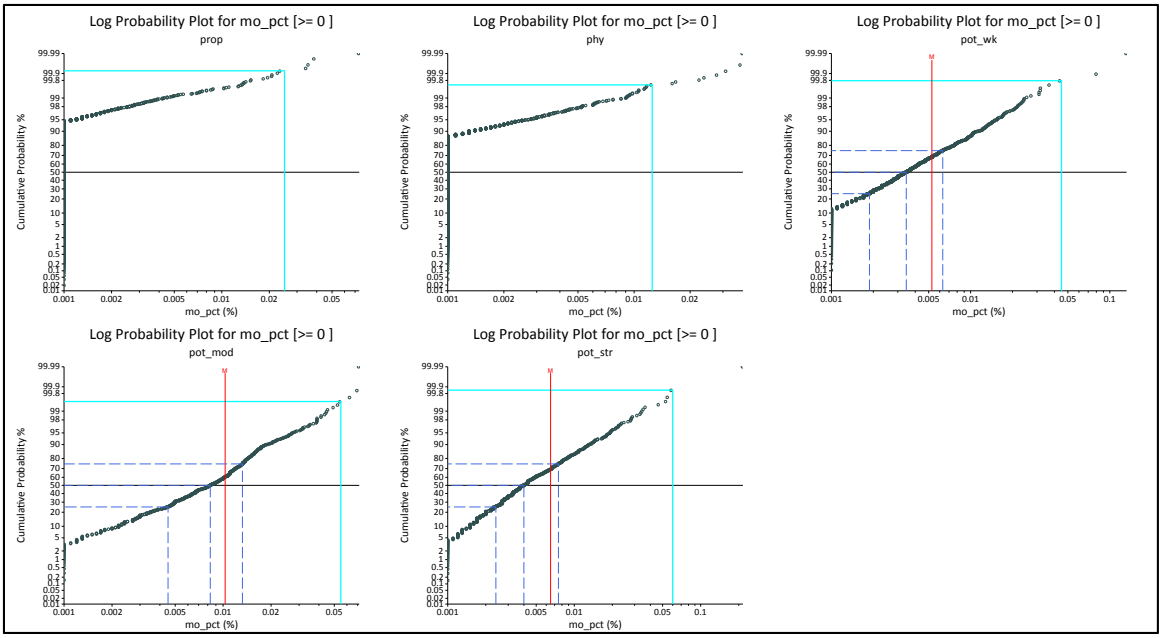


Figure 13.34: Kemess East Log Probability Plots - Molybdenum (6 m composites)

(Source: SRK, 2016)

A full set of contact analyses plots were run for all contact possibilities, to identify the correct sample sharing to be utilized during grade estimation. Figure 13.35 is an example of how the molybdenum grade population behaves at the potassic moderate / potassic strong alteration domain contact. Due to the relatively small step in molybdenum mean values at short distances from the contact, the final estimation routine treats this contact as a soft boundary. Thus sample sharing is allowed across this contact.

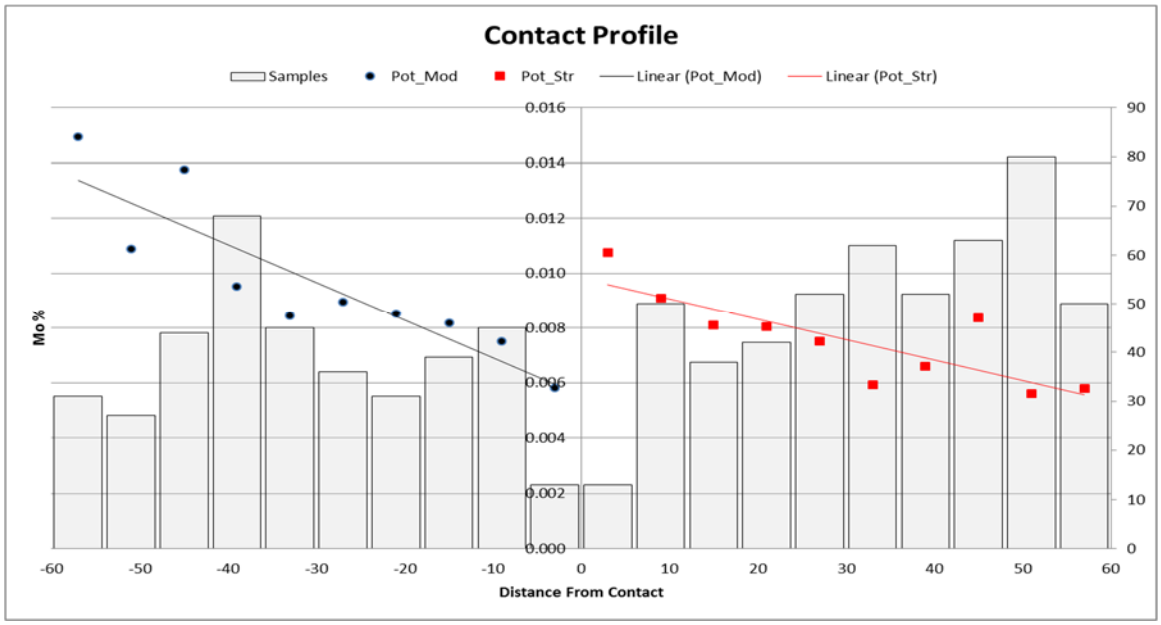


Figure 13.35: Kemess East Potassic Moderate vs Potassic Strong Contact Analysis - Molybdenum (6 m composites)

(Source: SRK, 2016)

Table 13.30 indicates the full matrix of contact analysis used in the moly resource estimation.

Table 13.30 Kemess East Contact Analysis Matrix - Molybdenum

Copper	Prop	Phy	Pot Wk	Pot Mod	Pot Str
Propylitic		Soft	Hard	Hard	Hard
Phyllic	Soft		Hard	Hard	Hard
Potassic Weak	Hard	Hard		Soft	Soft
Potassic Mod	Hard	Hard	Soft		Soft
Potassic Strong	Hard	Hard	Soft	Soft	

Specific Gravity

The Kemess East resource database contained 29,700 specific gravity composites, from 66 holes, within the KE resource area of interest (AOI). These values were utilized to perform a block estimate of specific gravity for the KE resource model.

Statistical EDA was performed on the 3.0 m specific gravity composites. Populations were found to be more strongly correlated to the lithology domains, as opposed to the alteration domains.

The statistical review highlighted the Hazelton and Takla could be combined into one estimation domain, while the Blake Lake Intrusive (BKe), and the Post Mineral Intrusive, should be treated as individual domains. The descriptive statistics for the data are presented in Table 13.31.

Table 13.31: Kemess East descriptive statistics of SG data - (3 m Composites)

Domain	# Samples	Min Density (g/cm^3)	Max Density (g/cm^3)	Mean Density (g/cm^3)	Cv
Black Lake Intrusive	3,033	2.22	3.30	2.67	0.04
Hazelton	7,507	2.27	3.38	2.75	0.04
Takla	6,205	2.24	3.68	2.78	0.04
Post Mineral	987	2.32	3.03	2.65	0.03

Post domaining, the composites were statistically reviewed and capping applied if required. Figure 13.36 shows the final estimation domain histograms, and Figure 13.37 shows the final estimation domain log probability plots.

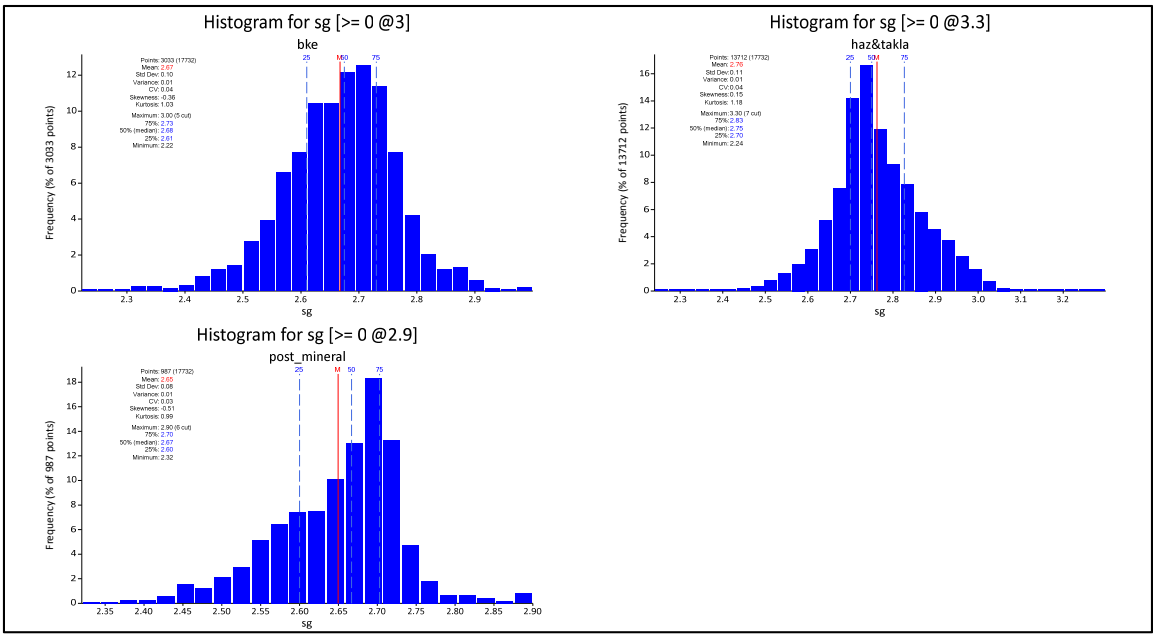


Figure 13.36: Kemess East Specific Gravity Histograms (3m Composites)

(Source: SRK, 2016)

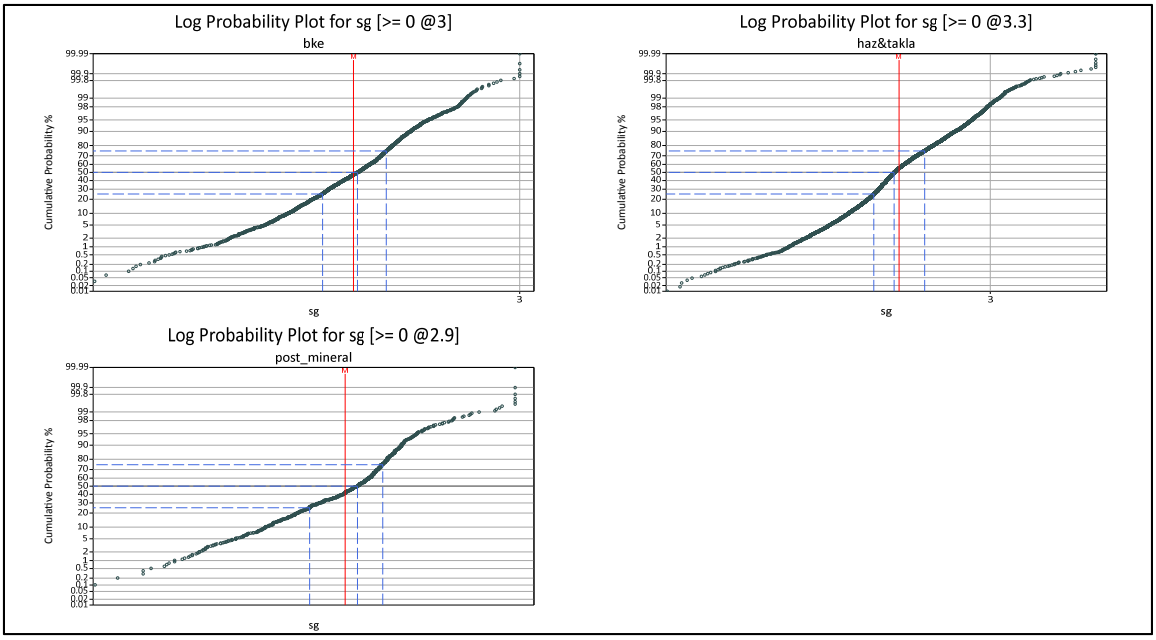


Figure 13.37: Kemess East Specific Gravity Log Probability Plots (3m Composites)

(Source: SRK, 2016)

13.4.5 Evaluation of Outliers

Post-compositing and post estimation domain generation, the database was evaluated for potential sample population outliers. Histograms and log probability plots were utilized to define breaks in the sample populations. The outlier evaluation was completed by estimation domain.

As the statistics indicated low variability sample populations, limited outliers were identified. Grade capping was applied to the outliers, and any value that exceeded the grade cap was systematically reduced to the grade cap value. A script was constructed in Vulcan software to populate a capped variable, and this variable was subsequently utilized in grade estimation.

Table 13.32, Table 13.33, Table 13.34, and

Table 13.35 identify the composite capping descriptive statistics for copper, gold, silver, and molybdenum respectively.

Table 13.32 Kemess East Capping Statistics - Copper (6 m composites)

Copper (%)	Prop	Phy	Pot WK	Pot Mod	Pot Str
Pre Cap Mean	0.012	0.035	0.158	0.352	0.470
Pre Cap CV	2.65	0.983	0.669	0.377	0.339
Cap Value	0.40	0.40	0.625	0.725	1.10
Number Capped	4	3	3	3	2
Post Cap Mean	0.012	0.035	0.158	0.352	0.470

Copper (%)	Prop	Phy	Pot WK	Pot Mod	Pot Str
Post Cap CV	2.33	0.967	0.663	0.376	0.334

Table 13.33 Kemess East Capping Statistics - Gold (6 m composites)

Gold (g/t)	Prop	Phy	Pot WK	Pot Mod	Pot Str
Pre Cap Mean	0.028	0.041	0.124	0.306	0.689
Pre Cap CV	2.771	1.121	0.959	0.414	0.448
Cap Value	1.00	0.35	0.75	0.75	1.90
Number Capped	4	4	5	3	4
Post Cap Mean	0.028	0.041	0.123	0.305	0.688
Post Cap CV	2.519	0.971	0.846	0.402	0.441

Table 13.34 Kemess East Capping Statistics - Silver (6 m composites)

Silver (g/t)	Prop	Phy	Pot WK	Pot Mod	Pot Str
Pre Cap Mean	0.360	1.110	1.000	1.900	2.180
Pre Cap CV	1.600	1.440	0.710	0.440	0.360
Cap Value	5.50	8.50	4.00	5.50	5.70
Number Capped	10	20	5	1	2
Post Cap Mean	0.350	1.060	1.000	1.900	2.170
Post Cap CV	1.26	1.110	0.690	0.440	0.360

Table 13.35 Kemess East Capping Statistics - Molybdenum (6 m composites)

Molybdenum (%)	Prop	Phy	Pot WK	Pot Mod	Pot Str
Pre Cap Mean	0.0004	0.0007	0.0053	0.0103	0.0065
Pre Cap CV	4.514	2.892	1.321	0.888	1.587
Cap Value	0.025	0.0125	0.045	0.055	0.060
Number Capped	4	8	2	3	1
Post Cap Mean	0.0004	0.0006	0.0051	0.0102	0.0063
Post Cap CV	3.583	2.236	1.052	0.862	1.114

13.4.6 Geostatistical Analysis and Variography

Snowden Supervisor software was utilized to run experimental variography for each variable within each of the four estimation domains. Once a direction of maximum continuity was defined, the directional variograms were fit with variogram models. Normal Score Transformation was utilized to best fit the experimental variograms with spherical models, or exponential models for molybdenum, and the models were then back transformed.

Upon completion of the variogram model fitting, the models were exported directly from Supervisor software in Vulcan format and the models were subsequently imported directly into the Vulcan estimation parameter files. As both gold and silver displayed strong correlation with copper in all estimation domains except the phyllic, the copper continuity orientations were superimposed on the gold and silver dataset. This will ensure the resultant block model will maintain this correlation. As the correlation in the phyllic zone was much weaker, continuity orientations were modeled independently.

The variogram models utilized for estimation are reported in Table 13.36, Table 13.37, Table 13.38, and Table 13.39 for copper, gold, silver, and molybdenum respectively.

Copper

The Variogram models used in the Ordinary Kriging estimation of copper are listed below in Table 13.36.

Table 13.36: Kemess East Variogram Models – Copper

Domain	Nugget Effect	Direction		Spherical Model 1		Spherical Model 2	
				Sill	Range	Sill	Range
	Gamma (h)	(Vulcan BPD Angles)		Gamma (h)	(m)	Gamma (h)	(m)
Phyllic	0.29	Bearing Plunge Dip	159 -10 168	0.34	34 43 22	0.37	184 250 92
Pot Weak	0.12	Bearing Plunge Dip	73 -12 -152	0.33	73 54 39	0.55	221 347 174
Pot Mod	0.12	Bearing Plunge Dip	125 -2 -160	0.32	93 50 25	0.56	199 219 107
Pot Strong	0.20	Bearing Plunge Dip	86 -5 -150	0.30	67 109 36	0.49	142 189 74

Gold

The Variogram models used in the Ordinary Kriging estimation of gold are listed below in Table 13.37.

Table 13.37: Kemess East Variogram Models – Gold

Domain	Nugget Effect	Direction		Spherical Model 1		Spherical Model 2	
				Sill	Range	Sill	Range
	Gamma (h)	(Vulcan BPD Angles)		Gamma (h)	(m)	Gamma (h)	(m)
Phyllic	0.14	Bearing Plunge Dip	44 -29 -102	0.31	78 30 26	0.55	506 307 437
Pot Weak	0.18	Bearing Plunge Dip	73 -12 -152	0.43	118 72 68	0.40	169 199 93
Pot Mod	0.17	Bearing Plunge Dip	125 -2 -160	0.22	93 43 21	0.60	180 202 80
Pot Strong	0.18	Bearing Plunge Dip	86 -5 -150	0.36	67 109 39	0.46	87 189 74

Silver

The Variogram models used in the Ordinary Kriging estimation of silver are listed below in Table 13.38.

Table 13.38: Kemess East Variogram Models – Silver

Domain	Nugget Effect	Direction		Spherical Model 1		Spherical Model 2	
				Sill	Range	Sill	Range
	Gamma (h)	(Vulcan BPD Angles)		Gamma (h)	(m)	Gamma (h)	(m)
Phyllic	0.27	Bearing Plunge Dip	60 52 -115	0.33	109 77 23	0.40	270 163 136
Pot Weak	0.16	Bearing Plunge Dip	73 -12 -152	0.31	82 100 28	0.53	253 387 156
Pot Mod	0.13	Bearing Plunge Dip	125 -2 -160	0.38	125 93 28	0.50	202 219 130
Pot Strong	0.21	Bearing Plunge Dip	86 -5 -150	0.30	46 109 17	0.49	88 221 70

Molybdenum

The Variogram models used in the Ordinary Kriging estimation of molybdenum are listed below in Table 13.39.

Table 13.39: Kemess East Variogram Models – Molybdenum

Domain	Nugget Effect	Direction		Exponential Model 1		Exponential Model 2	
				Sill	Range	Sill	Range
	Gamma (h)	(Vulcan BPD Angles)		Gamma (h)	(m)	Gamma (h)	(m)
Propylitic & Phyllic	0.29	Bearing Plunge Dip	165 20 -16	0.62	50 208 82	0.08	230 743 910
Potassic (Wk, Mod & Str)	0.26	Bearing Plunge Dip	110 3 -170	0.40	88 61 43	0.34	229 294 250

13.4.7 Block Model

A block model was constructed in Vulcan software as a standard block model. Two percent variables were coded to the model; a percent below topography and a percent inside the post mineralization dykes. Due to the narrow nature of the modeled post mineralization dykes, a percent model was generated to represent the dyke volumes, and a script was run to calculate the final block estimate diluted by the post mineralization dyke volumes. The block model was not rotated. The model parameters are shown in Table 13.40.

Table 13.40 Kemess East Block Model Parameters - Local Mine Grid

Local Mine Grid	Min	Max	Cell Size	No. Cells
East	11,300	12,500	10	120
North	15,600	17,000	10	140
Elevation	0	2,000	10	200

13.4.8 Specific Gravity Estimation

The specific gravity estimation was constrained by the previously defined lithology domains. Ordinary Kriging interpolation was applied to estimate block specific gravity values, as shown in Table 13.41 and Table 13.42. An average domain density value was applied to any blocks where the below estimation parameters were not satisfied.

Table 13.41: Kemess East Specific Gravity Variogram Models

Domain	Nugget Effect	Direction		Exponential Model 1		Exponential Model 2	
				Sill	Range	Sill	Range
	Gamma (h)	(Vulcan BPD Angles)		Gamma (h)	(m)	Gamma (h)	(m)
BKe & B	0.16	Bearing Plunge Dip	235 -1.7 175	0.18	278 116 20	0.66	334 156 265
HAZ & Takla	0.18	Bearing Plunge Dip	15 1.7 170	0.26	126 13 33	0.56	200 135 204

Table 13.42: Kemess East Specific Gravity Estimation Parameters

Domain	Default Mean	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Brg	Plunge	Dip	Major	Semi	Minor
BKe & B	2.67	5	18	4	280	-1.0	15	250	150	100
HAZ	2.76	5	18	4	280	-1.0	15	250	150	100
Takla	2.76	5	18	4	280	-1.0	15	250	150	100
Post Dyke	2.65	No estimation – All Post Mineral Rocks assigned the default average								

13.4.9 Grade Estimation

For estimation of copper, gold, silver, and molybdenum values, a multi-pass estimation routine was used. An anisotropic search routine was used for the estimation, based on the results of the variography. The rotations used for the search ellipses are the same as the respective variogram model rotation for each variable in each domain.

The estimation was generated using Maptek Vulcan software. Ordinary Kriging (OK) was used for copper, gold, silver, and molybdenum for the four domains (Phyllic, Potassic Weak, Potassic Moderate, and Potassic Strong).

The grade of the post mineral intrusive dykes was set to a default detection limit grade. Discretization was set to 4x, 4y, and 2z for all estimations.

The estimation parameters for copper are shown in Table 13.43, for gold in Table 13.44, for silver in Table 13.45, and for molybdenum in Table 13.46.

Table 13.43 Kemess East Estimation Parameters - Copper

Domain	Pass	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Bearing	Plunge	Dip	Major	Semi	Minor
Phyllic	1	4	12	6	159	-10	168	100	100	50
	2	4	10	4				175	200	100
Potassic Weak	1	4	12	6	73	-12	-152	100	100	60
	2	4	10	4				200	250	150
Potassic Moderate	1	4	12	6	125	-2	-160	100	100	50
	2	4	10	4				200	200	100
Potassic Strong	1	4	12	6	86	-5	-150	75	75	40
	2	4	10	4				150	150	75

Table 13.44 Kemess East Estimation Parameters - Gold

Domain	Pass	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Bearing	Plunge	Dip	Major	Semi	Minor
Phyllic	1	4	12	6	44	-29	-102	100	100	50
	2	4	10	4				250	150	150
Potassic Weak	1	4	12	6	73	-12	-152	100	100	60
	2	4	10	4				200	250	150
Potassic Moderate	1	4	12	6	125	-2	-160	100	100	50
	2	4	10	4				200	200	100
Potassic Strong	1	4	12	6	86	-5	-150	75	75	40
	2	4	10	4				150	150	75

Table 13.45 Kemess East Estimation Parameters - Silver

Domain	Pass	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Bearing	Plunge	Dip	Major	Semi	Minor
Phyllic	1	4	12	6	60	52	-115	100	100	50
	2	4	10	4				250	150	150

Domain	Pass	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Bearing	Plunge	Dip	Major	Semi	Minor
Potassic Weak	1	4	12	6	73	-12	-152	100	100	60
	2	4	10	4				200	250	150
Potassic Moderate	1	4	12	6	125	-2	-160	100	100	50
	2	4	10	4				200	200	100
Potassic Strong	1	4	12	6	86	-5	-150	75	75	40
	2	4	10	4				150	150	75

Table 13.46 Kemess East Estimation Parameters - Molybdenum

Domain	Pass	Composites			Search Rotation (Vulcan BPD)			Search Ranges (m)		
		Min	Max	Max / Hole	Bearing	Plunge	Dip	Major	Semi	Minor
Prop & Phyllic	1	4	12	6	165	20	-90	75	150	150
	2	4	10	4				150	300	300
	3	3	8	3				225	500	500
Potassic (All)	1	4	12	6	110	3	-170	75	100	75
	2	4	10	4				150	200	150
	3	3	8	3				225	250	200

13.4.10 Model Validation

Various measures have been implemented to validate the resultant resource block model. These measures include the following:

- Visual comparison of drill hole composites with resource block grade estimates by zone, both in plan and section;
- Statistical comparisons between block and composite data, and comparison to alternative interpolation method (ID2); and
- Swath plot analysis (drift analysis) comparing the OK model with the underlying composite data, the declustered composite data, and the alternative interpolation model (ID2).

The block model was initially validated visually by comparing block grades against drill hole composite data in cross-section and plan view for the individual domains. Figure 13.38 and Figure

13.39 are representative sections showing the visual comparison of block estimates with the composite data, for each variable.

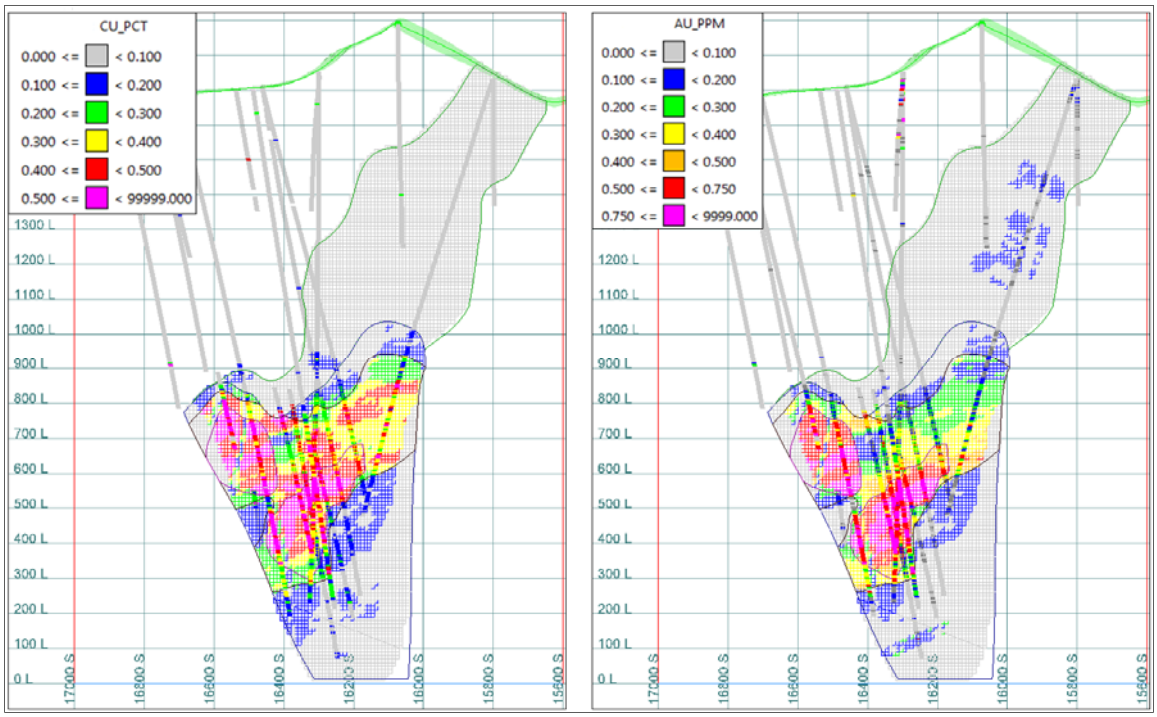


Figure 13.38 Kemess East Section 11,860E - Copper (Left) Gold (Right)

(Source: SRK, 2016) Note – Grid spacing on section is 100m Vertical and 200m Horizontal

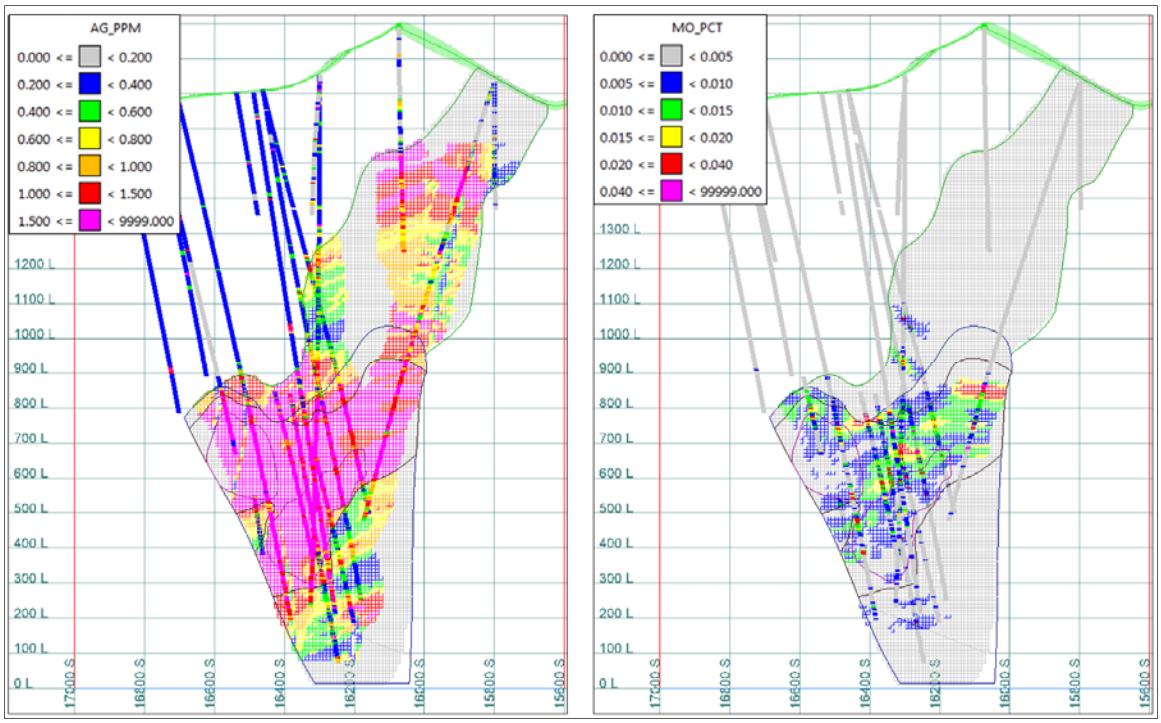


Figure 13.39 Kemess East Section 11,860E - Silver (Left) Molybdenum (Right)

(Source: SRK, 2016) Note – Grid spacing on section is 100m Vertical and 200m Horizontal

Global comparisons of the mean grade of the blocks against the mean grade of the un-declustered and declustered drillhole composites are outlined in Table 13.47.

Table 13.47 Kemess East Model Statistical Comparison

Domain	Variable	6.0 m Comps	NN Model	ID2 Model	OK Model
Phyllic	Cu %	0.035	0.031	0.030	0.030
	Au g/t	0.041	0.045	0.044	0.044
	Ag g/t	1.11	1.05	1.03	1.04
	Mo %	0.0007	0.0006	0.0005	0.0006
Potassic Weak	Cu %	0.158	0.132	0.137	0.137
	Au g/t	0.124	0.101	0.106	0.106
	Ag g/t	1.00	0.81	0.84	0.84
	Mo %	0.0053	0.0046	0.0048	0.0048
Potassic Moderate	Cu %	0.352	0.336	0.334	0.334
	Au g/t	0.306	0.288	0.285	0.284
	Ag g/t	1.90	1.87	1.88	1.88
	Mo %	0.0103	0.0095	0.0094	0.0093
Pot Strong	Cu %	0.470	0.456	0.454	0.453
	Au g/t	0.689	0.641	0.643	0.641
	Ag g/t	2.18	2.11	2.08	2.08
	Mo %	0.0065	0.0063	0.0061	0.0061

The model was imported into Snowden Supervisor software in order to statistically validate the model globally and by estimation domain. Swath plots were generated to validate the spatial trends within the model, as shown in Figure 13.40, Figure 13.41, Figure 13.42, and Figure 13.43 for copper, gold, silver, and molybdenum respectively. As part of the validation process, histogram comparisons, log probability plot comparisons, grade-tonnage comparisons, and QQ plot comparisons were reviewed.

It was concluded that the model does an adequate job of matching the data, and that no global bias is present, as the OK model generates the same tonnes and grade as the NN model at a zero cut-off, with no shift in the mean value on the histogram comparisons. Locally the estimations were reviewed by graphing the mean grade of the variable against the easting, northing and elevation for each of the domains (swath plots). As expected the mean grade of the composited samples is more variable than the mean grade of the blocks which are smoothed by kriging. The block averages in general bisect the sample plot indicating no bias between the model estimate and the samples. Larger variation between the means can be correlated with a reduction in the number of samples.

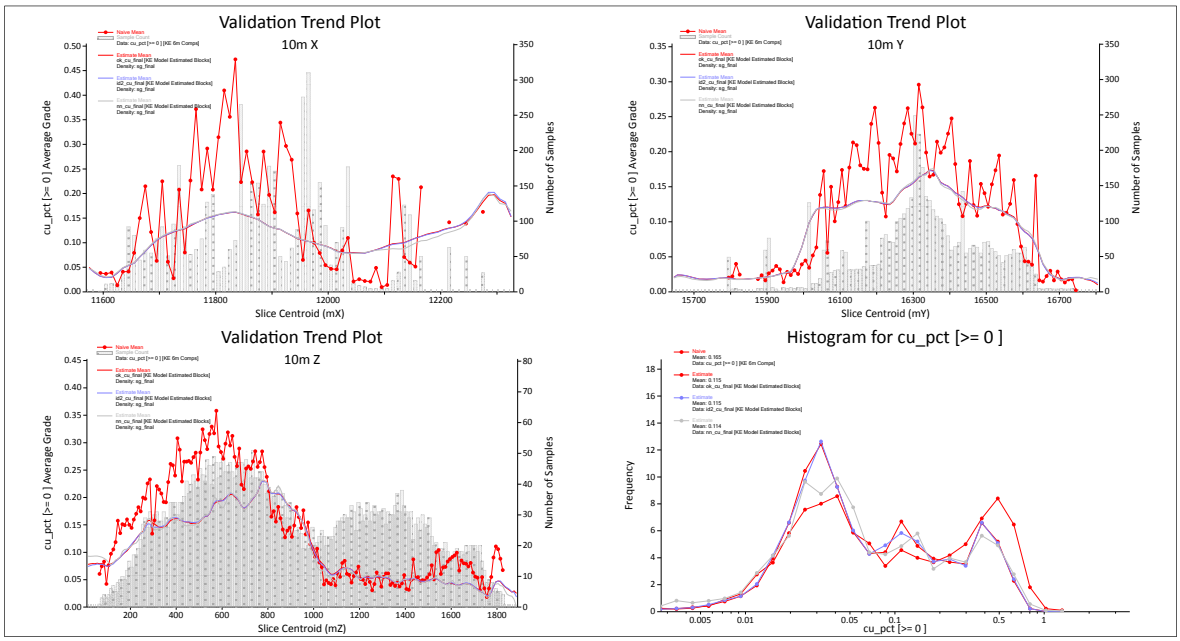


Figure 13.40 Kemess East Model Validation – Copper

(Source: SRK, 2016)

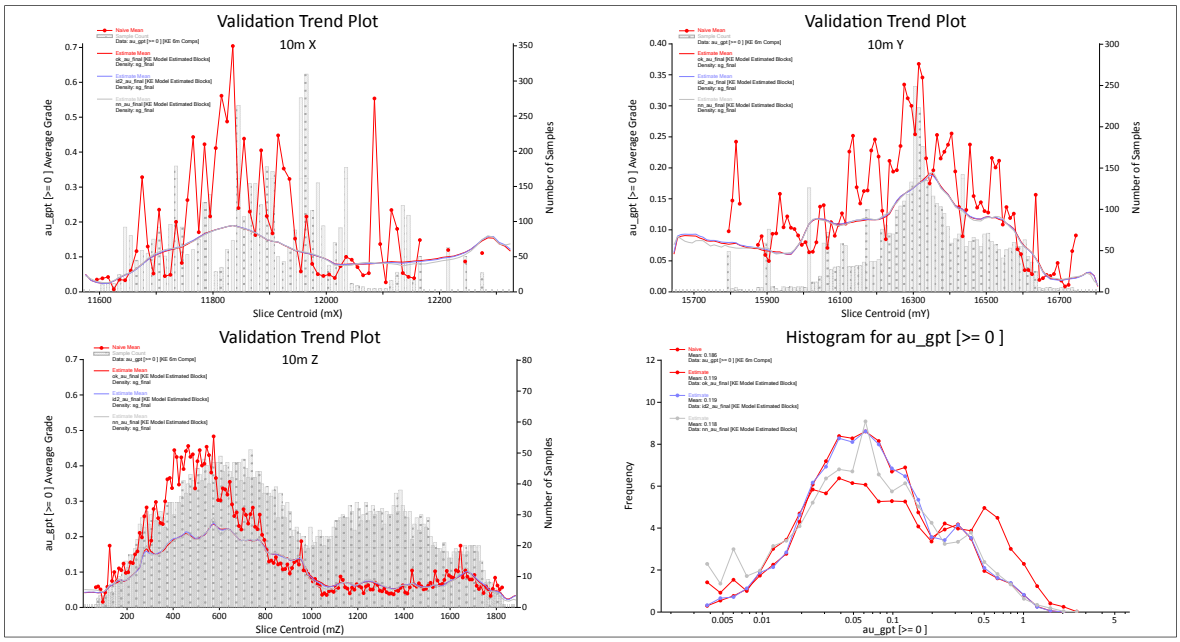


Figure 13.41 Kemess East Model Validation – Gold

(Source: SRK, 2016)

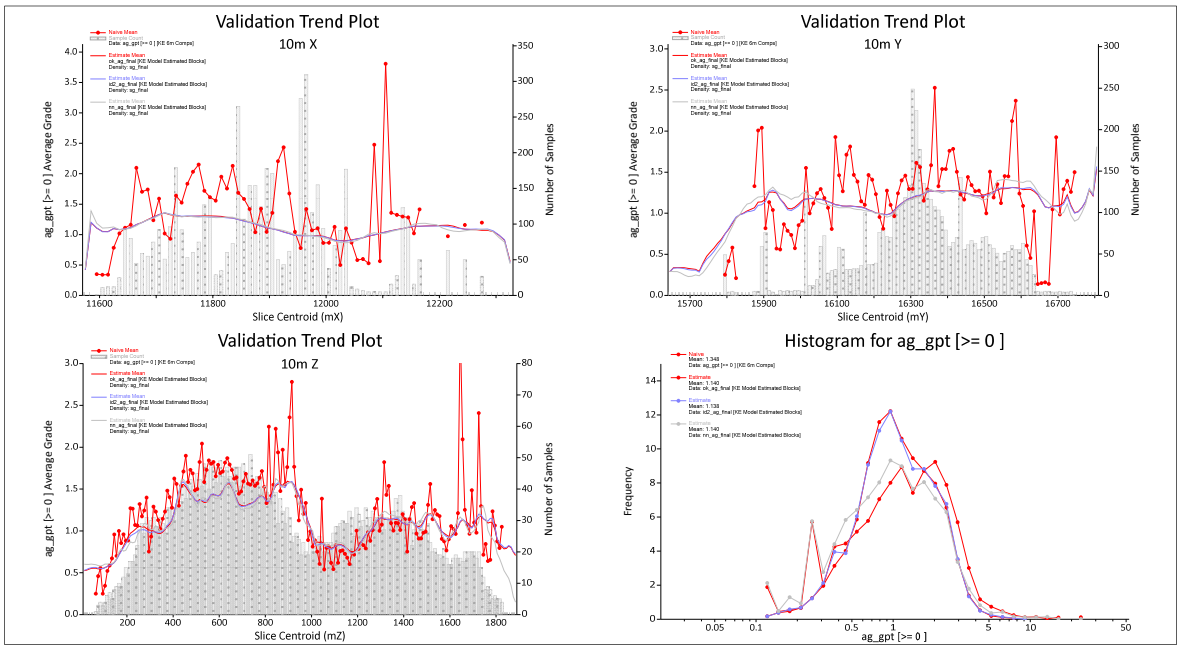


Figure 13.42 Kemess East Model Validation – Silver

(Source: SRK, 2016)

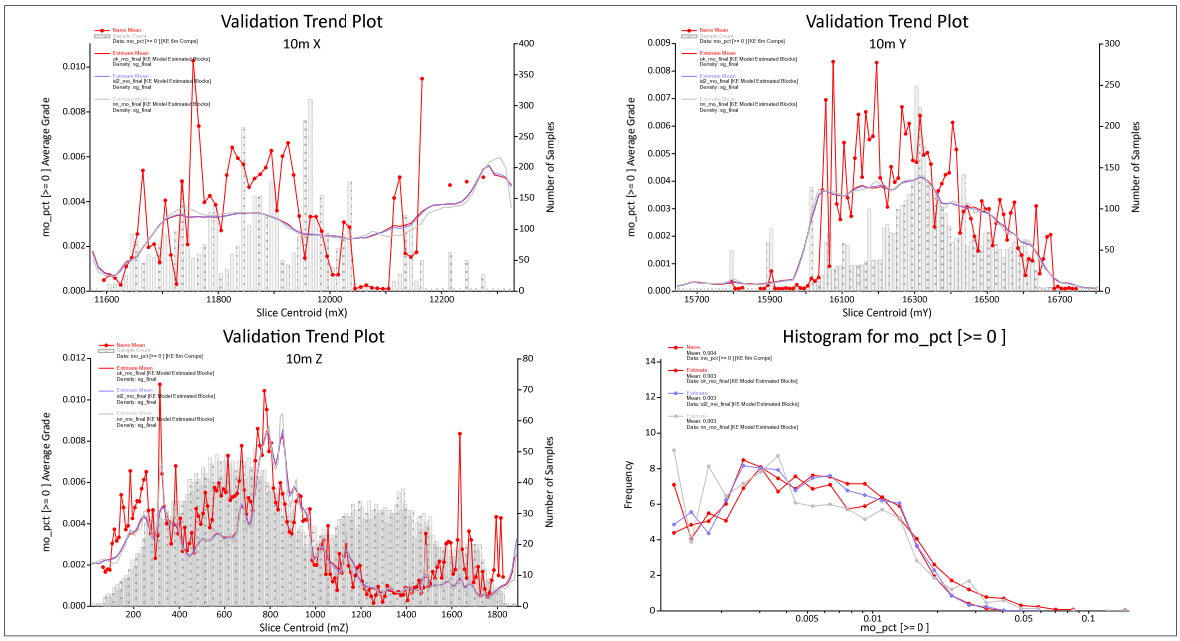


Figure 13.43 Kemess East Model Validation – Molybdenum

(Source: SRK, 2016)

13.4.11 Mineral Resource Classification

The Kemess East Resources are classified under the categories of Measured, Indicated, and Inferred according to standards as defined by the "CIM Definition Standards - for Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on May 10, 2014.

Classification of the resources reflects the relative confidence of the grade estimates. This is based on several factors including; sample spacing relative to geological and geostatistical observations regarding the continuity of mineralization, mining history, specific gravity determinations, accuracy of drill collar locations, accuracy of drillhole downhole survey data, quality of the assay data, and many other factors which influence the confidence of the mineral estimation.

The classification parameters are defined in relation to the number of drillholes used to estimate the block grades and the block-composite separation distance. These classification criteria are intended to encompass zones of reasonably continuous mineralization. At this stage no blocks have been classified as Measured.

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 entirely by core drilling on sections spaced at approximately 50 m to 80 m.

The classification of Indicated and Inferred Mineral Resources is based on geological confidence, drillhole spacing, and quality of the estimates. Porphyry copper deposits are characterised by large volumes of continuous grade over large distances radially distributed around and within the source intrusion. Post-mineral modification has offset the mineralization, i.e. Kemess East Offset Fault and dykes have removed mineralized volumes. Confidence in the estimation domains supports an Indicated classification. This classification has been applied to the estimation of gold, copper, silver, and molybdenum within the best-defined and highest grade domains.

The drill hole spacing of the Indicated resource is predominantly 75 m x 75 m between drill holes, with a small central area of approximately 50 m x 50 m spacing. Occasionally there is greater than 100 m spacing between drill holes at lower elevations due to down-hole deviation. The Inferred Mineral Resource classification has drill hole spacing ranging from 50 m x 100 m to 110 m x 130 m.

A post-process cleaning was conducted on the model classification to remove isolated indicated blocks surrounded by inferred, and isolated inferred blocks surrounded by indicated. A manual edit was performed by generating an indicated and inferred polygon for each level of the block model, and all blocks within the polygons were converted to their respective categories. This cleaning generates a contiguous volume of indicated and inferred blocks.

For Kemess East an additional processing of the resource classification was applied. As highlighted in Kemess East downhole survey (Section 9.2.4), 14 holes were identified to have lower confidence downhole survey data. Of these 14 holes only a handful (<5) are within the currently defined resource area of interest. These holes are currently based on manually adjusted downhole survey data. As such, SRK performed indicator estimation to estimate the relative influence these holes had on the block estimate. Any block inside the indicated polygon where greater than 70% of the overall estimate (copper and gold were assessed) was influenced by these holes, the block was downgraded to inferred. A review of the estimation domain volumes as related to these holes, was also conducted. SRK determined that the mineralized volumes were not materially impacted by these holes, as the holes were generally located internally to the mineralized volumes and surrounded by holes with higher confidence downhole survey data, indicating these drillholes were not resulting in extrapolation of the mineralized volumes.

13.4.12 Mineral Resource Sensitivity

In order to assess the sensitivity of the Kemess East resource to changes in NSR cut-off grade, SRK summarized tonnage and grade above cut-off at a series of increasing NSR cut-offs by resource category.

The sensitivity analysis for Indicated blocks within the Kemess East resource model is provided in Table 13.48. It can be observed that the resource is reasonably insensitive to NSR cut-off values in the increment from C$16.00/t NSR to C$18.00/t NSR, which is likely the NSR range of interest. The resource base cut-offs are shown in bold.

The sensitivity analysis for Inferred blocks within the Kemess East resource model is provided in Table 13.49. It shows the resource is reasonably insensitive to NSR cut-off values in the increment

between C$16.00/t NSR to C$18.00/t NSR, which is likely the NSR range of interest. The resource base cut-offs are shown in bold.

Grade tonnage curves are illustrated in Figure 13.44 and Figure 13.45 for Indicated and Inferred resources respectively. These tonnages may not all be classified as Indicated and Inferred resource as this is reliant on the economic cut-offs chosen and geometry considerations. At higher cut-off the uncertainty increases.

Table 13.48: Kemess East Indicated Mineral Resource Sensitivity

Cut-off Value (C$/t NSR)	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Contained Metal	
				Copper (000's lbs)	Gold (000's oz)
10.0	48,460	0.35	0.42	372,640	660
11.0	46,210	0.36	0.44	367,670	660
12.0	44,380	0.37	0.46	363,080	650
13.0	42,830	0.38	0.47	358,900	640
14.0	41,450	0.39	0.48	354,920	640
15.0	40,600	0.39	0.49	352,120	640
16.0	39,950	0.40	0.49	350,010	630
17.0	**39,270**	**0.40**	**0.50**	**347,550**	**630**
18.0	38,580	0.41	0.51	344,980	630
19.0	37,950	0.41	0.51	342,610	620
20.0	37,230	0.41	0.52	339,550	620

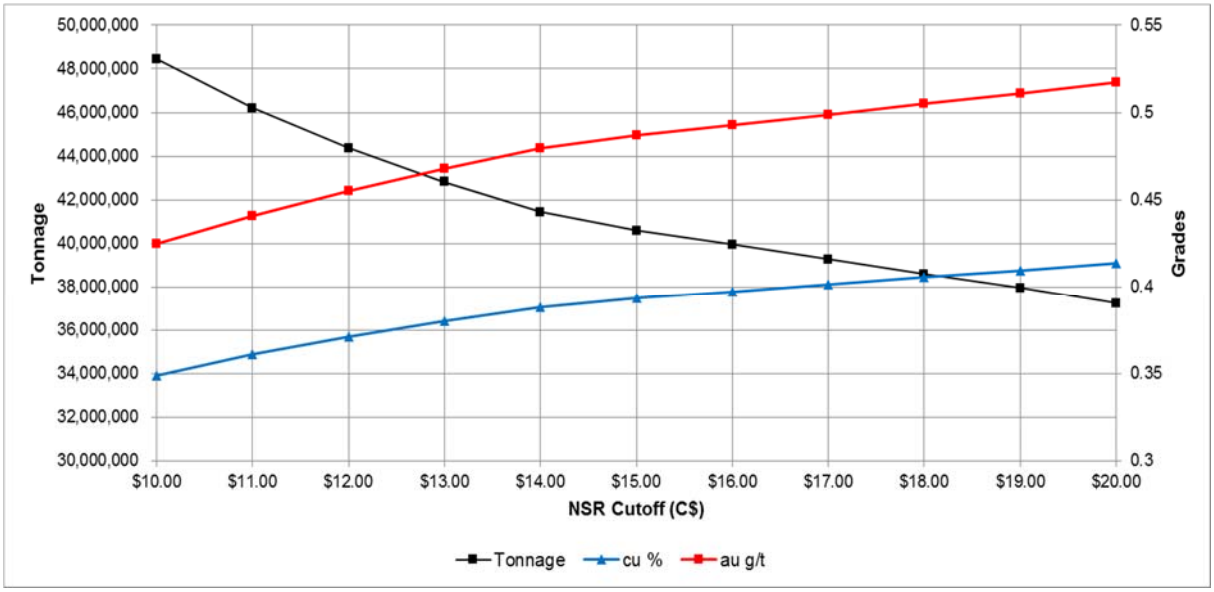


Figure 13.44: Kemess East Indicated Mineral Resource Sensitivity Plot

(Source: SRK, 2016)

Table 13.49: Kemess East Inferred Mineral Resource Sensitivity

Cut-off Value (C$/t NSR)	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Contained Metal	
				Copper (000's lbs)	Gold (000's oz)
10.0	138,360	0.32	0.32	970,910	1,440
11.0	133,160	0.33	0.33	958,780	1,430
12.0	127,390	0.34	0.34	944,760	1,410
13.0	122,230	0.35	0.35	931,280	1,390
14.0	118,310	0.35	0.36	919,930	1,370
15.0	114,750	0.36	0.37	908,700	1,360
16.0	111,730	0.36	0.38	898,330	1,350
17.0	**109,670**	**0.37**	**0.38**	**890,540**	**1,340**
18.0	107,920	0.37	0.38	884,110	1,330
19.0	106,370	0.37	0.39	877,980	1,320
20.0	104,410	0.38	0.39	869,630	1,310

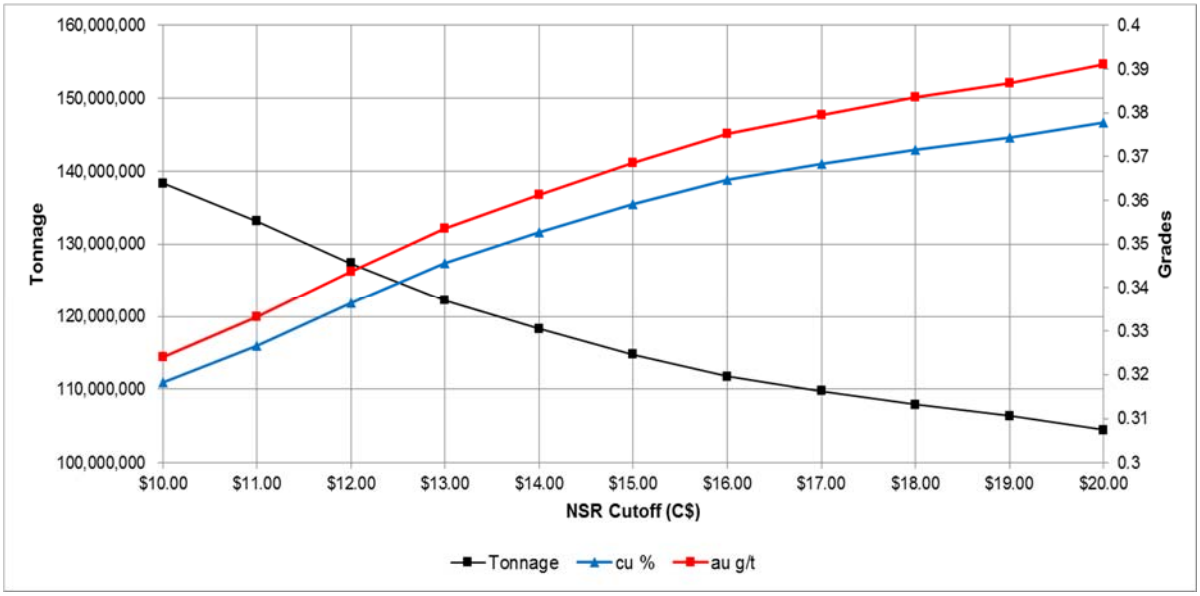


Figure 13.45 Kemess East Inferred Mineral Resource Sensitivity Plot

(Source: SRK, 2016)

13.4.13 Mineral Resource Statement

The Mineral Resources for the Kemess East (KE) copper-gold-silver-moly project, located in north-central British Columbia, have been estimated by SRK at 39,270 kt at an average grade of 0.40% Cu, 0.50 g/t Au, 1.99 g/t Ag, and 0.008% Mo classified as Indicated Mineral Resources; with an additional 109,670 kt at an average grade of 0.37% Cu, 0.38 g/t Au, 1.99 g/t Ag, and 0.009% Mo classified as Inferred Mineral Resources. The Mineral Resources are stated above a C$17.00/t NSR cut-off.

The Mineral Resources are reported in accordance with CSA, NI 43-101, and have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. The resource estimate was carried out by Chad Yuhasz, P.Geo, APEGBC, Senior Consultant (Resource Geology), SRK Consulting. Mr. Yuhasz has over 13 years of operational and consulting experience in the minerals industry, specifically in Mineral Resource estimation, production geology, feasibility studies, and economic evaluations. Mr. Yuhasz is a Certified Professional Geologist and a Qualified Person as defined by international reporting codes.

The effective date of this Mineral Resource estimate is 29 February, 2016 and is based on all data available. The Mineral Resource statement for the KE deposit is presented in Table 13.50. The resources for the Project are derived from the Vulcan block model using density values assigned as described in the estimation section. Mineral Resources are reported constrained to a reasonable prospects for economic extraction solid.

Table 13.50: Mineral Resource Statement, Kemess East Copper-Gold-Silver-Molybdenum Deposit, Northwest British Columbia, Canada, 29 February, 2016

Resource Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Moly Grade (% Mo)	Contained Metal			
						Copper (000's lbs)	Gold (000's oz)	Silver (000's oz)	Moly (000's lbs)
Measured									
Indicated	39,270	0.40	0.50	1.99	0.008	347,550	630	2,510	7,050
Measured + Indicated	**39,270**	**0.40**	**0.50**	**1.99**	**0.008**	**347,550**	**630**	**2,510**	**7,050**
Inferred	109,670	0.37	0.38	1.99	0.009	890,540	1,340	7,010	20,860

Notes:

1. Resources stated are contained within a reasonable prospects for economic extraction solid above C$17.00/t NSR cut-off. A variable specific gravity value was assigned by domains for all model blocks.
2. NSR calculation is based on assumed copper, gold and silver prices of US$3.20/lb, US$1,275/oz and US$21.00/oz, respectively. Molybdenum was not assigned any value in the NSR calculation.
3. Mineral Resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.
4. Contained metals are in situ and undiluted, and do not include metallurgical recovery losses.
5. 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 Resources estimated will be converted into Mineral Reserves.

The “reasonable prospects for eventual economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off value, taking into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considers that portions of the Kemess East resource are amenable for underground extraction via panel cave mining development.

The 2016 Mineral Resource estimate has been prepared by identifying the portion of the resource model that could potentially be mined using panel cave mining methods, treating indicated and inferred blocks as potentially economic. The study identified a volume potentially suitable for block/panel cave mining at a Net Smelter Return (NSR) cut-off value of C$17.00/t.

The cost parameters were selected based on experience and benchmarking against the Kemess North Project. The reader is cautioned that the results from the cave optimization are used solely for the purpose of testing the “reasonable prospects for eventual economic extraction” by a panel cave and do not represent an attempt to estimate Mineral Reserves. The commodity prices (Table 13.51) are used to assist in the preparation of a Mineral Resource estimate and to select an appropriate resource reporting cut-off NSR value. Although molybdenum was modeled, it was not assigned value in the NSR calculation due to lack of metallurgical testing.

Table 13.51: Kemess East Commodity Prices, and Recovery Assumptions, used to Calculate NSR Values and Reasonable Cut-off

Commodity	Value	Unit
Copper	3.20	US$ per pound
Gold	1,275.00	US$ per ounce
Silver	21.00	US$ per ounce
Molybdenum	8.36	US$ per pound (Not used)
Copper Recovery	91%	Assumed Recovery
Gold Recovery	72%	Assumed Recovery
Silver Recovery	65%	Assumed Recovery
Molybdenum Recovery	NA	Not Applicable

13.4.14 Previous Mineral Resource Estimates

The previous mineral resource disclosure on the Kemess East deposit is shown below in Table 13.52. The previous mineral resources are stated at a C$0.00/t NSR cut-off, inside potentially economically mineable cave solids, designed at a C$15.00/t NSR. In general, there are no material changes to the updated mineral resource.

Table 13.52 Kemess East Mineral Resource Statement, Copper-Gold-Silver-Moly deposit, Northwest British Columbia, Canada, December 31, 2014

Resource Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Moly Grade (% Mo)	Contained Metal			
						Copper (000's lbs)	Gold (000's oz)	Silver (000's oz)	Moly (000's lbs)
Measured									
Indicated	55,864	0.41	0.52	2.01	0.01	503,663	939	3,601	9,261
Measured + Indicated	**55,864**	**0.41**	**0.52**	**2.01**	**0.01**	**503,663**	**939**	**3,601**	**9,261**
Inferred	117,152	0.34	0.38	1.79	0.01	871,407	3,360	13,723	33,662

Notes:

1. 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 resources estimated will be converted into mineral reserves.
2. Resources stated as contained, above a C$0.00/t NSR cut-off within potentially economically mineable cave solids designed at C$15.00/t NSR cut-off.
3. NSR calculation is based on assumed copper, gold and silver prices of US$3.00/lb, US$1,300/oz and US$20.00/oz, respectively.
4. Mineral resource tonnage and contained metal have been rounded to reflect the accuracy of the estimate; numbers may not add due to rounding.
5. Mineral resources are exclusive of mineral reserves.

Contained metals are in situ and undiluted, and do not include metallurgical recovery losses.

14 Mineral Reserve Estimates

The Mineral Reserve estimate has been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM Standards on Mineral Resources and Mineral Reserves and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

The Mineral Reserve estimate was developed from the resource model "SRK_KUG_FS2015_ENG.bmf" which was prepared by SRK Consulting in September 2015.

The Mineral Reserve statement for the Kemess Underground (KUG) project is presented in Table 14.1.

Table 14.1: Mineral Reserve Statement, Kemess Underground Gold-Copper-Silver Deposit Project, 29 February, 2016

Reserve Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Contained Metal		
					Copper (000's lb)	Gold (000's oz)	Silver (000's oz)
Proven	0	0.00	0.00	0.00	0	0	0
Probable	107,310	0.27	0.54	1.99	629,595	1,868	6,878
Proven and Probable	**107,310**	**0.27**	**0.54**	**1.99**	**629,595**	**1,868**	**6,878**

Notes:

1. Estimated at US$2.50/lb Cu, US$1,200/oz Au and US$16.00/oz Ag, using an exchange rate of 0.75 US$:C$, cut-off NSR Value of C$15.30/t and a shut-off NSR Value of C$17.30/t of ore. Metallurgical recoveries and other parameters are shown in Table 14.2.
2. Mineral Reserve tonnage and mined metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

14.1 CIM Definitions & Guidelines for Mineral Reserves

14.1.1 Mineral Reserve

Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve.

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

Mineral Reserves are those parts of Mineral Resources which, after the application of all mining factors, result in an estimated tonnage and grade which, in the opinion of the Qualified Person(s) making the estimates, is the basis of an economically viable project after taking account of all relevant processing, metallurgical, economic, marketing, legal, environment, socio-economic, and

government factors. Mineral Reserves are inclusive of diluting material that will be mined in conjunction with the Mineral Reserves and delivered to the treatment plant or equivalent facility. The term 'Mineral Reserve' need not necessarily signify that extraction facilities are in place or operative or that all governmental approvals have been received. It does signify that there are reasonable expectations of such approvals.

14.1.2 Probable Mineral Reserve

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

14.1.3 Proven Mineral Reserve

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect potential economic viability.

14.2 Sources of Information

The Mineral Reserve has been estimated using the resource block model prepared by Chad Yuhasz, P.Geo., an employee of SRK Consulting. Mr. Yuhasz is also the QP for the KUG Mineral Resource estimate.

SRK prepared an estimate of the NSR value for each block in the resource model. This was used to drive the production schedule. The NSR calculation estimates the theoretical value received for concentrate produced from each tonne of material mined, net of all costs and losses incurred in the off-site transportation and processing of concentrate.

AuRico provided the following inputs:

- Geological interpretations of lithological domains, structures and faults;
- Process plant recoveries, concentrate grades, concentrate transportation and treatment charges; and
- G&A costs.

SRK provided the mining cost estimate, commodity prices and exchange rate. KWM provided the processing cost estimate.

SRK used the software program PCBC™ to prepare the caving production schedule, which included provision for mixing of various mineralized and unmineralized materials within the cave as mining progresses.

14.3 Net Smelter Return Model

To enable a single cut-off parameter to be used when defining the economic portion of the resource to be mined, SRK estimated the value (NSR value) received for concentrate produced from each tonne of resource, net of all costs and losses incurred in the off-site transportation and processing of concentrate.

Metal prices, metallurgical recoveries, transportation and treatment charges, and other values used in the NSR calculation are presented in Table 14.2. It should be noted that metal prices and concentrate transport unit cost in Table 14.2 were not used for base case economic analysis of the KUG project.

Table 14.2: Basis for the NSR estimate

NSR Parameter	Units	Value
Exchange rate	US$ per C$	0.75
Copper price	US$/lb	2.50
Gold price	US$/oz	1,200
Silver Price	US$/oz	16.00
Metallurgical recovery – Copper	%	91
Metallurgical recovery – Gold	%	72
Metallurgical recovery – Silver	%	65
Copper Concentrate Payables – Copper	%	96.5
Copper Concentrate Payables – Gold	%	97.5
Copper Concentrate Payables – Silver	%	90.0
Copper Concentrate Deductions – Copper	%	1.0
Copper Concentrate Deductions – Gold	%	NIL

NSR Parameter	Units	Value
Copper Concentrate Deductions – Silver	%	NIL
Concentrate moisture content	%	8
Concentrate grade	%	22
Concentrate transport	US$/t (wet)	214.97
Concentrate treatment	US$/t (wet)	80
Cu metal refining charge	US$/lb Cu	0.08
Metal refining charge – Gold	US$/oz Au	6.00
Metal refining charge – Silver	US$/oz Ag	0.40

14.4 Initial Selection of Mining Outlines

The panel caving method requires the orebody to be undercut, enabling the ore to collapse and fragment (cave) into underlying drawpoints, where it is extracted. The shape and size of the area to be undercut (the footprint) was determined by using PCBC™ to evaluate the economics of individual drawpoints based on a nominated cost of establishing each drawpoint (C$376,000) with the recoverable NSR value of the overlying column of ore, after deducting mining, processing, and all other onsite costs. A nominal cost of C$15.30/t was used for this purpose. Table 14.3 shows the PCBC mixing parameters and Table 14.4 shows details for the production rate curve used by PCBC for the mineral reserve estimate.

Table 14.3: PCBC Mixing Parameters

Mixing Parameter	Value	Description
MIX_HORIZON	150	The degree of mixing increases with this value
CYCLES	3	More cycles result in increases mixing.
BASE_FROZEN	0.5	Fraction of base slice that is unable to be extracted. This increases mixing as the material is slowly released due to erosion.
HT_FROZEN	180	The fraction “frozen” in each slice decreases linearly from the BASE_FROZEN fraction to zero at the specified height.
EROSION	0.03	The “frozen” material within a slice that is released as a proportion of the material moving through the slice.

Table 14.4: Production Rate Curve

% Extraction	t/m²/day	t/day	mm/day
0	0.10	23	37
10	0.25	56	93
20	0.35	79	130
30	0.46	104	170
100	0.46	104	170

The footprint was designed to encompass those drawpoints with a positive net value. Further details on the footprint are provided in Section 15.6.

The target draw heights for the ore columns were determined by applying a cut-off value (C$15.30/t) to the ore within the draw column. A drawpoint is closed once it is mined to the height that yields the maximum NPV according to PCBC™ which takes into account the operating margin (contained NSR value – the cut-off value) and drawpoint production rates. However, as the production schedule was limited to 110 Mt due to a constraint on the available storage space for tailings in the existing Kemess South open pit, the shut-off the was not a driving factor in determining the optimal KUG cave volume.

14.5 Adjustment for Ore Loss

In SRK's opinion, mining losses will occur in addition to those simulated by the PCBC™ software, in particular the following:

- Over the life of the project there will be deviations from the draw control rules used in the simulations that will result in waste entering the draw earlier than expected;
- Other than in the best ground conditions, it is expected that some drawpoints will become damaged beyond economic repair before reaching their shut-off grade; and
- Towards the end of the mine life there will be some drawpoints that have not reached their shut-off grade, but the total number of drawpoints in this condition will be insufficient to justify continued operation of the mine.

In SRK's opinion, the impact of these factors will be minimal as the available quantity of mineralised material above the C$15.30/t cut-off is greater than the 110 Mt mine plan constraint (imposed by tailings deposition capacity and independent of cave mining economics). In other words, losses incurred as a consequence of these risks can be generally offset by drawing additional ore (>C$15.30/t) from the remaining drawpoints.

14.6 Development Ore

A small quantity of ore, not included in the panel cave modeling work, will be mined during development. This material is included in the overall KUG reserve estimate.

The estimated quantity of development ore is shown in Table 14.5.

Table 14.5: Summary of KUG Development Ore

Ore Source	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	NSR Value (C$/t)
Development	1,551	0.21	0.36	1.49	25.1

14.7 Total Ore Mined

The total quantity of ore mined from development and from the panel cave drawpoints is shown in Table 14.6.

Table 14.6: Summary of KUG Total Ore Mined

Ore Source	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	NSR Value (C$/t)
Development	1,551	0.21	0.36	1.49	25.1
Production					
Undercut	3,325	0.28	0.50	1.81	34.6
Drawbell	1,723	0.24	0.43	1.62	29.5
Caving	100,783	0.27	0.55	2.01	35.3
Total	**107,381**	**0.27**	**0.54**	**1.99**	**35.1**

Source: KUG 2015 T&G by activity physicals.xlsx, received 20160120

14.8 Mineral Reserve Classification

The Indicated and Inferred resources lying within the vertical projection of the KUG cave footprint that are expected to cave are shown in Table 14.7 . Cave footprints for the resource and reserve differ, with the resource footprint being larger than that of the reserve.

Table 14.7: Mineral Resources within the Vertical Projection of the KUG Cave Footprint

Resource Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Contained Metal		
					Copper (000's lbs)	Gold (000's oz)	Silver (000's oz)
Measured							
Indicated	246,400	0.22	0.42	1.75	1,195,300	3,328	13,866
Measured + Indicated	**246,400**	**0.22**	**0.42**	**1.75**	**1,195,300**	**3,328**	**13,866**
Inferred	21,600	0.22	0.40	1.70	104,700	277	1,179

Note: Totals do not equal the sum of the components because of rounding adjustments.

As there are no Measured Resources in the mine plan, a Probable Reserve classification has been applied to the reserve.

The Inferred Mineral Resources were not included in the Mineral Reserve estimate.

The above Mineral Resource estimate (presented in Section 3) is inclusive of the Mineral Reserve estimate presented in Section 4. Due to the complexity of block cave scheduling SRK is of the opinion that reporting resources inclusive of the reserves is a more accurate approach for reporting.

14.9 Mineral Reserve Estimate

The Mineral Reserve statement for the Kemess Underground Project is presented in Table 14.8. The Reserve has been estimated using the resource block model prepared by SRK Consulting (September 2015), the contents of which form the basis of the Mineral Resource Estimate summarised in Table 14.7.

The entire Mineral Reserve is included in the Mineral Resource estimate.

The reserve has been reported using the CIM Definition Standards on Mineral Resources and Mineral Reserves Definitions and Guidelines.

Table 14.8: Mineral Reserve Statement, Kemess Underground Gold-Copper-Silver Deposit, Northwest British Columbia, Canada, 29 February, 2016

Reserve Category	Quantity (000's t)	Copper Grade (% Cu)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)	Contained Metal		
					Copper (000's lb)	Gold (000's oz)	Silver (000's oz)
Proven	0	0.00	0.00	0.00	0	0	0
Probable	107,310	0.27	0.54	1.99	629,595	1,868	6,878
Proven and Probable	**107,310**	**0.27**	**0.54**	**1.99**	**629,595**	**1,868**	**6,878**

Notes:

1. Estimated at US$2.50/lb Cu, US$1,200/oz Au and US$16.00./oz Ag, using an exchange rate of 0.75 US$:C$, cut-off NSR Value of C$15.30/t and a shut-off NSR Value of C$17.30/t of ore. Metallurgical recoveries and other parameters are shown in Table 14.2.
2. Mineral Reserve tonnage and mined metal have been rounded to reflect the accuracy of the estimate, and numbers may not add due to rounding.

15 Mining Methods

15.1 Summary

The mine design and mine development schedule was prepared by AuRico and was reviewed by SRK Consulting. SRK used the PCBCTM modelling software to generate a cave production schedule and provided Mineral Reserve estimate based on the block caving method.

The opportunity to exploit the Kemess Underground (KUG) gold-copper-silver deposit by underground mining methods was evaluated by a preliminary economic analysis (PEA) completed by AMC Mining Consultants in July 2011 (AMC, 2011). This PEA considered longhole open stoping, sub-level caving and block cave mining methods, with the block cave approach recommended due to superior economics. As a result, AuRico (then Northgate Minerals) elected to conduct a Feasibility Study (FS) for the Kemess Underground (KUG) deposit based on block caving. The initial FS was completed in 2013 (SRK, 2013).

The KUG mine will be located approximately 6.5 km north of the existing Kemess South site. This gold-copper-silver porphyry deposit hosts a probable Mineral Reserve of 107.3 Mt within a selected PCBC™ cave volume that has footprint dimensions of approximately 570 m east-west and 90 m to 300 m north-south. Mineralization extends from approximately 200 m to 550 m below surface. The typical range of column heights and NSR values is shown in Figure 15.1.

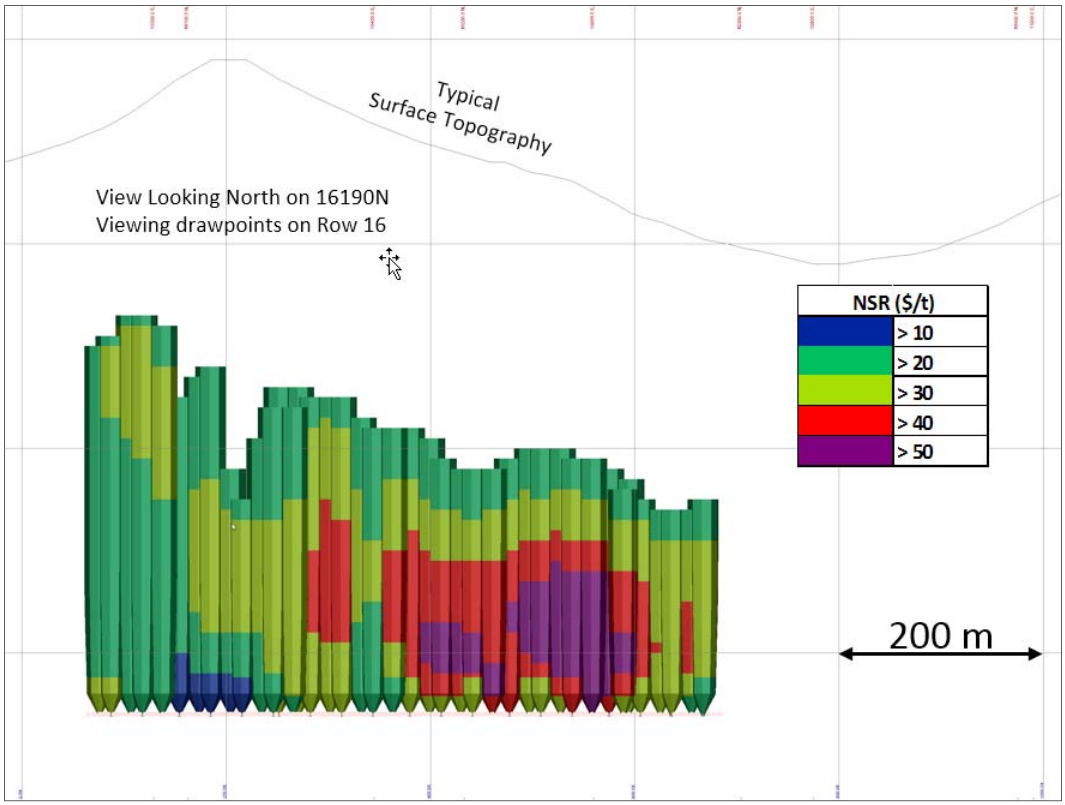


Figure 15.1: Typical Range of Panel Cave Column Heights
(Source: SRK, 2016)

AMC (2011) evaluated several potential mining methods and recommended cave mining as the mining method of choice, primarily because of its superior economics. Figure 15.2 presents an overall view of the KUG mine layout.

The KUG panel cave mine was designed using Gemcom's PCBC™ software. The planned production schedule calls for 107.3 Mt of ore mined at a head grade of 0.27% Cu and 0.54 g/t Au and 1.99 g/t Ag.

Geotechnical information was reviewed; cavability and fragmentation parameters established based on empirical observations and Itasca Consulting Group Inc.'s (Itasca) numerical modeling, with this work carried out as part of the 2013 KUG FS (SRK, 2013).

Triple declines will be developed from surface comprising access, intake air, and conveyor declines. The access decline will provide access for personnel, equipment, and materials/consumables.

The final design establishes a single extraction level at the 1140 mRL (Figure 15.3) that includes 582 drawpoints (291 drawbells) over a footprint that is approximately 570 m east-west and 90 m to 300 m north-south. The cave will be initiated in the highest grade ore in the east of the orebody and progress to the southwest over the life of the mine. An 18 m high undercut will be established above the undercut level, with this level at the 1158 mRL and accessed via ramps from the extraction level.

Ore will be recovered on the extraction level using 8.6 m³ (17 t) load-haul-dump (LHD) machines and dumped into one of four 55" x 42" jaw crushers located to the south of the footprint. Transfer conveyors will then transport ore to the main underground conveyor.

The main underground conveyor, 3.2 km long and rising 305 m vertically, will transport ore from the transfer conveyors and transfer ore to a 4.9 km surface conveyor that will drop 98 m vertically overall. A total of 9 Mt per year (Mt/y) of ore will be discharged onto the existing crushed ore stockpile located at the existing Kemess South process plant site.

Infrastructure required for the mine will include a 6.0 m wide x 5.0 m high intake air decline and 5 m diameter exhaust air raise and associated fan pairs, with each pair capable of delivering approximately 400 m³/sec. The access and conveyor declines will have a dedicated ventilation system separate from the main mine ventilation system for safety reasons.

Other mine infrastructure will include two dewatering pumps, each capable of pumping 88 L/s. Only one pump will be used at any one time with the other as back-up.

There will be an underground maintenance facility to service and maintain the underground mobile equipment. Major equipment rebuilds will be done at the surface workshop that was previously used for the open pit. Table 15.1 presents a complete list of underground infrastructure.

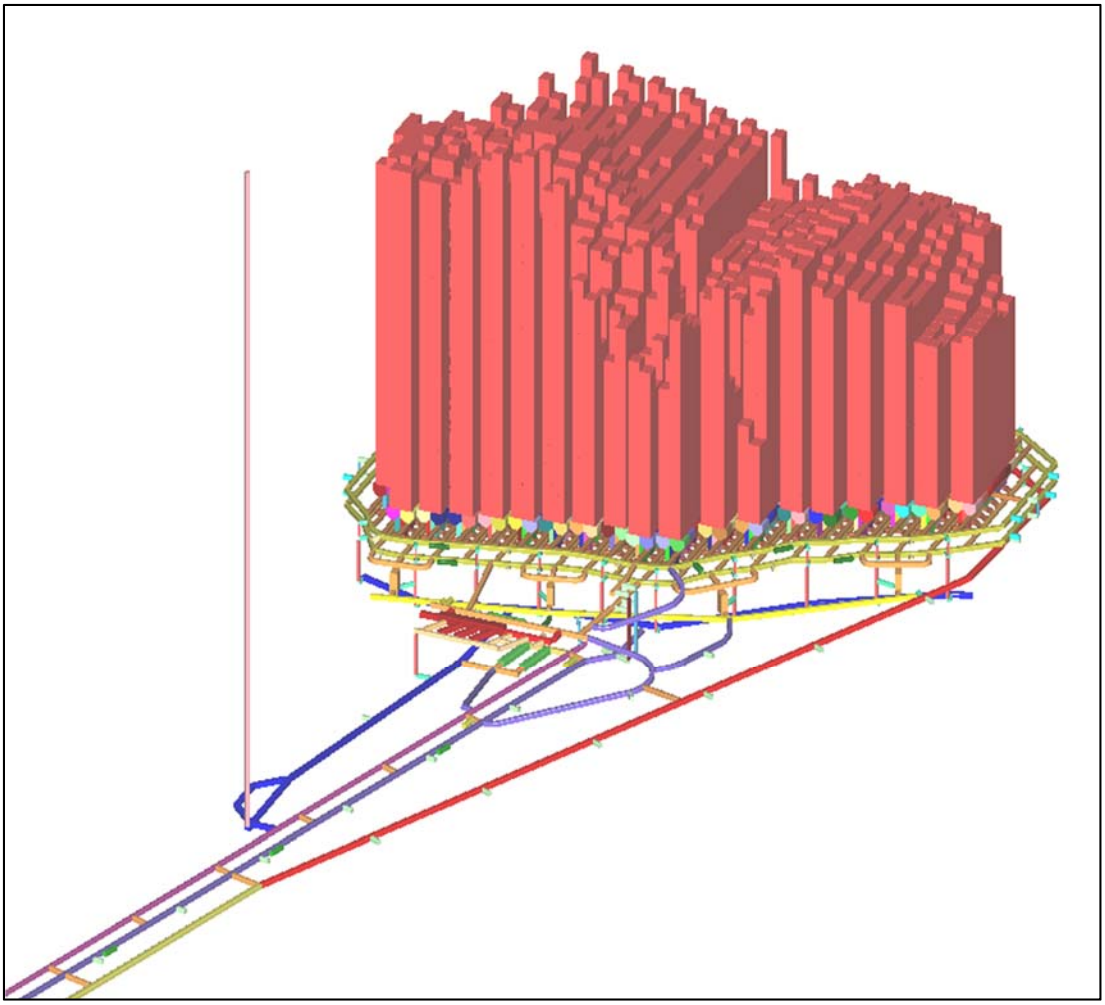


Figure 15.2: Isometric View of Panel Cave (Looking Northwest)

(Source: SRK, 2016)

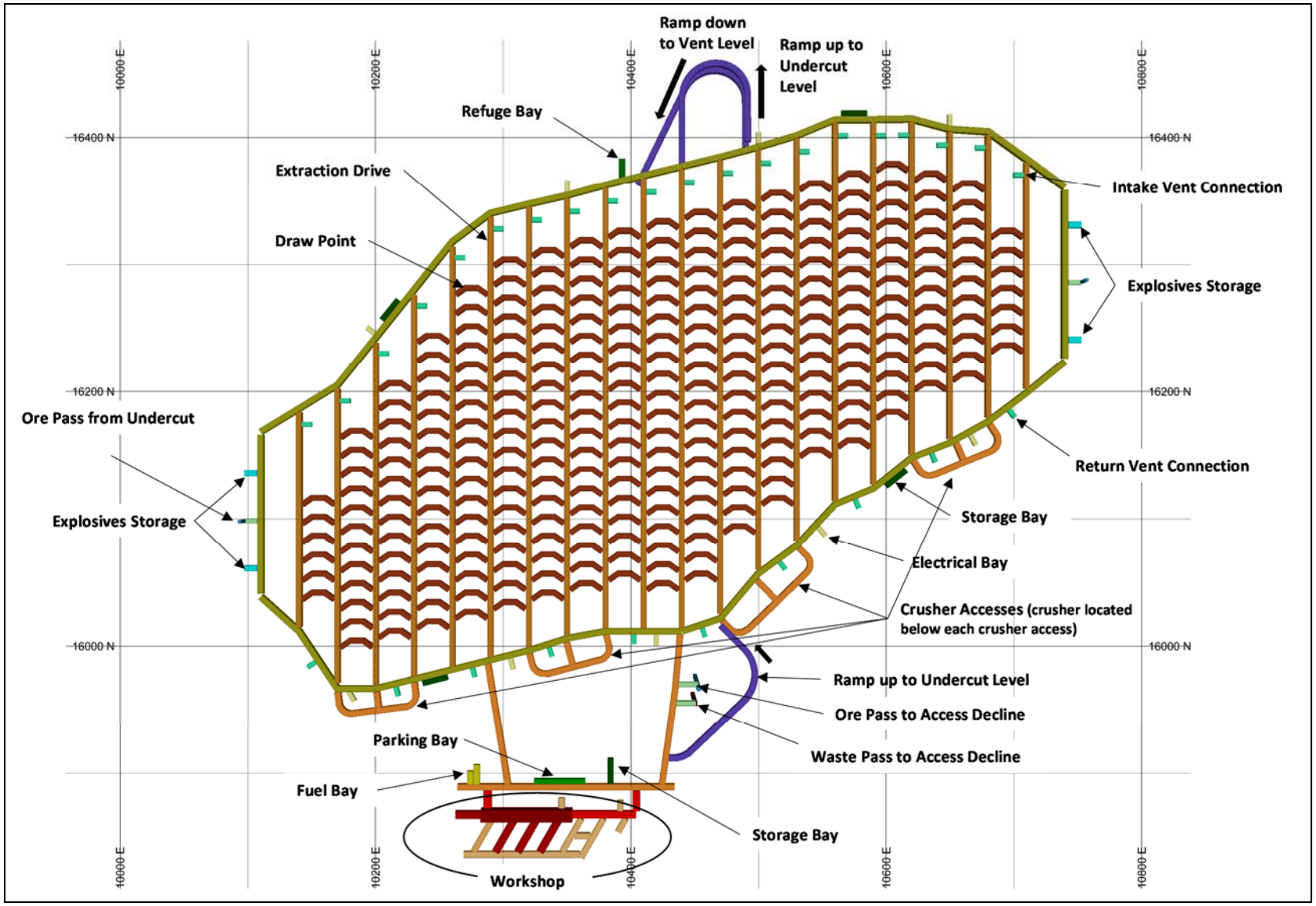


Figure 15.3: Extraction Level Layout
(Source: AuRico, 2016)

Table 15.1: Underground Infrastructure

Item	Description
Primary Ventilation	Two main intake fans and two main exhaust fans, each capable of approx. 200 m³/s at 2.4 kPa to 2.7 kPa.
Conveyor	3,200 m long x 1,067 mm wide, speed of 2.9 m/s.
Crushers	Four 55 " x 42 " jaw crushers delivering a nominal 80% minus 165 mm product.
Maintenance	Workshop capable of accommodating ten pieces of equipment c/w 20 t crane, wash bay, drill shop, offices, stores and refuge station.
U/G Storage and Warehousing	Eight storage bays, one fuel bay, and four sets of explosive magazines.
Mine Dewatering	Two pumps, each capable of 88 L/s.
Electrical Power Supply Distribution:	Two main 25.0 kV power cables down declines to main substation.
Lunchroom and Refuge Stations	Three each.

Table 15.2 shows the annual lateral development and raise requirements to fully develop the KUG mine, totaling 47,751 m lateral development and 2,196 m raise development.

Table 15.3 shows the annual KUG mine production with first ore produced in Year -2 and full production achieved in Year 3, with full production occurring over ten years. Total ore mined is 107.3 Mt at 0.27% Cu, 0.54 g/t Au, and 1.99 g/t Ag for 285.7 kt Cu, 1,868 koz Au, and 6,878 koz Ag.

Table 15.2: Lateral Development Schedule

Item	Units	Total	Year -4	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Total Lateral Development	m	47,751	1,347	5,510	7,331	8,503	7,214	6,343	4,582	4,420	2,501	0
Total Raise Development	m	2,196	0	0	890	469	228	202	126	142	139	0

Table 15.3: Production Schedule

Name	Units	Total	Year -4	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
Undercut Ore	t	3,324,864	0	0	0	398,694	773,027	657,363	501,346	530,017	464,417	0	0	0	0	0	0	0	0
Drawbell Ore	t	1,722,946	0	0	0	192,598	410,350	334,652	262,075	279,986	243,286	0	0	0	0	0	0	0	0
Development Ore	t	1,550,802	0	0	223,802	377,953	244,737	237,634	163,053	165,457	138,166	0	0	0	0	0	0	0	0
Cave Ore	t	100,782,885	0	0	0	66,169	2,009,623	4,610,044	8,093,200	8,032,121	8,155,313	9,015,093	9,011,443	9,011,075	9,003,991	9,014,949	8,999,986	9,021,130	6,738,749
TOTAL Ore	t	**107,381,498**	**0**	**0**	**223,802**	**1,035,413**	**3,437,736**	**5,839,693**	**9,019,675**	**9,007,581**	**9,001,181**	**9,015,093**	**9,011,443**	**9,011,075**	**9,003,991**	**9,014,949**	**8,999,986**	**9,021,130**	**6,738,749**
TOTAL Waste	t	**2,975,734**	**91,018**	**371,232**	**324,822**	**336,812**	**592,749**	**582,021**	**309,779**	**352,595**	**14,705**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
TOTAL Mine	t	**110,357,232**	**91,018**	**371,232**	**548,625**	**1,372,226**	**4,030,485**	**6,421,714**	**9,329,455**	**9,360,176**	**9,015,886**	**9,015,093**	**9,011,443**	**9,011,075**	**9,003,991**	**9,014,949**	**8,999,986**	**9,021,130**	**6,738,749**
Cu Grade	% Cu	0.27	0.00	0.00	0.14	0.26	0.34	0.34	0.34	0.32	0.29	0.26	0.25	0.25	0.25	0.24	0.23	0.21	0.20
Cu	t Cu	285,660	0	0	319	2,701	11,632	19,842	30,634	29,021	26,250	23,823	22,703	22,387	22,112	21,449	20,300	19,126	13,363
Au Grade	g/t Au	0.54	0.00	0.00	0.22	0.46	0.66	0.73	0.76	0.70	0.61	0.55	0.51	0.48	0.47	0.46	0.43	0.40	0.38
Au	oz Au	1,867,859	0	0	1,568	15,312	72,879	137,109	220,437	202,271	176,785	158,444	148,229	140,436	136,271	133,306	125,684	117,394	81,732
Ag Grade	g/t Ag	1.99	0.00	0.00	1.26	1.77	2.10	2.25	2.30	2.30	2.15	2.05	1.99	1.96	1.94	1.88	1.79	1.69	1.59
Ag	oz Ag	6,878,658	0	0	9,055	58,937	231,930	422,864	666,481	667,084	622,465	595,442	575,998	569,043	560,905	544,918	517,633	490,921	344,982
Ore NSR	C$/t	35.1	0.0	0.0	16.4	31.9	43.4	46.0	47.0	43.9	39.0	35.2	33.2	32.1	31.4	30.6	28.9	27.1	25.3

Figure 15.4 shows ore NSR (C$/t terms) by quarter. Lower values in early quarters are associated with initial development to the south of the footprint. However, as development and related undercutting and drawbell opening pushes to the east of the footprint, average ore NSR values increase rapidly to peak at almost C$48/t in Q18 of ore production.



Figure 15.4: Ore NSR by Quarter

(Source: SRK, 2016)

15.2 Mine Geotechnics

15.2.1 Geotechnical

Geotechnical assessments, particularly caveability and fragmentation, are of upmost importance when determining the suitability of a rock mass for cave mining. SRK (Vancouver office) personnel were engaged to carry out this assessment due to their experience and expertise related to cave mining. SRK`s scope included reviewing of existing geotechnical reports and data, collecting and reviewing additional geotechnical data, providing geomechanical mine design parameters, caveability assessment, fragmentation assessment, and ground support recommendations. The following sub-sections summarise this scope.

15.2.2 Geotechnical Assessment

Data used for geotechnical assessment included 2011 data collected by SRK (seven oriented core geotechnical boreholes), 2010 data collected by AMC (29 geotechnical boreholes, 26 oriented cores), and 2003 data collected by Knight Piesold (10 oriented core geotechnical boreholes). The seven 2011 geotechnical boreholes, which were drilled and logged under SRK supervision, comprised three boreholes in the cave footprint area and four boreholes along the planned triple decline alignment; with the three footprint area boreholes drilled at different orientations to provide

an optimum amount of geotechnical information. The 2011 boreholes were drilled and logged as part of a confirmatory geotechnical program used to assess the previously collected geotechnical data. Acoustic (ATV) and optical (OTV) televiewer surveys were also carried out for one cave footprint and two decline alignment boreholes by DGI Geoscience.

Geotechnical logging by SRK staff for the 2011 programme was carried out for subsequent use of the In-Situ Rock Mass Rating (IRMR) method, after Laubscher & Jakubec (2000). Information recorded included:

- Drill core parameters: Total core recovery, rock quality designation, orientation offset;
- Rock mass parameters: Intact rock strength, microdefect intensity & strength, matrix alteration, veining intensity;
- Joint parameters: Discontinuity elevation-orientation & type-character, joint condition; and
- Large scale structures: Location, orientation, physical properties.

Review of historic geotechnical logging databases resulted in modification of fracture frequency and rock quality designation (RQD) values collected during the 2010 drilling program.

In-field and laboratory testing of core from all three geotechnical drilling programmes included point load tests, direct shear tests, uni-axial compressive strength tests, tri-axial compressive strength tests, and Brazilian tensile strength tests; covering all lithology and alteration types.

As a result of SRK`s assessment of existing and collected geomechanical data, five geotechnical domains were identified, as follows:

- Black Lake Intrusives (being the main orebody);
- Hazelton Formations (being a barren wedge to the north of the footprint);
- Takla Group, Potassic Alteration (above the Black Lake Intrusives);
- Takla Group, Phyllic Alteration (above the Takla Potassic); and
- Takla Group, Sulphate Leach (above the Takla Phyllic).

In addition, SRK modelled the fault zone (around the Kemess North and East Bounding Faults) and syenite dykes as separate geotechnical domains.

Figure 15.5 shows the relative locations of the various lithological and alteration zones at KUG.

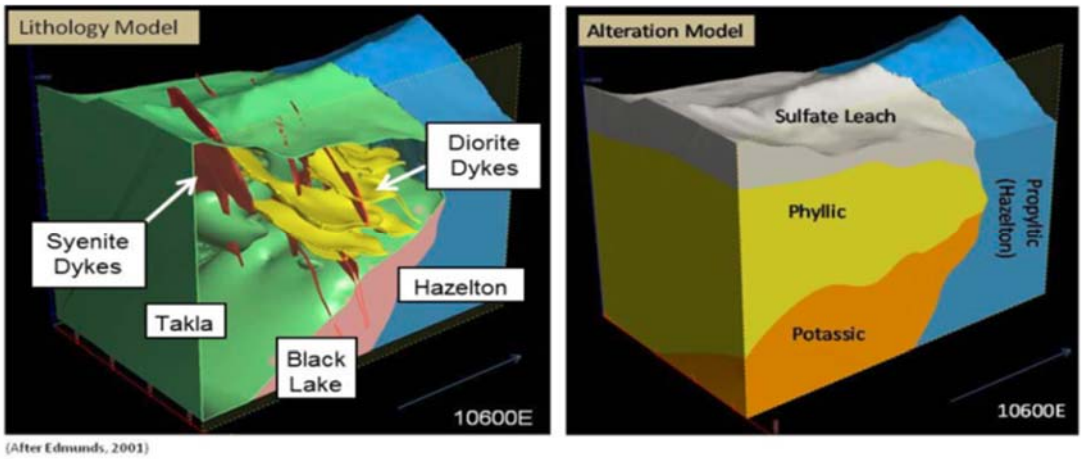


Figure 15.5: Lithological & Alteration Zones at KUG

(Source: SRK, 2016)

The location, width of deformation, and character of the Kemess North & East Bounding Fault damage zone was reviewed using pierce points from 73 boreholes; with strength rates qualitatively as weak, intermediate, or strong. The resultant 3D model generally trends E-W in the west and N-NW in the east, with the upper section dipping to the north and the lower section dipping to the south, and is ±60 m wide. This fault damage zone intersects the north side of the cave footprint and is, therefore, considered the most significant structural feature affecting the KUG cave. A zone of weakness has also been identified, associated with pre-mineralisation diorite dykes/sills. Surface structural trends were derived by SRK from satellite imagery and McKinley (2006) outcrop mapping structural data; N-S, E-NE, E-SE, NW, and NE trending. The N-S trends associated with surface ridges were not identified from borehole data, and should be further investigated as underground geotechnical borehole data becomes available.

A structures database was compiled using 2011 SRK core logging, 2011 SRK televiewer, and 2003 Knight-Piesold data. As a result, stereographic plots were generated for each of the five geotechnical domains and joint sets identified.

While no KUG site-specific in situ stress measurements have been carried out, in situ stress conditions are inferred from other data, specifically the 2004 Atomic Energy of Canada Limited (AECL) in situ stress testing programme below the Kemess South open pit. This testing involved nine tests in four boreholes, concluding that maximum principal in situ stress is aligned N-S (k1 = 1.26), intermediate principal in situ stress is aligned sub-vertical (= 1.00), and minor principal in situ stress is aligned roughly E-W (k2 = 0.66). At a depth of 500 m below surface, SRK estimates maximum, intermediate, and minor principal in situ stresses to be σ_{max}=15 MPa to 20 MPa, σ_{int}=14 MPa, and σ_{min}=8 MPa to 10 MPa respectively. KUG-specific in situ stress measurements will be needed once access to the cave footprint is established.

Geotechnical domains were derived by considering geological and geotechnical properties including alteration, rock quality designation (RQD), intact rock strength (IRS), fracture frequency,

joint character, and rock mass rating (RMR). Alteration and lithology were identified as having most control over rock mass quality. The following comments are provided for each of the main geotechnical domains:

Black Lake Intrusives: This domain contains minor zeolite veining which may affect the rock mass strength locally as these veins have low shear and tensile strength. While allowing for increased caveability, the presence of zeolite veins requires increased ground support locally. This domain contains the highest copper and gold grades, represents the initial ore unit to be caved, and is bound to the north by the fault damage zone. (Figure 15.6)



Figure 15.6: Black Lake Intrusives, Borehole KN11-01 (722 m to 737 m)
(Source: SRK, 2013)

Hazelton: This domain contains excessive zeolite veining which is likely to significantly affect (i.e. reduce) rock mass strength. While allowing for increased caveability, the presence of zeolite veins requires increased ground support. This domain occurs to the north of the fault damage zone, and does not contain mineralisation (Figure 15.7).



Figure 15.7: Hazelton, Borehole KN11-05 (662 m to 672 m)
(Source: SRK, 2013)

Takla Group, Potassic Alteration: This domain contains few zeolite veins. This domain contains relatively high copper and gold grades, represents the second ore unit to be caved, and is bound to the north by the fault damage zone and overlies the black lake intrusive (Figure 15.8).



Figure 15.8: Takla Potassic, Borehole KN11-05 (455 m to 463 m)
(Source: SRK, 2013)

Takla Group, Phyllic Alteration: This domain contains few zeolite veins although the phyllic alteration generally creates relatively weak joint strength. This domain contains relatively low copper and gold grades, represents the final ore unit to be caved, and is bound to the north by the fault damage zone and overlies the potassic alteration zone (Figure 15.9).



Figure 15.9: Takla Phyllic, Borehole KN11-05 (177 m to 187 m)
(Source: SRK, 2013)

Takla Group, Sulphate Leach: This domain is not included in the optimal cave volume due to relatively low copper and gold grades, but is expected to create dilution to the underlying cave due the relatively weak rock mass. This domain extends to surface above the phyllic domain and is bound to the north by the fault damage zone (where it deepens) (Figure 15.10).



Figure 15.10: Takla Sulphate Leach, Borehole KN11-01 (83 m to 92 m)

(Source: SRK, 2013)

Table 15.4 summarises rock mass data for each of the geotechnical domains.

Table 15.4: Geotechnical Domains, Rock Mass Data

Rock Mass Domain	% Drilled[1]	IRS (MPa)[2]	Intact UCS (MPa)	Intact UCS_{30th} (MPa)[3]	Tensile Strength (MPa)	Intact Young's Modulus (GPa)	Poisson's Ratio	Density (g/cm^3)	m_i[4]	RQD	FF/OF <10/m[5]	FF/J <10/m[5]	JC_{RMR90}	RMR_{90} / GSI	GSI_{30th}[3]	IRMR[6]
Takla - Phyllic	16%	90 (s.d. 61)	53 (37-76) (n=4)	42	-	24.6 (s.d. 15) n=4	0.20 (s.d. 0.06) n=4	2.80	15*	94 (s.d. 14)	3.5 (s.d. 1.8)	0.8 (s.d. 1.9)	21 (s.d. 4)	60 (s.d. 14)	50	58 (s.d. 17)
Takla - Potassic	40%	119 (s.d. 44)	91 (46-125) (n=9)	75	8.6 (s.d. 3.7) n=12	44.5 (s.d. 10) n=4	0.23 (s.d. 0.07) n=4	2.93	15	97 (s.d. 7)	4.3 (s.d. 1.9)	0.7 (s.d. 1.3)	21 (s.d. 4)	64 (s.d. 10)	60	64 (s.d. 13)
Black Lake	20%	117 (s.d. 41)	106 (37-272) (n=13)	85	7.6 (s.d. 3.6) n=2	31.1 (s.d. 18) n=6	0.16 (s.d. 0.04) n=6	2.74	15	98 (s.d. 4)	4.3 (s.d. 1.6)	0.5 (s.d. 0.9)	21 (s.d. 5)	65 (s.d. 9)	62	67 (s.d. 11)
Hazelton	17%	145 (s.d. 18)	130 (59-237) (n=10)	99	8.3 (s.d. 4.5) n=5	51.7 (s.d. 25.6) n=4	0.27**	2.70	16	99 (s.d. 3)	5.9 (s.d. 2.7)	0.5* (s.d. 0.5)	23 (s.d. 3)	69 (s.d. 7)	65	68 (s.d. 6)
Sulphate Leach	5%	40	As Takla Phyllic	-	-	As Takla Phyllic	As Takla Phyllic	2.60*	As Takla Phyllic	60* (s.d. 30)	-	-	14	40 (s.d. 13)	32	30*
Faults	-	25	50*	-	-	-	-	2.60*	As Takla Phyllic	70*	-	-	14	40*	32*	30*
Syenite[7]	2%	-	150*	120*	-	22.5 (-) n=1	0.15 (-) n=1	2.80*	20*	93 (s.d. 18)	5.9 (s.d. 2.8)	1.1 (s.d. 1.7)	21 (s.d. 4)	50*	35*	45*

Notes:

1. % drilled refers % linear metres drilled within the cave zones from the 2003, 2010, and 2011 datasets.
2. IRS (intact rock strength) estimate from field obseravtions only.
3. 30th refers to 30th percentile of data distribution (refer to figures and text). GSI considered equivalent to RMR90.
4. mi value determined from tensile and UCS testing (supported by typical ranges for similar rock types).
5. <10/m indicates these values exclude heavily fractured zones (i.e. faults from the background values used in fragmentation assessment).
6. IRMR does not consider open fractures here; natural joints only. This value is the conservative end of the accepted IRMR ranges.
7. Syenite contacts roughly 30% damaged (shearing, gouge etc); reduced rock mass parameters generated for this domain.

*represent estimated values (from photos and data) where data alone is insufficient for parameter determination.

** Refer to text regarding lab data interpretation

15.2.3 Caveability Assessment

The ability of a rock mass to cave is of paramount importance for any planned cave mining operation. A key parameter is the undercut dimensions required to initiate and sustain caving. In addition to SRK`s empirical assessment based on Laubscher`s method (1990), Itasca (Minneapolis office) was also contracted to carry out numerical modeling using FLAC3D.

Overall, both empirical assessment and numerical modelling approaches show the KUG deposit will cave readily at a hydraulic radius (HR) well within the complete footprint dimensions.

Empirical assessment of caveability uses rock mass competency values (RMR or IRMR) to which mining-related adjustments are applied to derive modified RMR (MRMR) values (Laubscher & Jakubec, 2000). Adjustments relate to weathering, joint orientation, mining induced stress, blasting, and the presence of water and/or ice. For the KUG situation, horizontal stresses are not expected to be of significance given the relatively shallow deposit depth and prevailing tectonic environment.

Three alternative joint/fracture frequency scenarios were considered when evaluating caveability, being:

- Optimistic - natural joints + 100% artificial breaks,
- Expected - natural joints + 50% artificial breaks, and
- Coarse - natural joints only.

For the domains that require caving, the HR required to achieve caving is in the range 23 to 33 across the range of MRMR values. This is well within the HR of 100 that’s associated with the complete KUG cave footprint area of ±600 m x 100-300 m. In fact, this range is also well within a HR of 50 that represents just one quarter of the complete KUG cave footprint area.

Figure 15.11 and Figure 15.12 show the empirical assessment results on stability diagrams for the Black Lake Intrusives, Hazelton, Takla Potassic, and Takla Phyllic geotechnical domains. MRMR (y-axis) is plotted against HR (x-axis). The yellow lines denote the most likely (expected) case for each geotechnical domain, while the blue and red lines represent the pessimistic and optimistic cases respectively. The red dashed line represents a cave area of approximately 300 m x 150 m.

For the Black Lake Intrusives and both Takla geotechnical domains, the area associated with the most likely case (HR = 28) is 112 m x 112 m. For the Hazelton, it is 120 m x 120 m.

The overlying sulphate leach domain has pessimistic, expected, and optimistic MRMR values of 25, 22, and 21; associated HR values are 13, 12, and 11 respectively.

These values do not include any adjustment for the effect of large scale structure on caveability, which is expected to reduce the required HR by 5% to 10%.

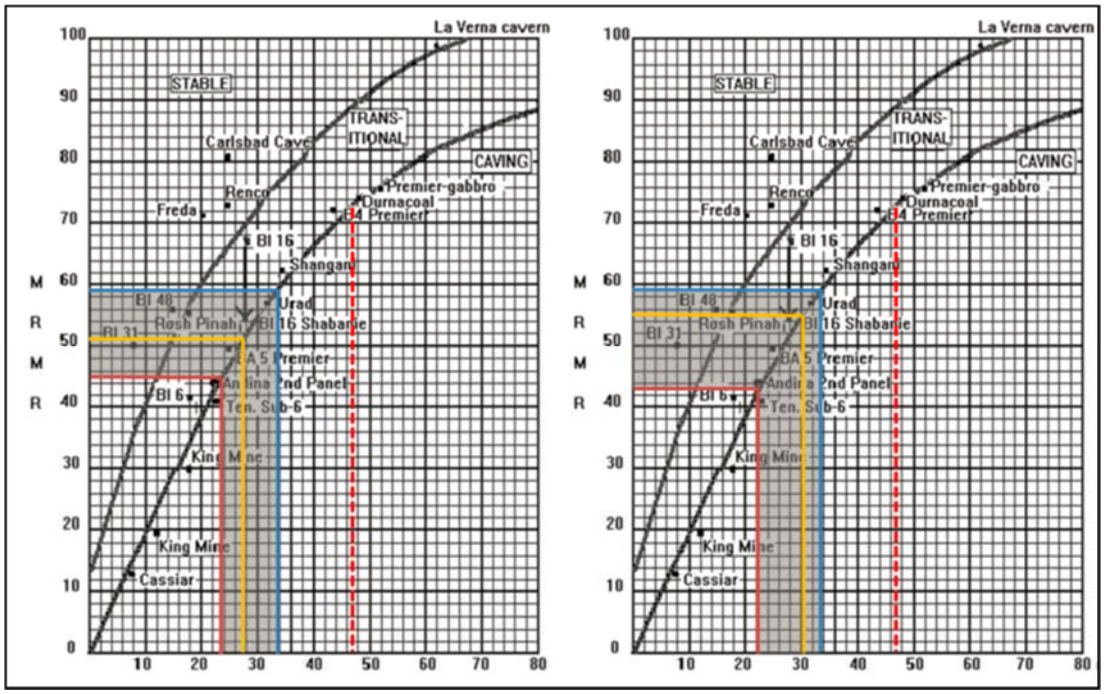


Figure 15.11: Stability Diagrams for Black Lake Intrusives (left) and Hazelton (right)

(Source: SRK, 2016)

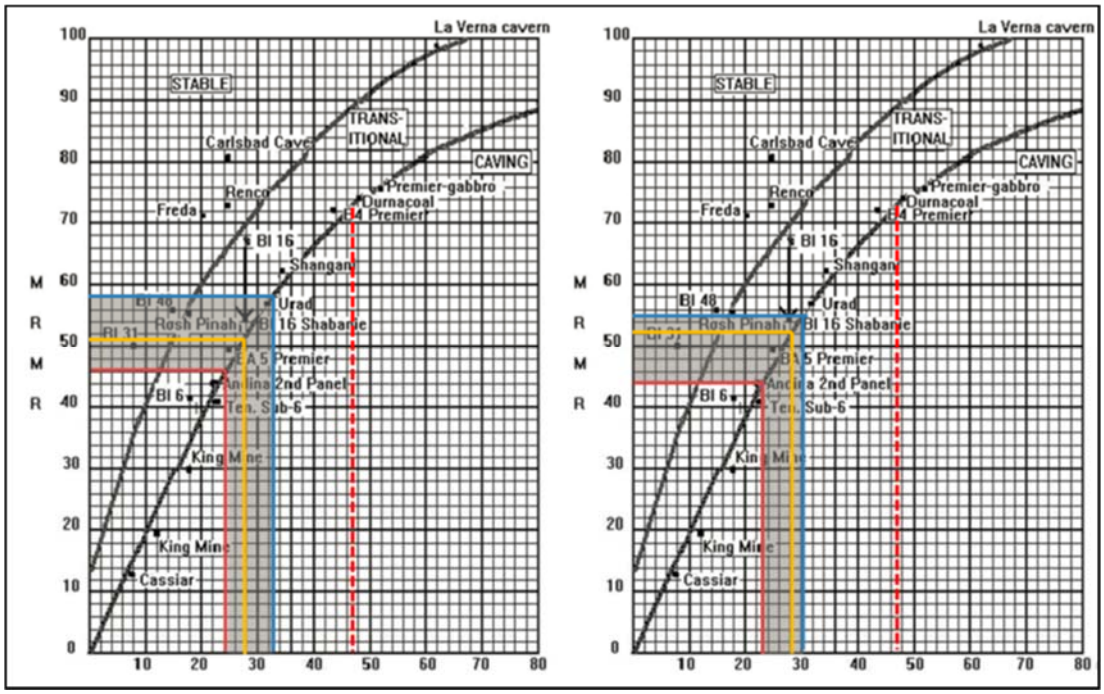


Figure 15.12: Stability Diagrams for Takla Potassic (left) and Takla Phyllic (right)

(Source: SRK, 2016)

Numerical modelling of the KUG cave was based on rock mass properties provided by SRK. The underlying graphic (Figure 15.13) shows the results of the Itasca modelling. This shows the KUG deposit will cave given the planned dimensions, and that the cave will propagate to surface 1.5 years after cave initiation. The yellow iso-surfaces represent 0.5 m vertical displacements.

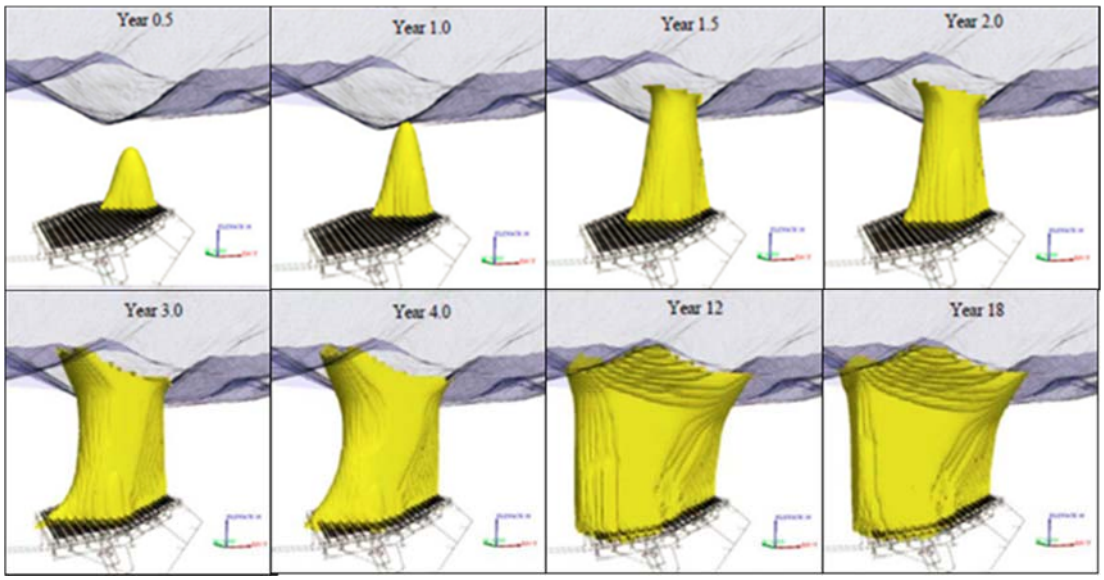


Figure 15.13: 0.5 m Vertical Displacement Iso-surface

(Source: SRK, 2016)

In addition to the abovementioned cave numerical modelling, Itasca modelling estimated the extent of the subsidence cone. This is shown by the final isosurface in Figure 15.13 and represents a factor of safety of 1.5. As a result of this analysis, a stand-off distance of at least 50 m is recommended for infrastructure.

15.2.4 Fragmentation Assessment

Fragmentation is a key factor in determining both efficiency and effectiveness of any planned cave mining operation, affecting production capacity and ore recovery. In addition to SRK's empirical assessment using Block Cave Fragmentation – version 3.05 (BCF v3.05) software, Itasca (Minneapolis office) was also contracted to carry out numerical modelling using discrete fracture network (DFN) and synthetic rock mass (SRM) approaches with Particle Flow Code (PFC).

Primary fragmentation refers to the size distribution of rock fragments as they are released from the cave back, with rock mass properties, joint properties, cave geometry, and induced stresses being key factors. Secondary fragmentation results from further size reduction of caved rock as it moves down the draw column, with cave pressure, draw height, and aspect ratio being key factors.

BCF software was used to estimate fragmentation for the Black Lake, Hazelton, Takla Potassic, and Takla Phyllic geotechnical domains. The following rock mass properties were applied (Table 15.5).

Table 15.5: Rock Mass & Joint Properties for Fragmentation Assessment

Domain	MRMR	Mi [1]	Intact Rock Strength	Fracture/Veinlet Frequency	Fracture/Veinlet Condition	IBS [2]
Black Lake	51	17	106	10	20	50
Hazelton	55	19	130	20	20	51
Takla Potassic	52	15	91	10	20	43
Takla Phyllic	51	15	53	10	20	25

Note:

1. Mi = Hoek Brown parameter constant. Mi value determined from tensile and UCS testing (supported by typical ranges for similar rock types).
2. IBS = Intact Block Strength. Scaled parameter of Rock mass strength from Laubscher 1990

Joint set parameters applied were number, dip, spacing, and condition of joints. Cave geometry factors applied included dip angle (45° constant) and dip direction of cave face. Induced stress factors applied included dip stress, strike stress, and normal stress. For the KUG rock mass, induced stresses are expected to be 1.50x to 1.75x the in situ stresses.

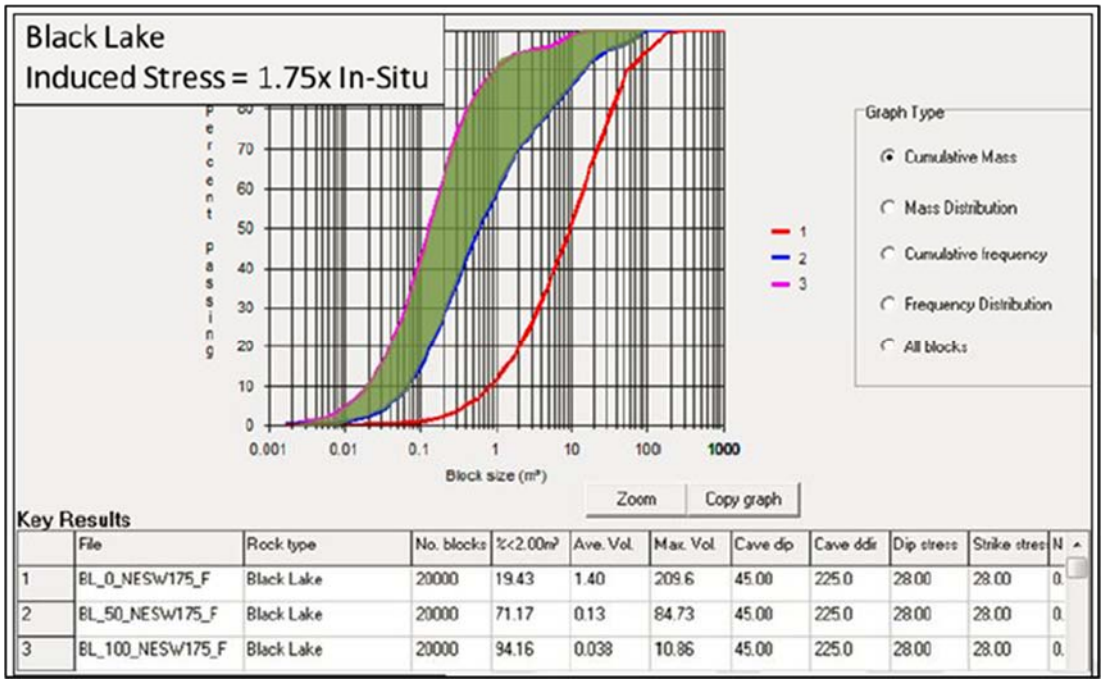
Three different cave propagation directions were evaluated (N-S, E-W, and NE-SW), with the associated dip and strike stress values shown below (Table 15.6).

Table 15.6: Dip & Strike Stress Values for Various Cave Propagation Directions

Description	Cave Propagation Direction					
	N-S		E-W		NE-SW	
Induced Stress Factor	1.50	1.75	1.50	1.75	1.50	1.75
Dip Stress (MPa)	32	37	18	21	24	28
Strike Stress (MPa)	18	21	32	37	24	28

The three joint conditions outlined in the previous section (Caveability Assessment) were used to assess primary fragmentation as these are considered to provide a reasonable range, with actual fragmentation expected to be bound by the curves representing “expected” (natural + 50% artificial breaks) and “optimistic” (natural + 100% artificial breaks) joint conditions.

Figure 15.14 shows the effect of the different joint conditions on primary fragmentation distribution for the Black Lake domain; line 1 (red) = pessimistic, line 2 (blue) = expected, and line 3 (purple) = optimistic. The hatched green area between the expected and optimistic lines represents the most likely fragmentation distribution range. For this particular BCF run, the NE-SW cave propagation direction was chosen.



	File	Rock type	No. blocks	%<2.00m³	Ave. Vol.	Max. Vol.	Cave dip	Cave ddir	Dip stress	Strike stres	N
1	BL_0_NESW175_F	Black Lake	20000	19.43	1.40	209.6	45.00	225.0	28.00	28.00	0.
2	BL_50_NESW175_F	Black Lake	20000	71.17	0.13	84.73	45.00	225.0	28.00	28.00	0.
3	BL_100_NESW175_F	Black Lake	20000	94.16	0.038	10.86	45.00	225.0	28.00	28.00	0.

Figure 15.14: Black Lake Primary Fragmentation Distribution for Varying Joint Conditions

(Source: SRK, 2016)

Sensitivity of primary fragmentation to induced stress factors (1.50 and 1.75x in situ stresses) was also evaluated, with results showing relative insensitivity. As such, the 1.5 factor was used to evaluate the effect of cave propagation direction on primary fragmentation.

Table 15.7 shows the results for the "expected" (50% artificial breaks) and "optimistic" (100% artificial breaks) joint conditions. Results show that for the two main ore bearing domains, Black Lake and Takla Potassic, E-W and N-S cave propagation directions result in an increased amount of -2 m^3 primary fragmentation compared to the NE-SW direction. Based on these results and orebody grade distribution, an E-W cave propagation direction was chosen for KUG.

Table 15.7: Primary Fragmentation % Passing 2m^3 by Cave Propagation Direction

Domain	Artificial Breaks (%)	Stress Factor	Cave Propagation Direction		
			E-W (%)	N-S (%)	NE-SW (%)
Black Lake	50	1.5	79	79	69
	100		90	92	93
Hazelton	50		87	88	88
	100		98	98	98
Takla Potassic	50		84	80	70
	100		97	92	90
Takla Phyllic	50		90	77	82
	100		97	90	91

Secondary fragmentation runs were carried out for each of the four geotechnical domains based on NE-SW cave propagation direction, 50% artificial breaks, and 1.5x stress factor; for draw heights of 20 m, 50 m, 100 m, and 200 m, and draw rates of 10 cm/day and 20 cm/day. Figure 15.15 shows the results for 10 cm/day draw rate. The red line represents primary fragmentation; the blue line represents 20 m draw height; through to the green line representing 200 m draw height.

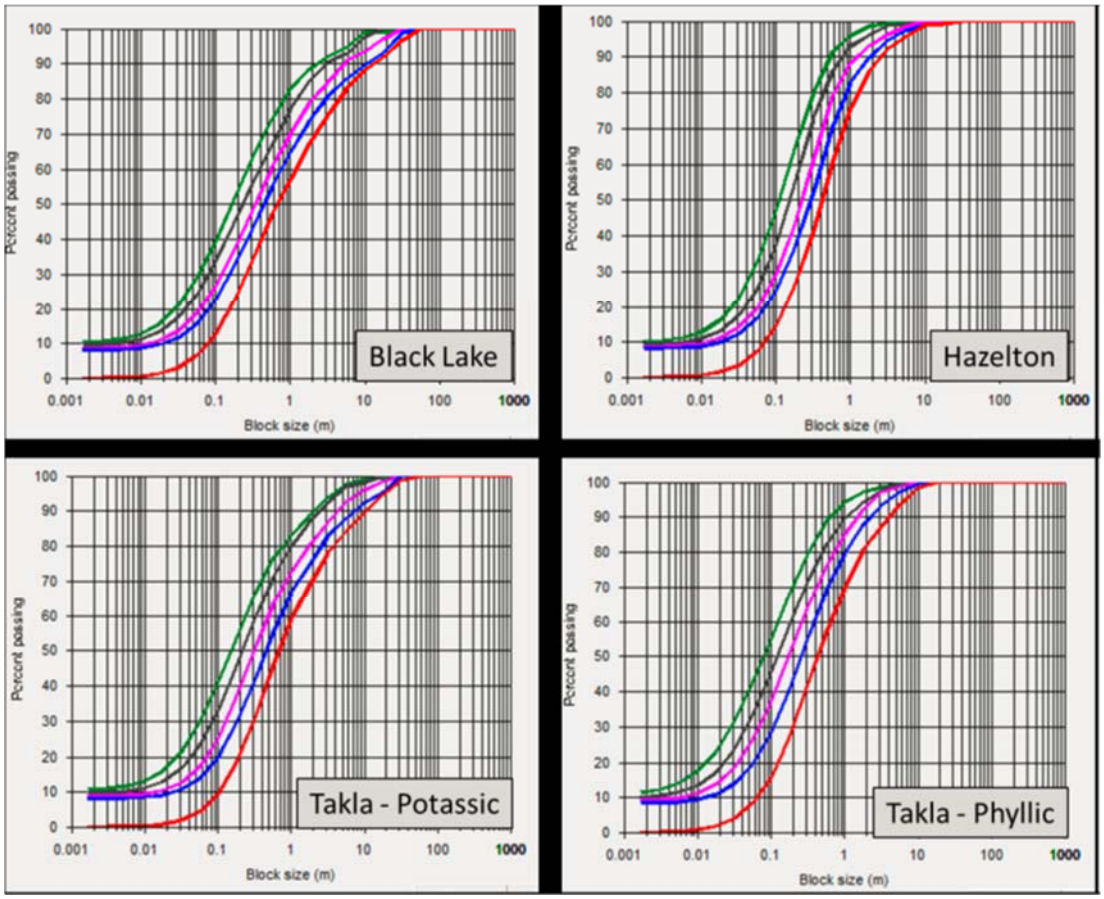


Figure 15.15: Secondary Fragmentation Distribution by Geotechnical Domain

(Source: SRK, 2016)

Given that the chosen cave propagation direction is E-W, actual fragmentation is expected to be represented by 50% to 100% artificial breaks and the stress factor is estimated to be 1.50x to 1.75x in situ stresses, Figure 15.15 is considered to represent a somewhat conservative view of secondary fragmentation.

Figure 15.15 shows that the Takla Phyllic domain is expected to fragment finer than the underlying Takla Potassic and Black Lake domains, causing preferential draw of this unit.

A comparison of draw rate effects on secondary fragmentation shows negligible difference between the 10 cm/day and 20 cm/day draw rates. The rate used in PCBC™ to produce the final cave production schedule was 17 cm/day.

DFN numerical modeling evaluates 3D blocks caused by open joints, with Itasca's base case assuming a conservative fracture frequency of 0.5 fractures/m and primary fragmentation by joint extension only. Results showed a primary fragmentation 50th percentile of 1.6 m to 1.9 m equivalent diameter, which is considered an upper bound as fragmentation due to internal fracturing was not modelled.

SRM numerical modeling is the direct modeling of rock fragmentation due to the evolution of stresses within the cave, using PFC (Itasca 2007). Results showed a primary fragmentation 50th percentile of 0.8 m to 1.6 m equivalent diameter, equating to 60% to 85% passing 2 m^3 respectively.

In order to consider both joint extension (DFN) and block breakage (SRM) mechanisms on primary fragmentation, the DFN and SRM results need to be combined. Doing so would put the results closer to the 70% to 85% passing 2 m^3 primary fragmentation results obtained using an empirical approach.

The aforementioned BCF fragmentation results were used to estimate hang-up frequency and average size for each domain at varying draw heights. Table 15.8 shows results for the 20 cm/day draw rate.

The Black Lake domain is expected to generate the largest frequency and largest average size of hang-ups, although no high hang-ups (above the drawpoint brow) are expected. The Hazelton domain is expected to generate relatively few and small hang-ups, although this domain is expected to produce the largest high hang-ups. The Takla Potassic domain is considered to sit between the Black Lake and Hazelton domains in terms of hang-up frequency and average size, albeit closer to the Black Lake domain. The Takla Phyllic domain is expected to be relatively insignificant in terms of hang-ups.

The hang-up frequency and average size data was combined with height of draw (HOD) and geotechnical domain cave production data from PCBC™ to estimate the number and average size of low and high hang-up events per day for each of the four geotechnical domains on a quarterly basis. This data was then used to estimate secondary breakage equipment, labour, and consumables requirements for each period in the cave's life.

Table 15.8: Hang-Up Frequency & Size by Geotechnical Domain

Draw Rate 20 cm/day	% Hangup				Average Size (t)							
Draw Height (m)	20	50	100	200	0		50		100		200	
					L	H	L	H	L	H	L	H
Black Lake	8	7	5	4	26.4	--	19.62	--	15.04	--	15.81	--
Hazelton	3	2	1	1	5.83	36.9	8.47	--	8.89	--	9.79	--
Takla - Potassic	--	5	4	3	14.35	23.42	16.73	--	16.7	--	16.01	--
Takla-Phyllic	--	--	1	1	9.9	--	10.4	--	8.6	--	11.6	--

15.2.5 Ground Support

SRK provided ground support recommendations for lateral and raise development outside the cave footprint, and for lateral development inside the cave footprint. Support recommendations were also provided for main infrastructure excavations. The following is a summary of these recommendations.

Development headings outside the footprint, including all three declines, where drift widths are 4.0 m to 6.0 m, ground support will consist of:

- 2.4 m long fully-grouted rebar (22 mm) on 1.5 m x 1.5 m pattern across back and shoulders, with 100 mm square face plates,
- 100 mm apertures #6 gauge galvanised welded wire mesh across back and shoulders, and
- 2.4 m long galvanised friction anchors (39 mm) in walls.

It is expected that ±15% of the aforementioned development will experience poorer ground conditions which will require the following:

- Reduce rebar spacing to 1.2 m x 1.2 m across back and shoulders;
- Extend mesh to floor, securing with friction anchors in walls; and
- 75 mm shotcrete on back, shoulders and walls (to floor level).

At intersections where the span exceeds 6 m, full column grouted cable bolts on 1.5 m x 1.5 m pattern should be installed prior to the breakaway being developed. Cable bolt length is expected to be 6 m, but actual conditions will determine length. Where possible, development of intersections in poor ground should be avoided, and the number of 4-way intersections should be minimised.

Where passing bays are developed along the access decline, additional ground support will be required due to the 9 m span; consisting of 5 m long full column grouted cable bolts on 2.0 m x 2.0 m pattern throughout the passing bay. Initial cable bolts should be installed prior to slashing the passing bay.

Crusher chamber support recommendations, assuming good ground conditions, are:

- 3.0 m long fully-grouted rebar (22 mm) on 1.5 m x 1.5 m pattern across back and walls, with 100 mm square face plates;
- 100 mm apertures #6 gauge galvanised welded wire mesh across back and walls,
- 75 mm to 100 mm shotcrete on back and walls;
- 9 m long full column grout cable bolts on 2.5 m x 2.5 m pattern on back; and
- 6 m long full column grout cable bolts on 2.5 m x 2.5 m pattern on walls.

Assuming fair quality rock mass along the length of the 5 m diameter return air raise, this raise will be unsupported. However, this requirement may change following drilling of geotechnical and/or pilot holes.

Ground support requirements for development inside the cave footprint are based on two rock mass environments, being:

- MRMR 40 to 60, representing 70% of development headings, and
- MRMR 30 to 40, representing 30% of development headings.

Table 15.9 shows the ground support recommendations for these two rock mass environments.

Table 15.9: Footprint Development Ground Support Recommendations

Development Type	Maximum Recommended Profile	Rock Mass Environment	
		MRMR 40-60	MRMR 30-40
Undercut Drift	4.5 mW x 4.0 mH	2.4 m long split sets on 1.5 m x 1.5 m pattern within 2m of floor. Welded mesh	Plus 25 mm shotcrete/fibrecrete
Production Drift	5.0 mW x 4.5 mW	2.4 m long fully grouted rebar on 1.2 m x 1.2 m pattern to floor level. Expanded mesh. 50 mm shotcrete or fibrecrete	Plus 50 mm shotcrete or fibrecrete, total 100 mm
Intersection	>6 m span	6 m long fully grouted cable bolts on 1.5 m x 1.5 m pattern	Plus 50 mm shotcrete or fibrecrete
Drawpoint Bullnose	n/a	Horizontal straps or cable slings encased in shotcrete/fibrecrete	Plus 50 mm shotcrete or fibrecrete
Drawpoint Brow	3.8 mW x 3.2 mH	6m long fully grouted cable bolts on 1.5 m x 1.5 m pattern. Rigid steel arches, encased in concrete, to 4 m from brow	n/a

Upon commencement of decline development at KUG, a ground control management plan (GCMP) will be compiled that incorporates changes relating to ground support as a result of experience gained during initial development.

It should also be noted that additional geotechnical drilling will occur as access development progresses towards the main infrastructure and cave footprint area in order to provide both excavation-specific and increased amounts of rock mass data.

15.3 Geotechnical Monitoring

Monitoring of cave progression is critical for safety and production, with the following systems recommended for KUG.

Time domain reflectometry (TDR) cables will be installed in surface drill holes to allow monitoring of cave propagation, with at least 20 such installations required across the cave volume.

Surface subsidence will be monitored using fixed survey prisms and periodic airborne surveys.

Extensometers and convergence monitors will be used to monitor pillar and drive deformation on both the undercut and extraction levels. Extensometers will be real-time monitors while convergence monitors will be manually read. These monitors will also be installed in major excavations.

A seismic system will be installed to continuously monitor the inelastic response of the rock mass to cave mining. This system provides information related to both the cave front and abutment stress intensity, with the latter allowing rock burst risk to be assessed.

The production management system will also be useful for geotechnical monitoring as it provides a tonnage basis from which draw column heights can be derived.

15.4 Mudrush and Airblast Assessment

15.4.1 Introduction

The mudrush phenomenon can be described as “an uncontrollable, sudden inrush of wet muck into the underground workings that may or may not result in injuries, loss of life, or loss of property”. Wet muck comprises a mixture of water, mud (fines and water), and broken rock in proportions such that it can flow if disturbed.

An airblast can be defined as “rapid displacement of large quantities of air, often under pressure, in a constrained underground environment caused by a fall of ground or other material”.

Cave mining is being proposed to extract the KUG orebody. Block or panel caving methods, as with any mining method that creates surface subsidence or craters, are potentially susceptible to wet muck flow.

The rock mass at KUG will require natural caving in moderately competent rocks and it will have the potential for hangups, voids and, in extreme cases, airblasts.

15.4.2 Industry Experience

Many caving and sublevel caving mines have experienced mudrushes and some have experienced airblasts.

The likelihood of airblasts can be reduced to acceptable levels relatively easily (with a “no void” policy and comprehensive monitoring procedure) and consequences could be, to a certain extent, minimized by leaving sufficient muckpile above the loading points. The likelihood of a mudrush cannot be eliminated (only minimized by proper draw control), but the consequences of mudrush could be significantly reduced. There are mines that operate safely in high mudrush-risk

environments (for example Freeport, Indonesia), but effective Standard Operating Procedures (SOP) and Trigger Action Response Plans (TARP) must be implemented.

15.4.3 Mudrush and Airblast Risk Evaluation

In order to evaluate the risk of mudrush, it is necessary to answer the following questions:

- Is there potential to generate wet muck at the KUG mine?
- What disturbance could mobilize wet muck that would result in an uncontrollable flow?
- What could be the extent of the wet muck problem and what is the likelihood that a flow could occur during the life of mine?
- What are the safety and economic consequences of a wet muck flow?
- What can be done to manage the risk?

In order to evaluate the risk of airblast, it is necessary to answer the following questions:

- Under what conditions can an airblast occur?
- What is the stability/caveability of the rockmass?
- What is the opportunity to create a confined void that can collapse?
- What are the safety and economic consequences of an airblast?
- What can be done to manage the risk?

In order to answer these questions, it is necessary to understand the geology, physical properties of the rocks, hydrological and hydrogeological regime, as well as production objectives and strategies. Although there is reasonable understanding of contributing factors AuRico does not currently have any physical data to determine the ability of various materials to produce fines and clays. A comprehensive testing program and analysis of existing data needs to be undertaken and the results compared with other operations that have experienced wet muck flow events. Until the testing is undertaken only a high level qualitative risk assessment is possible.

An independent study of the groundwater and surface water shows that the water inflows to the cave could be sufficient in volume to cause saturation of a portion of the muck if the permeability of the muckpile is low; i.e. so that drainage is inhibited. If the caved column is drawn erratically, the fines generated from overburden and weathered rocks could form a “seal” that could cause water accumulations in the caved mass, potentially forming pockets of wet and/or saturated muck.

Based on these findings, SRK concludes that there is a potential risk at KUG of wet muck ingress if these pockets reach the production drawpoints. In the case of high rainstorm events or freshet, the fines could be washed out with water through the cave and potentially enter the drawpoints.

In terms of flowability of the wet muck, laboratory tests need to be completed to understand under what conditions (moisture content and material mix) the continuous flow will occur.

The assessment of the KUG rock mass caveability concluded that the rock mass is moderately competent and caving will occur when 10,000 m² to 15,000 m² has been undercut.

Because of an only moderately competent rock mass and a large footprint, the risk that the cave will not propagate all the way to the surface resulting in airblast or that a portion of orebody will form an overhang is very low.

15.5 KUG Mudrush Risk Discussion

A simple qualitative risk analysis was conducted to identify the level of risk in the context of the expected mine life. The mine life is presented graphically in Figure 15.16 on an annual basis. The results are summarized in Table 15.10.

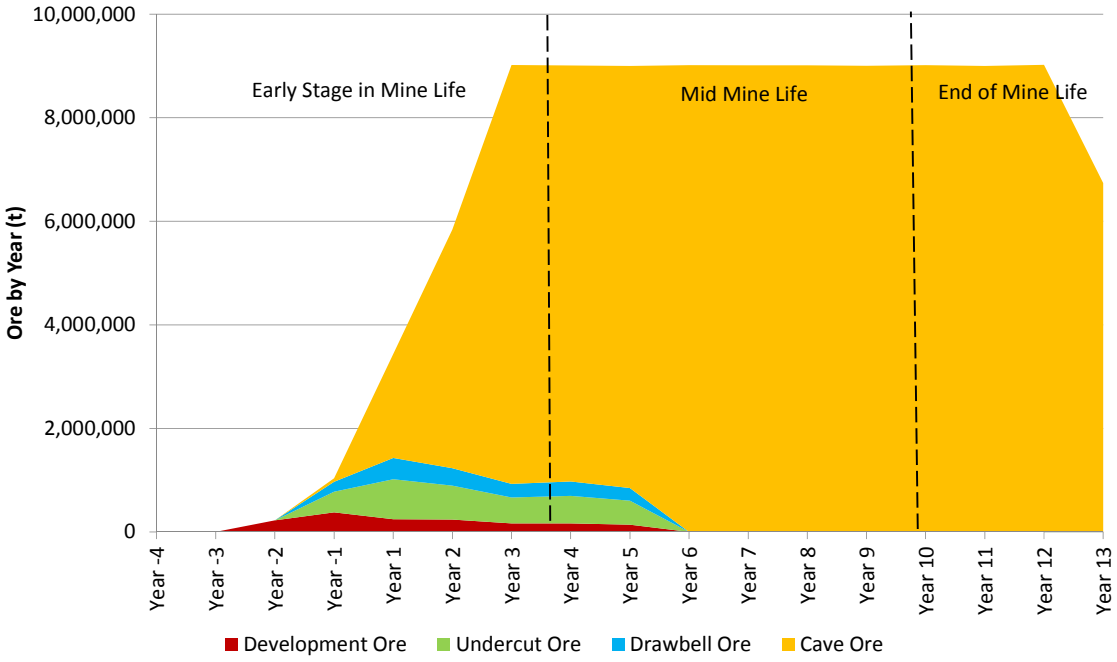


Figure 15.16: KUG production schedule and mine life time periods

(Source: SRK, 2016)

In terms of mudrush, SRK conclude that in the case of KUG, it is possible that a small scale event can occur any time after the cave reaches the surface and a crater is formed if there is an extreme rainfall event or during the freshet. However, this is manageable since such an event would only coincide with record rainfall and it is expected that wet muck would have very high water content. The amount of mud generating materials left behind will increase; hence, this type of event will trigger potentially larger volume mudrushes with time (Table 15.10).

Table 15.10: Summary of the Mudrush Risk Assessment for KUG

Wet Muck Flow Size	Level of Risk		
	Early Stage in Mine Life *	Mid Mine Life	End of Mine Life
Small (up to 50m^3)	Moderate	Moderate	High
Medium (50-1,000m^3)	Low	Moderate	Moderate
Large (more than 1,000m^3)	Low	Moderate	Moderate

Note: (*) This risk level is related to cave initiation – before the continuous caving occurs.

15.6 Planned Mining Methods

15.6.1 Cave Design

The cave layout was re-evaluated in context of the updated resource model using Gemcom's PCBC™ software to determine the highest value extraction level elevation and lateral extents, cave volume (draw column heights), and mining sequence. This section summarises the comparison of alternative cave layouts and identification of the highest value layout.

The process of identifying the highest value cave layout did not consider previous layouts from the 2013 Feasibility Study (SRK 2013), however, many of the constraints and inputs were left unchanged from the final values used in that study.

The key constraint from that study was tailings storage limitations in the Kemess South Pit of approximately 100 Mt. Removing this constraint from the final cave design resulted in a schedule containing 143 Mt with an average NSR of C$30.94/t.

The following key parameters were unchanged from the 2013 feasibility study (SRK 2013).

- **Maximum LOM cave quantity**: 100 Mt
- **Operating Cost** (Mining, Processing + G&A): C$15.3/t
- **Discount Rate**: 5%
- **Vertical Mining Rates**
- **Mixing Settings**

The following key parameters were changed from the 2013 feasibility study (SRK 2013).

- **Resource Model**: September 2015 Block Model
- **Metal Prices** (US$)
    - Cu = 2.50 C$/lb
    - Au = 1,200 C$/oz
    - Ag = 16 C$/oz

- **NSR**: NSR was calculated for each block

The following outlines the sequence of tasks undertaken to produce an optimised production schedule (in NPV terms).

1. **Highest value cave volume:** To identify the highest value cave volume, five oversized drawpoint layouts spanning the likely maximum extraction level extents were created at vertical intervals of 20 m from an elevation of 1120 mRL to 1200 mRL.
2. **Slice files**:
    - Slice files were created for each layout using the updated resource model and NSR values.
    - The slice files were premixed and adjusted for the undercut tonnage according to the method and settings used in the 2013 FS (SRK 2013).
3. **Height of Draw**: A Height of Draw (HOD) surface that contained the highest value 100 Mt was generated for each slice file based on the calculated NSR, operating costs, development costs and discount rate. Figure 15.17 shows an example of one of the HOD surfaces created using this method.

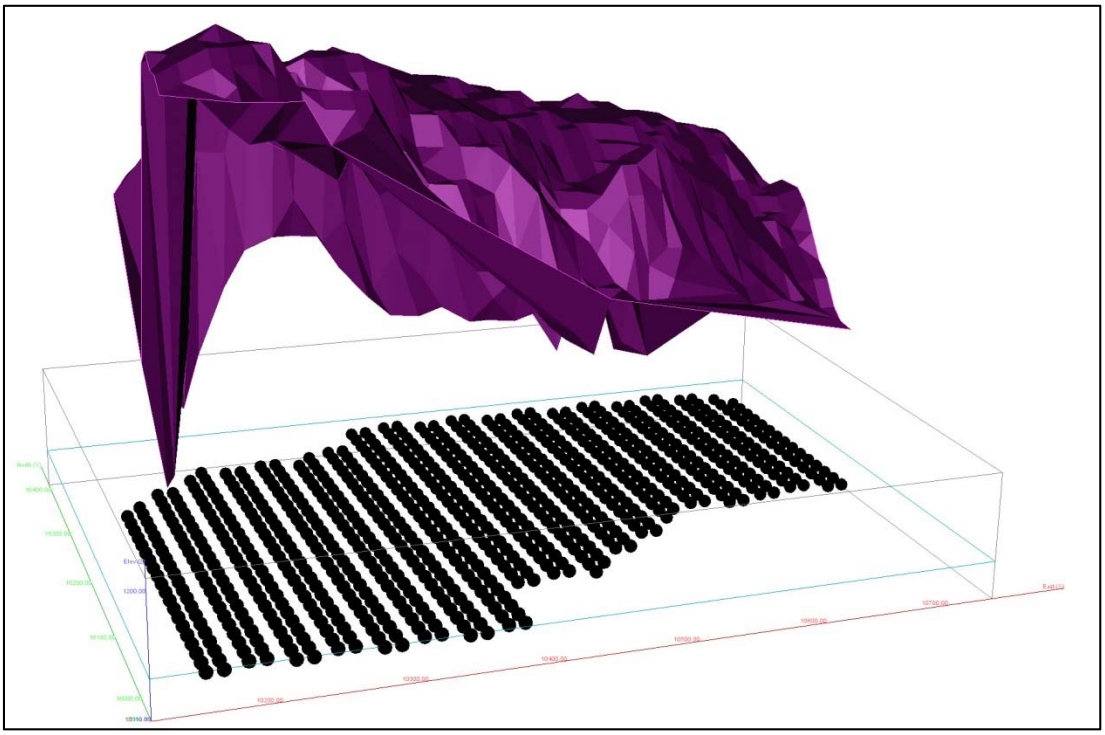


Figure 15.17: Example of HOD Surface Using an Extended Drawpoint Layout

(Source: SRK, 2016)

4. **Preliminary Production Schedules:** A number of preliminary production schedules were generated for each slice file using the HOD surfaces.
    - Three different target schedules were used to vary drawpoint construction rates and target production. All schedules used the same development sequence as the 2013 FS (SRK 2013) with a north-south cave front starting in

the east and progressing to the west. A comparison of the schedules is provided in Table 15.11.

- In all cases, an extraction level located at the 1140 mRL was found to generate the highest value in the production schedules.

Table 15.11: Comparison of Schedules

Design	Elevation (m)	Target Schedule Option NPV (C$M)		
		1	2	3
KMSS0a	1120	-	1,220.2	1,241.6
KMSS0b	1140	1,239.0	1,236.2	1,257.3
KMSS0c	1160	1,192.2	1,218.3	1,239.1
KMSS0d	1180	1,079.1	1,152.5	1,170.3
KMSS0e	1200	934.3	1,021.7	1,037.8

Description of Target Schedule Options

1. Drawpoints opened in two distinct phases at 30 per quarter
2. Drawpoints opened at 30 per quarter for first 12 quarters, 15 per quarter thereafter
3. Same as 2 with production constraints removed from early quarters
4. NPV was based on a 5% discount rate, no capital costs were included

5. **Optimised footprint outline & drawpoint layout**: The HOD surface (from task 3) was used to optimise the footprint outline and drawpoint layout (created in task 1). These were further refined to increase the grade early in the production schedule. Figure 15.18 shows the final drawpoint layout.

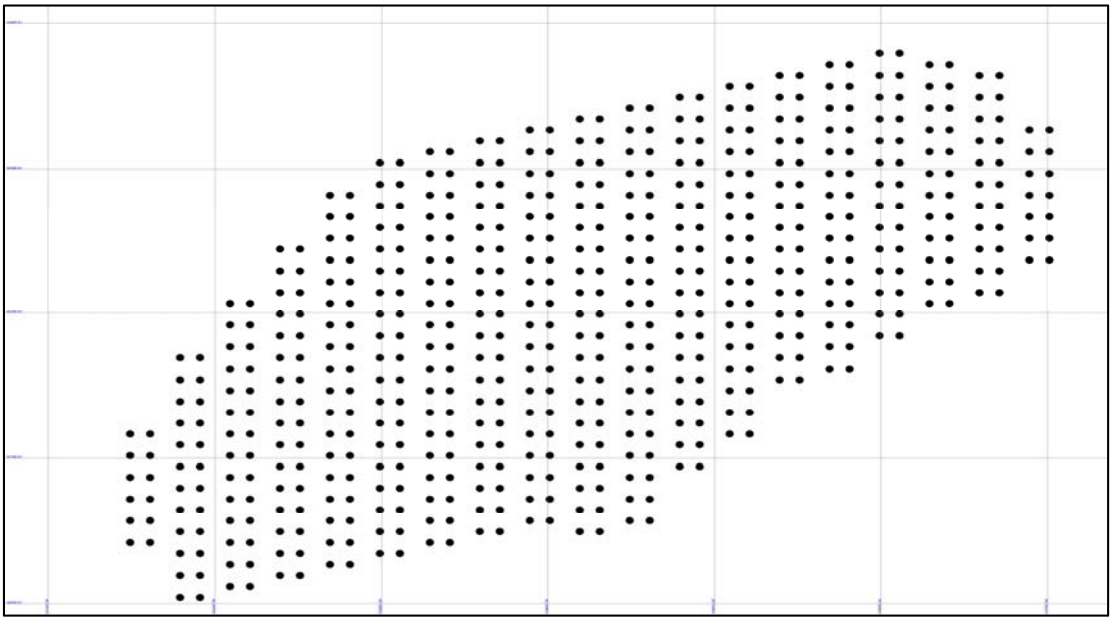


Figure 15.18: KUG footprint (drawpoint layout) at the 1140 mRL extraction level

(Source: SRK, 2016)

6. **Development sequences**: Various development sequences (incorporating different cave front geometry and direction of advance) were evaluated in PCBC^{TM} to identify the sequence with highest value. Only marginal value differences resulted from altering the geometry of the cave front. To simplify development and production scheduling, a cave front that progresses east-west was chosen. For a more complete discussion on cave scheduling, refer to Section 15.8.3 (Production).

A 3-D view of the final cave design is presented in Figure 15.19. This shows that draw columns are higher on the west (left) side compared to the east (right) side, and higher on the north (far) side compared to the south (near) side.

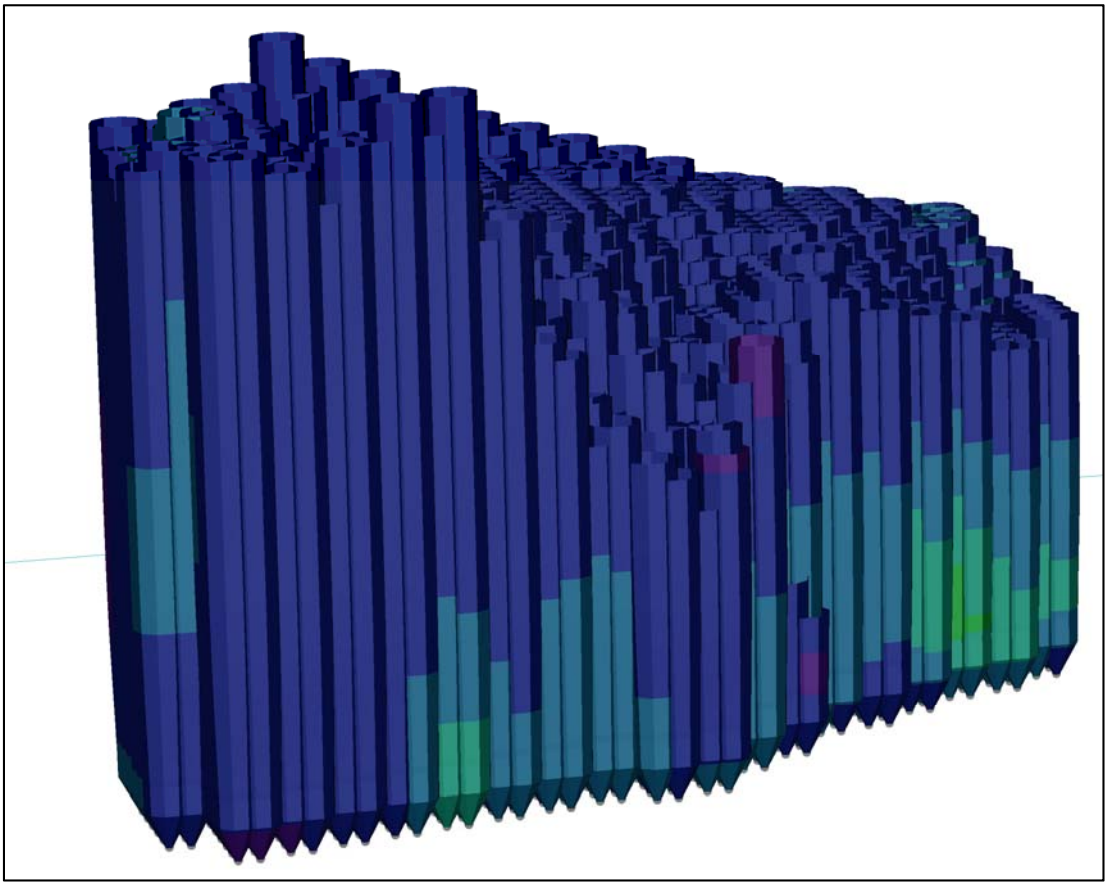


Figure 15.19: Isometric view of panel cave (looking northeast)

(Source: SRK, 2016)

15.7 Mine Design

15.7.1 Mine Access

This section focuses on the surface area in the Kemess Lake Valley immediately around the triple decline portals as well as the triple decline system.

AMEC Foster Wheeler (Burnaby, BC) has provided a design, schedule, and cost estimate for the approximately 4,165 m access corridor, comprising the surface haul road and access tunnel, from

the existing Kemess South facilities to the planned triple decline portal location. In addition to providing access to the portals, this corridor provides routing for hauling waste (and early ore from KUG), the overland conveyor (to transport ore from KUG to the existing processing facility), 25 kV electrical power supply to KUG, and 200 mm dewatering pipeline from KUG. While this corridor provides primary access to KUG, an alternative access/egress is available via an existing exploration road that traverses the hill directly north of the Kemess South open pit.

All planned Kemess Lake Valley infrastructure will be located in the Kemess Lake watershed.

Given the relatively steep topography, the access road and conveyor follow separate routes upon exiting the access tunnel north portal. The standard access road width of 12 m has been maintained in this area allowing two-way traffic along this section of road. The access road crosses above a buried section of the conveyor (CV02) adjacent to the transfer tower (between conveyors) so as not to impede access for equipment, materials, and personnel during operations.

Portal area facilities include the following, with their relative locations shown on Figure 15.20.

- CV01 to CV02 transfer tower;
- CV02 conveyor from conveyor decline portal area to access tunnel north portal;
- Development-construction contractor offices, workshop and stores;
- Helipad;
- Fans, air heaters and propane tank farm;
- Waste and ore stockpiles;
- Laydown area;
- Electrical substation;
- Water handling (contact and non-contact water);
- Stripped organics stockpiles; and
- Ventilation raise access road.

The mine development-construction contractor is expected to mobilise to site to commence access corridor construction from existing Kemess South facilities to the access tunnel in Year 5/Q4. The mine development schedule has their underground construction scope occurring over the period Year -4/Q1 to Year 4. It is expected the contractor will establish all office, workshop, and stores facilities required to support their underground scope in the Kemess Lake Valley following development of the access tunnel.

The primary twin intake air fans and air heaters for the production ventilation system will be located at the top of the intake air decline, providing intake air to the extraction and undercut levels. Propane tanks supplying the air heaters will be located a short distance away and downslope from the portals.

A single exhaust fan located at the conveyor decline portal will ventilate the decline ventilation system (comprising the access and conveyor declines) once this system is fully established. The propane-fueled intake air heater for the decline ventilation system will be located directly adjacent to the access decline portal.

Ore and waste stockpiles are located in close proximity to the access decline portal in order to facilitate rapid turnaround of underground haul trucks. Capacity for the ore and waste stockpiles is approximately 46 kt, which is in excess of peak stockpiling requirements of 4,500 t/d ore and 2,500 t/d waste.

A general laydown area is provided directly adjacent to the contractor's facilities. Access to this laydown is on its south side via the access road. In addition to being an overflow for the contractor's storage requirements, this area will serve as a temporary storage for owner equipment and supplies.

The electrical substation is located directly adjacent to the conveyor on the south side of the Kemess Lake Valley to provide electrical power to all area facilities, including conveyors head and tail motors, and the underground mine (KUG). Transformers are required to change the 25 kV supply to 4.16 kV. The 25 kV supply will be by armoured cable running on the conveyor structure through the access tunnel and into the Kemess Lake Valley.

Water handling facilities include the 200 mm dewatering pipeline from underground (which runs up the conveyor decline), contact water sump, pump house (for discharge of contact water to the Kemess South open pit), sedimentation pond for non-contact water, runoff diversion/collection ditches (for contact and non-contact water), and culverts. Drainage ditches and culverts are designed to handle a one in 50-year, 24-hour event.

The KUG dewatering pipeline will discharge into the contact water sump which is located in a natural topographic low point below the ore-waste stockpile area. The sump will be lined and measure approximately 40 m long x 20 m wide x 4 m deep. The sump will be designed to allow settling during a 10-year storm event and provide overflow capacity for a 200-year storm event. A pump house will be located adjacent to the sump to pump contact water to the KUG TSF.

The non-contact water sedimentation pond is located in the valley bottom and naturally collects all surface water in the area. The capacity of the pond will be increased by the construction of a small berm and decant structure at the natural outlet to allow settling during a 10-year storm event. A spillway will be constructed on the right abutment to provide overflow capacity for a 200-year storm event. The pond itself will not be lined as it is a natural pond. Two stripped organics stockpiles will be maintained in the Kemess Lake Valley area for remediation upon mine closure.

Access to the primary (production system) return ventilation raise will be by a 4,800 m long access road. The road will follow existing exploration roads up the hillside above the triple declines for approximately 3,600 m where a spur road approximately 1,200 m long will be constructed to connect with the ventilation raise. The road width is 6 m, allowing single lane traffic only which is considered adequate given its limited use as primary return ventilation fans are planned to be located underground in close proximity to the raise.

The triple decline system layout is shown on Figure 15.21. The access decline (5.0 m to 5.5 mW x 5.0 m to 5.5 mH) is located 16.5 m east of the conveyor decline (4.5 mW x 4.5 mH), and the intake air decline (6.0 mW x 5.0 mH) is located 18.0 m east of the access decline. The declines have insufficient separation distance to meet BC mining regulations for means of egress; an exemption will be required. All declines are inclined at -9.5%. Connecting cross-cuts (5.0 mH x 5.0 mW) are provided every 150 m along the length of the declines between the access and conveyor declines and every 300 m between the access and intake air declines to facilitate initial development by providing an effective ventilation network and storage areas; resulting in a total of 30 cross-cuts. The elevation of the three portals is 1,383 mRL, and the elevation of the bottom of the access decline is approximately 1,082 mRL, an elevation change of 301 m.

The access decline will be used for transport of all personnel and materials/consumables, and waste trucking (and some ore trucking prior to commissioning of the conveyor system). As such, 20 m long passing bays are provided every 300 m along its length; resulting in a total of ten passing bays. Given that conveyor and intake air decline development muck will be hauled via the access decline, re-muck bays are provided every 150 m along the length of the access decline.

Access to the extraction level is via the access decline at the 1,099 mRL, with access to the conveyor level from the extraction level access ramp. The access decline's final connection to the intake air decline is at the 1,142 mRL. The conveyor decline connects to the return ventilation level at the 1,132 mRL.

There are two temporary electrical substations provided for in cross-cuts #7 and #14 between the access and conveyor declines, and the main electrical substation is in cross-cut #21.

There is a fuel storage tank in cross-cut #20, fed from surface via a pipeline installed in the conveyor decline. Fuel will be delivered in batches to this tank and transferred to fuel bays elsewhere in the mine workings by fuel truck.

There are two truck loadouts at the bottom of the access decline, with the tips located on both the extraction (1,140 m) and undercut (1,158 m) levels, providing approximately 500 t storage capacity in each of the passes. This equates to approximately 40% of the peak waste handling requirement per shift (of 2,500 t/d) or 22% of the peak ore handling requirement per shift (of 4,500 t/d).

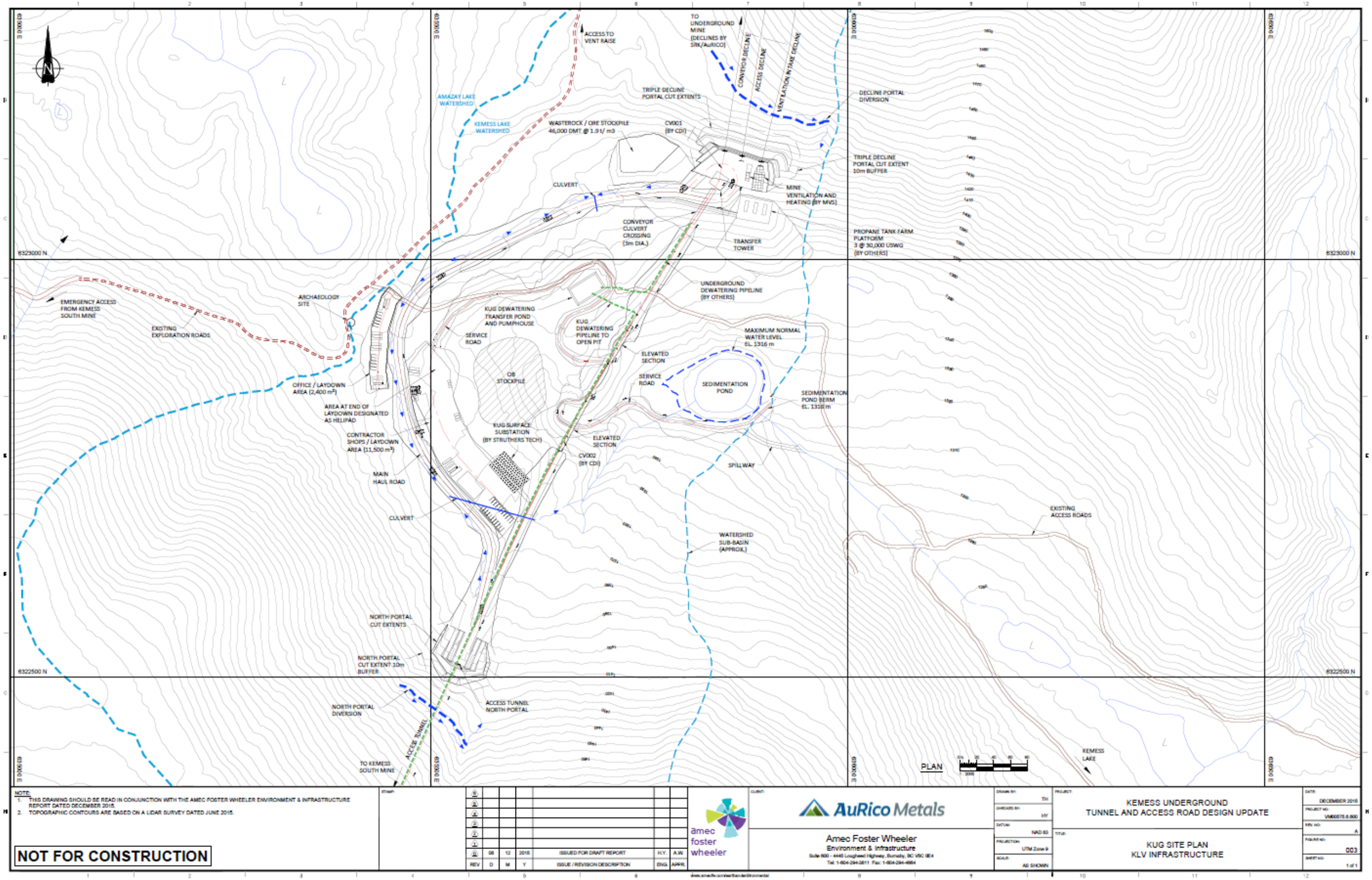


Figure 15.20: Portal area layout (Kemess Lake Valley infrastructure)
(Source: AuRico, 2016)

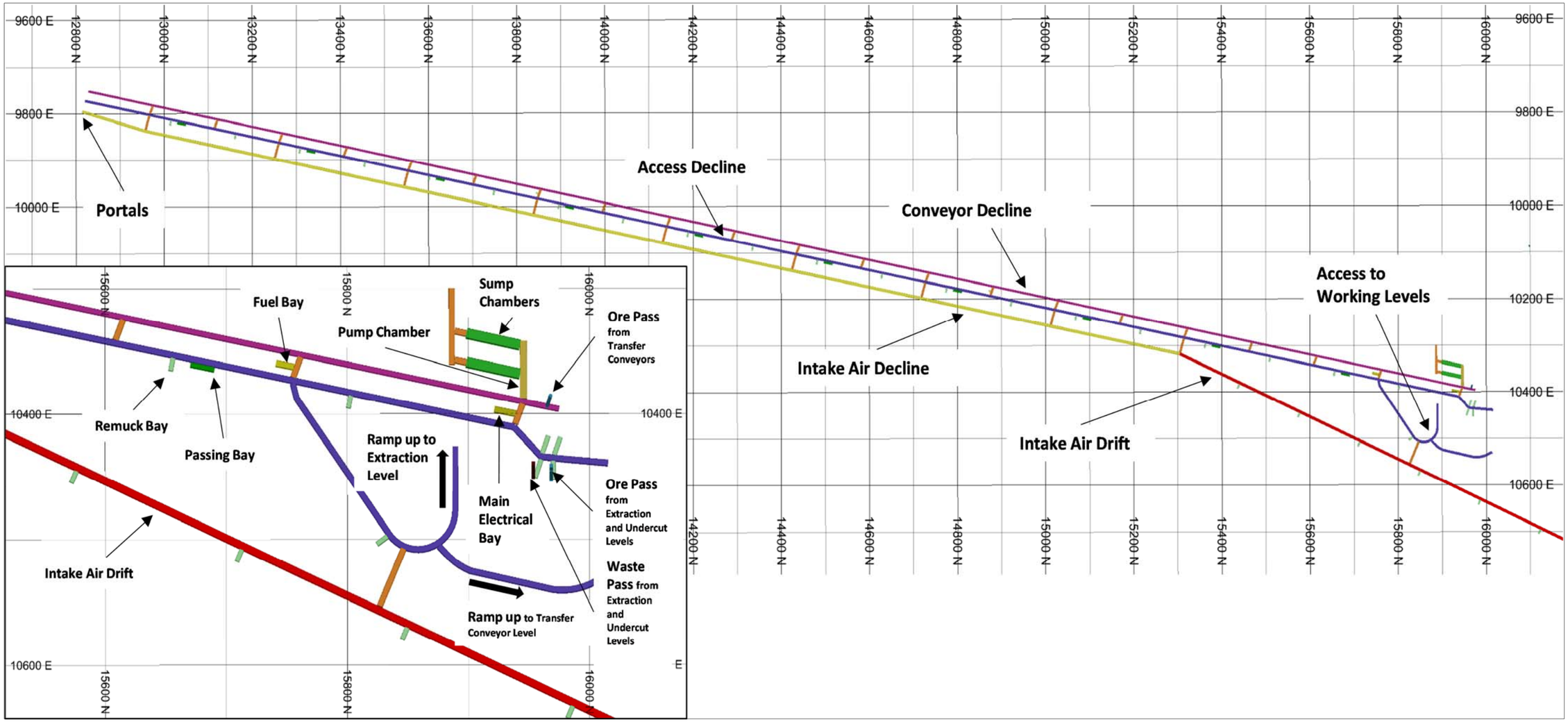


Figure 15.21: Decline system layout
(Source: AuRico, 2016)

15.7.2 Extraction Level

The extraction level is located at the 1,140 mRL and is accessed via the access decline at the 1,099 mRL, and is shown on Figure 15.22. From this main access, there are accesses both east and west of the workshop to the cave footprint. There are 20 extraction drives (4.5 mW x 4.5 mH) on 30 m centreline spacing providing access to 582 drawpoints (4.5 mW x 4.5 mH) or 291 drawbells, with caved ore being fed via the drawbells from above the undercut level. All ore will be trammed using diesel-drive LHDs to crusher tips located south of the extraction footprint, feeding one of four primary jaw crushers. A single (5.0 mW x 5.0 mH) rim tunnel is provided around the extraction footprint. Mobile equipment traffic will be limited in the south rim tunnel as priority will be for production LHDs feeding the primary jaw crushers.

Rim tunnels are provided for 360° access to the cave footprint, with all non-LHD traffic using these for access; including supervisory, secondary breakage, maintenance, technical services, rehabilitation, and construction traffic. The north rim tunnel can be accessed via the undercut level without having to travel along the south rim tunnel on the extraction level. Explosives and accessories (or cap) magazines are provided on both the east and west rim tunnels for easy access by secondary breakage personnel and equipment.

A refuge chamber is provided on the north rim tunnel just west of the ramp to the undercut level. This is in addition to a refuge chamber located at the main workshop on this level.

The main workshop is located on this level in close proximity to the level access; this facility is described in Section 15.9.7. There is a direct ventilation connection from the workshop to the underlying return air level. The fuel bay located close to the workshop is intended for use by all mobile equipment. The underground control room and shifters offices are also located in close proximity to the workshop.

Ventilation to this level is provided via 20 intake (2.4 m diameter) air raises located at the north end of each extraction drive. Air is routed to the return air level via ten return (3.0 m diameter) air raises situated just south of the south rim tunnel.

Five electrical bays are provided along the south rim tunnel, and four are provided along the north rim tunnel, to facilitate both development and production operations.

A total of four storage areas are provided, primarily for use during development and construction, two on the south rim tunnel and two on the north rim tunnel.

Drawbell dimensions were provided by SRK based on consideration of fragmentation and stability of workings and are shown by Figure 15.23. The width of the pillar across the major apex is 20 m, with the drawbell being 10 m in that dimension. Across the minor apex, the drawbells connect at a height of 10 m above the extraction level floor to form a continuous trough; this design was chosen as it is more favourable in terms of oversize being able to move towards the drawpoint but does result in a reduced minor apex pillar compared to drawbells connecting at the undercut level elevation. There is approximately 6,640 t associated with each drawbell, totalling 1.932 Mt associated with all 291 drawbells.

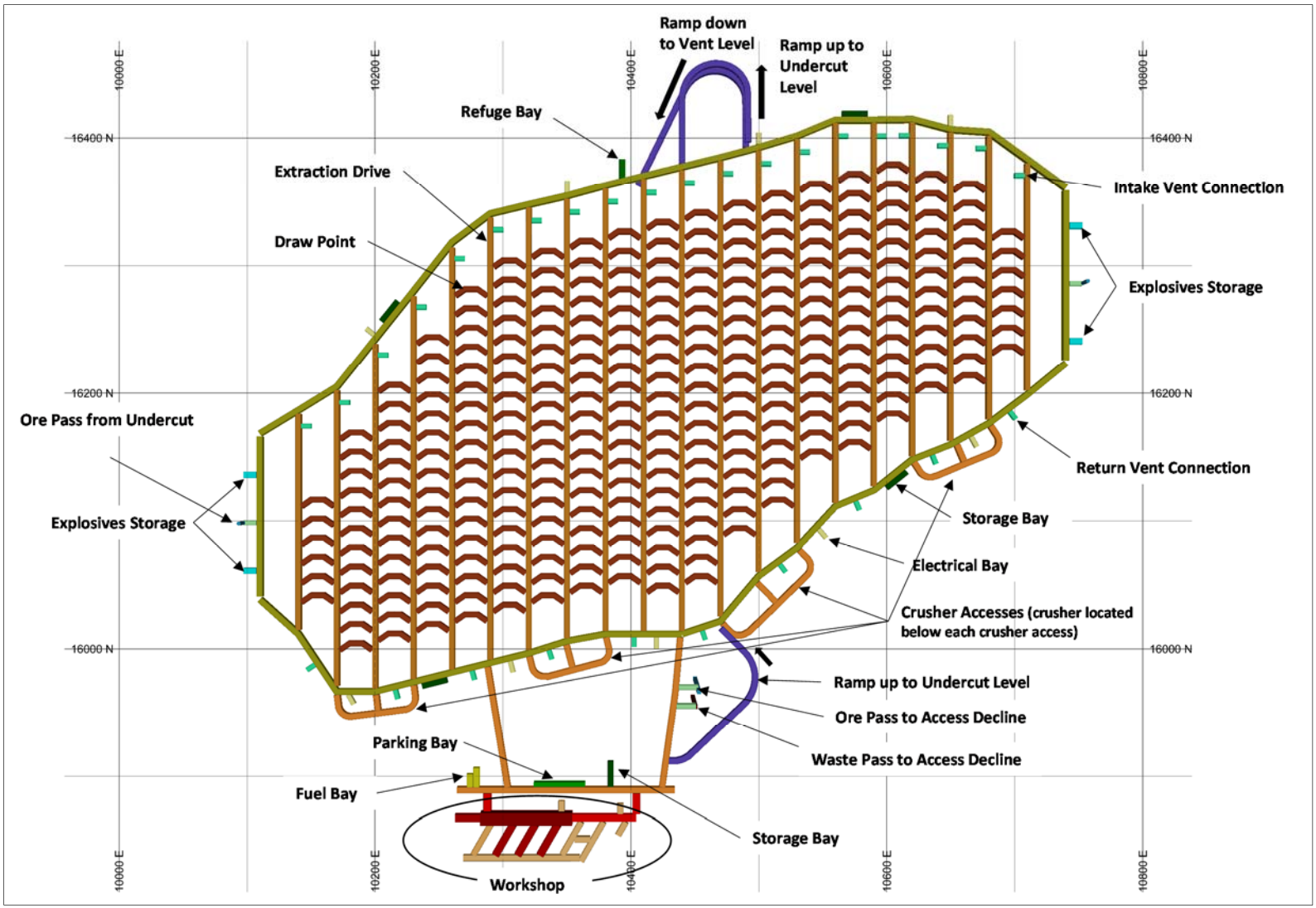


Figure 15.22: Extraction level layout

(Source: AuRico, 2016)

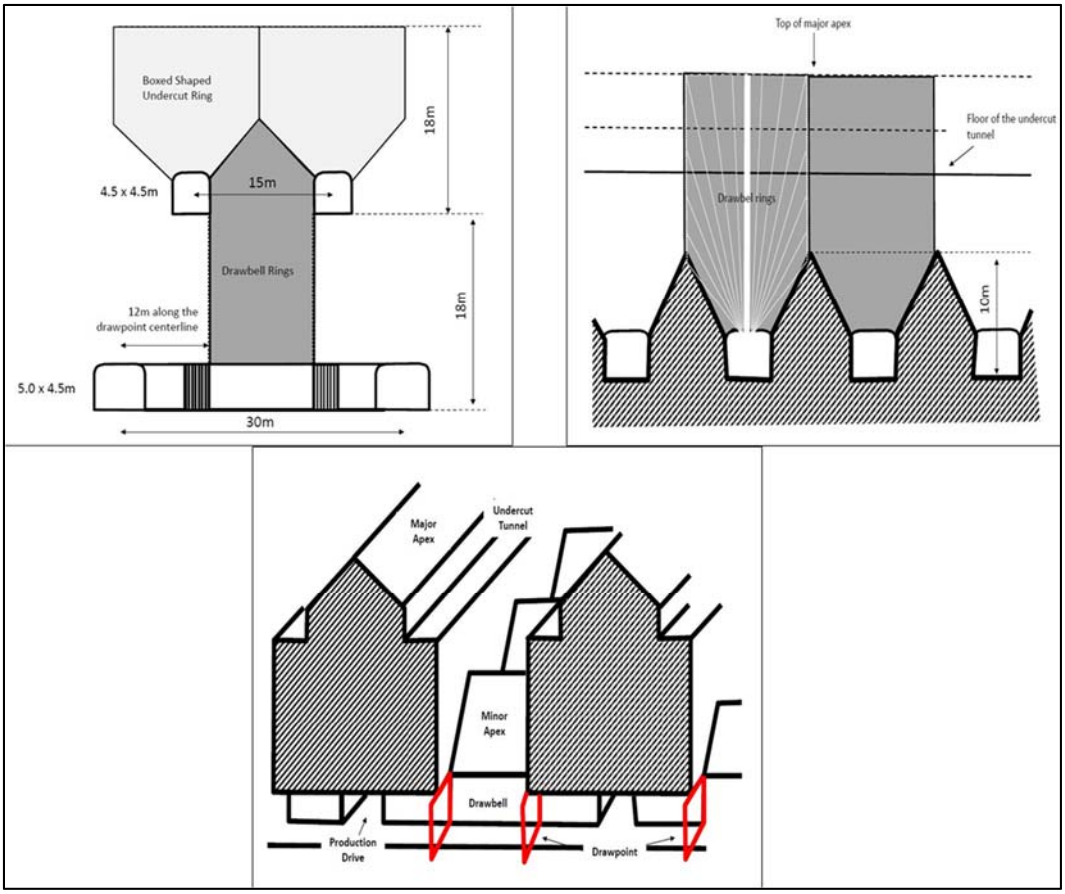


Figure 15.23: Drawbell geometry

(Source: SRK, 2016)

15.7.3 Undercut Level

The undercut level is located at the 1,158 mRL and is accessed via ramps from the extraction level, and is shown on Figure 15.24. There are two access ramps, one each north and south of the cave footprint. There are 38 undercut drives (4.5 mW x 4.5 mH) on 15 m centreline spacing.

North, south, east, and west rim tunnels (5.0 mW x 5.0 mH) provide 360° access to the cave footprint. In addition to these drifts providing access for development and undercutting personnel-equipment, they also provide an alternative access to the extraction level.

As per the extraction level, explosives and accessories (or cap) magazines are provided on both the east and west rim tunnels for easy access by development and undercutting personnel and equipment.

Since a refuge chamber is located on the north side of the extraction level, a refuge chamber is provided on this level just east of the south side ramp to the extraction level.

Ventilation to this level is provided via eight intake (3.0 m x 3.0 m) air raises located along the north rim tunnel. Air is routed to the return air level via ten return (3.0 m diameter) air raises situated just south of the south rim tunnel; with these raises also serving the extraction level.

Five electrical bays are provided along the south rim tunnel, and four are provided along the north rim tunnel, to facilitate both development and undercutting operations.

A total of four storage areas are provided, primarily for use during development.

The undercut will be taken to a height of 18 m above the floor of the undercut level as shown on Figure 15.24. This is relatively high and is recommended to avoid potential problems arising from oversize rocks in the drawpoints, associated with the low stress environment at KUG. There is approximately 12,850 t of undercut associated with each drawbell, totalling 3.739 Mt associated with all 291 drawbells.

Only 50% of in situ undercut tonnes will be mucked from the undercut level, with the remainder mucked with the underlying drawbells. This is to provide a buffer to protect against airblast risk.

15.7.4 Conveyor Level

The conveyor level is accessed via a ramp from the access ramp between the access decline and the extraction level, and is at the 1,116 mRL. Ore will be delivered to transfer conveyors on this level from four primary jaw crushers (55” x 42”) that are fed directly by production LHDs on the overlying extraction level. The location of the crusher chambers and alignment of the transfer conveyors needs to be finalized during the detailed engineering stage. SRK considered the current design to be acceptable in this study for the cost estimation and scheduling purposes with the condition that the design will be revised prior to the mine development.

Crushed ore is discharged from the transfer conveyors to a surge bin (1,000 t capacity) from which it is transferred to the underlying conveyor (CV-01) in the conveyor decline for transport to surface.

Intake air is provided to this level via an intake air raise at the east end of the level with connections to the return air level at each crusher chamber.

15.7.5 Ventilation Level

The ventilation level (Figure 15.25) is located at both the 1,120 mRL and 1,090 mRL (ungraded) for the intake and return sections respectively. However, as this level will also serve as the main drainage level, the low point will be at approximately 1,082 mRL with 1% gradient applied from the high point (which is the far northwest end of the intake air level).

The intake air level is initially accessed from the intake air decline and later also by a ramp from the north side of the extraction level. The two primary intake air fans are located on surface, transporting intake air to the north side of the cave footprint. Intake air is delivered to the undercut level via eight intake air raises (3.0 x 3.0 m) and to the conveyor level via 20 intake air raises (2.4 m diameter). The ventilation level drifts, both intake and return, are 6.0 mW x 5.0 mH.

The return air level is initially accessed from the conveyor decline by a cross-cut (4.5 mW x 4.5 mH) that provides access to the bottom of the primary return air raise (753 m long, 5 m diameter), primary return air fans, and return air level. The two primary return air fans are located in parallel drifts (8.0 mW x 8.0 mH). The return air level drift that feeds the primary return air fans is 6.0 mW x 5.0 mH, and is located on the south side of the cave footprint. Return air is delivered to this drift via 10 return air raises (3.0 m diameter) that connect to both the undercut and extraction levels.

The main dewatering facility is located at the low point of the return air level (1,082 mRL), which is in close proximity to the workshop. This facility comprises two sump chambers and a pump chamber, which are accessed from the conveyor decline. Given the elevation difference between intake and return sections of the ventilation level a bulkhead is not required where drainage water is transferred between these sections (to the south-east of the cave footprint).



Figure 15.24: Undercut Level Layout
(Source: AuRico, 2016)

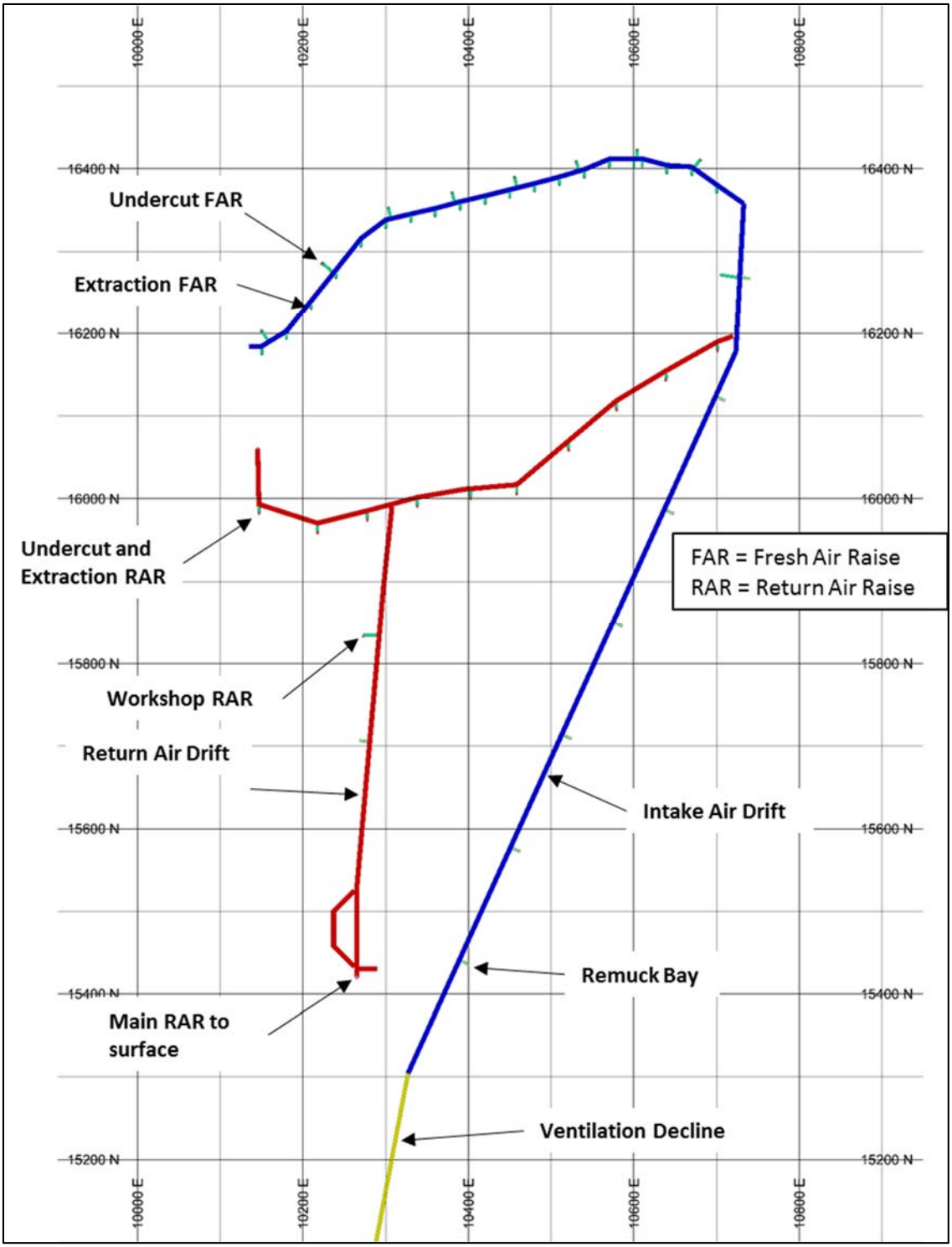


Figure 15.25: Ventilation Level Layout

(Source: AuRico, 2016)

15.8 Mine Schedule

This section discusses all aspects of the mine schedule, including development, construction, and production.

The combined mine development-production schedule was compiled using EPS (Enhanced Production Scheduler) software, a MineRP product. However, the cave production schedule was developed using PCBC™ software, which is recognised as industry standard for cave production scheduling; with PCBC™ schedule data transferred to EPS using a PCBC™-EPS data transfer plug-in developed by MineRP.

For this report, all mine schedule data is reported by year, with quarterly breakdown provided in the appendices to this report. Commercial production is defined as equivalent annual production being at least 60% of target production, with target production being 9.0 Mt/y or 25 kt/d based on 360-day per year operation.

15.8.1 Lateral Development & Construction

Development is scheduled to facilitate the timely ramp-up in production while ensuring infrastructure is in place to support this ramp-up. In conjunction, practical limitations associated with single and multi-heading development situations were recognised to ensure a realistic development schedule. Development is assumed to start in Q4 (i.e. 1 October) Year -4 in order to allow construction/development of the access road, access tunnel, establishment of Kemess Lake Valley facilities and triple decline portals construction ahead of development.

Table 15.12 summarises the lateral development schedule. There is a total 47,751 m of lateral development, which includes a variety of standard heading sizes as well as major excavations. Based on the stated heading and excavation sizes, lateral development equates to 2.923 Mt. For trucking calculations, 5% overbreak allowance is added to design lateral development tonnages.

Initial access to KUG is via triple declines, being an access decline, conveyor decline and intake air decline. The development rate over the initial six quarters averages 457 m/month, so two development face rigs (jumbo) will be required. While this period of development is effectively single heading, cross-cuts between declines (every 150 m to 300 m along their length), re-mucks, and passing bays will allow each jumbo to average approximately 229 m/mo. It is planned that owner development crews will be utilised for lateral development.

Table 15.12: Lateral Development Schedule

Item	Units	Total	Year -4	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Crusher Chambers	m	118	0	0	0	50	0	43	0	25	0	0
Sump Chamber	m	90	0	0	0	90	0	0	0	0	0	0
Pump Chamber	m	46	0	0	46	0	0	0	0	0	0	0
Workshop Gallery	m	72	0	0	42	30	0	0	0	0	0	0
Conveyor Decline (4.5 mW x 4.5 mH)	m	3,217	414	1,643	1,161	0	0	0	0	0	0	0
Decline/Ramp Level (5.5 mW x 5.5mH)	m	4,879	414	1,643	2,254	0	379	190	0	0	0	0
Drawpoint (4.5 mW x 4.5 mH)	m	8,238	0	0	0	1,031	1,996	1,771	1,251	1,353	835	0
Electrical Bay (4.0 mW x 4.0 mH)	m	170	0	5	12	47	36	24	12	26	9	0
Electrical Bay (5.0 mW x 5.0 mH)	m	18	0	0	18	0	0	0	0	0	0	0
Explosives Store (5.0 mW x 5.0 mH)	m	80	0	0	0	40	0	0	0	0	40	0
Extraction Level (5.0 mW x 4.5 mH)	m	5,884	0	0	0	1,671	1,265	1,081	939	758	170	0
Fuel Bay (5.0 mW x 5.0 mH)	m	40	0	0	40	0	0	0	0	0	0	0
Inner Rim (5.0 mW x 5.0 mH)	m	3,452	0	0	268	1,134	432	360	300	358	599	0
Intake Air Level (6.0 mW x 5.0 mH)	m	4,379	414	1,642	943	868	157	92	69	180	15	0
Miscellaneous (5.0 mW x 5.0 mH)	m	1,704	65	358	821	170	127	63	44	57	0	0
Outer Rim (5.0 mW x 5.0 mH)	m	0	0	0	0	0	0	0	0	0	0	0
Passing Bay (4.0 mW x 5.0 mH)	m	240	20	110	110	0	0	0	0	0	0	0
Refuge Bay (5.0 mW x 4.0 mH)	m	30	0	0	15	0	15	0	0	0	0	0
Remuck (4.0 mW x 5.0 mH)	m	459	20	110	220	60	29	0	0	0	0	0
Return Air Level (6.0 mW x 5.0 mH)	m	1,350	0	0	1,015	167	0	168	0	0	0	0
Storage Bay (4.5 mW x 4.5 mH)	m	179	0	0	20	65	15	0	59	20	0	0
Undercut Level (4.5 mW x 4.5 mH)	m	11,947	0	0	0	2,736	2,634	2,443	1,839	1,527	768	0
Vent Raise Connect (4.0 mW x 4.0 mH)	m	752	0	0	17	267	130	106	68	118	45	0
Workshop Access (6.0 mW x 5.0 mH)	m	80	0	0	80	0	0	0	0	0	0	0
Workshop Bay (6.0 mW x 6.0 mH)	m	103	0	0	53	50	0	0	0	0	0	0
Workshop Store (5.0 mW x 4.2 mH)	m	196	0	0	196	0	0	0	0	0	0	0
Workshop Weld Bay (5.0 mW x 4.5 mH)	m	28	0	0	0	28	0	0	0	0	0	0
Total Lateral Development	**m**	**47,751**	**1,347**	**5,510**	**7,331**	**8,503**	**7,214**	**6,343**	**4,582**	**4,420**	**2,501**	**0**

In Q7 (i.e. Q2 Year -2), the average monthly development increases to 555 m, requiring a third jumbo to be utilised. In this quarter, development of the intake and return air levels commences, effectively providing two additional high priority headings.

In Q8 (i.e. Q3 Year -2), the average monthly development rate increases to approximately 675 m/month with a fleet of three jumbos maintained until Q19 (Q2 Year 2). In addition to continuation of triple decline and ventilation level development, quarter eight sees development from the access decline to the extraction (1,140 mRL) level. The latter part of this quarter sees initial development of the main dewatering excavation.

The installation of major underground infrastructure is expected to be completed by contractor personnel, using special equipment where necessary, right after completion of bulk excavations. The following bulk excavations are developed from quarter eight:

- The main dewatering infrastructure, comprising twin sumps and a pump chamber, is excavated in Q8 to Q10, with construction-installation occurring immediately thereafter.
- The workshop is situated on the extraction level, and is excavated in Q9 to Q10, again with construction-installation occurring immediately thereafter.
- Excavation of the first two (east side) primary jaw crusher chambers from Q10 (Q1 Year -1); allowing the remainder of Year -1 for crusher installation ahead of cave production being crushed from the start of Year 1.

As can be seen, the period comprising Q8 to Q10 has somewhat intensive bulk excavation requirements, with an intense period of infrastructure construction-installation in the following months. This period also sees the main primary return air raise completed, with this raise developed over Q7 to Q9. Primary fan installation will occur in Q9 to Q11.

As previously mentioned, access to the extraction level is developed in quarter eight. In quarter nine, development of the southern rim tunnel on the extraction level commences. In addition, access to the undercut level is developed in Q9. Q10 (Q1 Year -1) sees initial undercut level drift development followed by initial extraction level drift and drawpoint development in Q11. This development is focussed to the east of the footprint as this is where the high grade core of the KUG orebody is located. This development is facilitated by the intake and return air level drifts being in close proximity from Q10 (to the south-east of the footprint), thereby providing through ventilation to the undercut and extraction levels.

From Q20 (Q3 Year 2), average monthly development decreases such that two jumbos are needed through the remainder of Year 2 to lateral development completion in Year 5; with both jumbos planned to be owner operated. As expected, the vast majority of lateral development over this period is within the footprint of the cave with multiple headings providing for high jumbo productivity.

The third and fourth primary jaw crusher excavations occur from Q20 (Q3 Year 2) and Q28 (Q3 Year 4) respectively. These are scheduled to be completed and primary crushers installed ahead of their required use.

Overall, initial development is scheduled to provide timely access to the KUG orebody. Once the cave footprint area is accessed, development is scheduled to allow an average five drawbells per month to be opened along with the associated (overlying) undercutting; adhering to the PCBC™-derived drawbell opening sequence. Following the push for production ramp-up, development slows to a rate required to match the cave production schedule derived using PCBC™ cave scheduling software. Figure 15.26 shows the lateral development metres by quarter.

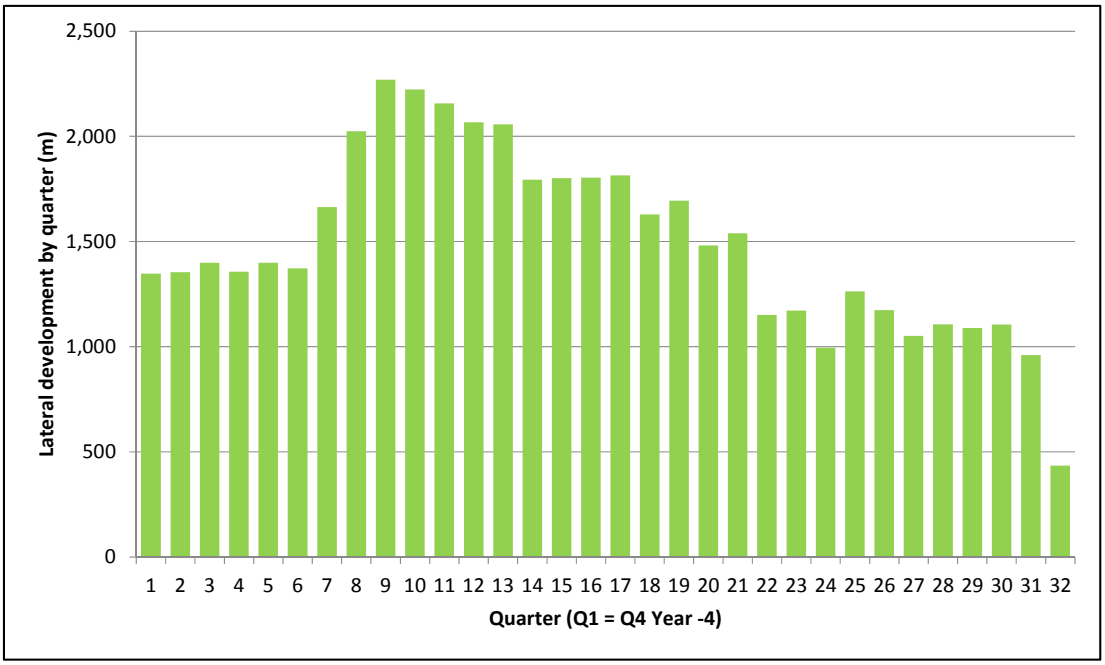


Figure 15.26: Lateral development by quarter

(Source: SRK, 2016)

Development rates are assigned to lateral development activities based on their relative priority, reflecting the importance of accessing the KUG cave in a timely manner. This is of critical importance to overall project economics due to the relatively long development lead time.

Highest priority headings include the triple declines, inter-level ramps, rim tunnels (on the extraction and undercut levels), and the intake-return air level; all of which are scheduled at 4.5 m/day or 135 m/mo. It is of critical importance to the mining schedule that these headings are prioritised, with all considered critical path activities.

Intermediate priority headings include triple decline cross-cuts, initial extraction drives, and initial undercut drives; all of which are scheduled at 3.0 m/day. The most eastern extraction and undercut drives are included in this group as their development is required to commence undercutting, drawbell opening, and (ultimately) cave production; covering extraction drives 1 to 5, and undercut drives 1W to 5E-5W.

Lowest priority headings include all remaining undercut and extraction drives, drawpoints, electrical bays, stores, passing bays, refuge bays, re-mucks, and ventilation raise cross-cuts; all of which are scheduled at 2.0 m/day.

Bulk excavation is scheduled at 100 m^3/d, with crusher chambers, dewatering infrastructure, and workshop scheduled using this rate. For a 6 mW x 6 mH heading this rate equates to an advance rate of less than 3 m/day.

15.8.2 Vertical Development

Table 15.13 summarises the vertical development schedule. There is a total 2,196 m of vertical development comprising raisebore (5.0 m diameter), drop raise (2.4 m to 3.5 m diameter) and Alimak raise (3.0 m x 3.0 m). Based on the stated heading and excavation sizes, raise development equates to 70 kt. For trucking calculations, 5% overbreak allowance is added to design raise development tonnages.

As previously mentioned, development of the primary ventilation return air raise occurs from Q7 to Q9. This is considered important in order to ensure the main primary ventilation circuit is available in Q10 when the intake and return air levels converge in the south-east area of the footprint. Installation of the primary return fans can occur in Q8 and Q9 as the ventilation level layout (on return air side) allows this to occur independently of ongoing raise and ventilation level development. The primary intake fans can be installed earlier as they are located on surface adjacent to the intake air decline portal.

There are 20 intake air raises, each 16 m long, planned between the intake air level and extraction level, and eight 34 m long intake air raises planned between the intake air and undercut levels. In addition, there are ten 64 m long return air raises planned between the undercut-extraction levels and return air level. The eight 34 m long intake air raises and ten 64 m long return air raises will be developed by the Alimak raising method. The 16 m long intake air raises are planned to be developed by drop raise. These raises are developed as the various levels reach the location of each raise, resulting in raise development occurring in all years from Year -2 to Year 5.

Provision has also been made for 3.0 m x 3.0 m Alimak waste and ore passes between the undercut and extraction levels, with truck loading chutes located at the bottom of the underlying access decline. The development schedule has these passes developed from Q9 (Q4 Year -2) allowing their use for development muck handling ahead of the ore crushing-conveying system being available.

Development rates are assigned to raise development activities based on practical advance rates.

The primary return air raise from surface is scheduled at an overall average of 4 m/day for piloting and reaming, resulting in a duration of approximately six months for the 753 m long raise. This activity is critical path as its development and associated primary fan installation is required for development in the cave footprint area.

Drop raise and Alimak raise activities are both scheduled at an overall average 2 m/day. These activities are not critical path raising activities.

Table 15.13: Raise Development Schedule

Item	Units	Total	Year -4	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6
Raisebore	m	753	0	0	753	0	0	0	0	0	0	0
Drop Raise	m	522	0	0	3	197	78	76	46	62	59	0
Alimak Raise	m	921	0	0	133	272	150	126	80	80	80	0
Total Raise Development	**m**	**2,196**	**0**	**0**	**890**	**469**	**228**	**202**	**126**	**142**	**139**	**0**

15.8.3 Production

In the context of Kemess underground, production is primarily considered to comprise cave production (above the undercut). Ore is also derived from development, undercutting and drawbell opening with quantities associated with each source summarised in Table 15.14.

Total ore tonnes associated with the KUG Feasibility Study (FS) is 107.3 Mt.

It should be noted that 50% of undercutting tonnes is assumed to remain in place to mitigate airblast risk and mucked with the underlying drawbell.

In addition to the aforementioned ore tonnes, there is a total 2,976 kt waste comprising 1,443 kt development, 938 kt undercutting, and 596 kt in drawbells.

The cut-off between development ore and waste was set at C$6.0/t NSR which is the approximate total estimated cost associated with ore processing for this study.

All development, undercutting, drawbell opening, and production activities had NSR values (in C$/t terms) calculated within the block model. The block model script is presented in Appendix B. The following formula is a close approximation of the block model script and is based on the same input parameters.

NSR (C$/t) = [60.5 x Cu grade (%)] + [35.2 x Au grade (g/t)]

The 60.5 value applied to the Cu grade was derived using 91% Cu recovery, revenue of US$2.50/lb Cu or C$3.35/lb Cu, 96.5% payable Cu, 22% Cu in concentrate, US$80/t concentrate treatment charge, US$0.08/lb Cu refining charge, US$215/t concentrate transport charge, and 8% moisture content.

The 35.2 value applied to the Au grade was derived using 72% Au recovery, revenue of US$1,200/oz Au or C$1,596/oz Au, 97.5% payable Au, and US$4.56/oz Au refining charge. There is no associated transport and treatment charges as these are carried by the Cu.

It should be noted that not all the foregoing values were used for economic evaluation of the KUG project. The reader is advised to refer to Section 12 for details.

Table 15.14 summarises the production schedule by year.

Table 15.14: Annual Production Schedule

Description	Units	Total	Year -4	Year -3	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13
Undercut Ore	t	3,324,864	0	0	0	398,694	773,027	657,363	501,346	530,017	464,417	0	0	0	0	0	0	0	0
Drawbell Ore	t	1,722,946	0	0	0	192,598	410,350	334,652	262,075	279,986	243,286	0	0	0	0	0	0	0	0
Development Ore	t	1,550,802	0	0	223,802	377,953	244,737	237,634	163,053	165,457	138,166	0	0	0	0	0	0	0	0
Cave Ore	t	100,782,885	0	0	0	66,169	2,009,623	4,610,044	8,093,200	8,032,121	8,155,313	9,015,093	9,011,443	9,011,075	9,003,991	9,014,949	8,999,986	9,021,130	6,738,749
TOTAL Ore	t	**107,381,498**	**0**	**0**	**223,802**	**1,035,413**	**3,437,736**	**5,839,693**	**9,019,675**	**9,007,581**	**9,001,181**	**9,015,093**	**9,011,443**	**9,011,075**	**9,003,991**	**9,014,949**	**8,999,986**	**9,021,130**	**6,738,749**
Undercut Waste	t	938,031	0	0	0	102,119	265,496	271,619	136,880	161,917	0	0	0	0	0	0	0	0	0
Drawbell Waste	t	595,625	0	0	0	69,597	157,479	180,073	78,622	102,017	7,838	0	0	0	0	0	0	0	0
Development Waste	t	1,442,077	91,018	371,232	324,822	165,096	169,774	130,329	94,278	88,661	6,867	0	0	0	0	0	0	0	0
Cave Waste	t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
TOTAL Waste	t	**2,975,734**	**91,018**	**371,232**	**324,822**	**336,812**	**592,749**	**582,021**	**309,779**	**352,595**	**14,705**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
TOTAL Mine	t	**110,357,232**	**91,018**	**371,232**	**548,625**	**1,372,226**	**4,030,485**	**6,421,714**	**9,329,455**	**9,360,176**	**9,015,886**	**9,015,093**	**9,011,443**	**9,011,075**	**9,003,991**	**9,014,949**	**8,999,986**	**9,021,130**	**6,738,749**
Cu Grade	% Cu	0.27	0.00	0.00	0.14	0.26	0.34	0.34	0.34	0.32	0.29	0.26	0.25	0.25	0.25	0.24	0.23	0.21	0.20
Cu	t Cu	285,660	0	0	319	2,701	11,632	19,842	30,634	29,021	26,250	23,823	22,703	22,387	22,112	21,449	20,300	19,126	13,363
Au Grade	g/t Au	0.54	0.00	0.00	0.22	0.46	0.66	0.73	0.76	0.70	0.61	0.55	0.51	0.48	0.47	0.46	0.43	0.40	0.38
Au	oz Au	1,867,859	0	0	1,568	15,312	72,879	137,109	220,437	202,271	176,785	158,444	148,229	140,436	136,271	133,306	125,684	117,394	81,732
Ag Grade	g/t Ag	1.99	0.00	0.00	1.26	1.77	2.10	2.25	2.30	2.30	2.15	2.05	1.99	1.96	1.94	1.88	1.79	1.69	1.59
Ag	oz Ag	6,878,658	0	0	9,055	58,937	231,930	422,864	666,481	667,084	622,465	595,442	575,998	569,043	560,905	544,918	517,633	490,921	344,982
Ore NSR	C$/t	35.1	0.0	0.0	16.4	31.9	43.4	46.0	47.0	43.9	39.0	35.2	33.2	32.1	31.4	30.6	28.9	27.1	25.3
Waste NSR	C$/t	1.4	0.0	0.0	0.5	1.4	1.4	2.2	2.4	2.1	3.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0

Figure 15.27 shows the mix of ore tonnes by year. This graph clearly shows the significance of cave ore tonnes to KUG, representing 94% of total ore tonnes. Development, undercutting, and drawbell ore tonnes represent relatively minor ore proportions, with their significance greater during production ramp-up.

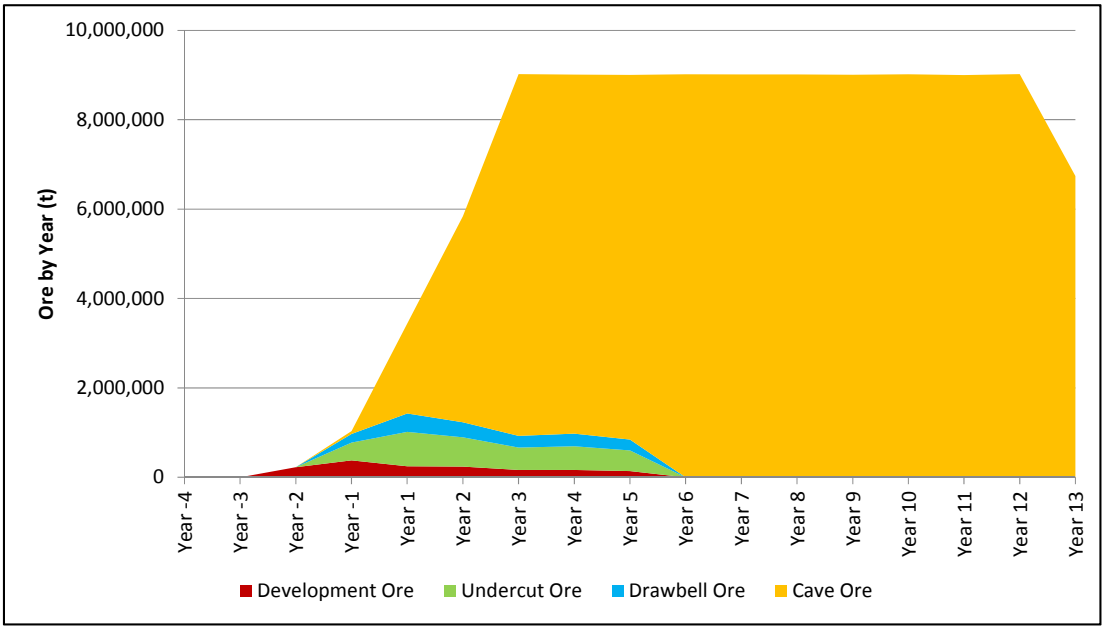


Figure 15.27: Ore production by year

(Source: SRK, 2016)

Tonnes and grades associated with undercutting and drawbell opening were derived from evaluating solids representing undercutting and drawbells associated with each of the 291 drawbells (or 582 drawpoints) against the KUG block model. As such, this volume of rock was excluded from cave production schedule runs using PCBC™ software. PCBC™ was then run against the optimised cave volume above the 1,176 mRL.

The PCBC™ schedule is based on five drawbells, accessed by ten drawpoints, being opened each month. The sequence of drawbell opening is from east to west between extraction drifts and from south to north along each extraction drift. This simplistic approach allows separation of development and undercutting-drawbell opening, and relatively straightforward scheduling.

As previously mentioned, initial development and production focus is to the east of the KUG orebody as this is the location of a higher grade/value core. This is shown by a plot of ore NSR values by quarter in Figure 15.28. Lower values in early quarters are associated with initial ore development to the south of the footprint. However, as development and related undercutting and drawbell opening pushes to the east of the footprint, average ore NSR values increase rapidly to peak at almost C$48/t in Q25.

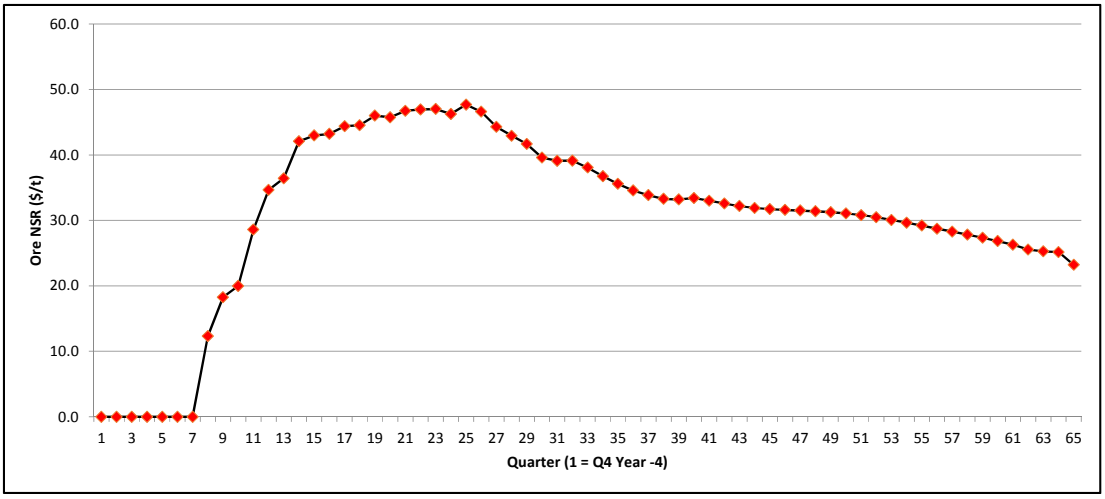


Figure 15.28: Ore NSR by Quarter

(Source: SRK, 2016)

From Table 15.14, it is seen that development ore is first mined in Year -2 which is sourced from the intake air decline/level, return air level, conveyor decline, access decline, and sump-pump chambers. Thereafter, as development moves into the footprint area, the amount of waste development decreases and ore development increases, with ore derived from all levels and excavations.

Undercutting commences in Year -1 and immediately builds to a steady state rate in excess of 200 kt/qtr. Only 50% of undercutting tonnes is mucked from the undercut level (as mentioned previously), so undercut mucking rates are 50% of these tonnages. It is planned that this tonnage will be transferred to the access decline via an ore pass located south of the footprint.

Drawbell opening also commences in Year -1 and immediately builds to a steady state rate in excess of 130 kt/qtr. In addition, 50% of the aforementioned undercut tonnage is mucked with this tonnage from the extraction level.

Overall, undercutting and drawbell development tonnage amounts to approximately 110 kt/month. This amount of material is required to be drilled, blasted and mucked.

Cave production commences in Year -1 (Q4) and builds over approximately three years to reach the overall target production rate of 9 Mt/y in Year 3; which includes 0.9 Mt in Year 3 from ore development, undercutting, and drawbell opening. In Year 4 and Year 5, cave production averages 8.1 Mt/y, with the cave contributing the full 9 Mt/y from Year 6. This rate is maintained for seven years until the end of Year 12, after which cave production declines due to depletion of draw columns. Final cave production is scheduled to occur in Year 13.

The following two issues associated with the cave production schedule are noted:

- A 110 Mt cave tonnage constraint was applied due to tailings storage limitations in the Kemess South Pit. If it proves possible (and profitable) to increase tailings storage capacity beyond 110 Mt, KUG mine life could be extended beyond Year 13.

- When PCBC™ runs occurred, deductions from the 9 Mt/y production target for development, undercutting, and drawbell opening tonnages were not accounted for (as the overall mine schedule was not available at that time). As such, the PCBC™ cave production schedule was adjusted in EPS to achieve the 9 Mt/y target. This was done by deferring cave production from all draw columns, starting with the earliest draw columns and working through the cave in date sequence.

Table 15.15 shows the production rate curve used in PCBC™ to derive the cave production schedule.

Table 15.15: Production Rate Curve

% Extraction	t/m^2/day	t/day	mm/day
0	0.10	23	37
10	0.25	56	93
20	0.35	79	130
30	0.46	104	170
100	0.46	104	170

This build-up in draw column productivity with increasing extraction is considered conservative, suggesting increased cave production is possible; particularly given cave breakthrough to surface after approximately 18 months from start of caving. This also increases cave production flexibility particularly with regards to blending different geometallurgical zones within the orebody.

15.8.4 Labour

Labour for the mining aspects of the KUG project is comprised of a mining contractor component and an owner-mining component. The contractor's scope includes underground infrastructure construction, raise development, and material haulage on surface. The owner's operations workforce is introduced in Year -4 and is responsible for all underground lateral development and production mining activities.

The labour schedule was prepared on a quarterly basis. An annualised summary of the on-site labour schedule is presented in Table 15.16. This table shows average personnel levels for each year, however maximum personnel levels for each year were used for the G&A cost estimate.

15.8.5 Mobile Equipment

The mobile equipment fleet for KUG is completely diesel-powered. The production LHDs will be operated in a conventional/manual (operator in cabin) arrangement and also a semi-autonomous (operator in control room) arrangement.

The mobile equipment schedule was prepared on a quarterly basis. The annualised summary of the mobile equipment schedule is presented in Table 15.17.

Table 15.16: Summary of Annual Labour Schedule

Labour Category	Rotation	Yr -4	Yr -3	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13
Owner																		
Operations Staff	4:3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Operations Staff	14:14	6	6	7	9	9	9	9	9	9	7	7	7	7	7	7	7	7
Operations Hourly	14:14	43	43	53	84	98	101	100	95	70	40	40	38	37	36	35	34	30
Maintenance Staff	4:3	3	3	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Maintenance Staff	14:14	4	4	5	5	5	5	5	5	5	5	5	5	5	5	5	4	4
Maintenance Hourly	14:14	13	13	23	31	34	34	34	34	32	29	28	28	28	27	27	27	26
Technical Staff	4:3	4	5	7	7	7	7	7	7	7	7	7	7	7	7	7	7	7
Technical Staff	14:14	4	4	10	11	11	11	11	11	11	9	9	9	9	9	9	9	8
Total Owner Labour		**80**	**81**	**109**	**154**	**171**	**174**	**173**	**168**	**140**	**104**	**103**	**101**	**100**	**98**	**97**	**95**	**89**
Contractor																		
Operations Staff	4:3	0	0	0	1	0	1	1	1	1								
Operations Staff	21:21	0	2	0	4	1	1	1	1	2								
Operations Hourly	21:21	3	5	8	17	7	8	6	6	5								
Maintenance Staff	21:21	0	0	0	0	0	0	0	0	0								
Maintenance Hourly	21:21	0	0	0	0	0	0	0	0	0								
Total Contractor Labour		**3**	**7**	**8**	**21**	**7**	**9**	**8**	**8**	**7**								
Maximum Labour on Site at the Same Time for Camp Size																		
Maximum Owner Labour		80	81	109	154	171	174	173	168	140	104	103	101	100	98	97	95	89
Maximum Contractor Labour		3	7	8	21	7	9	8	8	7								

Table 15.17: Summary of Annual Mobile Equipment Schedule

Owner Equipment Fleet	Model Version	Yr -4	Yr -3	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr -13
Jumbo, 2 boom	DD421	2	2	4	4	3	3	2	2	2	0	0	0	0	0	0	0	0
Production Drill	DL431	0	0	0	4	4	4	3	3	2	0	0	0	0	0	0	0	0
Aries ITH Drill with V30 Reamer	DU 411 - Aries	0	0	0	1	1	1	1	1	1	0	0	0	0	0	0	0	0
Emulsion Loader	MC 605 VE	1	1	2	3	4	2	2	2	2	0	0	0	0	0	0	0	0
Rockbolter	DS311	2	2	3	3	3	2	2	2	2	0	0	0	0	0	0	0	0
Cablebolter	DS421	1	1	2	2	2	1	1	1	1	0	0	0	0	0	0	0	0
Shotcrete Sprayer	SS3	1	1	1	1	1	1	1	1	1	0	0	0	0	0	0	0	0
Transmixer	TM3	1	1	2	3	3	2	2	2	1	0	0	0	0	0	0	0	0
Secondary Breaker, Water Cannon	WC-3	0	0	0	1	1	1	0	0	0	0	0	0	0	0	0	0	0
Secondary Breaker, Blockholer	BH-3	0	0	0	1	1	3	3	3	3	3	3	3	3	3	3	2	2
Secondary Breaker, Mobile Rockbreaker	RB3	0	0	0	1	2	2	2	2	2	2	2	2	2	2	1	1	1
Secondary Breaker, Splitter	DB120	0	0	0	0	1	2	2	1	1	1	2	2	1	1	1	1	1
Stationary Rockbreaker at Crushers	MRSTE35	0	0	0	1	2	3	3	4	4	4	4	3	3	3	3	3	3
LHD, 3.7 m3, 6.7 t	LH307	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
LHD, 8.6 m3, 17 t (manual)	LH517	2	2	3	6	6	6	6	5	4	1	1	0	0	0	0	0	0
LHD, 8.6 m3, 17 t (automated)	LH517	0	0	0	0	2	5	8	7	8	8	8	8	8	8	8	8	8
Haulage Truck, 50 t	TH550	2	2	4	9	4	3	3	3	2	0	0	0	0	0	0	0	0
AutoMine System for LHDs	AutoMine LHD	0	0	0	0	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader	Veekmas FG 7 C	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Scissor Lift	SL3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2	2
Boom Truck	BT3	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Flat Deck Truck	Toyota Landcruiser	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mechanics Truck	Toyota Hurth	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Fuel/Lube Truck	PC3	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Personnel Carrier (9 per.)	Toyota Van	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Supervisor/Engineering Vehicle	Toyota	2	5	7	8	8	8	8	8	8	7	7	7	7	7	7	7	7
Septic Vaccuum Truck	0	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Forklift/Telehandler	535-140	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Personnel Carrier (16 per.)	FL3	2	2	2	3	4	4	4	4	3	2	2	2	2	2	2	2	2
Cassettes System Prime Mover	Multimec MF 100	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Contractor Equipment Fleet	**Model Version**	**Yr -4**	**Yr -3**	**Yr -2**	**Yr -1**	**Yr 1**	**Yr 2**	**Yr 3**	**Yr 4**	**Yr 5**	**Yr 6**	**Yr 7**	**Yr 8**	**Yr 9**	**Yr 10**	**Yr 11**	**Yr 12**	**Yr -13**
Surface Loader, 7 m3	Cat966	1	1	1	1	1	1	1	1	1								
Surface Truck, 41 t	Cat745C	2	2	3	4	3	3	3	2	2								
Scissor Lift					1		1	1	1	1								
Boom Truck			1		1		1	1	1	1								
Flat Deck Truck			1		1		1	1	1	1								
Supervisor/Engineering Vehicle			1		2		2	2	2	2								
Forklift/Telehandler			1		1		1	1	1	1								

15.9 Mine Infrastructure

15.9.1 Access Road & Tunnel

As discussed in Section 5.6.1, the KUG deposit is accessed via triple decline tunnels with portals located at the north end of the Kemess Lake Valley (KLV) just south of the watershed divide with the Amazay Lake Valley. The existing Kemess South Mine infrastructure will be connected to the portals via an access and utility corridor along the west side of the open pit to the ridge above the KS open pit and then down to the head of the Kemess Lake Valley using a short access tunnel to maintain conveyor design grades. The corridor provides both access to the decline portals as well as routing for the overland conveyor system, 25 kV power cable, and underground dewatering pipeline.

The corridor will serve as the main access route to the portals throughout KUG mining operations. During the development period the corridor will also serve as a haul road for transportation of waste rock (and ore produced during initial development).

Although primary access to the portals will be provided by the corridor, a secondary (emergency) access route is also available using the existing exploration roads that traverse the hillside directly north of the Kemess South open pit. These roads will be used for construction access to initiate clearing, grubbing, and portal construction activities, eliminating the need for a pilot road for much of the alignment.

The corridor is roughly 5 km in length measured from the existing live ore stockpile at the north end of the existing Kemess South process plant to the face of the KUG triple decline portals at the north end of the Kemess Lake Valley.

Figure 15.29 shows the general layout of the access corridor.

15.9.2 Kemess Lake Valley Infrastructure

The area along the access road between the decline portals and access tunnel will be used to house the main mine development and construction offices and laydowns as well as waste rock and ore stockpiles from underground development. The contractor laydowns will be used for staging underground equipment to support the development and construction activities as well as the active mining operations. Also incorporated into the KLV infrastructure are a propane tank farm for heating of underground ventilation, sedimentation pond and underground dewatering transfer pond, diversion ditches above the portals, and conveyor transfer tower and conveyor elevated sections which are detached from the main portal access road area.

15.9.3 Ventilation

Ventilation system design for the KUG Mine was carried out by Mine Ventilation Services (MVS) engineers using VnetPC Pro+ ventilation system modeling software. This section provides a summary of their work.

The ventilation system design for the KUG Mine includes two primary ventilation circuits that are kept completely separated from each other.

Approximately 400 m^3/s has been estimated to meet the ventilation needs for the KUG Mine; airflow requirements were derived from estimating air flow based on 0.06 m^3/s per kW of rated engine power. It is expected that EPA Tier IV engines will be utilised with low sulphur diesel.

The haul/convey ventilation circuit draws approximately 70 m^3/s of fresh air down the access decline, which splits at the bottom of the ramp; approximately 40 m^3/s provides ventilation for the shop and is exhausted through a short raise down to the exhaust sub-level, while the remainder is split between the bottom of the ore passes and the sumps. That air travels up the conveyor decline prior to being exhausted at the portal. The fan installation for this circuit is located in a bulkhead located at the portal of the conveyor decline.

The production ventilation circuit is fed from the primary fan installation at the top of the intake air decline, which leads directly to the Intake Air Level. Two 1,000 hp fans operated in parallel at 3.2 kPa will provide approximately 345 m^3/s of air to the production circuit. Fresh air on the intake air level is split between the conveyor, undercut, and extraction levels, respectively.

A series of 2 m diameter raises connect from the intake air level up to each extraction drive while 3 m X 3 m raises provide fresh air to the undercut level. Each of the production raises will be controlled via a “butterfly” regulator located on the extraction level and a drop-board regulator on the undercut level. Extraction drives will also have a door located at the northernmost end of the drive that will control airflow into/through the North Rim Drift.

For active extraction drives, the intake raise regulator is opened until the desired airflow is achieved (this can be determined based on the airflow quantity sensor that will be installed in the drive). When the drive is inactive, meaning that the production LHD is moving to another drive, the loader operator can close the intake regulator before leaving the area and open the regulator at the destination

The active drives on the extraction level are provided with a minimum of 20 m^3/s. The first drive is adjusted to 50 m^3/s with the door left open in order to ventilate the rim drift with sufficient ventilation to allow for transportation of equipment around the extraction level. Once leaving the drives on the southern side of the level, air is collected in the South Rim Drift and exhausted through the exhaust raises located at the crushers. These exhaust raises transport air down to the exhaust air level, where it is then carried out of the mine through the primary exhaust fan installation and the dedicated 5 m diameter exhaust raise to surface. The two 1,000 hp exhaust fans remove approximately 425 m^3/s of air from the mine at 2.7 kPa. Locating the primary exhaust fans underground will require permission from the BC Mining and Minerals Division.



Figure 15.29: General Layout of the Access Corridor
(Source: AuRico, 2016)

The undercut level is ventilated with fresh air from the most recently excavated intake raise. Air is distributed to the working areas via auxiliary ventilation systems (fans and ducts), and then removed through the most recently excavated exhaust raise directly to the exhaust level.

The four crushers are each ventilated via a regulator in the exhaust raise access drift. These regulators can be adjusted as necessary to provide the optimal ventilation flow in the crusher area and in the South Rim Drift.

There will be three primary fan installations at the KUG Mine. A single fan installed in a bulkhead at the top of the conveyor decline ensures that air will flow from the crusher up to the portal in the same direction as the ore throughout the length of the conveyor (homotropal flow). Two fans located in parallel will be located at the top of the intake air decline in order to push fresh air into the mine through the intake air decline. Parallel exhaust fans located at the bottom of the exhaust raise will exhaust the air from the mining (production circuit) as well as from the access decline and workshop. The primary intake and exhaust installations will consist of two identical fans installed in parallel for further operational flexibility. During development of the declines, and before the exhaust circuit through the primary exhaust fans is established, a single intake fan will be run to provide the necessary ventilation for development activities. The use of parallel fan installations at the KUG Mine will also allow the operation of a single fan when production, maintenance or operational demands require it, or in the event of an unplanned fan outage. Note that in the case of the primary intake and exhaust fan installations, a change in the operation of one fan will require adjustment of the opposite installation in order for the system to remain balanced.

Ventilation modeling of auxiliary ventilation fan and duct combinations were performed using DuctSIM software. For the ventilation of the triple decline development, 1 m diameter flexible (blowing bag) duct was paired with 75 kW fans in parallel (two fans/ducts per entry). This combination meets the demands of the decline development as well as other development requirements of the mine once the bottom of the decline is reached. The recommended combination of fan and duct is capable of providing 20 m^3/s to active faces up to 1,200 m away at operating pressures that range from 2-3 kPa. The airflow quantity will vary based on the length of the ducts, from over 30 m^3/s within 350 m of the fans to 20 m^3/s at 1,200 m distant.

The following types of ventilation infrastructure are recommended to be installed as part of the ventilation system design for the KUG Mine.

- Vane-axial fans,
- Fan monitoring sensors,
- Air quality sensors,
- Air quantity sensors,
- Psychrometric sensors,
- Thermal imaging sensors, and
- Automated equipment doors.

For the KUG Mine, the most practical and cost-effective choice for air heating are propane-fueled, direct air heaters. MVS engineers performed several calculations to determine the amount of air heating required at the mine based on the average monthly temperatures reported by AuRico. The heat required for the intake air decline is based on the expected temperature rise necessary to heat the ambient air to -5 °C prior to its introduction underground while operating at or close to maximum capacity (i.e., after decline development has been completed and both fans are operating). This also provides sufficient heating capacity to provide +2 °C air to the declines during development operations. Air in the Access Decline will be heated to +2 °C for the duration of the mine life.

Figure 15.30 depicts an isometric view of the ventilation model schematic for the KUG Mine.

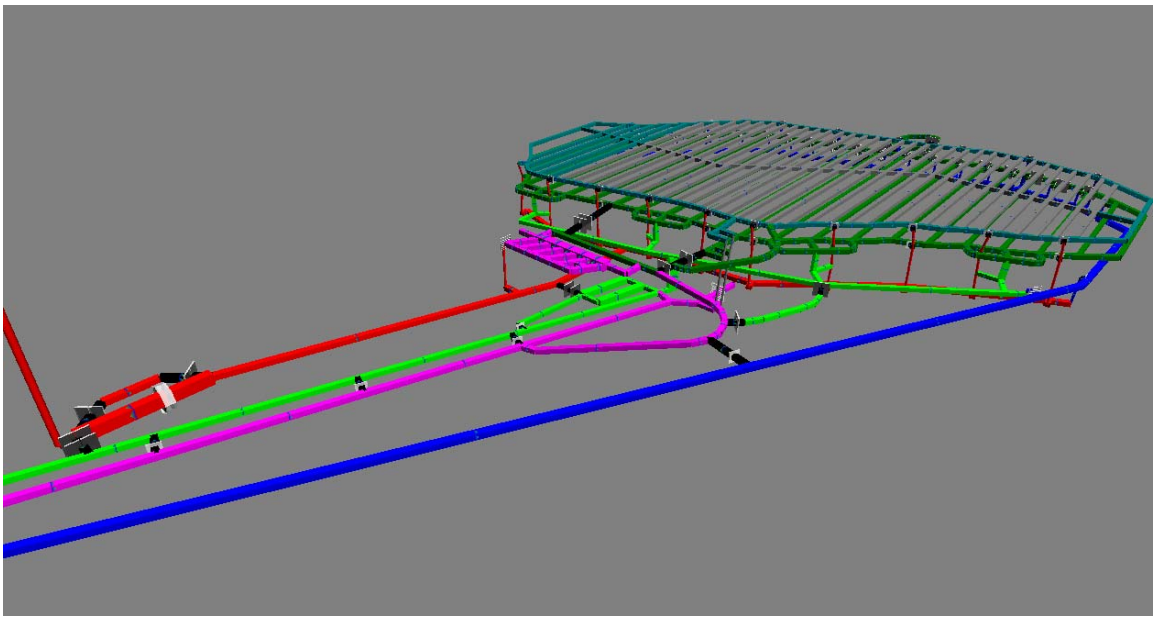


Figure 15.30: Ventilation model schematic for the KUG mine (isometric view)

(Source: AuRico, 2016)

15.9.4 Primary Crushing

Four single-toggle jaw crushers will be located underground at KUG on the south side of the caving footprint. Each jaw crusher will be 55” x 42” (1.4 m x 1.07 m) with 200 kW of installed power, equivalent to a Metso model C150. Based on manufacturer’s capacity tables, each crusher is capable of processing 375 tph to 750 tph, depending on a closed-side-setting between 5” and 8” (125 mm and 200 mm).

For an expected average ore production of 9 Mt/y, this is equivalent to 24,600 tpd and at 75% availability, a total of 1,367 tph (or 342 tph per crusher). Topsize to each crusher is expected to be 1 m and controlled with a 1 m square opening, scalping grizzly which will bypass any oversize material for secondary breaking. A stationary rock breaker will be installed adjacent to the crusher to break oversize material.

Grizzly screen undersize will pass to the crusher feeders which will screen out any -150 mm fines with vibrating feeders. It is expected that around 30% of the crusher feed will be +150 mm and need to be crushed. As the cave footprint develops, between two and four crushers will be operating at one time, but will have sufficient capacity to achieve 9 Mt/y even with two operating. The jaw crusher size was selected on the maximum feed size and is oversized in terms of capacity and installed power.

It is assumed that three jaw crushers would be operating during full commercial production. Each crusher will be operated 333 days per year assuming one maintenance shift per week plus six days additional maintenance downtime.

With a closed-side-setting of 150 mm, it is expected the product size will have an 80% passing (P80) size of 165 mm with a P20 of 10mm. It is recognised that a jaw crusher has a tendency to pass slabby material with a coarser product size than a gyratory crusher operating at the same setting. Due to the excess jaw crusher capacity, it is expected that each crusher will operate with a tighter setting – for example, 125 mm or possibly 100 mm.

Ore from the extraction level development, drawbell blasting and initial caving will be mucked directly by 8.6 m3 manually-operated diesel LHDs to the nearest crusher located outside of the production footprint. Ore from the undercut level development and undercut blasting will be mucked to the ore passes located near a crusher. This ore will be re-mucked at the bottom of the ore passes on the extraction level and dumped into a crusher.

Two transfer conveyors will be used to convey crushed ore to a surge bin located above the conveyor decline. The surge bin will have a live capacity of approximately 1,000 t. There are no other surge piles/bins provided in the crusher-conveyor system. Ore from the surge bin will be transferred to the underground conveyor (CV-01) via a belt feeder. Crushed ore will then be conveyed to a crushed ore stockpile near the existing processing facility via a 3.21 km long decline conveyor and 4.93 km long overland conveyor.

Other equipment associated with each crusher installation include an 80 t capacity overhead crane (for installation and maintenance), dust collector and control room.

Ventilating air for the crusher chambers will be exhausted directly to the return air level via a dedicated connection to the return air raises.

15.9.5 Conveying

Conveyor system design for KUG was carried out by Conveyor Dynamics, Inc. (Bellingham, USA) using proprietary modeling software. This section summarizes the overall conveyor system.

A site plan showing the system appears in Figure 15.31. After conveyor CV-01 exits at the conveyor decline portal, the surface conveyor CV-02 negotiates the surface terrain by passing through a tunnel before skirting to the west of the existing KS open pit void to discharge into Kemess South's former open pit truck dump hopper. From there it is conveyed into the ore processing facility using the existing Kemess South infrastructure.

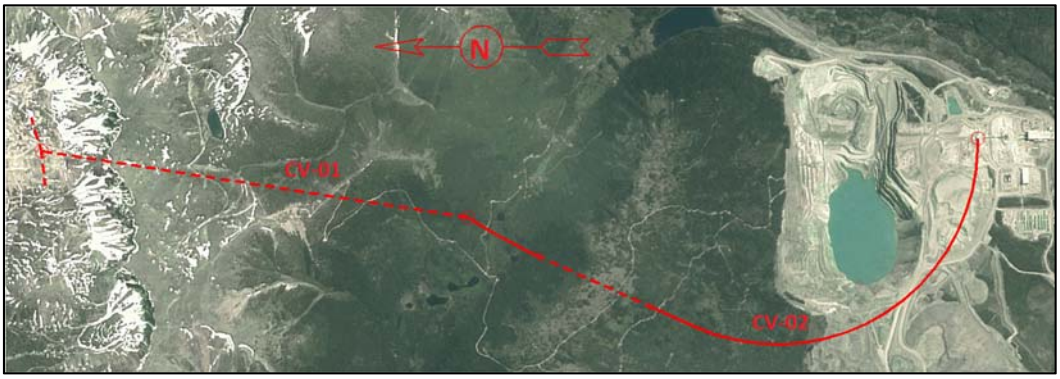


Figure 15.31: Conveyor system site plan

(Source: CDI, 2016)

The conveyor system flow sheet (Figure 15.32) comprises of a discharge conveyor from each jaw crusher, underground transfer conveyors running East/West, an underground ore storage bin (1,000 t capacity), belt feeder, an underground decline conveyor CV-01, a transfer station, and a surface conveyor CV-02.

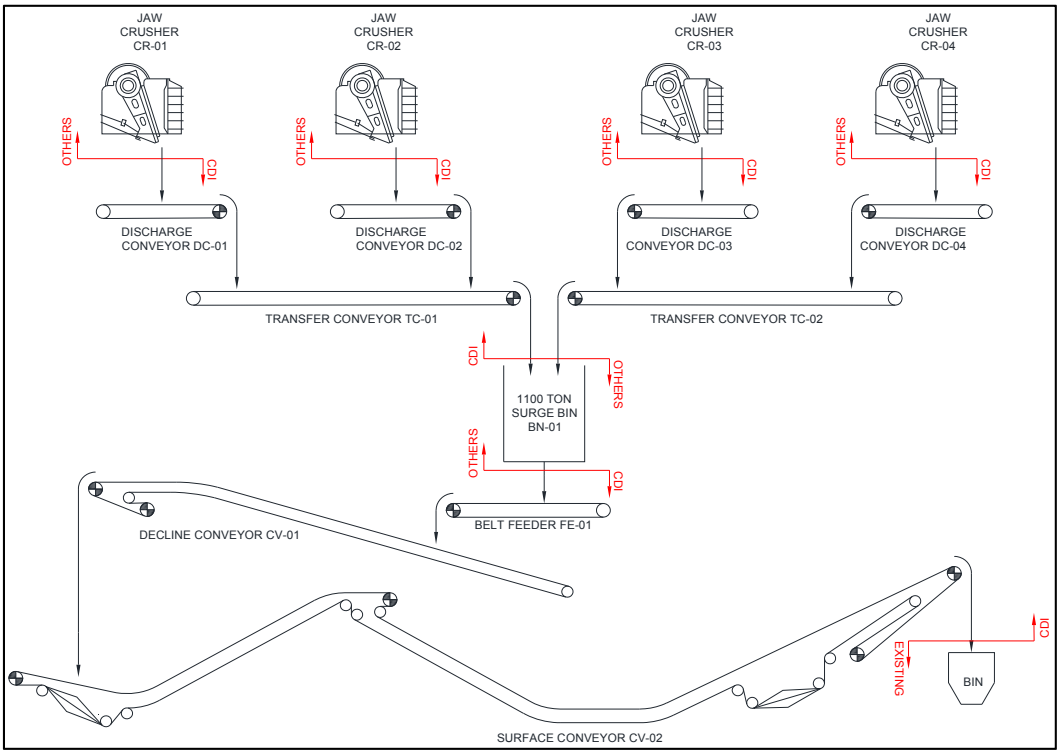


Figure 15.32: Conveyor system flow diagram

(Source: CDI, 2016)

The conveyors will operate continuously with scheduled minor maintenance generally required every two weeks for a single shift. Major scheduled maintenance shutdowns are typically required bi-annually.

To deliver 9.0 Mt/y, CDI has assumed the conveyor system will operate on the following design basis. Conveyor system operating basis is shown in Table 15.18.

Table 15.18: Conveyor System Operating Basis

Operating Basis Summary	Tonnes per hour	Hours per year	Mtonnes per year
Operating hours at 100% capacity	1,627	2,370	3.86
Operating hours at 80% capacity	1,302	3,950	5.14
Operating hours at 0% capacity	0	1,580	0.00
Total		**7,900**	**9.00**

Crusher Discharge Conveyors CD-01/02/03/04

The crusher discharge (CD) conveyors are located within each of the jaw crusher stations to accept ore from the grizzly underflow and the primary jaw crusher discharge. Each crusher discharge conveyor is approximately 20 m long and rises 2.5 m to discharge onto the transfer conveyor at a design capacity of 1,500 t/h. A tramp iron magnet is located above the discharge pulley to remove any ferrous tramp material.

The CD conveyors use a 1,400 mm (55”) wide, 45 degree trough, ST-1000 N/mm belt that operates at 1.5 m/s and is powered by a single 75 kW (100 hp) six-pole motor with a traction fluid coupling. Conveyor tension is achieved using a fixed hydraulic take-up located at the tail.

Transfer Conveyors TC-01/02

The transfer conveyors (TC) (Figure 15.33) receive ore from the crusher discharge. Transfer conveyor TC-01 covers the Eastern side of the ore body and gathers ore from CD-01 and CD-02. Similarly, TC-02 covers the Western side of the ore body and gathers ore from CD-03 and CD-04.

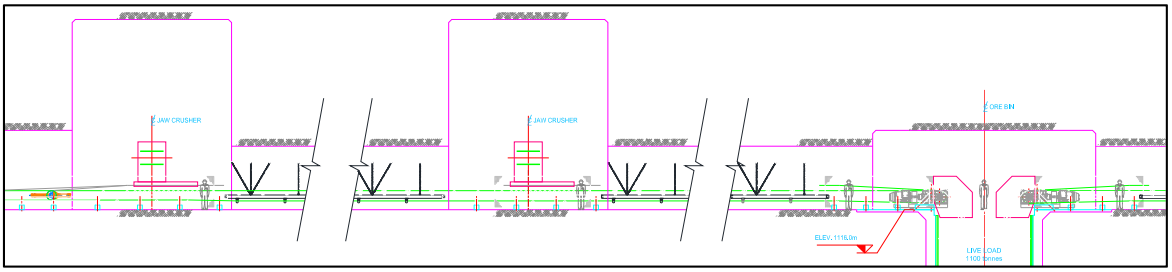


Figure 15.33: Transfer conveyor arrangement

(Source: CDI, 2016)

Each transfer conveyor is aligned straight and horizontal, being approximately 260 m long to discharge into the underground ore storage bin BN-01 at a design capacity of 1,500 t/h. A tramp metal detector is located after the last feed point to detect any tramp material that was not extracted by the upstream tramp magnets. Belt weightometers are located after each of the two load points for upstream process control.

The TCs use a 1,400 mm (55”) wide, 45 degree trough, ST-1000N/mm belt that operates at 2.0 m/s and is powered by a single 75 kW (100 hp) six-pole motor with a traction fluid coupling. Conveyor tension is achieved using a gravity take-up located at the tail.

Belt Feeder FE-01

The curved chute that discharges onto the underground conveyor (CV-01) will help prevent puncture damage and reduce belt wear. This feeder will be powered by a 200 kW hydraulic motor and operates at 0.35 m/s. The belt feeder is shown in Figure 15.34.

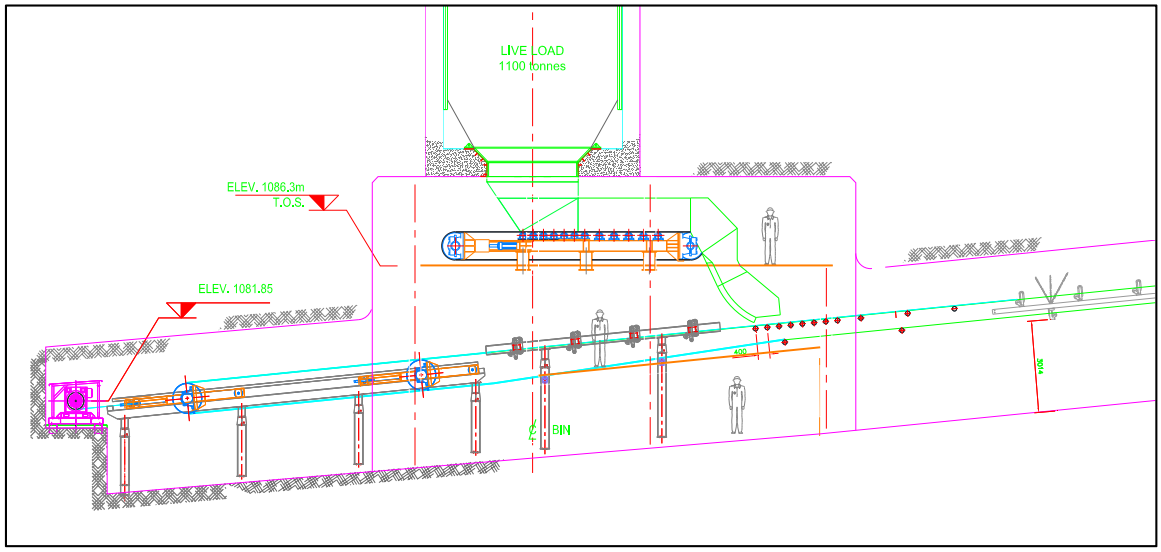


Figure 15.34: Belt feeder and CV-01 tail

(Source: CDI, 2016)

Decline Conveyor CV-01

The underground decline conveyor, CV-01, hangs from the roof of the tunnel by chains (Figure 15.35). It follows a straight +9.5% (uphill) alignment from below the belt feeder. This conveyor is 3,209 m long and gains approximately 305.5 m in elevation.

The decline conveyor CV-01 uses a 1,067 mm (42”) wide, 45 degree trough, ST-3800N/mm belt that operates at 2.9 m/s and is powered by two 1006 kW (1350 hp) six-pole motors with variable voltage, variable frequency (VVVF) drive control (one on the primary pulley and one on the

secondary pulley). The high tension pulley shell diameter is 1,250 mm, and the low tension pulley shell diameter is 916 mm. Conveyor tension is achieved using a winch take-up located at the tail.

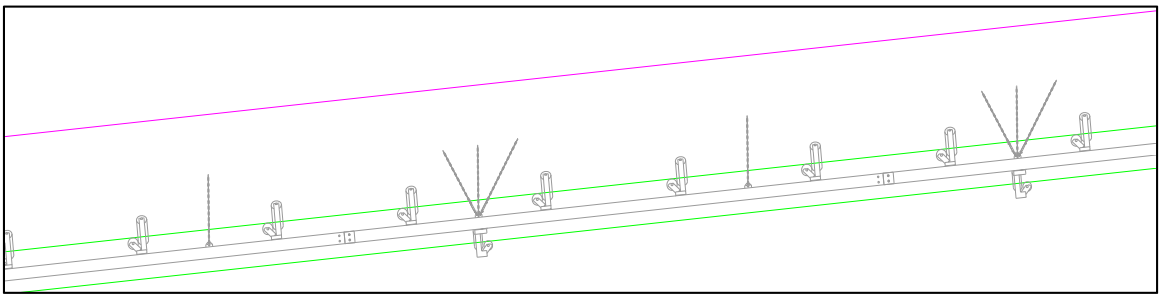


Figure 15.35: Conveyor CV-01 tunnel stringer

(Source: CDI, 2016)

Surface Conveyor CV-02

The surface conveyor, CV-02, leaves the transfer station and crosses the Kemess Lake Valley before entering the access tunnel on the south side of the valley. CV-02 exits the access tunnel on the west side of the Kemess South open pit, with CV-02 then wrapping around the south-west corner of the Kemess South open pit before discharging in the existing Kemess South open pit truck dump (Figure 15.36). From there, the material is stockpiled by the existing Kemess South stacker conveyor.

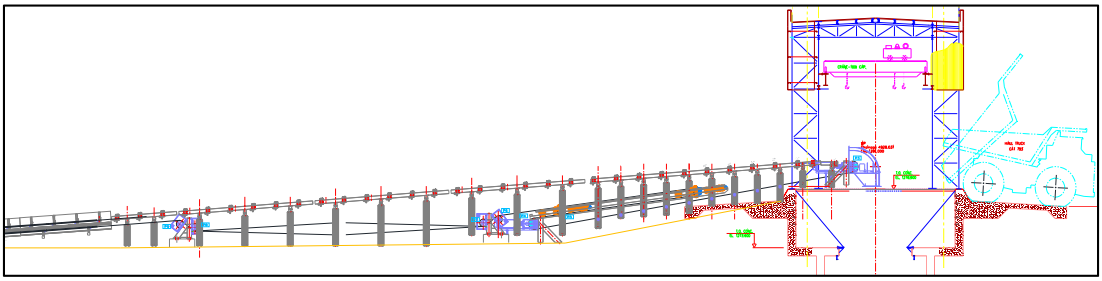


Figure 15.36: Conveyor CV-02 head end discharge to existing truck dump station

(Source: CDI, 2016)

The surface conveyor CV-02 uses a 1,067 mm (42”) wide, 45 degree trough, ST-1600N/mm belt that operates at 4.0 m/s and is powered by five 373 kW (500 hp) six-pole motors with VVVF drive control. The drives are located at the tail, mid station and head end due to the complex vertical geometry involved. The conveyor also features head and tail end belt turnovers. The high tension pulley shell diameter is 914 mm, and the low tension pulley shell diameter is 610 mm. Conveyor tension is achieved using a gravity take-up located at the head end.

AMEC Foster Wheeler (Burnaby, Canada) designed the access corridor from the existing Kemess South facilities to the planned triple-decline portal location in the Kemess Lake Valley. Conveyor CV-02 connects the existing Kemess South facilities to the planned triple declines portal location. Where CV-02 cannot be aligned with the access road, due to conveyor curvature constraints, a 5 m wide road is provided for CV-02 installation and maintenance.

15.9.6 Water Management

The dewatering system design for Kemess underground was carried out by Tetra Tech (Toronto, Canada) for the 2013 feasibility study (SRK 2013). Their report can be found in Appendix B. This section provides a summary of their work.

The dewatering facility will be located at the lowest elevation of the Mine, being the 1,082 mRL, which is the low point on the ventilation-drainage level; allowing gravity drainage to the two main sumps. Each sump comprises settling and clean water zones, with each clean water zone having at least a two hour capacity.

Water inflow estimates have been provided by Lorax Environmental Services Ltd covering all stages of the Mine's life from initial decline development, through cave development, and through to cave breakthrough to surface. Maximum inflow is estimated to occur just after cave breakthrough, conservatively estimated at 70 L/s. Thereafter, steady-state inflow is conservatively estimated to be 38 L/s. In addition, the 2013 FS (SRK 2013) estimates mining process water inflow of 13 L/s.

Based on estimated inflows, two 574 kW pumps will be installed in the pump chamber. During peak inflow (70 L/s), one pump will need to run for 10 h followed by 2 h idle, removing 88 L/s against a 400 m head. During steady-state inflow (38 L/s), one pump will need to run for 1.7 h followed by 2 h idle, removing 92 L/s against a 411 m head. As such, one pump and one sump will always be on standby. For both situations, a 200 mm pipe is required for dewatering, delivering water to the dewatering transfer pond adjacent to the conveyor decline portal, with the pipe located in the conveyor decline. Settled water will then be transported to the KUG tailings storage facility via a 200 mm pipeline installed on the overland conveyor (CV02) structure.

Lorax has also estimated inflow to the KUG workings of 654 L/s over 48 h for the 100-year storm event; being an extreme worst case as it assumes maximum precipitation and snowmelt occurs simultaneously which is considered extremely unlikely. One pump running full-time will take just under 13 days to dewater the ventilation level. It may be possible to dewater using both pumps, reducing the dewatering period to less than eight days, although in-pipe water velocity increases to 4.1 m/s which is considered excessive. The ventilation-drainage level has a water storage capacity of ±89 ML, which with ongoing dewatering during a 100-year event is considered adequate to handle the total 113 ML inflow.

Total capacity for a single sump is approximately 5 h at a steady-state inflow rate of 38 L/s, providing initial storage in the event of a failure of the standby sump. Thereafter, the ventilation-drainage level would be used for water storage.

During initial decline development, estimated maximum water inflow is 8.4 L/s. As such, 9.6 kW face (diaphragm) pumps and 43 kW staged (centrifugal) pumps will be used for dewatering. The staged pumps will be located in redundant re-muck bays in the access decline.

15.9.7 Maintenance Facilities

The underground workshop design for KUG was carried out by Tetra Tech (Toronto, Canada) for the 2013 KUG Feasibility Study. Their report can be found in Appendix B. This section provides a summary of their work.

The main workshop will be located on the extraction level, south of the cave footprint, with ready access from the access decline. Facilities will include:

- Main gallery with 20 t overhead crane;
- Two service bays with inspection pits;
- One service bay with 5 t capacity monorail;
- Welding bay;
- Wash bay, with oil separator;
- Engine, drill and electrical repair areas; with 2 t capacity monorail;
- Warehouse;
- Assorted stores – tyres, tools, cylinders, lubes;
- Offices and refuge chamber;
- Electrical substation; and
- Parking bay directly outside the workshop.

The workshop is ventilated from the access decline and has a dedicated connection to the underlying return air level. Fire doors at both accesses to the workshop will be used to control ventilation during normal and emergency situations. The workshop will also be equipped with a fire suppression system.

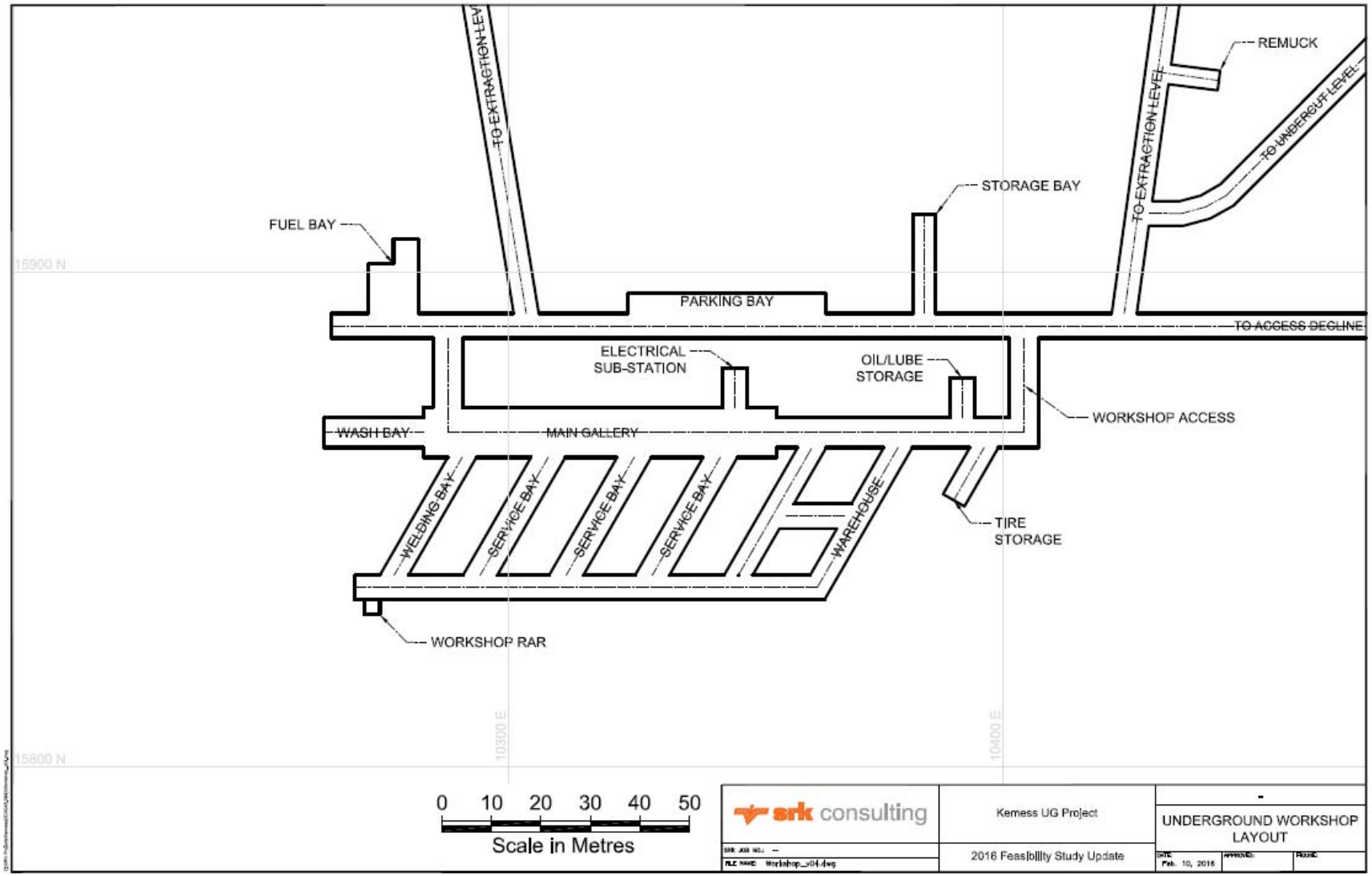


Figure 15.37: Underground maintenance facilities

(Source: SRK, 2016)

A total of ten large items of mobile equipment can be accommodated in the workshop at any one time. This compares to a peak maintenance load of 11 units during Year 1 to Year 5. Any shortfalls will be handled by (a) some breakdown maintenance being carried out in-the-field, (b) some maintenance such as light vehicles being carried out on surface, and (c) the average availability of new equipment exceeding 80%.

15.9.8 Electrical

The proposed electrical infrastructure begins with a feed from the existing KS 25 kV switchgear lineup at the KS 25 kV substation. The electrical feed will be primarily by overhead line, with a portion of armoured cable fixed to the access tunnel back/roof, to power the Kemess Lake Valley (KLV) substation at the triple decline portal. The electrical system is presently designed with a 20 MVA capacity to comfortably support the predicted underground and conveyor loads.

The KLV substation is to consist of a switchgear lineup to service two 25 kV cables, one each down the access and conveyor declines to the Kemess underground (KUG) substation. Moreover, the KLV surface substation will have provision for redundant 25/4.16 kV transformers (presently located at the KS processing plant, to be re-purposed) to power the surface infrastructure and overland conveyors, and two 1,250 kW diesel generators to provide emergency backup and/or construction power.

The KUG underground substation located at the bottom of the declines will step down from 25 kV to 4160 V for powering the underground crushers, ventilation fans, and portable substations. The portable substations will be used for low voltage, 600 V distribution.

This project will make use of a fibre optic network backbone to provide communication for protection devices, process equipment, operations, offices, and underground, including wireless communication to the remotely operated LHDs. The mine shall utilize a high resistance grounding scheme to comply with safety requirements of CSA M421.

Electrical power requirements for other aspects of the KUG project will be supplied by the existing 230 kV grid supply from the Mackenzie substation, which was used to provide power for the previous KS operation. These requirements include process plant, admin/workshop/warehouse complex, camp and kitchen, fuel tank farm, KUG TSF dewatering pumps, water treatment plants, and site water management pumps.

Approximately 500 m of 25 kV 1c500 MCM cables are anticipated between the existing process plant and the beginning of the overhead line. The overhead line uses 477 Hawk ACSR conductors, running approximately 3,300 m to the access tunnel. Once at the access tunnel, the system will transition back to 25 kV cable to the KLV substation at the main portals. From the KLV substation, there will be another cable feed down each of the access and conveyor declines, approximately 4,200 m each. Conductors are sized according to the projected load-flow and voltage drop requirements.

The power line shall consist of forty poles supporting a 477 Hawk ACSR overhead run with a typical 85 m span between poles. The overhead run will extend from the KS 25 kV substation near the end of the conveyor to the access tunnel approximately 3.3 km distant.

The poles have been planned to be installed along the south and west side of the access road. This choice has been made to ensure that the poles will have minimal disturbance from vehicular traffic and construction of the conveyor. AMEC Foster Wheeler has provided information to state that most of the poles will be on a declined slope and that the anchors will be placed into the ground towards the access road. For the purposes of this study, a 3 m clearance from the access road has been provided.

Two poles will have pole-top transformers mounted to convert the voltage for the heat trace system required for the KUG dewatering pipeline. The heat trace feeds will go into small E-houses at the side of the access road in order to install the required infrastructure.

Depending on the location, the 25 kV single conductor 500 MCM cabling will be either buried in rigid PVC conduit, strapped to cable-trays along the conveyor system or suspended inside the underground tunnels via messenger cables anchored to the rock. Where possible, the method to suspend the cables by messenger wire is the preferred cost effective option.

The KLV substation, located close to the triple decline portals, is a key electrical infrastructure item for this project. This location will:

1. Receive power from the existing grid supply
2. Provide backup generators to power equipment during construction
3. Provide backup power for emergency upset conditions
4. Provide feeds for powering the underground operation
5. Power the construction / office area in the Kemess Lake Valley
6. Provide network infrastructure for surface and underground communications
7. Provide power for a section of the dewatering pipe heat trace; and
8. Provide redundancy for maintenance purposes.

The KLV substation shall make use of two existing 6 MVA 25 kV:4160 V transformers currently at the Kemess site, re-purposed for surface construction and mine development-construction loading. Additional smaller transformers shall be installed to service the loads at the required voltage levels.

The main conveyor motors will be fed from this location on the 4,160 V lineup with VFDs. Additionally, during the construction phase of the KUG mining operation, ventilation and other electric-drive mining equipment will be fed from this location.

The KLV substation will also comprise two 1,250 kW diesel generators to provide power during portal construction and initial KUG development. These will be left in place after construction to provide power during emergencies or if the 25 kV feed from the Kemess South substation has disconnected.

It is planned that the KLV substation will be initially powered by diesel generators, and thereafter transferred to the 25 kV feed from the process plant as soon as the 25 kV overhead line and required cabling are installed.

The 25 kV supply will be stepped down to 4,160 V initially, and 4160 V cables will be used for providing required electrical power supply during development of the triple declines. Once the declines are complete, these 4,160 V cables will be bypassed by permanent 25 kV cables run to the main underground substation to provide the main KUG electrical power feed. The 4,160 V cables will remain in place however, for powering loads such as ventilation and dewatering infrastructure that are located in close proximity to the conveyor decline.

The underground substation for the KUG Mine will be a central location for the electrical services for the mine; currently planned to be located in the lowermost cross-cut between the access and conveyor declines. This substation shall consist of two 12 MVA dry-type transformers rated for an underground environment. The main transformers will provide for the 4,160 V and 600 V lineups. The major loads will be 4,160 V and include the production system main exhaust ventilation fans, primary jaw crushers, dewatering pumps, and the transfer conveyors.

Auxiliary ventilation fans will be serviced from this location as the mine development progresses.

Most of the mine electrical requirements are located on the extraction level. The extraction level will be fed from two separate cable runs, developing a ring-like 4,160 V infrastructure across eight E-rooms on the extraction level. This infrastructure will be constructed from early Year -1 through Year 5 of the project schedule. Each E-room will consist of one 1 MVA mobile substation with 600 V disconnects in order to service the expected loads. Each mobile substation will be able to support up to two electric-drive LHD (if required) and two drill jumbo units simultaneously.

Each mobile substation will have up to six feeds which will support the 600 V requirements. In order to operate electric-drive LHDs (if used for KUG), a take-off unit has been accounted for at each end of every extraction drive. There are forty take-off units to be mounted permanently along the extraction level from Year -1 to Year 5.

The KUG Mine dewatering pipe will be mounted on the conveyor structure across the KLV to the take-off point to the west of the KS open pit for in-pit water disposal. As the conveyor structure will be uncovered and exposed to the elements, it is necessary to provide heat tracing for the length of the dewatering pipe from the conveyor decline portal. The total distance of the heat trace is approximately 3,500 m

The heat trace will be broken into five sub-systems, each providing heat trace along the dewatering pipe for up to 700 m each. Each sub-system will be split into separate feeds of approximately 150 m cabled sections. A three-phase 150 kVA transformer has been provided for at three of the sub-system locations. Two sub-system locations occur along the length of the overhead power line and 150 kVA single-phase pole-top transformers have been provided for at each overhead line sub-system location.

To account for the power line and dewatering pipeline being on opposite sides of the KUG access road, the heat trace feeds will be installed in a conduit below the access road from the supply transformers to the conveyor structure and then mounted along the dewatering pipe.

In order to control the heat trace system, the communication/control infrastructure required to operate the conveyor system will be utilized. As the heat trace control requirements will be negligible in comparison to the conveyor system, this approach is considered cost-effective.

15.9.9 Storage & Materials Handling

The underground storage and materials handling requirements for KUG was carried out by Tetra Tech (Toronto, Canada) for the 2013 KUG Feasibility Study (see Appendix B). This section provides a summary of the work.

There will be a total of eight 20 m long concrete floored storage bays for the laydown and storage of ground support and ventilation materials; four on each of the extraction and undercut levels, with two on the north rim tunnel and two on the south rim tunnel.

There will be a total of four explosives and four explosives accessories (i.e. cap) magazines; two on each of the extraction and undercut levels, located on the east and west rim tunnels. Explosives magazines will hold both packaged and bulk explosives required for both development and production activities. All magazines will be constructed in adherence with British Columbia mining and construction regulations.

The underground workshop will have dedicated storage for spares/consumables such as tyres, lubes, cylinders, and tools. There will also be a warehouse located at the workshop for storage of high to medium use maintenance parts.

Diesel will be batch piped to a storage tank located in one of the lower decline system cross-cuts. From there, diesel will be delivered to the fuel bay/station located adjacent to the workshop on the extraction level by fuel truck.

Concrete and shotcrete will be delivered from the surface batch plant directly to points of use underground by transmixer. Shotcrete will be transferred to mobile shotcrete units. The batch plant will be located to the east of the Kemess South process plant building.

Flatbed trucks will be used to transport palletised materials from the existing Kemess South warehouse to the Kemess Lake Valley laydown area. Materials will generally be transported underground using boom trucks, although scissor lifts and light vehicles may also be used. Explosives will be transported underground using approved vehicles, with explosives and accessories transported separately.

Shift crews will be transported underground using a fleet of 16-person personnel carriers, with transport directly from the mine dry located at the existing Kemess South admin/warehouse/workshop facility to the parking area outside the underground workshop.

15.9.10 Refuge Stations

There will be three permanent refuge stations in the Kemess underground workings; one at the workshop, one on the extraction level north rim tunnel, and one on the undercut level south rim tunnel. They will be constructed and equipped as per British Columbia mining regulations, and sized to accommodate all persons working in the vicinity. Refuges will also serve as lunch rooms.

15.9.11 Monitoring & Control Systems

In addition to the geotechnical monitoring systems outlined in Section 15.3, the following monitoring infrastructure and systems will be required for effective control of Kemess underground mine operations.

A combination of 12-pair fibre optic (main trunk) distribution cables and 6-pair fibre optic (secondary branch) drop cables will be installed in parallel with the electrical reticulation system. Network communications and control racks will be installed at various electrical substation, control room, and ventilation raise locations. These control racks act to terminate the fibre optic cable, allowing breakout of communication and control services as required.

The main functions provided by the fibre optic network include:

- Data communications – process control, automation, and computer network; and
- Voice communications – fixed and mobile phones.

Functions are deployable in both hardwire and wireless formats.

The resulting network provides a unified communications backbone, supporting advanced mine monitoring and control.

The extraction level production LHD mucking operation will be controlled using a system such as Sandvik's AutoMine® system.

16 Recovery Methods

16.1 Summary

Four single-toggle jaw crushers will be installed underground in the KUG mine. Ore will be conveyed from the crushers to the conveyor decline portal where it will be transferred to an overland conveyor for delivery to the existing process plant ore feed stockpile.

Based on the metallurgical testwork that has been completed for the KUG project, the existing Kemess South process plant flowsheet has the unit operations required to process the ore and achieve the predicted results summarised in Section 16.2. There are some additional equipment requirements, primarily for fine regrinding, described in the following text, but the main preparation efforts will focus on reconditioning the existing equipment.

One SAG mill/ball mill grinding line was removed from the KS grinding circuit. The targeted process plant throughput for the KUG project is estimated to average 24,600 t/d. The simplified flowsheet in Figure 16.1 illustrates the revised flowsheet. A copper flotation concentrate containing gold and silver values will be produced and transported to smelters for processing. Tailings will be discharged to the KS open pit which will be the KUG tailings storage facility (TSF) and is expected to have sufficient volume capacity for the duration of the KUG operations. Process reclaim water will be pumped to the KS process plant facility from the KUG TSF.

16.2 Flowsheet Selection

The simplified process flowsheet in Figure 16.1, has been designed to utilize as much of the existing KS process plant equipment as possible to minimize additional capital expenditures.

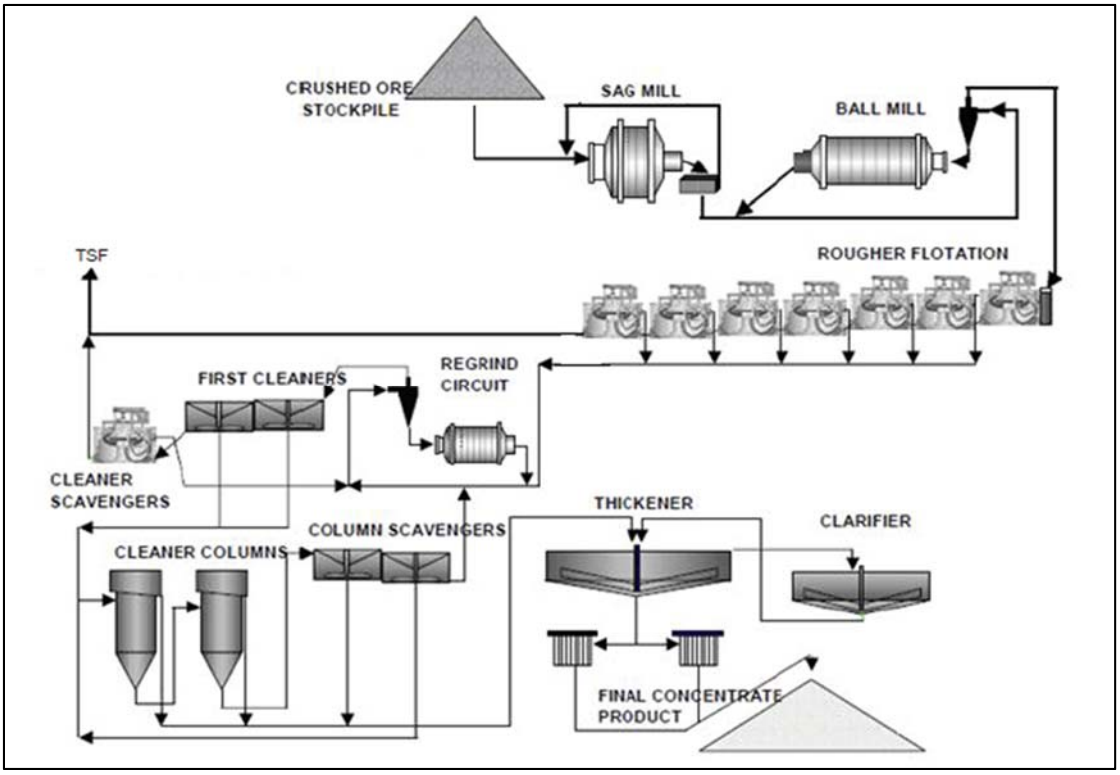


Figure 16.1: KUG simplified flowsheet

(Source: SRK, 2013)

16.3 Process Design Criteria

The process design criteria used for assessing the existing flowsheet is based on metallurgical testwork, information from previous operations experience for Kemess South, vendor information, and reference to similar operations.

16.4 Primary Crushing

Primary jaw crushers will be located underground, with up to four 55” x 42” (1.4 m x 1.07 m), single-toggle jaw crushers operating to deliver 9 Mt/y. Details of the primary crusher installation are discussed in Section 15.9.4.

16.5 Overland Conveyor

Both underground and overland conveying and conveyor design criteria are discussed in Section 15.9.5 of this report.

16.6 Kemess South Mill

16.6.1 Primary Stockpile

Ore will be transferred from the overland conveyor to the coarse ore stockpile adjacent to the KS process plant using the existing stacker conveyor. The coarse ore stockpile has a live storage capacity of 74,200 t. Ore will be reclaimed from the stockpile by three existing 6 ft x 20 ft (1.83 m x 6.1 m) apron feeders delivering ore to a 60" (1.52 m) wide SAG mill feed conveyor.

16.6.2 Primary & Secondary Grinding

The grinding circuit will utilize the existing 10.36 m x 4.65 m (34 ft x 15.25 ft) SAG mill driven by twin 4,474 kW (6,000 hp) motors. The SAG mill is fixed speed and operated in series with a closed-circuit 6.7 m x 11.1 m (22 ft x 36.5 ft) ball mill, also driven by twin 4,475 kW (6,000 hp) motors. Coarse material or pebbles discharging from the SAG mill (screen oversize) is returned to the mill without being crushed. Slurry discharging from the SAG mill (screen undersize) and ball mill are collected in the cyclone feed pump box and pumped to the ball mill cyclopac for closed circuit ball mill operation. The grinding circuit will have a nominal capacity of 24,600 t/d.

The existing cyclopac in the grinding circuit had been retrofitted with gMAX® 660 mm (26”) cyclones prior to the shutdown of KS operations. The cyclopac consists of 12 cyclones of which nine are typically operational. There will not be any changes to the existing cyclone pumps or cyclopacs for processing of KUG ore.

16.6.3 Flotation and Fine Grinding

Cyclone overflow from the aforementioned grinding circuit will be processed in the 16 existing 127 m3 Wemco rougher flotation cells. The circuit layout has two lines of two parallel rows each containing eight flotation cells. During operation of KS, some of the Wemco rougher flotation cells were refurbished with Outokumpu tank cell mechanisms. There are a total of four banks of eight rougher cells (32 cells) and a feed distribution tank which permits selection and operation of each rougher flotation bank independently. No additional rougher flotation modifications are required for processing ore from the KUG deposit.

Rougher flotation concentrate will be pumped to the concentrate regrind circuit to produce a P80 of 20 µm product for final cleaning. The existing 4,475 kW ball mill regrind will not be sufficient to handle the regrinding requirements that result from the increased mass pull and the fine target product size. A two-stage regrind circuit will be configured with the existing ball mill regrind operated as the first stage. Cyclones will be used to increase mill feed density and remove the fines component prior to the regrind ball mill. For this feasibility study, the second regrind stage is based on using two 3,000 kW IsaMill™ units to take advantage of the higher efficiencies offered by stirred mills. These fine grinding mills will operate as a single pass unit in open circuit without classification.

Product from the rougher concentrate regrinding circuit will be pumped to the first cleaner mechanical flotation cells. These cells were purchased used, and were installed and retrofitted into the KS process plant. Allowance has been provided to rebuild the first cleaner flotation cells for use on the KUG project. The first cleaner concentrate will be pumped to the two existing 3.4 m diameter

x 12.2 m high column cells which are arranged in series. Column cell final concentrate will be pumped to the concentrate thickener. Column cell tailings will be delivered to one line of the existing mechanical column scavenger cells. The line contains 12 Denver cells, each 17 m3 capacity, that will need to be rebuilt. First cleaner tailings will flow to the cleaner scavenger flotation cells from which the concentrate will be pumped back to regrind circuit.

The combined tailings from the cleaner scavenger flotation cells and the column scavenger cells will be pumped to the tailings facility in the KS open pit (KUG TSF) for subaqueous disposal. This will be a new pumping system for the KUG project included in the plant CapEx and Opex estimates as well as tailings discharge in the KUG TSF. Rougher flotation tailings will be pumped to the KUG TSF using the existing tailings pumping system.

16.6.4 Concentrate Dewatering

Final copper concentrate will be pumped to an existing 18.3 m diameter high capacity WesTech thickener circuit where it will be thickened to 70% solids w/w. The original 10.8 m diameter thickener acts as a clarifier for the concentrate thickener before the overflow water is returned to the process. Thickened concentrate will be stored in the two existing 4.88 m diameter x 4.00 m high agitated stock tanks prior to being pumped to the filter presses.

Two existing Baker Hughes plate and frame filters each containing 135 m^2 of filter area will dewater the concentrate to a target moisture content of 8%. The dry concentrate will be conveyed to the existing concentrate storage area before being loaded onto trucks via a front-end loader.

A new overhead crane and sump pump dedicated to the dewatering area have also been specified in the KUG process plant CapEx estimate.

16.7 Reclaim Water and Process Water Pond

Process water will be reclaimed from the KUG TSF. The reclaim water pumping system will use the existing vertical turbine pumps and deliver it to the existing process water pond.

16.8 Tailings Management

For the KUG project, the process plant tailings will be pumped to the KUG TSF (i.e. KS open pit). This facility is discussed in Section 17.3 of this report.

17 Project Infrastructure

17.1 Summary

The Kemess South Mine was brought into production as a 52,000 t/d open pit mine in 1998 and produced continuously until 2011.

It was a fully integrated complex that also included:

- Primary crusher, stacker conveyor, and ore stockpile;
- 52,000 t/d process plant (or concentrator) producing Cu-Au concentrate;
- Administration, workshop/service, and warehouse complex;
- Camp with accommodation for in excess of 300 on-site personnel;
- Tailings storage facility (TSF) with associated tailings and reclaim water pipeline systems;
- 380 km 230 kV electrical power line from BC Hydro's Kennedy siding sub-station;
- 1.6 km airstrip capable of handling most STOL (short take-off & landing) aircraft and Hercules transport planes when heavier loads were required;
- 380 km all-season access road from Mackenzie, and
- Rail load-out facility at Mackenzie.

One half of the grinding circuit has been removed, effectively reducing grinding capacity to approximately 25,000 t/d. However, the remaining facilities are intact, having been under care and maintenance since 2011 and are available to support the KUG project. Where refurbishment, upgrade or addition to existing facilities is required, CapEx has been estimated and provided for under processing, G&A and site services cost estimates.

For KUG tailings storage, it is planned to use the Kemess South open pit, with the open pit having capacity for approximately 107 Mt tailings and 3.0 Mt waste rock from KUG development. To achieve this storage capacity, a 25 m high dam (the "east dam") is required to be constructed at the east end of the open pit. A spillway will be constructed in the south-west area of the pit to allow discharge of excess water once KUG operations have ceased while ensuring adequate water cover of PAG tailings.

Access to the KUG mine workings (via triple declines) will be provided by an access corridor comprising an access road west of the Kemess South open pit, access tunnel through the hill to the north of the open pit, and access road across the Kemess Lake Valley. Electrical reticulation will be extended along this corridor to provide grid power to the KUG mine. Mining contactor facilities will be established in the Kemess Lake Valley following development of the access tunnel. The conveyor used to transport crushed ore from the KUG mine will be installed along the access corridor with discharge to the existing Kemess South primary crusher surge hopper (with crusher removed). An overview of the surface infrastructure is presented in Figure 17.1.



Figure 17.1: Site Surface Infrastructure
(Source: AuRico, 2016)

17.2 Existing Infrastructure at the Project Site

The Kemess South Mine site previously supported a 52,000 t/d open pit mining and ore processing operation. The infrastructure from this operation is still intact and is currently being maintained by an on-site care and maintenance crew. The camp, administration complex, ore processing facility, and related support infrastructure, such as the airstrip, access road, power supply, and concentrate handling infrastructure, are more than adequate for the proposed KUG project. The following discussion of facilities is focussed on features relevant to the KUG project.

17.2.1 Camp

The existing camp includes a kitchen, five 36-42 person bunkhouse units, potable water facility, sewage facility, backup generators, site services maintenance shops, and recreational facilities. This camp was used for the Kemess South Mine and is approximately 17 years old. It is estimated that with upgrades to six bunkhouses and replacement of a further two bunkhouses, the camp will be able to support the projected life of the KUG project. Irreparable bunkhouses were removed and disposed of as part of the ongoing Kemess South reclamation effort. Costs for upgrades and replacements, including kitchen and recreation facilities, have been estimated and included with the G&A CapEx estimate. On-site labour complements will fluctuate over the KUG project life as shown in Figure 17.2. The camp footprint is appropriately sized for the KUG project with adequate provision to house the catering staff in addition to the project related labour complement. The peak in the camp labour complement occurs in Q3 Year-1 (Q19 in Figure 17.2) due to peak underground development, underground construction activities, and process plant start-up.

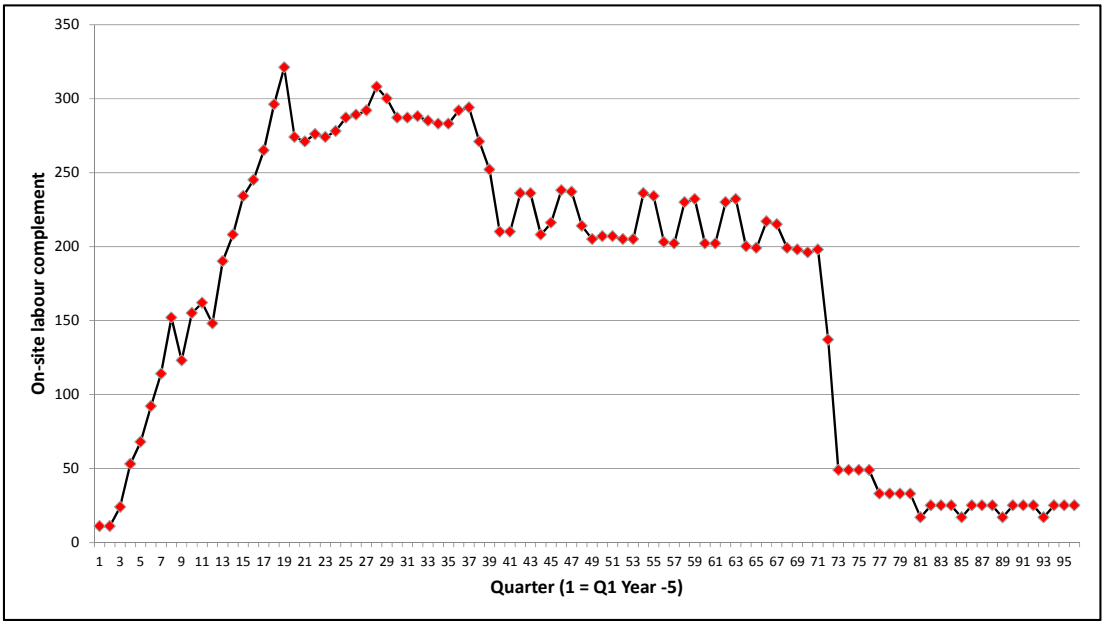


Figure 17.2: Site labour complement over KUG life of mine

(Source: SRK, 2016)

17.2.2 Power Line

There is a 380 km, 230 kV power line from the BC Hydro Kennedy substation to the mine site and related stepdown transformers, backup diesel generators, and infrastructure to power the process plant and Kemess South site. AuRico owns the power line and all related site electrical infrastructure. The power line is currently in use providing power for Kemess South care and maintenance, and agreements are in place to be able to provide sufficient electrical power for the KUG project. Annual inspections are current and the line is in good condition. The main substation feeding the Kemess power line was recently upgraded to enable feeding both Kemess and the Mount Milligan projects.

17.2.3 Process Plant

The previous ore processing facilities processed in excess of 52,000 t/d of ore similar to that of KUG (Figure 17.3) and are being maintained by care and maintenance personnel. Basic fire suppression systems are operational and mills are being rotated at regular intervals. One grinding line has been removed but the remaining line is sufficient for the KUG project (with fine grinding mill additions) and the remainder of the process plant is intact.

The barges previously used for reclaiming water from both tailings facilities are still on-site and intact.



Figure 17.3: SAG mills – only one line required for KUG project
(Source: AuRico, 2016)

17.2.4 Administration and Workshop Complex

The administration/surface workshop complex is intact and operational. The existing open pit mine dry facilities will be expanded to provide mine dry facilities for the KUG project as the existing facilities are too small. Associated costs have been estimated and included with the G&A CapEx estimate.

Warehousing and a significant workshop, including a fully equipped machine shop, are established in the complex carried over from the Kemess South open pit operation.

Fuel storage for support of a large scale open pit mining operation is intact and operational.

All of the accounting, engineering, and administration offices from the Kemess South open pit operation are still available for the KUG project.

The KUG project is budgeted to be self-sufficient from an accounting and administration point of view with limited corporate support required.

17.2.5 Access Road

The 380 km access road from Mackenzie is being maintained and used on a seasonal basis for ongoing site reclamation. Lower portions of the road, not maintained by AuRico service local First Nations communities and are used extensively for forestry operations. There is a combined industry/government road user group that co-ordinates maintenance of the various sections of the access road from Mackenzie up to the mine site.

Each spring the road is inspected and bridges and culverts are upgraded or replaced as required. The access road agreements with the various provincial ministries are all in good standing.

Road maintenance costs are included in the G&A costs. Beyond regular maintenance, no major upgrades are seen as required for the KUG project.

This road will be used to bring construction and operating supplies to site and truck out concentrate for trans-shipment at the Mackenzie rail head facility.

17.2.6 Airstrip

There is an all-weather airstrip at site that was routinely used by Twin Beech aircraft to transport personnel on and off site during the operation of the Kemess South Mine. Dash 8 aircraft have also used the strip in the past, as well as some heavy haul type STOL aircraft.

The airstrip continues to be maintained and used by the care and maintenance crew as their main site access route. The airstrip will not require any additional improvements for the proposed KUG project.

A regional airline company provided charter air service for the operation of the Kemess South open pit operation and continues to provide charter service to the Kemess site during care and

maintenance. It is anticipated that they, or others, would be able to provide the required service for the proposed KUG project.

Current charter pricing has been used for estimating flight costs for the KUG project and are included in G&A cost estimates.

17.2.7 MacKenzie Rail Loadout Facility

AuRico retains a trans-shipment facility at Mackenzie, adjacent to a rail spur, to facilitate concentrate rail shipment. For the Kemess South operation, grinding media has also been brought to Mackenzie via rail car from various suppliers. During initial construction of Kemess South, oversized loads were transported by rail to avoid highway size and weight restrictions. It is planned to use this facility to ship concentrate from the KUG project to market. There are no improvements required for this facility.

17.3 Tailings Facility

For the KUG project, the process plant tailings will be pumped to the existing mined-out Kemess South open pit. The capacity of the Kemess South pit has been determined to be sufficient for the life of mine tailings and waste rock volumes from the KUG project; approximately 107 Mt of ore and an additional 3.0 Mt of PAG waste rock over an operating mine life of 13 years.

17.3.1 Overview

The waste management plan for the KUG project includes disposal of the process plant tailings and mine development waste rock within the mined-out portion of the Kemess South open pit, designated as the KUG Tailings Storage Facility (TSF), in a similar manner to the previous operations (Figure 17.4). However, in order to accommodate the total volume of tailings produced, the pit rim must be raised beyond its current level. This will be accomplished by construction of the East Dam (starter dam construction anticipated to be completed by end of Year 8) to an ultimate crest elevation of 1,275 mRL in order to provide adequate tailings, free water, and flood storage. Water will be reclaimed from the tailings facility for use in ore processing. Operational constraints dictate that a water treatment facility (for removing selenium from impacted water) will need to be built and in operation by Year 1 when the process plant starts up. This is required as part of the TSF water balance which predicts a net water balance surplus throughout the proposed operations due to the accumulation of water in the proposed tailings facility since cessation of the previous Kemess South mining operations.



Note: KS PAG waste and tailings deposited into KS Pit. View looking west

Figure 17.4: Kemess South open pit

(Source: AuRico, 2016)

17.3.2 Tailings Dam

The East Dam is designed with a downstream rockfill shell, a central low permeability till core and appropriate filters separating the core from the downstream shell. Tailings and waste rock deposited into the impoundment will provide upstream support for the upper core of the dam. Upon completion, a wide NAG above-water tailings beach and fresh water cap will be developed and an open channel closure spillway with invert at 1,271 mRL will be constructed through the south-west perimeter of the impoundment. The East Dam will be constructed in lifts using a four-stage operation dictated by the method of construction and gradual accumulation of tailings and water in the facility (Figure 17.5).

- **Stage 1** — Starter dam is designed to be constructed primarily as a compacted basal till embankment with 2H:1V upstream and downstream slopes and keyed into foundation material. The starter dam will be constructed to 1,260 mRL.
- **Stage 2** — Intermediate dam, constructed to elevation 1,265 m, will be raised by the downstream method with 10 m wide compacted till core to 1,265 mRL.

- **Stage 3** — Intermediate dam, constructed to 1,270 mRL will be raised by the centreline method with 10 m wide compacted till core to 1,269 mRL, upon which the till core will be tapered with a downstream slope of 1H:1V to 1,270 mRL.
- **Stage 4** — Final dam, constructed to 1,275 mRL, will continue to be raised by the centreline method with the core tapering at 1H:1V to a final core width of 5 m at 1,274 mRL. Upon closure, the till core will be capped with a 1 m thickness of fine NAG rockfill.

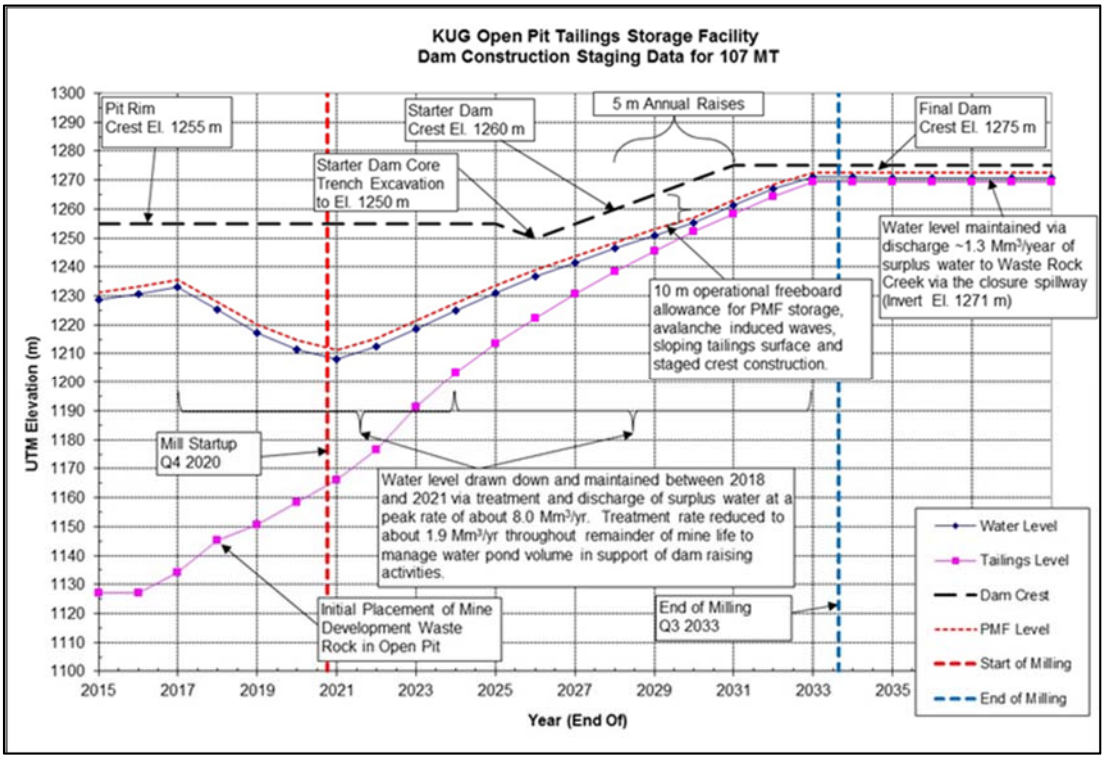


Figure 17.5: Dam construction phasing and water balance

(Source: AMEC FW, 2016)

17.3.3 Operational Water Management

Underground water management design is by Tetra Tech. In brief, mine water will be collected in sumps underground and pumped via a 200 mm line to a transfer pond at the portal, and then to the Kemess South pit.

Water management on surface will be focused on the Kemess South pit, which will also serve as the TSF for the project. Since the end of Kemess South mining operations in 2011, the open pit has been allowed to recharge with a combination of natural runoff and diversion water collected in the various sumps and ponds around the Kemess site. Based on the current site water management scheme, it is predicted that the additional water accumulating within the open pit will form a substantial water balance surplus throughout the proposed KUG operations necessitating treatment and discharge of about 46 Mm^3 to make room for storage of the life of mine (LOM) tailings

production. It was originally assumed that the excess water could be discharged prior to KUG start-up and that the TSF would operate as a zero-discharge facility during the operating life of the mine. A subsequent review demonstrated that a revision to this approach was required.

A more conservative scenario has been selected for this study which assumes that discharges from the Kemess South pit will begin prior to process plant start up and will require treatment for dissolved metals. Assuming process plant start up in Year -1/Q4, the sequence will be as follows:

- A dewatering barge and pipeline capable of removing water at a peak rate of 1,100 L/s will be installed in Year -4. The pipeline will be routed across the Kemess South mine site and down the main site access road to a diffuser structure located in Attichika Creek.
- A water treatment plant capable of treating 120 L/s will be constructed in Year -1 and connected to the main dewatering pipeline that discharges to Attichika Creek.
- The diversion ditch along the pit highwall will be reconnected in Year -2.
- Water will be withdrawn from the pit and discharged at a rate of up to 8.0 Mm^3 per year from Year -3 through Year 1. Treatment and discharge at a rate of 1.9 Mm^3 per year will begin in Year 1 and continue throughout the 13-year mine life and 6-year closure period.
- The KUG TSF closure water pond will be maintained by the closure spillway constructed in bedrock at the southwest side of the pit. The volume of the closure water pond is anticipated be about approximately 2.5 Mm^3.

17.3.4 Design Considerations

The tailings containment structures have been designed in accordance with the Canadian Dam Association, Dam Safety Guidelines using Probable Maximum Flood (PMF) and Maximum Credible Earthquake (MCE) criteria during construction, operation and closure of the facility. As the tailings facility is an exhausted open pit, stability analyses were performed and additional provision was made for wave generation in the event of pit wall failure.

17.3.5 Tailings Storage Facility Expansion Potential

With the construction of a dam along the southwestern portion of the pit containment facility, it would be possible to deposit an additional 15 Mt of tailings in the KUG TSF. This would entail an additional lift on the East Dam and subsequent extension of the dam toe towards Kemess Creek which constrains further expansion of the TSF to about 15 Mt or a 10 m raise of the facility (i.e. East Dam crest to 1285 mRL).

17.4 Water Treatment

17.4.1 Water Treatment Plant

The KUG project water treatment plant (WTP) is designed to remove contaminants from impacted surface water streams prior to discharge into Attichika creek. The design criteria for the WTP was determined by comparing ERM Consultants' 95th percentile operations base case water quality estimates with existing pit water quality assays and the BC Water Quality Guidelines issued May

2015 (WQG) in order to determine which constituents would require treatment. BioteQ Environmental Technologies Inc. then designed a treatment process to remove the constituents to the BC WQG concentrations at the plant effluent (end of pipe). Table 17.1 shows the feed water quality estimates for which the plant was designed (BioteQ, 2015a).

Table 17.1: KUG Water Quality Estimates, 95th Percentile Operations Base Case

Data Type	Estimate
Data Source	ERM Consultants
Constituent	mg/L
SO4	1,423
Cl	359
Ag	0.0749
Al	4.29
As	0.04
Cd	0.0037
Co	0.0096
Cu	0.0336
Fe	1.74
Mo	3.73
Pb	0.078
Se1	0.08
Zn	0.81

1 Selenium is present in two forms: selenite and selenate. This figure is the total selenium.

The water treatment process comprises two distinct systems and is shown in Figure 17.6:

1. A metals precipitation and filtration system to treat all of the metal constituents listed in Table 17.1 except selenate selenium
2. BioteQ's Selen-IX™ selenium ion exchange system to treat the selenate selenium

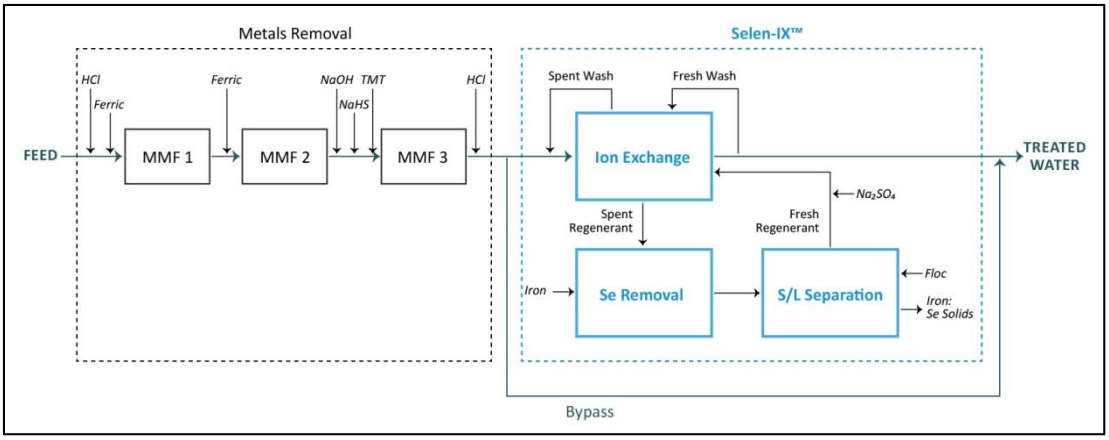


Figure 17.6: KUG WTP Process Block Diagram

(Source: AMEC FW, 2016)

The metals precipitation stage involves three stages of reagent addition and multimedia filtration to precipitate the metals out of solution. The metal precipitates collected in the multimedia filters are then deposited in the KUG TSF where they will settle to the bottom mingled with tailings and waste rock.

The next stage of treatment is Selen-IX™, which is a selenium treatment system developed by BioteQ for the selective removal of selenate from mine impacted water to levels as low as <1 ppb. Selen-IX™ was selected for the following reasons:

- Selen-IX™ can reach <1 ppb of selenium in the plant effluent, meaning that treated water reaches environmental targets at end of pipe without relying on dilution. The ability to discharge water without relying on dilution water allows greater flexibility in water management for the KUG project and reduces the risk of non-compliance.
- Previous work on other pilot projects and preliminary costs presented to AuRico showed significant life cycle cost savings through the use of Selen-IX™ as compared to other selenium treatment systems.
- Selen-IX™ has been approved by regulators as an acceptable selenium treatment process in permits for other mines in British Columbia.

These three benefits were sufficient incentive for AuRico to select Selen-IX™ and pilot this treatment approach for the KUG project.

The overall process comprises two main parts including ion exchange and electrochemical reduction and precipitation of selenium. Both of these process parts are purely physico-chemical and insensitive to water temperature. The first stage of the system uses a strong base anion exchange resin in the sulphate form to selectively remove selenium from water and produce discharge with residual selenium level as low as < 1 ppb. Once the resin is saturated with selenium, the resin is regenerated using sodium sulphate brine solution. During regeneration, selenium loaded on the resin is stripped off the resin into a small volume of the sodium sulphate brine

containing selenium at the concentration of at least one order of magnitude higher than the incoming plant feed.

The selenium-rich brine then enters the second part of the process where selenium is reduced and concurrently precipitated with iron released from iron anodes inside electrocells. The iron:selenium solids are then separated from the brine solution in a clarifier. The clarifier overflow is recycled back to the ion exchange regeneration stage while the underflow solids are dewatered in a conventional plate and frame filter press.

The WTP capacities and startup dates defined by ERM's site water model are shown in Table 17.2 (ERM, 2015).

Table 17.2: KUG WTP Capacities and Startup Dates

Plant	Treatment capacity (l/s)	Operational start date
Metals precipitation plant	121	Year 1
Selen-IX™	50	Year -3

The Selen-IX™ plant will be installed in Year -4 and located on the ground floor of the existing process plant building. The metals treatment plant will be installed in Year -1 on a newly constructed second floor of the process plant building (above the Selen-IX™ plant). Both plants will be predominantly supplied as pre-fabricated modules to minimize on-site construction.

17.4.2 Water Treatment Residue and Residue Management

Water treatment for the KUG project generates three solid by-product streams:

- Metals removal residue,
- Primary Selen-IX™ residue resulting from the primary treatment of the Selen-IX™ brine, and
- Secondary Selen-IX™ residue resulting from the secondary treatment of the Selen-IX™ brine.

Table 17.3 summarizes the amount and character of these streams. These estimates are based on the 95th percentile operations base case water composition and so the actual masses generated over the KUG mine life are likely to be significantly less due to the lower average amount of contaminants in the feed water for treatment.

Table 17.3: Residue Character Summary

Description	Mass per Water Treated (g/m3)	Total Life of Mine (t)	Total Tailings/Waste Rock Mass (%)	Iron Content wt (%)	Se Content wt (%)
Metals Removal residue	50	2,000	0.002%	38.9	<0.1%
Primary Selen-IX™ residue	500	4,000	0.004%	43.8	<0.1%
Secondary Selen-IX™ residue	4	31	0.00003%	36.8	<0.1%

The preferred option for the management of these residues is to dispose of them with the tailings being disposed of in the KUG tailings storage facility. The residue resulting from water treatment represents only a small portion of the overall mass of tailings; approximately 0.006% of the mass of the 107 Mt of tailings expected to be generated and stored in the KUG tailings storage facility over the life of the mine. Underwater disposal of the water treatment residue in the tailings facility mixed in a zone of low hydraulic conductivity minimizes the potential for metals release. Other options include trucking this material off-site to a landfill facility or depositing them in the void space left by the Kemess underground workings.

18 Market Studies and Contracts

Exen Consulting Services (Exen) was contracted by AuRico Metals to review the marketing and logistics of copper concentrates from the Kemess underground project. Their review is summarized below.

18.1 Concentrate marketability

Based on the transportation logistics, the most suitable market for the Kemess concentrate will be the overseas Asian market. The longer term outlook for concentrate demand is reasonably favourable, considering that many of the projects waiting the go ahead are in higher risk environments. There currently are no sales contract for this project. The slightly lower than optimal copper grade of 22% is offset by the extremely low concentrations of penalty elements making the KUG concentrate readily marketable and not subject to any penalties. The gold grade is projected to be 20–50 g/t which will result in a payable metal factor of about 97.0%.

18.2 Metal Price review

The metal market for copper looks to be fairly neutral from the supply and demand side. A longer term price of US$2.50/lb is forecast for copper. Exen did not venture an opinion on gold prices, but AuRico has judged that a longer term price of US$1,200/oz is reasonable to use as a base case.

18.3 Refining Charges

Exen recommends using the following concentrate treatment and refinery charges:

- Treatment charge: US$80.00/dmt concentrate
- Copper refining charge: US$0.08/lb
- Gold refining charge: US$6.00/oz
- Silver refining charge: US$0.40/oz

18.4 Concentrate Transport Logistics

As done for the Kemess South operation, it is suggested to truck bulk concentrate via the 380 km access road to MacKenzie where Kemess retains a railhead transhippment facility. Several options were examined from MacKenzie onward including trucking to the port of Stewart, rail to a smelter in eastern Canada, rail to a smelter in the United States, and rail to ports in British Columbia. Rail via a port in British Columbia followed by ocean freight overseas to Asia appears to be the most economical route and was considered as the base case.

Exen suggests an all inclusive transportation cost of US$132.14/dmt.

19 Environmental Studies, Permitting, and Social or Community Impact

19.1 Introduction

The KUG project will require approval from provincial and federal governments prior to receipt of permits, licenses and authorizations to construct and operate the proposed mine.

The KE deposit is currently at the exploration stage, so only permits related to exploration drilling are currently required. This work is subject to an exploration agreement with the TKN First Nations.

Reclamation and closure preparation of Kemess South has been undertaken annually via the progressive reclamation strategy in the approved Mine Reclamation and Closure Plan. AuRico has been recognized for being pro-active in reclamation and closure at the mine site, and has been the recipient of a number of reclamation awards from the British Columbia Technical and Research Committee on Reclamation and the Mining Association of British Columbia.

19.1.1 Environmental Permitting

The environmental assessment and permitting framework for metal mining in Canada is well established. The federal and provincial environmental assessment processes provide a mechanism for reviewing major projects to assess their potential impacts. AuRico examined and scoped the permitting and environmental requirements of the KUG project in the context of the ongoing regulatory requirements of the Kemess South mine.

The KUG project exceeds production rate thresholds for a gold mine under the Canadian Environmental Assessment Act 2012; therefore, a federal environmental assessment (EA) was triggered. The project does not automatically trigger the need for a provincial review under the BC Environmental Assessment Act. However, as a federal EA is required and the potential exists to de-risk project approvals through the provincial EA process with prescribed timelines, AuRico requested the KUG project be designated provincially as a reviewable project. The project is currently in the Application phase of a substituted EA process. Favourable decisions are expected to be reached before the end of 2016.

Before construction commences, the KUG project will be required to obtain a number of new provincial and federal licenses/permits. A number of existing permits for the Kemess South Mine have been in place since 1996 that remain in good standing, but may require amendment or renewal before construction begins.

As part of the EA process, AuRico is considering potential effects on several valued components of the natural and human environment including, among other components, aquatic and terrestrial ecosystems, current use of lands and resources for traditional purpose. From an environmental perspective the most substantive potential impacts of the project are associated with the long term management of waste rock, tailings, mine water, and process water due to high quality fish habitat in potential receiving environments, flow rates in potential receiving environment streams that vary substantially through the year with relatively low winter flows, and acknowledgement that the Tse

Keh Nay ascribe great value to water bodies in the receiving environment such as Thutade Lake and the Finlay River. With the application of appropriate engineering design, project planning and implementation of mine and environmental management plans, it is anticipated that the project will avoid any significant environmental effects.

19.1.2 Aboriginal and Public Consultations

AuRico has engaged substantively with the surrounding communities to inform them of plans and gain their support for the KUG project. Since 2010, AuRico has continued to meet regularly and consult with the Tse Keh Nay prior to and during the EA process. These meetings helped to identify Tse Keh Nay members' Aboriginal rights and interests that may be affected by the KUG project, provide updated information about the KUG project, and build relationships. Consultations to date have been positive to support the EA review and permitting process.

AuRico undertook communications and consultations with the public between 2011 and 2015. This includes residents of northwest BC communities including Mackenzie, Smithers, Terrace, Prince George, as well as the local governments representing these communities; interest groups and non-governmental organizations; commercial interests and license and tenure holders; and recreational land-users.

19.1.3 Closure and Post-closure Water Management

At closure, mine dewatering will cease and water-retaining bulkheads will be installed in the declines to flood the underground workings. Once groundwater levels have returned to stable levels, contact water from the underground workings will discharge via natural groundwater flow paths to East Cirque Creek (situated about the KUG cave zone). Groundwater quality in the area of the underground mine is not expected to change materially from present conditions and long-term water treatment is not anticipated.

During initial closure phase, excess water within the KUG TSF will require ongoing treatment and controlled discharge to Attichika Creek. Water quality within the KUG TSF will improve over time with ongoing treatment. Once water quality within the KUG TSF reaches concentrations that would allow for untreated discharge to the receiving environment, active water treatment would cease and the project would transition to post-closure. In post-closure the KUG TSF will have a free overflow to Waste Rock Creek. It is anticipated that on-going water treatment will be required for six years following the end of KUG operations.

19.2 Environmental Regulatory Setting

The environmental assessment and permitting framework for metal mining in Canada is well-established. The federal and provincial environmental assessment (EA) processes provide a mechanism for reviewing major projects to assess their potential impacts. Before construction commences, the project undergoes a construction and operating licensing/permitting phase. The project is then regulated through all phases (construction, operation, closure, and post closure) by both provincial and federal departments and agencies. The legislated and procedural aspects of the process are discussed in the following sections.

19.3 Environmental Assessment Process

19.3.1 British Columbia Environmental Assessment Act Requirements

In British Columbia, the Environmental Assessment Office (EAO) manages the assessment of proposed major projects as required by the British Columbia (BC) Environmental Assessment Act (BC EAA, 2002). The BC Environmental Assessment Act (2002) requires that projects which exceed thresholds defined in the Reviewable Projects Regulation (BC Reg. 370/2002) must undergo an EA and obtain an Environmental Assessment Certificate (EAC) before a project can proceed. Section 7(1) of the BC EAA (2002) also allows a proponent who proposes a project that is not a reviewable project under the BC EAA or under the regulation to apply for the project to be designated as a reviewable project.

Therefore, mineral mining projects in BC become reviewable in three ways:

1. The Reviewable Projects Regulation (BC Reg. 370/2002) provides for a mining project to be automatically reviewable if it meets the following thresholds:
    a. Any new mine with 75,000 tonnes of mineral ore production per year; and
    b. Modifications to a mine that result in either 50% or more increase in area of mine disturbance, or 750 ha or more new disturbance.
2. Ministerial Designation by the Minister of Environment who has the authority to direct the review of a project which is not automatically reviewable under the Reviewable Projects Regulation (BC Reg. 370/2002).
3. Proponent "opt-in" whereby a proponent may request that the EAO consider designating its project as a reviewable project, and the EAO agrees with and orders such a designation.

With respect to this project, the KUG project did not meet thresholds for a modification to an existing facility under section 7(1) of the Reviewable Projects Regulation (BC Reg. 370/2002). Although the production capacity exceeds the regulation trigger of 75,000 tonnes per year of mineral ore, it will not exceed the threshold of either 750 hectares of new land disturbance or a 50% increase in land disturbance. Accordingly, the proposed project did not automatically trigger the need for a provincial review under the BC Environmental Assessment Act as a reviewable project. Aurico considered this fact along with the potential to de-risk project approvals through the EA process with prescribed timelines, and therefore AuRico requested the KUG project to be designated as a reviewable project, and on February 7, 2014 under section 7(3) of the Act was designated as a reviewable project.

Once the project was deemed reviewable, a Project Description (PD) was submitted to the EAO in February 2014, and the EAO issued an Order under section 10(1)(c) of the Act, requiring that the project undergo an EA and obtain an EAC. The EAO subsequently issued an Order under section 11 of the Act, which specifies the scope of the project, the scope of the assessment and establishes reporting and consultation requirements during the EA process. The provincial EA process is supported by three key documents: a Valued Component (VC) Scoping Guidance document, an Application Information Requirements (AIR) document which acts as a terms of reference, and an

Application for an Environmental Assessment Certificate which contains the effects assessment analyses, mitigation, and environmental management plans.

As per the Valued Component Scoping Guidance document, AuRico submitted a draft of the Valued Component Scoping Summary to the EA working group for review on 6 June 2014. After making revisions to address the EA working group comments, the draft Valued Component Scoping Summary was posted for public comment from 26 November 2014 to 5 January 2015. No public comments were received and the November 2014 copy of the VC Scoping Document was approved by the EAO.

AuRico submitted a draft of the AIR document to the EA working group for review on 10 October 2014. After undergoing rounds of review by the EA working group, the AIR was approved by the EAO in January 2016 and posted on the EAO's Project Information Centre (EPIC). The EA application is complete and was submitted for pre-screening to the Tse Keh Nay on 11 January 2016, and on 3 March 2016 the application was submitted to the EAO for screening review. The 180 day EA application review period will begin once the EAO completes a screening review of the application to determine concordance with the AIR.

The BC EAA and accompanying regulations and guidance documents establish the overarching regulatory framework for undertaking EAs in BC. Within this framework, the scope, procedures, and methods of each assessment are tailored specifically to the circumstances of the proposed project. This approach allows for each assessment to focus on key issues relevant to the project when determining whether or not the project should proceed.

The decision to approve or reject a provincial EA for a mining project is made by the Minister of Energy and Mines and by the Minister of Environment. In making their decision, the ministers must consider the assessment report (prepared by the EAO) and any recommendations accompanying the assessment report, and may consider any other matters they consider relevant to the public interest. If the project is allowed to proceed, an EAC is issued under Section 17(3) of the EAA with conditions to the certificate that the ministers consider necessary. An EAC is comprised of general provisions, a description of the physical works and activities of the project (Schedule A, Certified Project Description [CPD]), and conditions (Schedule B, Table of Conditions [ToC]). Issuance of an EAC allows statutory decision makers to issue necessary permits and authorizations to proceed with physical activities and development of project components considered in the EA.

19.3.2 Canadian Environmental Assessment Act, 2012 Requirements

The legislative basis for federal EA requirements is the Canadian Environmental Assessment Act, 2012 (2012). The Regulations Designating Physical Activities (SOR/2012-147) identify thresholds for projects that may be subject to a federal EA process.

With respect to this project, the KUG project exceeds production rate thresholds for a gold mine of 600 t/d (section 16 (c)), therefore a PD was submitted to the CEA Agency, for the purposes of determining whether a federal EA was required for the Designated Project. The PD was then screened to determine if there are potential adverse effects in areas of federal jurisdiction under section 5 of CEAA (2012) and was posted for public comment. Subsequent to these steps, the

federal Minister of the Environment determined that a federal EA must be completed. On 8 April 2014 the Minister of the Environment notified AuRico that a federal EA would be required, and approved the substitution of the federal EA by that of the Government of BC. Although the procedural aspects of substituted EA projects are undertaken by the EAO, both governments still make a decision to approve or deny the issuance of an EA Certificate or EA Decision Statement. The substituted EA process is described in more detail in following sections.

Under section 52 of CEAA (2012), the federal decision to approve or reject the EA for designated projects where the CEA Agency is the responsible authority, is made by the Minister of the Environment. In making a ministerial decision for substituted projects, the information presented in EAO's final Assessment Report and the implementation of mitigation measures are considered with respect to the potential for the designated project to cause significant adverse environmental effects. Significant adverse environmental effects are not anticipated in relation to the KUG project. If it is decided that the designated project is not likely to cause significant adverse environmental effects, the Minister of the Environment will issue an EA Decision Statement containing the conditions that AuRico must meet, including any mitigation measures and/or follow-up programs. Federal authorities are prohibited from exercising any power or performing any duty or function that could permit a project to be carried out in whole or in part unless the EA Decision Statement indicates that the designated project is not likely to cause significant adverse environmental effects or that significant adverse environmental effects are justified in the circumstances.

19.3.3 Substituted Assessment Process for the Designated Project

Substituted EA projects are conducted by the EAO and CEA Agency in accordance with a Memorandum of Understanding (BC EAO 2013a) that establishes expectations, roles, and procedures for implementing substituted EA review processes in BC. On 19 February 2014, the EAO requested substitution for the KUG project under section 32 of CEAA (2012), committing to fulfill the conditions for substitution under section 34 of CEAA (2012). On 8 April 2014 the Minister of the Environment approved the substitution of the federal EA by that of the provincial EA process administered by the BC EAO.

The following conditions must be met for a substituted EA process:

1. The substituted process will include a consideration of the factors set out in subsection 19 (1) of CEAA (2012).
2. The public will be given an opportunity to participate in the environmental assessment.
3. The public will have access to records in relation to the environmental assessment to enable their meaningful participation.
4. At the end of the environmental assessment, BC will submit a report to the CEA Agency that includes the findings and conclusions of the environmental assessment with respect to the factors as set out in subsections 19 (1) of CEAA (2012).
5. The report will be made available for public review.

The federal Minister of the Environment has also established additional conditions in its substitution decision for the proposed project, as follows:

1. BC will involve expert federal authorities in the BC process.
2. BC will provide the environmental assessment report to the CEA Agency within the time frame that will enable the Minister to make decisions under subsection 52 (1) of CEAA (2012) within the time limits set out in CEAA (2012).
3. BC will conduct procedural aspects of Aboriginal Consultation in accordance with the process set out in the Memorandum of Understanding.
4. BC will make available to Aboriginal groups funding provided by the CEA Agency to support Aboriginal Consultation during the substituted environmental assessment (BC EAO 2013c).

The substituted assessment process consists of three phases (Figure 19.1):

- Phase 1: Substitution Decision – The PD is reviewed by both governments and, if requested, a substitution decision is made.
- Phase 2: Pre-Application & Application Review – The EAO works with the proponent to conduct the EA according to prescribed timelines. The CEA Agency acts as a liaison, reviewing and providing feedback as necessary on relevant sections of EA documents, including participating in the review of the draft Assessment Report and proposed conditions. The EAO produces a final Assessment Report containing the findings and conclusions of the EA process.
- Phase 3: Decision Process – The EAO and CEA Agency make a joint briefing to decision-makers and refer the project for an EA decision. Both governments coordinate the announcement and issuance of EA approvals. Once the decisions have been taken, the Assessment Report will be made available to the public.

AuRico has successfully completed Phase 1 of the substituted EA process and submitted an Application for an Environmental Assessment Certificate to the BC EAO on 3 March 2016. Following completion of the initial application screening on April 22nd, it is expected that that project will enter the Application Review stage in mid-Q2 2016.

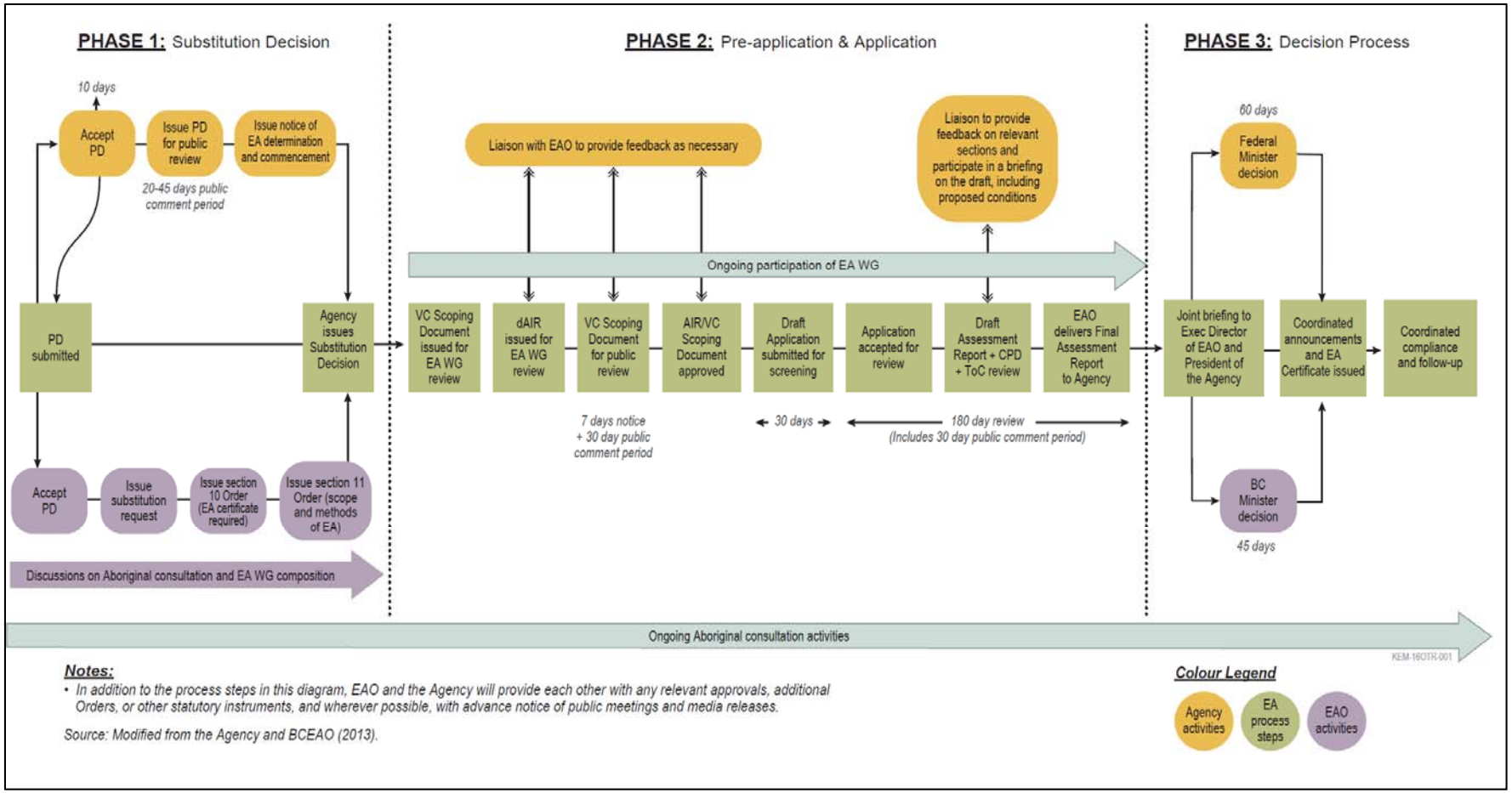


Figure 19.1: Assessment process for substituted environmental assessments

(Source: ERM, 2016)

19.4 Environmental Licensing Process

Following a successful EA, the KUG project will be required to obtain a number of provincial and federal licenses/permits. This process can generally be initiated, in part, during the final stages of the EA. If all applications for licenses and permits are completed, submitted, and reviewed prior to the EAC issuance, then the statutory decision makers are able to write the authorizations following a Ministerial decision allowing the project to proceed. This drafting of final licenses and permits may take an additional period of two months. AuRico intends on submitting provincial permit applications required for the KUG project during the Application Review stage. The permitting process will be led by the BC Major Mines Permitting Office (MMPO). No decisions on permits can be made by provincial agencies until the EAC is granted. Federal permit authorizations will also be submitted synchronously.

A number of existing permits for the Kemess South Mine have been in place since 1996 that remain in good standing, authorizing the use of roads, the power line, explosives magazines, camp and process plant related facilities/functions (e.g., nuclear gauge and boiler use). Table 19.1 summarizes the permits held by AuRico for the Kemess South Mine that are also applicable to the KUG project; these permits may require amendment or renewal before construction begins.

Table 19.2 presents the list of provincial and federal authorizations, licences, and permits that are anticipated to be required for the construction and/or operation of the KUG project. The list is not intended to be exhaustive and does not suggest the permitting methodology which may include amendment of existing permits or the requirement for new applications. AuRico has been engaged with the MMPO since August 2015 and will continue to meet with the appropriate governmental agencies to discuss the permitting requirements and approach to the KUG project.

Table 19.1: Existing Kemess South Mine Authorizations

Authorization	Date Issued	Statute	Authorized by
Project Approval Certificate M96-03	29-Apr-96	Environmental Assessment Act (S.B.C. 1994)	BC Environmental Assessment Office
Permit approving the Work System and Reclamation Program (Permit M206)	12-Jul-96	Mines Act (1996f)	BC Ministry of Energy and Mines
Mining Lease (354991)	-	Mineral Tenure Act (1996e)	BC Ministry of Energy and Mines
Effluent Discharge Permit (PE 15335):	08-Dec-98	Environmental Management Act (EMA) (2003),Municipal Sewage Regulation (BC Reg. 129/99)	BC Ministry of Environment
Solid Waste Permit (PR 14928)	09-Sep-97	Environmental Management Act (2003), Municipal Waste Water Regulations (BC Reg. 87/2012)	BC Ministry of Environment
Water License (Conditional) (C110454): Kemess Lake	Re-issued July 23, 2015	Water Act (1996g), Water Protection Act (1996h)	BC Ministry of Forests, Lands and Natural Resource Operations
Water License (Conditional) (C118554): East Kemess Creek, South Kemess Creek, Kemess Creek, Serrated Creek and Attichika Creek	Re-issued July 23, 2015	Water Act (1996g), Water Protection Act (1996h)	BC Ministry of Forests, Lands and Natural Resource Operations
Road Use Permit (RUPOTHO 8003): Omineca Resource Access Road	-	Forest Act (1996b)	BC Ministry of Forests, Lands and Natural Resource Operations (bonding by MOTI)
Special Use Permit (S24513): Access Roads to the power line	26-Oct-06	Forest Act (1996b); Provincial Forest Use Regulations (B.C. Reg. 176/95)	BC Ministry of Forests, Lands and Natural Resource Operations
License of Occupation (703497): Power Line alignment	04-Feb-97	Land Act (1996d)	BC Ministry of Forests, Lands and Natural Resource Operations
Special Waste Generator Identification number: BCG07761	18-Feb-98	Environmental Management Act (2003), Hazardous Waste Regulation (BC REG. 63/88)	BC Ministry of Environment
Gas Permits (2172133)	31-Oct-15	Safety Standards Act (2003)	BC Safety Authority
Boiler, Pressure Vessel and Refrigeration Permit (Various)	12-Feb-15	Safety Standards Act (2003)	BC Safety Authority
Food Premises Permit (36704)	-	Public Health Act (2008), Food Premises Regulation (BC Reg. 210/99)	BC Ministry of Health, Northern Health Authority
Source Water Approval (Potable Water)	29-Jun-15	Drinking Water Protection Act (2001)	Northern Health Authority
Authorization	**Date Issued**	**Statute**	**Authorized by**
Radio License (80221125)	-	Radiocommunication Act (1985c)	Industry Canada
Nuclear Substances License – Fixed gauges (15710-1-17.0)	Re-issued November 10, 2015	Nuclear Safety and Control Act (1997)	Canadian Nuclear Safety Commission
Explosives Magazine License (BC-682): The main magazine storage of explosives and detonators including avalanche control explosives and detonators	Re-issued July 3, 2015	Explosives Act (1985a)	Natural Resources Canada
Corporate Firearm Licence (11750426.0010)	-	Firearms Act (1995), Authorizations to Carry Restricted Firearms and Certain Handguns Regulations (SOR/98-207)	Justice Canada

Table 19.2: Anticipated Provincial and Federal Authorizations, Licences, and Permits Required for Kemess Underground Project

Authorization	Date Issued	Statute	Authorized by
Project Approval Certificate M96-03	29-Apr-96	Environmental Assessment Act (S.B.C. 1994)	BC Environmental Assessment Office
Permit approving the Work System and Reclamation Program (Permit M206)	12-Jul-96	Mines Act (1996f)	BC Ministry of Energy and Mines
Mining Lease (354991)	-	Mineral Tenure Act (1996e)	BC Ministry of Energy and Mines
Effluent Discharge Permit (PE 15335):	08-Dec-98	Environmental Management Act (EMA) (2003),Municipal Sewage Regulation (BC Reg. 129/99)	BC Ministry of Environment
Solid Waste Permit (PR 14928)	09-Sep-97	Environmental Management Act (2003), Municipal Waste Water Regulations (BC Reg. 87/2012)	BC Ministry of Environment
Water License (Conditional) (C110454): Kemess Lake	Re-issued July 23, 2015	Water Act (1996g), Water Protection Act (1996h)	BC Ministry of Forests, Lands and Natural Resource Operations
Water License (Conditional) (C118554): East Kemess Creek, South Kemess Creek, Kemess Creek, Serrated Creek and Attichika Creek	Re-issued July 23, 2015	Water Act (1996g), Water Protection Act (1996h)	BC Ministry of Forests, Lands and Natural Resource Operations
Road Use Permit (RUPOTHO 8003): Omineca Resource Access Road	-	Forest Act (1996b)	BC Ministry of Forests, Lands and Natural Resource Operations (bonding by MOTI)
Special Use Permit (S24513): Access Roads to the power line	26-Oct-06	Forest Act (1996b); Provincial Forest Use Regulations (B.C. Reg. 176/95)	BC Ministry of Forests, Lands and Natural Resource Operations
License of Occupation (703497): Power Line alignment	04-Feb-97	Land Act (1996d)	BC Ministry of Forests, Lands and Natural Resource Operations
Special Waste Generator Identification number: BCG07761	18-Feb-98	Environmental Management Act (2003), Hazardous Waste Regulation (BC REG. 63/88)	BC Ministry of Environment
Gas Permits (2172133)	31-Oct-15	Safety Standards Act (2003)	BC Safety Authority

Authorization	Date Issued	Statute	Authorized by
Boiler, Pressure Vessel and Refrigeration Permit (Various)	12-Feb-15	Safety Standards Act (2003)	BC Safety Authority
Food Premises Permit (36704)	-	Public Health Act (2008), Food Premises Regulation (BC Reg. 210/99)	BC Ministry of Health, Northern Health Authority
Source Water Approval (Potable Water)	29-Jun-15	Drinking Water Protection Act (2001)	Northern Health Authority

19.5 Environmental Considerations

The KUG project is in essence the addition of an undeveloped ore body to an existing mining project. As such, it has the advantage of being able to make use of significant environmental data gathered throughout two previously completed EAs as well as 15 years of data gathered during development and operations.

The Kemess South Mine completed an environmental assessment prior to its commissioning in Q2 1998. The results of this assessment and subsequent permits identified a number of environmental management plans which guided the operation of the Kemess South Mine and documented, through a variety of monitoring programs, the interactions of the mine and the environment. Additional recent baseline environmental (biological and physical components) and social data have been gathered for the proposed KUG project. All of this data is available to support the KUG project through the required EA and permitting/licensing requirements.

The EA process was followed as described in the preceding environmental sections. Scoping was conducted to ensure that the EA application focused on the issues with the greatest potential to cause significant adverse effects on selected valued components (VCs). A list of VCs was compiled and reviewed early in project planning based on the KUG project design, local area and region where the KUG project is located, and publicly available scientific information. VCs selected for the KUG project effects assessment are summarized in Table 19.3. The VCs are grouped into five assessment pillars: environment, social, economic, heritage, and health. These VCs represent a broad spectrum of the environmental components that require consideration in the KUG project planning and development.

The KUG project infrastructure (including the KUG TSF) is within the Kemess Creek watershed. Kemess Creek flows from the northeast to the southwest immediately south of the Kemess South mine site into Attichika Creek, which is a tributary of Thutade Lake. Thutade Lake outlets northward and enters the Finlay River. The Kemess Underground subsidence zone above the ore body is within the Attycelley Creek watershed, including the East Cirque Creek subwatershed. Attycelley Creek flows westerly and enters the Finlay River, approximately 200 metres downstream from the Thutade lake outlet.

There will be loss and disturbance to terrestrial areas associated with KUG, however, this will be minimized by optimizing the use of existing infrastructure features from the Kemess South Mine. The Kemess South Mine footprint previously resulted in a terrestrial area loss and alteration of 2,296 ha. The KUG project is expected to result in an additional terrestrial area loss of less than 100 ha which includes the 35 ha projected KUG cave subsidence zone.

From an environmental perspective the most substantive potential impacts are associated with the long term management of waste rock, tailings, mine water, and process water. This perspective is based upon: high quality fish habitat in potential receiving environments such that stringent limits on effluent water quality will be applied; flow rates in potential receiving environment streams vary substantially through the year with relatively low winter flows; and Tse Keh Nay ascribe great value to water bodies in the receiving environment such as Thutade Lake and the Finlay River.

Table 19.3: Valued Components Selected for the Environmental Assessment

Assessment Pillar	Subject Area	Valued Components
Environment	Hydrogeology	Groundwater quantity Groundwater quality
	Surface hydrology	Surface hydrology
	Surface water quality	Surface water quality
	Terrain and soils	Terrain stability Soil quantity Soil quality
	Terrestrial ecology	Alpine and Parkland Ecosystems Forested Ecosystems Wetland Ecosystems Red and Blue-listed Ecosystems Harvestable plants; Rare plants and lichens and associated habitat
	Fish and aquatic habitat	Adfluvial bull trout Dolly Varden Rainbow trout Periphyton Benthic invertebrates Sediment quality
	Wildlife	Woodland caribou Mountain goat Moose Grizzly bear Hoary marmot Furbearers (using American marten and wolverine) Migratory landbirds Migratory waterbirds Raptors Bats Western toad
Economic	Economic	Aboriginal labour market conditions Non-Aboriginal labour market conditions
Social	Social	Community well-being Aboriginal community well-being
Health	Human health	Human health
Heritage	Heritage resources	Physical and cultural heritage resources (including any structure site or thing of historical, archaeological or architectural significance) Paleontological resources
Canadian Environmental Assessment Act (CEAA) section 5(1)(c)	Effects of changes to the environment on Aboriginal peoples	Health and socio-economic conditions Physical and cultural heritage Current use of lands and resources for traditional purposes Any structure, site or thing of historical, archaeological, paleontological or architectural significance

During construction, operation and closure, discharge and treatment of excess supernatant in the KUG TSF is a primary consideration which results in water treatment being required for dissolved metals and selenium; staging of discharge to match natural streamflow fluctuations in the receiving environment during the construction phase; and limiting discharge to the open water period of approximately May through October. During post-closure, the flooding of the underground mine and potential long-term effects on the aquatic environment in the Attycelley Creek watershed are important. Potential near-field receiving environments of Central and East Cirque Creeks have relatively low importance in terms of aquatic habitat due to naturally elevated levels of many water quality parameters of potential concern; however, there are aquatics and fish habitat downstream in Amazay Lake and Attichika Creek that warrant protection. In addition, AuRico has an agreement with the Tse Keh Nay to avoid water quality impacts on Amazay Lake. To avoid potential impacts to Central Cirque Creek and Amazay Lake, hydraulic bulkheads will be strategically placed to ensure that all contact water from the KUG cave and subsidence zone will discharge to East Cirque Creek where post-closure quality of the contact water in the underground is not anticipated to be substantially different from current conditions.

There are a number of wildlife species of interest and conservation concern in the KUG project area. These wildlife species include moose, woodland caribou, grizzly bear, and wolverine. All of these species, except moose, are provincially blue-listed threatened species. Woodland caribou and grizzly bear are federal species of special concern and protected under Schedule 1 of the Species at Risk Act (2002b). The potential environmental impacts of the KUG project will have mitigative measures as necessary to address potential impacts to wildlife. These effects will be largely mitigated by traffic and waste management strategies.

The potential environmental impacts of the KUG project will have mitigative measures as necessary to address these potential impacts. All impacts can be mitigated through the implementation of good engineering practices, industry best practices, and adaptive management plans. Some of these can also be managed as per existing management strategies in accordance with existing authorizations and permits for the Kemess South Mine.

19.6 Social Setting

The Project is located in the Peace River Regional District of northern British Columbia. The closest community by road is the District of Mackenzie (400 km south of the Project, population 3,500). There are also small settlements accessed by forest service roads, including Germansen Landing (230 km south of the mine, population approximately 44) and Manson Creek (250 km south of the mine, population approximately 40). The closest communities by air are the Aboriginal communities of Kwadacha (alternately known as Fort Ware; approximately 70 km northeast of the mine, population 300), Tsay Keh (alternately known as Ingenika Point; approximately 120 km south-east of the mine, population 220), and Takla Lake (approximately 180 km south-west of the mine, population 400).

The Project is located within the traditional territories of the Takla Lake First Nation (TLFN) and Tsay Keh Dene Nation (TKDN). The traditional territory of the Kwadacha Nation (KwN) is adjacent to and downstream from the Project. In relation to the Project, these three First Nations are jointly represented by the Tse Keh Nay (TKN) based on their common history, culture and language and

deep social and political ties. The Gitxsan Wilp Nii Kyap traditional territory lies adjacent to the Project along the western boundary. The Metis Nation of BC (MNBC) is also represented throughout the regional area including “chartered community” associations in Smithers, Terrace, Chetwynd, Prince George, and Fort St James. The Project is also located within the Treaty 8 disputed area.

The Project is located within a trapline registered to members of the Takla Lake First Nation. A trapline compensation agreement was in place during the operation of the Kemess South Mine, and a similar arrangement has been re-established for the KUG Project. AuRico is also negotiating an Impact-Benefit Agreement with the TKN, which may include provisions related to employment and training, land use, and other topics.

Workers and contractors for the Kemess South mine were from northern BC including the Smithers and Prince George areas. Kemess South also operated workforce flights from communities such as Williams Lake, Kamloops, Kelowna, and Vancouver, and the remainder of the workforce resided in various communities throughout the province. The distribution of employee locations and their corresponding percentage of the total work force changed over the life of the Kemess South operations. Aboriginal people made up 10% to 18% of the former Kemess South mine workforce for any given year. For the KUG Project, AuRico expects to hire a minimum 50% of the workforce from northern BC, and a minimum 10% of the workforce from Aboriginal groups.

The closure of the Kemess South mine resulted in job losses for the workforce. Job losses were mitigated through a successful transition program via the Kemess Joint Workplace Transition Committee. This employee-focused program provided resources and communication to ensure that all workers were prepared for mine closure and had access to new opportunities including job search assistance.

The development of the KUG Project would rejuvenate the economic and social benefits provided by the Kemess South mine. The Project will provide employment, procurement, and government revenue benefits on a similar scale to those provided by the Kemess South mine. The KUG project would provide significant positive economic benefits, as well as contributing incrementally to diversifying the northern BC economy in the face of the current downturn in the forestry sector due to market conditions and the pine beetle infestation, and current cyclical downturn in the coal and metal mining sectors.

19.6.1 Aboriginal Consultations

The BC EAO delegates the procedural aspects of the Province’s duty to consult with Aboriginal groups to proponents during the EA process. Requirements for Aboriginal consultation requirements are identified in the Section 11 Order. The three TKN First Nations are the primary Aboriginal groups that AuRico must engage with during the EA process. Additional engagement activities, though to a lower level, are required for the Gitxsan Wilp Nii Kyap, the MNBC, and the eight Treaty 8 member First Nations.

Prior to entering the EA process, AuRico initiated consultations with the TKN in 2010 with a meeting held in Prince George. The purpose of the meeting was to describe the proposed KUG project and

discuss TKN's preferred approach to consultation, including possible agreements with TKN related to the KUG project. AuRico continued to meet with TKN and following a series of 10 meetings in Prince George, the TKN communities, and Toronto, signed an Interim Measures Agreement (IMA) in 2012.

The Senior Implementation Committee (SIC) and the Environmental Management Committee (EMC), established by the IMA, have been meeting on a monthly (and bi-weekly when needed) basis since May 2013. These committees provide a forum for discussing TKN members' potential Aboriginal rights and interests that may be affected by the KUG project, provide updated information about the KUG project, and build relationships. A subcommittee on employment and contracting opportunities was also established within the SIC.

The BC EAO established an EA Working Group in May 2014 which includes representatives of the Schedule B Aboriginal groups (i.e., TLFN, TKDN, and KwN), federal and provincial government agencies, and local government (i.e., District of Mackenzie, the Town of Smithers and the Peace River Regional District). A number of Working Group meetings and teleconferences were held between June 2014 and December 2015 (Table 19.4). In addition, AuRico has hosted site visits for TKN representatives and members of TKN First Nations in 2011, and 2013 to 2015.

Since the start of the EA process, AuRico has hosted nine meetings about the KUG Project in TKN communities. Table 19.5 summarizes the community meetings AuRico has held with TKN communities to date. In addition, there was one meeting with TKN in Prince George.

An Aboriginal Consultation Plan (ACP) was developed in accordance with the Section 11 Order which delegates procedural aspects of the duty to consult to the proponent via an ACP. In addition, an Interim Pre-Application Aboriginal Consultation Report (ERM 2015) documents AuRico's consultations with the TKN and other Aboriginal groups from 2010 until 2014. A second report was submitted to the BC EAO in February 2016. TKN reviewed and provided input to the ACP as well as the reports.

Through a collaborative approach established between AuRico and TKN, AuRico shared the draft VC Scoping Summary and draft AIR first with the TKN for their review and comment prior to submitting these documents to the BC EAO. In November 2014, AuRico discussed the valued components related to the Project and the status of the EA process at community meetings in Kwadacha, Tsay Keh, and Takla Landing.

Table 19.4: Summary of Working Group Meetings

Meeting Date	Topics	Location	Aboriginal Group	Government Agencies
June 25, 2014	• Site visit	Site visit	TLFN TKDN KwN	BC EAO BC Ministry of Energy and Mines (BC MEM) BC Ministry of Environment (BC MOE) BC Ministry of Forests, Lands and Natural Resource Operations (BC FLNRO) Environment Canada (EC)
June 26, 2014	• EA Process and Working Group • Project Overview • Valued Components	Prince George	TLFN TKDN KwN	BC EAO BC MEM BC MOE BC FLNRO EC Natural Resources Canada (NRCan) Transport Canada (TC) NH
July 21, 2014	• Fish Tissue Sampling	Teleconference	TLFN TKDN KwN	BC EAO BC MOE Health Canada (HC)
July 24, 2014	• Health	Teleconference		NH
July 29, 2014	• Aquatic Predators	Teleconference	KwN TKN (Technical Coordinator)	BC EAO BC FLNRO
August 5, 2014	• Hydrogeology • Hydrology • Geochemistry • Surface Water Quality • EA Methodology	Vancouver	TLFN KwN TKN (Technical Coordinator and Advisor)	BC EAO BC FLNRO EC NRCan
October 9, 2014	• TK/TLUS • Human Health • Noise Modelling • Air Quality Modelling • Drinking Water Assessment • Country Foods Risk Assessment	Teleconference	TLFN KwN TKN (Technical Coordinator and Advisor)	BC EAO HC NH
November 20, 2014	• Aboriginal Consultation	Teleconference		BC EAO CEA Agency
March 17, 2015	• Approach to Assessment of Discharge Options • Review of Proposed Discharge Options • Preliminary Water Balance • Surface Water Quality and Treatment Options • Aboriginal Consultation	Vancouver		BC EAO BC MEM BC MOE BC FLNRO EC NRCan

Meeting Date	Topics	Location	Aboriginal Group	Government Agencies
July 16, 2015	• Discharge Alternatives Process • Community Meeting Outcomes • Water and Water Discharge • Water Quality Modelling Results • Treatment Options and Economics • Evaluation of Alternatives	/ancouver	KwN ᵀKDN ᵀLFN ᵀKN Advisor)	BC EAO BC FLNRO EC
October 23, 2015	• Aquatic predators	ᵀeleconference		BC EAO BC FLNRO BC MOE
November 5, 2015	• Attichika Creek Water Quality Discharge Predictions	/ancouver		BC EAO BC MOE BC MEM BC MMPO
November 12, 2015	• ORAR in context of AIR	/ancouver		BC EAO BC FLNRO BC MMPO
November 20, 2015	• Assessment of Alternative Technologies for Tailings Storage	/ancouver		BC EAO BC MEM BC MOE BC MMPO
December 1, 2015	• ORAR monitoring	ᵀeleconference	ᵀLFN	BC EAO

Through the EMC, the TKN has had the opportunity to review and comment on the draft baseline reports that support the EAC Application. AuRico engaged the TKN in the assessment of various water discharge alternatives, with the aim of having the TKN involved in the decision-making process and ensuring that TKN values and information were represented in the selection of the water discharge location. Furthermore, AuRico has established the following agreements with the TKN: Interim Measures Agreement (June 2012), Exploration Agreements, Joint Communication Plan (March 2015), Environmental Assessment Conduct Agreement (June 2014), and Employment and Contract Opportunity Framework (March 2015).

AuRico has also had discussions with Gitxsan Nation and Wilp Nii Kyap, a hereditary house of the Gitxsan Nation, since 2011.

In 2015, AuRico sent a letter via email to the Chief of Wilp Nii Kyap, each of the eight Treaty 8 First Nations, and MNBC providing an update about the Project and describing the process for collecting information and assessing potential effects in accordance with Section (5)(1)(c) of CEAA (2012). AuRico provided each group with a summary report for review outlining its understanding of the group's existing health and socio-economic conditions, current use of lands and resources for traditional purposes, and physical and cultural heritage in relation to the KUG project, as well as the related Aboriginal rights and interests, and areas of missing or unavailable information. AuRico also provided a letter and draft report for review describing the assessment of potential effects of environmental changes caused by the KUG project on Wilp Nii Kyap, each of the Treaty 8 First Nations, and MNBC, in accordance with Section 5(1)(c) of CEAA (2012).

Table 19.5: Summary of Community Meetings in Tse Keh Nay Communities

Date	Location	Venue
09-Jul-15	Takla Landing	Potlach House
07-Jul-15	Tsay Keh	Training Centre
06-Jul-15	Kwadacha	Community Health and Recreation Centre
13-Nov-14	Takla Landing	Potlatch House
06-Nov-14	Tsay Keh	Training Centre
05-Nov-14	Kwadacha	Administrative Building/ Band Office
08-May-14	Takla Landing	Potlatch House
06-May-14	Tsay Keh	School
05-May-14	Kwadacha	Aatse Davie School
30-Apr-14	Prince George (TKN)	Coast Hotel

19.6.2 Public Consultations

Public consultation requirements for the provincial EA process are set out in the BC Environmental Assessment Act (2002) Public Consultation Policy Regulation (BC Reg. 373/2002). The Regulation identifies public consultation that must be undertaken by a proponent, and requirements related to public notice, public access to information and public comment periods.

AuRico undertook communications and consultations with the public between March 2011 and September 2015. As described in the Public Consultation Plan for the KUG project, the term “public” includes: residents of northwest BC communities including Mackenzie, Smithers, Terrace, Prince George, as well as the local governments representing these communities; interest groups and non-governmental organizations; commercial interests and license and tenure holders; and recreational land-users.

Consultation activities included a public comment period on the VC Scoping Summary, open houses in four communities in northern BC, and meetings with municipal governments. Land users and tenure holders with interests in the Project area were contacted, and some provided information via interviews and/or questionnaires supporting a land use baseline study.

The EAO notified the public in November 2014 regarding the opportunity to comment on the VC Scoping Summary. The BC EAO held a public comment period between November 2014 and January 2015. The notice identified methods for submitting comments as well as the locations for viewing copies of document, and how to submit comments.

AuRico provided project overviews to local governments in Terrace, Smithers, Prince George, District of Mackenzie and the Peace River Regional District. AuRico has also provided project updates through meetings with the District of Mackenzie (Table 19.6).

AuRico hosted open houses for the public in order to share information about the KUG project. Local governments and organizations (e.g., chambers of commerce) were advised of these events and invited to participate. Table 19.7 lists the dates and locations of public community meetings. These events were additional and complementary to community events held in Aboriginal communities.

AuRico presented the project to the public and other stakeholders at meetings, conferences and through panel discussions. Table 19.8 summarizes AuRico’s participation and presentations in conferences and panels up to September 2015.

Table 19.6: Meetings and Presentations to Local Governments

Organization	Date	Type of Communication	Topic
City of Terrace	12-Nov-13	Presentation at Council Meeting (Terrace, BC)	Project Overview
Peace River Regional District	14-Nov-13	Presentation at Board Meeting (Dawson Creek, BC)	Project Overview
City of Prince George	18-Nov-13	Presentation at Council Meeting (Prince George, BC)	Project Overview
District of Mackenzie	25-Nov-13	Presentation at Council Meeting (Mackenzie, BC)	Project Overview
City of Smithers	26-Nov-13	Presentation at Council Meeting (Smithers, BC)	Project Overview
District of Mackenzie	29-Jan-14	Meeting (Vancouver, BC)	Project Update
District of Mackenzie	02-May-14	Meeting (Mackenzie, BC)	Project Update
District of Mackenzie	30-Jan-15	Meeting (Vancouver, BC)	Project Update

Table 19.7: AuRico Public Community Meetings

Date	Location	Venue
28-Apr-14	Terrace, BC	Northwest Community College
29-Apr-14	Smithers, BC	Northwest Community College
30-Apr-14	Prince George, BC	Coast Hotel
01-May-14	Mackenzie, BC	St. Peter's Church

Table 19.8: AuRico Participation in Conferences and Panels

Organization	Date	Communication Method (Location)	Topic
District of Mackenzie Trade Show	2/3-May- 14	Booth (Mackenzie)	Project Overview
City of Smithers	09-May-14	Panel Discussion during Mining Week (Smithers)	Resources for Community Panel
Minerals North 2014	23-May-14	Presentation at Minerals North (Vanderhoof)	Project Overview
Premier's Resource Forum	21/22-Jan-15	Booth (Prince George)	Project Overview
AME BC Mineral Exploration Roundup	J26/27-Jan-15	Core shack booth (Vancouver)	Drill results
Kemess Exploration Group Conference	14/15-Apr-15	Booth and presentation (Kamloops)	Project Overview
Minerals North 2015	27/29-May-15	Booth, presentation and participation in core shack (Mackenzie)	Project Overview, drill results

19.7 Closure and Post-closure Water Management

The Kemess South Mine operated from 1998 until 2011. Parts of the mine have undergone reclamation whereas other areas have been maintained on a care and maintenance basis for use in potential future projects. Reclamation activities since closure of the Kemess South Mine have included re-contouring, capping and revegetation of the waste rock dumps, decommissioning of the tailings storage impoundment and associated works, and reclamation and revegetation of other lands affected by mine development. The scope of the closure and post-closure water management plan for the KUG project includes all new disturbances created by the KUG project and does not incorporate the areas described in the Reclamation and Closure Plan for the Kemess South Mine (Northgate 2010).

19.7.1 Underground Mine

At closure of the KUG project, mine dewatering will cease and water-retaining bulkheads will be installed approximately midway along the triple declines to control the groundwater flow path once the underground mine has flooded. The underground mine workings will continue to flood and the rebound of groundwater to stable levels are predicted to require approximately 36 years. Groundwater will ultimately drain via natural groundwater flow paths to East Cirque Creek. The strategic placement of the water-retaining bulkheads will ensure that contact groundwater will not discharge to Central Cirque Creek or to Amazay Lake. Groundwater modeling results show that there is no groundwater flow path (seepage) to Amazay Lake when the water-retaining plugs are installed midway along the declines (Lorax Environmental 2015a). Groundwater conditions will

reach a new, post-mining equilibrium condition; the water table configuration and groundwater flow paths will be permanently altered from the pre-mining conditions.

Once the workings flood and discharge to the East Cirque Creek, there will be additional monitoring to confirm that surface water quality in the creek is not a concern to the health of fish and aquatic resources in the downstream receiving environment. Treatment of decant from the subsidence zone is not anticipated as water quality modeling studies show small increases in concentrations for some contaminants of potential concern (COPC) above baseline concentrations, but there are no significant residual effects predicted from this discharge on surface water quality (ERM 2015; Lorax Environmental 2015b). Therefore, long-term water treatment is not anticipated.

19.7.2 KUG Tailings Storage Facility

Tailings and mine development waste rock will be stored in the Kemess South open pit (the proposed KUG TSF). The potentially acid generating (PAG) waste materials will be stored under a water cover to prevent metal leaching / acid rock drainage (ML/ARD). At closure, a non-acid generating (NAG) tailings beach extending from the East Dam to the supernatant pond will be present on the eastern end of the KUG TSF. As the KUG mine approaches the end of its operational life, a large above-water beach will exist on the east end of the KUG TSF.

During operations, process water from the KUG TSF supernatant pond will be reclaimed for use as mill process water and excess water treated and discharged to Attichika Creek. The sludge produced from the water treatment plants during operations will be sub-aqueously stored in the KUG TSF.

The closure phase will extend for the period of time (currently predicted to be six years) required for ongoing treatment of water within the KUG TSF and controlled discharge to Attichika Creek. Excess water in the KUG TSF supernatant pond will continue to be treated in the closure phase and thereafter until the water quality meets discharge criteria. The discharge rate from the KUG TSF will reduce to approximately 1.5 Mm3/yr (96 L/s) in the closure phase and be treated and discharged over a six month period. Continued operation of two water treatment plants will be required for metals removal (rated to treat 120 L/s) and selenium removal (rated to 50 L/s) throughout the active closure phase.

Once water quality within the KUG TSF reaches concentrations that would allow for untreated discharge to the receiving environment, active water treatment would cease and the KUG project would transition to post-closure. No water treatment is expected in post-closure as water quality modelling results indicate that there are no COPC downstream of the proposed discharge location (ERM 2015).

When water quality in the KUG TSF meets discharge criteria without treatment, the upslope diversion ditch will be re-graded to original elevation and this will allow catchment runoff and melt-water to flow into the KUG TSF and out through the closure spillway to Waste Rock Creek and ultimately to Attichika Creek. Long-term water treatment is not anticipated.

20 Capital and Operating Costs

20.1 Capital Cost Estimate

This section presents items associated with project capital costs. All figures are in Canadian dollar terms unless otherwise stated. The estimate was prepared to a level of accuracy consistent with feasibility study standards.

A total of C$950M in capital is estimated (CapEx) to be spent over the life of the project as summarized in Table 20.1.

Full pre-production expenditure including capitalised pre-production operating costs is estimated to be approximately C$524M to first ore production.

The total estimated expenditure to commercial production including capitalised pre-production operating expenditure (OpEx) is C$687M. This is partially offset by pre-commercial revenue such that the net cash requirement for the project development is C$604M (after adjustments for working capital and taxes).

Ongoing sustaining capital is approximately C$263M, including underground mobile equipment replacements and ongoing underground development.

Table 20.1: Summary Capital Cost Estimates

Item	To First Production (C$M)	Pre-Commercial[1] (C$M)	Sustaining (C$M)	Life of Mine (C$M)
Project Owners Costs	7	7	0	7
Mine	205	266	207	473
Mill	31	38	13	51
Site Service	13	13	9	22
Conveyor	40	40	0	40
UG Electrical	18	19	0	19
UG Ventilation	10	10	1	11
Access Corridor	36	36	0	36
Water Treatment Build	13	13	0	13
TSF and Pipeline Build	7	8	32	39
Reclamation Cost	0	0	1	1
Total Capital	**380**	**449**	**263**	**712**
Capitalised Pre-production Opex (COC)	144	238	0	238
Total Capital (Incl. COC)	**524**	**687**	**263**	**950**
Pre-commercial Net Revenue	N/A	(83)	N/A	(83)
Net Capex (Incl pre-commercial Opex and Net Revenue)	**524**	**604**	**263**	**867**

[1]Pre-commercial values include “to first production”

20.1.1 Basis of the Capital Cost Estimate

Currency

The estimates are expressed in Real 2016 Canadian dollars (C$). US dollar costs were converted using an exchange rate of 75.0 US cents per Canadian dollar.

No escalation or inflation factors were used in forward estimates.

Responsibility

The major contributors to the various sections have been outlined in Table 20.2.

Table 20.2: Responsibility for Capital Cost Estimates

Item	Responsibility
Owner's Costs	AuRico
Tailings Storage Facility	AMEC
Access Corridor	AMEC
Conveyor Installation Surface and Underground	CDI/AMEC
Process Plant	KWM
Sustaining Capital	AuRico/SRK/KWM/AMEC
Mine Closure and Rehabilitation	ERM/AuRico
Environmental Studies/Permitting/Social License	AuRico/SRK/ERM
Mine Development and Capital Installation	SRK
Mine Equipment	SRK
Mine Ventilation	SRK/MVS
Mine Electrical Reticulation	Struthers Technical
Water Treatment Plants	BioteQ
Contingency	AuRico/AMEC/CDI/SRK

Exclusions from the Capital Cost Estimate

The following items were specifically excluded from the Capital Cost Estimate:

- Project finance interest, leasing, off-take agreements, reclamation bonding, and other financing arrangements and costs. 100% equity financing is assumed;
- Escalation due to currency inflation;

- Exchange rate fluctuations;
- All license fees;
- Allowances for special incentives (schedule/safety or others);
- Sunk costs;
- Future scope changes and modifications of the proposed FS design; and
- Unforeseen delays due to weather or labour action.

20.1.2 Underground Mine Capital Cost Estimate

Summary of Assumptions for Estimate

It will take approximately four years to develop the KUG project, from first development to commercial production. Access will be via a corridor from the existing Kemess South (KS) plant site to the portal entrances.

Approximately 47,750 m of lateral development will be required to access approximately 107 Mt of ore that will be mined from the KUG deposit using the panel caving mining method. Access to the KUG mine will be by triple declines; one providing general access, another providing the intake air for mine ventilation, and the third for a conveyor delivering 9 Mt/y of ore to the existing Kemess South processing plant.

Ore will be mucked from the extraction level drawpoints and dumped directly to one of four primary jaw crushers located to the south of the cave footprint. Crushed ore will then be transported by one of two transfer conveyors and placed on a 3.2 km long underground conveyor that will transport the ore to surface. The material will then be transferred to a 4.9 km long overland conveyor that will deliver the ore to a stockpile adjacent to the Kemess South processing plant.

AuRico provided all the designs and schedule for the development of the panel cave mine, and SRK provided the cave production schedule. SRK developed the CapEx estimate for the items related to the underground mine design and schedule. The estimate for the underground mine capital costs were based on the following:

- Project development plan,
- Underground mining equipment list and allowance for major equipment rebuilds and equipment replacement,
- Budget quotes for the major equipment obtained by SRK from vendors,
- Budget quotes obtained by SRK from mining contractors for vertical development and major infrastructure, and
- SRK in-house cost database.

Other mining CapEx estimates are based on mining cost service information. No equipment was considered for lease. Underground equipment was deemed to be at or near the end of its lifecycle at the time of mine closure and therefore was not given a salvage value.

All contractor costs include labour. Contractor indirect labour cost includes contractor management and supervision. The owner labour cost was estimated as separate cost item based on labour rates and staff requirements for each mining activity including mine development, construction, maintenance, services, engineering, management and supervision.

The underground capital costs are presented in Table 20.3.

Table 20.3: CapEx Estimate – Underground Mine

Item	To First Production (C$M)	Pre-Commercial (C$M)	Sustaining (C$M)	Life of Mine (C$M)
Contractor Mobilization	1	1	0	1
Contractor Labour (without Raise Development and Construction)	2	2	2	4
Contractor Vertical Development (including labour)	7	9	3	11
Contractor Construction	6	6	3	9
Contractor Surface Trucking	5	6	4	10
Contractor Surface Crushing	3	3	3	6
Owner Development	54	78	65	143
Equipment Purchase and Replacement	66	86	61	147
Equipment Rebuilds	4	8	45	53
Owner Labour	58	67	21	87
UG Electrical	18	19	0	19
UG Ventilation	10	10	1	11
UG Conveyor Construction	18	18	0	18
Total Underground Capital (excl. capitalised Opex)	**251**	**313**	**208**	**520**

Underground Mine Development

All underground development is included in the capital cost estimate.

It was assumed that all underground lateral development from the triple decline portals would be carried out by owner crews. All underground raise development and construction of major infrastructure was assumed to be carried out by a mining contractor(s).

Underground mine development costs were estimated from first principles and were compared to the quotation documents provided by two mining contractors.

Where a mining contractor was assumed to be utilized, it was assumed the contractor would provide labour, equipment, and drilling consumables, with the remaining consumables provided by the owner. A 12% markup has been applied to all contractor expenses.

Equipment and material costs were taken from vendor quotes. Labour, fuel, and power costs were provided by AuRico. Fuel is assumed to cost C$0.91/litre; electric power is assumed to cost C$0.058/kWh as currently paid by AuRico.

Underground Mine Mobile Equipment

It was assumed the contractor would use its own equipment for all assigned development and construction work.

The estimated lease rates for the contractor equipment were based on the following inputs:

- Equipment budget price,
- 60 months for equipment depreciation period,
- 3% interest rate,
- 1.5% insurance, and
- 12% contractor mark-up.

The purchase of a mobile mining equipment fleet will be required for the underground mining operation by the owner. Mobile equipment costs were developed from vendor budgetary quotations. Unless provided by each vendor, the following assumptions were made for the additional expenses at initial equipment purchase:

- 5% of equipment budgetary price to cover initial parts stock,
- 4% of equipment budgetary price to cover freight and on-site assembly, and
- 2% of equipment budgetary price to cover equipment commissioning and training.

Equipment life-cycle operating hours were based on manufacturers' recommendations and on SRK's experience.

The schedule for major equipment rebuilds and replacement was based on equipment operating hours and anticipated equipment life provided by vendors. It was assumed that all equipment will have one major rebuild (60% of initial price) before replacement.

Underground Mine Infrastructure

Excavation/development costs for mine infrastructure development were estimated from first principles.

Equipment and/or related installation costs were provided by mining contractors, Mine Ventilation Services Inc. (MVS), and Conveyor Dynamics Inc. (CDI). CapEx associated with purchase and

installation of mine electrical reticulation and control-communication infrastructure was provided by Struthers Technical Solutions.

MVS provided the equipment costs for the mine ventilation and associated heating systems; SRK provided the associated excavation/development and installation costs.

CDI provided the equipment costs and installation costs for the underground and surface conveyors.

Metso provided a quote for the underground jaw crushers; a mining contractor provided a quote for crusher installation costs.

The costs for all other mine equipment were provided by vendor quotes; SRK provided the development and installation costs.

20.1.3 Plant and Infrastructure

Introduction

The following sub-sections summarize CapEx estimates provided for process plant modifications (KWM Consulting), Kemess G&A and site services (AuRico), tailings and waste rock storage (AMEC Foster Wheeler), access corridor (AMEC Foster Wheeler), discharge pumps & pipelines systems (AMEC Foster Wheeler), and water treatment (BioteQ).

In addition, CapEx estimates related to indirect costs, escalation, owner costs, contingencies, and process plant commissioning are discussed.

Process Plant modifications

KWM Consulting provided the CapEx estimate to enable the existing KS process plant to process 9 Mtpa for the KUG project. The estimate is presented in Table 20.4.

Table 20.4: CapEx Estimate – Process Plant

Item	Total (C$M)
Initial	
KS Primary Crusher (removal) and Reclaim	0.17
SAG Mill and Ball Mill Grinding	1.93
Rougher Flotation	0.35
Cleaner Flotation and Regrind	28.87
Thickening and Filtering	0.36
Tailings	1.34
Mill General	3.97

Item	Total (C$M)
Contingency	0.79
Sub-total	**37.77**
Sustaining Capital	
3rd Fine Grind Mill (Yr 7)	**13.19**
Sub-total	**13.19**
Total	**50.96**

Basis of Estimate:

- Budget equipment pricing for new equipment received from vendors.
- Plant modifications and repairs developed with AuRico Care & Maintenance personnel.
- Preliminary quantities for demolition developed from arrangement drawings.
- Sustaining capital has been allocated in Year 7 to expand the regrind capacity.

Kemess G&A and Site Services

As part of this Feasibility Study, both G&A and site services CapEx requirements have been estimated with the assumption that the KUG project advances without delay.

G&A CapEx includes all camp (including kitchen, septic system and recreation), mine dry and communications system related facilities. Camp CapEx is timed to ensure sufficient rooms are available as the camp population increases, with a peak 339 rooms available to handle the peak requirement of 321 on-site personnel in Year -1. Mine dry facilities are expanded in Year -4 to provide 300 locker-basket sets and 50 shower heads. The communication system is also replaced in Year -4 with land-based wireless repeater stations originating in Mackenzie.

Site services CapEx covers all mobile equipment not accounted for under mining and processing. This includes pick-up trucks, loaders, excavators, trucks, dozers, graders, bobcats, boom trucks, service vehicles, buses, water truck, sand truck, snow blower, and emergency vehicles. Purchase and replacement costs are included in this estimate.

Site services CapEx totals C$14.745M during construction and operations phases, with C$5.227M spent during the construction phase (Years -4 to -1) and an average C$730k per year spent during the operations phase (Years 1-13).

A summary of the G&A and site services components for the construction phase is presented in Table 20.5.

Capitalized pre-production OpEx has also been estimated for G&A and site services activities, with these reported elsewhere along with capitalized pre-production OpEx for mining and processing activities.

Table 20.5: CapEx Estimate – G&A and Site Services

Item	Yr -4 (C$M)	Yr -3 (C$M)	Yr -2 (C$M)	Yr -1 (C$M)	Total (C$M)
G&A					
Camp	1.57	1.86	1.93	0.00	5.36
Mine Dry	0.48	0.00	0.00	0.00	0.48
Communications	1.61	0.00	0.00	0.00	1.61
Sub-total	**3.66**	**1.86**	**1.93**	**0.00**	**7.45**
Site Services					
Sub-total	**1.19**	**2.25**	**1.70**	**0.13**	**5.26**
Total G&A and Site Services	**4.86**	**4.10**	**3.62**	**0.13**	**12.71**

Tailings and Waste Rock Storage Facilities

AMEC Foster Wheeler provided the CapEx estimate for the KUG tailings and waste rock storage facility. All of the mine development waste and tailings will be deposited in the former Kemess South open pit. This facility was used to store tailings from the processing of low grade stockpiles associated with the previous Kemess South operations.

A summary of the CapEx estimate (inclusive of 15% contingency) is presented in Table 20.6.

Table 20.6: CapEx Estimate – Tailings and Waste Rock Storage Facilities

Item	LOM (C$M)
TSF CapEx Estimate	
Starter Dam Construction (El. 1260)	13.81
Dam Raise Construction (El. 1265)	4.70
Dam Raise Construction (El. 1270)	3.93
Dam Raise Construction (El. 1275)	4.15
Diversion Channel Construction	0.66
Spillway Construction	0.70
Construction Monitoring (QA/QC)	1.57
Site Investigation, Detailed Design & Instrumentation	1.55
Reclamation	1.16
Total	**32.24**

Water Discharge Pumps & Pipelines Systems

AMEC Foster Wheeler provided the CapEx estimate to construct the KS open pit (KUG TSF) dewatering pipeline and additional pipelines to convey water for metals and selenium treatment with discharge to Attichika Creek. All associated pumps are included in these estimates.

The estimated costs (inclusive of 30% contingency) for these items are shown in Table 20.7.

Table 20.7: CapEx Estimate – Water Discharge Pumps & Pipelines Systems

Item	LOM (C$M)
Water Discharge & Pipelines Estimate	
Install Main Discharge Pipeline and Diffuser to Attichika Creek	3.55
Relocate and Refurbish Existing KS TSF Pumping Barge	1.14
Install Metals Treatment Plant Pipeline and Retrofit Barge	0.36
Install Selenium Treatment Plant Pipeline	0.20
Install Treated Water Discharge Pipeline	0.25
Contingency (30%)	1.64
Total	**7.13**

Water Treatment

The CapEx estimate for the water treatment facilities were based on the following:

- Plant mass balance based on the design criteria and capacities described in Section 8.4 and Selen-IX™ pilot plant operating data generated during a pilot operating campaign (BioteQ, 2015b);
- Budgetary quotations obtained for the highest capital cost equipment items including:
    - Ion exchange columns
    - Electrocells;
- Equipment cost estimates for all remaining plant equipment (e.g. pumps, tanks, agitators, clarifiers) generated by a cost model that uses an extensive database of process equipment quotations obtained by BioteQ over the last ten years; and
- Installed costs (including piping and instrumentation) were estimated by a factored approach based on previous BioteQ projects (installed cost = 3.0x equipment cost).

A summary of the capital costs for the water treatment facilities is presented in Table 20.8.

Table 20.8: CapEx Estimate – Water Treatment Plants

Item	LOM (C$M)
Selen IX	
Process Equipment	3.40
Installation	7.34
Subtotal	**10.74**
Metals Removal	
Process Equipment	0.83
Installation	1.66
Subtotal	**2.49**
Total	
Process Equipment	4.23
Installation	9.00
Total	**13.23**

Access Corridor

AMEC Foster Wheeler provided the CapEx estimate to establish the road and conveyor access to the triple declines portal location (a distance of ~4.5 km). The estimated cost (inclusive of 15% and 25% contingency for the surface works and tunneling works, respectively) for these items is shown in Table 20.9.

Table 20.9: CapEx Estimate – Access Corridor

Item	LOM (C$M)
Access Road Corridor	
Final Design	0.58
Access Road to South Portal	3.75
Access Tunnel	
South Portal	4.00
North Portal	3.44
Access Tunnel	10.86
KLV Access Roads and Laydowns	2.99
Triple Decline Portal and Laydowns	5.21
Conveyor Foundations	2.58
Vent Raise Access Road Extension	0.21
Surface Water Management	0.35
Reclamation of portal area and access corridor	0.70
Construction Monitoring QA/QC	1.73
Total	**36.39**

Indirect Cost Development

EPCM

To the greatest extent practicable, AuRico intends to manage contractors directly. As such, provision is included in site G&A and technical services cost estimates to provide personnel for this requirement. In addition, for specialist tasks such as tailings dam and conveyor construction, cost provision has been included for external EPCM services.

Freight

Where necessary, consultants providing CapEx estimates have included associated freight costs. In addition, an annual freight cost is provided in the G&A cost estimate for all phases of the KUG project including pre-production.

Spares

Where necessary, consultants providing CapEx estimates have included associated cost of spares.

Surface Mobile Equipment

AuRico has an existing fleet of surface mobile equipment currently being utilized for care and maintenance activities. This fleet will be used for site services activities during all phases of the KUG project.

Pre-production CapEx associated with this fleet for replacement units is C$5.7M.

Temporary Construction Facilities

The Kemess site is a former large-scale open pit mining and processing operation. As such, there are extensive facilities available to support temporary construction activities including a workshop, warehouse, laydowns, offices, and camp. Additional facilities are expected to be limited to facilities required at the access tunnel south portal and in the Kemess Lake Valley to support KUG mine development and construction activities.

Escalation

Estimates were prepared in real terms and escalation due to inflation is therefore implicit in cash flow modeling in real terms. Any additional escalation of costs is assumed to be accounted for in underlying estimates and is not added in subsequent modeling.

Site Services and G&A Costs

Owner's costs related to G&A and site services activities during the pre-production phase of the KUG project were estimated by AuRico, totaling C$106.7M. These were capitalised for the purposes of cash flow and tax modeling.

Corporate Owners' Costs

The economic model compiled by SRK also includes C$2.4M for KUG-related in-fill and geotechnical drilling, and C$4.7M for attributed AuRico corporate costs during the pre-production period.

Contingencies

All capital estimates were received inclusive of contingency. Pre-commercial production contingency equates to 12.5% of pre-commercial production capex or 8.2% of pre-commercial production capex plus opex.

Pre-commercial Operating Expenditure

A period of operations prior to the achievement of commercial production is assumed. Commercial production is defined as 60% of nameplate capacity for the mill. The operating costs incurred during this period, including the Site Services and G&A discussed in Section 20.2 total C$238M and are capitalised for the purposes of cash flow analysis and taxation calculations.

20.2 Operation Costs

This section presents items associated with project operating costs. All figures are in Canadian dollar terms unless otherwise stated. The estimate was prepared to a level of accuracy consistent with feasibility study standards.

20.2.1 Summary and Common Assumptions

Summaries of the KUG operating cost estimates during commercial production and total for life-of-mine (LOM) are shown in Table 20.10Table **Error! Reference source not found.**.

Table 20.10: LOM Operating Costs

Description	Commercial Production (C$M)	Commercial Production (C$/t ore)	Life of Mine (C$M)	Life of Mine (C$/t ore)
Underground	513	4.89	594	5.54
Mill and Processing	598	5.69	613	5.71
Main Ventilation	34	0.33	43	0.40
Conveyor	18	0.18	19	0.17
Water Treatment	27	0.26	46	0.43
Site Services OpEx	86	0.82	131	1.22
G&A OpEx	221	2.11	358	3.33
Total Operating Cost	**1,498**	**14.27**	**1,804**	**16.80**

The major assumptions used for the cost estimation were provided by AuRico as follows:

- **Diesel fuel price** at C$0.91/litre as per current prices AuRico pays to the suppliers.
- **Power cost** at C$0.058/kWhr as per current prices AuRico pays to the suppliers.
- **Labour:** AuRico provided all owner labour rates which were sourced from other Canadian mining operations.
- **Shift Rotation:** senior staff: four days on and three days off; hourly: two weeks on and two weeks off.

It is assumed for the purposes of this study that all travel, to and from site for all employees, is from Smithers or Prince George.

In SRK's opinion, the fuel price used in this study is lower than used in the projects of similar locations. However, the fuel price is what AuRico currently pays to a supplier so was used for cost estimation in this study.

20.2.2 Underground Mining Operating Costs

Underground mining operating costs were estimated for an owner operating scenario. Productivity, equipment operating hours, labour, supply requirements, and costs were calculated for each cost category, such as: production drilling and blasting, mucking and trucking, secondary breaking, underground crushing and conveying, mine services, and maintenance. The labour cost estimates were based on the labour rates provided by AuRico and the labour requirements estimated by SRK for each period of mine development and production.

The operating costs have been developed on a first principle approach and they have been collated under their respective activities.

Supply costs were based on estimated supply consumption and recent Canadian supplier's prices for drill and steel supplies, explosives, ground support, and services supplies. Maintenance consumables, such as parts, tires, etc., as well as fuel, lube and power were included in equipment operating costs and are part of mine development, production, and services costs.

Consumables and equipment operating costs were obtained from suppliers for most items. AuRico and SRK experience was used for the remaining items.

Mobile equipment maintenance costs were increased by 15% over suppliers' estimates to account for damage not anticipated by those suppliers. Consumables costs were increased by 10% to account for material wastage and miscellaneous use.

The total operating cost from the commencement of commercial production averages C$5.39/t ore including ventilation and conveying.

Mine Operating Input Data

Table 20.11, Table 20.12 and Table 20.13 show the input data used for estimation of productivities, equipment and labour requirements, and costs.

It was assumed that the semi-autonomous LHD operators will be located on surface and will have a different shift schedule from the underground workers.

Table 20.11: Shift Schedule Inputs – Production Rates

Operating Factors	Unit	Quantity
Days / Year	days/year	365
Crusher Weekly Maintenance	shift/week	1
Crusher Annual Maintenance	days/year	6
Crusher Total Maintenance	days/year	32
Crusher Operating Days	days/year	333
Mine Operating Days	days/year	360
Average Mining Rate	t/yr	9,000,000
Nominal Mining Rate	t/day	25,000

Table 20.12: Shift Schedule Inputs – Basic Shift Data

Operating Factors	Unit	Quantity
Working Days per Week	ea.	7
Shifts Per Day	ea.	2
Shift Length	hr	11
Shift Change / Travel From Dry	hr	0.5
Equipment Inspection / Fueling	hr	0.75
Lunch / Coffee Breaks	hr	0.25
Equipment Parking / Reporting / Travel To Dry	hr	0.25
Subtotal Non Productive Time	**hr**	**2.0**
Usable Time/Shift	hr	9.0
Shift Efficiency	%	82%
Usable Minutes/hr	min	50
Operational Efficiency (50 Min In Hr)	%	83%
Effective Work Time Per Shift	hr	7.5
Effective Work Time Per Day	hr	15.0
Work Time Efficiency	hr	68%
Usable Time Per Year	hr	6,480
Hours Worked /Year	hr	5,400
Hours Worked on Rotation Per Man /Year	hr	2,013
Basic Hours Man-year (52wx40hrs)	hr	2,080

Table 20.13: Shift Schedule Inputs – Shift Data for Semi-Autonomous Equipment

Operating Factors	Unit	Quantity
Working Days per Week	ea.	7
Shifts Per Day	ea.	2
Shift Length	hr	12
Shift Change / Travel From Dry	hr	0.25
Lunch / Coffee Breaks	hr	0.75
Subtotal Non Productive	hr	1.0
Usable Time/Shift	hr	11.0
Shift Efficiency	%	92%
Usable Minutes/hr	min	52.5
Operational Efficiency (52.5 min In hr)	%	88%
Effective Work Time Per Shift	hr	9.6
Effective Work Time Per Day	hr	19.25
Work Time Efficiency	hr	80%

20.2.3 Production Costs

Panel caving production costs were developed from first principles and SRK's experience, where required.

Production drill and blast costs include cost to drill and blast the drawbells and undercut level. It was assumed that all material from extraction and undercut level development, drawbell, undercut and 10 m of initial caving will be mucked by manually operated 8.6 m^3 LHDs and dumped to the ore passes located south of the cave footprint. When the primary jaw crushers are available for direct tipping by production LHDs, the semi-autonomous LHDs will be used to muck the ore from the drawpoints and dump it directly into jaw crushers. Every drawpoint has been given a tonnage profile from PCBC™. The mucking costs reflect the tramming distances to the relevant ore pass or crusher for every tonne of ore based on the mine schedule.

The secondary breaking costs were estimated based on the amount of high and low hang-ups at drawpoints and amount of oversize material to be broken to get through the 1.0 m x 1.0 m grizzly openings. Secondary breakage activities require one water cannon and maximum of three block holers to deal with high and low hang-ups, and maximum of four mobile rockbreakers to deal with drawpoint oversize throughout the period of production. It was assumed that 50% of drawpoint oversize material will be broken by mobile rockbreakers in the drawpoints with the remainder broken by stationary rockbreakers installed on the top of each crusher.

The underground crushing cost includes operating costs of the primary jaw crushers. It was estimated that three jaw crushers would be operating at the same time during full commercial production.

The mine services and maintenance costs include all services mobile equipment fleet, mine auxiliary ventilation, mine dewatering, fixed equipment maintenance, and assumption for miscellaneous expenses and drawpoints rehabilitation. The labour requirements were estimated for each mining activity.

These costs for the full life of mine including the pre-commercial period are shown in Table 20.14.

Table 20.14: OpEx Estimate – LOM Mining Costs Incl. Non-commercial Period

Item	Life of Mine (C$M)	Unit Cost (C$/t ore)
Mining Costs		
Production Drill & Blast	34.56	0.32
Production Mucking	106.19	0.99
Trucking	14.08	0.13
Secondary Breaking	13.35	0.12
Underground Crushing	14.23	0.13
Mine Services and Maintenance	96.96	0.90
Sub-total	**279.38**	**2.60**
Labour		
Operating Hourly Labour	123.41	1.15
Maintenance Hourly Labour	82.85	0.77
Operating Staff	41.53	0.39
Maintenance Staff	25.91	0.24
Technical Staff	41.33	0.38
Sub-total labour	**315.04**	**2.93**
TOTAL (LOM includes non-commercial period)	**594.42**	**5.54**

20.2.4 Mine Labour Cost Basis

Mining employees at the KUG operation are divided into two categories: salaried and hourly personnel. Salaried personnel include engineering, technical, and supervisory staff with the majority of these on four days on and three days off rotation. The hourly personnel will work a two weeks on and two weeks off rotation schedule. Hourly personnel includes labour for mine development, production, services, and maintenance.

The labour requirements were estimated for each mining activity based on the estimated operating productivities, cycle times, number of equipment operators per unit, maintenance and services requirements.

The labour rates were provided by AuRico and include base wage/salary, production bonus, which varies from 5% to 30% of the base depend on employee category, and 30% burden of the base for all employee categories.

20.2.5 Process Plant

The battery limits for the process plant operating costs are from the discharge of the overland conveyor into the existing KS primary crusher dump pocket (with crusher removed) to the discharge of tailings and reclaim of process water to and from the KUG TSF and the discharge of the concentrate conveyor in the concentrate load-out shed. The intention is to locate the water treatment plant, for water discharge to the environment, in the existing process plant building but operations of this plant are excluded from the process plant operating cost estimate.

The KS process plant has an operating history and the development of the operating cost for KUG ore processing has been based on some of the data from 2009 and 2010 as detailed in the Basis of Estimate. Vendor pricing from 2015 was obtained for consumables.

Table 20.15 summarizes the process plant operating costs. When in full production, the process plant operating costs are projected to be C$5.69/t.

Table 20.15: OpEx Estimate – Process Plant

Item	Unit	Year -1	Year 1	Year 2	Year 3	Year 4 to 10
Processing Plant						
Total ore feed to mill	ktpa	1,227	3,366	5,951	8,751	9,000
Total Cost						
Labour	C$M	1.48	3.56	6.61	8.44	8.61
Power	C$M	2.56	6.50	11.82	16.04	16.41
Consumables	C$M	2.62	7.15	12.66	18.51	19.03
Maintenance	C$M	0.96	2.63	4.65	6.84	7.03
Total	C$M	**7.63**	**19.84**	**35.75**	**49.83**	**51.08**

Item	Unit	Year -1	Year 1	Year 2	Year 3	Year 4 to 10
Unit Cost						
Labour	C$/t milled	1.21	1.06	1.11	0.96	0.96
Power	C$/t milled	2.09	1.93	1.99	1.83	1.82
Consumables	C$/t milled	2.14	2.12	2.13	2.12	2.11
Maintenance	C$/t milled	0.78	0.78	0.78	0.78	0.78
Total	C$/t milled	**6.21**	**5.89**	**6.01**	**5.69**	**5.68**

The operating costs for Year 1 to Year 3 have assumed a base of fixed costs and the remainder as variable cost based on production rates (power, consumables, and operating supplies). Note that the actual costs applied in the Technical Economic Model during these years reflect the schedule under evaluation and may vary slightly from these costs but are consistent with the assumption of fixed and variable costs presented here.

Basis of Estimate

Labour

The labour costs include all staff and hourly personnel, operations and maintenance, assigned to the process plant. The labour list was developed together with AuRico management. Rates and benefits for each position were provided by AuRico.

For Year -1 to Year 2, labour has been adjusted to move from operating the process plant for two-week ore campaigns (two-weeks operation, two-weeks maintenance) to continuous production. During the campaign operation, 24-hour loader operators have been included in the process plant operations crew to manage the stockpile loading when the overland conveyor is operating and reclaiming ore to feed the SAG mill when the process plant is operating. The loader operators are phased out for Year 4.

The process plant staffing is based on the following:

- Year -1, 43 people (start of mill staffing 01 July, 21.5 people on annual basis)
- Year 1, 52 people
- Year 2, 77 people
- Year 3, 76 people

The staffing has been developed to provide the opportunity to develop and train personnel as the capacity is being ramped up.

Electrical Power

A simplified equipment list was developed from the existing plant flowsheets that included all of the primary power consumers in the process plant. The list was modified to include the new processes

(fine grinding mills, cleaner tails pumps, Knelson concentrator) that have been added into the original flowsheets and to remove the redundant equipment (SAG mill, ball mill) that have been removed from the plant. A 3% factor was included in the total power load to include miscellaneous consumers, building lighting, heating, etc. Load factors were assigned to the equipment to generate the power consumption for the various unit operations (Table 20.13).

The cost of C$0.058/kWh was provided by AuRico based on recent billings.

Table 20.16: OpEx Estimate – Process Plant Electrical Power (9.0 Mt/year)

Description	Power Utilized (MWh/yr)	Energy Cost (C$/yr)	Energy Cost (C$/t milled)
Ore Handling	3,852	223,416	0.02
Grinding and Reclaim	139,491	8,090,478	0.90
Copper Flotation and Regrind	112,456	6,522,448	0.72
Concentrate Thickening and Filtration	1,463	84,854	0.01
Reagents	303	17,574	0.00
Tailings	9,998	579,884	0.06
Process Services	6,879	398,982	0.04
General Services (lighting, heating, etc., 3%)	8,233	477,514	0.05
Total	**282,675**	**16,395,150**	**1.82**

For Years -1 to Year 2, estimates for the electrical power costs were based on 12% fixed power costs and 88% variable costs with the plant throughput.

Consumables

The costs for consumables were developed using a combination of 2009 data, testwork results and 2015 vendor data.

Grinding media costs for the SAG mill, ball mill and regrind mills were based on a combination of consumption kg/kWh and 2015 pricing from a local media vendor. Ceramic media consumption rates and costs were provided by the equipment vendor.

Reagent consumption rates were based on historic unit consumption rates and recent metallurgical testwork. Vendor pricing for 2015 was used to determine the costs.

Mill liner costs were estimated based on costs provided by the liner supplier and the schedule for mill relines.

Annual operating supplies were developed based on the 2009 costs for the various unit operations. Operating costs were factored from 2009 to 2015 using the Consumer Price Index (CPI).

Table 20.17 summarizes the consumables costs.

Table 20.17: OpEx Estimate – Process Plant Consumables (9.0 Mt/year)

Description	Annual Cost (C$M)	Unit Cost (C$/t milled)
Grinding Media	14.11	1.57
Reagents	1.59	0.18
Annual Operating Supplies	0.50	0.06
Liners	3.26	0.36
Total	**19.47**	**2.16**

For Years -1 to Year 2, consumable costs have been factored based on plant throughput.

Maintenance

The process plant maintenance costs have been developed based on the 2009 actual maintenance costs distribution. These costs apply to material costs (including contracts) as maintenance labour is included in the process plant labour and materials considered consumables (mill liners) are included with the consumables. Table 20.18 summarizes the plant maintenance costs.

Table 20.18: OpEx Estimate – Process Plant Maintenance (all throughputs)

Plant area	2009 Costs (C$M)	Production Adjustment	2015 Projected Annual Cost (C$M)	Unit Cost (C$/t milled)
Plant General	1.38	1.00	1.53	0.17
Assay and Sample Prep Lab	0.02	1.00	0.02	0.00
Metallurgical	0.00	1.00	0.00	0.00
Primary Crusher (apron feeder and stockpile feed conv)	0.31	0.60	0.21	0.02
Stockpile Reclaim	0.54	0.60	0.36	0.04
Grinding	3.22	0.60	2.14	0.24
Grinding (External Services)	0.72	0.60	0.48	0.05
Flotation	0.56	1.00	0.63	0.07
Dewatering	0.17	1.00	0.19	0.02
Tailings	2.35	0.60	1.57	0.17
Reclaim Water	0.21	0.60	0.14	0.02
Total	**9.47**	**0.77**	**7.25**	**0.81**

Where appropriate, maintenance costs have been adjusted for operations at reduced throughput. The 2009 costs were then adjusted for the changes in CPI. For 2009 to 2015, the CPI adjustment was 1.11.

For Year -1 to Year 2, maintenance costs were factored based on plant throughput, so unit costs are assumed to remain constant.

Unit Operations

The Kemess operation has historically recorded the operating costs based on the unit operations. Table 20.19 provides a preliminary distribution of the annual process OpEx by unit operation or activity.

Table 20.19: OpEx Estimate – Process Plant Distributed Items

Item	Average Annual Cost (C$M)	Unit Cost (C$/t milled)
Process plant – Distributed items		
General and Administration	4.87	0.54
Metallurgy and Assaying	1.62	0.18
Reclaim	1.38	0.15
Grinding	29.66	3.30
Flotation	9.64	1.07
Dewatering	0.69	0.08
Tailings	3.04	0.34
Reclaim Water	0.19	0.02
Total	**51.08**	**5.68**

20.2.6 Water Treatment

Operating costs of the water treatment facilities were estimated based on the following:

- Consumables consumption rates were calculated from the plant mass balance, informed by pilot operation on two feed waters:
    - 95th percentile KUG water composition estimate, to simulate the water during process plant operation, and
    - Existing pit water, spiked with selenium to simulate the water to be encountered during the initial years of operation prior to process plant startup.
- Power consumption was estimated from equipment requirements
    - Electricity rate of C$0.06/kWh.
- Bulk costs for key consumables were obtained from vendors, in particular the following:
    - Iron anodes and

    - Sodium sulphate.

- Labour rates typical for a mine in northern BC
    - Rates include vacation and statutory holiday pay, and other payroll burden.
- Reagent shipping costs from the lower mainland to site of C$225/tonne.
- BioteQ will operate both the metals removal and Selen-IX™ plants with the same operating crew.
- Water treatment residues are assumed to be transported from the treatment plant to the KUG TSF. This has not been accounted for in the cost model as this figure is likely insignificant.

Total LOM water treatment costs are C$46.34M including post-closure costs. This equates to C$0.43 per tonne of ore. During commercial production operating costs are C$0.26 per tonne.

20.2.7 General and Administration

As part of this Feasibility Study, both G&A and site services OpEx requirements have been estimated with the assumption that the KUG project advances without delay. Construction (pre-production), operating and closure phases OpEx have been estimated. It should be noted that costs from site operating experience, including ongoing care and maintenance, were applied (and adjusted) where considered representative of future KUG project-related site conditions.

G&A OpEx comprises costs related to G&A-related labour (General Manager, accounting, concentrate sales, procurement, warehouse, IT, HR, H&S, security and environment), general site electricity consumption (so excluding mining and processing), flights, camp, insurances, business occupancy taxes, communication, freight/transport, environmental works and engineering.

A summary of the G&A OpEx estimate is presented in Table 20.20.

Table 20.20: OpEx Estimate – G&A Summary

Item	Period	Total Cost (C$M)	Average Annual Cost (C$M)	Unit Cost (C$/t milled)
G&A				
Construction phase	Yr -5 to Yr -1	80.77	16.15	n/a
Operating phase	Yr 1 to Yr 13	236.47	18.19	2.23
Closure phase	Yr 14 to Yr 19	40.50	6.75	n/a
Total	**Yr -5 to Yr 19**	357.77	n/a	n/a

Site services OpEx comprises costs related to labour required for camp maintenance, back-up generator maintenance, road maintenance, snow removal, small construction works, on-site supplies transport, site water management, garbage disposal, airstrip maintenance and operations, and associated fixed and mobile equipment maintenance. Costs related to mobile equipment

maintenance, contractors (for specialist maintenance needs), Omineca Resource Access Road maintenance, chemicals and gases, and powerline brushing are included.

A summary of the site services OpEx estimate is presented in Table 20.21.

Table 20.21: OpEx Estimate – Site Services Summary

Item	Period	Total Cost (C$M)	Average Annual Cost (C$M)	Unit Cost (C$/t milled)
Site services				
Construction phase	Yr -5 to Yr -1	25.95	5.19	n/a
Operating phase	Yr 1 to Yr 13	91.05	7.00	0.86
Closure phase	Yr 14 to Yr 19	14.37	2.39	n/a
Total	**Yr -5 to Yr 19**	**131.37**	**n/a**	**n/a**

Table 20.22 and Table 20.23 provide a breakdown of total and unit cost (C$/t) site services G&A OpEx for the operating phase of the KUG project.

Table 20.22: OpEx Estimate Breakdown – Site Services

Item	Life of Mine (C$M)	Unit Cost (C$/t)
Labour	$62.38	$0.58
Mobile Equipment	$23.06	$0.21
Contractors	$9.75	$0.09
ORAR maintenance	$7.47	$0.07
Chemicals & gases	$3.95	$0.04
Powerline brushing	$8.00	$0.07
Miscellaneous	$16.75	$0.16
Total Site Services OpEx	**$131.37**	**$1.22**

Table 20.23: OpEx Estimate Breakdown – G&A

Item	Life of Mine (C$M)	Unit Cost (C$/t)
Labour	$62.02	$0.58
Flights	$48.75	$0.45
Camp	$89.20	$0.83
Electricity	$39.30	$0.37
Insurances	$6.09	$0.06
Business occupancy tax	$8.70	$0.08
Communications	$6.60	$0.06
Freight & transport	$43.06	$0.40
Environmental works (reclamation)	$9.29	$0.09
Environmental monitoring – KS	$6.37	$0.06
Environmental monitoring – KUG	$13.58	$0.13
Engineering – studies	$2.31	$0.02
Engineering – detailed	$5.81	$0.05
Miscellaneous	$16.70	$0.16
Total G&A OpEx	**$357.77**	**$3.33**

21 Economic Analysis

This section presents items associated with the economic evaluation of the project. All figures are in Canadian dollar terms unless otherwise stated.

21.1 Summary

The economic analysis was undertaken using an Excel-based, discounted cash flow (DCF) model that modelled cash flows by quarter (three-month periods). The nominal discount rate of 5% was used with an exchange rate (US$ per C$) of 0.75.

Long term commodity pricing used was US$2.50/lb for copper, US$1,200/oz for gold, and US$16.00/oz for silver. Table 21.1 summarises the base case assumptions.

The cash flow model generally uses 2016 real Canadian dollars.

Note that on instruction from AuRico, an assumption of 0% inflation was used throughout the model.

The payback periods are shown in

Table 21.2. The discounted payback period is 5.6 years from the start of commercial production and 4.1 years for a non-discounted basis from commercial production.

Net Present Value (NPV) of the project cash flow after tax is C$258M at a discount rate of 5%. The undiscounted cash flow is C$695M after tax. The IRR is 11.9%.

Sensitivities to the base case were run for changes to commodity prices (revenues), costs and foreign exchange rate. The results can be found in Section 21.8.2.

Table 21.1: Base Case Assumptions

Base Case Pricing	Unit	Long-Term
Copper	US$/lb	$2.50
Gold	US$/oz	$1,200
Silver	US$/oz	$16.00
Exchange rate	US$ per C$	0.75
Modeled Recoveries to Concentrate		
Copper Recovery to Concentrate	%	91
Gold Recovery to Concentrate	%	72
Silver Recovery to Concentrate	%	65

Table 21.2: Payback Periods

Payback Periods	From 1st production (years)	From commercial production (years)
Discounted Payback Period	6.3	5.6
Non-discounted payback period (real)	4.8	4.1
Nominal payback period	4.8	4.1

21.2 Revenues

21.2.1 Production Schedule

Revenues were derived from production schedules developed by AuRico which is based on the cave production schedule provided by SRK. The schedule provided tonnes of ore and grades of economic minerals to enable revenues to be derived.

21.2.2 Commodity Prices

Commodity prices used for the base case economic model are shown in Table 21.1.

21.2.3 Mill Recoveries

Mill recoveries were estimated by KWM Consulting Inc. and are shown in Table 21.1.

21.2.4 Treatment and Refining Costs, Payable Metal Assumptions, and Freight

These costs summarized in Table 21.3 and Table 21.4, were supplied in a marketing study by Exen Consulting. Various alternate destinations were studied and compared for the most favourable combination of freight costs and economic terms. For the base case, exporting though Prince Rupert, B.C., to a destination of Japanese, Korean, and Northern Chinese smelters was assumed. Life of mine treatment, refining and payable costs presented in this section are in Canadian dollars (US$ per C$ of 0.75),

Table 21.3: Treatment and Refining Costs

TCRC and Freight	Unit Costs (US$)	LOM Costs (C$M)
Treatment Charge	$80 per DMT	126
Copper Refining Costs	$0.08 per payable lb	58
Gold Refining Costs	$6.00 per payable ounce	10
Silver Refining Costs	$0.40 per payable ounce	2
Total Treatment and Refining Costs		
Freight Costs (base case)	$132.14 per DMT	208
Total TCRC and Freight		**405**

Table 21.4: Payable Metals Assumptions and Net Costs

Payable Metals Deductions and Costs	Effective Payable Rate	LOM Costs (C$M)
Copper	95.5%	87
Gold	97.0%	65
Silver	90.0%	9
Total Effective Payable Costs		**161**

21.3 Operating Costs

Operating costs were imported into the model in the form of worksheets from the underlying cost models. Details of the underlying operating costs can be found in Section 20.

The modeled operating costs are shown in Table 21.5 and Table 21.6.

Table 21.5: Unit Operating Costs

Description	Life of Mine (C$/t ore)	Commercial Production (C$/t ore)
Underground	$5.54	$4.89
Mill and Processing	$5.71	$5.69
Main Ventilation	$0.40	$0.33
Conveyor	$0.17	$0.18
Water Treatment	$0.43	$0.26
Site Services OpEx	$1.22	$0.82
G&A OpEx	$3.33	$2.11
Total Unit Cost	**$16.80**	**$14.27**

Table 21.6: Total Operating Costs

Description	Life of Mine (C$M)	Commercial Production (C$M)
Underground	594	513
Mill	613	598
Main Ventilation	43	34
Conveyor	19	18
Water Treatment	46	27
Site Services OpEx	131	86
G&A OpEx	358	221
Total Operating Costs	**1,804**	**1,498**

Operating costs and sustaining capital costs are shown in co-product terms of cost per unit of metal equivalent in the following tables.

Table 21.7: Cash Operating Costs per lb Cu Equivalent

Cost Item	Life of Mine (C$/lb)	Commercial Production (C$/lb)	Life of Mine (US$/lb)	Commercial Production (US$/lb)
Mining	0.50	0.44	0.37	0.33
Primary Ventilation	0.04	0.03	0.03	0.02
Conveyor	0.02	0.02	0.01	0.01
Subtotal UG Mining	**0.55**	**0.48**	**0.41**	**0.36**
Offsite Costs (TCRC & Freight)	**0.34**	**0.34**	**0.25**	**0.25**
Processing plant	0.51	0.51	0.38	0.38
Water Treatment	0.04	0.02	0.03	0.02
Site Services	0.11	0.07	0.08	0.06
G & A	0.30	0.19	0.22	0.14
Total Cash Costs	**1.84**	**1.62**	**1.38**	**1.21**

Table 21.8: Cash Operating Costs per oz Au Equivalent

Cost Item	Life of Mine (C$/oz)	Commercial Production (C$/oz)	Life of Mine (US$/oz)	Commercial Production (US$/oz)
Mining	238.0	210.3	178.5	157.8
Primary Ventilation	17.2	14.1	12.9	10.5
Conveyor	7.5	7.5	5.6	5.7
Subtotal UG Mining	**262.7**	**231.9**	**197.0**	**174.0**
Offsite Costs (TCRC & Freight)	**162.0**	**161.8**	**121.5**	**121.3**
Processing plant	245.3	244.9	184.0	183.7
Water Treatment	18.6	11.1	13.9	8.3
Site Services	52.6	35.3	39.5	26.5
G & A	143.2	90.7	107.4	68.0
Total Cash Costs	**884.3**	**775.7**	**663.3**	**581.7**

Table 21.9: All-in Sustaining Costs per lb CuEq

Cost Item	Life of Mine (C$/lb)	Commercial Production (C$/lb)	Life of Mine (US$/lb)	Commercial Production (US$/lb)
Mining	0.50	0.44	0.37	0.33
Primary Ventilation	0.04	0.03	0.03	0.02
Conveyor	0.02	0.02	0.01	0.01
Subtotal UG Mining	**0.55**	**0.48**	**0.41**	**0.36**
Offsite Costs (TCRC & Freight)	**0.34**	**0.34**	**0.25**	**0.25**
Processing plant	0.51	0.51	0.38	0.38
Water Treatment	0.04	0.02	0.03	0.02
Site Services	0.11	0.07	0.08	0.06
G & A	0.30	0.19	0.22	0.14
Sustaining Capital	0.22	0.22	0.16	0.17
Total	**2.06**	**1.84**	**1.55**	**1.38**

Table 21.10: All-in Sustaining Costs per oz AuEq

Cost Item	Life of Mine (C$/oz)	Commercial Production (C$/oz)	Life of Mine (US$/oz)	Commercial Production (US$/oz)
Mining	238.0	210.3	178.5	157.8
Primary Ventilation	17.2	14.1	12.9	10.5
Conveyor	7.5	7.5	5.6	5.7
Subtotal UG Mining	**262.7**	**231.9**	**197.0**	**174.0**
Offsite Costs (TCRC & Freight)	**162.0**	**161.8**	**121.5**	**121.3**
Processing plant	245.3	244.9	184.0	183.7
Water Treatment	18.6	11.1	13.9	8.3
Site Services	52.6	35.3	39.5	26.5
G & A	143.2	90.7	107.4	68.0
Sustaining Capital	105.2	107.7	78.9	80.8
Total	**989.5**	**883.3**	**742.1**	**662.5**

21.4 Capital Costs

Capital costs were imported into the model in the form of worksheets from the underlying cost models. Details of the underlying capital costs can be found in Section 20.1. The modelled capital costs are shown in Table 21.11. First process plant production is achieved in Q4 Year -1.

Commercial production is defined as 60% of nominal plant capacity reached for the first time. This is reached in Q4 Year 1.

For the purposes of economic modelling, operating costs were capitalised for the period prior to commercial production. Post-closure costs were also capitalised and are included in the life-of-mine figures shown in Table 21.11.

Table 21.11: Total Capital Costs

Item	To First Production (C$M)	Pre-Commercial[1] (C$M)	Sustaining (C$M)	Life of Mine (C$M)
Project Owners Costs	7	7	0	7
Mine	205	266	207	473
Mill	31	38	13	51
Site Service	13	13	9	22
Conveyor	40	40	0	40
UG Electrical	18	19	0	19
UG Ventilation	10	10	1	11
Access Corridor	36	36	0	36
Water Treatment Build	13	13	0	13
TSF and Pipeline Build	7	8	32	39
Reclamation	0	0	1	1
Total Capital	**380**	**449**	**263**	**712**
Capitalised Pre-production Opex (COC)	144	238	0	238
Total Capital (Incl. COC)	**524**	**687**	**263**	**950**
Pre-commercial Net Revenue	N/A	(83)	N/A	(83)
Net Capex (Incl pre-commercial Opex and Net Revenue)	**524**	**604**	**263**	**867**

1 Pre-Commercial values inclusive of those “To First Production”

21.5 Tax and Tax Depreciation

Taxes and Tax depreciation were modelled by treating the project as a standalone business. Existing balances in expenditure categories were advised by the client.

The major categories of taxes modelled are:

1. British Columbia Minerals Tax was modelled using a net current proceeds rate of 2.0% and a net revenue tax rate of 13.0%. An imputed interest rate of 1.6% was assumed for the purposes of the tax credit account. This is based on historic rates.
2. British Columbia Corporate Tax was modelled at a rate of 11%.
3. Canadian Federal Corporate Tax was modelled at a rate of 15%.
    a. The BC Minerals Tax is deductible for both Federal and Provincial Corporate tax. The corporate tax rates are simply additive (neither deductible for the other).
    b. Tax depreciation for capital expenditure was estimated in accordance with the general principles used in British Columbia for mining taxation, including the use of accelerated depreciation for certain categories. The British Columbia Mining tax was estimated using the guidelines published by the Government of British Columbia.

Tax credits from post-closure expenditure were flowed through to the cash flow of the project. This is achievable under current legislation through the use of a Qualifying Environmental Trust (QET), as discussed in Section 21.7.

The estimate of the LOM tax payable under these categories is shown in Table 21.12.

Table 21.12: Federal and Provincial Taxes

Federal and Provincial Taxes	LOM (C$M)
BC Provincial Corporate Tax	99
Federal Corporate Tax	135
BC Minerals Tax	100
Total Taxes	**334**

21.6 First Nations Project Participation

AuRico advised that an Interim Measures Agreement (IMA) with the relevant First Nations groups is in place and provided details of methodology for estimating the payments that would be due from the cash flows of the project. The agreement provides for fixed, milestone-driven, and variable payments, dependent on net cash flow.

21.7 Post-Closure Costs

Post-closure costs are assumed funded from a QET, established for this purpose. The model assumes cash contributions to the QET for the three years prior to closure sufficient to cover all post-closure costs. These contributions are deducted against income for tax purposes. A 3% rate of return on the balance of the trust is modelled as additional revenue and is subject to additional tax. This overall treatment was developed in consultation with the client. SRK considers it to be a reasonable proxy for detailed tax planning suitable for a Feasibility Study.

21.8 Project Valuation

On the basis of the revenue and costs outlined above, the project is estimated to have a post-tax net present value (NPV) (at 5% Discount Rate) of C$258M and a post-tax internal rate of return of 11.9%.

21.8.1 Payback Periods

The cumulative project cash flow becomes positive in real terms 4.1 years after commencement of commercial production. On a discounted basis at 5%, the project has returned the investment 5.6 years after commencement of commercial production (Table 21.13).

Table 21.13: Payback Periods

Payback Periods	From 1st production (years)	From commercial production (years)
Discounted Payback Period	6.3	5.6
Non-discounted payback period (real)	4.8	4.1

21.8.2 Sensitivity Analysis

Sensitivity to Price

Table 21.14 and Table 21.15 show sensitivity of the project's post-tax NPV and IRR to both gold and copper price assumptions. Note that although the NPVs are expressed in Canadian dollars, the unit prices shown in this table are in US dollars. The value at alternate prices is derived from altering only commodity prices in the model. Taxes, First Nations payments, and working capital are automatically updated, but no correlation between commodity prices and input costs is modelled.

Table 21.14: NPV Sensitivity to Gold and Copper Prices

		Gold Price ($US/oz)						
		$950	**$1,050**	**$1,150**	**$1,200**	**$1,250**	**$1,350**	**$1,450**
Copper Price ($US/lb)	**$3.50**	$361	$422	$484	$516	$546	$608	$669
	$3.25	$296	$358	$420	$451	$482	$544	$605
	$3.00	$231	$294	$356	$387	$417	$479	$541
	$2.75	$166	$229	$291	$322	$353	$415	$477
	$2.50	$99	$163	$226	**$258**	$289	$351	$412
	$2.25	$29	$97	$161	$192	$224	$286	$348
	$2.00	($47)	$26	$94	$126	$158	$221	$284
	$1.75	($125)	($50)	$23	$59	$92	$156	$219

Table 21.15: IRR Sensitivity to Gold and Copper Prices

		Gold Price ($US/oz)						
		$950	**$1,050**	**$1,150**	**$1,200**	**$1,250**	**$1,350**	**$1,450**
Copper Price ($US/lb)	**$3.50**	14.1%	15.4%	16.6%	17.2%	17.7%	18.8%	19.9%
	$3.25	12.7%	14.1%	15.3%	15.9%	16.5%	17.7%	18.8%
	$3.00	11.3%	12.7%	14.0%	14.7%	15.3%	16.5%	17.7%
	$2.75	9.8%	11.3%	12.7%	13.3%	14.0%	15.2%	16.5%
	$2.50	8.1%	9.7%	11.2%	**11.9%**	12.6%	13.9%	15.2%
	$2.25	6.3%	8.0%	9.7%	10.4%	11.2%	12.6%	13.9%
	$2.00	4.1%	6.2%	8.0%	8.8%	9.6%	11.1%	12.5%
	$1.75	1.8%	4.0%	6.1%	7.1%	7.9%	9.6%	11.1%

Sensitivity to Costs

Table 21.16 and Table 21.17 show the sensitivity of the project's post-tax NPV and IRR to variation in overall cost assumptions.

Table 21.16: NPV Sensitivity to Capital and Operating Costs

		Capital Costs						
		-30%	**-20%**	**-10%**	**0%**	**10%**	**20%**	**30%**
Operating Costs	**30%**	$123	$97	$73	$50	$26	($1)	($36)
	20%	$192	$166	$142	$120	$98	$76	$50
	10%	$260	$235	$211	$189	$167	$147	$126
	0%	$329	$304	$279	**$258**	$236	$217	$197
	-10%	$398	$372	$348	$325	$305	$286	$267
	-20%	$466	$440	$416	$393	$373	$353	$335
	-30%	$536	$509	$485	$462	$442	$423	$405

Table 21.17: IRR Sensitivity to Capital and Operating Costs

		Capital Costs						
		-30%	**-20%**	**-10%**	**0%**	**10%**	**20%**	**30%**
Operating Costs	**30%**	9.4%	8.4%	7.5%	6.8%	6.1%	5.4%	4.7%
	20%	11.4%	10.3%	9.4%	8.5%	7.8%	7.1%	6.5%
	10%	13.4%	12.2%	11.1%	10.2%	9.4%	8.7%	8.0%
	0%	15.4%	14.1%	12.9%	**11.9%**	11.0%	10.3%	9.5%
	-10%	17.3%	15.9%	14.7%	13.6%	12.6%	11.8%	11.0%
	-20%	19.3%	17.7%	16.4%	15.2%	14.2%	13.3%	12.4%
	-30%	21.3%	19.5%	18.1%	16.8%	15.7%	14.7%	13.9%

Sensitivity to Foreign Exchange Rate

To explore the sensitivity of project economics to foreign exchange (FX) assumptions, a range of FX rates was considered around the base case of US$0.75 = C$1.00.

Note that in the cash flow model the varied FX rates are only applied to revenue i.e. C$ denominated Cu and Au commodity prices were flexed by the FX variation. Offsite costs including freight were also flexed in C$ terms. For simplification in this sensitivity analysis it is assumed that all project capital and operating costs stay constant in terms of Canadian dollars, regardless of FX variation.

The effect of not considering the response of operating and capital costs to FX variation is that the sensitivity and risk is likely overstated. Some reduction in C$ costs could reasonably be assumed where the C$ strengthens against the US$, mitigating the reduction in value shown. Conversely, some increase in C$ costs could reasonably be assumed should the C$ weaken, reducing the upside. This is complex to model and requires detailed consideration of cost exposure to FX in the short term, as well as a view how costs respond over time under purchasing power parity (PPP) principles.

Table 21.18 shows the sensitivity of the project's post-tax NPV and IRR to variation in exchange rate. It can be seen that the project approaches breakeven at an FX rate of between 0.90 and 0.95 as the strengthening C$ results in lower nominal C$ revenues (prices are assumed constant in US$ terms). Lower costs would tend to mitigate this reduction in value, as discussed above.

Table 21.18: NPV & IRR Sensitivity to Foreign Exchange Rate

US$ per C$	0.65	0.7	0.75	0.8	0.85	0.9	0.95	1
Project NPV (C$M)	$454	$349	**$258**	$177	$105	$37	($28)	($88)
Project IRR	16.00%	13.90%	**11.90%**	10.00%	8.20%	6.50%	4.70%	2.90%

22 Adjacent Properties

There are no properties currently under development or exploitation adjacent to the Kemess site.

23 Other Relevant Data and Information

This section was used in the July 2015 technical report to report the Kemess East (KE) resource estimate. Reporting related to the KE deposit is now included in previous sections of this report as appropriate.

The reader is reminded that reporting for the KE deposit is limited to the updated resource estimate whereas reporting for the KUG project includes all aspects of the associated feasibility study.

24 Interpretation and Conclusions

This section summarises the main conclusions, risks, opportunities as well as updates to the KUG project since the previous Feasibility Study, completed in 2013.

On the basis of the revenue and costs outlined in Sections 20 and 21, the project is estimated to have a post-tax NPV of C$258M at a 5% discount rate and a post-tax internal rate of return of 11.9%. The cumulative project cash flow becomes positive in real terms 4.1 years after commencement of commercial production. On a discounted basis at 5%, the project has returned the investment 5.6 years after commencement of commercial production.

24.1 Project Risks and Opportunities

24.1.1 Summary

SRK conducted a risk and opportunities assessment workshop for the Kemess Underground project. Risks and opportunities were quantified for both likelihood (probability of occurrence) and consequence (impact of occurrence). Risks and opportunities were provided by SRK, AuRico, and the other external consultants listed in this report.

The consequences were assessed over six categories; safety impact, revenue impact, revenue deferral/advance, capital cost, operating cost, and schedule delay/advance. The opinion of the likelihood of occurrence was based on professional opinion based on the current understanding of the project. The responsible party for each risk and opportunity ranked is denoted in the risk and opportunity register. Risks and opportunities are ranked under the assumption that mitigation methods mentioned in this study are put in place, and that industry standard practices are generally implemented.

The risks and opportunities outlined in this section were considered at the asset level only. Effects at the corporate level, which may impact cost of capital or corporate reputation, were not considered as part of this assessment and therefore are not reflected in the risk and opportunity rankings.

A total of 59 items were assessed and each was considered as both a potential risk and opportunity. The number of items assessed should not be an indication of the risk or opportunity around the project, but of the thoroughness of the assessment.

No risks or opportunities scored above a moderate risk ranking. A moderate risk represents a threat to the project and effective additional mitigation measures are recommended. The highest risk scored 10/100 and the highest opportunity scored 8/100. The low values of the risk and opportunity scoring are typical for a project which is in the feasibility study stage. No technical risks were identified that were significant enough to recommend that the project does not proceed and the Kemess Underground is consider to be of low technical risk in the context of other caving projects.

Risk and Opportunity Rating

Risks and opportunities were assessed over six major consequence categories and with a range of consequences with five levels of impact. These are outlined below.

The six consequence categories considered during the preparation of the risk and opportunity matrix are presented in Table 24.1.

The consequence categories were calibrated specifically for the Kemess Underground Project so that the impact on NPV was approximately the same for each category assessed. For example, a revenue loss of between 2% and 5% per annum has a similar impact on NPV as a revenue deferral of between 10% and 20% per annum. Each risk and opportunity was assessed using the same scale.

Table 24.1: Summary of Single Factor Consequence Rankings

Category	Insignificant	Moderate	Major	Severe/Extensive	Catastrophic/Extreme
Safety Impact	Reportable Injury	Single Lost-Time Injury	Multiple Lost-Time Injuries or Permanent Disability	Single Fatality	Multiple Fatalities
Revenue Impact	X < 2% over LOM	2% < X < 5% over LOM	5% < X <10% over LOM	10% < X < 20% over LOM	X > 20% over LOM
Revenue Deferral / Advance	X < 10% per year	10% < X <20% per year	20% < X < 30% per year	30% < X < 40% per year	X > 40% per year
Project Capital Costs	X < 10%	10% < X < 20%	20% < X < 40%	40% < X < 60%	X > 60%
Project Operating Costs	X < 5%	5% < X < 10%	10% < X < 20%	20% < X < 30%	X > 30%
Schedule Delay / Advance	X < 1 year	1 year < X < 2 years	2 year < X < 3 years	3 year < X < 5 years	X > 5 years

The likelihood of consequences was assessed on the ranges presented in Table 24.2.

Table 24.2: Summary of Risk and Opportunity Likelihoods

Category	Likelihood
Rare	< 5%
Unlikely	5%–15%
Moderate	15%–50%
Likely	50%–90%
Almost Certain	> 90%

Likelihood and consequence ratings were used to produce a risk or opportunity rating out of 100 for each of the risks, allowing the risks to be categorized into five levels. A summary of the risk and opportunity rankings and their associated levels is presented in Table 24.3.

Table 24.3: Summary of Risk and Opportunity Ratings

Ranking	Risk	Opportunity
0 to 6	Insignificant	Insignificant
6 to 10	Moderate	Moderate
11 to 25	Major	Major
26 to 50	Severe	Extensive
50 to 100	Catastrophic	Extreme

Risks and opportunities are ranked under the assumption that mitigation methods mentioned in this study are put in place, and that industry standard practices are generally implemented. The **five levels of risks** are defined as follows.

- **Insignificant:** Risk is considered well-managed. Monitoring of the assumptions is recommended.
- **Moderate:** Risk represents a threat to the project. Effective additional mitigation measures are recommended.

- **Major:** Risk represents a material threat to the project. It is recommended that the project not proceed until additional mitigation has been assured.
- **Severe:** Risk renders the project, as it is currently characterised, unlikely to succeed. A full strategic revision of the project plan is recommended.
- **Catastrophic:** Risk is of such likelihood and consequence that abandonment of the project is strongly recommended.

The **five levels of opportunities** are defined as follows.

- **Insignificant:** Impact of the opportunity is considered to be of minimal importance to the project.
- **Moderate:** Impact of the opportunity represents an improvement in the project should it be realized. Further investigation into exploiting this potential opportunity should be undertaken.
- **Major:** Opportunity represents a material improvement in the project should it be realized. It is strongly recommended that further investigation into exploiting this potential opportunity should be undertaken.
- **Extensive:** Opportunity renders project, as it is currently characterised significantly sub-optimal, with major opportunity for economic improvement. A full strategic revision of the project plan is recommended prior to further work being undertaken.
- **Extreme:** Opportunity is of such likelihood and consequence that re-engineering the entire project to better understand the impacts is strongly recommended to ensure appropriate capital investment decisions have been made.

24.1.2 Key Project Risks

This section describes all project risks that were deemed to be of moderate significance or greater (>6 score). A moderate risk represents a threat to the project and effective additional mitigation measures are recommended. No risk or opportunity ranked greater than moderate which is typical for projects at this level of study.

Drawpoint Failure: Drawpoint failures could occur due inadequate construction, excess secondary blasting or poor draw control management. These failures would result in moderate revenue losses and increased operating costs for the project. It is noted that some drawpoints may achieve increased height of draw from the base case which will offset some of this risk.

Draw Cone Interaction: There is a moderate risk that the drawpoint spacing is too wide for draw cone interaction. This issue would result in ore loss and premature entry of dilution resulting in lower overall project revenue.

Mudrush Risk: Caving projects in wet climates carry an increased risk of mudrush events occurring. The Kemess Underground Project is exposed to such a risk. A mudrush event would impact revenue and costs, as well as potentially causing fatalities, representing a moderate risk.

Loader Automation: This study assumes that autonomous loaders achieve a certain level of productivity. There is a reasonable likelihood that the assumed level of productivity is not achieved, representing a moderate risk to the project.

Mining Dilution: There is a moderate risk that mining dilution entry occurs sooner than has been modelled in PCBC™ due to a lack of drawcone interaction or by over-mucking individual drawpoints, leading to lower ore grades. It should be noted that this risk manifests itself as a symmetrical opportunity in the case of the KUG.

Underground Development Schedule: There is a moderate risk that the assumed development advance rate of 135 m/month may not be realized. The underground development is a critical path item for the project schedule; a slower advance rate would result in both a schedule delay and increased capital costs.

Recovery and Concentrate Grade: Limited metallurgical testwork in relation to the effectiveness of regrinding KUG ore has been undertaken. There is a moderate risk that the energy requirements for the regrind are greater than anticipated resulting in a coarser grind size than required to achieve the recovery predicted in this study, reducing the gold revenue. Installation of additional regrind mills would likely resolve this, with the associated CapEx.

Commodity Price: There is a moderate risk that the long term commodity prices are lower than those used for project evaluation.

Labour Market: There is a moderate risk that improved economic conditions could conceivably lead to an increase in the demand for skilled labour, leading to a shortage of employees with suitable qualifications and experience to achieve the designed project. This would manifest itself in increases in both capital and operating costs.

Electricity Cost: There is a moderate risk that higher electricity costs to be realized than those assumed in this study resulting in higher capital and operating costs. It should be noted the uncertainty around the estimate represents a more significant downside than upside.

Fuel Cost: There is a moderate risk that fuel costs increase above those used for this study resulting in increases in both capital and operating costs.

24.1.3 Key Project Opportunities

This section describes all project opportunities that were deemed to be of moderate significance or greater (>6 score). A moderate opportunity represents a material opportunity to the project and effective additional mitigation measures are recommended.

Mining Dilution: There is a moderate opportunity that mining dilution entry occurs later than has been modelled in PCBC™ due to good drawcone interaction or well executed drawdown strategy, leading to higher ore grades. It should be noted that this opportunity manifests itself as a symmetrical risk in the case of the KUG.

Draw Rate: There is a high probability and therefore a moderate opportunity that the cave rock mass competency allows for an increased draw rate as the cave matures.

Commodity Price: There is a moderate opportunity that the long term commodity prices are higher than those used for project evaluation.

Labour Market: There is a moderate opportunity that economic conditions could conceivably lead to an decrease in the demand for skilled labour, this could lead to a surplus of potential employees with suitable qualifications and experience to achieve the designed project. This would manifest itself in decreases in both capital and operating costs. The timing of the project in conjunction with end of mine life of similar operations in western Canada provides further possibility that the required skilled labour becomes available to the project.

Mineral Reserve Increase: There is a moderate opportunity to increase the Mineral Reserve if the KUG TSF can be engineered to hold more tailings and/or identify alternative tailings storage location(s). This could be due to increasing the proportion of indicated resource that can be classified as probable reserve or drawing inferred material through drawpoints and realizing estimated ore values.

Fuel Cost: There is a moderate opportunity that fuel costs decrease below those used for this study resulting in decreases in both capital and operating costs.

Table 24.4: Summary of Key Risks and Opportunities

ID	Description	Likelihood	Safety Impact	Revenue Impact	Revenue Deferral/Advance	Capital Costs	Operating Costs	Schedule Delay	Ranking
Risks									
GTC01r	Drawpoint Failure	15%–50%	-	10% < X <20% over LOM	-	-	X < 5%	-	10
GTC02r	Draw Cone Interaction	15%–50%	-	5% < X <10% over LOM	-	-	-	-	6
GTC04r	Mudrush Risk	5%–15%	Single fatality	-	X < 10% per year	X < 10%	X < 5%	-	9
UGM01r	Loader Automation	15%–50%	-	-	X < 10% per year	X < 10%	X <5%	-	6
UGM04r	Mining Dilution	15%–50%	-	5% < X <10% over LOM	-	-	-	-	6
UGM08r	Underground Development Schedule	15%–50%	-	-	-	10% < X < 20%	-	X <1 year	6
PRO06r	Recovery and Concentrate Grade	50%–90%	-	2% < X <5% over LOM	-	- X < 10%	-	-	8
EXM01r	Commodity Price	15%–50%	-	10% < X < 20% over LOM	-	-	-	-	8
EXM04r	Labour Market	15%–50%	-	-	-	10% < X < 20%	5% < X < 10%	-	8
EXM06r	Electricity Cost	15%–50%	-	-	-	X < 10%	5% < X < 10%	-	6
EXM05r	Fuel Cost	15%–50%	-	-	-	X < 10%	5% < X < 10%	-	6
Opportunities									
UGM04o	Mining Dilution	15%–50%	-	5% < X <10% over LOM	-	-	-	-	6
UGM11o	Draw Rate	> 90%				X < 10%	X < 5%		7
TMG06o	Mineral Reserve Increase	50%–90%		5% < X <10% over LOM					8
EXM01o	Commodity Price	15%–50%	-	10% < X < 20% over LOM	-	-	-	-	8
EXM04o	Labour Market	15%–50%	-	-	-	10% < X < 20%	5% < X < 10%	-	8
EXM05o	Fuel Cost	15%–50%	-	-	-	X < 10%	5% < X < 10%	-	6

24.2 Conclusions

24.2.1 KUG Mineral Resource Estimate

The Mineral Resource estimate presented in Section 13 represents an update to the Kemess Underground Mineral Resource estimate and includes new 3D models of geological structure, lithology and alteration. In addition, a new interpretation of grade estimation domains, a change in grade estimation parameters and a new block size were incorporated in the models.

The Mineral Resource model was prepared by SRK considering 149 drillholes drilled by AuRico Metals and predecessor companies during the period of 1976 to 2011. The structural and lithological model was prepared by AuRico Metals, while the alteration model and grade estimation was prepared by SRK Vancouver.

SRK is of the opinion that the current drillhole information is sufficiently reliable and accurate to interpret with confidence the boundaries of the porphyry mineralization, and the assay data are sufficiently reliable to support the Mineral Resource estimation. The audit showed no significant errors from the resource area regarding the recording of tabulated analytical data. The Kemess Underground database for the 2016 resource was verified and can be relied upon for resource estimation.

It was concluded that the resource model reasonably matches the data, while reviewing the results on section. Locally the estimations were reviewed by graphing the mean grade of the variable against the easting, northing and elevation for each of the domains (Swath Plots). The block averages in general bisect the sample plot indicating no bias between the model estimate and the samples.

To assess the sensitivity of the resource to changes in NSR cut-off, SRK summarized tonnage and grade above cut-off at a series of increasing NSR cut-offs by resource category. It can be observed that the resource is reasonably insensitive to cut-off values in the increment from C$14.0/t NSR to C$16.0/t NSR, which is likely the range of interest.

SRK considers that the blocks located within the conceptual panel cave show “reasonable prospects for eventual economic extraction” and can be reported as a Mineral Resource.

24.2.2 KE Mineral Resource Estimate

The Mineral Resource estimate presented in Section 13 represents an update to the Kemess East Mineral Resource estimate and includes new 3D models of geological structure, lithology and alteration. In addition, a new interpretation of grade estimation domains, and a change in grade estimation parameters were incorporated in the models.

The Mineral Resource model was prepared by SRK considering 66 drillholes drilled by AuRico Metals and predecessor companies during the period of 2005 to 2015. The structural and lithological model was prepared by AuRico Metals, while the alteration model and grade estimation was prepared by SRK Vancouver.

SRK is of the opinion that the current drillhole information is sufficiently reliable and accurate to interpret with confidence the boundaries of the porphyry mineralization, and the assay data are sufficiently reliable to support the Mineral Resource estimation. The audit showed no significant errors from the resource area regarding the recording of tabulated analytical data. The Kemess East database for the 2016 resource was verified and can be relied upon for resource estimation.

It was concluded that the resource model reasonably matches the data, while reviewing the results on section. Locally the estimations were reviewed by graphing the mean grade of the variable against the easting, northing and elevation for each of the domains (Swath Plots). The block averages in general bisect the sample plot indicating no bias between the model estimate and the samples.

To assess the sensitivity of the resource to changes in NSR cut-off, SRK summarized tonnage and grade above cut-off at a series of increasing NSR cut-offs by resource category. It can be observed that the resource is reasonably insensitive to cut-off values in the increment from C$16.0/t NSR to C$18.0/t NSR, which is likely the range of interest.

SRK considers that the blocks located within the conceptual panel cave show “reasonable prospects for eventual economic extraction” and can be reported as a Mineral Resource.

24.2.3 KUG Mineral Reserve Estimate

SRK is of the opinion that the Mineral Reserve estimate has been completed to a level appropriate for feasibility studies.

To enable a single cut-off parameter to be used when selecting the economic parts of the resource to be mined, SRK estimated the NSR value received for concentrate produced from each tonne of resource, net of all costs and losses incurred in the off-site transportation and processing of concentrate.

The Mineral Reserve Estimate stated in this report is constrained by the size of the facility that can receive the tailings or 107 Mt. The Mineral Resources statement includes 246 Mt of indicated material of which a significant proportion could be converted into Mineral Reserves, providing alternative tailings storage location(s) could be identified.

24.2.4 Mining Method

The selection of cave mining as the preferred underground mining method was made in 2011 as part of a PEA completed by AMC Mining Consultants. The 2013 FS and this 2016 update continued with this decision. SRK`s scope included reviewing existing geotechnical reports and data, collecting and reviewing additional geotechnical data, providing geomechanical mine design parameters, caveability assessment, fragmentation assessment, and ground support recommendations.

Numerical modelling of the KUG was based on rock mass properties provided by SRK. Itasca modelling results show the KUG deposit will cave given the planned dimensions and the cave will propagate to surface 1.5 years after cave initiation.

The severity of hang-ups varies between geotechnical domains, with the Black Lake Intrusive expected to generate the largest frequency and largest average size. The Hazelton and Takla Potassic domains are expected to generate relatively few and small hang-ups and the Takla Phyllic domain is expected to be relatively insignificant in terms of hang-ups.

The KUG workings are accessed by a triple decline system, with declines for access (of personnel, equipment and materials), conveying and intake air ventilation. This approach is adopted due to challenging topography in the project area and to minimize the environmental impact of the KUG project. The primary ventilation circuit is completed by a dedicated return air raise to surface from south of the cave footprint.

The mine layout includes undercut and extraction levels. The undercut level is located at the 1,158 mRL and is accessed via ramps from the extraction level. The extraction level is located at the 1,140 mRL and is accessed via the access decline at the 1,099 mRL.

Caving zone ore will be mucked from drawpoints by 8.6 m^3 (17t) semi-autonomous LHDs and dumped into jaw crushers located to the south of the cave footprint. Transfer conveyors will then transport ore to the main underground conveyor.

The main underground conveyor, 3.2 km in length, will transport ore from the transfer conveyor to a transfer located adjacent to the triple decline portals. An overland conveyor, 4.9 km in length, will then transport ore to the existing crushed ore stockpile ahead of the process plant.

Total ore tonnes associated with the KUG Feasibility Study is 107.3 Mt, with target production being 9.0 Mt/y or 25 kt/d based on 360-day per year operation. Potential exists to increase production beyond 9.0 Mt/y based on the number of drawpoints, with secondary crushing required to achieve increased production. Production is currently constrained by underground ventilation and tramming limitations, and potentially by processing plant throughput.

The ingress of water into the underground workings is likely to increase when the cave propagates into the “broken zone”. Extreme inflows resulting from snowmelt and rainfalls will be mitigated by using the lower ventilation drifts for storage. Extreme inflow events that flood the ventilation level may result in short-term losses of production.

24.2.5 Processing/Process Plant

The KUG project intends to process 9 Mt/y (24,600 t/d equivalent) through one of the two original Kemess grinding circuits that processed Kemess South material. The “A side” grinding circuit has been removed. The existing Kemess “B side” grinding circuit will be used to process the ore following primary crushing underground and stockpiling ahead of the concentrator. The original flotation, thickening, and concentrate handling facilities remain, while the tailings will be pumped to the Kemess South open pit where tailings were being deposited near the end of the Kemess South operation from 2009 to 2011.

A comparison of the Kemess South and KUG ores has been made to determine the suitability of the Kemess South flowsheet and equipment. From the last two years of operating data, the Kemess

concentrator achieved the equivalent of more than 9 Mt/y of throughput on one side of a parallel circuit with a Kemess South ore type similar to KUG ore.

A significant mineralogical difference in the KUG ore is its higher pyrite to chalcopyrite ratio. This will result in greater mass flow of rougher concentrate to the regrind circuit than similar Kemess South Hypogene ore. In addition, the finely disseminated nature of the copper and gold particles necessitates a finer regrind size than the Kemess regrind circuit had historically achieved.

Existing regrind circuit capacity will not be sufficient to handle the greater rougher concentrate mass pull combined with the finer grind size requirements of the underground ore. A series of fine grinding tests were completed to estimate the energy required for the regrind circuit. A stirred mill regrind circuit has been included in the flowsheet following the existing regrind ball mill, and capital has been allocated in the study.

The predicted metallurgical results have been based on a comparison of the locked cycle test results from a 2003 program and more recent testwork in 2011. It is expected that the KUG concentrator will generate a copper concentrate of 22% Cu at 91% copper recovery, 72% gold recovery, and 65% silver recovery.

The copper concentrate produced from the KUG orebody will contain both gold and silver credits and is expected to be free of deleterious elements, not subject to any penalties, and marketable to both smelters and traders.

24.2.6 Surface Infrastructure

A dedicated access corridor is to be developed to connect the existing Kemess South facilities to the KUG workings, with access via the west side of the Kemess South open pit and through an access tunnel (865 m length) to the Kemess Lake Valley. In addition to providing access for personnel, equipment and materials, this access corridor will contain the electrical powerline for the underground mine, ore conveyor and mine dewatering pipeline.

For KUG tailings storage, it is planned to use the Kemess South open pit, with a capacity for approximately 107 Mt tailings and 3.0 Mt waste rock from underground development. To achieve this storage capacity, a 25 m high dam (East Dam) is required to be constructed at the east end of the open pit. A spillway will be constructed in the south-west area of the pit to allow discharge of excess water once KUG operations have ceased while ensuring adequate water cover of PAG tailings.

Additional capacity is potentially available in the Kemess South open pit, but this would require further raising the East Dam, constructing an additional dam in the South saddle area and reconfiguring the closure spillway.

For water management, a more conservative scenario has been selected for this update compared with the earlier 2013 FS. Based on extensive modeling, the scenario assumes discharge from the Kemess South pit will begin prior to process plant start-up and will require treatment for selenium and dissolved metals. The water treatment plants were designed by BioteQ Environmental

Technologies Inc. to remove contaminants from impacted underground and surface water streams prior to discharge into Attichika Creek.

The selenium treatment plant will be installed in Year -4 and located on the ground floor of the existing process plant building. The metals treatment plant will be installed in Year -1 on a newly constructed second floor of the same building. Both plants will be predominantly supplied as pre-fabricated modules to minimize on-site construction.

Existing Kemess South infrastructure that can be utilized for the KUG project includes the grid-connected electrical powerline and access road (both originating at Mackenzie, approximately 400km south-east of Kemess), admin-workshop-warehouse complex, camp (including kitchen, sewage, potable water supply and recreation facilities), airstrip, site roads, and fuel tank farm.

24.2.7 Environmental and Social

The KUG project is in essence the addition of an undeveloped ore body to an existing mining project. As such, it has the advantage of being able to make use of significant environmental data gathered throughout two previously completed Environmental Assessments (EA) as well as 15 years of data gathered during development and operations.

AuRico has engaged substantively with the surrounding communities to inform them of plans and gain their support for the project. Since 2010, AuRico has continued to meet regularly and consult with the Tse Keh Nay (TKN) prior to and during the EA process. Consultations to-date have been positive to support the EA review and permitting process. TKN and AuRico signed an Interim Measures Agreement in May 2012 and are currently negotiating an Impacts and Benefits Agreement.

At KUG closure, mine dewatering will cease and water-retaining bulkheads will be installed within the declines to flood the underground workings. Once groundwater levels have rebound to stable levels, contact water from the underground will discharge via natural groundwater flow paths to East Cirque Creek. Groundwater quality in the area of the underground mine is not expected to change materially from present conditions and long-term water treatment is not anticipated.

KUG tailings and waste rock will be stored in the Kemess South open pit. The PAG waste materials will be stored under a water cover to prevent metal leaching or acid rock drainage. As the mine approaches the end of its operational life, a large above-water NAG tailings beach will be constructed at the east end of the KUG TSF. At closure, this NAG tailings beach will extend from the East Dam to the supernatant pond at the west end of the KUG TSF.

24.2.8 Economic Evaluation

A total of C$950M is estimated to be spent in capital over the life of the project. Preproduction expenditure is estimated to be approximately C$524M to first ore production, and an additional C$163M to achieve commercial production which is anticipated to be offset by C$83M in pre-commercial production revenue to give a total C$604M net cash required for project development to achieve commercial production. Ongoing sustaining capital is approximately C$263M, including underground mobile equipment replacement and ongoing underground development.

The economic analysis was undertaken using an Excel-based DCF model that modelled cash flows by quarter (three-month periods). A nominal discount rate of 5% was used with an exchange rate (US$ per C$) of 0.75. Long term commodity pricing used was US$2.50/lb for copper, US$1,200/oz for gold and US$16.00/oz for silver. SRK believes the metal prices that have been chosen are realistic price scenarios reflecting industry norms.

The cost of labour is the primary driver of the operating costs. The total unit operating costs after the commencement of full commercial production are estimated at C$14.27/t. This cost is made up of a mining cost of C$5.39/t, a processing cost of C$5.69/t, a water treatment cost of C$0.26, and a site services and G&A cost of C$2.93/t.

The major assumptions used for the cost estimation are C$0.91/litre for diesel fuel, C$0.50/litre for propane and C$0.058/kWhr for electrical power. The labour rates were provided by AuRico and include base wage/salary, production bonus and 30% burden of the base for all employee categories. Shift rotations are four days on and three days off for senior staff and two weeks on and two weeks off for all other employees.

25 Recommendations

This section identifies opportunities that can increase confidence in the success of the KUG project and add economic value, as well as opportunities that can increase the confidence in the KE resource estimate. Additional details are found in the body of this report.

25.1 Mineral Resources

The mineral resources for both the KUG and KE deposits are considered quite well defined. However, confidence in both resource models could be increased, by replacing key drillholes that are currently based on unreliable downhole survey resuts, with a drillhole that is accurately surveyed. Drilling to better define the post mineralization structures, as well as drilling to better define the potassic alteration zones, could also lead to a higher confidence resource model as well as potentially increasing the resources.

25.2 Mine Design

Caveability and fragmentation will vary from the base case because there is always a degree of uncertainty scaling up from empirical and numerical modelling. Additional rockmass data would reduce that uncertainty and enhance confidence in the representativeness of the given information.

A fully integrated mud rush risk assessment is required prior to commencement of caving. It should be noted that the presence of a mud rush risk does not make mining infeasible.

Development is the largest contributor to mine capital costs. Internal ramps, portions of the perimeter drifts, and even the maintenance facility are examples of potential cost reductions without compromising the project.

25.3 Mineral Reserves

There is an opportunity to significantly increase the reserves beyond that stated in this report, depending on available tailings storage capacity. The tailings facility can be optimized to determine how much material can ultimately be stored within or alternative storage location(s) identified.

With additional tailings storage capacity, a significant proportion of the 246 Mt of indicated material in the Mineral Resources can be converted into Mineral Reserves.

25.4 Processing

Future metallurgical testwork could provide further optimization of the target primary and regrind sizes for the various ore zones in the deposit.

The addition of a secondary crusher ahead of the existing process plant has the potential to increase throughput by 10% or higher to a nominal 9.9 Mt/y depending on ore hardness. At this throughput, this approach is revenue neutral, but if higher throughputs than 9.9Mt/y are achieved it is expected to increase project NPV.

25.5 Environmental and Social

It is recommended the KUG project be advanced through the BC environmental assessment process with a high quality EA Application, thereby ensuring optimal review and approval timelines. In addition, AuRico should continue a high level of consultation with the TKN First Nations, other Aboriginal groups and the public in order to build support for the project.

Contingency plans should be developed for water management during operations and post-closure to ensure discharge water quality guidelines are achieved. End land use objectives for the KUG project are recommended to be consistent with the former Kemess South Mine.

26 Acronyms and Abbreviations

A	amp
A*b	hardness value from SMC test
ACP	aboriginal consultation plan
ACSR	aluminum conductor steel-reinforced
AECL	Atomic Energy of Canada Limited
AIR	application information requirements
AMC	AMC Mining Consultants
AME	Association of Mineral Exploration
AMEC	AMEC Foster Wheeler
AMSL	above mean sea-level
APEGBC	Association of Professional Engineers and Geoscientists of British Columbia
ARD	acid rock drainage
ATV	acoustic televiewer
AOI	area of interest
AuRico	AuRico Metals Inc.
BC	British Columbia
BCF	block cave fragmentation
BioteQ	BioteQ Environmental Technologies
BPD	bearing, plunge, dip
BWI	Ball Mill Work Index
C	Centigrade
C$	Canadian dollar
CapEx	capital expenditure
CDI	Conveyor Dynamics Inc.
CEA	Canadian environmental assessment

CEAA	Canadian environmental assessment agency
CIM	Canadian Institute of Mining and Metallurgy
COPC	contaminants of potential concern
CPD	certified project description
CPI	consumer price index
CSA	Canadian Standards Association
d	day
DCF	discounted cashflow
DCS	distributed control system
DFN	discrete fracture network
DMT	dry metric tonne
DXF	drawing exchange format
EA	environmental assessment
EAA	environmental assessment act
EAC	environmental assessment certificate
EAO	environmental assessment office
EDA	experimental data analysis
EMC	environmental management committee
EPA	environmental protection agency
EPCM	engineering procurement and construction management
EPIC	EAO's project information centre
EPS	enhanced production scheduler
ERM	ERM Consultants Canada
Exen	Exen Consulting Services
F_{80}	80% passing (feed size)
ft	feet

FIFO	fly-in, fly-out
FS	feasibility study
G&A	general and administrative (expenses)
GCMP	ground control management plan
h	hour
HOD	height of draw
HR	hydraulic radius
HSSE	health, safety, security and environmental
IBS	intact block strength
IMA	interim measures agreement
in	inches
IRMR	in-situ rock mass rating
IRR	internal rate of return
IRS	intact rock strength
KE	Kemess East
KLV	Kemess Lake Valley
KN	Kemess North
KS	Kemess South
kt	kilotonnes (‘000s of tonnes)
kW	kilowatts
KwN	Kwadacha Nation
KUG	Kemess Underground
kV	kilovolts
KWM	KWM Consulting
lb	pound
LCT	locked cycle test (flotation)

LHD	load haul dump
LOM	life of mine
m	metre
MCC	motor control centre
MCM	thousands of circular mils
MCE	maximum credible earthquake
mm	millimetre
ML	metal leaching
MMPO	major mines permitting office
mRL	metres from reduced level (i.e. elevation)
MRMR	modified RMR
mW, mH, mL	metres high; metres wide; metres long
M	Mega or million
MNBC	Metis Nation of BC
MVS	Mine Ventilation Services
NAG	non acid-generating
NPV	net present value
NSR	net smelter return
OpEx	operating expenditure
ORAR	Omineca Resource Access Road
PD	project description
PFD	process flow diagrams
PLC	programmable logic controller
PVC	polyvinyl chloride
OTV	optical televiewer
oz	ounce (troy)

PAG	potentially acid-generating
PMF	probable maximum flood
RMR	rock mass rating
RQD	rock quality designation
Q	quarter (three month period)
QAQC	quality assurance, quality control
P_{80}	80% passing (size)
PEA	preliminary economic assessment
PFC	particle flow code
SAG	semi-autogenous grinding
SIC	senior implementation committee
SOP	standard operating procedures
SRM	synthetic rock mass
SRK	SRK Consulting Canada Inc.
STOL	short take-off and landing
Struthers	Struthers Technical Solutions
t	tonnes
TARP	trigger action response plans
TDR	time domain reflectometry
ToC	table of conditions
TLFN	Takla Lake First Nation
TKDN	Tsay Keh Dene Nation
TKN	Tse Keh Nay
TSF	tailings storage facility
µm	micron
UCS	unconfined compressive strength

UG	underground
US$	United States dollar
UTM	universal transverse mercator
V	volt
VC	valued component
VFD	variable frequency drive
VVVF	variable voltage, variable frequency
WQG	water quality guidelines
WTP	water treatment plant
Y, y	year

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Date and Signature Page

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All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.