NI 43-101 Technical Report for the Kemess Underground Project, British Columbia, Canada

Report Prepared for

AuRico Gold Inc.

Report Prepared by

SRK Consulting (Canada) Inc.
2CN025.001
April 1st, 2013

NI 43-101 Technical Report for the Kemess Underground Project, British Columbia, Canada

AuRico Gold Inc.

1076 Main Street, Suite 2
P.O Bag 3519
Smithers, BC V0J 2N0

SRK Consulting (Canada) Inc.
Suite 2200-1066 West Hastings Street
Vancouver, B.C., Canada V6E 3X2

e-mail: vancouver@srk.com
website: www.srk.com

Tel: +1.604.681.4196
Fax: +1.604.687.5532

SRK Project Number 2CN025.001

Effective date: December 31, 2012
Signature date: April 1, 2013

Authored by:

Andrew Witte, PEng
Senior Geotechnical Engineer (AMEC)

Chris Bostwick, FAusIMM
Senior Vice President (AuRico Gold Inc.)

Gordon Skrecky, PEng
Project Manager (AuRico Gold Inc.)

Harold Bent, PGeo
Environmental Manager (AuRico Gold Inc.)

Jarek Jakubec, CEng MIMMM
Corporate Consultant (SRK)

Jeffrey Volk, FAusIMM, CPG
Director, Reserves and Resources (AuRico Gold Inc).

Kenneth Major, PEng
President (KWM Consulting Inc.)

Pacifico Corpuz, PEng
Senior Mining Engineer (Tetra Tech WEI Inc.)

Peer Reviewed by:

Gilles Arseneau, Ph.D., PGeo.
Associate Consultant (SRK)

Executive Summary

Responsibility

This report was prepared by SRK with primary contributions from AuRico Gold Inc, AMEC, Conveyor Dynamics Inc, KWM Consulting, Lorax Environmental Services Ltd., and Tetra Tech Inc. Each Qualified Person is responsible for their respective sections, SRK does not accept liability for the statements, findings, and opinions expressed in the portions of this report authored by other contributors.

Introduction

SRK Consulting (Canada) Ltd. was retained by AuRico to prepare a Canadian Securities Administrators (CSA) National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101) compliant technical report on the Kemess underground project (KUG) located in north-central British Columbia, approximately 250 km north of Smithers and 430 km northwest of Prince George (Figure i). AuRico Gold Inc (AuRico) holds the surface rights to the property on which the KUG is located. It holds these rights through its mineral claims and mining leases.

The property was host to the former Kemess South open pit mine. During the life of that mine, 2.975 Moz of gold and 749 Mlb of copper were recovered from 218 Mt of ore. Open pit mining ceased in 2010 on depletion of the mineral reserves, processing of low grade stockpile continued until March 2011. The new mineral resource lies approximately 6.5 km to the north of the existing Kemess processing complex and 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 tree line. Access to the area overlying the deposit is currently limited to summer months only. The processing plant and accommodation camp are located at an elevation of approximately 1,200 m and are accessible year round.

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 onsite infrastructure consists of offices, maintenance facilities, a 300 man accommodation camp, concentrator, raw ore stockpile areas, access and service roads, airstrip, explosives depot, and tailings storage facilities.

The Kemess South mine operated on a fly-in fly-out basis with the majority of employees working on a two weeks on, two weeks off cycle, commuting from regional centres and Vancouver. The airstrip can accommodate most short takeoff and landing (STOL) aircraft types. Lockheed C-130 Hercules (Hercules) aircraft have occasionally used the airstrip for heavy lifts.

An AuRico owned, 380 km power line originating in Mackenzie, provides power to the mine site via the BC Hydro grid.

Process water is reclaimed from the tailings facility and from the Kemess South open pit. Potable water is sourced from permitted wells and treated through an onsite water treatment plant.

Figure i: Kemess property location.

AuRico currently maintains the mill and infrastructure facilities on care and maintenance pending decisions regarding the Kemess underground deposit and other potential projects in the area.

The top of the KUG potential economic mineralized zone is about 150 m below the surface of mountainous terrain and extends down to at least 600 m below the surface. The footprint of the orebody as outlined by current underground mining studies is approximately 540 m long and up to 230 m wide. The terrain of the Kemess underground (KUG) deposit is illustrated in Figure ii.

Figure ii: Surface terrain of the KUG deposit.

Resource Estimate

The KUG mineral resource estimate is shown in Table i. The mineral resources are contained within a portion of the resource block model judged to be potentially minable by block caving.

Table i: Mineral resource statement, Kemess copper-gold-silver deposit, northwest British Columbia, Canada, December 31, 2012.*

Resource category	Tonnes (000's)	Cu grade (%)	Au grade (g/t)	Ag grade (g/t)	Contained metal		
					Cu (000's lb)	Au (000's oz)	Ag (000's oz)
Measured	0	0.00	0.00	0.00	0	0	0
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 cutoff. 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, 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.

(6) Contained metals are in situ and undiluted, and do not include metallurgical recovery losses.

Mining

The Kemess block cave underground mine will be located approximately 6.5 km north of the Kemess plant site. A proposed 4 km road and overland conveyor will lead to the portal entrances. Access to the underground will be via twin 3.25 km long declines. One decline will be dedicated to a conveyor, while the other will provide general access.

Infrastructure required for the mine will include 5 m diameter intake and exhaust raises and associated fans capable of delivering 400 m^3/s. The main declines will have a dedicated ventilation system separate from the main mine for safety reasons.

Other mine infrastructure will include two pumps to manage water inflows, which will be capable of pumping 70 L/s each. Only one pump will be used at any one time with the other as a backup and standby.

There will be one maintenance facility capable of doing most of the underground mobile maintenance. Major equipment rebuilds will be done at the surface shops that were previously used for the open pit.

The Kemess block cave mine was designed using Gemcom's GEMS™ Footprint Finder module, and PCBC™ software. The planned production schedule calls for approximately 100 Mt of ore mined, at a head grade of 0.28% copper and 0.56 g/t gold.

Geotechnical information was reviewed and cavability and fragmentation parameters established based on empirical observations and Itasca Consulting Group Inc.'s (Itasca) numerical modeling.

The final design establishes a single extraction level that includes 640 drawpoints over a footprint that is approximately 540 m east to west and 230 m north to south . 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.

The ore will be recovered on the 1160 extraction level using 10.7 m^3 load-haul-dump (LHD) machines (21 t). They will deliver the ore directly to a single 54" x 75" gyratory crusher centrally located immediately south of the orebody. The material will be crushed and then put directly on to a 42" (1067 mm) wide conveyor belt that will be powered by three 900 hp (671 kW) motors.

The 3.4 km long underground conveyor will rise 230 m vertically and transfer the ore to another 4.6 km surface conveyor that will be powered by four 500 hp (373 kW) motors and will drop 59 m vertically. A total of 9 Mtpa will be discharged onto a stockpile located at the plant site.

Table ii shows the build-up of the ore tonnage profile to full commercial production by quarter. Table iii shows the production profile by year. This project will take 5 years, from commencement of construction, to achieve commercial production. Operations continue for 12 years once full production is achieved.

Table ii: Pre-commercial production schedule by quarter.

Item	Total	Qtr 1	Qtr 2	Qtr 3	Qtr 4	Qtr 5	Qtr 6	Qtr 7	Qtr 8	Qtr 9	Qtr 10	Qtr 11	Qtr 12	Qtr 13	Qtr 14	Qtr 15	Qtr 16	Qtr 17
		Q4 Year –5	Q1 Year –4	Q2 Year –4	Q3 Year –4	Q4 Year –4	Q1 Year –3	Q2 Year –3	Q3 Year –3	Q4 Year –3	Q1 Year –2	Q2 Year –2	Q3 Year –2	Q4 Year –2	Q1 Year –1	Q2 Year –1	Q3 Year –1	Q4 Year –1
Undercut ore (t)	603,983											41,198	94,630	96,273	89,964	91,240	97,101	93,578
Drawbell ore (t)	1,076,369												137,183	184,347	190,358	202,421	183,698	178,362
Development ore (t)	830,346								24,494	82,542	119,703	110,773	93,957	93,721	96,640	71,401	65,418	71,697
Cave ore (t)	1,515,661												19,633	81,265	169,443	281,471	402,700	561,149
Total ore (t)	4,026,359								24,494	82,542	119,703	151,970	345,403	455,606	546,406	646,533	748,917	904,785
Undercut waste (t)	99,714											7,041	10,600	7,093	20,384	20,225	15,565	18,805
Drawbell waste (t)	244,077												26,501	43,106	35,491	31,294	51,279	56,406
Development waste (t)	1,002,413	57,059	59,521	60,060	58,711	61,065	60,179	109,862	154,694	114,568	61,265	22,775	31,148	28,688	24,110	61,513	20,794	16,400
Total waste (t)	1,346,203	57,059	59,521	60,060	58,711	61,065	60,179	109,862	154,694	114,568	61,265	29,815	68,249	78,887	79,986	113,033	87,638	91,611
Total tonnes	5,372,563	57,059	59,521	60,060	58,711	61,065	60,179	109,862	179,188	197,110	180,967	181,786	413,652	534,493	626,392	759,566	836,555	996,396
Cu grade (%)	0.34								0.14	0.16	0.18	0.24	0.32	0.33	0.35	0.37	0.37	0.37
Cu (t)	13,771								35	132	217	363	1,092	1,491	1,899	2,369	2,785	3,388
Au grade (g/t)	0.69								0.27	0.25	0.30	0.41	0.58	0.63	0.70	0.75	0.78	0.79
Au (oz)	89,389								209	667	1,136	2,025	6,414	9,286	12,288	15,630	18,735	23,000
Ag grade (g/t)	2.06								0.95	1.06	1.32	1.64	2.06	2.02	2.08	2.14	2.16	2.20
Ag (oz)	266,500								748	2,817	5,070	8,016	22,862	29,644	36,617	44,513	52,082	64,131
Total Ore NSR ($/t)	34.8								14.0	14.3	16.5	22.4	30.5	32.6	35.3	37.7	38.6	39.1

Table iii: Annual commercial production schedule.

	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Undercut ore (t)	1,451,305	369,541	348,193	406,820	326,752								
Drawbell ore (t)	3,104,108	739,607	750,983	809,886	803,632								
Development ore (t)	859,388	263,621	241,027	195,414	159,327								
Cave ore (t)	90,932,306	4,062,465	7,159,300	7,567,162	7,678,592	9,001,078	9,001,316	9,003,067	9,000,678	9,005,076	9,003,157	8,405,710	2,044,706
Total ore (t)	96,347,107	5,435,233	8,499,502	8,979,282	8,968,303	9,001,078	9,001,316	9,003,067	9,000,678	9,005,076	9,003,157	8,405,710	2,044,706
Undercut waste (t)	196,944	74,898	89,877	28,593	3,576								
Drawbell waste (t)	497,278	189,758	169,081	102,852	35,587								
Development waste (t)	298,284	83,404	84,965	88,613	41,301								
Total waste (t)	992,506	348,060	343,923	220,058	80,465								
Total tonnes	97,339,613	5,783,293	8,843,425	9,199,339	9,048,767	9,001,078	9,001,316	9,003,067	9,000,678	9,005,076	9,003,157	8,405,710	2,044,706
Cu grade (%)	0.28	0.37	0.35	0.32	0.31	0.29	0.27	0.26	0.25	0.24	0.23	0.22	0.21
Cu (t)	267,105	19,896	29,347	28,994	28,078	25,848	23,876	23,035	22,493	21,689	21,011	18,503	4,335
Au grade (g/t)	0.55	0.80	0.76	0.69	0.63	0.59	0.53	0.50	0.47	0.45	0.42	0.40	0.38
Au (oz)	1,715,767	139,761	208,525	198,044	180,559	171,673	153,072	143,950	136,205	128,968	122,471	107,715	24,823
Ag grade (g/t)	2.05	2.27	2.33	2.35	2.31	2.24	2.09	2.01	1.94	1.83	1.68	1.65	1.63
Ag (oz)	6,345,007	396,255	637,710	677,068	667,372	648,745	605,102	581,650	561,383	529,522	486,662	446,377	107,163
Total ore NSR ($/t)	28.1	39.0	37.0	33.8	31.7	29.6	26.8	25.5	24.5	23.4	22.4	21.1	20.2

Kemess Underground Mineral Reserve Estimate

The mineral reserves are shown in Table iv and are based on the indicated mineral resources prepared by AuRico.

Table iv: Mineral Reserve Statement, Kemess Copper-Gold- Deposit, Northwest British Columbia, Canada, December 31, 2012.*

Reserve category	Tonnes (000's)	Cu grade (%)	Au grade (g/t)	Ag grade (g/t)	Contained metal		
					Cu (000's lb)	Au (000's oz)	Ag (000's oz)
Proven	0	0.00	0.00	0.00	0	0	0
Probable	100,373	0.28	0.56	2.05	619,151	1,805	6,608
Proven and Probable	*100,373*	*0.28*	*0.56*	*2.05*	*619,151*	*1,805*	*6,608*

Notes*

(1) Estimated at US$3.00/lb Cu, US$1,300/oz Au and $23.00/oz Ag using a cutoff NSR Value of C$15.3/t and a shut-off NSR Value of C$17.3/t of ore. Metallurgical recoveries and other parameters are shown in Tables 22.2, 22.3 and 22.4 .

(2) Mineral reserve tonnage and recovered metal have been rounded to reflect the accuracy of the estimate, numbers may not add due to rounding.

Chris Elliott, FAusIMM, Principal Mining Consultant, SRK generated the mineral reserve estimate according to the *CIM Standards on Mineral Resources and Mineral Reserves Definitions and Guidelines (2010).*

The mineral reserve estimate is based on a block cave mining method that includes a portion of the mineral resource that lies directly above its footprint. Gemcom's PCBC™ software was used to generate a fully diluted production schedule that includes both internal and external dilution at a zero grade.

Processing

The original Kemess processing plant achieved 55–65 ktpd and generated a 20–23% Cu concentrate from the Kemess South open pit ore. Recoveries were 70–85% copper and 55–70% gold with ore grades of 0.1–0.2% Cu and 0.2–0.4 g/t Au.

This compares to an anticipated 22% Cu concentrate for the KUG ore. Recoveries are forecast to be 91% Cu and 72% Au, with resource grades averaging 0.28% Cu and 0.56 g/t Au.

The KUG project will process 24.65 ktpd (9 Mtpa equivalent). The primary crushing plant will be located underground instead of on surface and a conveyor system will deliver the ore 8 km (the converyor is 8 km long, but the distance from the orebody to the plant is 6.5 km ~~as~~ the crow flies") to a stock pile located directly ahead of the concentrator. Grinding will be through half of the original circuit (previously there were two parallel grinding circuits). Flotation, thickening, and concentrate handling facilities remain the same. Tailings will be pumped to the Kemess South pit where tailings were deposited near the end of operations in 2010.

A comparison of the Kemess South and KUG ores was made to determine the suitability of the Kemess South flowsheet and equipment.

Operating data from the last two years of operations suggests that the Kemess concentrator achieved 9 Mtpa on each of two parallel grinding circuits processing ore that was thought to be

similar to KUG material. Subsequent bond work index tests (BWI) conducted on the new orebody also suggest that 9 Mtpa can be achieved with this ore. Nevertheless, SRK recommends additional tests to confirm semi-autogenous (SAG) mill throughput (at P80 150 µm) to be confident that the same equipment can achieve the desired tonnage on a range of underground material.

There are also some mineralogical differences between the two ores.

One difference is the higher pyrite to chalcopyrite ratio in the KUG ore. This will result in greater mass flow of rougher concentrate to the re-grind circuit than similar Kemess South hypogene ore. The original flotation circuit remains to process less than half of the previous tonnage and this will be sufficient for the KUG material.

Another difference is the finely disseminated nature of the copper and gold particles. This difference will necessitate a finer re-grind size than the Kemess re-grind circuit historically achieved. Test work indicates that metallurgical performance deteriorates as the re-grind stream coarsens from 80% (P80) passing 15 µm.

To achieve optimal recovery and a reasonable Cu concentrate grade, the primary grind is expected to be P80 at 150 µm and the re-grind to be P80 at 12 to15 µm. A 22% Cu concentrate is targeted because higher grade concentrates result in significant gold losses.

The existing re-grind circuit capacity will not be sufficient to handle the greater, rougher concentrate mass pull (20–23% of the mill feed) combined with the finer grind size requirements of the underground ore (a P80 at 15 µm or finer). A stirred mill has been added to the flowsheet following the existing re-grind ball mill. SRK recommends additional test work to be conducted by the supplier to verify the tonnage, feed size, and expected product size.

Surface Infrastructure

The Kemess Mine was originally brought into production as a 52,000 t/d open pit mine in 1998 (pre-stripping began in 1997) and produced continuously until 2011.

It was a fully integrated complex that included:

- A 1.6 km air strip capable of handling most STOL aircraft and Hercules transport planes when heavier loads were required, and
- A concentrator capable of processing 52,000 t/d. Half of the grinding circuit has been removed, but the rest of the facility, as listed below, has been kept on a care and maintenance program:
    - A coarse ore stockpile area,
    - An open pit maintenance facility,
    - An administration and service complex and accommodation for over 300 personnel,
    - A Company owned 380 km power line (230 kV) and associated transformers, and
    - An open pit that will continue as the tailings storage facility (TSF) for the KUG project.

These facilities have been maintained on a care and maintenance program following the cessation of processing activities in 2011 while reclamation of the Kemess South mine site continues.

Environmental, Permitting, and Closure General Setting

The Kemess property mined and processed over 200 Mt of ore from the Kemess South open pit from 1998 to 2011. Operating permits have been in place since 1996 and remain in place. The project's footprint has been progressively reclaimed on an annual basis. AuRico has been recognized for being pro-active in the reclamation and closure of the Kemess South mine. They have been the recipient of a number of reclamation awards from the British Columbia Technical and Research Committee on Reclamation (TRCT) including:

2010 – British Columbia Jake McDonald Annual Mine Reclamation Award, recognition for Outstanding Reclamation Achievements at the Kemess South mine, and

2001, 2008, and 2009 – Metal Mining Citation — recognition for Outstanding Achievement in mine reclamation at Kemess South mine.

The Kemess South mine was also the recipient of the 2010 Mining and Sustainability Award from the Mining Association of British Columbia (MABC). This sustainability award honours industry leadership in responsible mining.

The environmental assessment (EA) and permitting framework for metal mining in Canada is well established. The EA process provides a mechanism for reviewing major mine projects to assess their potential impacts. Following successful completion of the EA process, a project enters the permitting phase. The project is then regulated through all phases (construction, operation, closure, and post closure) by both federal and provincial departments and agencies.

The KUG project is in essence the addition of an undeveloped underground ore body to an existing mine operation. 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 existing environment for the proposed development, including the biological and physical components has been characterized. Any potential environmental impacts from the proposed project will have mitigable measures as necessary to address these impacts.

AuRico and the Tse Keh Nay (TKN) have cooperatively initiated studies in the areas identified by TKN to further address other environmental effects.

The most significant potential impacts from a regulatory perspective are associated with the long-term management of waste rock, tailings, mine water, and process water. The potential impacts and proposed mitigation associated with these components will be managed as per the existing Kemess South management strategies that have been developed to meet the requirements of existing authorizations and permits. The physical infrastructure for the long term storage and mitigation is addressed in detail elsewhere in this study.

The existing waste management practices associated with the Kemess South mine and those proposed for the KUG project represent industry best practices.

From a regulatory perspective AuRico has not identified any activities associated with the KUG project that will result in potential adverse environmental impacts that cannot be mitigated through the implementation of good engineering practices and management plans.

Aboriginal and Public Consultations

To date consultations by AuRico have focused on First Nations, specifically the Tse Keh Nay (i.e., Takla Lake, Kwadacha, and Tse Keh Dene First Nations). The majority of the feedback for the previously proposed Kemess North project coming from mine employees, northern BC businesses, local governments, chambers of commerce, economic development organizations, and mining industry organizations was positive and supportive of the project.

Closure and Post Closure Water Management

At closure, the mine will be flooded and water-retaining bulkheads established at the portals. The pipeline from the mine will also be re-configured to deliver mine water directly to the water treatment plant, rather than to the pit. The water treatment plant will be re-configured for long-term use in annual seasonal campaigns, projected to be operational during the months of June and July.

After milling and tailings deposition ceases, and with mine water is no longer entering the pit, water quality in the South Kemess pit is expected to improve. Once it meets discharge criteria without treatment, the pit water will be allowed to flow to Waste Rock Creek via the TSF closure spillway.

It is expected to take roughly 14 years for the mine to flood to the portal level. Thereafter, water will be withdrawn from the mine annually and campaigned seasonally through the water treatment plant. According to estimates provided by Lorax, the total annual volume entering the mine after closure will be 0.9 Mm^3. Roughly that volume will need to be removed and treated each year. It is assumed that the treated water will initially be discharged to Attichika Creek.

If the quality of the treated mine water is demonstrated to be sufficient for discharge via the pit and its closure spillway, it would be routed there and the pipeline to the Attichika Creek decommissioned.

It must be noted that the KUG closure plan does not consider any site remedial work other than that directly associated with the KUG feasibility study. For example the Kemess South tailings, waste dumps, and water runoff are not included. These facilities are covered under the existing South Kemess closure plan. Closure for the purposes of this report primarily involved treating water from the underground mine in perpetuity.

Project Execution

This project will be managed by AuRico personnel. Contractors will be retained under AuRico direction to construct the initial road access to the portal entrances. Another contractor will be retained under AuRico personnel direction to do the initial capital development including driving the twin declines, establishing the ventilation raises, and developing and installing the underground infrastructure.

CAPEX Estimation

All capital and operating costs estimates are based on late 2012 dollars and US$1 = C$1.

The total life of mine capital cost for this project is estimated to be $683 M (Table v). This includes pre-production capital of $502 M, sustaining capital of $146 M, and water treatment capital and closure costs or $35 M, for a total of $683 M.

There are additional pre-production mine, process, and G&A costs totaling $293 M that are offset by revenues of $344 M. Total pre-commercial production net cash flow is negative $452 M.

OPEX Estimation

The cost of labour is the primary driver of all of the operating costs. 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 two week on-site and two week off-site schedule. 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 and weekend coverage will be maintained by rotating a minimum team to ensure coverage of all essential areas.

The total unit operating costs after the commencement of full commercial production is estimated at $14.56/t. This is made up of a mining cost of $6.07/t, a processing cost of $5.00/t, and a G&A cost of $3.50/t (Table vi).

Mining costs are summarized in Table vii. The labour component is just under 60% of all direct and indirect mine operating costs.

Table v:LOM capital cost estimates ($M).

Description	Year –6	Year –5	Year –4	Year –3	Year –2	Year –1	Year 1+	Total
Underground capital								
Mine development	4.6	19.3	40.4	17.4	10.9	7.1	7.8	107
Mining labour	0.1	0.3	11.9	18.6	19.3	22.4	25.3	98
Mobile equipment — purchases/rebuilds	1.8	0.0	17.8	20.6	8.6	17.1	72.5	138
Contractor indirects	2.4	9.7	9.7	9.7	4.9	0.0	0.0	36
Conveyor	0.0	10.0	10.0	10.0	0.1	0.1	0.0	30
Other	6.5	5.3	18.3	42.7	25.7	24.8	12.2	136
Processing plant	0.0	0.0	5.9	15.6	0.0	0.0	0.0	21
Pre-construction owner's costs	4.5	0.0	0.0	0.0	0.0	0.0		5
Camp renovation costs	1.9	1.2	1.3	0.0	0.0	0.0	0.0	4
Access road	5.9	8.8	13.2	0.0	0.0	0.0	0.0	28
Tailings storage facility	0.0	0.0	0.0	0.0	0.0	0.0	28.5	29
Surface admin	3.3	1.4	0.0	0.0	0.0	0.0	0.0	5
Power line along conveyor to U/G	1.2	0.0	0.0	0.0	0.0	0.0	0.0	1
Closure and water treatment capital	0.0	0.0	0.0	10.0	0.0	0.0	34.8	45
Total capital cost (excl. capitalized opex)	32	56	129	145	69	71	181	683
Capitalised mining operating costs	0.0	0.0	0.0	8.9	19.5	26.4	0.0	55
Capitalised processing operating costs	0.0	0.0	0.0	17.0	27.7	44.6	0.0	89
Capitalised G&A costs	12.6	15.8	24.5	30.7	32.9	32.5	0.0	149
Less pre-commercial net revenue	0.0	0.0	0.0	(28.1)	(114.8)	(200.6)		
Pre-production net pre-tax expenditure	45	72	153	173	35	(26)		

Table vi: Operating costs.

Operating costs	LOM ($M)	Unit costs ($/t)
Underground	554.6	$6.07
Concentrator	456.6	$5.00
G&A	319.4	$3.50
Total operating costs	**1,330.6**	**$14.56**

Table vii: Mining cost by function ($M).

U/G direct production costs by process	LOM costs ($M)	Unit costs ($/t)
Drawbell drill & blast	6.0	*$0.07*
Undercut drill & blast	7.7	*$0.08*
Secondary breaking	15.6	*$0.17*
Undercut mucking	1.4	*$0.01*
Production mucking	120.2	*$1.32*
Equip. condition monitoring	1.2	*$0.01*
Operating labour	98.4	*$1.08*
Total underground production costs	250.4	*$2.74*

Indirect underground mining costs	LOM costs ($M)	Unit costs ($/t)
Mining labour	113.1	*$1.24*
Maintenance labour	45.9	*$0.52*
Mining staff	47.4	*$0.50*
Electricity	20.3	*$0.22*
Propane	19.9	*$0.22*
Fixed equipment — parts/materials	16.7	*$0.18*
Mine development/rehab	13.1	*$0.15*
Maintenance staff	13.6	*$0.14*
Diesel fuel	8.0	*$0.09*
Mobile equipment – parts/materials	6.1	*$0.07*
Total other underground costs	304.2	*$3.33*

Grand total UG operating costs	554.6	*$6.07*

The processing cost for commercial production averages $5.00/t. Once full production of 9 Mtpa is achieved the unit cost will fall to $4.96/t (Table viii). The end of life mine unit operating cost increases as production decreases. This increases the overall cost of production from $4.96/t to $5.00/t.

Table viii: Processing cost by function ($M).

Mill operating costs by category	Cost per year at 9 Mtpa ($M)	$/t milled
General & administration	9.0	$1.00
Overland conveyor & reclaim	1.8	$0.19
Grinding	25.4	$2.83
Flotation	2.9	$0.32
Other (assay lab/tailings/reclaim/metallurgy)	5.5	$0.62
Total mill summary at 9 Mtpa	**44.6**	**$4.96**

Mill operating costs for LOM	LOM costs ($M)	$/t milled
Total mill opex (LOM)	**456.6**	**$5.00**

Detailed G&A costs are identified in Table ix. The main drivers for these costs are a direct reflection of the total site manpower. The costs are shown from the start of commercial production.

Table ix: General and administrative costs.

On-site G&A	Costs ($M)	Unit costs ($/t)
Flights & camp	109.3	$1.20
Admin costs	190.0	$2.08
Environmental	20.2	$0.22
Total G&A costs	319.4	$3.50

Economic Analysis

The economic analysis was undertaken using an Microsoft™ Excel® spreadsheet DCF model that modelled cash flows by quarter (i.e., 3 month periods). The nominal discount rate used was 5% and an exchange rate for the US$ versus the C$ of 1:1. Long-term commodity pricing used was $1,300/oz for gold, $3.00/lb for copper, and $23/oz for silver (Table x).

The payback periods are shown in Table xi. The discounted payback period is 5 years from the start of commercial production and 3.5 years for the non-discounted commercial production. Net cash flow after tax is $134 M at a discount rate of 5%. The undiscounted cash flow is $390 M after tax. The IRR is 10%.

Sensitivities to the base case were run for changes to commodity prices (revenues) and costs.The results can be found in Table 22.7 and Table 22.8.

Table x: Base case assumptions.

Base case pricing		Long-term
Copper	$/lb	**$3.00**
Gold	$/oz	**$1,300**
Silver	$/oz	**$23.00**
Exchange rate	USD:CAD	**1:1**
Modeled recovery to concentrate		
Copper recovery to concentrate		**91%**
Gold recovery to concentrate		**72%**
Silver recovery to concentrate		**65%**

Table xi: Payback periods.

Payback periods (years)	From construction start	From commercial production start
Discounted payback @ 5%	10.0	5.0
Non-discounted Payback (real)	8.5	3.5

Interpretations and Conclusions

SRK considers that there are both risks and opportunities for this project. This feasibility study has demonstrated the KUG project will generate a positive non-discounted pre-tax cash flow of C$627 M, ($268 M at a 5% discount rate), or an after tax net present value of C$390 M ($134 M at a 5% discount rate. The after tax IRR is 10% based on 2012 costs; however, the long lead time to production does mean that costs and revenues could be higher or lower than anticipated.

SRK is confident that the KUG orebody will cave, but there is always a degree of uncertainty in scaling up empirical and numerical modelling over the life of a cave and the fragmentation profile cannot be defined with certainty. This may lead to an underground cave ramp up schedule that varies from the base case with associated downside risks.

The potential exists to reduce capital costs in the underground. Development is the largest contributor to those costs and has yet to be optimized. Similarly there needs to be a review of the infrastructure required to access the underground. Ventilation infrastructure also needs to be reviewed.

First Nation and other community concerns may represent a significant risk to the project; however, AuRico and its predecessor Northgate Minerals have engaged with the surrounding First Nation and other communities to inform them of plans and gain their support. SRK has been told that the results to date are positive. First Nation and other community engagement needs to continue in order to avoid potentially significant delays in permitting and approvals.

In summary, the project economics are positive, and the project is located in a political jurisdiction that is pro-mining and where the regulatory environment is understood. To date, SRK is not aware of any fatal flaws for the project.

Recommendations

This report did not try to optimize this project. There was only one main trade-off study that was done (regarding cave design), and others are warranted. The following recommendations can increase confidence and value in this project. This section identifies those opportunities in point form. Additional details are found in the body of the report.

Mineral Reserves:

- There is the opportunity to significantly increase the reserves beyond that stated in this report. Mineral resources are available to support a larger mineral reserve. These reserves are constrained by the size of the facility that can receive the tailings. A trade-off study to optimize the size of the tailings facility is all that is required to increase mineral reserves. No costs are anticpated to be required other than the cost of the trade off study for the Tailings Management Facility (TMF).

Underground Design and Infrastructure:

- It is recommended that more geotechnical information be gathered and analyzed as development advances (including driving drifts throughout the ore before design parameters are finalized). Cavability and fragmentation will vary from base case because there is always a degree of uncertainty scaling up empirical and numerical modelling. Additional data would reduce that uncertainty and enhance confidence in the representativeness of the given information. The technical staff required are included in the project capital costs.
- A fully integrated mud rush risk assessment is required prior to commencement of caving. The presence of mud rush risk does not make mining infeasible. What it does mean is that programs and policies will have to be put into effect to manage these events. The cost of this additional work is anticipated to be $30,000.
- It is recommended that the use of remotely operated LHDs be considered. This is a proven technology that is routinely used in the mining industry. The extraction level layout is considered suitable for their use. This technology has the potential to reduce operating labour, LHD maintenance, and LHD capital costs, and improve safety.
- It is recommended that a trade-off study be conducted to determine the capital and operating benefits associated with replacing the planned intake air raise with an intake air decline. The potential exists to save time on the development schedule and up to $20M in operating costs if heating costs can be eliminated or reduced. This issue requires further study.
- Development is the largest contributor to the capital costs. It is recommended that a detailed review of the proposed infrastructure be undertaken to see if those costs can be reduced. Internal ramps, portions of the perimeter drifts, and even the maintenance facility are examples of potential cost reductions without compromising the project. Existing in house resources will be used for this trade-off study and so no additional costs are anticipated.

Processing:

- It is recommended by SRK that more hardness tests be conducted on the KUG orebody to confirm the representativeness of the work that has been completed and the appropriateness of the assumptions that have been made.

- KWM Consulting has demonstrated, through their work using the BWI method of analysis, that the hardness of the ore should not be a constraint. Nevertheless, it is recommended by SRK that further analysis be done using other methods specific to SAG mills to confirm these results.
- SRK has noted that the existing re-grind capacity will not be sufficient to handle the greater rougher concentrate mass pull (20–23% of the mill feed) combined with the finer grind size requirements for the underground ore (P80 @ 15 µm or finer). Provision has been made to add another mill to the re-grind circuit. SRK recommends that specific test work be conducted by the manufacturer of that mill to verify the tonnage, feed size, and expected product size.

Surface Infrastructure:

- It is recommended that a trade-off study be completed to determine if additional tailings storage capacity is available and economically viable in the Kemess South open pit. A decision was made in this study to minimize capital. A trade-off study might demonstrate that material can be placed in this facility without major capital cost implications.
- It is recommended that a trade-off study be conducted to determine if a tunnel can replace the planned portal access road and overland conveyor. There are several impacts that need to be considered including the development schedule to production, capital costs, and operating costs. The manpower savings could be significant in terms of the potential savings in travel time to and from the underground workplaces.

Environmental Permitting and Closure:

- Community concerns represent a possible risk to the project; however AuRico and its predecessor Northgate Minerals have engaged the Tsay Keh Nay to develop a positive relationship that will be mutually beneficial, and thereby gaining support for the advancement of the Kemess underground project. When the regulatory review process has been finalized, AuRico will initiate broader community engagement in the region.
- This project will require both provincial and federal approvals, and may require federal and/or provincial EA.
- It must be noted that the KUG closure did not consider any site remedial work other than that directly associated with the KUG feasibility study. For example the Kemess South tailings and waste dumps and water runoff are not included. Closure of these facilities is covered under the existing Kemess closure plan. Closure for the purposes of this report primarily involves treating water from the underground mine in perpetuity.
- SRK recommends that AuRico continue to be proactive in its efforts to engage the community so that it can avoid delays in permitting and approvals. The costs for this initiative are already included in the reclamation work being undertaken at the site and so no additional costs are required.
- SRK is unaware of any other significant factors and risks that may affect access, title, or the right or ability to perform the exploration work recommended for the Kemess UG Project.

The costs of the studies discussed above are included in Table xii

Table xii: Recommendations Budget Forecast

Description	Work Requirements	Cost $
Underground Design And Infrastructure		
Ongoing geotechnical evaluations	This work is ongoing and included in capital costs	$0
Mudrush assessment		$30,000
Remotely operated lhds		$20,000
Trade off study for main ventilation intake raise		$20,000
Trade off study to reduce development	Will be done with existing in house resources.	$0
	Subtotal	$70,000
Processing		
Hardness tests		$25,000
Specific sag mill tests		$50,000
Manufacturer regrind testwork		$25,000
	Subtotal	$100,000
Surface Infrastructure		
Trade off study on tailings storage capacity		$100,000
Trade off study on underground access		$130,000
	Subtotal	$230,000
Environmental Permitting And Closure		
Community engagement	This work is ongoing and included in capital costs	$0
	Subtotal	$0
	Total	$400,000

Table of Contents

List of Tables

List of Figures

Appendices

1 Introduction

1.1 Terms of Reference and Purpose of the Report

This technical report was prepared for AuRico Gold Inc. (AuRico) by SRK Consulting (Canada) Inc. (SRK) to summarize the results of a feasibility study (FS) conducted to assess the Kemess underground project (KUG), located on AuRico's property in north-central British Columbia.

This technical report was written by qualified persons (QP) as detailed below. Any previous technical reports or literature used in the compilation of this report are referenced throughout the text.

The following individuals, by virtue of their education, experience, and professional association, are considered qualified persons (QP) for this report, as defined in the Canadian Securities Administrators National Instrument 43-101 (NI 43-101), and are members in good standing of appropriate professional institutions. The QP's responsible for specific sections are listed in Table 1.1

The contract 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: Qualified persons and areas of responsibility.

Name of qualified person	Company	Area of responsibility	Section(s) of responsibility
Jeffrey Volk, CPG, FAusIMM	AuRico	Mineral resource estimation, quality assurance/quality control (QA/QC), geology	6-11, 13, and 23
Jarek Jakubec C,. Eng MIMMM	SRK	Project Principal Director, geotechnical engineering, mine design	1, 2, 15.1-15.3, 24, and 25
Chris Bostwick, FAusIMM	AuRico	Mine design, mine planning	15.4, 15.5, 15.7, 22.2.1, 22.5
Pacifico (Virgil) Corpuz.P.Eng	Tetra Tech	Underground infrastructure, construction and development	15.6.2, 15.6.4–15.6.8, 20.3.1-20.3.4, 21.2, 21.2.1, 21.4.3, and 21.4.4
Andrew Jennings, P.Eng	CDI	Conveyor design surface and underground	15.6 and 16.5
Gordon Skrecky P. Eng	AuRico	Surface infrastructure	3, 4, 5, 17, 18, 20.4.3, 20.4.6, 20.4.7, 20.4.8, 20.4.9, 21.4.1, 21.4.2, 21.6, and 22.2.4
Ken Major P.Eng	KWM Consulting	Metallurgy and mineral processing	12, 16.1-16.3,16.6, 16.7, 20.4.2, 20.4.10, 21.5, and 22.2.3
Andrew Witte, P. Eng	AMEC	Water storage And waste management engineering	16.8, 20.4.4, 20.4.5
Harold Bent, P. Geo	AuRico	Permitting, environment, social impact assessment, and stakeholder consultation	19
Chris Elliott, FAusIMM	SRK	Project Economics and Mineral Reserves	14, 15.6, 15.6.1, 20, 20.1, 20.2, 20.4.1, 21.1, 21.3, 22.1-22.4 and 22.6

1.2 Qualifications of Consultants (SRK)

The SRK Group comprises over 1,100 professionals, offering expertise in a wide range of resource engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This fact permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues. SRK has a demonstrated 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. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

1.2.1 Sources of Information and Extent of Reliance

SRK has received data and information from a number of sources, including:

- AuRico – Resource model, mine design and mine planning, permitting, environment, social impact assessment, and stakeholder consultation;
- Tetra Tech – Underground infrastructure, construction and development;
- Mine Ventilation Services (MVS) – Mine ventilation design;
- Conveyor Dynamics Inc (CDI) – Surface and underground conveyor design;
- KWM Consulting – Metallurgy and mineral processing; and
- AMEC – Water management and waste management engineering

While SRK has reviewed these data and information and has relied on the assumptions and conclusions provided therein, each qualified person is responsible for their sections of the report, SRK does not accept liability for the statements, findings, and opinions expressed in the portions of this report authored by other contributors.

1.3 Units of Measure

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. All currency values are Canadian dollars (C$) unless otherwise stated.

The units of measure presented in this report are metric units. Gold values are reported in parts per billion (ppb) or parts per million (ppm). Gold is also reported in grams per tonne (g/t). Copper is reported in decimal percent (%). Tonnage is reported as metric tonnes (t), unless otherwise specified. This report uses abbreviations and acronyms within mineral industry standards. Explanations are located in Section 27.

1.4 Site Visits

A summary of site visits and areas of focus during the site visit(s) is provided in Table 1.2

Table 1.2: Summary of site visits.

Name of qualified person	Company	Date of visit	Site visit activities
Jeffrey Volk, CPG, FAusIMM	AuRico	March 11, 2013	Tour of open pit, mill, warehouse, admin complex and maintenance shop; fly-over of site and location of new infrastructure and geology review
Jarek Jakubec C. Eng MIMMM	SRK	May 16–19, 2011	Reviewed available geotechnical reports, and existing drillhole database for consistency. Reviewed project location, selected drill core, and had discussions with geological staff.
Chris Bostwick, FAusIMM	AuRico	June 20, 2012	Tour of open pit, mill, warehouse, admin complex and maintenance shop; fly-over of site and location of new infrastructure.
Andrew Jennings P. Eng	CDI	June 25–27, 2012	Walk conveyor route reviewed feasibility study plans.
Pacificico Corpuz P. Eng	Tetra Tech	June 25-27, 2012	Site familiarization and visit proposed portal entrance location
Gordon Skrecky P. Eng	AuRico	July 2006–October 2011 continuously.	Worked at operating site
Ken Major P. Eng	KWM Consulting	August 1999 to May 2009, Intermittent visits	Occassional process consulting for commissioing (PWC), various engineeirng modifications and process optimizations (Northgate), Kemess North (open pit) Feasibility Study (Hatch)
Andrew Witte P. Eng	AMEC	October 22–29, 2011, June 25–27, 2012, September 17–19, 2012	Toured open pit area to layout feasibility level site investigation program for waste management facilities and review highwall diversion schemes. Walked portions of Kemess Lake access road alignment and decline portal areas. Flew over subsidence zone and vent raise areas. Toured general mine infrastructure including camp, mill and service complexes.
Harold Bent P. Geo	AuRico	1998–2003 continuously, 2003–2013 regular as needed visits	Worked at operating site; 2003 onward — oversight of environmental programs
Chris Elliott FAusIMM	SRK		Has not been to site

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 Farris, Vaughan, Wills and Murphy LLP (Farris), as expressed in a legal opinion provided to AuRico, formerly Northgate Minerals on September 30, 2009. AuRico has provided an update letter from internal counsel dated March 01, 2013 confirming that all land title information as at the time of the Farris opinion are up to date and in legal good standing. A copy of the title opinions is provided in Appendix A. The reliance applies solely to the legal status of the rights disclosed in Sections 3.1 and 3.2 below.

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

3 Property Description and Location

3.1 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. 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 2002–2010 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.

Source: http://www.th.gov.bc.ca/popular-topics/distances/bcmap.html

Figure 3.1: Project location.

3.2 Mineral Titles

AuRico holds mineral title to 57 claims totaling 29,285 Ha. AuRico also has leasehold on an additional 4 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 Table 3.1.

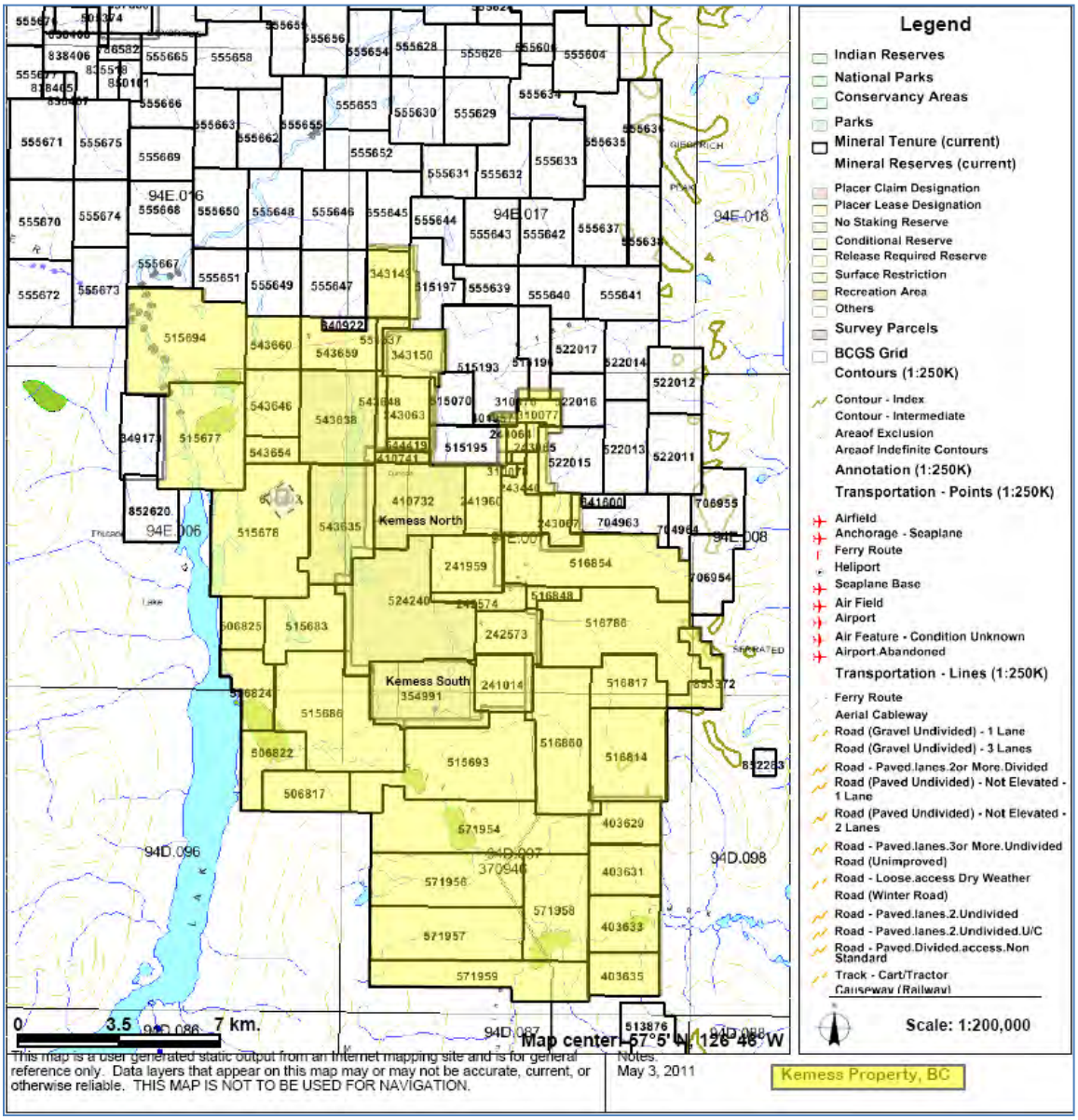


Figure 3.2: Mineral tenure map showing AuRico claim boundaries (yellow shading) and claim numbers.

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

Tenure number	Claim name	Owner ID (Interest)	Company	Tenure sub type	Good to date	Status	Area (ha)
241014	SEM #1	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	400
241959	NEK 3	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	500
241960	NEW KEMESS 3	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	375
242573	DU 2	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	500
242574	NEK 4	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	350
243063	CAN 1	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	500
243064	DUNC 1	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	100
243065	DUNC 2	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	100
243066	DUNC 3	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	150
243067	CREEK	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	300
243440	ALISON 1	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	500
304706	GOZ 1	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	25
304707	GOZ 2	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	25
310076	DUN 1	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	225
310077	DUN 2	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	225
310078	DUN 3	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	225
355408	MILL CREEK 4	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	25
401957	UN 1	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	50
403629	BEAR 6	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	500
403631	BEAR 8	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	500
403633	BEAR 12	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	500
403635	BEAR 16	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	375
405949	LAT 1	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	25
414229	DUNC 4	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	25
414230	DUNC 5	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	25
414231	UN 2	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	25
414232	UN 3	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	25
506817	TLK 1	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	423
506822	TLK 2	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	423
506824	TLK 3	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	388
506825	TLK 4	261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	282
515677		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	1108
515678		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	1443
515683		261048 (100%)	AuRico	Claim	2022/dec/11	GOOD	669
515686		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	1428
515693		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	1534
515694		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	1353
516786		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	1392
516814		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	864
516817		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	441
516848		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	106
516854		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	1197
516860		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	1075
543635		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	898
543638		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	862

Tenure number	Claim name	Owner ID (Interest)	Company	Tenure sub type	Good to date	Status	Area (ha)
543646		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	440
543648		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	106
543654		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	176
543659		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	422
543660		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	351
544419		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	70
571954		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	1341
571956		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	988
571957		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	989
571958		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	847
571959		261048 (100%)	AuRico	Claim	2022/dec/14	GOOD	830
853372	KEMESS CREEK 1	261048 (100%)	AuRico	Claim	2022/aug/31	GOOD	264
57							29285
Tenure		**Owner ID**		**Tenure**			**Area**
354991		261048 (100%)	AuRico	Lease	2016/sep/15	GOOD	862
410732		261048 (100%)	AuRico	Lease	2016/sep/29	GOOD	950
410741		261048 (100%)	AuRico	Lease	2016/sep/29	GOOD	106
524240		261048 (100%)	AuRico	Lease	2016/dec/22	GOOD	1565
4							3483

3.2.1 Nature and Extent of Issuer's Interest

AuRico's predecessor, Northgate Minerals., acquired the Property from receivership from the former operator Royal Oak Mines (Royal Oak) in, 2000. Royal Oak had previously acquired the property from El Condor LTD in, 1996. These claims cover the resources and reserves at the Kemess Project in the North part of the Property. AuRico is the 100% owner of these claims which are currently held in good standing

3.3 Royalties, Agreements and Encumbrances

There are currently no royalty or other underlying agreements with any third parties. SRK is not aware of any liens or encumbrances on the property that would impact project implementation.

3.4 Environmental Liabilities and Permitting

Most of the permits previously in place for the Kemess South operation are still in good standing including road use, power line, explosives, camp related, and mill related such as nuclear guage and boiler permits. Many of these are listed below. Some of the key permits to be obtained include a new mining permit, shown as M206 below, and an amendment to the existing permit to deposit tailings into the exhausted open pit, PE 15335, to allow the construction of an embankment to increase the tailings capacity. The mining permit covers the operation of the facility during production as well as the eventual closure and reclamation of the site.

Table 3.2: Table of current permits.

Permit number	Description
01-7829.01-99	Finlay Rd.
01-7829.08-99	Finlay–Osilinka Rd.
01-9147.01-99	Thutade Forest Service Rd.
01-SS22414-97	24 small road sections, i.e., Nation Dump Rd., Modeste Mainline, Blackpine FSR, Manson Dump Rd.) Special Use Permits (SUP) MOF — Issued February 21, 1997
80221125	Radio license — issued March 25, 1997
09-12586-99	Radioisotope license
10943	MoTH road use permit (367.3 km to 435.22 km), road use permits (RUP) MOF — Issued December 10, 1996
110454	Conditional water license Kemess Lake
110851	Conditional water license (east Kemess Creek, Serrated Creek, and south Kemess Creek.)
AR15157	Refuse and air contaminants from the construction camp — issued September 9, 1997
BCG07761	Special waste consignor identification number — issued February 18, 1998 Explosives storage and use permits
Numerous	Boiler operator certificates
C116035	Conditional water license Kemess Creek
M206	Approving work system and reclamation program
M96-03	Project approval certificate — issued April 29, 1996
No. 1168	Magazine storage for avalanche explosives and detonators - issued February 14, 1997
No. 682	The main magazine storage of explosives and detonators — issued January 21, 1998
OTH00123	Gas Permit — issued October 27, 1998, renewed annually
PE 15257	Air emissions — pending
PE 15335	Tailings storage facility and associated works, RBC, mill and accommodation site, runoff and open pit water — issued December 8, 1998. (main effluent permit)
PR14928	Refuse to the ground/active waste rock dump — issued July 29, 1997
S17447	Mine site access roads
S22850	Cheni Mine Road transfer (includes old S13088)
S24513	Power line access roads

Source: Spillway completed Fall, 2012

Figure 3.3: Photo showing spillway and related TSF reclamation.

3.4.1 Required Permits and Status

Presently, Kemess South has a reclamation & closure plan document with the BC Ministry of Energy and Mines. The reclamation & closure plan is updated every five years. During the operational phase, the primary focus of the reclamation & closure plan is reclamation. Upon mine closure, the reclamation & closure plan will focus on permanent closure. AuRico is updating its reclamation & closure plan that will lay out the closure plan and time lines for implementation. Significant reclamation works have already been completed for the Kemess South project including a final spillway for the previous tailings facility. No significant amount of reclamation and revegetation of disturbed area are expected to be required for the proposed Kemess underground.

3.5 Other Significant Factors and Risks

SRK is not aware of any other significant factors or risks associated with the proposed mine development at this site.

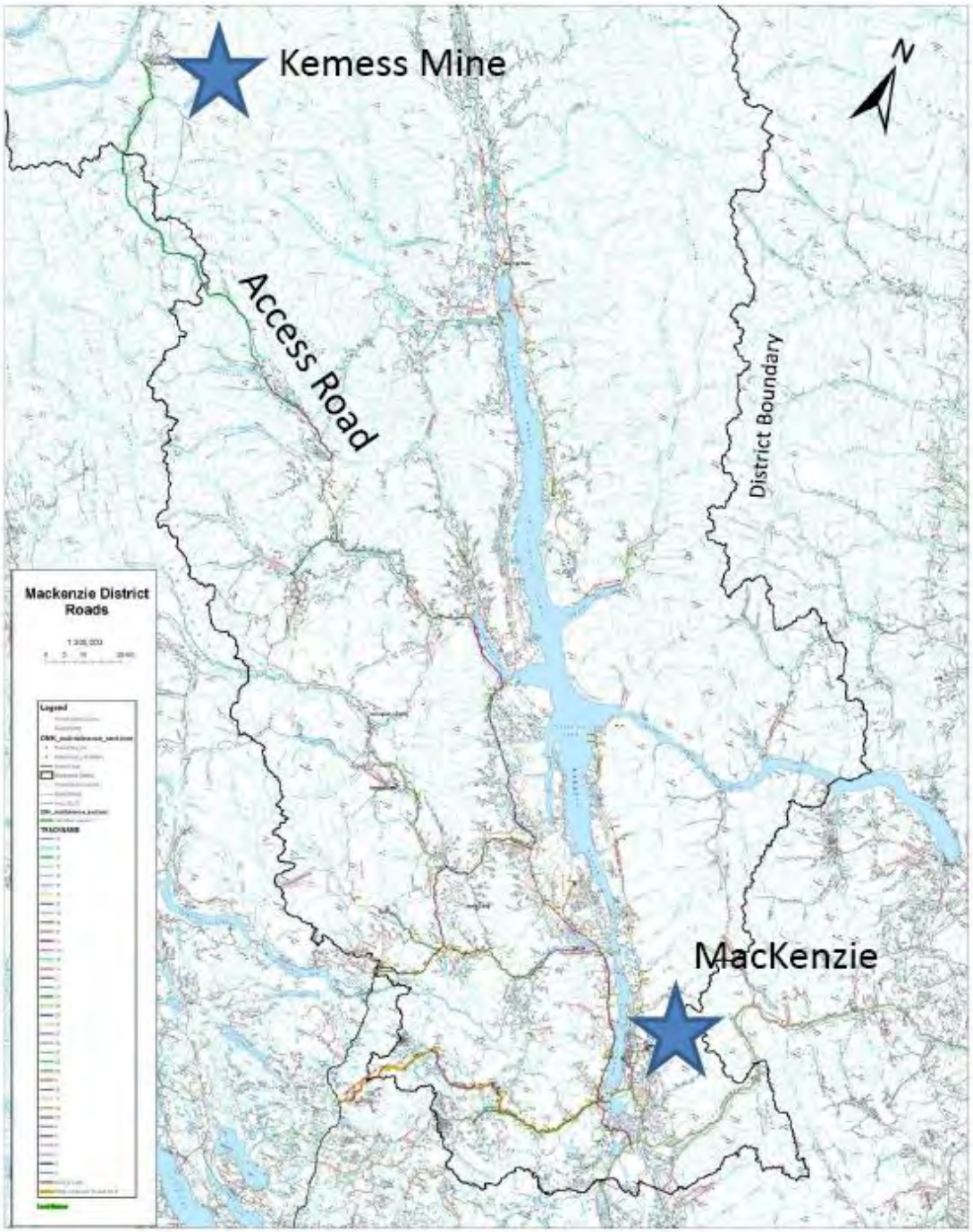
4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

4.1 Accessibility

All season road access to the Kemess property is via the Omineca resource access road from either the town of Mackenzie, or from Fort St James. Operations at Kemess South were serviced by scheduled year round flights from surrounding communities and from Vancouver.

The Kemess underground deposit lies approximately 5 km to the north of the existing Kemess concentrator and accommodation camp. 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. A natural rock glacier is evident in the western cirque. Vehicular access to the area overlying the deposit is limited to summer months only. Helicopter access is possible during winter. The processing plant and accommodation camp are located at an elevation of approximately 1,200 m and are accessible all year round.

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.



Source:http://www.for.gov.bc.ca/ftp/DMK/external/!publish/!Web/RoadsInfo/DMK_Roads_maintainance%20sections_300000_apr20_11.pdf

Figure 4.1: Road Access to Mine Site

4.2 Local Resources and Infrastructure

Existing on-site infrastructure consists of offices, maintenance facilities, a 300-person accommodation camp, mill, crushers and raw ore stockpile areas, access and service roads, airstrip, explosives depot, and tailings storage facilities.

The Kemess South mine operated on a fly-in fly-out basis with the majority of employees working on a two weeks on, two weeks off cycle, commuting from regional centres and from Vancouver.

The airstrip can accommodate most 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.

Process water is reclaimed from the tailings facility and from the Kemess South open pit.

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

AuRico currently keeps the mill and infrastructure facilities on care and maintenance pending decisions regarding the Kemess underground deposit and other potential projects in the area.

The surface rights on the Kemess property are owned in part by AuRico but are predominantly public lands owned by the Province of British Columbia. All areas of proposed activities fall either on AuRico private land or on unpatented mining claims controlled by the Province. In the latter case 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

4.3 Climate and Length of Operating Season

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. Current care and maintenance activities site access during winter months is only by fixed wing aircraft, although road access is still maintained during the non-winter months.

4.4 Physiography

Kemess mine 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 be 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. The Kemess area 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. Commonly, snow does not leave the higher elevations until late June.

Figure 4.2: Views looking south west towards Thutade Lake from Duncan Ridge.

4.5 Sufficiency of Surface Rights.

AuRico has surface rights for the area of the proposed development through the currently registered mineral leases. AuRico is of the opinion that surface rights are sufficient for all planned construction and operational activities.

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 in 2011.

5.2 Previous Development, and Production

There is no previous production from the Kemess underground area. Production from the nearby Kemess South deposit is detailed below.

5.3 Kemess South

The history of the Kemess property records earlier discoveries in the Kemess North area.

However, 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, in 2000 Northgate Exploration a predecessor company to AuRico 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	Waste Mined (t)	Ore Milled (t)	Grades		Metal produced	
				Mill Head			
				Cu %	Au g/t	Cu tonnes	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 North

Northgate consolidated the exploration information on the Kemess North 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 North are summarized in Table 5.2.

Table 5.2: Kemess North 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 8 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 trenching. A resource of 157 Mt at 0.37g/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 (7 holes on Kemess North, 4 holes on NOR1, 5 holes at Hilda, 18 holes on the Bear Claims, and 6 holes at Duncan Ridge). Hole KN-05-24 discovers Kemess North offset.
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.

Period	Company	Work completed
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 North deposit.
2011	Au Rico Gold Inc.	Completed 2,207 m of diamond drilling in 3 holes on Kemess North 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 Northgate, 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 NI43-101 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.

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-goldporphyry 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 North 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 North 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 deposit. Figure 6.1 shows the district geology, major intrusive masses, and disposition of the district's deposits.

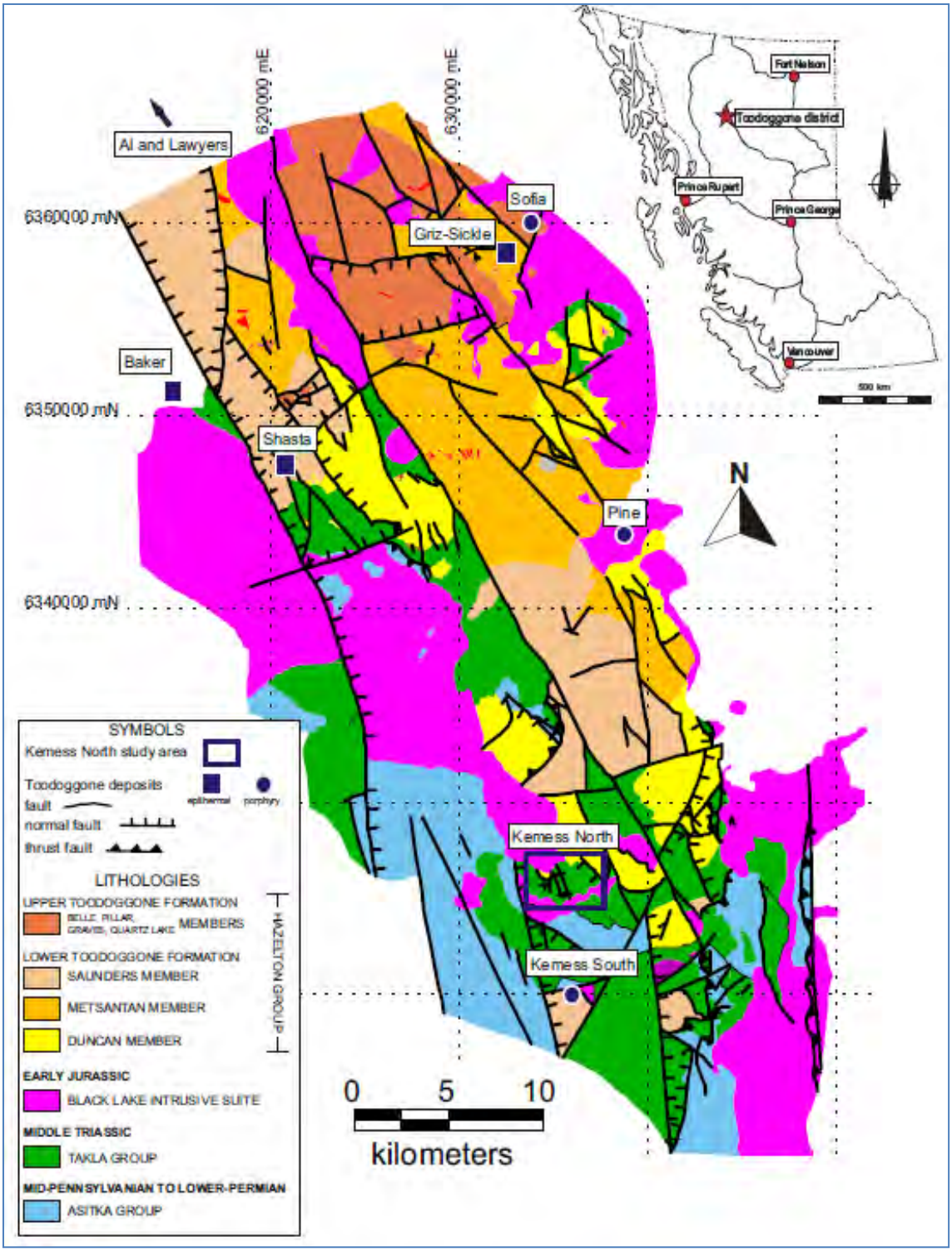


Figure 6.1: Toodoggone district geology (McKinley 2006).

6.2 Property Geology

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 a distinctive feldspar porphyritic unit (bladed feldpsar porphyry – BFP). Mantling the northern and eastern limits of the Kemess North 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 North area.

The Kemess North pluton beneath the East Cirque hosts the bulk of the copper-gold mineralization for the Kemess underground 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.

In the 2010 drilling another phase of the Black Lake intrusive suite was identified in the Kemess North area. Pre-mineral sills and dykes of hornblende porphyritic diorite are prevalent in the altered Takla volcanic rocks that overlie the Kemess North pluton.

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 crosscut 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.

6.2.1 Structure

The most prominent structure traversing the Kemess underground project area is the Kemess North (KN) Fault, an east-west trending south dipping reverse fault that truncates the Kemess North pluton and associated mineralization at depth. The KN Fault is complex in that there is a related north dipping secondary structure that demonstrates apparent normal offset with at least 1000 m of southside up apparent displacement. At least three steeply dipping, northwest trending normal faults have been inferred from surface mapping and drilling, transect the Kemess North property. Fault spacing ranges from 500 to 1,500 m.

6.2.2 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 contents 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.

6.2.3 Mineralization

Gold-copper mineralization forms an inclined tabular zone that is centred on the East Cirque porphyritic monzodiorite, which from 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 (10660E to 10180E). Between 10260E and 10160E 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 that 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 6.2 shows the mapping of surface alteration in the deposit area.

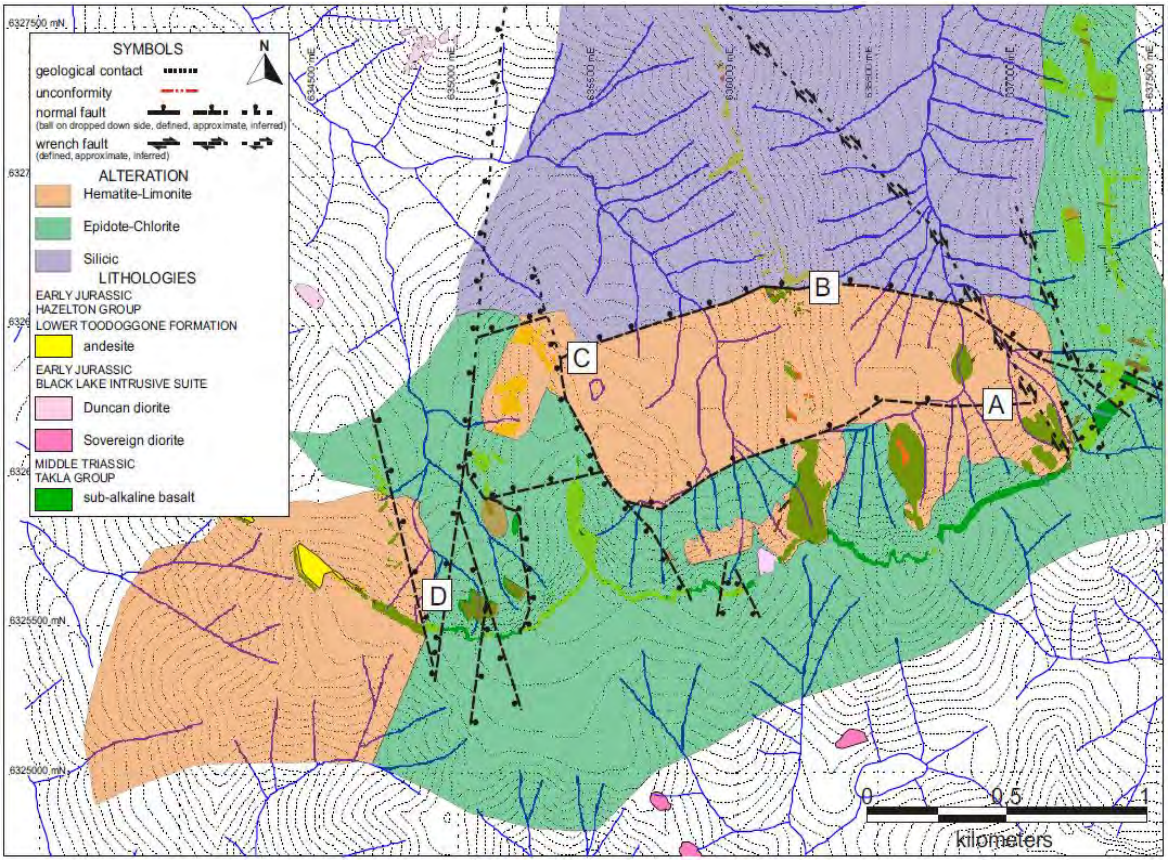


Figure 6.2: Surface alteration mapping at Kemess North (McKinley 2006).

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 low temperature phenomena. McKinley in 2006 completed a useful paragenetic study of the vein-types which is summarized in Figure 6.3. This vein classification scheme was adopted by exploration and has been in use since 2010.

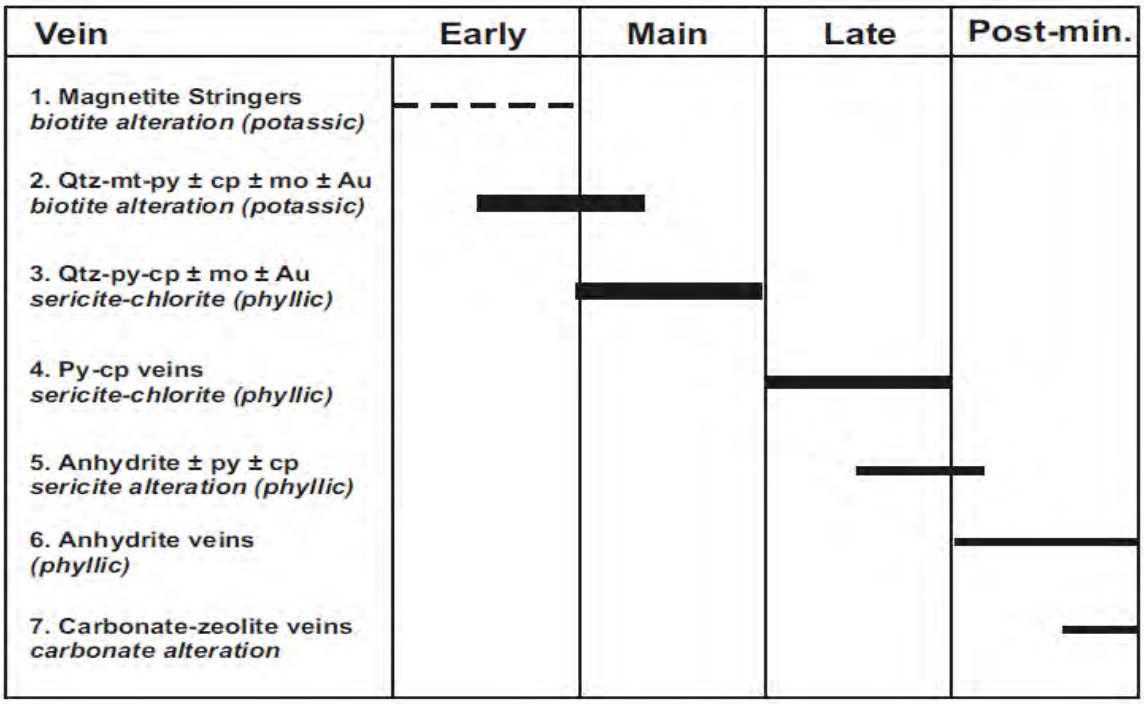


Figure 6.3: Vein Paragenisis at Kemess North (McKinley 2006).

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 stock work and quartz magnetite replacements. Gold and copper grades variably diminish outward into the hanging wall 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.

7 Deposit Types

Kemess North is a large 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 North 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 better 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.

8 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 North area since 2001.

Table 8.1: Drill campaign statistics at Kemess North

Year	Holes	Metres	Year	Holes	Metres
1975	5	589	2002	59	33,338
1976	8	1,477	2003	53	20,479
1989	5	732	2004	23	7,125
1990	12	2,204	2005	12	7,398
1991	27	7,059	2006	17	8,632
1992	25	4,520	2007	27	17,713
2000	12	4,269	2010	30	16,439
2001	16	8,386	**Total**	**331**	**140,933**

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 soil or rock sampling or trenching at Kemess North. Surface work has been confined to access road 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 North project.

The procedures followed in the field and through the interpretation stage of exploration have been professional. 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: Exploration employees/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

Job function	Supervisor	Contractors
		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 2011		Driftwood Hy-Tech

9 Drilling

Since May 2000, there have been ten summer drill programs completed in the Kemess North 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 drill holes 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. Because the 2011 program was completed for confirmatory geotechnical purposes and does not impact the resource work it will be treated separately.

The Kemess underground project (KUG) 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 drill hole spacing is less than 100 m and became quite well covered with the additional 2010 and 2011 holes.

The current KUG resource database contains 146 drill holes for a total of 67,157 m and an average length of 460 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 property. Historically, drilling an HQ diameter hole (64 mm core) to act as a casing for NQ (48 mm core), which usually was used to complete the hole, solved 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 Procedures

The procedures used to locate exploration drill holes 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 dug. The orientation of the drill hole 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 drill holes 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 property 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.

9.1.1 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 minerlized 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 drill holes. The mineralization is generally confined to three main lithologies, hypogene 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.

9.1.2 Collar Survey

Survey control for the drill hole 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 drill holes completed to confirm the geotechnical properties of Kemess North were verified later on by GPS using a base station with sub-centimeter accuracy.

9.1.3 Down Hole 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 drill holes.

The 2010 program relied upon non-magnetic methods such as Flex-It electronic gyro and Deviflex to survey the down-hole traces. All 2010 drill holes 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: Historic holes with adjusted trajectories.

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. A plan showing the drill hole collars and traces in the Kemess Underground area is shown in Figure 9.1.

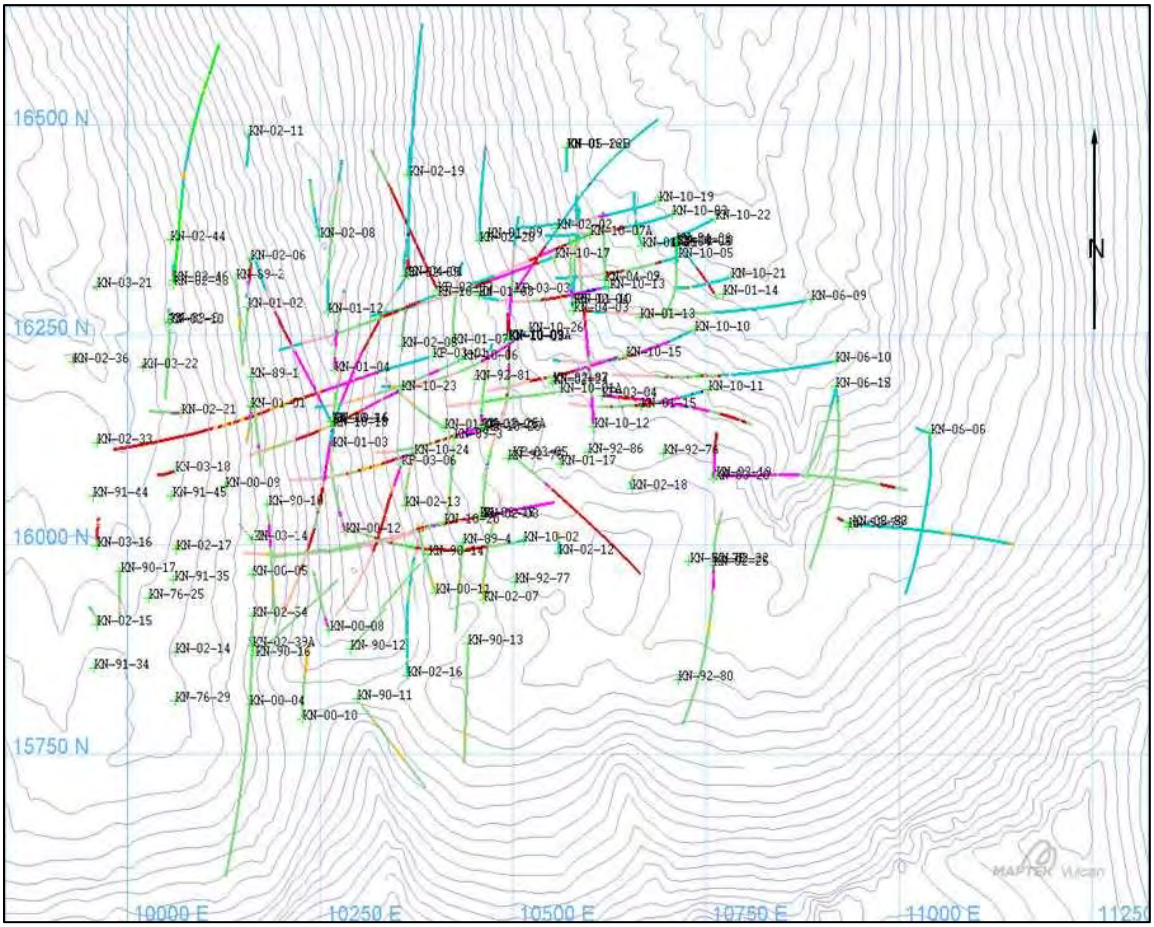


Figure 9.1: Drill hole locations — 146 holes in the Kemess underground area.

10 Sample Preparation, Analyses, and Security

10.1 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 and forms the basis of the mineral resource. More than 75% of the assays have been completed since 2001.

Samples from the Kemess North 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 at site insertion of blind duplicate, blank, and standard samples.

Sample intervals were determined by a staff geologist according to lithology, and ranged from 0.3 to 2.0 m, with the average length of samples being 1.92 m. Because of the low-grade nature of the mineralization and difficulty determining potential ore from non-ore material, the entire drill hole is sampled. Once in a uniform rock type, sample spacing was generally 2.0 m. The maximum 2.0 m sample length was seleceted 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, foremely ALS Chemex (ALS). For the 2003 program, a sample-bucking facility was built near the camp area and run under supervision of Kemess's Chief Assayer during the subsequent programs.

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. In 2008 a pulverizer was added to the preparation process at Kemess and pulps were submitted for analysis for both 2008 and 2010 programs.

At ALS, the –10 mesh samples (pre-2008) were pulverized to 85% passing 75 µm (PUL-22) prior to assay. 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, 32,506 samples comprise the entire Kemess underground database. Since 2000, 28,498 samples have been submitted to ALS for copper and gold analyses which represent more than 87% of all the analyses. Since 2003, silver has been

analysed for by multi-element ICP package provided by ALS. There are 16,016 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 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 Bulk Density Data

A great deal of effort has gone into the determination of the bulk density (BD) for the deposit. In total, more than 9,950 measurements have been performed and a mean BD of 2.75 was determined. 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 drilling samples had SG determined by air and water methods performed while logging.

For resource purposes, the drill hole BD was composited to 6 m lengths. The length-weighted average BD were compared by domain before and after compositing, and there was insignificant change. Table 10.1 summarizes the BD data by resource estimation domain on a length-weighted basis.

Table 10.1:Bulk density determinations by resource estimation domain.

Domain	Sample type	Number of samples	Minimum (g/cm^3)	Maximum (g/cm^3)	Mean (g/cm^3)	Std dev	Variance	Coefficient of variation
Domain 1								
Raw data	Core	1,056	2.04	3.23	2.69	0.14	0.02	0.05
6 m composites	Core	477	2.06	3.04	2.67	0.14	0.21	0.54
Domain 2								
Raw data	Core	3,946	1.88	3.74	2.79	0.15	0.22	0.05
6 m composites	Core	1,792	1.88	3.63	2.76	0.16	0.03	0.06
Domain 3								
Raw data	Core	2,035	1.40	3.48	2.75	0.14	0.02	0.05
6 m composites	Core	704	1.40	3.08	2.74	0.14	0.02	0.05
Domain 4								
Raw data	Core	1,035	2.50	3.28	2.80	0.12	0.02	0.04
6 m composites	Core	354	2.51	3.10	2.80	0.10	0.01	0.04
Domain 6								
Raw data	Core	210	1.17	2.86	2.62	0.14	0.02	0.05
6 m composites	Core	119	1.17	2.76	2.59	0.18	0.03	0.07
Domain 7								
Raw data	Core	1,683	2.26	3.12	2.69	0.10	0.01	0.04
6 m composites	Core	934	2.26	3.12	2.68	0.11	0.01	0.04

Domain 6 corresponds to post-mineral intrusive rocks (syenite) and a default value of 2.59 was used here.

10.4 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 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, 1,536 samples were submitted for quality control purposes as blind blanks, standards, or duplicates. This represents approximately one in every 26 samples, or 3.9% of the samples collected 2002 through 2011 from all areas of exploration drilling on the Kemess property. 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.

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).

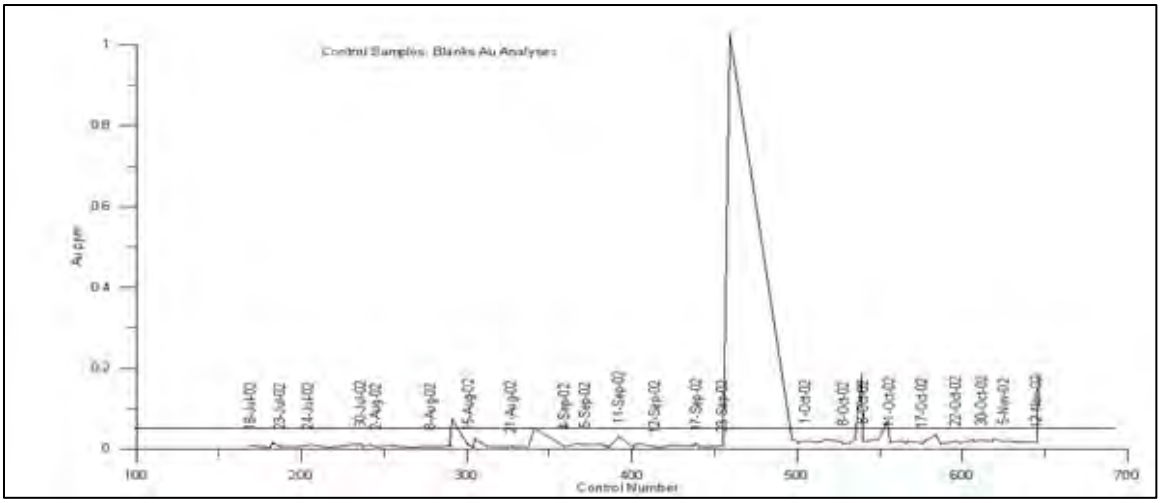


Figure 10.1: 2002 blanks over time – gold.

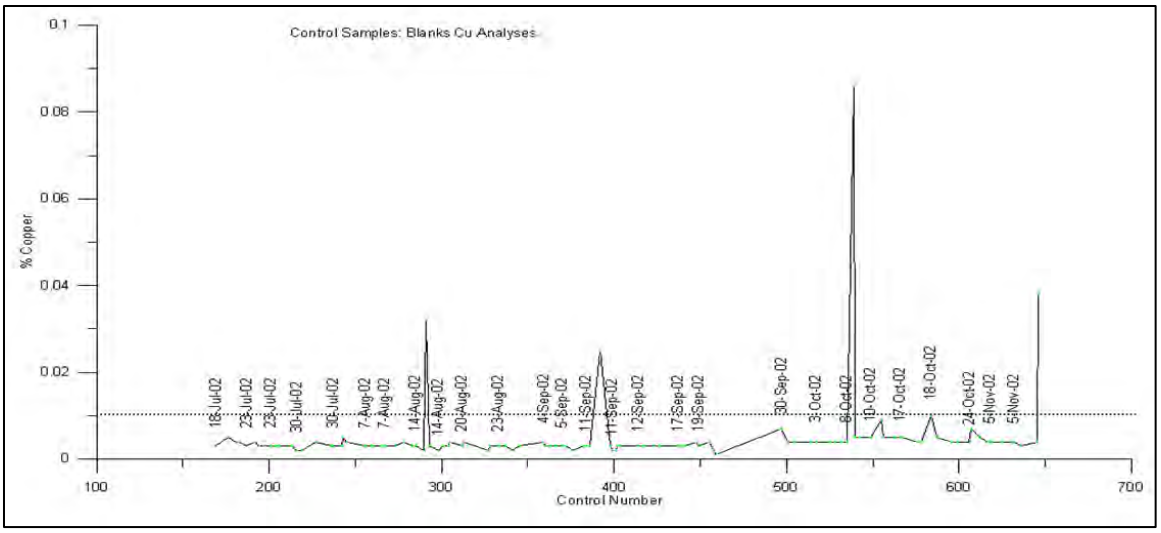


Figure 10.2: 2002 blanks over time – copper.

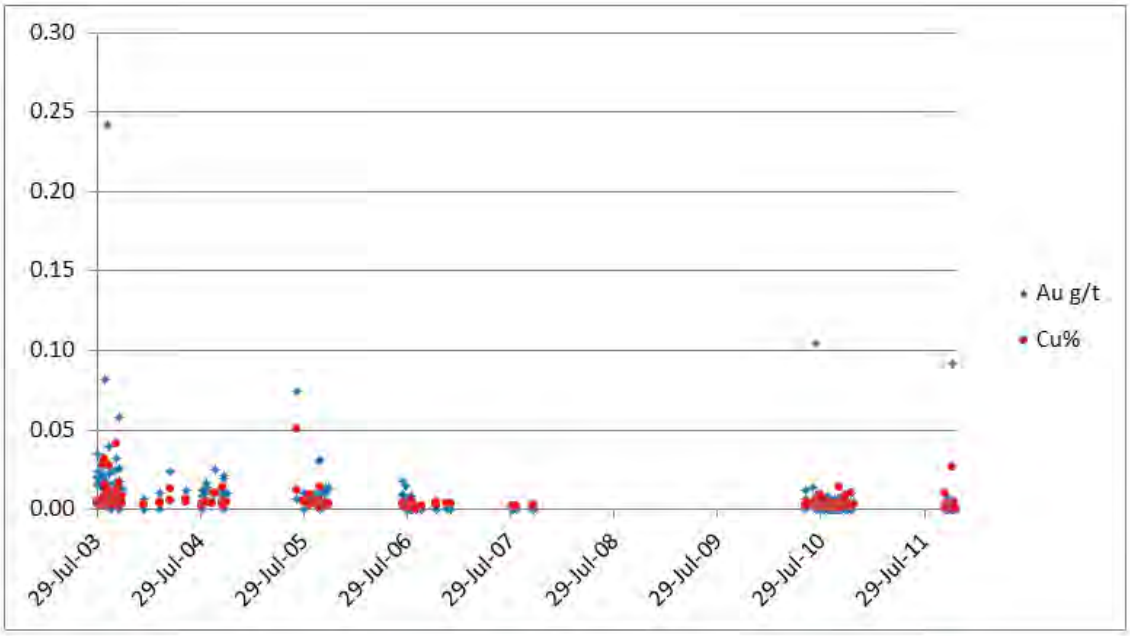


Figure 10.3: Blanks over time 2003–2011

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 norrow 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 blank copper outliers are typically coincident with these gold outliers, which support the conclusion that this is a natural, in-situ, geologic variation. The few outliers, not exhibiting these characteristics, do not represent any significant or systematic contamination.

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 10.2) 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 standards are presented in Figure 10.4 to Figure 10.6.

Table 10.2: 2002–2011 certified RockLabs standards.

Standard	Mean Au ppm	SD Au ppm	Lower Au limit	Upper Au limit
GTS-2	0.26	0.01	0.23	0.29
OX-5	0.97	0.033	0.87	1.07
OXD-27	0.416	0.05	0.265	0.567
OXE-21	0.651	0.026	0.572	0.73
OXH-29	1.298	0.03	1.207	1.389
S-2	1.535	0.054	1.373	1.697
SE-19	0.583	0.026	0.504	0.662
SE-29	0.597	0.016	0.548	0.646
SE-44	0.606	0.006	0.578	0.634
SE-58	0.607	0.019	0.55	0.67
SF-12	0.819	0.024	0.746	0.892
SH-24	1.326	0.043	1.196	1.456
SH-35	1.323	0.044	1.189	1.461
SH-55	1.375	0.045	1.234	1.51

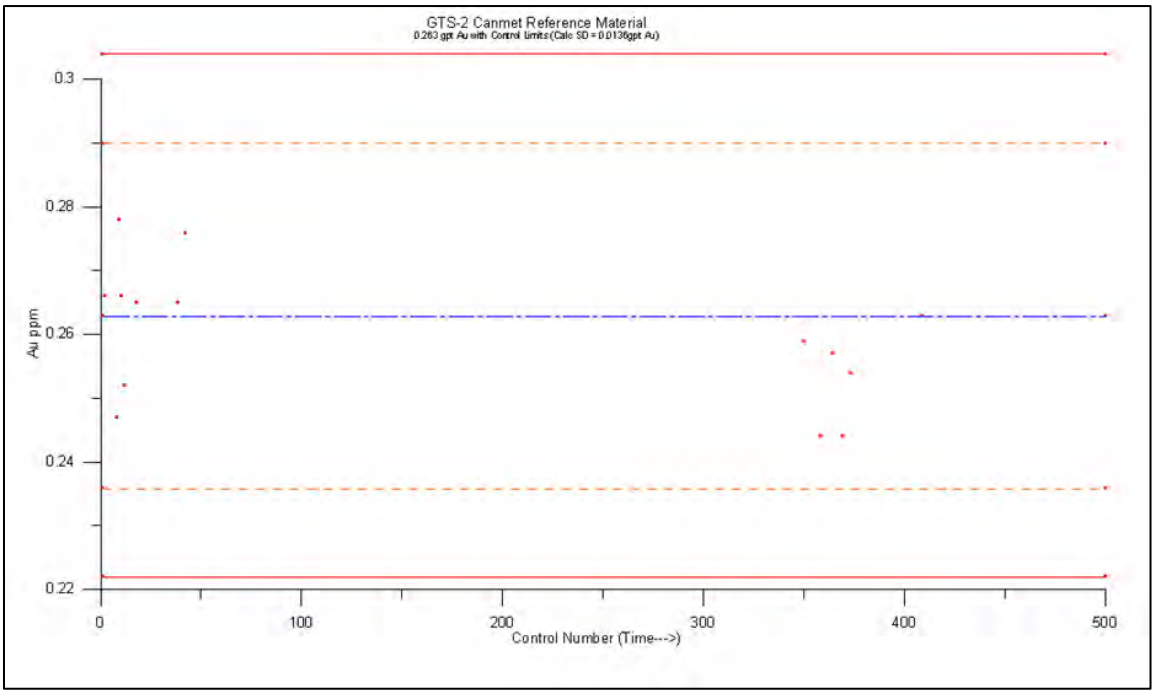


Figure 10.4: Standard GTS-2 over time: – gold (2002).

Note: Blue line is the expected value for the standard, orange dash lines represent 2 standard deviations from the expected value and the red lines indicate a 3 standard deviation from the expected value.

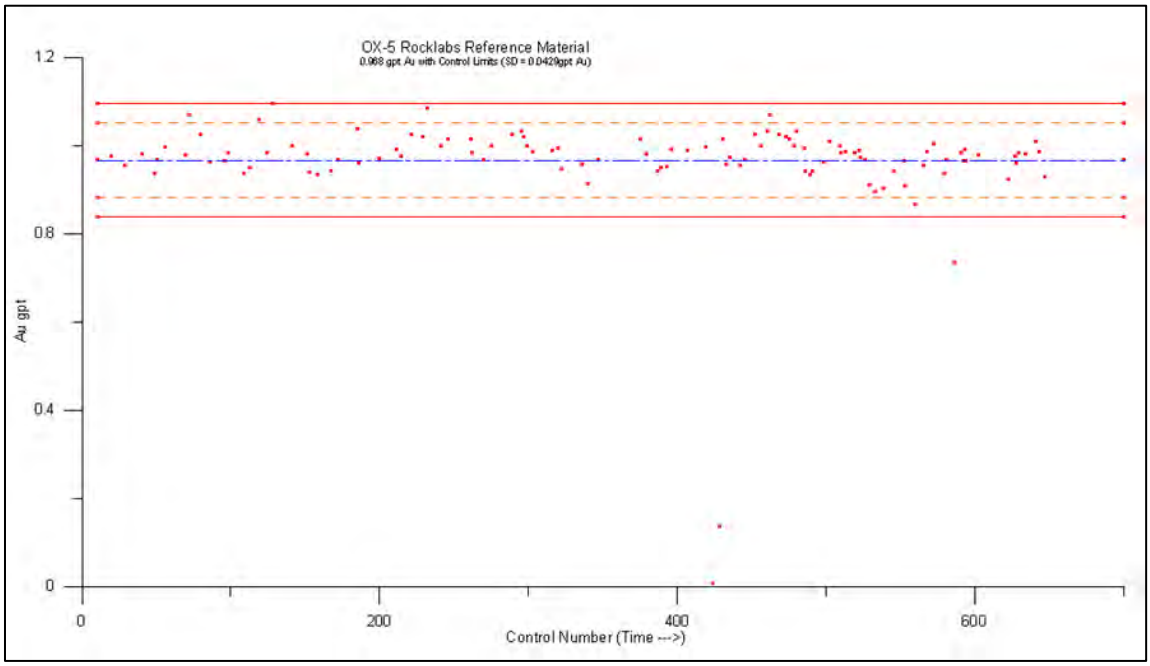


Figure 10.5: Standard OX-5-2 over time – gold (2002).

Note: Blue line is the expected value for the standard, orange dash lines represent 2 standard deviations from the expected value and the red lines indicate a 3 standard deviation from the expected value

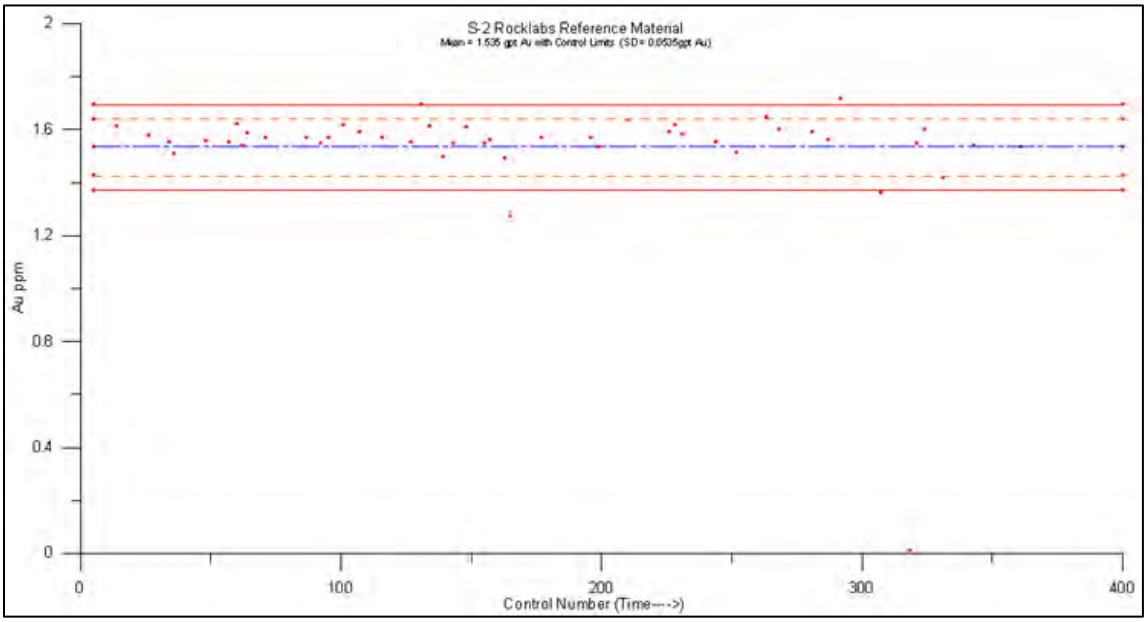


Figure 10.6: Standard S-2 over time – gold (2002).

Note: Blue line is the expected value for the standard, orange dash lines represent 2 standard deviations from the expected value and the red lines indicate a 3 standard deviation from the expected value

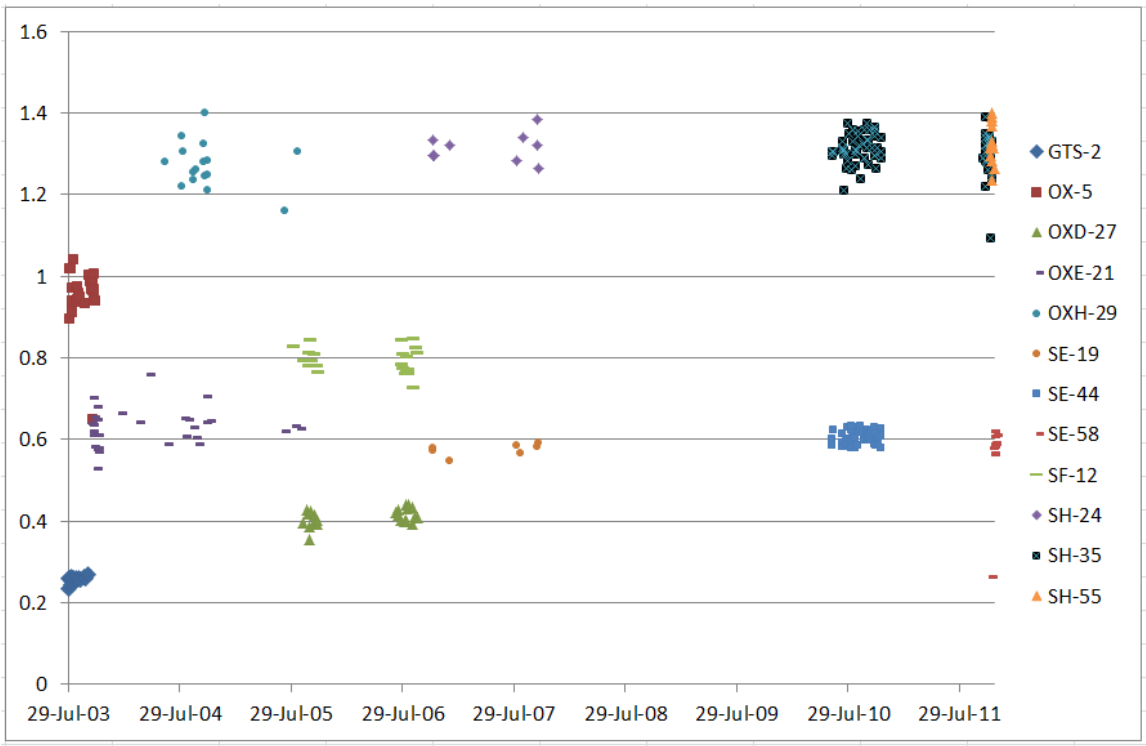


Figure 10.7: Standards over time – gold (g/t) (2003–2011).

Although not intended as copper standards, all standards were analyzed for copper and monitored along with the gold results, as shown in Figure 10.8.

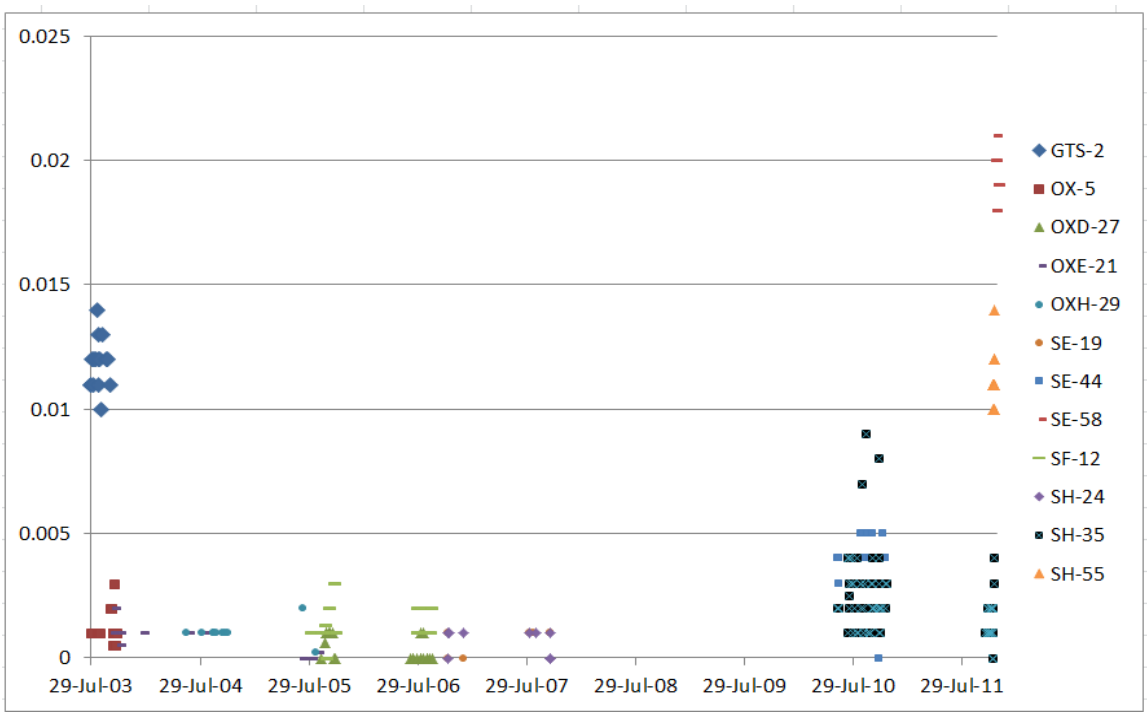


Figure 10.8: Standards over time – copper (%) (2003–2011).

Based on the performance of the standards, over the entire course of the 2002–2011 programs, and laboratory investigations of outliers, all mainstream gold and copper assay results associated with these standards are considered accurate.

Combined preparation and analytical precision was examined using matched pairs created by taking a second 250 g riffle-split, referred to as a ~~r~~eject-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. Typical examples of scatter plots of matched-pair assays, in this case from the 2010 program, are presented in Figure 10.9.

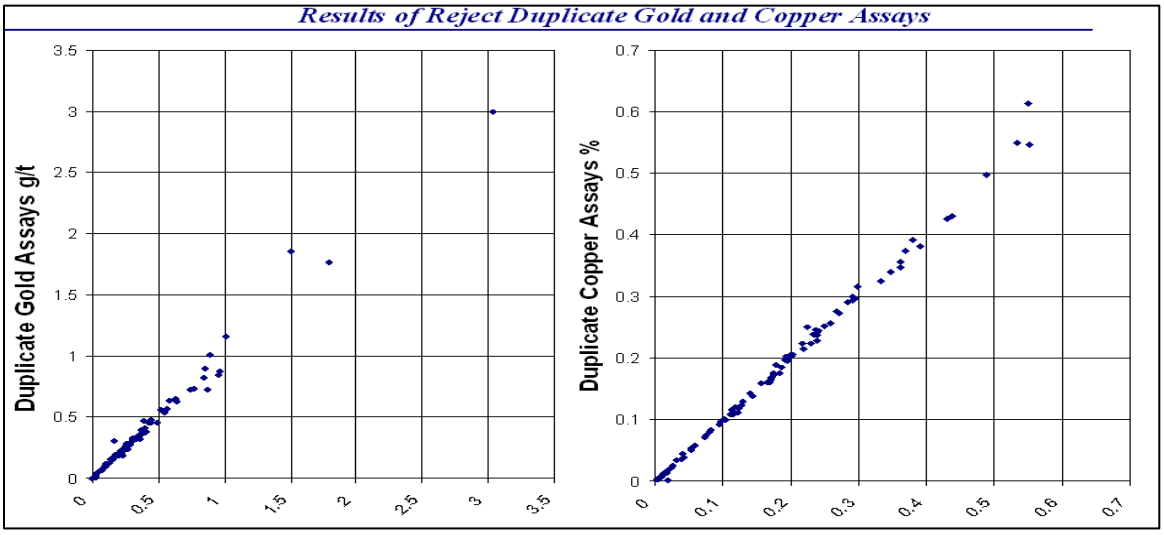


Figure 10.9: 2010 matched pair results for gold and copper reject duplicates.

Outliers, defined as those matched pairs with a grade differences over 0.1 g/t gold, or 0.1 % copper, and greater than 25% precision, were identified and investigated. Comparisons between ICP signatures of each of the outlier matched pairs, as well as surrounding samples, were used to determine if these were erroneous data points or valid outliers representing real nugget effects. Errors were resolved and invalid outliers were removed before precision analysis was performed.

Absolute precision was calculated using the Thompson-Howarth regression analysis, represented by the red lines on Figure 10.10 and Figure 10.11.

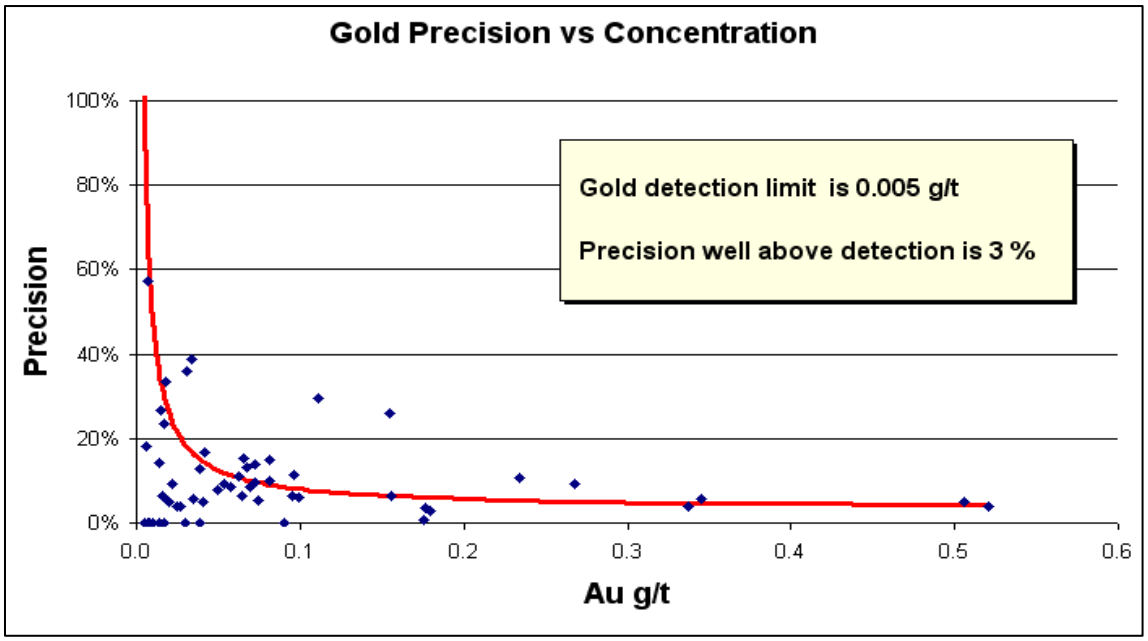


Figure 10.10: 2007 gold precision vs. concentration plot.

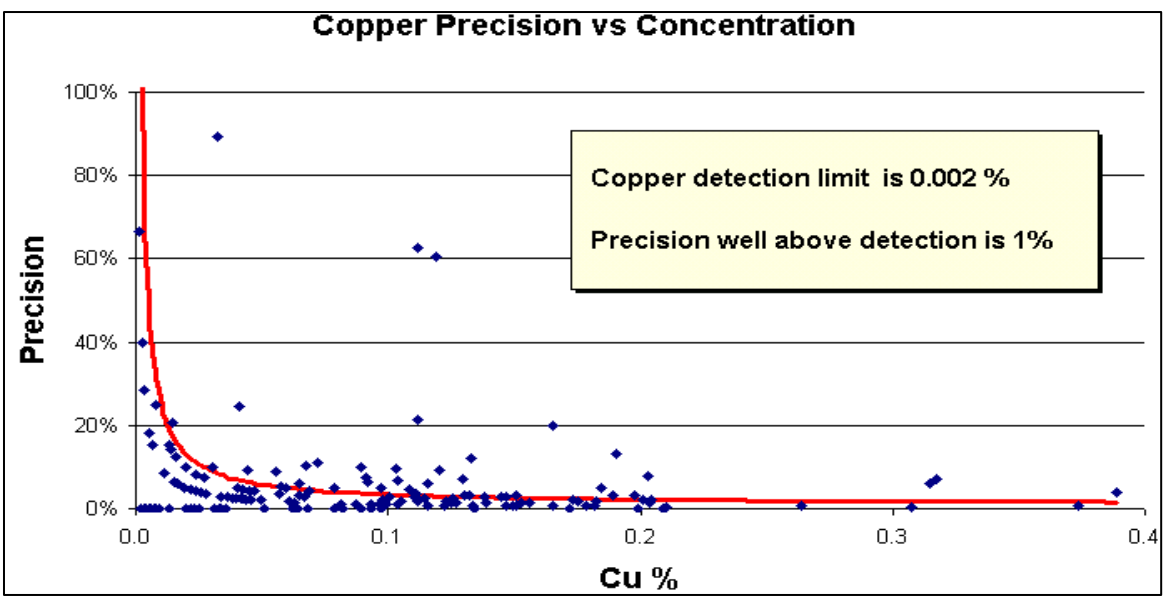


Figure 10.11: 2003 copper precision vs. concentration plot.

Precision analysis results for the 2002 through 2010 programs, from all areas of exploration drilling on the Kemess Property, are summarized in Table 10.3. Very good levels of precision were achieved consistently over the course of these drilling programs. Precision levels from all programs are comparable.

Table 10.3: Preparation and analysis precision summary.

Year	Total samples	Duplicates	Au precision @ 0.5 g/t	Cu precision @ 0.4 %
2002	16,342	340	7%	3%
2003	11,918	199	7%	1%
2004	7,302	165	5%	3%
			Au precision @ 0.4 g/t	**Cu precision @ 0.2 %**
2005	6,188	123	9%	3%
2006	5,664	78	7%	3%
2007	8,086	67	4%	2%
2010	7,473	90	6%	2%

Similar precision analysis of 225 reject-duplicates, from 2003 through 2011 and restricted to the Kemess North underground block cave area, yielded gold precision of 8% at 0.4 g/t and copper precision of 2% at a concentration 0.2% (Figure 10.12 and Figure 10.13).

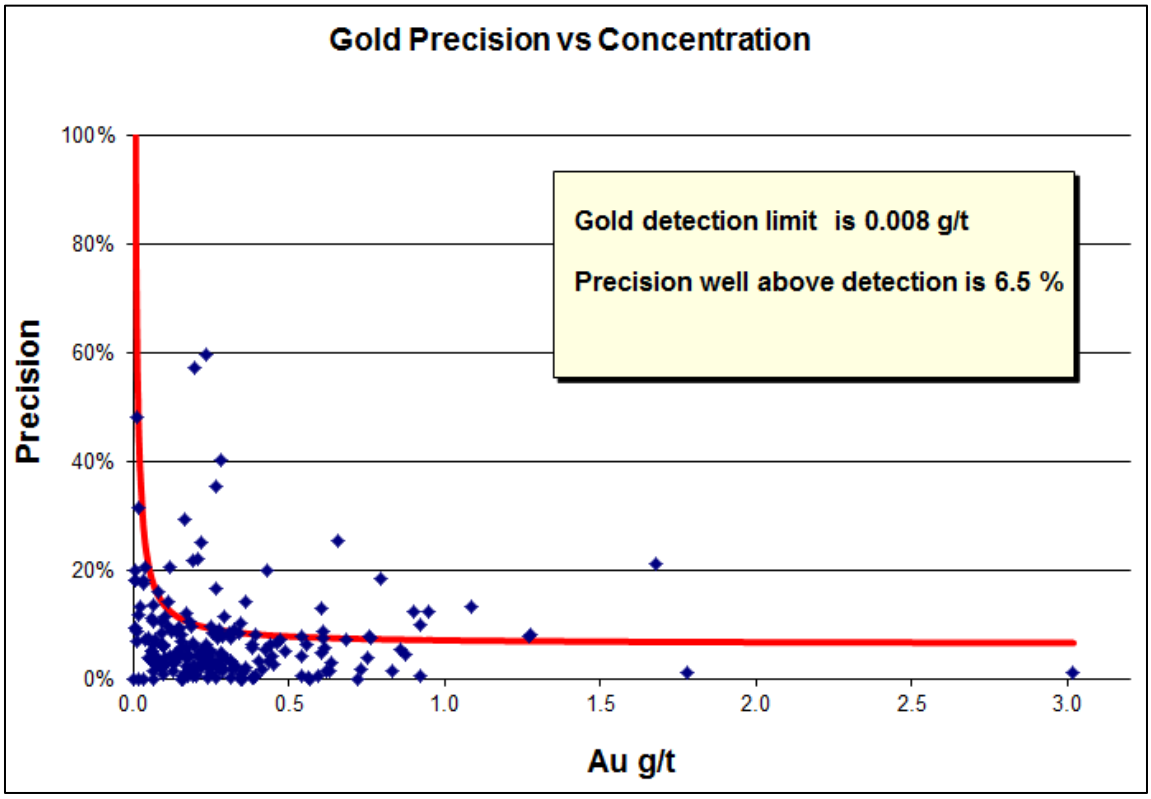


Figure 10.12: 2003–2011 Kemess North underground gold precision vs. concentration plot.

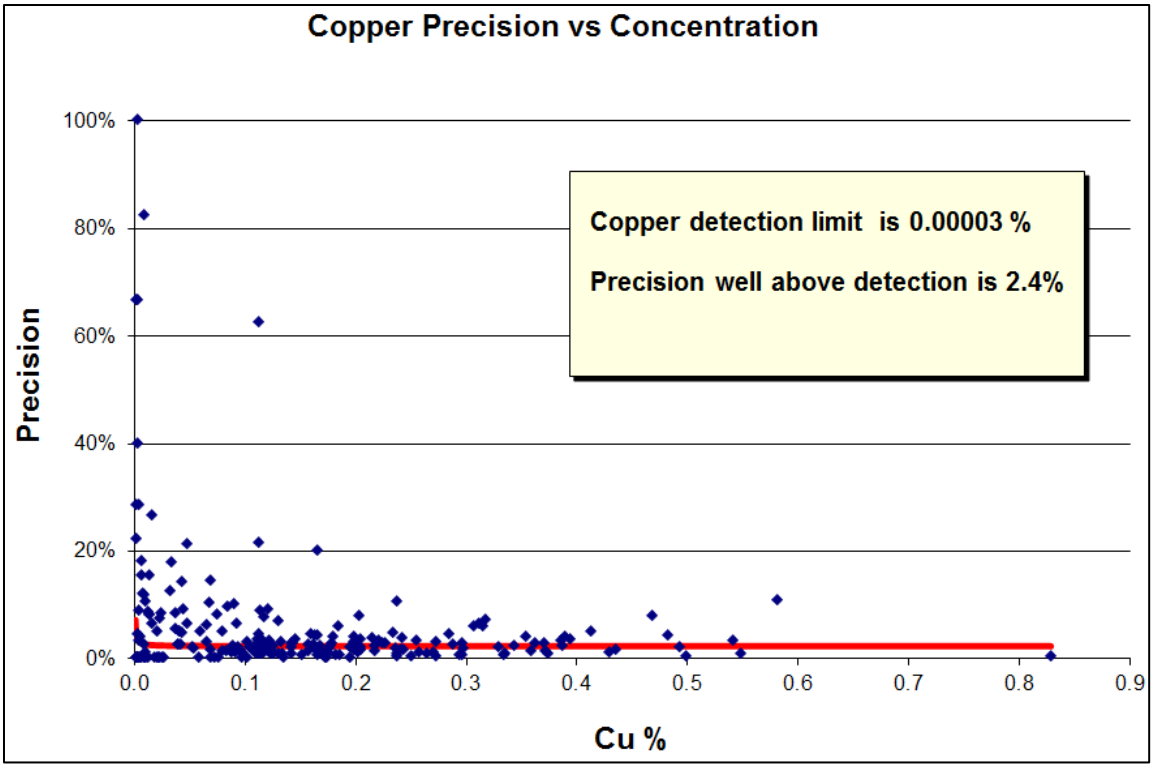


Figure 10.13: 2003–2011 Kemess North underground copper precision vs. concentration plot.

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

11 Data Verification

There have been numerous data verification programs on the Kemess North open pit project, and the reader is referred to Section 13 of both the June 2004 and May 2005 technical report for background detail.

In late June 2011, the Northgate Exploration Group completed a 5% audit of the Kemess underground resource database to verify that analytical results have been entered correctly into the drill hole database used to prepare the February 2011 mineral resource estimate. The audit process and results are summarized in the following sections.

11.1 Method

Samples were selected for verification by first identifying those samples within the 2011 resource solid, and then by flagging 1 in 20 of those analyses for cross reference against their assay certificates via sample number. In total, 493 analyses were verified, with 490 analyses verified against the original assay certificates and three analyses dating from the 1992 drilling compared against the original databases.

11.2 Results

With the exception of the following data verification issues copper, gold, and silver results have been correctly entered. In one instance it was noted that copper analysis below the detection limit for 2001 drilling had an incorrect default value (0.002% should have been entered as 0.001% – KN-01-07 595.6–597.6 m) in post-mineral dyke material. The corresponding gold value had been entered correctly.

Further verification showed that many of the 2001 analyses lacked tag numbers in the database, resulting from a choice by the project managers at the time. This has been known since 2002, and the protocol is that when the tag information is discovered it is entered and the database updated.

Investigation of the detection limit issue provided more information regarding the missing tags. Importing the remaining tag numbers and comparing against the 2011 resource database yielded the following minor items with only two real errors where metal contents was a concerned:

- One key punch error in drill hole KN-01-10 at 272.28–274.00 m, sample #13210, 0.182% should be 0.155% Cu and 0.254 g/t should be 0.215 g/t Au. This error was caused by a carry down of the results for #13209 when updating re-logged diorite intervals. The error was corrected and is not considered material.
- There is an interval error in drill hole KN-01-13 at 586.4 m which caused a shuffling down by one record (2 m) of nine samples. It appears to have been a compositing artifact or error that has escaped detection previously. An artificial, but invalid interval was created resulting in moving values a further 2 m down the hole. The error was corrected with no material impact as it lies below the mineral resource boundary.

Four copper detection limit issues were identified, all in hole KN-01-07, 0.002% were corrected to 0.001% (an additional nine 2001 below detection limit of 0.002% in KN-01-07; were corrected).

Six gold detection limit differences in hole KN-01-04, 0.003% vs 0.0025% (database is correct).

Eleven gold detection limit differences in KN-01-06, 0.003% vs 0.0025% (database is correct).

11.3 Summary

The 5% audit showed no significant errors from the resource area regarding the recording of tabulated analytical data. The analytical database for the 2011 resource was verified and can be relied upon for resource estimation.

11.4 Verifications by AuRico – 2011 Drilling

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 drillholes 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 encountered in 2011 drilling.

Hole ID	Hole ID	Hole ID	
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 drill hole 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: Analytical results for 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.

11.5 Verifications by SRK

11.5.1 Site Visit

A site visit was conducted from May 16-19, 2011 by Jarek Jakubec, and Ryan Campbell both form SRK. They conducted a high level independent geotechnical review of selected core, and available reports. They checked the existing geotechnical drillhole database for consistencies and generally got familiar with the project.

11.5.2 Independent Verification Sampling

No independent sampling was done by SRK because previous independent sampling along with metallurgical test results easily confirmed that the deposit containted copper and gold mineralization in concentration levels reported by Aurico.

SRK is of the opinion that the data provided by AuRico along with the data quality assuarance and quality control (QA/QC) protocols establised by AuRico are of adequate quality and quantity to support the estimation of mineral resources and mineral resreves.

12 Mineral Processing and Metallurgical Testing

12.1 Summary

In January 2005 Northgate Exploration Ltd. completed a feasibility study for their Kemess North Project. The Kemess North deposit was to be mined as an open pit with primary crushing facilities located near the pit and the ore conveyed to the existing mill facilities that were to be expanded with the addition of a new grinding line. The metallurgical characteristics for the Kemess North ore showed through test programs to be very similar to the Kemess South hypogene ore such that minimal modifications were expected to the Kemess South mill process flowsheet. A metallurgy report for the Kemess North deposit was prepared by Northgate in 2003., In 2011, additional metallurgical work was initiated by Northgate (AuRico) at G&T Metallurgical Services Ltd.'s Kamloops BC lab to support the KUG project and verify the previous observations. With the underground block cave mine, the ore will now be developed from the bottom levels up, the opposite of the original open pit feasibility work. Additional drill core sampling was completed in 2012 with samples delivered to G&T for determination of grinding parameters. The metallurgical section has been extracted from the following reports:

- Kemess North Preliminary Metallurgical Testing, Amtel, June 2001;
- Characterization of Copper and Gold occurrences in Composites of the Kemess North Deposit, Amtel, July 2002;
- Gold Deportment in Copper Cleaner Scavenger Tailings, Amtel May 2003;
- Summary Report on Process Development Treatment of Ores from the Kemess North Deposit, Klaus Konigsmann P.Eng., May 2003;
- An Update of Metallurgical Testing of Kemess Ores, Lakefield, May 2003;
- Feasibility Study, Kemess North Project, Hatch Engineering, January 2005;
- The Grindability Characteristics of Samples from the Kemess Mine and Grinding Circuit Evaluation, (East Zone), SGS Lakefield, August 2008;
- Kemess Site Visit Report, KWM Consulting, May 2009;
- Concentrate Grade and Regrind Size Improvement on the East Pit Hypogene Ore at Kemess Mine, Powerpoint report, December 2010;
- Kemess Underground, SMC Test Report, March 2011;
- Kemess Underground PEA Report, AMC, July 2011;
- Metallurgical Assessment of Five Composite Samples from the Kemess Underground Project, G&T Metallurgical (KM2911), September 2011; and

The ores of KUG share a number of favourable characteristics with the hypogene ores of the Kemess South deposit:

- Both deposits carry -clean" sulphides without surface oxidation;
- Impurity elements will occur in extremely low concentrations;

- The sulphides will be coarse grained and are adequately liberated for rougher flotation at a grind of P80 at 150 µm;
- Average ball mill work indices will be low compared to a majority of porphyry deposits;
- The samples of Kemess North used in the 2003 program averaged 13.8 kWh/t. Ball mill work indices for Kemess South ranged from 13.8 to 15.0 kWh/t. In the recent 2012 G&T program, the average work indexes for the monzonite ores was 14.1 kWh/t and for the Takla ores was 15.3 kWh/t. Waste that could infiltrate the block cave draw down from the Toodoggone and Hazelton was hard averaging 20.3 kWh/t for the two samples tested. The JK analysis of the 2012 samples indicated that the monzonite samples tested primarily in the medium and moderate hardness classification where the Takla tested primarily in the moderate to hard classification;
- The average metal content for the KUG project will be expected to have a higher copper grade, at 0.28% Cu versus 0.22% Cu. Gold grades are expected to be lower at 0.56 g/t Au versus 0.75 g/t for South ores;
- The pyrite content of KUG will average 4% compared to 1% pyrite for the south ores. Since pyrite contains finely disseminated gold, pyrite rejection in flotation carries a larger portion of gold to tailings;
- Rougher concentrate mass pull is estimated to be 23%;
- The rougher flotation with KUG ores will be a bulk sulphide float. Rougher concentrates have to be re-ground to a P80 of ≤25 µm for cleaner flotation to produce quality concentrates;
- Existing flotation cell capacity and de-watering equipment will be adequate for treatment of the KUG ores;
- Concentrates will be free of deleterious impurities;
- The ultimate settling density of KUG tailings will be at least 66% solids at a pulp density of 1.74 g/cm^3; and
- The Net Neutralization Potential for all KUG composites will be negative, averaging –50 t $CaCO_3$ per 1000 t of tailings. The tailings will be acid generating unless stored under water. Note that the KUG feasibility study contemplates the KUG tailings being stored under water within the existing and permitted Kemess South open pit tailings facility.

The results of the 2003 cycle tests with seven composites of different Kemess North ore zones led to the following projection of plant performance for the KUG project (Table 12.1).

Table 12.1: 2003 projected plant metallurgy for Kemess North open pit ore.

Ore type	Copper concentrate grade Cu (%)	Copper recovery Cu (%)	Gold recovery Au (%)	Silver recovery Ag (%)	Ag/Au (in concentrate)
Middle zone ores	22–24	86	50–60	40–45	3.0
Lower zone ores	24–26	92	60–70	50–55	3.0

The most recent test work, 2011, completed at G&T Metallurgical Services Ltd. located in Kamloops, BC, resulted in rougher recoveries for the 34 flotation tests that were completed at 93.8% for copper and 90.2% for gold. These results are similar to the rougher results observed in the 2003 test program (Table 12.2). The cleaner results were dependent on feed grade and ore zone.

In Table 12.3, the locked cycle test results from the 2003 metallurgical program and the 2011 G&T test program are compared. The 2003 samples represent the deep level Kemess North open pit ore. The blended sample in the G&T test-work has been blended from the ore samples that represent the first eight years of the operation of the underground block cave project.

Table 12.4 provides a summary of the locked cycle tests completed in 2011 on a range of sample composites. This set of tests provided an opportunity to look at the influence of finer re-grinding of the rougher concentrate.

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

- About 91% of the feed copper and 73% of the feed gold in the blend composite were recovered to a final concentrate containing about 24% Cu and 41 g/t Au;
- The final copper recovery for cycles IV and V in the first series of 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%;
- In the second set of locked cycle tests, after adjustments for reagents or target re-grind 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 the gold losses were shown to occur in the cleaner scavenger tailings and were attributed to gold particles interlocked with rejected pyrite or as very 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 equal to 22% copper,
- Copper recovery equal to 91%, and
- Gold recovery equal to 72%.

Table 12.2: Test results from the 2011 G&T metallurgical test work program (open circuit tests).

Comp	# of samples	# of tests	Material	Average grades		Ratio	Rougher recovery			Cleaner recover (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	1.4	86.2	74.4
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: Locked cycle test comparison.

Sample		Head grades		Cu concentrate grades		Concentrate recoveries (%)	
Description	**Location**	**Cu (%)**	**Au (g/t)**	**Cu (%)**	**Au (g/t)**	**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 metallurgical test work.

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	54.9	93.9	71.4
		144	20	0.66	1.81	23.3	50.3	94.5	73.5
CP2	Medium grade monzonite	134	18	0.39	0.78	20.7	37.9	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	31.1	89.4	71.9
CP4	Mixed lithology	154	19	0.23	0.43	21.2	31.1	90.6	72.3
		154	16	0.23	0.42	24.8	36.4	89.7	75.9
CP5	Upper volcanics	127	21	0.25	0.44	21.9	24.7	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	40.5	91.3	73.2

The results of the locked cycle tests completed in 2011 on the composites for the individual ore domains supported this conclusion. This has been further supported by an analysis of the project metallurgy based on the distribution of ore that indicated recoveries of 91.3% and 71.9% for copper and gold respectively (Table 12.5).

Table 12.5: Domain locked cycle results, 2011 G&T.

Composite	Copper concentrate			Distribution
	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

Note: Hazleton waste is 4.4% of feed

12.2 Sample Description

For the 2003 metallurgical program the exploration geologists selected 1,737 samples from diamond drill cores representative of the Kemess North 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 program, five mineral types were identified and the samples were collected and composited as per the following description (Table 12.6).

Table 12.6: Composites descriptions 2011 KUG project.

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	0.4	0.24	1.69
CP4	402	Mixed lithology	0.46	0.23	1.98
CP5	548	Upper volcanics	0.45	0.24	1.89

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

The drill core samples that were used for the 2012 supplemental hardness tests were grouped into the similar five categories with additional samples representing CP6 representing the upper zones of the block cave, and Hazelton (Toodoggone), waste rock that may be drawn in with the block cave during the early years of ore recovery.

12.3 Mineralogical Characteristics of Kemess Underground Ores

The host rock of the Kemess North deposit show some clay and anhydrite formation in the near surface zone. Followed by chlorite and sericite alteration, with increasing depth silicification and the presence of quartz predominate. hematite/magnetite was also encountered.

The pyrite content of samples ranges from a low of 1.4% to a high of 8.1%. The pyrite was coarse grained and was essentially liberated at a primary grind of P80 at 150 µm. The pyrite will be clean (i.e., not activated) and can be rejected by conventional flotation techniques.

Chalcopyrite will range from 0.5 to 1.2%, (0.16–0.44% Cu).

The principal copper mineral will be chalcopyrite. It will occur in intimate association with pyrite. Fine grinding will be required (≤ 15 µm) to produce acceptable liberation of chalcopyrite from pyrite. Secondary copper minerals, bornite, and chalcocite, will occur as trace minerals and will not be of economic significance.

Gold will average 0.58 g/t (ranging 0.18–0.96 g/t).

Gold will occur as native gold and electrum. The silver content of electrum will range from 10 to 20%.

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

In general, the ores of the upper and medium depth mining horizons will be slightly lower in metal content than those of deeper zones.

- Upper and medium depth: 0.17–0.25% Cu, 0.28–0.40 g/t Au
- Deepest mining horizons: 0.22–0.40% Cu, 0.36–0.96 g/t Au

12.4 Specific Gravity and Ball Mill Work Indices

Specific gravity determinations were undertaken with the 2003 metallurgical test program with representative samples by pycnometer, the specific gravity averaging 2.78.

The bond ball mill work indexes were determined for all the composites in the 2003 program. The average ball mill work index for the Kemess North ore was determined to be 13.8 kWh/t.

A specific program looking at ore work indexes for the various ore regimes was completed in 2012 and determined that the average work index for the two main classifications of ore mineralization were:

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

The mill throughput will be influenced by ore hardness and the blended draw of ore through the block cave. In the early years of the block cave, there is a possibility of drawing Hazleton waste in with the ore. Two drill core samples were tested as part of the 2012 program and indicated work indexes of 18.3 and 22.2. kWh/t.

A preliminary production model was provided by AuRico based on expected quarterly blends. The average work index for the different rock types was applied to the annual blends (sequenced four quarters) and the blended work index developed is presented in Table 12.7. The table indicates that

the expected average annual blended work index would vary from 15.8 in the early years, due to the dilution with the Hazleton waste when the block cave is commissioned, to 14.8 in the later years where production will be primarily CP4.

Table 12.7: Blended work index.

		Blend/year														
Domain	**WI**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**10**	**11**	**12**	**13**	**14**	**15**
CP1	13.1	42.06%	41.71%	29.34%	19.23%	12.61%	7.47%	3.32%	0.87%	0.07%	0.00%	0.00%	0.00%	0.00%	0.00%	0.00%
CP2	14.9	0.01%	3.43%	12.85%	17.66%	17.84%	17.03%	16.35%	14.95%	12.17%	8.55%	4.56%	1.75%	0.56%	0.16%	0.00%
CP3	13.9	0.57%	4.10%	7.19%	11.22%	15.34%	19.49%	22.99%	22.18%	17.27%	10.74%	4.71%	1.31%	0.34%	0.08%	0.00%
CP4	14.8	1.90%	0.60%	0.31%	1.11%	2.11%	3.65%	7.23%	17.32%	28.30%	40.54%	59.20%	71.60%	79.76%	89.82%	98.21%
CP5	15.8	29.30%	32.12%	37.03%	42.30%	46.38%	48.16%	47.34%	43.12%	41.43%	39.58%	31.42%	25.31%	19.33%	9.94%	1.79%
Hazleton	20.3	26.17%	18.05%	13.27%	8.49%	5.72%	4.21%	2.77%	1.57%	0.77%	0.32%	0.11%	0.02%	0.00%	0.00%	0.00%
Blended ball mill work index		15.8	15.4	15.3	15.3	15.2	15.2	15.2	15.1	15.1	15.1	15.0	15.0	15.0	14.9	14.8

12.5 Fineness of Grind in Primary and Regrind Circuits

The target fineness of grind for the primary circuit will remain at the current P80 of 150 µm. Test work has shown that adequate liberation of the sulphides for rougher flotation will be achieved. Copper recovery in the rougher circuits will range from 90–96%. Gold recoveries in the rougher flotation will range from 85–92%, but about 15–20% 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. Batch tests were conducted in the 2003 test program in which the re-grind fineness was progressively increased.

Concentrate grades rise as fineness increases while metal losses in cleaner flotation worsen only slightly within the range of re-grinds tested. Typical results from the 2003 test program are tabulated in Table 12.8 and Table 12.9. 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.

Table 12.8: Determination of re-grind product (2003).

Sample test No.	Fineness re-grind	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.

Sample	Head grades		Cu concentrate grades		Concentrate recoveries	
	Cu (%)	**Au (g/t)**	**Cu (%)**	**Au (g/t)**	**Cu (%)**	**Au (%)**
From 2003 report						
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
G&T blended composite						
LCT1 (2011)	0.36	0.78	23.5	40.5	91.3	73.2

Copper concentrate grades increased in similar proportion for all samples tested with fine grinding of rougher concentrates. The target fineness for the re-grinding of rougher concentrates will be a P80 in the range of 12–15 µm.

The results from the flotation work completed during the 2011 program also determined that there was an advantage to finer re-grind size (Table 12.4) showing improved metallurgical performance at a 15 µm target product size (P80).

In 2010 the Kemess South operators completed an evaluation of millpebs (irregularly shaped steel grinding media) for grinding media in the re-grind circuit when production switched to the higher pyrite ratio east pit ore. This material required a finer liberation mesh to achieve acceptable recoveries than the previous hypogene ores. Observations using the millpebs media

included increased recoveries of copper and gold and a reduction of metals lost from overgrinding or sliming in the grinding circuit. Plans are to continue with the development of the millpebs media with the KUG ore requiring a finer re-grind size.

12.6 Grinding Mill Capacity

The KUG project has been based on reducing the primary processing capacity to mill 9 Mtpa. This will be accomplished with the availability of only one of the original grinding lines for processing the KUG Ore.

An analysis of the grinding operating data in specific years from 2001–2010 indicated that the specific grinding energy increased as the Kemess South pit deepened and again when the East Pit ore was processed (Table 12.10).

Table 12.10: Historical grinding energy requirements/

Year	Tonnes	Availability (%)	Power (kWh)	kWh/t	Power utilization (%)
2001	15,360,640	83.52	178,845,717	11.6	67.9
2002	17,308,131	88.71	223,799,692	12.9	80.0
2003	18,701,060	91.39	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 test report was used to develop a preliminary design work index for the KUG Project (Table 12.11). The average ball mill work index was determined to be 13.8 kWh/t. Using the energy requirements, it was determined that the production for KUG ore through the remaining SAG mill–ball mill train would be limited to 22,000 t/d.

Table 12.11: Kemess North bond ball mill work index tests. (kWh/t)

Description	Deposit location	Identity	# of samples	RM work index	Ball mill work index
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
	Upper zones SC6	3B-2	50	14.4	12.0

Description	Deposit location	Identity	# of samples	RM work index	Ball mill work index
		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: KVK Consulting, May 28, 2003 report

For the preliminary design of the KUG project the ball mill work index was based on the average work index measurements for the lower and middle ore zones. This is very similar to the Kemess South work index (Table 12.12, Ave WI=14.6 kWh/t) at 14.2 kWh/t.

Table 12.12: 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

With the recent grinding energy test work results from G&T, the existing grinding circuit capabilities were re-evaluated based on comparison to the mill operating data. Table 12.13 evaluates the Kemess South hypogene ore operating data. Plant data indicated that the mill power utilization was between 80–85% and that plant availability was 91.2%. Assuming a product grind P80 = 150 µm the SAG mill efficiency for power based calculations was estimated at 1.40.

In 2009, the east pit zone became the source of ore for the mill operations. This ore was believed to be harder and to have a higher ratio of pyrite to copper than the historical Kemess South ores. Samples of this ore were tested at SGS and the indication was that the ball mill work index was 14.9–15.3 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 12.14), similar to that for the hypogene ore, indicated that the SAG efficiency factor for 2010 was 1.50. For this evaluation the work index was assumed to be the average of the two samples, which was 15.1 kWh/t. Production through the mill was 18.75 Mt.

Table 12.13: 2003/2004 hypogene ore grinding circuit evaluation.

South data - hypogene ore 2003/2004							
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	t/h		Mill feed rate	2,267	t/h	
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 12.14: 2010 east zone ore grinding circuit evaluation.

East pit - hypogene ore 2010 **Ball mill work index details**							
East zone	14.9						
Southeast zone	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	t/h	
Mill feed rate	2,357	t/h		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		Mill power required	16,256	kW	
				Mill power required	21,800	hp	
				Mill power installed	24,000	hp	90.8%

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

The monzonite ore grinding model (Table 12.15) indicates that the grinding mills would use about 80% of the installed power to achieve the target throughput.

Table 12.15: Monzonite ore grinding model.

Kemess North monzonite work index 14.1 kWh/t							
9 Mtpa							
Primary SAG mill				**Secondary ball mill**			
Mill operating parameters and power required:				Mill power required:			
Annual feed tonnage	9	Mtpa		Daily feed tonnage	2,4658	t/d	
Daily feed tonnage	24,558	t/d		Mill availability	92	%	
Mill availability	92	%		Mill feed rate	1,117	t/h	
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%

Similarly, the Takla ore grinding model (Table 12.16) indicates the grinding mills would use about 87% of the installed mill power to achieve target throughput.

Table 12.16: Takla ore grinding model.

Kemess North Takla work index 15.3 kWh/t **9 Mtpa**							
Primary SAG mill				**Secondary ball mill**			
Mill operating parameters and power required:				Mill power required:			
Annual feed tonnage	9	Mtpa		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	t/h	
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%

Table 12.7 provided a preliminary assessment of how the ores will blend in the block cave. 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, the Hazelton is projected to supply about 27% of the feed. The average weighted work index for the year is increased to 15.8 kWh/t. Table 12.17 models the circuit for the year 1 ore blend and indicates that the grinding circuit would use 90% of the installed grinding power to achieve target throughput.

The plant throughput will fluctuate as the work index varies. Based on the test work 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 where the ability to reduce the critical size may have been beneficial but no long-term economic benefit was identified.

12.7 Flotation Characteristics

Table 12.18 highlights the differences in the ore and processing characteristics of the Kemess South (hypogene) and the KUG deposits.

The rougher flotation for KUG would essentially be a bulk sulphide flotation. The bulk 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 with the Kemess South hypogene ores, alters conditions in rougher and cleaner flotation.

Table 12.19 compares laboratory operating practices when treating the ores of Kemess South and KUG based on the 2003 test program.

Table 12.17: Year 1 blended ore grinding model.

Kemess underground year 1 work index 15.8 kWh/t **9 Mtpa**							
Primary SAG mill				**Secondary ball mill**			
Mill operating parameters and power required:				Mill power required:			
Annual feed tonnage	9	Mtpa		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	t/h	
Mill feed rate	1,117	t/h		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	15.8	kWh/t	
SAG mill work index	15.8	kWh/t		Diameter efficiency factor, EF3	0.91		
SAG efficiency factor	1.5			Feed size efficiency factor, EF4	1		
Transmission loss factor	1			Fineness of grind factor, EF5	1		
Unit power consumption	7.21	kWh/t		Low ratio of reduction, EF7	1.03		
Mill power required	8,052	kW		Transmission loss factor	1		
Mill power required	10,798	hp		Unit power consumption	7.22	kWh/t	
SAG mill power installed	12,000	hp	90.0%	Mill power required	8,059	kW	
				Mill power required	10,807	hp	
				Mill power installed	12,000	hp	90.1%

Table 12.18: 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
Kemess hypogene ore	0.22	0.65–1.0	3.0	1%	No	Primary 145 µm Re-grind 35–50 µm	13.8–15.0 12.5 in sample tested	pH 9.5
KUG ore, 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 µ Re-grind ≤15 µm	±13.8	First pH 10 Second pH 11 Third pH 11.5

Table 12.19: 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
South 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

Plant operations and equipment utilization will require some adjustments:

- 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 cleaner capacity will be sufficient for the KUG ore (i.e., at half the feed rate);
- Re-grind mill unit power input will be greater than current levels because of the finer grinding requirements. Steel consumption and power demand will increase proportionally. At 9 Mtpa and the increased mass pull the existing re-grind mill will not meet the requirements of the KUG project;
- pH in cleaner flotation will rise for pyrite depression without impeding gold flotation; and
- Sufficient concentrate capacity exists in the existing thickeners and pressure filters.

Gold recoveries to the copper concentrate will be 10 percentage points lower when treating KUG ores compared to those obtained for Kemess South 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 report of May 2003) has shown that ultra-fine 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 Kemess South hypogene ores the pyrite content is ±1%. For the KUG project the pyrite content is expected to be ±4%.

Table 12.5 summarizes the open cycle flotation results of the G&T test program completed in 2011. Additional cleaner flotation and locked cycle flotation tests were completed (Table 12.24). The observations and results from these tests generally support the conclusions from the 2003 test program.

Table 12.20 provides an estimate of the size for the rougher flotation concentrate re-grind mill for the KUG ore. The existing re-grind mill is a 6,000 hp ball mill with an estimated maximum power draw capability of 5,400 hp. The mill media charge was changed from balls to millpebs during the last couple of years of operation in an effort to reduce the losses due to sliming observed when the target re-grind size was set finer for the higher pyrite east zone ore. For the KUG ore, test work has shown that better metallurgical results are achieved if the re-grind size is a maximum P80 = 15 µm. With an expected rougher mass pull of 23%, the estimated re-grind power required from the existing ball mill re-grind would not be sufficient.

Using conventional bond ball mill calculations, the additional power required is estimated at 3,000 kW. Stirred mill technology has been shown to be more energy efficient at generating the finer grinds and at the same time minimizing slimes. Typically stirred mills require 60% or less of the equivalent ball mill power requirement, which would be 1,790 kW.

Table 12.20: Estimate of rougher concentrate re-grind power.

23% mass recovery mill power required 9 Mtpa		
	Existing 600 hp ball mill	New requirements
	24,658 tpd	24,658 tpd
Daily feed tonnage	5,671 tpd rougher concentrate	5,671 tpd rougher concentrate
Mill availability	92 %	92 %
Mill feed rate	257 t/h	257 t/h
Feed size, F80	150 µm	24 µm
Discharge size, P80	27 µm	15 µm
Ball mill work index	15 kWh/t	15 kWh/t
Diameter efficiency factor, EF3	0.91	0.91
Feed size efficiency factor, EF4	1.00	1.00
Fineness of grind factor, EF5	1.21	1.47
Low ratio of reduction, EF7	1.03	1.29
Transmission loss factor	1.00	1.00
Unit power consumption	15.66 kWh/t	11.62 kWh/t
Ball Mill power calculated	4,022 kW	2,984 kW
Ball Mill power installed/available	4,028 kW	N/A
Fine grind (stirred)mil powerl requirements	60.00%	1790 kW

12.8 Predicted Results

The predicted metallurgical results have been based on a comparison of the locked cycle test results from the 2003 program and more recent supporting work in 2011. For the purpose of the PEA completed in 2011 it was recommended to base the metallurgy on:

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

The results of the locked cycle tests, completed in 2011 on the composites for the individual ore domains (Table 12.21), supported the previous conclusion. This has been further supported by an analysis of the project metallurgy, based on the distribution of ore that indicated recoveries of 91.3% and 71.9% for copper and gold respectively (Table 12.21).

Table 12.21: Domain locked cycle results, 2011 G&T.

Domain	Copper concentrate			Distribution
	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	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

Note: Hazleton waste is 4.4% of feed

12.9 Concentrate Quality

Historically the copper concentrates produced for the Kemess South operation were good quality with low penalty elements. This is expected to continue with the KUG ore.

The copper concentrates produced will be of excellent quality. Copper concentrate will grade in the range of 22–23% Cu, 30–50 g/t Au, and 75–100 g/t Ag.

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

12.10 Tailings Characteristics

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

The net neutralization potential for KUG tailings will be negative at –50 t $CaCO_3$/1000 t of tailings. Tailings will be stored under water to avoid acidification of the pond.

12.11 Future Test-work

The current grind size in the plant has ranged from a P80 of 130 to 180 µm. The design criteria for the feasibility study will be a grind size of P80 of 150 µm. Both plant data for the past Kemess South mill operation as well as KUG test data from Lakefield Research and G&T were obtained to determine likely gold and copper recoveries at the design criteria grind size.

Based on the test program, a copper rougher recovery of 91.8% and a gold rougher recovery of 79.7% are projected for a grind size of P80 of 150 µm for the middle and deep level KUG ore. The locked cycle test work completed at G&T for the block cave resource has indicated similar results.Future testwork could provide further optimization of the target primary grind size for various ore zones in the deposit.

Addional grinding testwork should also be completed on the rougher concentrate to get a better handle on the unit energy (kWh/t) to regrind the concentrate and firm up the additional regrind power requirements.

13 Mineral Resource Estimates

13.1 Introduction

The mineral resource statement presented herein represents the first mineral resource estimate prepared for the KUG project in accordance with the Canadian Securities Administrators' National Instrument 43-101

The mineral resource model prepared by AuRico considers 146 core boreholes drilled by AuRico and predecessor companies during the period of 1976 to –2010. The resource estimation work was prepared by AuRico Gold employees and was reviewed and verified by Jeffrey Volk, CPG, FAusIMM, a ―Qualified Person" as this term is defined in National Instrument NI 43-101. The qualified person in this case is not independent of the issuer. .

This section describes the resource estimation methodology and summarizes the key assumptions utilized for the Kemess grade estimate conducted by AuRico. In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the gold and , copper, and silver mineral resources for the KUG project, and is suitable as a basis for a Feasibility level of study. The mineral resources have been classified in conformity with generally accepted Canadian Institute of Mining (CIM) ―Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines (2010) and are reported in accordance with the Canadian Securities Administrators' National Instrument 43-101. Mineral resources are not mineral reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

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

Leapfrog Mining 2.3 and Maptek™ Vulcan 8.0.3 were used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model, and tabulate mineral resources. Isatis ® was used for geostatistical analysis and variography, as well as grade estimation.

13.2 Topography

Topography was provided in DXF format to SRK based on a photogrammetric survey conducted by Eagle Mapping LTD in 2003. This survey is based on aerial photos flown on August 2, 1999 by the B.C. Survey. SRK has reviewed the topography surface in cross section comparing elevation between the drillhole collars and the topography surface. SRK found close agreement, and is of the opinion that the 3-D topography model is suitable for use in resource estimation and mine planning at a feasibility stage level.

13.3 Coordinate System

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

13.4 Drillhole Database

The 2011 resource estimate is based on all drilling completed to November 2010, comprising a total of 146 drill holes. Preparation of the estimate followed the completion of a 30-hole infill diamond drill program at KUG in 2010. The 2010 program was designed and executed to increase confidence in the grade distribution, location, and geotechnical characteristics of a target zone of mineralization that has been identified as potentially minable using underground mining methods.

The KUG resource database comprises 32,506 gold-copper analyses and 16,016 silver analyses. Silver was not considered an economic constituent of the original Kemess North Pit project so was not routinely analysed until 2003. In order to report the economic benefits of silver, any existing core or reject material from pre-2003 drilling located within the block cave was analysed for silver in 2010. Some gaps in silver analyses do exist due to previous metallurgical sampling. In these instances, where estimation conditions for silver depart from those for gold and copper, no silver estimation is made. Summary statistics for all drillhole information is provided in Table 13.1. Summary statistics for all assay intervals is provided in Table 13.3

Table 13.1: Summary Statistics for All Drillhole Data

Number	Total Length (m)	Min Length (m)	Max Length (m)	Avg Length (m)
146	67,157.59	16.75	1,206.40	459.98

Table 13.2: Summary Statistics for All Assay Interval Data

Metal	Number	Total Length (m)	Min Length (m)	Max Length (m)	Avg Length (m)
Cu	32,532	62,467	0.01	22.90	1.92
Au	32,705	62,931	0.01	28.90	1.92
Ag	14,687	28,569	0.10	22.90	1.95

13.5 Specific Gravity

Specific gravity (SG) data was collected by analysis of 10-15cm lengths of core; the value was assigned to the corresponding downhole sample length used for assaying. The number of SG values per hole varies; pre-2006 drillholes have sparse data compared to the 2006 and 2010 drill campaigns. The drillholes were composited to 6m. Further work is recommended to assess whether compositing is the right approach. The length-weighted average SG's were compared by domain before and after compositing, there was insignificant change (Table 13.3). The percentage of assay composites containing a SG value per domain is listed in Table 13.4. Three SG values are less than 2.0, and these values require further investigation (Table 13.5). The same domain codes were used for flagging the SG data as for the assayed variables. A default of 2.59g/cm^3 was set for the syenite dykes (domain 6).

Table 13.3: Summary Statistics of Specific Gravity Determinations - Raw Data and 6.0m Composite Data by Estimation Domain

Domain	Sample Type	Number of Samples	Minimum	Maximum	Mean	Std Dev	Variance	Coeff of Var
Sulfate Leached Zone								
Raw Data	Core	1,056	2.04	3.23	2.69	0.14	0.02	0.05
6m Composites	Core	477	2.06	3.04	2.67	0.14	0.21	0.54
Halo Zone								
Raw Data	Core	3,946	1.88	3.74	2.79	0.15	0.22	0.05
6m Composites	Core	1,792	1.88	3.63	2.76	0.16	0.03	0.06
Low Grade Zone								
Raw Data	Core	2,035	1.40	3.48	2.75	0.14	0.02	0.05
6m Composites	Core	704	1.40	3.08	2.74	0.14	0.02	0.05
High Grade Zone								
Raw Data	Core	1,035	2.50	3.28	2.80	0.12	0.02	0.04
6m Composites	Core	354	2.51	3.10	2.80	0.10	0.01	0.04
Post Mineral Intrusives								
Raw Data	Core	210	1.17	2.86	2.62	0.14	0.02	0.05
6m Composites	Core	119	1.17	2.76	2.59	0.18	0.03	0.07
Hazelton Fm. Volcanics								
Raw Data	Core	1,683	2.26	3.12	2.69	0.10	0.01	0.04
6m Composites	Core	934	2.26	3.12	2.68	0.11	0.01	0.04

Table 13.4: Number and Relative Percentage of Specific Gravity Data by Domain.

Domain	No. of SG Intervals	% of Total Domain
Sulphate Leached Zone	1,056	22
Halo Zone	3,946	27
Low Grade	2,035	42
High Grade	1,035	48
Post Mineral Intrusives	210	20
Hazelton Formation. Volcanics	1,683	36
TOTAL	9,965	

Table 13.5: Number of Specific Gravity Values less than 2.0.

Domain	Number of SG Intervals < 2.0	SG Value
Sulphate Leached Zone	0	0.00
Halo Zone	1	1.88
Low Grade	1	1.40
High Grade	0	0.00
Post Mineral Intrusives	1	1.17
Hazelton Fm. Volcanics	0	0.00

13.6 Lithology and Alteration Modelling

A lithologic model was constructed for the KUG deposit. Three alteration zones were also generated: the Sulphate Leach zone, the Phyllic zone, and the Potassic zone (in descending order vertically). The Kemess North Fault was constructed by merging north dipping and south dipping fault surfaces. The fault is a reverse fault with the older Takla formation thrust over the younger Hazelton formation.

The domains used for resource estimation include both the lithological contacts, formed by the post mineral intrusive dykes and the Tooddogone Formation/Hazelton Group volcanic, and by the alteration zone boundaries. The Sulphate Leach zone has been identified as a broken rock and rubble zone within the pre-mineral Takla Formation. This zone has been domained separately as there are significantly lower gold and copper values caused by leaching, and lower core recovery.

Porphyry copper deposits are characterised by large volumes of continuous grade over large distances, radially distributed around and within the source intrusion. Within and near the monzonite intrusion the grade is higher and decreases with increased distance from the intrusive contact.

To ensure metal is not smeared, a contact profile analysis was conducted to assess if high and low grade domains could be defined. The process assesses the average grade with respect to distance from lithologic and alteration contacts. Table 13.6 lists the contacts used in the analysis and whether a gradational or sharp boundary was observed.

Table 13.6: Contact Profile Analysis Results

Contact	Variable	Contact Boundary
Lithological – Monzonite + Takla	Au ppm	Gradational
	Cu %	Gradational
	Ag ppm	Gradational
Alteration – Potassic + Phyllic	Au ppm	Gradational
	Cu %	Gradational
	Ag ppm	Gradational
Grade – Halo zone + LG Domain	Au ppm	Sharp
	Cu %	Sharp
	Ag ppm	Sharp
Grade – LG domain + HG Domain	Au ppm	Sharp
	Cu %	Sharp
	Ag ppm	Inconclusive

The domains used for estimation are listed in Table 13.7: and illustrated in Figure 13.1. Each of the domains was estimated using hard boundaries where a block estimate only uses samples from within that domain. Coding of the drill holes has been carried out using these domain solids. The coding of each domain was visually validated in cross-section on-screen.

Table 13.7: Estimation Domains

Estimation Domains	Description	Boundary Type
Domain 1	Sulphate Leach zone	Hard
Domain 2	Halo zone	Hard
Domain 3	Low grade zone	Hard
Domain 4	High grade zone	Hard
Domain 6	Post mineral intrusive dykes	Hard
Domain 7	Hazleton Group volcanics	Hard

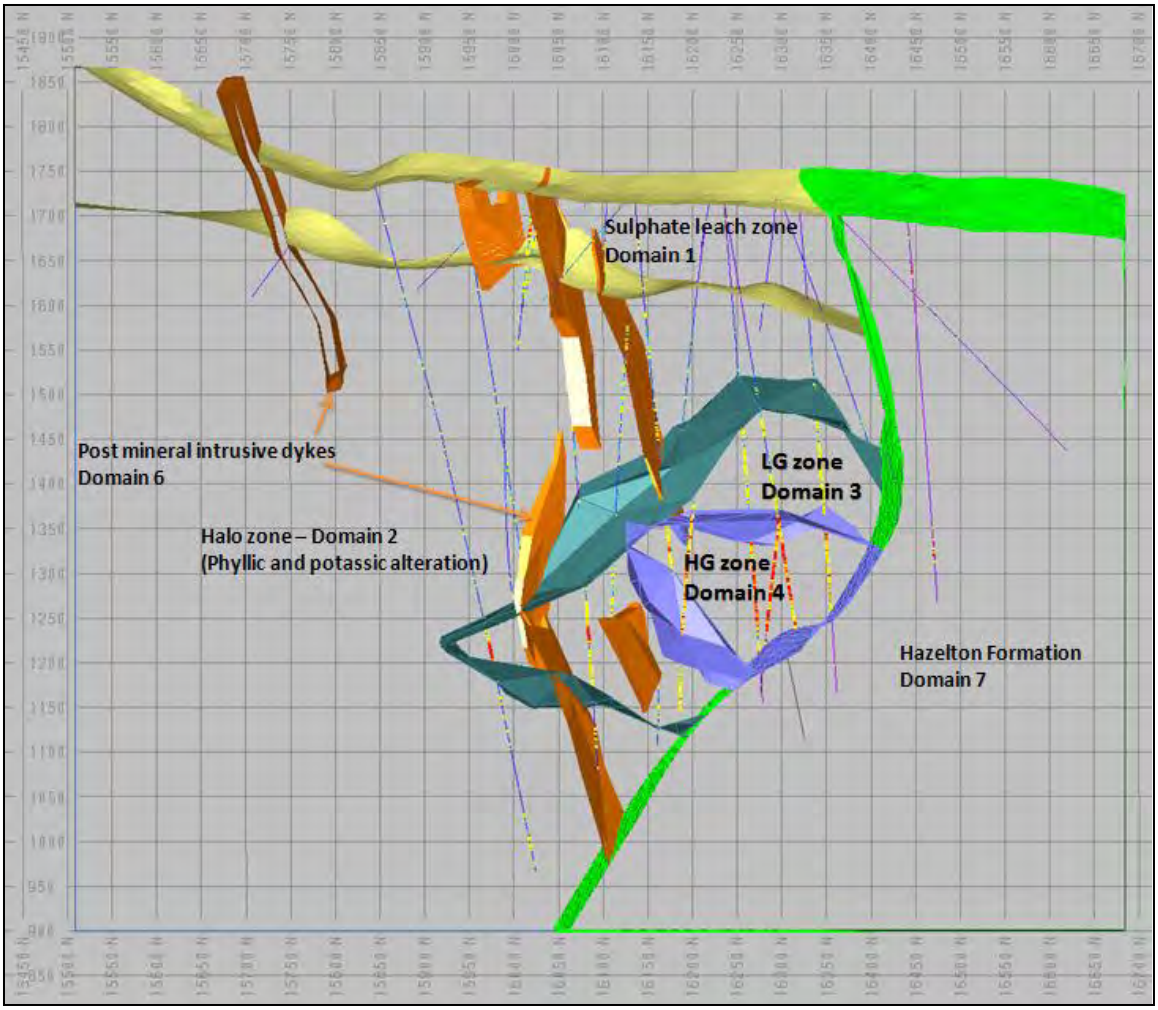


Figure 13.1: Estimation Domains – North-South Cross Section Viewed to the West

13.7 Compositing

Raw sample data have been composited to 6 m downhole intervals using the MineSight™ compositing procedures, with an allowable minimum composite of 3 m. The mean sample length is approximately 2 m for each domain. Compositing to 6 m allows up to three raw data intervals per composite. The compositing process creates 6 m composites of the primary assay intervals in a down-hole direction honouring the coded geological domains. The correlation between the various metals is as follows:

- Gold and copper ranges from 0.699 to 0.845,
- Gold and silver ranges from 0.406 to 0.622,
- Copper and silver ranged from 0.463 to 0.799.

Trend plots were generated for each of the domains to assess the assumption of stationarity for easting, northing, and elevation. No significant trends were interpreted for Domains 3 or 4. Domain 1 shows a decrease in grade in the north east portion of the domain for gold and copper.

This is also evident for Domain 2 for gold (easting) and copper (easting and elevation). It is recommended that this area be assessed for further domains.

13.8 Evaluation of Outliers

The raw copper and gold assay data were examined statistically for the presence of high grade outlier values that could potentially adversely affect grade estimation. Based upon examination of cumulative probability distributions, it was determined that assay capping was not necessary.

13.9 Exploratory Data Analysis

Summary statistical analysis was performed on the raw copper and gold data, in order to further evaluate the data on a domainal basis. Summary statistics for copper by domain are provided in Table 13.8. Summary statistics for raw gold assay by domain are provided in Table 13.9. It can be observed that on a grade-thickness basis, the High Grade, Low Grade and Halo Zones account for the majority of gold metal, A minimum copper grade of zero was entered into the data base instead of a below detection limit default value for domains 1 (Sulphate leached zone) and 2 (Halo zone). This corresponds to the assays of almost half the intervals drilled prior to 2004.

The remainder of the intervals were unsampled and assigned absent data values. The numbers of unassayed intervals is low as a percentage of the total domain dataset and is not considered material in terms of the global resource estimate.

Table 13.8: Summary Statistics – Raw Copper Assay Data by Estimation Domain

Domain	Cu Cutoff (%)	Statistics above Cut-off								Incremental Statistics Between Cut-offs		
		Total Meters	Incremental Pct	Min Grade (Cu %)	Max Grade (Cu %)	Mean Grade (Cu %)	Grade Thickness (%*m)	Standard Deviation	Coeff. of Variation	Total Meters	Mean Grade (Cu %)	Grade Thickness (%*m)
All Data	0.01	52,820	36.00%	0.01	5.17	0.16	8,513	0.15	0.91	19,014	0.05	898
	0.10	33,806	61.08%			0.23	7,615	0.15	0.66	32,262	0.20	6,536
	0.50	1,544	2.66%			0.70	1,078	0.29	0.41	1,407	0.64	897
	1.00	137	0.26%			1.32	181	0.57	0.43	137	1.32	181
Sulphate Leached Zone	0.01	6,388	67.36%	0.01	0.80	0.09	559	0.08	0.94	4,303	0.04	191
	0.10	2,085	32.22%			0.18	368	0.09	0.50	2,058	0.17	352
	0.50	27	0.42%			0.58	16	0.11	0.19	27	0.58	16
	1.00	0	0.00%			0.00	0	0.08	0.00	0	0.00	0
Halo Zone	0.01	23,939	41.36%	0.01	1.70	0.12	2,942	0.08	0.68	9,901	0.05	539
	0.10	14,039	58.28%			0.17	2,402	0.08	0.44	13,952	0.17	2,346
	0.50	87	0.34%			0.66	57	0.20	0.31	82	0.62	51
	1.00	4	0.02%			1.32	6	0.22	0.16	4	1.32	6
Low Grade Zone	0.01	7,962	3.74%	0.01	2.56	0.23	1,860	0.12	0.51	297	0.07	22
	0.10	7,665	93.12%			0.24	1,838	0.12	0.49	7,414	0.23	1,677
	0.50	250	2.89%			0.64	160	0.23	0.37	230	0.58	134
	1.00	20	0.25%			1.29	26	0.38	0.30	20	1.29	26
High Grade Zone	0.01	4,054	0.34%	0.02	4.01	0.40	1,627	0.22	0.54	14	0.08	1
	0.10	4,040	76.59%			0.40	1,626	0.22	0.54	3,105	0.31	967
	0.50	935	21.23%			0.71	660	0.23	0.33	861	0.66	566
	1.00	74	1.84%			1.26	94	0.36	0.28	74	1.26	94
Post Mineral Intrusives	0.01	5,517	19.95%	0.01	5.17	0.18	983	0.14	0.80	1,101	0.06	66
	0.10	4,417	78.81%			0.21	916	0.14	0.69	4,348	0.20	866
	0.50	68	1.18%			0.74	51	0.78	1.05	65	0.59	39
	1.00	3	0.06%			3.82	12	1.81	0.47	3	3.82	12
Hazelton Fm Volcanics	0.01	4,863	68.05%	0.01	2.09	0.11	539	0.18	1.60	3,309	0.02	77
	0.10	1,554	28.33%			0.30	462	0.22	0.73	1,378	0.24	327
	0.50	176	2.90%			0.77	135	0.29	0.38	141	0.65	91
	1.00	35	0.72%			1.26	44	0.26	0.21	35	1.26	44
External to Estimation Domains	0.01	96	92.91%	0.01	0.37	0.04	4	0.06	1.70	89	0.02	2
	0.10	7	7.09%			0.22	1	0.12	0.54	7	0.22	1
	0.50	0	0.00%			0.00	0	0.06	0.00	0	0.00	0
	1.00	0	0.00%			0.00	0	0.06	0.00	0	0.00	0

Table 13.9: Summary Statistics – Raw Gold Assay Data by Estimation Domain

Domain	Au Cutoff (g/t)	Statistics above Cut-off							
		Total Meters	Incremental Pct	Min Grade (Au g/t)	Max Grade (Au g/t)	Mean Grade (Au g/t)	Grade Thickness (g/t*m)	Standard Deviation	Coeff. of Variation
All Data	0.01	57,075	21.40%	0.01	15.75	0.29	16,718	0.35	1.20
	0.10	44,864	75.44%			0.36	16,150	0.37	1.03
	1.00	1,807	3.13%			1.70	3,065	0.88	0.52
	5.00	18	0.03%			6.58	118	2.69	0.41
Sulphate Leached Zone	0.01	7,102	21.02%	0.01	1.52	0.19	1,376	0.13	0.67
	0.10	5,609	78.89%			0.23	1,284	0.12	0.54
	1.00	7	0.10%			1.21	8	0.17	0.14
	5.00	0	0.00%			0.00	0	0.13	0.00
Halo Zone	0.01	24,538	20.88%	0.01	3.19	0.21	5,045	0.14	0.67
	0.10	19,415	78.97%			0.24	4,739	0.13	0.53
	1.00	37	0.15%			1.30	48	0.31	0.24
	5.00	0	0.00%			0.00	0	0.14	0.00
Low Grade Zone	0.01	7,969	0.54%	0.01	5.73	0.43	3,436	0.28	0.65
	0.10	7,926	97.04%			0.43	3,433	0.28	0.65
	1.00	193	2.40%			1.60	309	0.80	0.50
	5.00	2	0.03%			5.73	11	0.00	0.00
High Grade Zone	0.01	4,054	0.07%	0.07	6.12	0.91	3,673	0.66	0.73
	0.10	4,051	69.99%			0.91	3,673	0.66	0.73
	1.00	1,214	29.84%			1.68	2,037	0.72	0.43
	5.00	4	0.10%			5.97	24	0.15	0.03
Post Mineral	0.01	5,766	11.46%	0.01	5.38	0.29	1,695	0.23	0.77

Domain	Au Cutoff (g/t)	Statistics above Cut-off							
		Total Meters	Incremental Pct	Min Grade (Au g/t)	Max Grade (Au g/t)	Mean Grade (Au g/t)	Grade Thickness (g/t*m)	Standard Deviation	Coeff. of Variation
Intrusives	0.10	5,106	87.46%			0.33	1,670	0.22	0.67
	1.00	62	1.05%			1.46	91	0.78	0.54
	5.00	2	0.03%			5.38	11	0.00	0.00
Hazelton Fm Volcanics	0.01	7,493	63.28%	0.01	15.75	0.20	1,490	0.48	2.44
	0.10	2,752	32.80%			0.49	1,350	0.71	1.45
	1.00	294	3.78%			1.95	572	1.40	0.72
	5.00	10	0.13%			7.23	72	3.47	0.48
External to Estimation Domains	0.01	153	96.40%	0.01	0.16	0.03	4	0.03	1.08
	0.10	6	3.60%			0.13	1	0.02	0.16
	1.00	0	0.00%			0.00	0	0.03	0.00
	5.00	0	0.00%			0.00	0	0.03	0.00

13.10 Variogram Analysis

Traditional variograms were generated using composited data for gold and specific gravity (SG). Significantly less input data was available for silver for each of the domains resulting in erratic directional variograms. The high correlation between copper and silver allowed the two variables to be analysed together in an experimental cross-variogram.

Down-hole variograms with a 6 m lag were generated for each of the variables per domain. This assisted in defining the nugget when modelling the directional variograms. The major direction of continuity for gold, copper, silver, and SG, for Halo zone (Domain 2), Low grade, and High grade domains (Domain 3 and Domain 4 respectively), was E-W with a 35° southerly dip. No rotation was applied to the Leached zone (Domain 1) for gold, copper, silver, or SG; or SG for the Hazelton Group (Domain 7).

Experimental variograms were modelled using two nested spherical models and a nugget effect. An example of the modelled and experimental variogram for gold within the High grade domain is illustrated in Figure 13.2. An example of the modelled and experimental cross variogram for copper and silver for Domain 4 is shown in Figure 13.3.

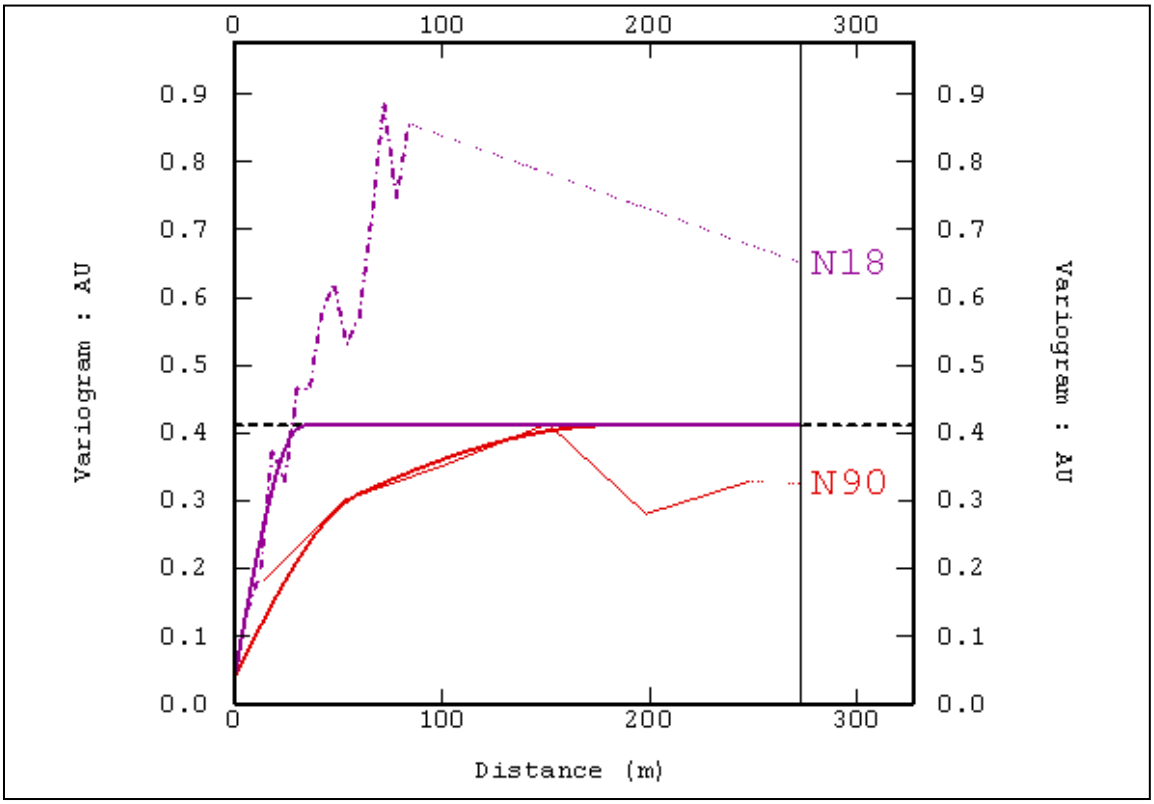


Figure 13.2: Experimental and Modelled Variogram of Gold for Domain 4 (High Grade)

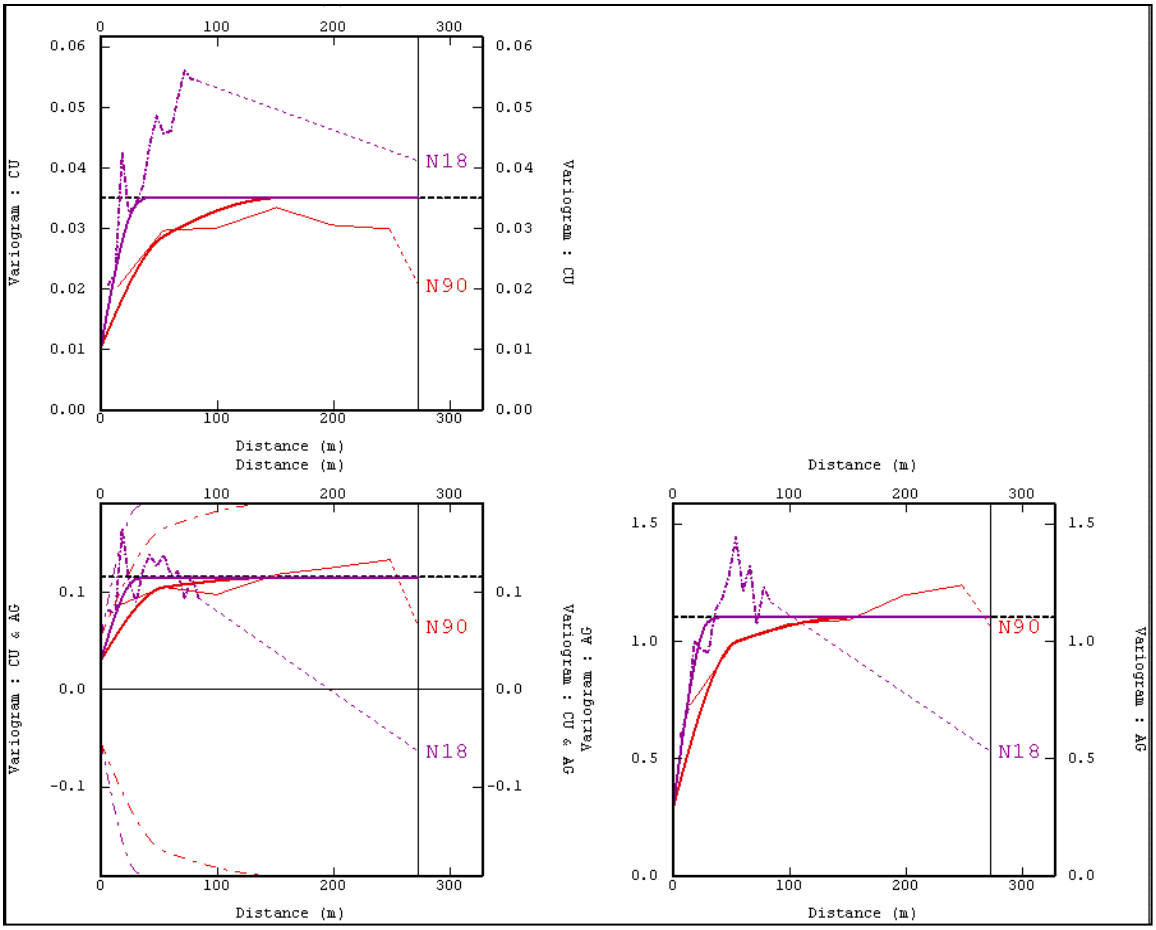


Figure 13.3: Experimental and Modelled Cross Variogram of Copper and Silver for Domain 4

The variogram models for gold and copper/silver for each domain are outlined in Table 13.10 and Table 13.11 respectively. The relative nugget effect for gold ranges from 10-25% and for copper-silver is 17-37%.

Table 13.10: Gold Variogram Parameters

Area	Domain Number	Axis 1	Axis 2	Axis 3	Nugget	1st Structure Spherical	Major Axis (X)	SMajor. Axis (Y')	Minor Axis (Z")	Variance	2nd Structure Spherical	Major Axis (X)	SMajor. Axis (Y')	Minor Axis (Z")	Variance	Total Variance	Nugget
Leached Zone	1	0	0	0	0.0024	1	50	50	100	0.003	1	250	250	140	0.0083	0.014	18%
Halo Zone	2	0	0	35	0.005	1	55	55	80	0.0072	1	340	340	150	0.0078	0.02	25%
LG Domain	3	0	0	35	0.0134	1	80	80	20	0.01	1	200	200	50	0.03	0.053	25%
HG Domain	4	0	0	35	0.04	1	60	60	30	0.18	1	185	185	35	0.192	0.412	10%

Table 13.11: Copper-Silver Cross Variogram Parameters

Area	Domain Number	Variable	Axis 1	Axis 2	Axis 3	Nugget	1st Structure Spherical	Major Axis (X)	SMajor. Axis (Y')	Minor Axis (Z")	Variance	2nd Structure Spherical	Major Axis (X)	SMajor. Axis (Y')	Minor Axis (Z")	Variance	Total Variance	Nugget
	1	Ag	0	0	0	0.083	1	60	60	30	0.2	1	400	400	50	0.087	0.37	22%
	1	Cu	0	0	0	0.0013	1	60	60	30	0.00051	1	400	400	50	0.0034	0.005	25%
Leached Zone	1	Ag/Cu	0	0	0	0.005	1	60	60	30	0.0015	1	400	400	50	0.0094	0.016	31%
	2	Ag	0	0	35	0.1136	1	50	50	60	0.2487	1	360	360	200	0.2827	0.645	18%
	2	Cu	0	0	35	0.001	1	50	50	60	0.00065	1	360	360	200	0.0043	0.006	17%
HG Domain	2	Ag/Cu	0	0	35	0.0084	1	50	50	60	0.0034	1	360	360	200	0.0318	0.044	19%
	3	Ag	0	0	35	0.1965	1	50	50	20	0.2535	1	200	200	50	0.2015	0.652	30%
	3	Cu	0	0	35	0.004	1	50	50	20	0.0026	1	200	200	50	0.0041	0.011	37%
LG Domain	3	Ag/Cu	0	0	35	0.0198	1	50	50	20	0.01788	1	200	200	50	0.022	0.06	33%
	4	Ag	0	0	35	0.2916	1	55	55	30	0.6	1	150	150	40	0.21	1.102	26%
	4	Cu	0	0	35	0.01034	1	55	55	30	0.012	1	150	150	40	0.0127	0.035	30%
Halo Zone	4	Ag/Cu	0	0	35	0.03007	1	55	55	30	0.065	1	150	150	40	0.02	0.115	26%

De-clustering test-work using a series of moving windows of a range of grid dimensions was undertaken to select a block size for estimation. Table 13.12 illustrates an example of the variation in the mean grade and standard deviation of the samples inside the range of block sizes for Domain 4. The drill holes generally are of even spacing, although there is closer spaced drilling in the north-east quadrant in the higher grade portion of the deposit, which is encompassed by Domain 4.

De-clustering does not have a large effect until the block size increases significantly. A block size of 30 m East x 20 m North x 20 m vertically was chosen for the estimation.

Table 13.12: De-Clustering Statistics for Copper and Gold for Domain 4

Domain	Step (m)	Easting (m)	Northing (m)	Elevation (m)	Raw mean	Weighted Mean	Raw Std Dev	Weighted St Dev
High Grade domain Cu	1	10	6	6	0.44	0.44	0.19	0.19
	2	20	13.11	13.11	0.44	0.44	0.19	0.19
	3	30	20.22	20.22	0.44	0.43	0.19	0.19
	4	40	27.33	27.33	0.44	0.43	0.19	0.19
	5	50	34.44	34.44	0.44	0.43	0.19	0.19
	6	60	41.56	41.56	0.44	0.43	0.19	0.19
	7	70	48.67	48.67	0.44	0.43	0.19	0.19
	8	80	55.78	55.78	0.44	0.43	0.19	0.19
	9	90	62.89	62.89	0.44	0.42	0.19	0.18
	10	100	70	70	0.44	0.42	0.19	0.18
High Grade domain Au	1	10	6	6	1.02	1.03	0.64	0.65
	2	20	13.11	13.11	1.02	1.02	0.64	0.64
	3	30	20.22	20.22	1.02	1.01	0.64	0.63
	4	40	27.33	27.33	1.02	1.00	0.64	0.62
	5	50	34.44	34.44	1.02	0.99	0.64	0.60
	6	60	41.56	41.56	1.02	0.98	0.64	0.61
	7	70	48.67	48.67	1.02	0.97	0.64	0.60
	8	80	55.78	55.78	1.02	0.97	0.64	0.60
	9	90	62.89	62.89	1.02	0.96	0.64	0.59
	10	100	70	70	1.02	0.92	0.64	0.56

13.11 Block Model Construction

A block model was constructed in MineSight™ software and imported into Isatis® software. The model parameters are provided in Table 13.13.

Table 13.13: Model Limits - Local Mine Grid

Local Mine Grid	Min (m)	Max (m)	Block Size (m)	No. of Blocks
East	9800	11360	30	52
North	15500	16700	20	60
RL	800	1900	20	55

The block model was coded with the same estimation domain wireframes as used to code the drill hole composites.

MineSight™ software does not sub-cell the parent block; it applies a percentage fill per block. A domain code and percent exists in each individual block. Where two or more domains are present, the block will be coded with all domains. The block percentage will not exceed 100%. The blocks for each domain were visually checked using MineSight™ software to ensure they were coded correctly (inside the individual domain wireframe). A comparison of the wireframes and block volumes is provided in in Table 13.14.

Table 13.14: Comparison of Estimation Wireframe and Model Volumes

Domain	Wireframe (m^3)	Block Model (m^3)	Variance (%)
Leached Zone	92,114,101	91,998,882	0.1
Halo Zone	680,238,311	680,104,579	0.0
LG Domain	46,810,131	46,774,517	0.1
HG Domain	11,733,958	11,715,358	0.2
Syenite	13,325,300	13,235,366	0.7
Hazelton	509,548,525	509,251,080	0.1

13.12 Grade Estimation

For grade estimation, a neighbourhood optimisation study was carried out to determine the appropriate search dimensions and minimum/maximum number of samples for interpolation. The test-work was carried out on the gold values for each domain separately. The change in slope of regression, standard deviation, mean value, and mean sum of positive weights was assessed for a range of sample numbers. A maximum sample value of 20 was used for the estimation. The variation between the number of samples for the slope of regression decreases approaching 20 samples. Negative kriging weights increase as the maximum number of samples is increased from 20.

Search parameters for gold and copper are provided in Table 13.15. The search parameters used for estimating SG are provided in Table 13.16. The rotations used for the search ellipses are the same as the respective variogram model rotation for each variable in each domain.

Table 13.15: Search Parameters for Gold and Copper

Area	Domain	Search Distance (m)			Rotation (degrees)			Sample Selection			
		X	Y	Z	axis 1	axis 2	axis 3	Min. Samples	Max Samples	Search Type	max. samples per Quadrant
Leached Zone	1	220	220	140	0	0	0	6	24	Quadrant	6
Halo Zone	2	285	285	200	0	0	35	6	36	Quadrant	9
LG Domain	3	220	220	70	0	0	35	6	24	Quadrant	6
HG Domain	4	130	130	50	0	0	35	6	20	Quadrant	5

Table 13.16: Search Parameters for SG

Area	Domain	Search Distance (m)			Rotation (degrees)			Sample Selection			
		X	Y	Z	axis 1	axis 2	axis 3	Min. Samples	Max Samples	Search Type	max. samples per Quadrant
Leached Zone	1	250	250	160	0	0	0	6	24	Quadrant	6
Halo Zone	2	390	390	270	0	0	35	6	36	Quadrant	9
LG Domain	3	270	270	85	0	0	35	6	24	Quadrant	6
HG Domain	4	170	170	65	0	0	35	6	20	Quadrant	5
Hazelton	7	500	500	200	0	0	0	6	20	Quadrant	9

The estimation was generated using Isatis® software. Ordinary Kriging (OK) was used for gold and SG and Co-kriging for copper and silver for the four domains (Leached, Halo, Low Grade, and High Grade). SG was also estimated using OK for the Hazelton domain. Default (low) grade values were used for the Hazelton Group volcanics (Domain 7), Leached zone outside the sub-domain (Domain 1), and Halo zone outside the sub-domain (Domain 2). Table 13.17 compares the number of blocks estimated against the total number of blocks per domain.

Table 13.17: Summary of Blocks Estimated and Blocks per Domain

Estimation Domains	Number of Blocks Estimated	Total number of blocks	% Blocks Estimated
Sulphate leached zone	5,223	10,715	49
Halo zone	27,163	63,921	42
Low grade	5,460	5,460	100
High grade	1,380	1,380	100

The Sulphate Leach zone and the Halo zone were sub-domained and the areas outside the sub-domains were not estimated. Default values were set to these blocks. The grade of the post mineral intrusive dykes (Domain 6) was set to the mean grade of the variables. Due to the limited data set for silver not all blocks for domains 1, 2, and 3 were estimated. These blocks were set to a default silver value of 0.1 g/t. Discretization was set to 5x, 4y, and 4z for all estimations.

13.13 Block Model Validation

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

- Visual comparison of drillhole composites with resource block grade estimates by zone both in plan and section;
- Statistical comparisons between block and composite data using histogram and cumulative distribution analysis;
- Generation of a comparative nearest neighbor (NN) model; and
- Swath plot analysis (drift analysis) comparing the OK models with the NN model.

13.13.1 Visual Inspection

Visual comparisons between the block grades and the underlying composite grades in plan and section show close agreement, which would be expected considering the estimation methodology employed. An example cross section and level plan showing block gold grades, gold composite grades and estimation domain boundaries are provided in Figures 13.4 and 13.5, respectively. An example cross section and level plan showing block copper grades, copper composite grades and estimation domain boundaries are provided in Figures 13.6 and 13.7, respectively.

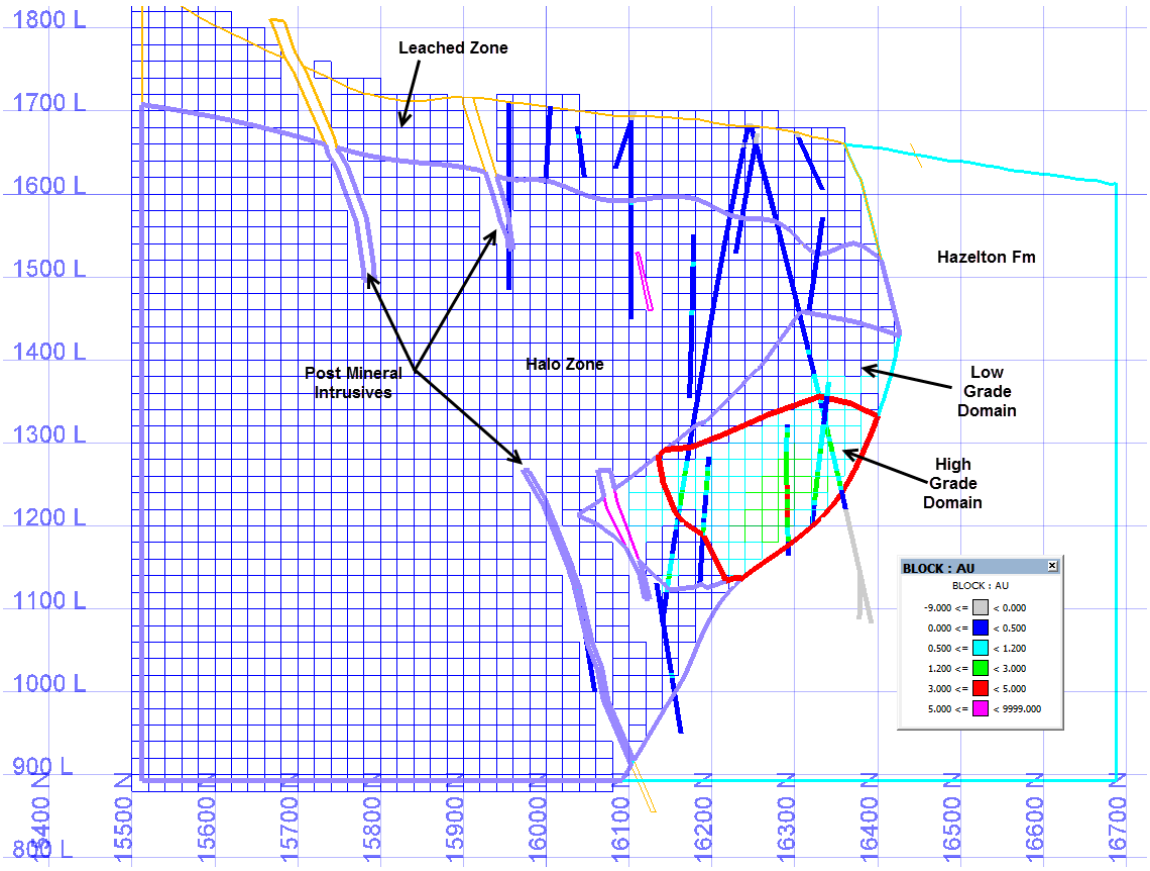


Figure 13.4: North-South Cross Section 10,500E Viewed to the West, Showing Block and Composite Gold Grades and Estimation Domains

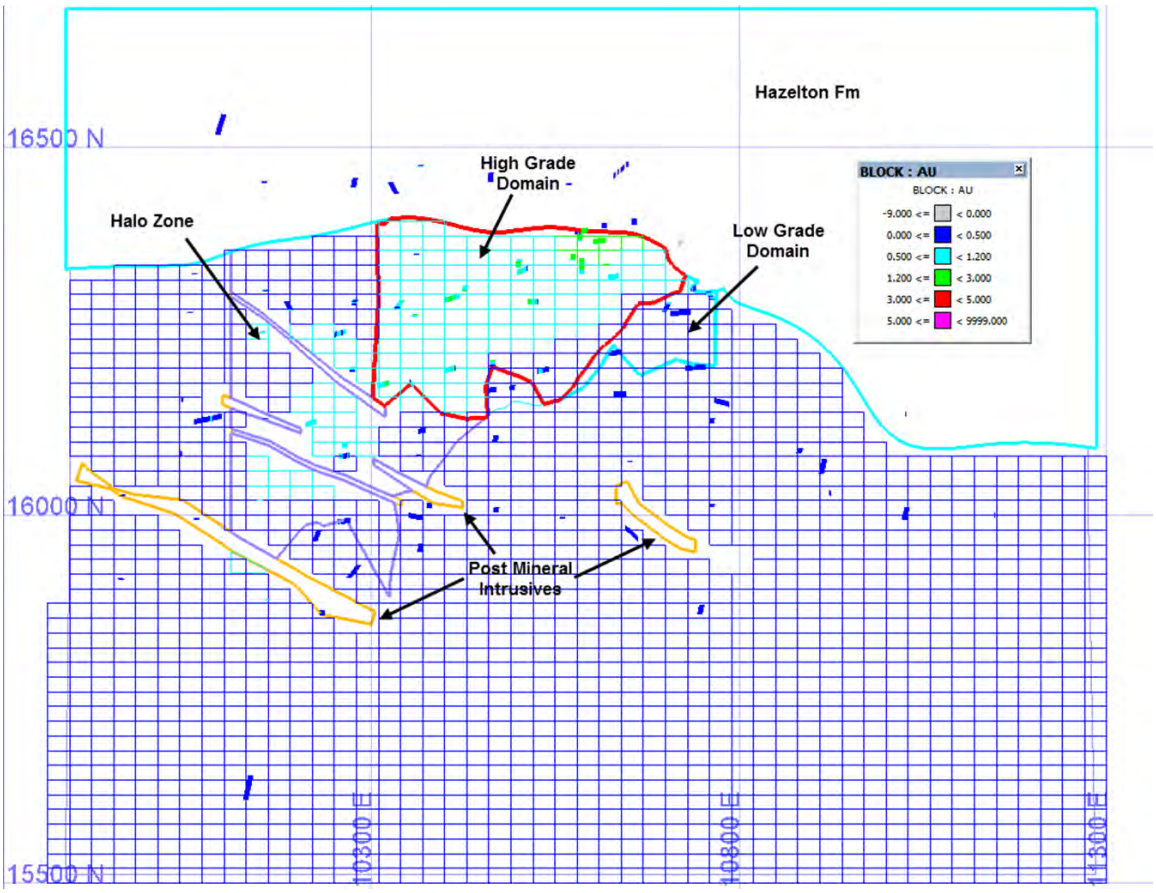


Figure 13.5: Level Plan at the 1,300 m Elevation, Showing Block and Composite Gold Grades and Estimation Domains

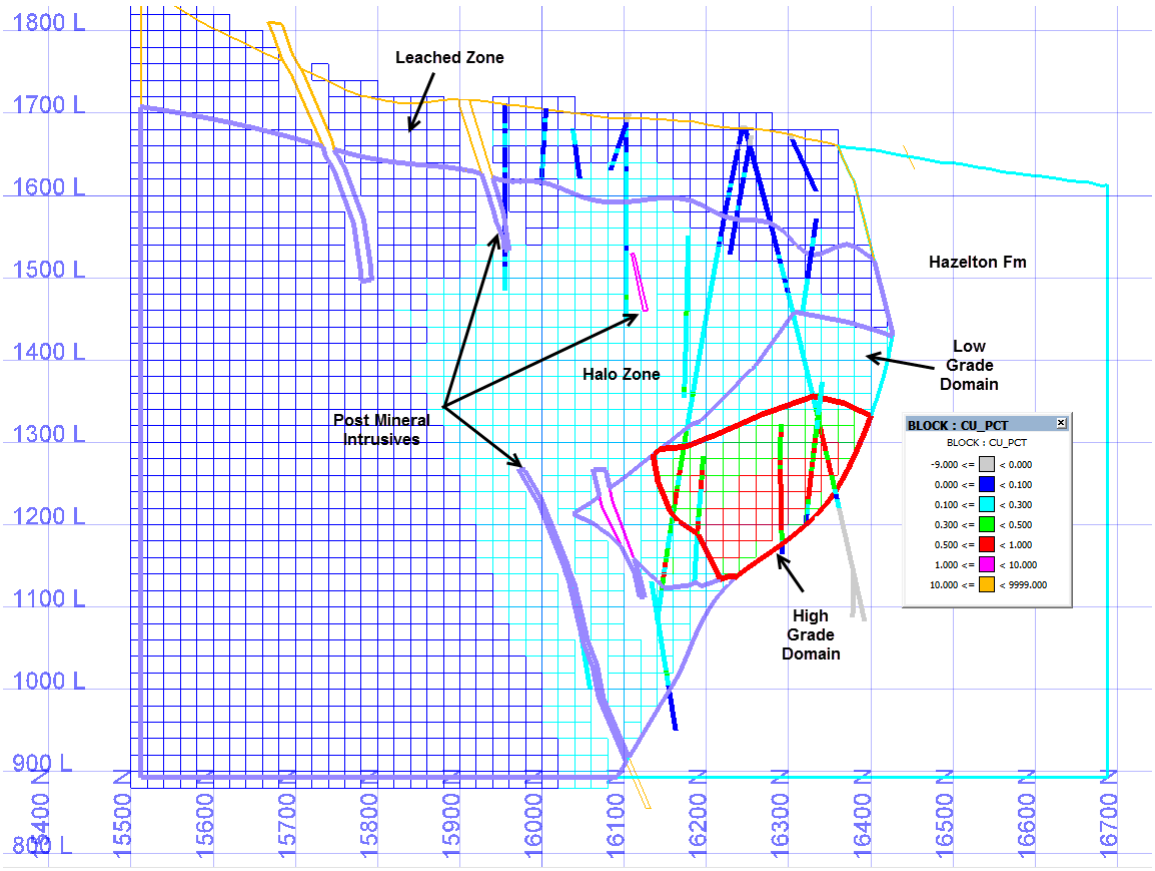


Figure 13.6: North-South Cross Section 10,500E viewed to the West, Showing Block and Composite Copper Grades and Estimation Domains

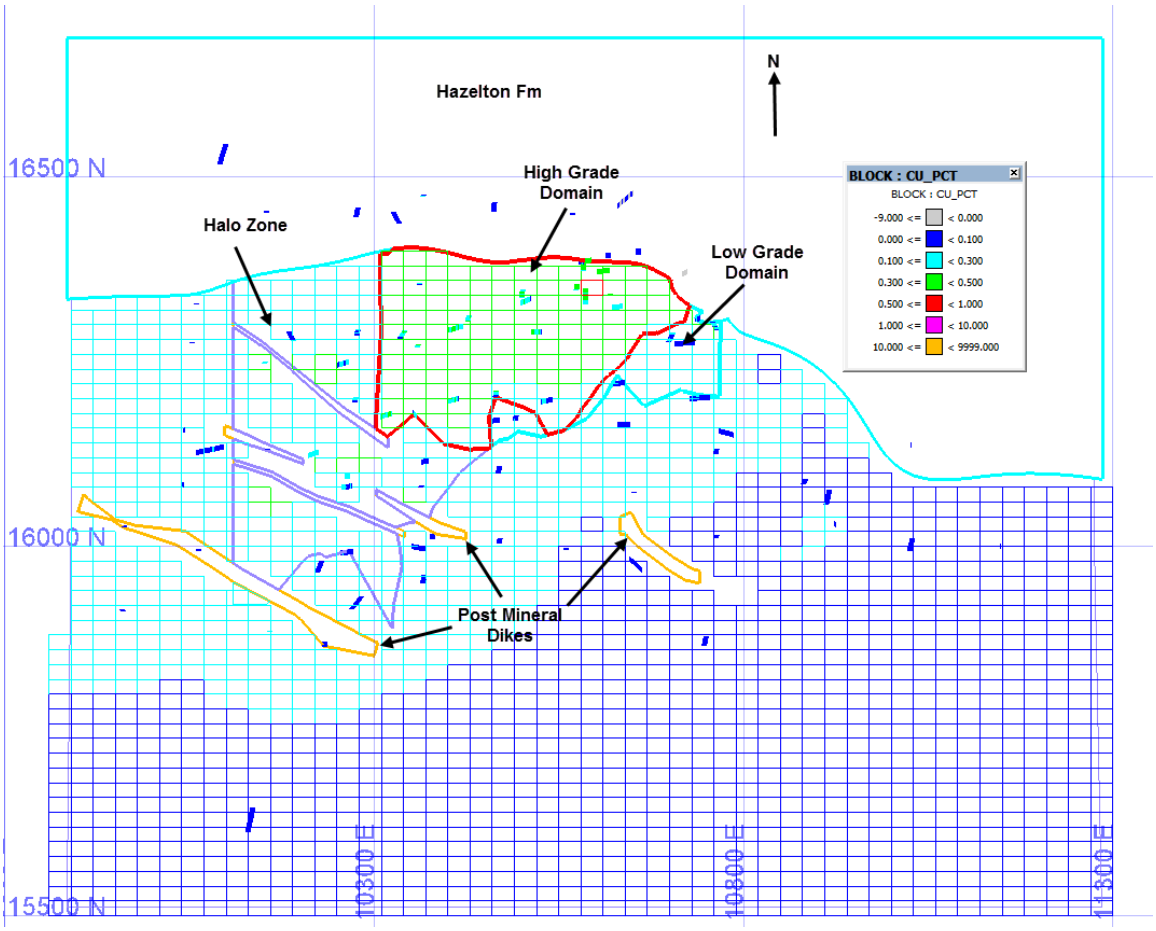


Figure 13.7: Level Plan at the 1,300 m Elevation, Showing Block and Composite Copper Grades and Estimation Domains

13.13.2 Block-Composite Statistical Comparison

Global comparisons of the mean grade of the blocks against the mean grade of the undeclustered and declustered drillhole composites are outlined in Tables 13.18 and 13.19. The percentage variations of the comparisons are within reason for each of the domains.

Overall, these comparisons show that the model grade distributions for silver and gold are appropriately smoothed when compared with the underlying composite distribution, and that the comparison of average grades show close agreement.

Table 13.18: Model Mean Grade Validation Table for Au and Cu.

Domain	Variable	Undeclustered 6m Comp	Declustered 6m Comp	Model Grade	Variance
Leached Zone	Au	0.193	0.191	0.18	6.2%
	Cu	0.083	0.083	0.08	3.9%
Halo Zone	Au	0.215	0.213	0.2	6.6%
	Cu	0.13	0.129	0.13	-1.0%
LG Domain	Au	0.458	0.453	0.46	-1.5%
	Cu	0.252	0.252	0.25	0.8%
HG Domain	Au	1.02	1.019	0.98	4.0%
	Cu	0.436	0.434	0.43	1.0%
	Ag	2.967	2.987	2.96	0.2%

Table 13.19 Model Mean Grade Validation Table for SG

Domain	Variable	Undeclustered 6m Comp	Declustered 6m Comp	Model Grade	Variance
Leached Zone	SG	2.671	2.662	2.63	1.20%
Halo Zone	SG	2.758	2.737	2.71	1.00%
LG Domain	SG	2.739	2.733	2.72	0.50%
HG Domain	SG	2.8	2.796	2.79	0.20%
Hazelton	SG	2.67	2.681	2.69	-0.30%

13.13.3 Comparison of Interpolation Methods

For comparative purposes, additional grades were estimated using NN interpolation methods. The results of the NN model are compared to the OK model at a 0 % copper cut-off grade for the KUG deposit are provided in Tables 13.20 and 13.21 for all indicated and inferred material, respectively. The results of the NN model are compared to the OK model at a 0 g/t gold cut-off grade for the KUG deposit are provided in Tables 13.22 and 13.23 for all indicated and inferred material, respectively. These comparisons confirm the conservation of metal at a zero cut-off, and shows close agreement on both a tonnage and grade basis for all zones. Although on a zone by zone basis, there are some variances, but on a global basis, the grade variances are not material.

Table 13.20: Comparison of OK and NN Tonnage and Grade at a Zero Copper Grade Cut-off – All Indicated Blocks by Estimation Domain

Model	Tonnes	Cu Grade (%)	Cu Lbs
Halo Zone			
OK	94,365,689	0.15	306,675,182
NN	94,365,689	0.15	310,706,724
% Diff	*0.00%*	*1.30%*	*1.30%*
Low Grade Zone			
OK	88,512,673	0.24	466,923,581
NN	88,512,673	0.24	473,957,468
% Diff	*0.00%*	*1.48%*	*1.48%*
Hi Grade Zone			
OK	36,766,799	0.40	322,309,440
NN	36,766,799	0.41	331,467,856
% Diff	*0.00%*	*2.76%*	*2.76%*
All Zones			
OK	219,645,161	0.23	1,095,908,203
NN	219,645,161	0.23	1,116,132,048
% Diff	*0.00%*	*1.81%*	*1.81%*

Table 13.21: Comparison of OK and NN Tonnage and Grade at a Zero Copper Grade Cut-off – All Inferred Blocks by Estimation Domain

Model	Tonnes	Cu Grade (%)	Cu Lbs
Halo Zone			
OK	409,021,905	0.12	1,108,505,511
NN	409,021,905	0.13	1,130,476,498
% Diff	*0.00%*	*1.94%*	*1.94%*
Low Grade Zone			
OK	28,397,362	0.22	139,918,232
NN	28,397,362	0.24	151,840,156
% Diff	*0.00%*	*7.85%*	*7.85%*
Hi Grade Zone			
OK	166,332	0.29	1,058,542
NN	166,332	0.35	1,299,053
% Diff	*0.00%*	*18.51%*	*18.51%*
All Zones			
OK	523,878,559	0.12	1,400,233,775
NN	523,878,559	0.13	1,446,436,288
% Diff	*0.00%*	*3.19%*	*3.19%*

Table 13.22: Comparison of OK and NN Tonnage and Grade at a Zero Gold Grade Cut-off – All Indicated Blocks by Estimation Domain

Model	Tonnes	Au Grade (g/t)	Au Oz
Halo Zone			
OK	94,365,689	0.26	782,162
NN	94,365,689	0.26	790,976
% Diff	*0.00%*	*1.11%*	*1.11%*
Low Grade Zone			
OK	88,512,673	0.43	1,220,003
NN	88,512,673	0.44	1,248,813
% Diff	*0.00%*	*2.31%*	*2.31%*
Hi Grade Zone			
OK	36,766,799	0.86	1,022,109
NN	36,766,799	0.90	1,061,626
% Diff	*0.00%*	*3.72%*	*3.72%*
All Zones			
OK	219,645,161	0.43	3,024,274
NN	219,645,161	0.44	3,101,415
% Diff	*0.00%*	*2.49%*	*2.49%*

Table 13.23: Comparison of OK and NN Tonnage and Grade at a Zero Gold Grade Cut-off – All Inferred Blocks by Estimation Domain

Model	Tonnes	Au Grade (g/t)	Au Oz
Halo Zone			
OK	409,021,905	0.19	2,448,814
NN	409,021,905	0.19	2,484,978
% Diff	*0.00%*	*1.46%*	*1.46%*
Low Grade Zone			
OK	28,397,362	0.38	349,451
NN	28,397,362	0.40	367,292
% Diff	*0.00%*	*4.86%*	*4.86%*
Hi Grade Zone			
OK	166,332	0.56	2,993
NN	166,332	0.71	3,791
% Diff	*0.00%*	*21.04%*	*21.04%*
All Zones			
OK	437,585,599	0.20	2,801,259
NN	437,585,599	0.20	2,856,061
% Diff	*0.00%*	*1.92%*	*1.92%*

13.13.4 Swath Plots (Drift Analysis)

A swath plot is a graphical display of the grade distribution derived from a series of bands, or swaths, generated in several directions through the deposit. Grade variations from the OK model are compared using the swath plot to the distribution derived from the NN grade model.

On a local scale, the NN model does not provide reliable estimations of grade, but on a much larger scale it represents an unbiased estimation of the grade distribution based on the underlying data. Therefore, if the OK model is unbiased, the grade trends may show local fluctuations on a swath plot, but the overall trend should be similar to the NN distribution of grade.

Swath plots have been generated in three orthogonal directions for distribution of gold and copper for all zones in the model area. Swath plots for gold along the EW, NS and vertical directions at KUG are shown in Figures 13.8 through 13.10, respectively. Swath plots for copper along the EW, NS and vertical directions at KUG are shown in Figures 13.11 through 13.13, respectively

There is good correspondence between both models in all orthogonal directions. The degree of smoothing in the IDW model is evident in the peaks and valleys shown in the swath plots, however, this comparison shows close agreement between the OK and NN models in terms of overall grade distribution as a function of X, Y and Z location.

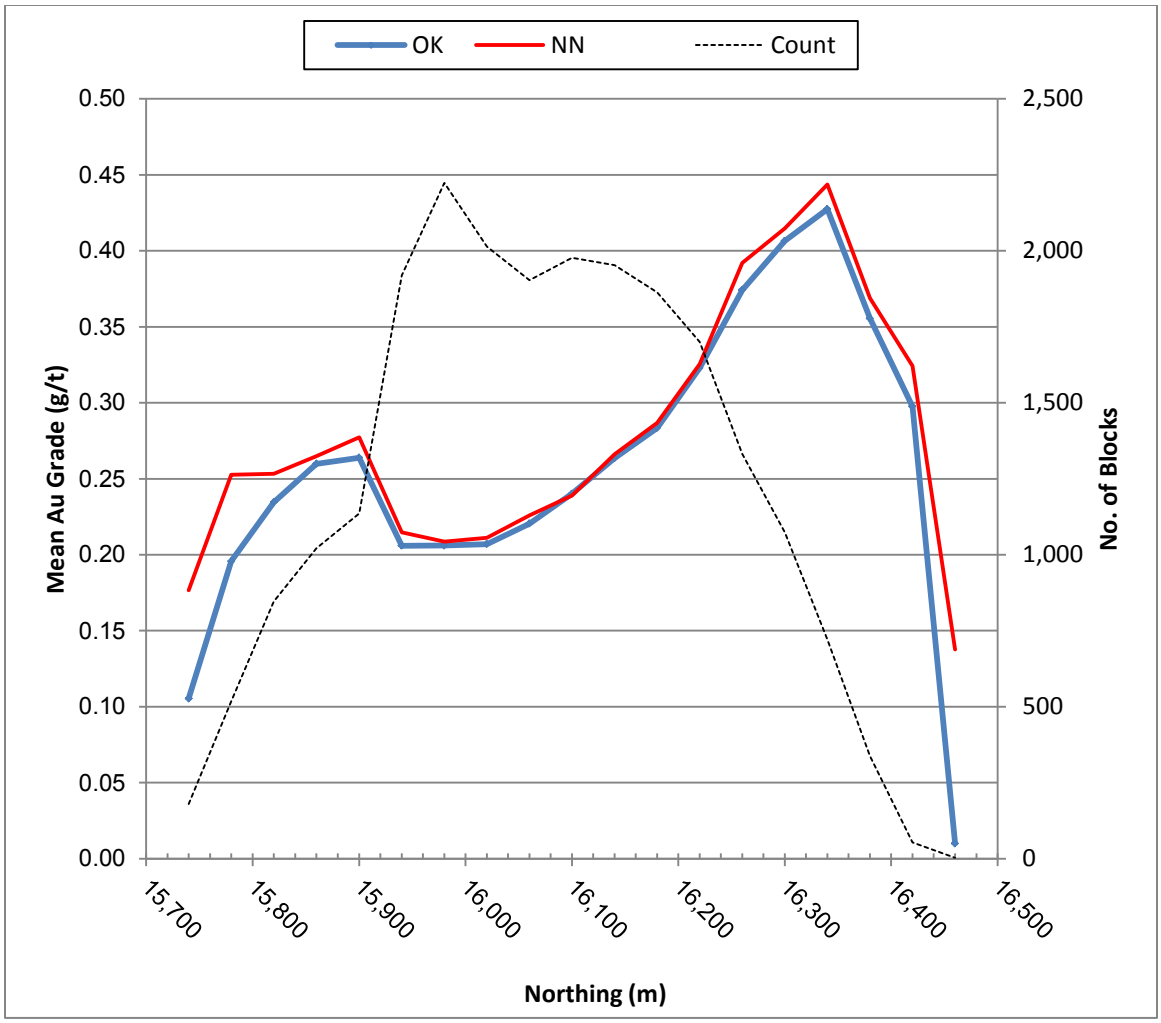


Figure 13.8: East-West Swath Plot, Comparing Ordinary Kriging and Nearest Neighbour Model Gold Grades

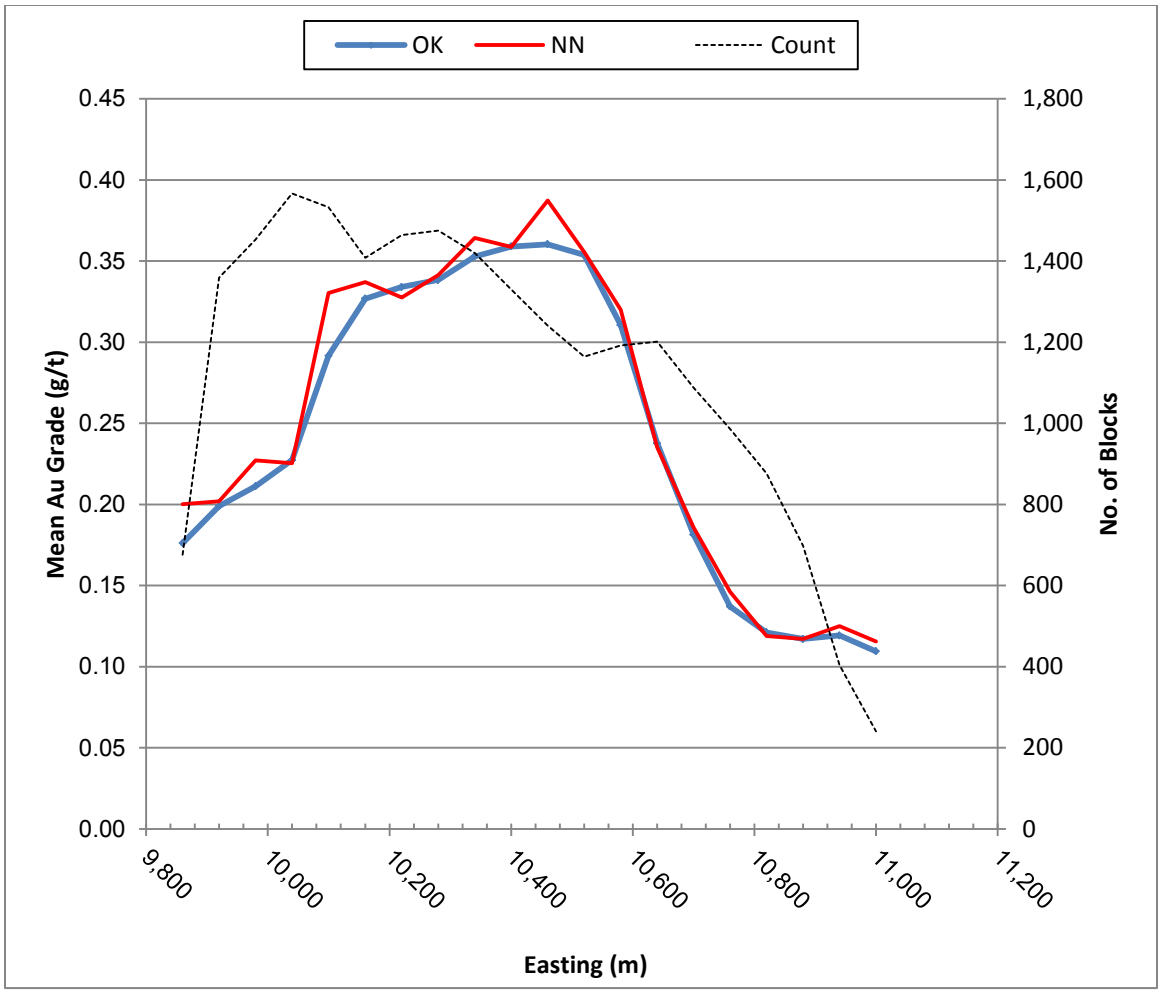


Figure 13.9: North-South Swath Plot, Comparing Ordinary Kriging and Nearest Neighbour Model Gold Grades

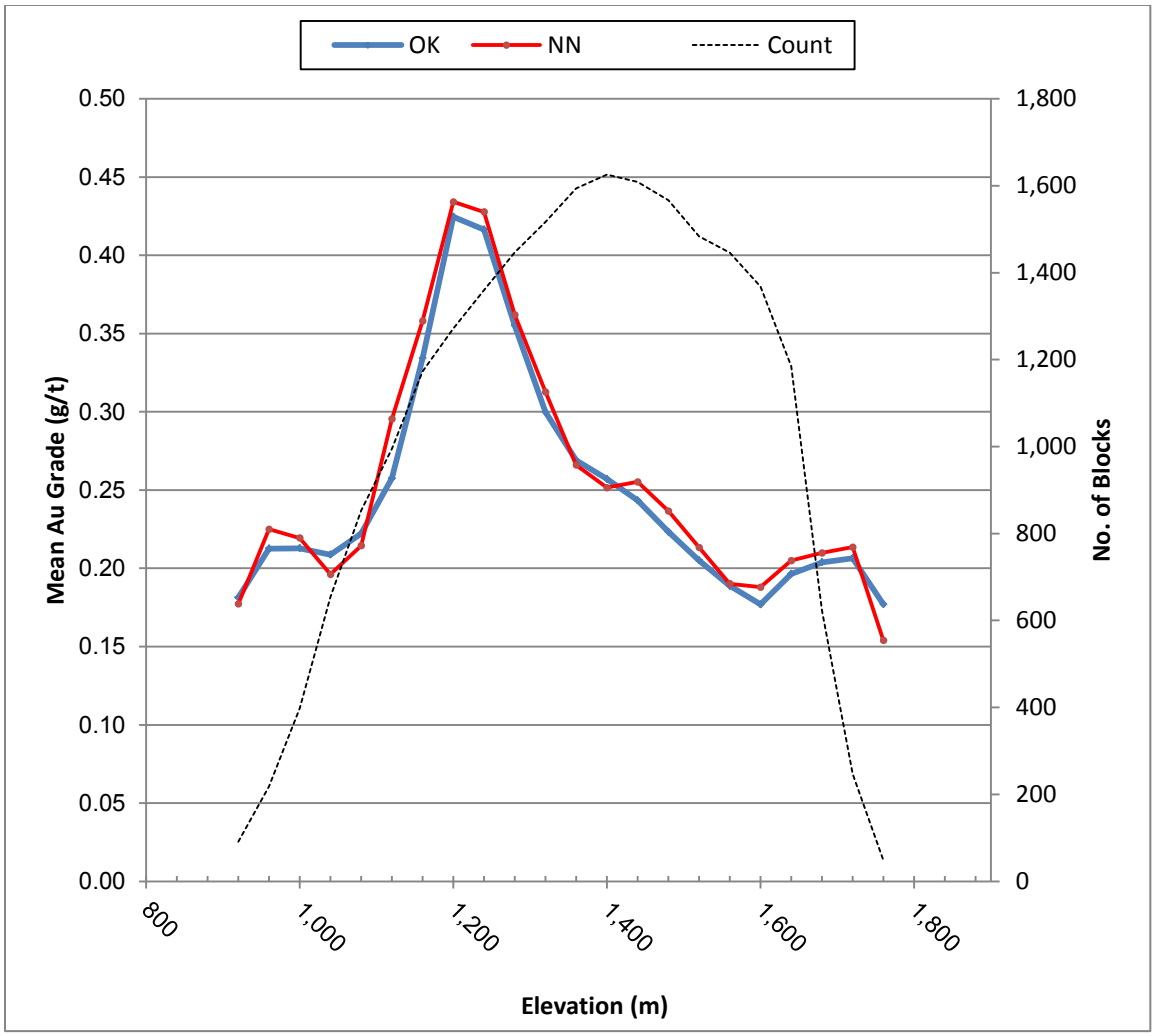


Figure 13.10: Vertical Swath Plot, Comparing Ordinary Kriging and Nearest Neighbour Model Gold Grades

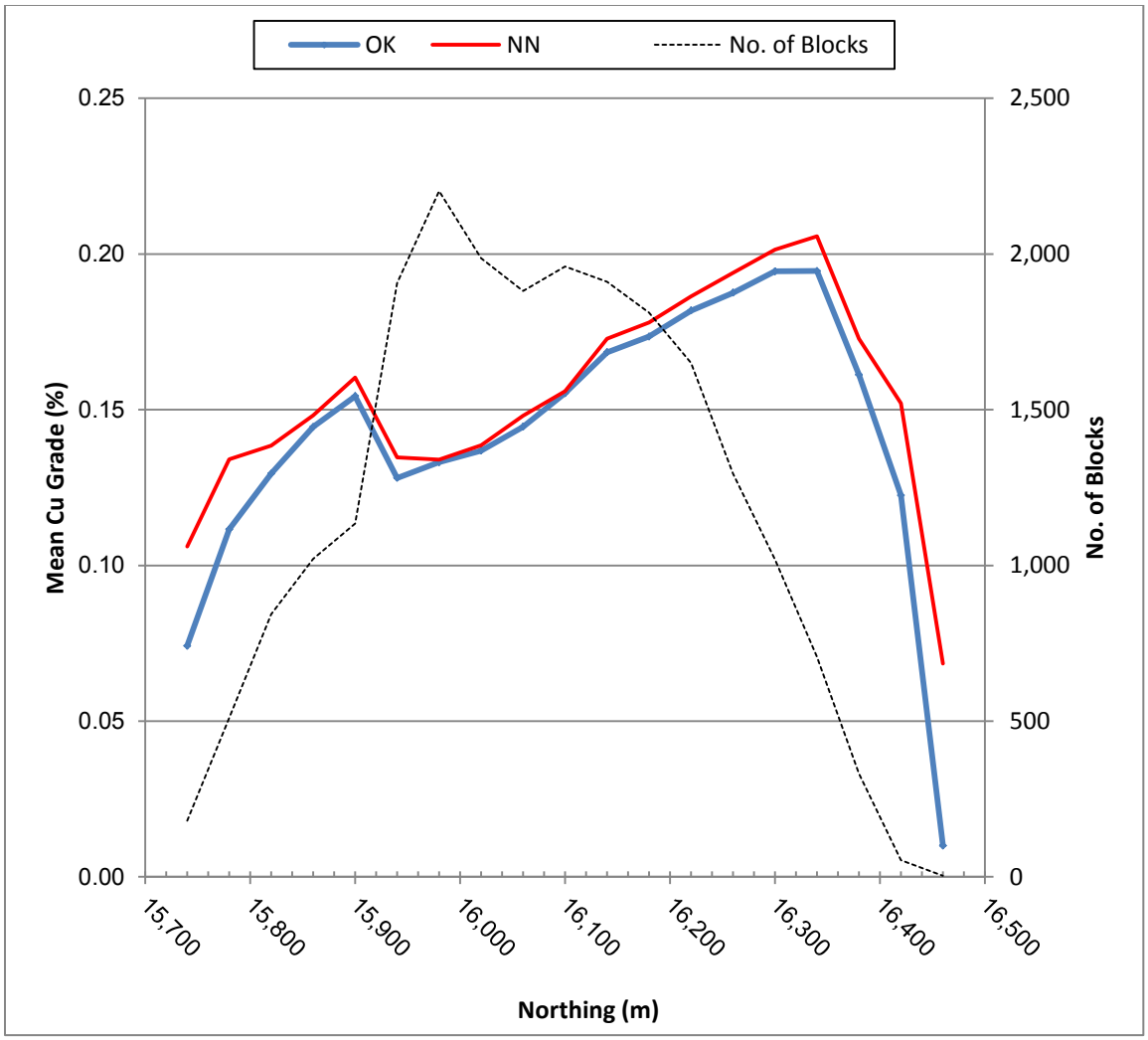


Figure 13.11: East-West Swath Plot, Comparing Ordinary Kriging and Nearest Neighbour Model Copper Grades

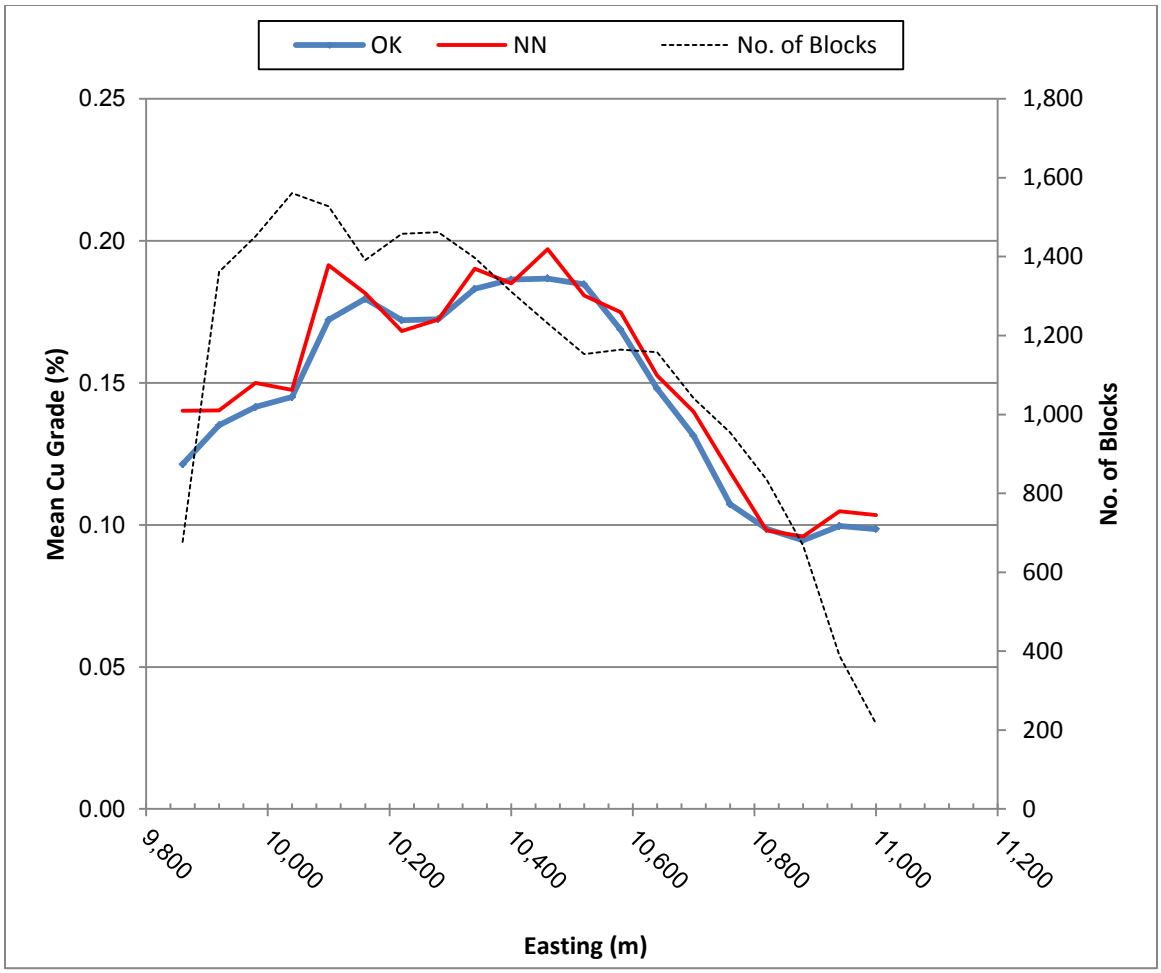


Figure 13.12: North-South Swath Plot, Comparing Ordinary Kriging and Nearest Neighbour Model Copper Grades

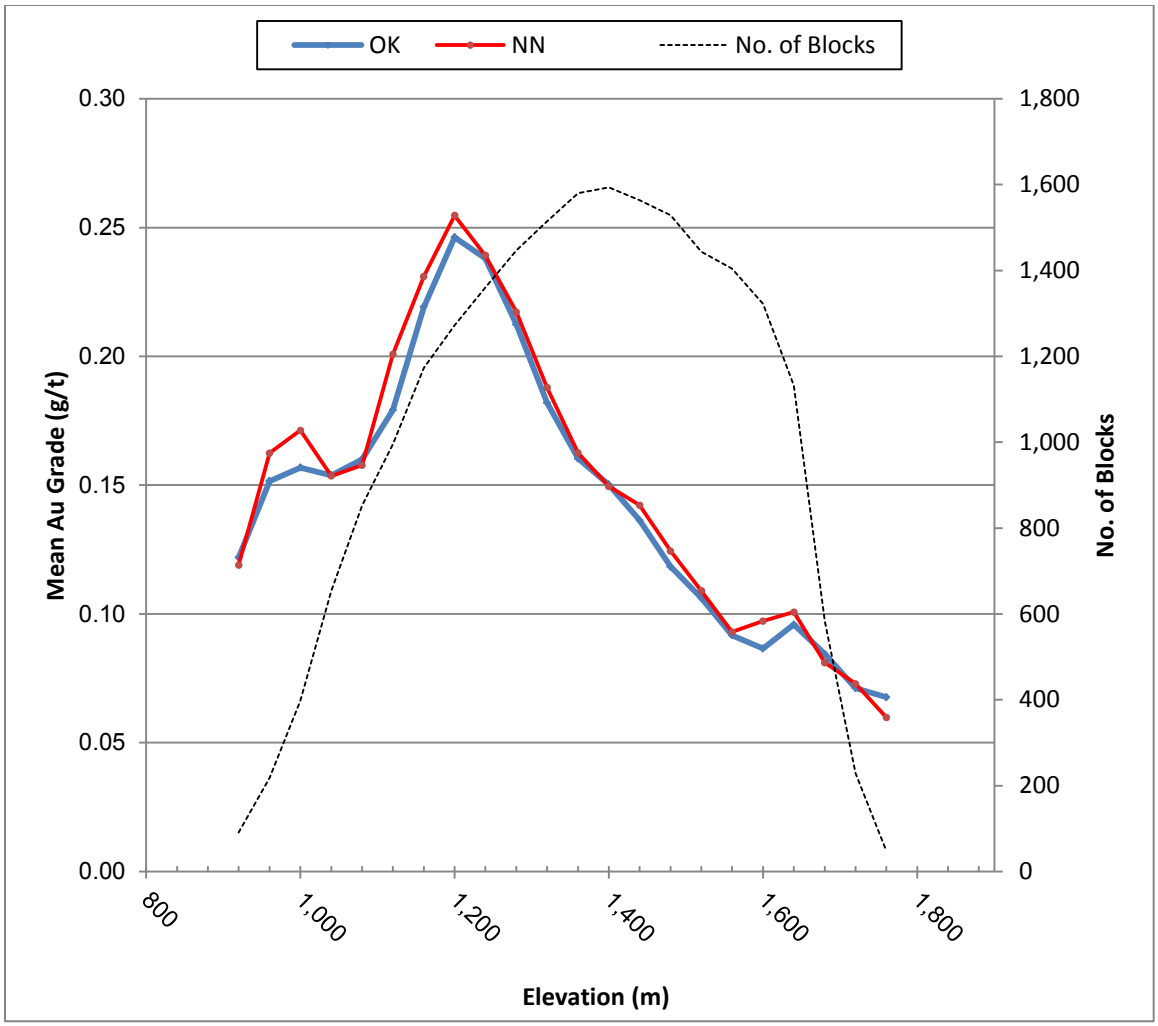


Figure 13.13: Vertical Swath Plot, Comparing Ordinary Kriging and Nearest Neighbour Model Copper Grades

13.14 Mineral Resource Sensitivity

In order to assess the sensitivity of the resource to changes in NSR cut-off grade, AuRico 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 resource model are provided in Table 13.24. It can be observed that the resource is reasonably insensitive to cut-off grades in the increment between $13.00 - $15.00/t NSR, which is likely the grade range of interest. The base case cut-offs are shown in bold.

The sensitivity analysis for Inferred blocks within the Kemess resource model are provided in Table 13.25. It can be observed that the resource is reasonably insensitive to cut-off grades in the increment between $13.00 - $15.00/t NSR, which is likely the grade range of interest. The base case cut-offs are shown in bold.

Table 13.24: Mineral Resource Sensitivity Table - All Indicated Blocks

Cut-off (NSR $/t)	Tonnes (000's)	Cu Grade (%)	Au Grade (g/t)	Contained Metal	
				Cu (000's lbs)	Au (000's Oz)
10.0	178,518	0.26	0.49	1,017,378	2,800
10.5	177,302	0.26	0.49	1,013,855	2,792
11.0	175,924	0.26	0.49	1,009,703	2,782
11.5	173,793	0.26	0.50	1,002,823	2,767
12.0	171,183	0.26	0.50	994,129	2,747
12.5	167,598	0.27	0.50	981,505	2,720
13.0	**162,299**	**0.27**	**0.51**	**962,525**	**2,676**
13.5	156,831	0.27	0.52	942,085	2,630
14.0	150,641	0.28	0.53	918,044	2,575
14.5	143,478	0.28	0.54	889,253	2,510
15.0	136,089	0.29	0.56	858,715	2,440
15.5	130,206	0.29	0.57	833,570	2,382
16.0	124,592	0.29	0.58	808,793	2,325
16.5	119,172	0.30	0.59	784,364	2,267
17.0	113,823	0.30	0.60	759,847	2,208

Table 13.25: Mineral Resource Sensitivity Table All Inferred Blocks

Cut-off (NSR $/t)	Tonnes (000's)	Cu Grade (%)	Au Grade (g/t)	Contained Metal	
				Cu (000's lbs)	Au (000's Oz)
10.0	9,403	0.20	0.36	42,097	109
10.5	9,158	0.20	0.37	41,384	108
11.0	9,095	0.21	0.37	41,182	107
11.5	8,959	0.21	0.37	40,736	106
12.0	8,806	0.21	0.37	40,225	105
12.5	8,422	0.21	0.38	38,889	102
13.0	**7,779**	**0.21**	**0.39**	**36,595**	**97**
13.5	7,471	0.22	0.39	35,438	94
14.0	7,043	0.22	0.40	33,738	91
14.5	6,570	0.22	0.41	31,806	86
15.0	6,034	0.22	0.42	29,584	81
15.5	5,390	0.23	0.43	26,840	75
16.0	5,020	0.23	0.44	25,313	71
16.5	4,485	0.23	0.45	23,010	65
17.0	4,008	0.24	0.46	20,877	59

13.15 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 December 17, 2010.

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. No blocks have been classified as Measured due to the lack of sufficient historic production data.

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 50–80 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, ie. 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 gold, copper, and silver within the block cave volume. Outside the block cave resource area silver has fewer data available for estimation and is of a lower confidence. The Leached zone of the Talka Formation, located above the block cave volume, has 83 intervals of lost core and has similarly been classified as Inferred.

The drill hole spacing of the Indicated resource is predominantly 80 x 80 m between drill holes, with a small area to the north-east of approximately 50 x 50 m spacing. Occasionally there is 100 x 120 m spacing between drill holes at lower elevations due to down-hole deviation. The spatial continuity is reasonable for each of the four domains (variogram ranges are greater than the drill spacing). The Inferred mineral resource classification has drill hole spacing ranging from 50 x 100 to 110 x 130 m.

Block classification has been based on a block in block out approach. If 50% of the block is inside the classification solid, the entire block will be classified into that particular category. Blocks with greater than 50% Hazelton (north of the Kemess North Fault) were classified as waste.

13.16.Mineral Resource Statement

The mineral resources for the Kemess Copper-Gold Project, located in north-central British Columbia, have been estimated by AuRico at 65,432 K t grading an average of 0.41 g/t gold and 0.24% copper classified as Indicated mineral resources; with an additional 9,969 K t grading an

average of 0.39 g/t gold and 0.21% copper classified as Inferred mineral resources. The mineral resources are stated above a $13.00/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 reviewed and verified by Jeffrey Volk, CPG, FAusIMM, Director – Reserves and Resources for AuRico Gold. Mr. Volk is a Certified Professional Geologist and a Qualified Person as defined by international reporting codes.

The effective date of this mineral resource estimate is December 31, 2012 and is based on all data available by January, 2011. The mineral resource statement for the Project is presented in Table 13.26. The resources for the Project are derived from the Isatis block model using density values assigned as described in Section 13.5.

Table 13.26: Mineral Resource Statement, Kemess Copper-Gold-Silver Deposit, Northwest British Columbia, Canada, December 31, 2012*

Resource Category	Tonnes (000's)	Cu Grade (%)	Au Grade (g/t)	Ag Grade (g/t)	Contained Metal		
					Cu (000's lbs)	Au (000's Oz)	Ag (000's Oz)
Measured	0	0.00	0.00	0.00	0	0	0
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 cutoff. 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.

The ―reasonable prospects for 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 grade taking into account extraction scenarios and processing recoveries. In order to meet this requirement, SRK considers that major portions of the KUG resource are amenable for underground extraction via block cave mining development.

The 2011 Mineral Resource Estimate has been prepared by identifying the portion of the resource model that could potentially be mined using block cave mining methods. This was determined by carrying out a preliminary block cave mining study using a model prepared prior to the 2010 drilling program. The study identified a volume potentially suitable for block caving at a

Net Smelter Return (NSR) cut-off value of $1300/t. The values have been calculated by applying the following parameters as defined in Table 13.27.

The wireframe encapsulating the volume was then expanded by 60 m along the western margin and 30 m elsewhere. The resulting wireframe (the ‐2011 Resource Outline") has a volume of approximately 71.5 x 10^6 m^3.

The cost parameters were selected based on experience and benchmarking against similar projects. The reader is cautioned that the results from the underground optimization are used solely for the purpose of testing the ‐reasonable prospects for economic extraction" by a block cave and do not alone represent an attempt to estimate mineral reserves. The results (6) are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off NSR value.

Table 13.27: Parameters Used to Calculate NSR Values and Reasonable Cut-Off

Parameter	Value	Unit
Gold price	1100.00	US$ per ounce
Copper price	2.80	US$ per pound
Silver price	20.00	US$ per ounce
Exchange rate	1.00	$US/$CND
Copper Recovery	90	%
Gold Recovery	68	%
Concentrate Grade (copper)	23	%
Silver Grade in concentrate	52.1	g/dmt
Concentrate Transportation	193.00	$/t
Concentrate Treatment charge	70.00	US$/t
Copper Refining charge	0.07	US$/lb
Payable copper	96.65	%
Payable gold	97.5	%
Gold refining charge	4.25	US$/oz
Silver refining charge	0.50	US$/oz
Mining costs	4.87	US$/t mined
Process cost	5.39	US$/t of feed
General and Administrative	3.45	US$/t of feed
Mining dilution	6	percent
Mining recoveries	94	percent
Assumed process rate	8.0	Feed Mt/y
Assumed mining rate	22,000	t/d

Initial estimates for underground bulk mining costs, combined with milling and general and administrative (G&A) costs consistent with the size of operation envisioned for the KUG, indicate that a mine could be economically viable with a NSR cut-off in the range of $13to $15/t of ore.

14 Mineral Reserve Estimate

The mineral reserve estimate for the Kemess underground project is presented in Table 14.1. The estimate was prepared by SRK using the ‐CIM Standards on Mineral Resources and Mineral Reserves Definitions and Guidelines".

Table 14.1: Mineral reserve statement, Kemess copper-gold-silver deposit, northwest British Columbia, Canada, December 31, 2012.*

Reserve category	Tonnes (000's)	Cu grade (%)	Au grade (g/t)	Ag grade (g/t)	Contained metal		
					Cu (000's lb)	Au (000's oz)	Ag (000's oz)
Proven	0	0.00	0.00	0.00	0	0	0
Probable	100,373	0.28	0.56	2.05	619,151	1,805	6,608
Proven and Probable	*100,373*	*0.28*	*0.56*	*2.05*	*619,151*	*1,805*	*6,608*

Notes*

(1) Estimated at US$3.00/lb Cu, US$1,300/oz Au using a cutoff NSR Value of $15.30/t and a shut-off NSR Value of $17.30/t of ore. Metallurgical recoveries and other parameters are shown in Table 22.2, 22.3, and 22.4.

(2) Mineral reserve tonnage and recovered metal have been rounded to reflect the accuracy of the estimate, numbers may not add due to rounding.

The reserve was estimated using the resource block model reviewed and validated by Jeffrey Volk, CPG, FAusIMM, an employee of AuRico. The contents of the model formed the basis of the mineral resource estimate summarised in KUG mineral resources.

The reserve has been estimated for a block caving mining method and takes into account the effect of mixing indicated resources with dilution from low-grade and barren material originating from within the cave outline and from overlying material.

15 Mining Methods

The opportunity to exploit the KUG Cu-Au deposit by underground mining methods was evaluated by a PEA completed by AMC in July 2011. This PEA considered long hole open stoping, sub-level caving, and block cave mining methods; the block cave approach was recommended due to superior economics. As a result, AuRico (then Northgate Minerals) decided to carry out a feasibility study (FS) for the KUG deposit based on block caving. This section discusses all mining-related aspects of the KUG FS. See Figure 15.1 for an overall view of the project.

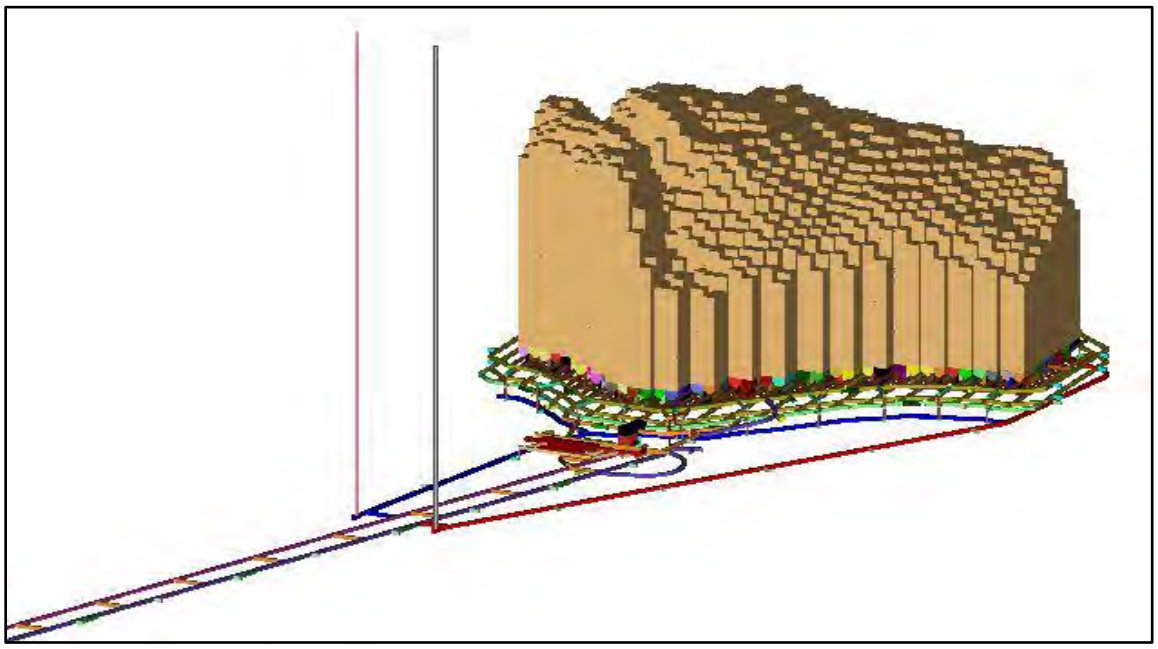


Figure 15.1: View of KUG, looking to the northwest.

Access to the cave is via a twin decline system, developed from portals approximately 3.5 km south of the KUG deposit: one decline for access and the other for ore conveying. A total of 640 drawpoints will be developed on the extraction level (1,160 m above mean sea level (AMSL)). The drawpoints will be used for the loading of undercut and cave ore from the overlying undercut level (1,178 m AMSL) via 320 drawbells. A fleet of load-haul-dump (LHD) units will tram drawbell, undercut, and cave ore to a single primary gyratory crusher located directly south of the cave footprint. Crushed ore will then be conveyed to a crushed ore stockpile near the existing processing facility via a 3.4 km long underground conveyor and 4.6 km long surface conveyor.

Mine infrastructure facilities are generally located to the south of the cave footprint to allow early establishment (following twin decline development) and ready access during operations. Intake and return air raises are developed from the surface, with collars located on the ridge to the south of the KUG deposit, with this primary circuit providing up to 400 m^3/s of air to the footprint area of KUG. Intake air is distributed via an intake air level (1,140 m AMSL) north of the footprint below the extraction level via 28 intake air raises to the extraction and undercut levels. Return air from these levels is collected via 10 return air raises to a return air level (also at 1,140 m AMSL) south of the footprint. In addition, the twin declines and underground workshop are ventilated by up to

70 m^3/s of air provided by exhaust fans located at the top of the conveyor decline. The underground workshop and main dewatering facility are also located south of the footprint.

A total of 47,473 m of lateral development and 2,444 m of raise development is required to extract the estimated KUG reserve, with development occurring over an eight year period. Commercial production, defined as 60% of the targeted 9 Mtpa production rate, is estimated to be achieved in the fifth year of development. In the following year, Year 2 of the schedule, ore production is estimated to be 8.5 Mt, with 7.1 Mt being cave ore. The target rate of 9 Mtpa is achieved from Year 3 to Year 10, with the cave contributing the full amount from Year 5. Production declines from Year 11 (8.4 Mt), with final cave production in Year 12.

KUG is estimated to contain the following probable reserves:

- 100.37 Mt ore,
- 0.28% Cu, equating to 280.9 kt Cu,
- 0.56 g/t Au, equating to 1.805 M oz oz Au, and
- Average $28.34/t NSR.

The optimal cave was defined using Gemcom's PCBC™ cave optimisation and production scheduling software. The Gemcom GEMS™ Footprint Finder module identified 1,160 m AMSL as the optimal location for the extraction level. A draw column shut-off value of $17.30/t was then applied to identify the optimal cave profile, which comprises 640 drawpoints or 320 drawbells. Multiple production sequences were evaluated to identify the highest value scenario, which starts as a chevron in the northeast of the footprint and becomes aligned north-south as the cave front moves west; this was simplified for mine scheduling to a north-south cave front that moves from the higher grade core east of the footprint to the west.

15.1 Mine Design

15.1.1 Geotechnical

Geotechnical assessments, particularly cavability and fragmentation, are of upmost importance when determining the suitability of a rock mass for cave mining. SRK 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, cavability assessment, fragmentation assessment, and ground support recommendations. The following sub-sections summarise this scope.

15.1.2 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 twin 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 for the 2011 programme was carried out for subsequent use of the in situ rock mass rating (IRMR) method, after Laubscher & Jakubec (2000). The recorded information included:

- Drill core parameters: Total core recovery, rock quality designation, and orientation offset;
- Rock mass parameters: Intact rock strength, microdefect intensity & strength, matrix alteration, and veining intensity;
- Joint parameters: Discontinuity elevation-orientation & type-character, and joint condition; and
- Large scale structures: Location, orientation, and 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.

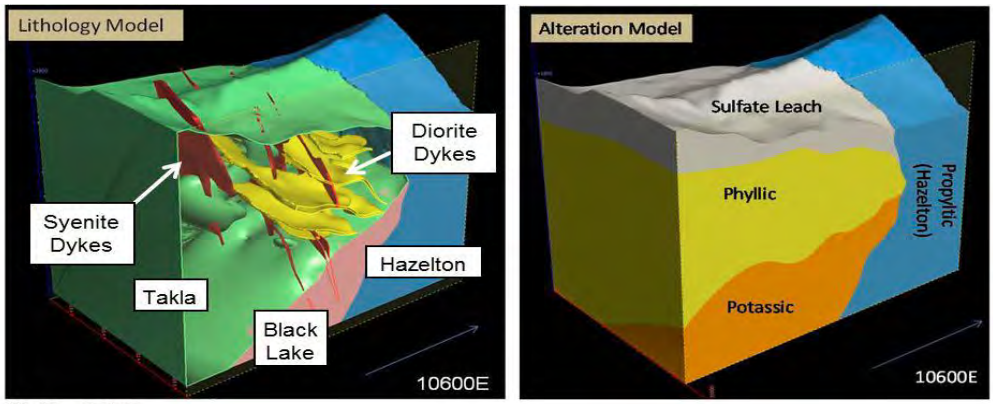
In-field and laboratory testing of core from all three geotechnical drilling programmes included point load tests, direct shear tests, uni-axial compressive strength tests, tri-axial compressive strength tests, and Brazilian tensile strength tests, covering all lithology and alteration types.

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 (i.e. around the Kemess North and East Bounding Faults) and syenite dykes as separate geotechnical domains.

Figure 15.2 shows the relative locations of the various lithological and alteration zones at KUG.



(After Edmunds, 2001)

Figure 15.2: Lithological and alteration zones at KUG.

The location, width of deformation, and character of the Kemess North and 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 NNW 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, ENE, ESE, 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 were 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 (k_1 = 1.26), intermediate principal in situ stress is aligned sub-vertical (= 1.00), and minor principal in situ stress is aligned roughly E-W (k_2 = 0.66). At a depth of 500 m below surface, SRK estimates maximum, intermediate, and minor principal in situ stresses to be σ_{max}=15–20 MPa, σ_{int}=14 MPa, and σ_{min}=8–10 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 table below (Table 15.1) summarises rock mass data for each of the geotechnical domains.

Table 15.1: 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

15.1.3 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 that the KUG deposit will cave readily at a hydraulic radius (HR) well within the complete footprint dimensions.

Empirical assessment of cavability 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 cavability, 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 23–33 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 x 300 m. In fact, this range is also well within an HR of 50 that represents just one quarter of the complete KUG cave footprint area.

Figure 15.3 and Figure 15.4 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 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 x 112 m. For the Hazelton, it is 120 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 cavability, which is expected to reduce the required HR by 5-10%.

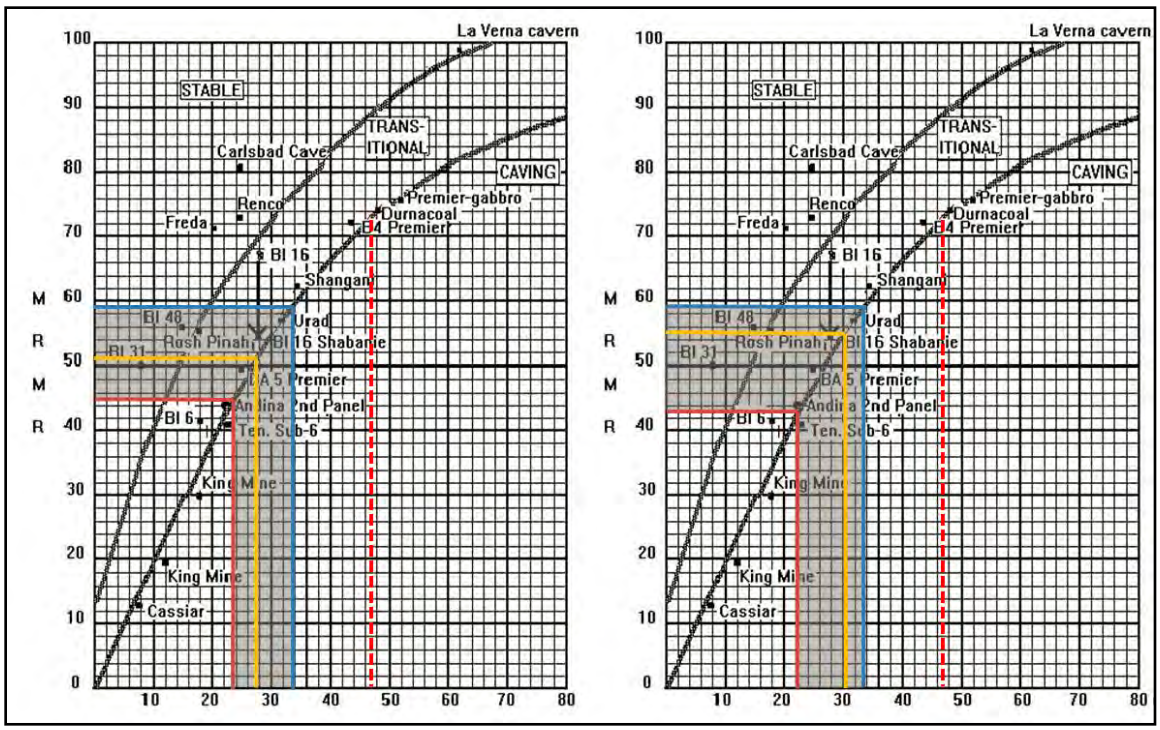


Figure 15.3: Stability diagrams for Black Lake intrusives (left) and Hazelton (right).

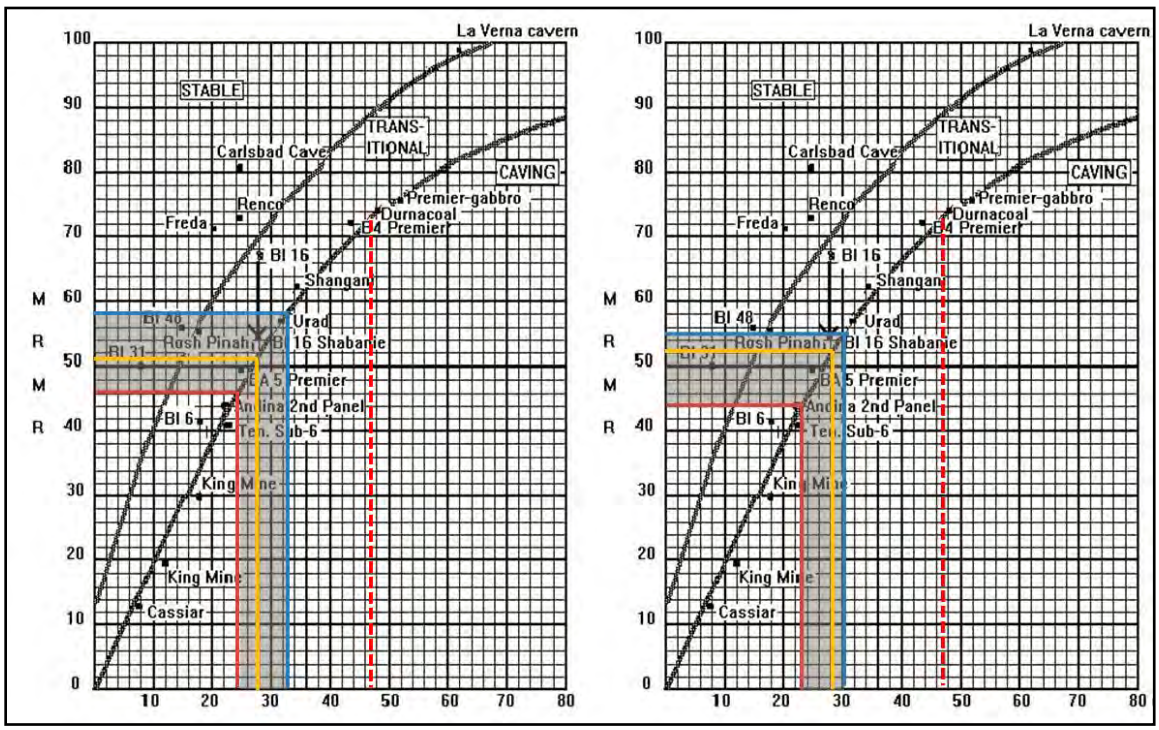


Figure 15.4: Stability diagrams for Takla Potassic (left) and Takla Phyllic (right)

Numerical modelling of the KUG was based on rock mass properties provided by SRK. The underlying graphic (Figure 15.5) 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 isosurfaces represent 0.5 m vertical displacements.

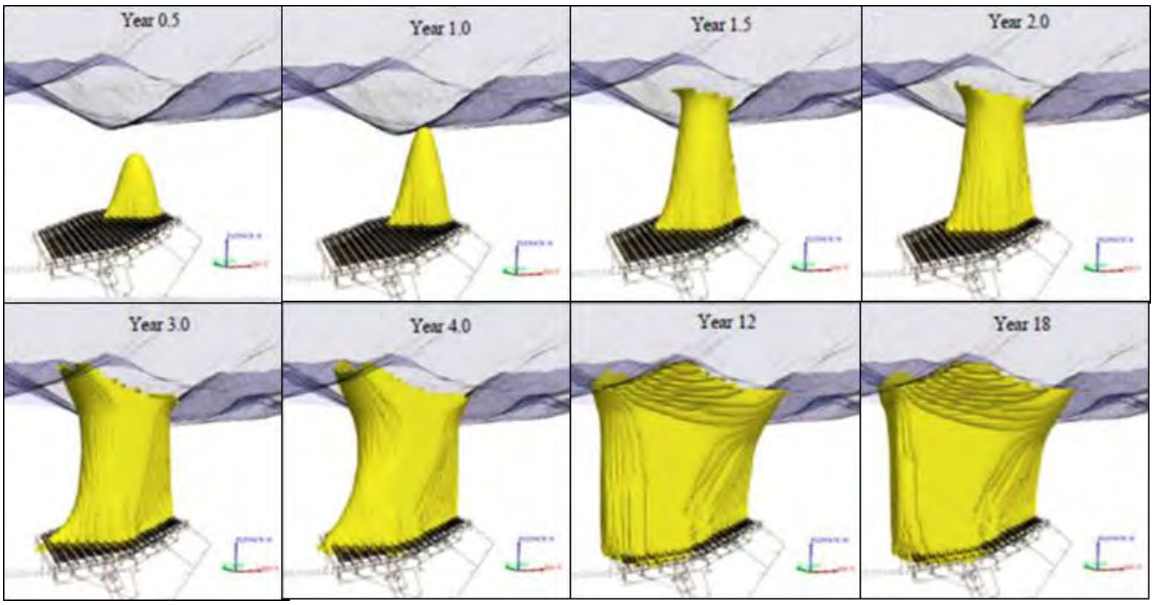


Figure 15.5: 0.5 m vertical displacement isosurface.

In addition to the above mentioned cave numerical modelling, Itasca modelling estimated the extent of the subsidence cone. This is shown by the final isosurface in Figure 15.5 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.

15.1.4 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 15.2).

Table 15.2: Rock mass and joint properties for fragmentation assessment.

Domain	MRMR	Mi[1]	Intact rock strength	Fracture/veinlet	Fracture/veinlet	IBS[2]
				Frequency	Condition	
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:

Mi = Hoek brown parameter constant. mi value determined from tensile and UCS testing (supported by typical ranges for similar rock types).

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 the dip angle (45° constant) and dip direction of the 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 1.50–1.75x 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 15.3):

Table 15.3: Dip and strike stress values for various cave propagation directions.

	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 (Cavability Assessment) were used to assess primary fragmentation as these are considered to provide a reasonable range. Actual fragmentation is expected to be bound by the curves representing expected (natural +50% artificial breaks) and optimistic (natural +100% artificial breaks) joint conditions. Figure 15.6 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.

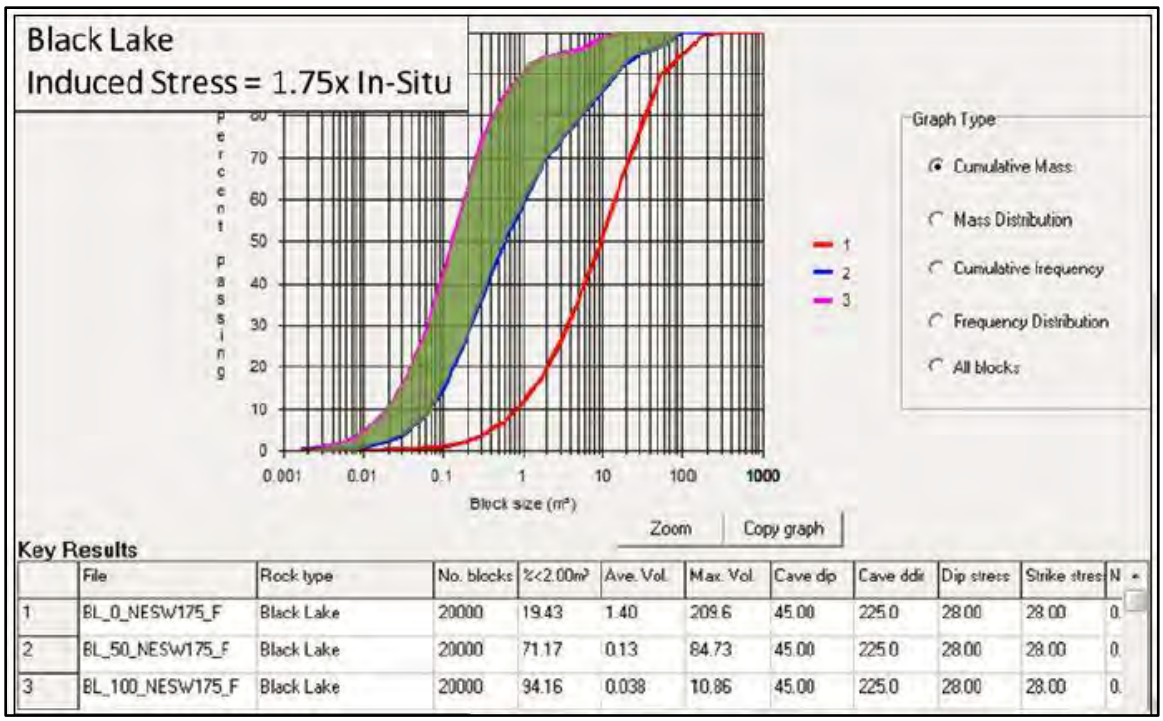


Figure 15.6: Black Lake primary fragmentation distribution for varying joint conditions.

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 1.50 factor was used to evaluate the effect of cave propagation direction on primary fragmentation. Table 15.4 lists 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 15.4: Primary fragmentation % passing 2 m^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 the NE-SW cave propagation direction, 50% artificial breaks, and 1.50x stress factor; for draw heights of 20, 50, 100, and 200 m, and draw rates of 10 cm/d and 20 cm/d. Figure 15.7 shows the results for 10 cm/d 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.

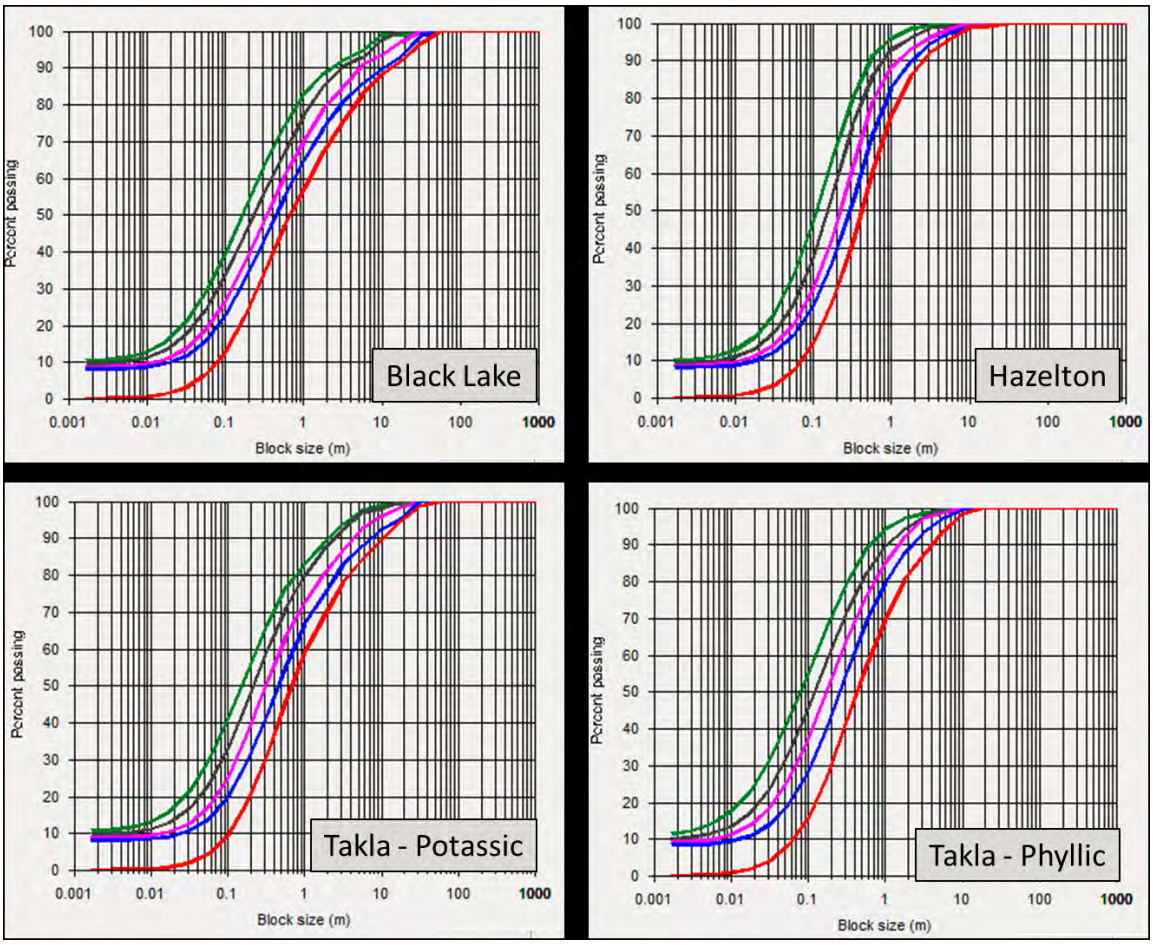


Figure 15.7: Secondary fragmentation distribution by geotechnical domain.

Given that the chosen cave propagation direction is E-W, actual fragmentation is expected to be represented by 50–100% artificial breaks and the stress factor is estimated to be 1.50–1.75x in situ stresses, Figure 15.7 is considered to represent a somewhat conservative view of secondary fragmentation.

Figure 15.7 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/d and 20 cm/d draw rates. The rate used in Gemcom's PCBC™ software to produce the final cave production schedule was 17 cm/d.

DFN numerical modelling evaluates 3D blocks caused by open joints, with Itasca's base case assuming a conservative fracture frequency of 0.5/m and primary fragmentation by joint extension

only. Results showed a primary fragmentation 50th percentile of 1.6–1.9 m equivalent diameter, which is considered an upper bound as fragmentation due to internal fracturing was not modelled.

SRM numerical modelling is the direct modelling of rock fragmentation due to the evolution of stresses within the cave, using PFC (Itasca 2007). Results showed a primary fragmentation 50th percentile of 0.8–1.6 m equivalent diameter, equating to 60–85% 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 70–85% 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 15.5 shows results for the 20 cm/d draw rate.

Table 15.5: Hang-up frequency and size by geotechnical domain.

20 cm/d draw rate	% Hang-up				Average size							
Draw height (m)	20	50	100	200	20		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	–

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 Gemcom PCBC™ software 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.

15.1.5 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 provide for main infrastructure excavations. The following is a summary of these recommendations.

Development headings outside the footprint, including both declines, where drift widths are 4–6 m, ground support will consist of:

- 2.4 m long fully-grouted rebar (22 mm) on 1.5 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 ±20% of the aforementioned development will experience poorer ground conditions which will require the following:

- Reduce rebar spacing to 1.2 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 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 four-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. It would consist of 5 m long full column grouted cable bolts on 2.0 x 2.0 m pattern throughout the passing bay. Initial cable bolts should be installed prior to slashing the passing bay.

Assuming good ground conditions crusher chamber support recommendations are:

- 3.0 m long fully-grouted rebar (22 mm) on 1.5 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,
- 75–100 mm shotcrete on back and walls,
- 9 m long full column grout cable bolts on 2.5 x 2.5 m pattern on back, and
- 6 m long full column grout cable bolts on 2.5 x 2.5 m pattern on walls.

Assuming fair quality rock mass along the length of the 5 m diameter intake and return air raises, these raises 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 40–60, representing 70% of development headings, and
- MRMR 30–40, representing 30% of development headings.

Table 15.6 shows the ground support recommendations for these two rock mass environments.

Table 15.6: 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 x 1.5 m pattern within 2 m 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 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 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	6 m long fully grouted cable bolts on 1.5 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 control 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.

15.1.6 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. 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.

15.2 Mud Rush Assessment

15.2.1 Introduction

The mud rush phenomenon can be described as ~~"a~~n 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 ~~"r~~apid 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 it will have the potential for hangups, voids and, in extreme cases, airblasts.

15.2.2 Industry Experience

Many caving and sublevel caving mines have experienced mud rushes and some have experienced airblasts.

The likelihood of airblasts can be reduced to acceptable levels relatively easily (with a ~~"n~~o void" policy and comprehensive monitoring procedure) and consequences could be, to a certain extent, minimized by leaving a sufficient thickness of muckpile above the loading points. The likelihood of a mud rush cannot be eliminated, only minimized by proper draw control, but the consequences of mud rush could be significantly reduced. There are mines that operate safely in high mud rush risk environments (e.g., Freeport), but proper standard operating procedures (SOP) and trigger Action response plans (TARP) have to be in place.

15.2.3 Mud rush and Airblast Risk Evaluation

In order to evaluate the risk of mud rush, 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 concluded 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 needs 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 10,000–15,000 m^2 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.

15.2.4 KUG Mud rush Risk Discussions

A simple qualitative risk analysis was conducted to identify the level of risk in the context of the expected mine life. The mine life was split into 3 time periods (Figure 15.8) and the results are summarized in Table 15.7.

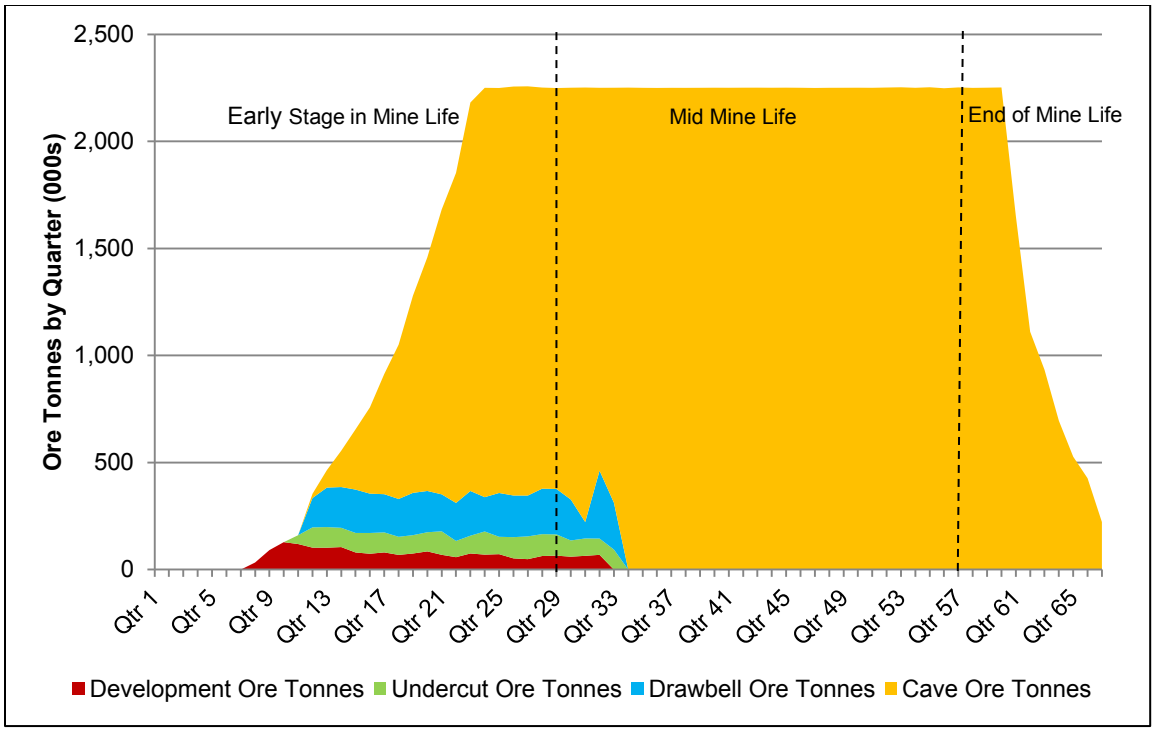


Figure 15.8: Kemess production schedule and mine life time periods.

In terms of mud rush, 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 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 mud rushes with time (Table 15.7).

Table 15.7: Summary of the mud rush risk assessment for Kemess.

Wet muck flow size	Level of risk		
	Early stage in mine life*	Mid mine life	End of mine life
Small (up to 50 m^3)	Moderate	Moderate	High
Medium (50–1,000 m^3)	Low	Moderate	Moderate
Large (more than 1,000 m^3)	Low	Moderate	Moderate

(*) This risk level is related to cave initiation, before the continuous caving occurs.

15.3 Cave Design

Gemcom PCBC™ software was used to compare alternative cave layouts, identify the optimal layout, and schedule cave production. This section summarises the comparison of alternative cave layouts and identification of the optimal layout.

The block model used for all Gemcom PCBC™ work was provided by AuRico. The 30 m (E-W) x 20 m (N-S) x 20 m (high) blocks were re-sized to 10 m x 10 m x 10 m blocks as smaller blocks are considered to provide improved mixing within the PCBC™ software.

One key constraint that was ultimately imposed on the final cave design was a maximum size of 100 Mt due to tailings storage limitations in the Kemess South pit. Without this constraint, an optimal cave size of ±140 Mt results.

15.3.1 Cave Layout Alternatives

Given the ±45° inclined nature of the KUG orebody along the KUG Fault, an inclined footprint layout was recommended for comparison against the flat footprint layout (base case). In addition, a hybrid footprint layout (i.e., part-inclined, part-flat) was also evaluated. This comparison was based on technical and economic parameters derived from the July 2010 PEA.

The PEA estimated a cost of $376k per drawpoint. Premiums of 33% and 15% were added to this cost for the inclined and hybrid footprint layouts respectively.

Initial Footprint Finder runs with no tonnage constraint showed the highest value hybrid layout to give a marginally higher total dollar value than the flat layout. The highest value inclined layout gave a marginally lower total dollar value than the flat layout. Best height of draw (BHOD) runs gave the same order: highest value for the hybrid layout followed by flat layout, although the relative differences were greater.

The flat and hybrid layouts were then compared with the 100 Mt constraint applied and the cave scheduled to progress from northeast to southwest. This also resulted in a higher value hybrid layout although the difference was once again marginal compared to the flat layout. Given this marginal value difference, the flat layout was adopted for the KUG FS. One further reason for this decision is that the increased development associated with the hybrid layout introduces the possibility of a longer ramp up in cave production, particularly as much of the additional development would need to occur in the fault zone area.

15.3.2 Optimised Cave Layout

For the flat layout, drawpoint spacing was reduced to 15 x 15 m compared to the 15 x 18 m used for the PEA as a result of SRK fragmentation assessment. For this drawpoint spacing, it is assumed that interaction between drawpoints occurs directly above the undercut.

A mining cost and shut-off value sensitivity analysis was carried out to (a) quantify the impact on overall cave size and value with increasing mining cost, and (b) identify the combination of mining cost and shut-off value that maximises cave value. Mining costs of $14.30/t, $15.30/t, and $16.30/t were used along with shut-off values that equalled the mining cost as well as $1.00/t and $2.00/t increments. (In PCBC™ terminology, mining cost includes OPEX for mining, processing, and G&A.)

All three scenarios result in caves with at least 100 Mt where mining cost equals shut-off value, showing that some degree of opex increase will not dramatically reduce cave size. However, at the higher mining costs, cave tonnages are below 100 Mt where the shut-off value exceeds mining cost and total cave values are reduced; if higher mining costs are experienced, a shut-off value equal to mining cost should be adopted.

Different tunnel orientations were also evaluated. Comparison of 160° and 180° azimuth extraction drives were considered, where the 160° orientation is a carry-over from the inclined and hybrid layouts. The 180° extraction drive orientation layout results in a marginally higher cave value and was adopted for the KUG FS. In addition, the 180° orientation layout gives a more favourable footprint shape (i.e., with regards to cavability).

Given the aforementioned outcomes, the elevation for the footprint that produces the highest cave value was re-evaluated: 1,120, 1,140, and 1,160 m elevations were considered. This comparison also allowed for the drawbell and undercut geometries recommendationed by SRK, with the height difference between extraction and undercut levels being 18 m and the undercut being 18 m high (from the bottom of the undercut drifts). This comparison resulted in the 1,160 m elevation being chosen for the extraction level, with the undercut level is at the 1,178 m elevation.

In order to finalise the cave design, updated economic inputs were provided. These are discussed in more detail in Section 15.5.3 (Production). The end result of these inputs is the following formula:

$$NSR = [45.16 \times Cu\ grade\ (\%)] + [28.07 \times Au\ grade\ (g/t)]$$

Block values were updated using this formula and three Footprint Finder runs were completed in order to verify that the optimal location for the extraction level is the 1,160 elevation. For the tonnage constrained runs, the optimal extraction level elevation was also confirmed at 1,160.

A BHOD analysis was carried out using shut-off values of $15.30/t–$18.30/t. As expected, increasing shut-off values resulted in decreased draw column heights and an increased number of drawpoints to maintain the 100 Mt cave tonnage target. The highest value cave occurred at a shut-off value of $17.30/t. This footprint shape was then smoothed to remove any potential for cave stall. This resulted in a footprint having 640 drawpoints over 20 extraction drives, which is shown in Figure 15.9 and represents the final footprint design.

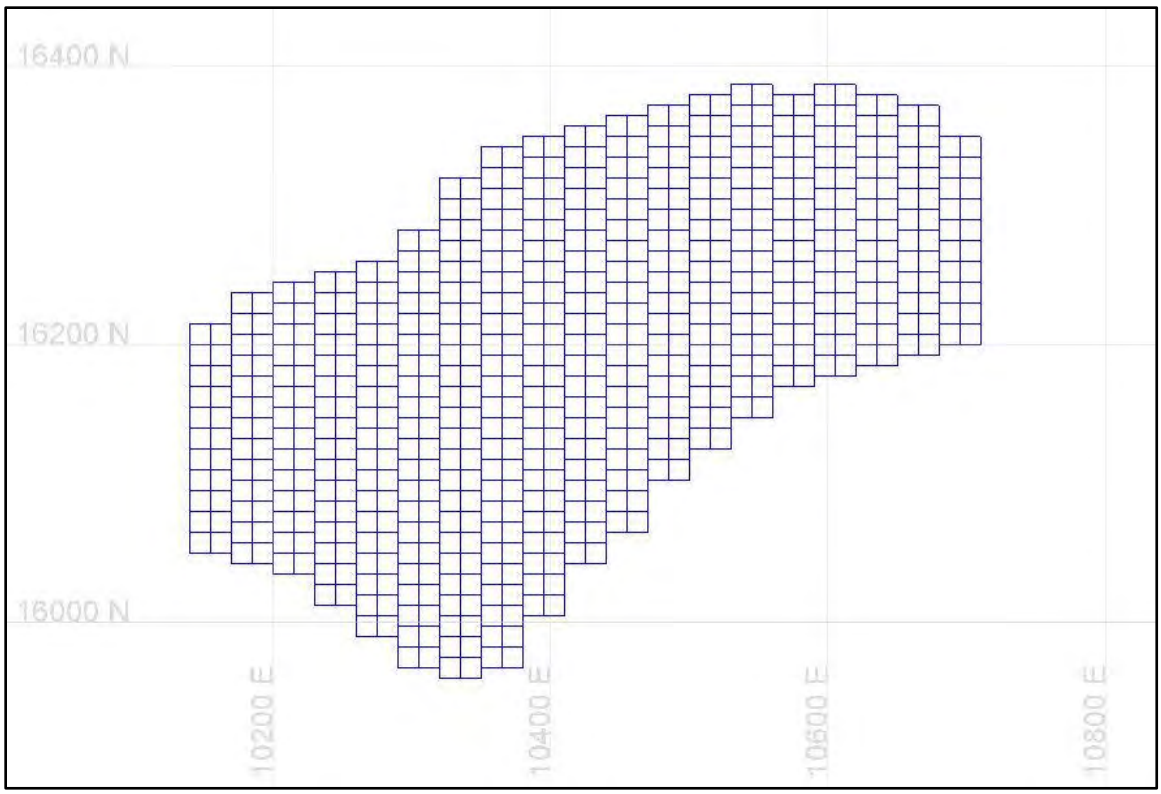


Figure 15.9: KUG footprint.

Various production schedules were then run to identify the highest value cave mining sequence. The key outcome of this exercise was that cave production should start in the east side of the cave as the highest value ore is at this side of the cave. Only marginal value differences result 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 15.5.3 (Production).

Figure 15.10 shows an orthogonal view of the final cave design looking northwest. 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. Draw columns range in height from 150 to 350 m.

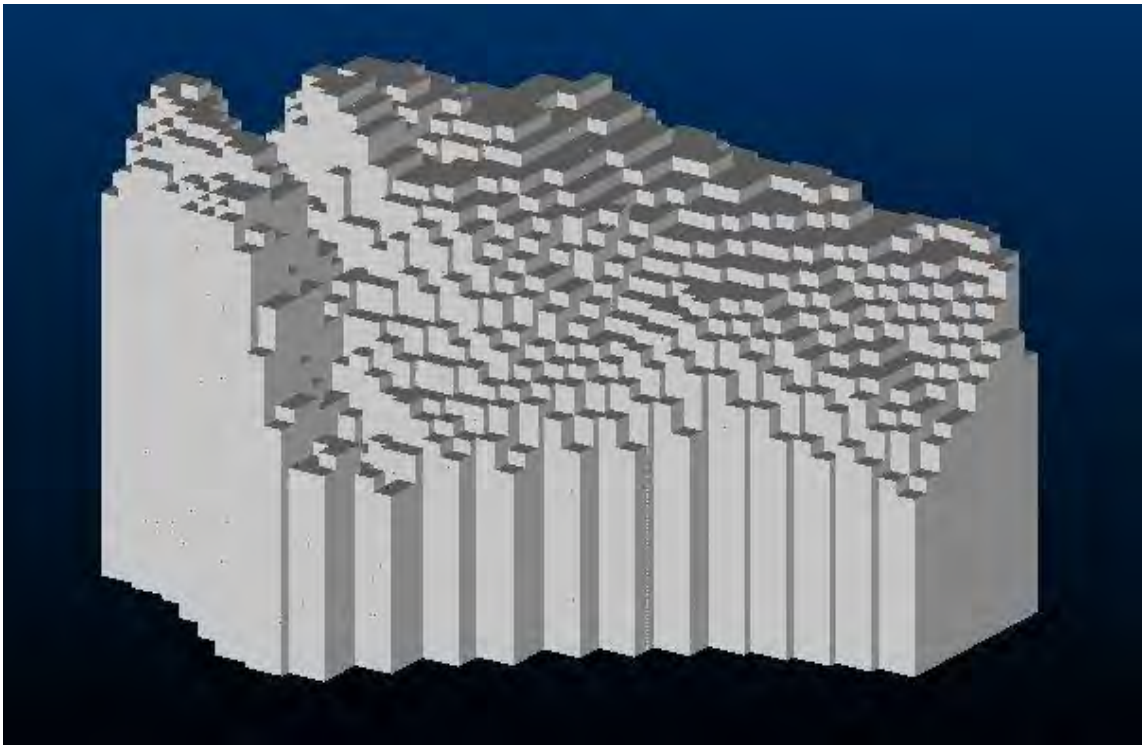


Figure 15.10: View of KUG cave, looking northwest.

This cave design results in dilution of 466 kt of sulphate leach zone and 5,100 kt of Hazelton domain material. This represents 5.9% of cave tonnage excluding undercut and drawbell tonnes or 5.5% of cave tonnage including undercut and drawbell tonnes. Effort will be made to remove the Hazelton domain dilution from the ore stream during operations.

15.4 Mine Access

This section focuses on the surface area immediately around the twin portals as well as the twin decline system.

AMEC Environment & Infrastructure (Burnaby, BC) has provided a design, schedule, and cost estimate for the 4,500 m access corridor from the existing Kemess facilities to the planned twin 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 (i.e., to transport ore from KUG to the existing processing facility), 13.8 kV electrical power supply to KUG, and 8" dewatering pipeline from KUG. While this corridor provides primary access to the KUG, an alternative access/egress is available via an existing exploration road that traverses the hill directly north of the Kemess South open pit.

Given the relatively steep topography, the access road and conveyors follow separate routes approaching the portal area. In order to keep within a maximum 10% gradient, the access road requires three switchbacks to traverse from the south to north side of the Kemess Lake Valley. The minimum road width has been reduced to 10 m in this area, from the standard 12 m, to reduce cut-and-fill requirements; mobile equipment operations along this section of road are expected to be safe as a result of the reduced travel speeds due to the switchbacks and

gradients. There are two locations where the access road crosses beneath the elevated conveyor, having clearances of 4.5 m (CV01) and 9.0 m (CV02) (Figure 15.11).

Portal area facilities include the following:

- Development construction contractor offices, workshop, stores,
- Batch plant,
- Fans and air heater for twin decline system,
- Waste and ore stockpiles,
- Laydown area,
- Electrical substation,
- Water handling,
- Stripped organics stockpiles, and
- Ventilation raises access road.

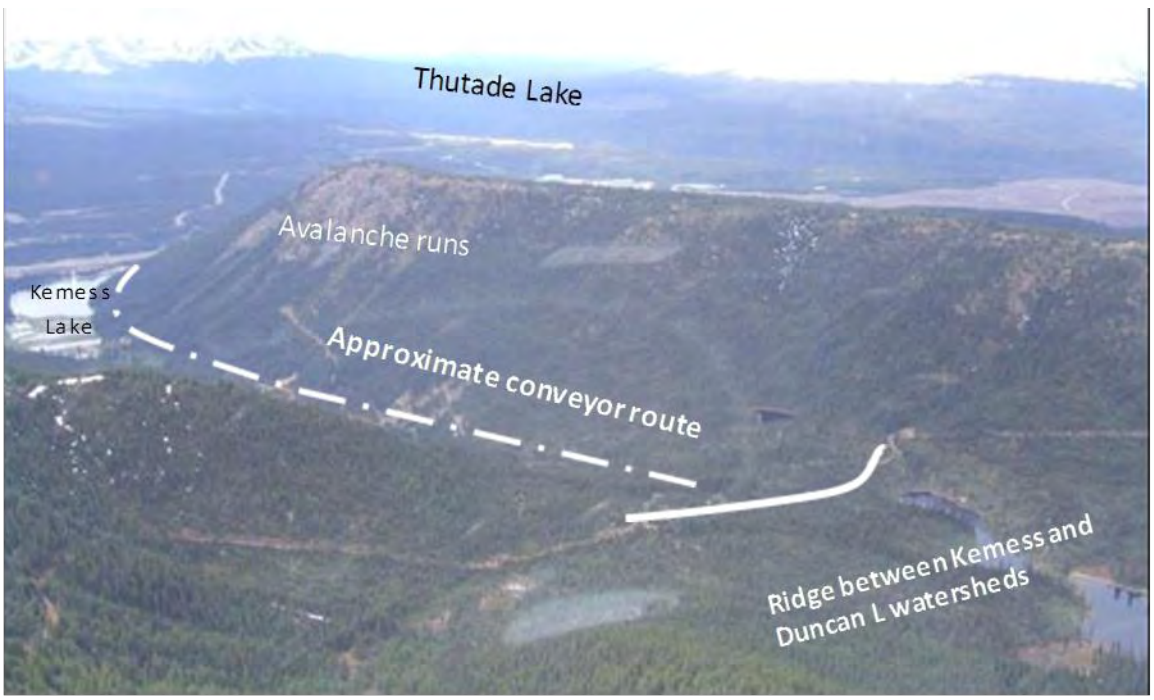


Figure 15.11: Road access in the Kemess Lake Valley showing previous avalanche runs.

The development construction contractor is expected to mobilise to site and establish portal area facilities in Q3 Year –5. The mine development schedule has their scope occurring over the period Q4 Year –5 to Q2 Year –1, a period of 15 quarters. It is expected that the contractor will establish all office, workshop, and stores facilities at the portal area required to support their underground scope.

The batch plant requires an area approximately 30 x 20 m and is located in close proximity to the access decline for ready transport of concrete underground by a transmixer truck. This plant has a capacity of 90 m^3/h of concrete.

Twin exhaust fans located at the conveyor decline portal will ventilate the declines system. The propane intake air heater along with the propane tank will be located directly between the twin portals. 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 70 kt and 90 kt respectively, which is in excess of peak stockpiling requirements of 1,785 t/d and 1,623 t/d respectively. Removal of material from these stockpiles will occur on the south side to limit traffic in the vicinity of the portals.

A general laydown area (#1) is provided a short distance from 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 centrally located to provide electrical power to portal area facilities, conveyor head and tail motors, and the underground (KUG). A transformer is required to change the 25 kV supply to 13.8 kV for use underground. The 25 kV supply will be by armoured cable running on the conveyor structure.

Water handling facilities include the 200 mm dewatering pipeline from underground (which runs up the conveyor decline), sedimentation pond, pump house (for discharge of water to the Kemess South open pit), runoff collection ditch, and culverts. The drainage ditch is designed to handle a 1:200 year, 24 hour event. The sedimentation pond is 100 m long x 20 m wide x 1.5 m deep, and can handle a 1:20 year, 24 hour event. Both the ditch and pond will be lined. All portal facilities will be located in the Kemess Lake watershed.

Two stripped organics stockpiles will be maintained in the portal area for remediation upon mine closure.

A 6,150 m long access road to the primary intake and return ventilation raises is planned to be constructed.It is an extension of the 1.5 km road to the Portal entrance. This road has been designed to have a maximum 10% grade to allow year round delivery of propane to the intake air heater. The road width is 6 m allowing single lane traffic only; passing bays will be provided periodically.

The access decline (5 mW x 5 mH) is located 30 m east of the conveyor decline (4.5 mW x 4.5 mH), and both declines are inclined at –8.3%. Declines connecting cross cuts (5 mH x 5 mW) are provided every 150 m along the length of the declines to facilitate initial development by providing an effective ventilation network and storage areas, resulting in a total of 21 cross cuts. The elevation of the portals is 1,390 m and the elevation of the bottom of the declines is approximately 1,130 m, an elevation change of 260 m.

The access decline will be used for transport of all personnel, materials, 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 10 passing bays. Given that

the conveyor 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 1,142 m elevation. The access decline also initially connects to the intake ventilation level at the 1,170 m elevation. The conveyor decline initially connects to the return ventilation level at the 1,169 m elevation.

There are two temporary electrical substations provided for in cross cuts 7 and 14, and the main electrical substation in cross cut 21.

There is a fuel storage bay in cross cut 20, fed from the surface via a pipeline installed in the conveyor decline.

There is a truck loadout at the bottom of the access decline, with the tip located on the extraction (1,160 m) level, providing approximately 200 t storage capacity. This equates to approximately 25% of the peak waste handling requirement per shift or 1,600 t/d. It is also possible to use this pass for ore handling providing ore is batched; there are four quarters in the mining schedule when this may be required, ahead of ore handling via the conveyor system.

15.4.1 Extraction Level

The extraction level is located at the 1,160 m elevation and is accessed via the access decline at the 1,142 m elevation, and is shown on Figure 15.13. From this main access, there are accesses both east and west of the primary crusher chamber. There are 20 extraction drives (5 mW x 4.5 mH) on 30 m centreline spacing providing access to 640 drawpoints (4.5 mW x 4.5 mH) or 320 drawbells, with caved ore being fed via the drawbells from above the undercut level. Since all ore is trammed using LHDs to the primary gyratory crusher, there will be significant LHD traffic on the extraction level. As such, inner and outer rim tunnels (5 mW x 5 mH) are provided to the south of the cave footprint to reduce LHD congestion. There are also passing bays situated on the east and west crusher access drifts to help in this regard.

The primary gyratory crusher is located south of the cave footprint between extraction drives 12 and 13. Generally, the east crusher tip is served by extraction drives 1–9, the central crusher tip by extraction drives 10–15, and the west crusher tip by extraction drives 16–20. The optimal location for the crusher is considered to be between drives 10 and 11, and this will be investigated further following the FS.

East, west, and north side rim tunnels (5 mW x 5 mH) are provided for 360° access to the cave footprint, with all non-LHD traffic using these for primary access, including supervisory, secondary breakage, maintenance, technical services, rehabilitation, and construction traffic. These rim tunnels can be accessed via the undercut level without having to travel along the southern inner or outer rim tunnels 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.

The main workshop is located on this level in close proximity to the level access; this facility is described in Section 4.7.5. 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 except production LHDs. Production LHD' will be re-fuelled at fuel bays

located both east and west on the outer rim tunnel. 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 air raises (2 m diameter) located at the north end of each extraction drive. Air is routed to the return air level via 10 return air raises (3 m diameter) situated just south of the south side outer rim tunnel.

Drawbell dimensions were provided by SRK based on consideration of fragmentation and stability of workings and are shown by Figure 15.12. 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 2.125 Mt associated with all 320 drawbells.

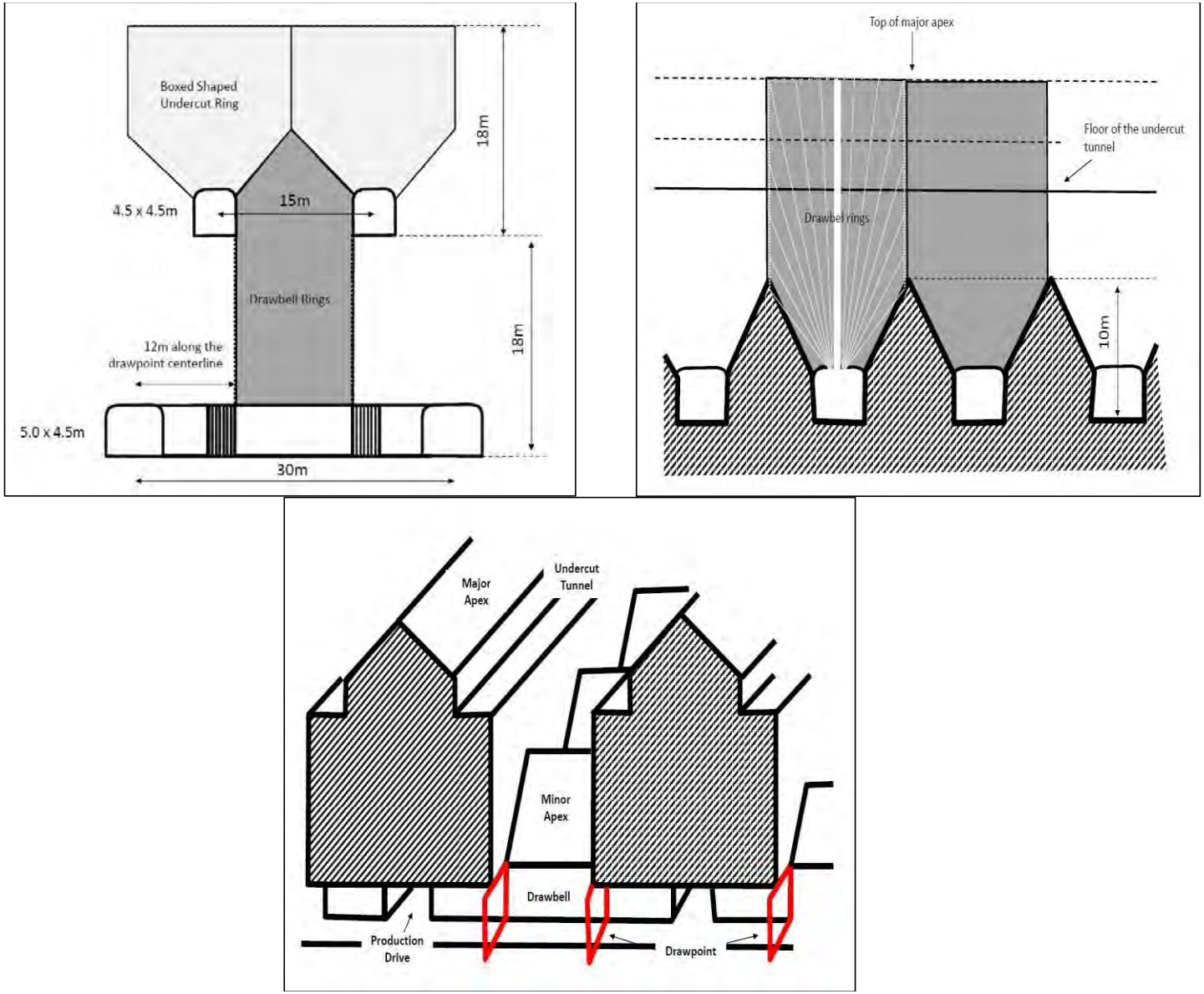


Figure 15.12: Drawbell geometry.

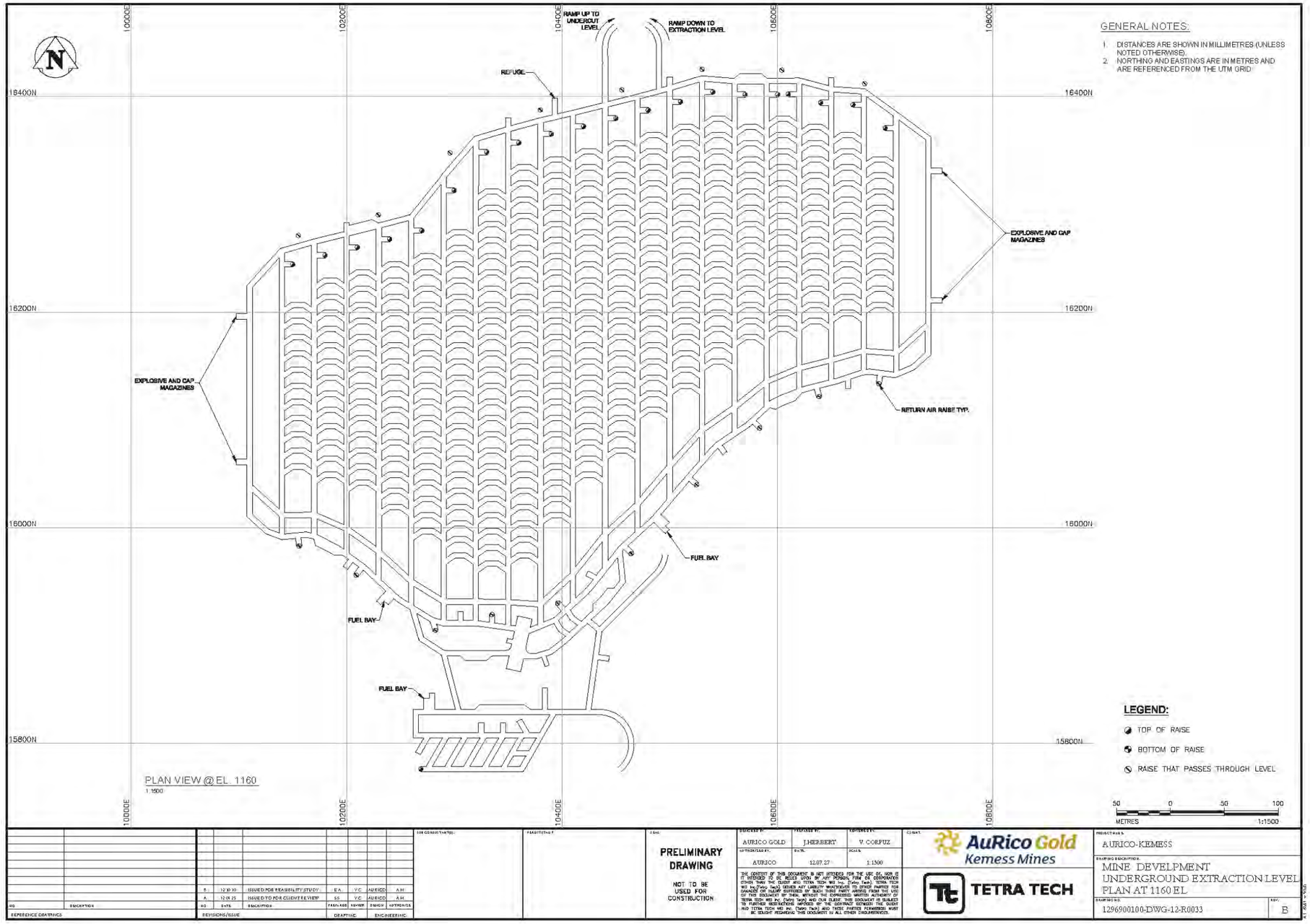


Figure 15.13: Extraction level layout.

15.4.2 Undercut Level

The undercut level is located at the 1,178 m elevation and is accessed via ramps from the extraction level. 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 mW x 5 mH) provide 360° access to the cave footprint. In addition to these drifts providing access for development and undercutting personnel equipment, they also allow other personnel equipment to avoid the parts of the extraction level utilised by production LHDs.

Ventilation to this level is provided via eight intake air raises (2 m diameter) located along the north rim tunnel. Air is routed to the return air level via ten return air raises (3 m diameter) situated just south of the south rim tunnel; these raises also serving the extraction level.

The upper left graphic on Figure 15.12 shows the undercut would be taken to a height of 18 m above the floor of the undercut level. This is relatively high and is recommended to avoid potential oversize problems associated with the low stress environment at KUG. There is approximately 12,850 t of undercut associated with each drawbell, totalling 4.110 Mt associated with all 320 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.

15.4.3 Ventilation Level

The ventilation level is located at the 1,140 m elevation (ungraded). However, as this level will also serve as the main drainage level, the low point will be at 1,126 m elevation 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 access decline by two cross cuts (4.5 mW x 4.5 mH) that provide access to the bottom of the primary intake air raise (720 m long, 5 m diameter), primary intake air fans, and intake air level. The two primary intake air fans are located in parallel drifts (8 mW x 8 mH), which then combine to transport intake air to the north side of the cave footprint. Intake air is delivered to the undercut level via eight intake air raises (2 m diameter) and to the extraction level via 20 intake air raises (2 m diameter). The ventilation level drift is 6 mW x 5 mH.

The return air level is initially accessed from the conveyor decline by two cross-cuts (4.5 mW x 4.5 mH) that provide access to the bottom of the primary return air raise (720 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 mW x 8 mH). The return air level drift that feeds the primary return air fans is 6 mW x 5 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 m diameter) that connect to both the undercut and extraction levels.

With overbreak, the section of the intake and return air levels immediately adjacent to the primary fans (i.e., on the opposite side to the primary air raises), will be 6 mW x 6 mH to accommodate higher air velocities that would be experienced in these sections.

The main dewatering facility is located at the low point (1,126 m elevation), which is south of the cave footprint and in close proximity to the primary crusher and workshop. This facility comprises

two sump chambers and a pump chamber, which are accessed from the conveyor decline. A bulkhead will be installed where the intake and return air levels meet, southeast of the cave footprint, to allow for transfer of drainage water and prevent ventilation short-circuiting.

15.5 Mine Schedule

This section discusses all aspects of the mine schedule, including development, construction, and production.

The combined development-production schedule was compiled using EPS (Enhanced Production Scheduler) software, a MineRP (formerly GijimaAST) product. However, the cave production schedule was developed using Gemcom's PCBC™ software, which is recognised as an industry standard for cave production scheduling. PCBC™ schedule data transferred to EPS employed a PCBC™-EPS data transfer plug-in developed by MineRP.

Prior to commercial production being declared, schedule data will be reported by quarter, and annually thereafter. Commercial production is defined as equivalent annual production being at least 60% of target production, with target production being 9.0 Mtpa or 25 ktpd based on a 360 day per year operation.

15.5.1 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 situations were recognised to ensure a realistic development schedule. Development is assumed to start in Q4 (October 1) Year –5 in order to allow construction of the access road, establishment of portal facilities, and portal construction in the spring and summer months ahead of development.

There is a total of 47,473 m of lateral development, which includes a variety of standard heading sizes as well as major excavations. Based on the stated heading sizes and excavation sizes, lateral development equates to 2.893 Mt. For trucking calculations, 10% overbreak allowance was added to design lateral development tonnages.

Initial access to KUG is via twin declines, one an access decline and the other a conveyor decline. The development rate over the initial six quarters averages 322 m/mo, 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 along their length), re-mucks, and passing bays will allow each jumbo to average 161 m/mo. It is planned that a development contractor will be utilised for initial development.

In quarter seven (i.e., Q2 Year –3), the average monthly development increases to 453 m, requiring a third jumbo to be utilised, operated by owner personnel. In this quarter, development of the intake and return air levels commences, effectively providing two additional high priority headings.

In quarter eight (i.e., Q3 Year –3), the average monthly development rate increases to approximately 720 m/mo, requiring a fourth jumbo to be utilised, operated by owner personnel. Each jumbo is now required to average 180 m/mo, with this requirement continuing through quarter 15 (i.e., Q2 Year –1). In addition to continuation of the twin decline and ventilation level development, quarter eight sees the development from the access decline to the extraction

(1,160 elevation) level. The latter part of this quarter sees the initial development of the main dewatering, workshop, and crusher chamber excavations.

The bulk excavation component of the lateral development is expected to be completed by development contractor personnel using specialist equipment where necessary. The following bulk excavations are developed from quarter eight:

- Excavation of the crusher chamber occurs over quarters 8–10, with completion in mid-quarter 10 (i.e., Q1 Year –2); allowing 4–5 months for crusher installation ahead of cave production starting in quarter 12 (i.e., Q3 Year –2);
- The main dewatering infrastructure, comprising twin sumps and a pump chamber, is excavated in quarters 8–9, with construction installation occurring immediately thereafter; and
- The workshop is situated on the extraction level and is excavated in quarters 8–9. Again with construction installation occurring immediately thereafter.

As can be seen, the period comprising quarters 8–10 has intensive bulk excavation requirements with an intense period of infrastructure construction installation in the following 3–4 months. This period also sees the main primary air raises completed, with the primary intake air raise developed over quarters 7–8 and the primary return air raise developed over quarters 9–10. Primary fan installation will occur in quarters 8–9.

As previously mentioned, access to the extraction level is developed in quarter eight. In quarter nine, development of inner and outer rim tunnels on the extraction level commences. In addition, access to the undercut level is developed in quarter nine. Quarter 10 (i.e., Q1 Year –2) sees the initial undercut level drift development followed by extraction level drift and drawpoint development in quarter 11. 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 quarter 10 (i.e., to the southeast of the footprint), thereby providing through ventilation to the undercut and extraction levels.

From quarter 16 (Q3 Year –1), average monthly development decreases such that three jumbos are needed through Year 2; all three jumbos are planned to be owner operated. Over this period, average monthly development equates to 163 m/mo per jumbo. As expected, the vast majority of lateral development over this period is within the footprint of the cave.

Year 3 has a total 4,878 m lateral development, which may be possible to develop with a fleet of two jumbos, equating to 203 m/mo per jumbo. This rate continues into Year 4, with the remaining 3,525 m developed over approximately nine months.

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 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 Gemcom's PCBC™ cave scheduling software.

Figure 15.14 shows the contractor and owner lateral development metres by quarter.

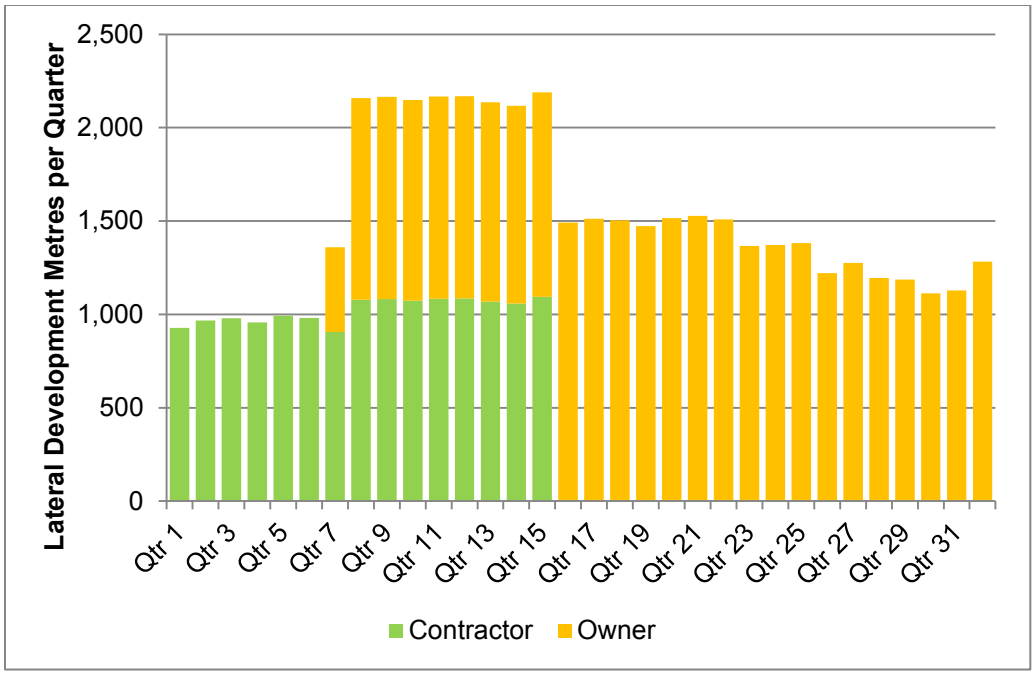


Figure 15.14: Contractor and owner lateral development by quarter.

A total of 15,120 m of lateral development is assigned to the contractor and 32,353 m assigned to owner development crews.

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 twin 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/d or 135 m/mo. It is of critical importance to the mining schedule that these headings are prioritised every shift, with all these headings considered critical path activities.

Intermediate priority headings include twin decline cross cuts, initial extraction drives, and initial undercut drives, all of which are scheduled at 3.0 m/d. 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. Extraction drives 1-5, and undercut drives 1W to 5E-5W are covered in this group.

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 are scheduled at 2.0 m/d.

Bulk excavation is scheduled at 100 m^3/d, with the crusher chamber, dewatering infrastructure, and workshop scheduled using this rate. For a 6 x 6 m heading this rate equates to an advance rate of less than 3.0 m/d.

15.5.2 Raise Development

There is a total 2,444 m of raise development comprising raisebore (5 m diameter), drop raise (2–3 m diameter), and Alimak raise (3 x 3 m). Based on the stated heading and excavation sizes,

raise development equates to 98 kt. For trucking calculations, 10% overbreak allowance is added to design raise development tonnages.

As previously mentioned, development of the primary ventilation raises occur over quarters 7–10, with the intake raise scheduled for development ahead of the return raise. An opportunity exists to bring this development forwards by one quarter by piloting both raises ahead of reaming the first (intake) raise. This is considered likely in order to ensure the main primary ventilation circuit is available in quarter 10 when the intake and return air levels converge in the southeast area of the footprint. Installation of the primary intake and return fans can occur in quarters eight and nine as the ventilation level layout (on intake and return air sides) allows this to occur independently of ongoing raise and ventilation level development.

There are 20 x 16 m long intake air raises planned between the intake air level and extraction level, and 8 x 34 m long intake air raises planned between the intake air and undercut levels. In addition, there are 10 x 34 m long return air raises planned between the undercut extraction levels and return air level. The 8 x 34 m long intake air raises will be developed either by Alimak or two leg drop raise. The remainder are planned to be developed by drop raise, with the 10 x 34 m long return air raises planned to be developed in two legs. These raises are developed as the various levels reach the location of each raise, resulting in raise development occurring in all years from Year –3 to Year 4.

Provision has also been made for waste and ore passes between the undercut and extraction levels with a single truck loading chute located at the bottom of the underlying access decline. The current development schedule has the ore pass developed in quarter 16 (i.e., Q3 Year –1) and waste pass developed in Year 1. However, to be of use they need to be developed in quarter nine or ten because of the following:

- Ore crushing conveying commences in quarter 12, after which the ore pass is only likely to be needed to transfer ore between the undercut and extraction levels, a total of approximately 2 Mt; and
- There is an ongoing need to transfer waste from both extraction and undercut levels to the underlying access decline to facilitate efficient truck haulage, a total of approximately 1.3 Mt after quarter 10.

Requirements for internal ore and waste passes, and the timing of their development, will be re-visited following completion of the FS.

Development rates are assigned to raise development activities based on practical advance rates.

The primary intake and return air raises from the surface are scheduled at an overall average of 4 m/d for piloting and reaming. Resulting in a duration of approximately six months for each of the 720 m long raises. These activities are critical path raising activities as their 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/d. These activities are not critical path raising activities.

15.5.3 Production

In the context of KUG, production is primarily considered to comprise undercutting, drawbell opening, and cave production (i.e., above the undercut). Ore is also derived from development, with tonnages associated with each source listed below in Table 15.8, Table 15.9 and Table 15.10:

Total ore tonnes associated with the KUG FS is 100.4 Mt.

It should be noted that 50% of the 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 2,338 kt waste comprising 1,300 kt development, 297 kt undercutting, and 741 kt in drawbells.

The cutoff between development ore and waste was set at $8.40/t NSR, which is the total cost associated with processing ($5.30/t) and G&A ($3.10/t) from the July 2011 PEA. As such, all development, undercutting, and drawbell opening activities had NSR values calculated using the following formula:

$$NSR = [45.16 \times Cu\ grade\ (\%)] + [28.07 \times Au\ grade\ (g/t)]$$

The 45.16 value applied to the Cu grade was derived using 91% Cu recovery, revenue of US$3.00/lb Cu, 96.5% payable Cu, 22% Cu in concentrate, US$70.00/t concentrate treatment charge, US$0.07/lb Cu refining charge, US$193.00/t concentrate transport charge, and 8% moisture content.

The $28.07 value applied to the Au grade was derived using 72% Au recovery, revenue of US$1,250.00/oz Au, 97.5% payable Au, and US$4.39/oz Au refining charge. There are no associated transport and treatment charges as these are carried by the copper.

The NSR calculation will be reviewed following the FS to account for any changes in concentrate transport, treatment, and refining charges.

It is noted that these values were changed in the financial analysis. The financial analysis used a concentrate transportation charge of $232.16 per dry meter tonne, a refining cost of US$6.00/oz, and a gold price of US$1,300/oz.

Table 15.8, Table 15.9 and Table 15.10 summarise the production schedule.

Table 15.8: Pre-commercial production schedule by quarter.

Item	Total	Qtr 1	Qtr 2	Qtr 3	Qtr 4	Qtr 5	Qtr 6	Qtr 7	Qtr 8	Qtr 9	Qtr 10	Qtr 11	Qtr 12	Qtr 13	Qtr 14	Qtr 15	Qtr 16	Qtr 17
		Q4 Year – 5	Q1 Year – 4	Q2 Year – 4	Q3 Year – 4	Q4 Year – 4	Q1 Year – 3	Q2 Year – 3	Q3 Year – 3	Q4 Year – 3	Q1 Year – 2	Q2 Year – 2	Q3 Year – 2	Q4 Year – 2	Q1 Year – 1	Q2 Year – 1	Q3 Year – 1	Q4 Year –1
Undercut ore tonnes	603,983											41,198	94,630	96,273	89,964	91,240	97,101	93,578
Drawbell ore tonnes	1,076,369												137,183	184,347	190,358	202,421	183,698	178,362
Development ore tonnes	830,346								24,494	82,542	119,703	110,773	93,957	93,721	96,640	71,401	65,418	71,697
Cave ore tonnes	1,515,661												19,633	81,265	169,443	281,471	402,700	561,149
Total ore tonnes	4,026,359								24,494	82,542	119,703	151,970	345,403	455,606	546,406	646,533	748,917	904,785
Undercut waste tonnes	99,714											7,041	10,600	7,093	20,384	20,225	15,565	18,805
Drawbell waste tonnes	244,077												26,501	43,106	35,491	31,294	51,279	56,406
Development waste tonnes	1,002,413	57,059	59,521	60,060	58,711	61,065	60,179	109,862	154,694	114,568	61,265	22,775	31,148	28,688	24,110	61,513	20,794	16,400
Total waste tonnes	1,346,203	57,059	59,521	60,060	58,711	61,065	60,179	109,862	154,694	114,568	61,265	29,815	68,249	78,887	79,986	113,033	87,638	91,611
Total tonnes	5,372,563	57,059	59,521	60,060	58,711	61,065	60,179	109,862	179,188	197,110	180,967	181,786	413,652	534,493	626,392	759,566	836,555	996,396
Cu grade (%)	0.34								0.14	0.16	0.18	0.24	0.32	0.33	0.35	0.37	0.37	0.37
Cu (t)	13,771								35	132	217	363	1,092	1,491	1,899	2,369	2,785	3,388
Au grade (g/t)	0.69								0.27	0.25	0.30	0.41	0.58	0.63	0.70	0.75	0.78	0.79
Au (oz)	89,389								209	667	1,136	2,025	6,414	9,286	12,288	15,630	18,735	23,000
Ag grade (g/t)	2.06								0.95	1.06	1.32	1.64	2.06	2.02	2.08	2.14	2.16	2.20
Ag (oz)	266,500								748	2,817	5,070	8,016	22,862	29,644	36,617	44,513	52,082	64,131
Total ore NSR ($/t)	34.8								14.0	14.3	16.5	22.4	30.5	32.6	35.3	37.7	38.6	39.1

Table 15.9: Annual commercial production schedule

Item	Total	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Undercut ore tonnes	1,451,305	369,541	348,193	406,820	326,752								
Drawbell ore tonnes	3,104,108	739,607	750,983	809,886	803,632								
Development ore tonnes	859,388	263,621	241,027	195,414	159,327								
Cave ore tonnes	90,932,306	4,062,465	7,159,300	7,567,162	7,678,592	9,001,078	9,001,316	9,003,067	9,000,678	9,005,076	9,003,157	8,405,710	2,044,706
Total ore tonnes	96,347,107	5,435,233	8,499,502	8,979,282	8,968,303	9,001,078	9,001,316	9,003,067	9,000,678	9,005,076	9,003,157	8,405,710	2,044,706
Undercut waste tonnes	196,944	74,898	89,877	28,593	3,576								
Drawbell waste tonnes	497,278	189,758	169,081	102,852	35,587								
Development waste tonnes	298,284	83,404	84,965	88,613	41,301								
Total waste tonnes	992,506	348,060	343,923	220,058	80,465								
Total tonnes	97,339,613	5,783,293	8,843,425	9,199,339	9,048,767	9,001,078	9,001,316	9,003,067	9,000,678	9,005,076	9,003,157	8,405,710	2,044,706
Cu grade (%)	0.28	0.37	0.35	0.32	0.31	0.29	0.27	0.26	0.25	0.24	0.23	0.22	0.21
Cu (t)	267,105	19,896	29,347	28,994	28,078	25,848	23,876	23,035	22,493	21,689	21,011	18,503	4,335
Au grade (g/t)	0.55	0.80	0.76	0.69	0.63	0.59	0.53	0.50	0.47	0.45	0.42	0.40	0.38
Au (oz)	1,715,767	139,761	208,525	198,044	180,559	171,673	153,072	143,950	136,205	128,968	122,471	107,715	24,823
Ag grade (g/t)	2.05	2.27	2.33	2.35	2.31	2.24	2.09	2.01	1.94	1.83	1.68	1.65	1.63
Ag (oz)	6,345,007	396,255	637,710	677,068	667,372	648,745	605,102	581,650	561,383	529,522	486,662	446,377	107,163
Total ore NSR ($/t)	28.1	39.0	37.0	33.8	31.7	29.6	26.8	25.5	24.5	23.4	22.4	21.1	20.2

Table 15.10: Life of mine summary production.

	Life of Mine Total
Undercut ore tonnes	2,055,288
Drawbell ore tonnes	4,180,477
Development ore tonnes	1,689,734
Cave ore tonnes	92,447,967
Total ore tonnes	100,373,466
Undercut waste tonnes	296,658
Drawbell waste tonnes	741,355
Development waste tonnes	1,300,696
Total waste tonnes	2,338,709
Total tonnes	102,712,175
Cu grade (%)	0.2798
Cu (t)	280,876
Au grade (g/t)	0.56
Au (oz)	1,805,156
Ag grade (g/t)	2.05
Ag (oz)	6,611,507
Total ore NSR ($/t)	28.34

Figure 15.15 shows the mix of ore tonnes by period. This graph clearly shows the significance of cave ore tonnes to KUG, representing 92% of total ore tonnes. Development, undercutting, and drawbell ore tonnes represent relatively minor ore proportions, with their significance greater during production ramp up.

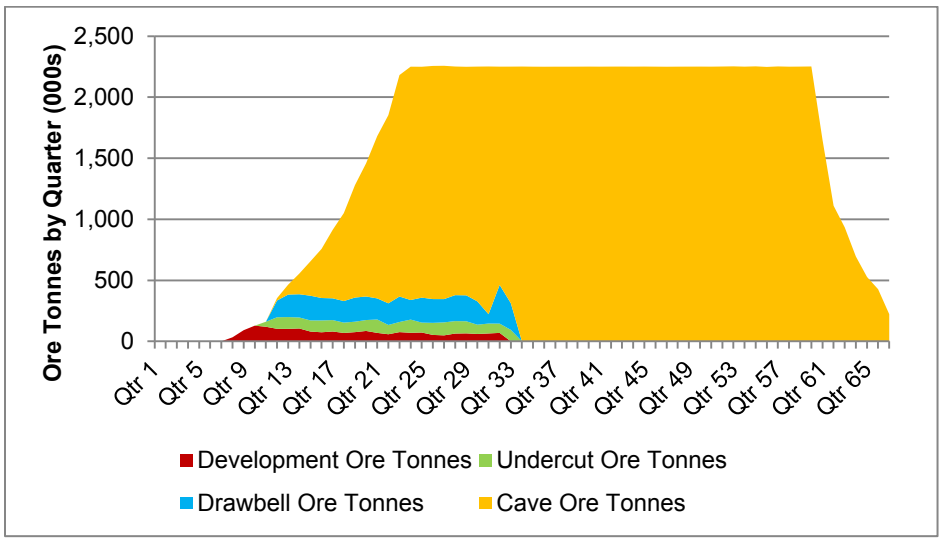


Figure 15.15: Ore tonnes (000's) by quarter.

Tonnes and grades associated with undercutting and drawbell opening were derived from evaluating solids representing undercutting and drawbells associated with each of the 640 drawpoints (or 320 drawbells) against the KUG block model. As such, this volume of rock was

excluded from the cave production schedule runs using Gemcom's PCBC™ software. PCBC™ was then run against the optimised cave volume above the 1,196 elevation.

The Gemcom PCBC™ schedule is based on five drawbells, accessed by 10 drawpoints, opened each month. The sequence of drawbell opening is from east to west between extraction drifts and from north to south 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 15.16. 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 $40/t in quarter 18 (i.e., Q1 Year 1).

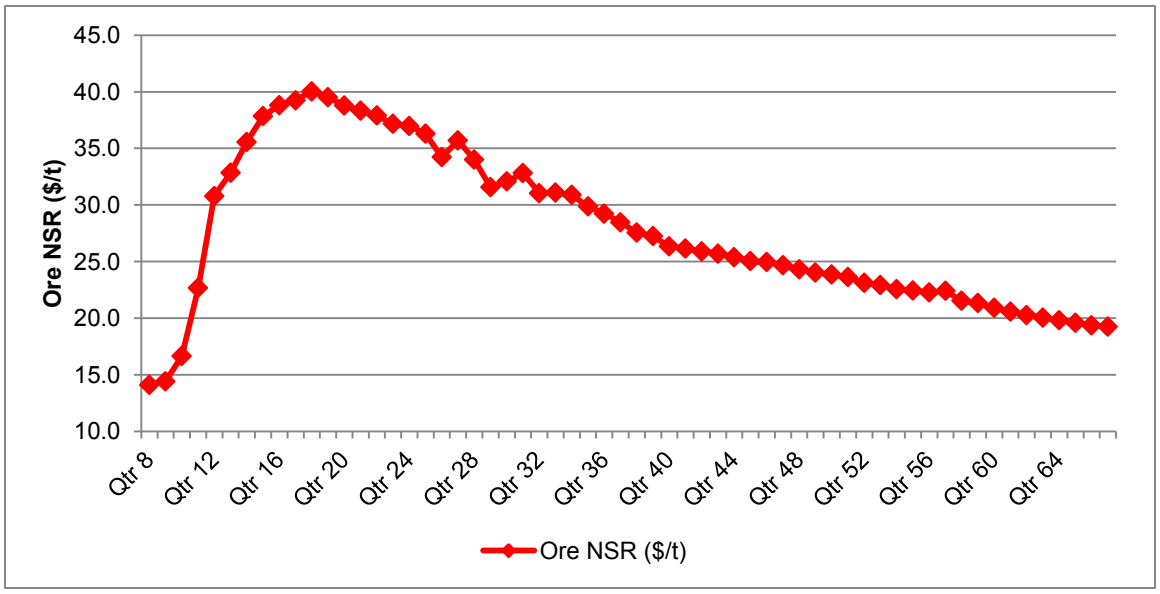


Figure 15.16: Ore NSR ($/t) by quarter.

From Table 15.8, it is seen that development ore is first mined in quarter eight (i.e., Q3 Year –3) which is sourced from the 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 quarter 11 (i.e., Q2 Year –2) and immediately builds to a steady state rate of approximately ±110 kt per quarter or 37 kt/mo. As only 50% of undercutting tonnes is mucked from the undercut level (as mentioned previously), the tonnage required to be drilled and blasted is double these tonnages. It is planned that this tonnage will be transferred to the extraction level via an ore pass located east of the crusher chamber.

Drawbell opening commences in quarter 12 (i.e., Q3 Year –2) and immediately builds to a steady state rate of approximately 230 kt per quarter or 77 kt/mo. Given that approximately 50% of this

tonnage is previously drilled and blasted as undercutting, only 50% requires drilling and blasting for drawbell opening. The entire tonnage is mucked on the extraction level.

Overall, undercutting and drawbell development tonnage amounts to approximately 114 kt/mo. This volume of material is required to be drilled, blasted, and mucked, a significant undertaking.

Cave production commences in quarter 12 (Q3 Year –2) and builds over 3½ years to reach the target production rate of 9 Mtpa by end of Year 2, which includes almost 1.4 Mt in Year 2 from ore development, undercutting, and drawbell opening. In Years 3 and 4, cave production exceeds 7.5 Mtpa, with the cave contributing the full 9 Mtpa from Year 5. This rate is maintained for 6 years until the end of Year 10, after which cave production declines due to depletion of draw columns. Final cave production is scheduled to occur in Year 13, although the profitability associated with 3.3 Mt ore in Year 12 and 0.65 Mt ore in Year 13 needs to be established.

The following discusses various issues associated with the cave production schedule:

- BHOD runs from Gemcom PCBC™ software suggested an optimal cave having in excess of 150 Mt. However, the 100 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 100 Mt, KUG mine life could be extended beyond Year 13.
- A delay of one quarter was applied to the start of cave production compared to what is possible, based on the development, undercutting, and drawbell opening schedule. This was done to ensure cave production does not commence before the crusher conveyor system is installed and commissioned by start of Q3 Year –2.
- When the PCBC™ runs occurred, deductions from the 9 Mtpa 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 Mtpa 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 15.11 shows the production rate curve used in PCBC™ to derive the cave production schedule:

Table 15.11: Production rate curve.

% extraction	$t/m^2/d$	t/d	mm/d
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 relatively conservative, suggesting increased cave production is possible. However, simulation of extraction level mucking activities showed that LHD traffic congestion on the extraction level, particularly around the crusher, will limit this potential.

15.5.4 Labour

This section discusses the mining-related labour required to develop, construct, and operate the KUG, including all contractor and owner management-supervisory, technical, operations, and maintenance personnel.

Initial total complement is 159 during early development through Q5, and then ramps up to a peak of over 400 by Q12 due to increases in owner labour for development, production, maintenance, technical, and management supervision. This level is maintained through Year 4 after which owner development, undercutting, and drawbell opening ceases. Years 5–10 have total complements reducing from 295 to 183, after which complements further decline as the end of the mine's life is reached.

Hourly labour for owner development is required from Q7 to Year 4, with complements varying with the number of owner development crews required. Q16 shows a step up in requirements asthe number of owner development crews increase from two to three along with cessation of contractor development. Development is completed in Year 4, after which owner development hourly labour is not required.

Owner production hourly labour is required from Q11 when undercutting, drawbell opening, and cave production commences. Peak complements occur in Years 1–4, declining thereafter as undercutting and drawbell opening is completed. Some of the decline is also associated with decreasing secondary breakage requirements as cave fragmentation improves.

Owner support hourly labour is required from Q1 to provide various essential services. As development, construction, and operations activity increases, the total complement increases to in excess of 70 from Q10 to Year 10. The complement declines thereafter as the various activities cease or wind down.

Owner maintenance hourly labour is required from Q1 to maintain equipment associated with owner support activities, with an initial complement of 4. As development and construction activities increase, this complement increases to 48 by Q14.

Owner operations staff labour is required from Q1 to oversee contractor development and construction activities as well as prepare for future owner mining, with an initial complement of 1 person. From Q6 to Q10 this complement increases to a peak of 16 as owner development, construction, and production operations commence. This peak is maintained through Year 4, declining thereafter as the various activities cease or wind down.

Owner maintenance staff labour is required from Q6 to oversee owner maintenance activities, with an initial complement of two personnel. This increases in Q7 to 7, which is then maintained throughout the mine's life.

Owner technical staff labour is also required from Q1 to provide technical support and oversight to contractor development and construction activities as well as to carry out mine planning, with an initial complement of six personnel. From Q5 to Q12 this complement increases to a peak of 16, which is then maintained through Year 4. In Years 5–6 there is a slight decline to 12, and a further slight decline thereafter to 10.

Contractor development hourly labour varies from 59 to 62 personnel from Q1 to Q15 depending on quarterly haul truck requirements. Contractor maintenance labour varies from 16 to 24 over

this period due to varying total mobile fleetsFrom Q16 there is no further requirement due to cessation of the contractor's scope.

Contractor staff labour complement is constant at 26 from Q1 to Q15, after which there is no requirement due to cessation of contractor scope.

15.5.5 Mobile Equipment

This section discusses the mining-related mobile equipment required to develop, construct, and operate KUG, including all contractor and owner mobile equipment.

Figure 15.17 shows steady state contractor development through Q1–15. Owner development starts once the overall development requirement reaches three jumbos from Q7, with two owner jumbos required by Q8–15. Contractor development ceases from Q16 with three owner jumbos required thereafter. Owner support mobile equipment is required throughout the mine's life, varying with development, construction, and production activities per period. Owner production mobile equipment is required from Q10, rapidly increasing thereafter as undercutting, drawbell opening, and cave production activities commence. Production mobile equipment shows a reduction from Year 5 as undercutting and drawbell opening activities cease, and secondary breakage requirements reduce due to improved cave fragmentation.

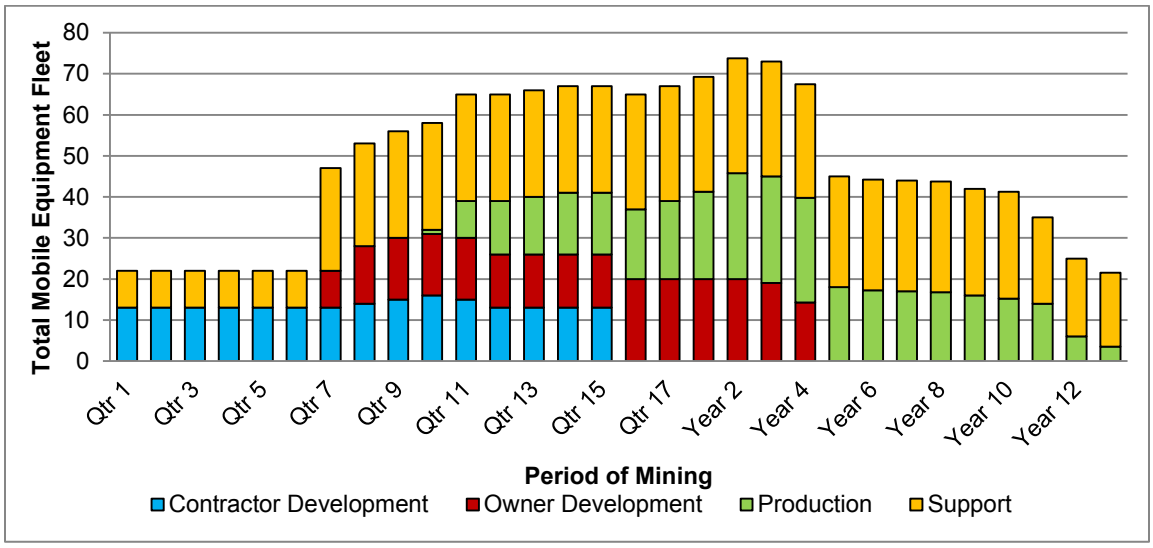


Figure 15.17: Mobile equipment fleet by period.

Contractor development runs from Q1–15, requiring two jumbos and supporting equipment throughout. The only variation in contractor mobile equipment relates to haul trucks, an increasing number is required as average haul distance increases and haulage requirements increase prior to the conveying system being commissioned in Q12.

Owner development commences in Q7 with one jumbo, increasing to two jumbos in Q8, further increasing to three jumbos in Q16, and maintains that fleet through to the end of owner development in Year 4. As for contractor development, supporting equipment requirements increase correspondingly. Also, as per the contractor development fleet, the owner development fleet requires an increasing number of haul trucks prior to commissioning of the conveyor system. There is another slight increase in owner haul trucks from Q16 through Year 2 as hauling requirements increase.

The owner production fleet requires just a single 8.6 m^3 LHD for mucking on the undercut level from Q11 through Year 4. The 10.7 m^3 LHD fleet is used for all extraction level mucking, building from just two units in Q11 to a peak of 13–14 units from Year 2–10. At steady state, two drills with V30 reamer head attachment, four longhole drill rigs, and one emulsion loader are required to maintain undercut and drawbell opening activity requirements through Year 4. Secondary breakage activities require one water cannon and one to two block holers through Year 1 to deal with high and low hang-ups, and one to five drill-split rigs and one mobile rockbreaker to deal with drawpoint oversize throughout the period of production.

The owner support equipment fleet varies throughout the mine's life with varying levels of development, construction, production, and maintenance. A single man carrier is needed from Q1, reaching two units in Q16 through Year 4. A single small LHD is provided from Q7 for clean up and other general duties. Two scissor lifts and one shotcrete unit are provided from Q7 for ongoing construction activities, including drawpoint construction. Four mobile maintenance vehicles are provided from Q7, declining to two in Year 11–13 as the total mobile fleet decreases. A single boom truck is provided Q1–6, increasing to two thereafter for the remainder of the mine's life, to provide for materials transport. One fuel truck is provided from Q7 to allow for re-fuelling of fuel bays using diesel piped to the diesel tank situated near the bottom of the declines. One transmixer is provided Q1–6, and two thereafter, for transporting concrete from the surface batch plant. A single grader is provided for the entire mine's life, which is considered sufficient as many roadways on the extraction level will be concreted. A single septic vacuum truck is provided for the mine's life. A single forklift is provided from Q7 to assist with materials handling. A fleet of nine light vehicles is provided for use by supervisory and technical personnel.

15.6 Mine Infrastructure

This section discusses the various mine infrastructure facilities and systems that are required to support KUG.

15.6.1 Ventilation

Ventilation system design for KUG was carried out by Mine Ventilation Services Inc. (MVS), Clovis, CA, USA, using VnetPC Pro+ ventilation system modelling software This section provides a summary of their work.

The final ventilation system for KUG essentially comprises two separate primary systems. One will ventilate the access decline and workshop using two identical fans (in parallel) at the portal area of the conveyor decline – referred to as the decline system. The other will ventilate the undercut and extraction levels (including the crusher chamber) by drawing fresh air through a 5 m raise from the surface, transferring fresh air to the working levels via an intake air level, and exhausting via a return air level and 5 m raise to the surface – referred to as the production system.

The twin 920 kW fans for the production systems will be Alphair 11200-AMF-7300s or their equivalent, each with a nominal capacity of 200 m^3/s at 3 kPa. A total requirement of 365 m^3/s has been estimated; these requirements were derived from estimating air flow based on 0.063 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 twin 186 kW fans for the decline system will be Alphair 8400-VAX-3150s or their equivalent, each with a nominal capacity of 70 m^3/s at 1.7 kPa.

The ventilation system was designed to keep air flow velocities within acceptable limits. In workings that will experience heavy traffic the maximum air velocity is 4 m/s. In the conveyor decline, the maximum velocity is 2 m/s. A minimum velocity of 1 m/s is provided in all active workings.

All primary fans are recommended to have variable speed drives (VSD) so that air flow can be adjusted to suit requirements.

Given seasonal ambient temperatures at Kemess, air heating will be required at the access decline portal and at the top of the intake air raise. This is required to provide a comfortable and safe working environment, avoid freezing of systems (such as water pipes), and avoid freeze-thaw that could create unstable ground conditions. A minimum temperature of +2°C is recommended. MVS recommends the use of direct-fired propane heaters for KUG. Heater running cost calculations were based on average monthly temperatures rather than extremes.

Once the final ventilation system requirements were established, based on development being completed and steady-state production being achieved, various snapshots were modelled to ensure suitable ventilation could be provided as the mine develops. Two of these interim stages have been reviewed in detail.

The first stage to be considered was the initial development of the twin access/conveyor declines. These twin declines will be advanced with the access decline as the main ventilation intake and the conveyor decline as the main exhaust. Twin exhaust fans capable of delivering 139 m^3/s will be installed at the conveyor decline portal entrance. Cross cuts will be driven every 150 metres to allow for a cross over of the ventilation circuit from the fresh air access decline to the return air in the conveyor decline. All development waste will trucked up the access decline to avoid the fans and bulkheads. Auxiliary fans and ducting can be used as the declines advance, with auxiliary ventilation equipment used to ventilate the active development faces beyond the last cross cut. Modelling with MVS's DuctSIM software showed that 1.4 m diameter flexible ducts can be paired with 50 kW, 1.4 m diameter fans that are capable of providing up to 20 m^3/s of air to active faces up to 660 m from the fans, more than adequate for decline development mucking.

Q1 Year –2 was also modelled as this is when the intake and return air levels meet at the southeast corner of the cave footprint, allowing the production ventilation system to be first activated. This provides a significant increase in the available air flow compared to having just the decline ventilation system: increasing from (up to) 139 m^3/s of air up to 500 m^3/s.

Following establishment of both primary ventilation systems, the decline ventilation system will provide 45 m^3/s of air to the workshop which exhausts directly to the return air level (which is part of the production ventilation system) and 25 m^3/s of air to the conveyor decline. This requires just one of the conveyor decline portal fans to be operating.

The identical primary intake and return fans for the production system are located underground in close proximity to the intake and return air raises. Intake air is moved to the extraction level via 20 intake air raises (2 m diameter) with each extraction drive having its own raise, providing at least 20 m^3/s of air to 13 different drives or rim tunnels. Intake air is moved to the undercut level via eight intake air raises (2 m diameter) with auxiliary ducting used to direct fresh air to active working faces. Regulators will be installed on all intake raises to manage the production system intake. Extraction level air flow will be further controlled by doors located at the north end of each extraction drive. Return air travels from north to south on the undercut and extraction levels and will be removed via 10 return air raises that connect both levels to the return air level. On the extraction level, a portion of the airflow exiting the extraction drives will be routed to ventilate the primary crusher, regulated to provide 40 m^3/s of air.

Locating the production ventilation system primary fans underground will require permission from the BC Mining and Minerals Division.

The following sensors are recommended to provide an effective ventilation system:

- Fan monitoring sensors,
- Air quality and quantity sensors,
- Psychometric sensors,
- Thermal imaging sensors, and
- Door sensors.

15.6.2 Crushing

Primary crusher design for KUG was carried out by Tetra Tech Inc. (Toronto, ON, Canada). This section provides a summary of their work.

There will be a single primary crusher located underground at KUG. Primary crusher tip points will be located on the extraction level (1,160 m), allowing tipping by LHDs tramming ore directly from drawpoints. There will be three tip points, one on each of the east, north, and west sides of the crusher serving different areas of the cave footprint. The east and west tip points have single access, while the north (or central) tip point has two access points. Figure 15.18 shows the tip point arrangement

The crusher will be a 54" x 75" gyratory crusher driven by a 450 kW motor. Figure 15.18 (plan view) and Figure 15.19 (section view) show the location of the crusher within the chamber. Assuming 333 crusher operating days per year (one shift per week, plus six days additional downtime), average throughput will be 27 ktpd or 1,500 t/h over an effective operating time of 18

h/d requiring a LHD to tip every 43 s on average. This size crusher will be able to handle this throughput as well as expected surges, producing a nominal –150 mm crushed ore product.

15.6.3 Conveying

Conveyor system design for KUG was carried out by Conveyor Dynamics Inc. (Bellingham, WA, USA) using proprietary modelling software. This section provides a summary of their work.

The conveying system will consist of an underground conveyor (fed from the primary gyratory crusher), a transfer station, and a surface conveyor that discharges onto the crushed ore pile ahead of the process plant. Only ore will be transported by conveyor, with waste hauled to surface by truck.

As per the primary crusher, the conveying system is designed to have a peak capacity of 1,627 t/h and a nominal capacity of 1,425 t/h averaging 1,500 t/h for 18 h/d over 333 days per year.

The belt feeder is shown on Figure 15.20.Figure 15.22. The curved chute that discharges onto the underground conveyor will help prevent puncture damage and reduce belt wear. This feeder will be powered by a 203 kW hydraulic motor.

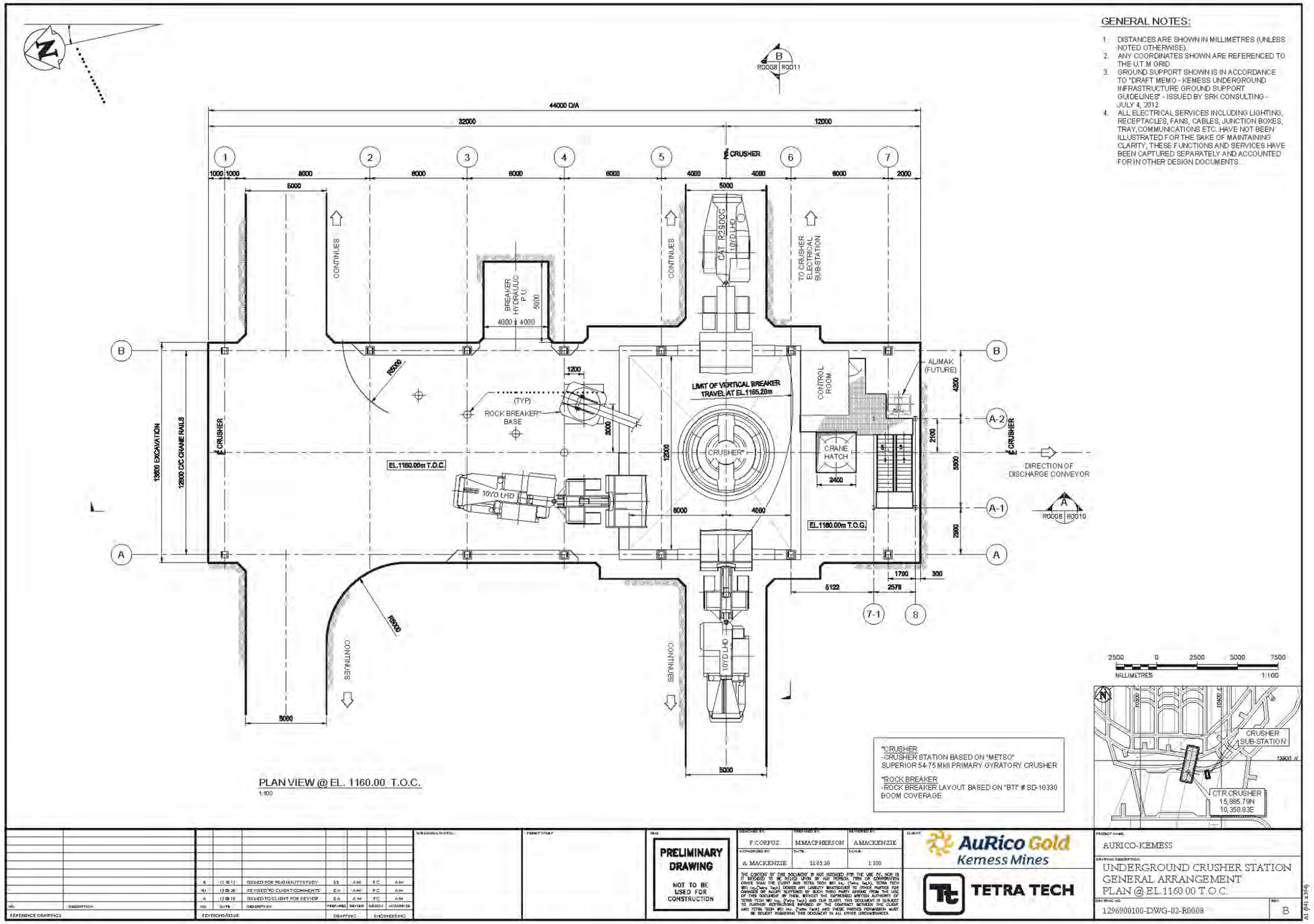


Figure 15.18: Plan view of crusher chamber on extraction level.

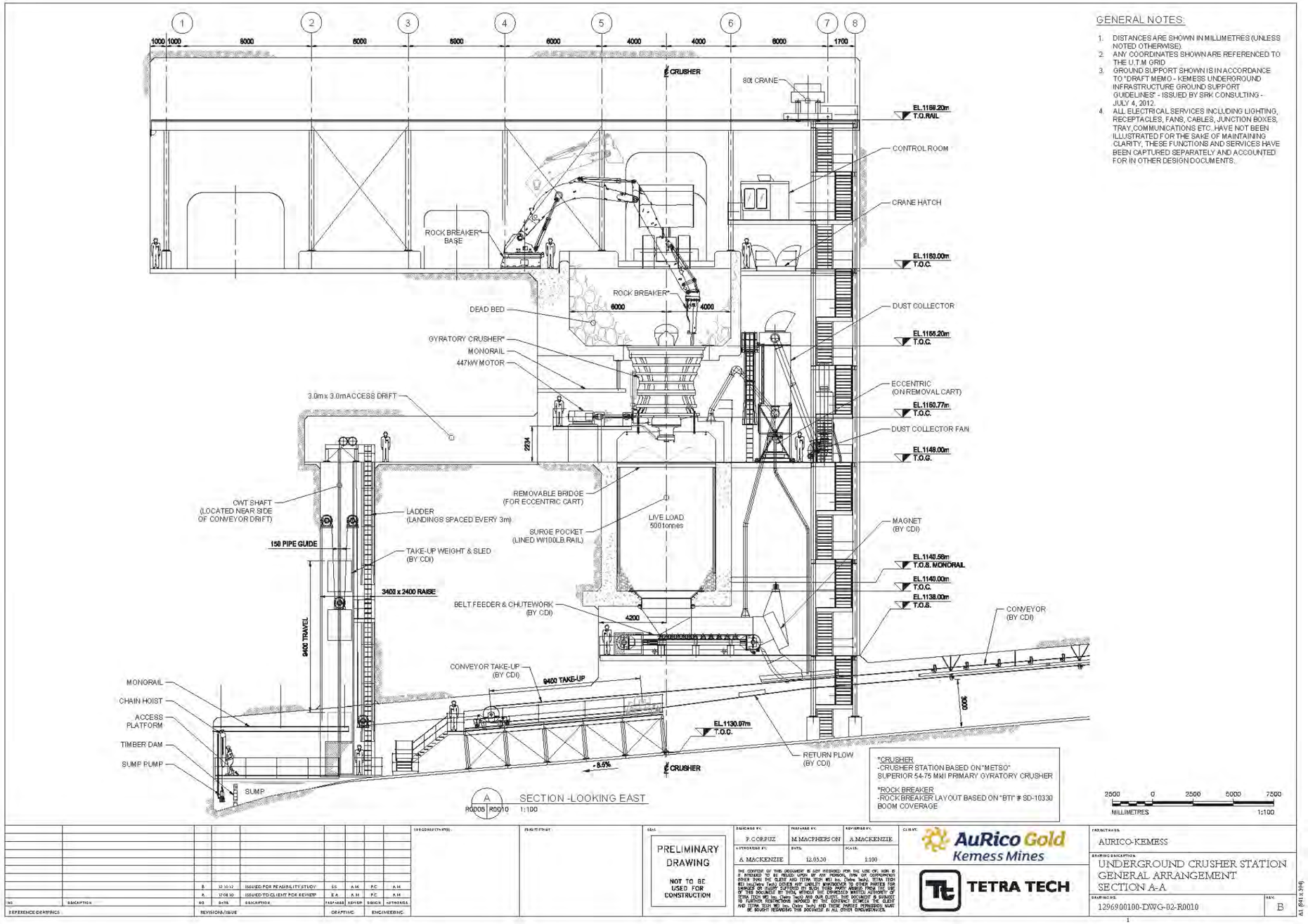


Figure 15.19: Section view of crusher chamber from extraction level to conveyor decline.

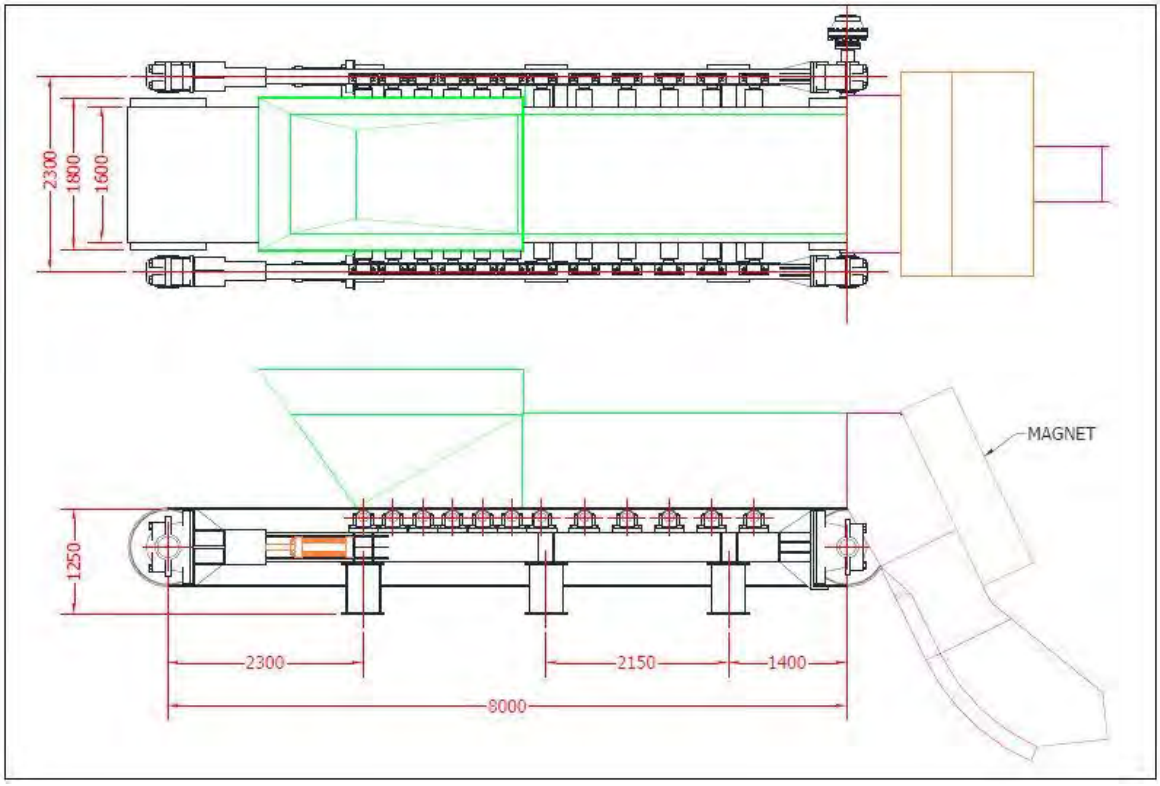


Figure 15.20: Belt feeder general arrangement.

The back-mounted underground conveyor, CV01, follows a straight +8.3% (uphill) alignment from below the belt feeder. This conveyor is 3,393 m long and gains approximately 230 m elevation overall (with the final surface portion being downhill). This conveyor is designed as an elevated (trestle mounted) section from the conveyor decline portal face to the CV01–CV02 transfer tower.

CV01 uses ST-3800N/mm belt that is 1,067 mm (42") wide, and is powered by three 671 kW inverter driven motors (with two 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. This belt will run at 2.9 m/s. The tail end gravity take-up arrangement is shown on Figure 15.19.

The surface conveyor, CV02, follows the Kemess Lake Valley en route to the existing processing facility, passing along the east side of the Kemess South open pit (which is the proposed tailings storage facility); resulting in a number of horizontal and vertical curve sections. This conveyor is 4,637 m long and loses approximately 59 m elevation overall, and has three elevated sections. The first is immediately following the CV01–CV02 transfer tower, the second is at the Kemess Lake outlet, and the third is where the conveyor discharges onto the stockpile. All elevated sections are accessable by walkways. There is a cut and cover section of conveyor just beyond the Kemess lake outlet where the access road crosses over the conveyor.

CV02 uses ST-1600N/mm belt that is 1,067 mm (42") wide, and is powered by 4 x 373 kW inverter driven motors (with 2 on the primary pulley, one on the secondary pulley and one on the tail pulley). The high tension pulleys shell diameter is 914 mm, and the low tension pulleys shell diameter is 610 mm. This belt will run at 4.0 m/s. This conveyor will also employ a gravity take up.

AMEC Environment & Infrastructure (Burnaby, BC) has provided a design for the 4,500 m access corridor from the existing Kemess facilities to the planned twin portal location. Conveyor CV-02 connects the existing Kemess facilities to the planned portals. Where CV02 cannot be aligned with the access road, due to conveyor curvature constraints, a 5 m wide road is provided for CV02 installation and maintenance.

The lowest belt factor of safety calculated by Conveyor Dynamics Inc. (CDI) is 3.5:1 for CV02, which occurs in the unlikely event that the conveyor starts with just the uphill sections loaded. The normal running safety factor of CV02 is 8.8:1. The lowest factors of safety modelled for CV01 all exceed 5:1.

15.6.4 Water Management

The dewatering system design for KUG was carried out by Tetra Tech Inc. (Toronto, ON, Canada). This section provides a summary of their work.

The dewatering facility will be located at the lowest elevation of the Mine, being 4–5 m below the 1,126 m elevation, which is the low point on the ventilation drainage level.This will allow gravity drainage to the two main sumps. Each sump comprises settling and clean water zones, with each clean water zone having at least two hours capacity.

Water inflow estimates were provided by Lorax Environmental Services Ltd. (Lorax) covering all stages of the mine's life: from initial decline development, through cave development, and through to cave breakthrough to the surface. Maximum inflow is estimated to occur just after cave breakthrough, conservatively estimated at 69.7 L/s. Thereafter, steady state inflow is conservatively estimated to be 38.4 L/s. In addition, Tetra Tech estimates mining process water inflow of 13.0 L/s.

Based on the estimated inflows, two 574 kW pumps will be installed in the pump chamber. During peak inflow (69.7 L/s), one pump will need to run for 10 h followed by a 2 h idle, removing 88 L/s against a 400 m head. During steady state inflow (38.4 L/s), one pump will need to run for 1.7 h followed by a 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,\. Water will be delivered to the sedimentation pond at the portal area, with the pipe located in the conveyor decline. Settled water will then be transported to the tailings storage facility via a 200 mm pipeline installed in the overland conveyor (CV02) structure.

Lorax has also estimated inflow of 654 L/s over 48 h for the 100 year storm event; an extreme worst case as it assumes maximum precipitation and snowmelt occuring simultaneously, 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 approximately 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 hours at a steady state inflow rate of 38.4 L/s, providing initial storage in the event of a power outage in 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.

15.6.5 Maintenance Facilities

The underground workshop design for KUG was carried out by Tetra Tech Inc. (Toronto, ON, Canada). This section provides a summary of their work. Figure 15.21 shows the general arrangement of the underground workshop.

The main workshop will be located on the extraction level, south of the primary crusher chamber, 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, and 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 access points 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.

A total of 10 large items of mobile equipment can be accommodated in the workshop at any one time. This compares to a peak maintenance load of 13–15 units during Years –1 to 4. 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%.

15.6.6 Electrical

The underground electrical reticulation system design for KUG was carried out by Tetra Tech Inc. (Toronto, ON, Canada). This section provides a summary of their work.

The portal area substation will provide 13.8 kV electrical power supply to KUG via two main feeder cables, one in each of the access and conveyor declines. Junction boxes located in cross cuts between the declines will allow these cables to be installed incrementally during initial development.

Where the declines meet the ventilation level cross cuts, the main electricity supply will be split into three separate legs for each decline supply. One leg will provide electrical power distribution

to the primary fans, another leg will provide electrical power distribution to one side (north or south) of the cave footprint, and the other leg continues down each decline and terminates in switchgear located at the main underground substation. This configuration will provide a high degree of redundancy and switching options to support routine system maintenance while minimising impact on operations.

Branch circuits from the main underground substation will link to peripheral substations that provide electrical supply to the main infrastructure facilities such as the primary crusher, dewatering pumps, and workshop using various transformers, breakers, and motor control centres. The larger electrical loads will be equipped with soft starts (i.e., reduced voltage motor starters) to minimise starting impact on both electrical and mechanical components.

Total underground electrical load is estimated to be 9.7 MW.

15.6.7 Storage & Materials Handling

The underground storage and materials handling requirements for KUG was carried out by Tetra Tech Inc. (Toronto, ON, Canada). This section provides a summary of their 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, 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, fuel trucks will deliver diesel to one of the three fuel stations located on the extraction level.

Concrete and shotcrete will be delivered from the surface (portal area) batch plant directly to points of use underground by a transmixer. Shotcrete will be transferred to mobile shotcrete units.

Flatbed trucks will be used to transport palletised materials from the existing Kemess South warehouse to the portal area 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. Explosives and accessories will be transported separately.

Shift crews will be transported underground using a fleet of 22 person man carriers, with transport directly from the mine dry located at the existing Kemess South warehouse workshop facility to the parking area outside the underground workshop.

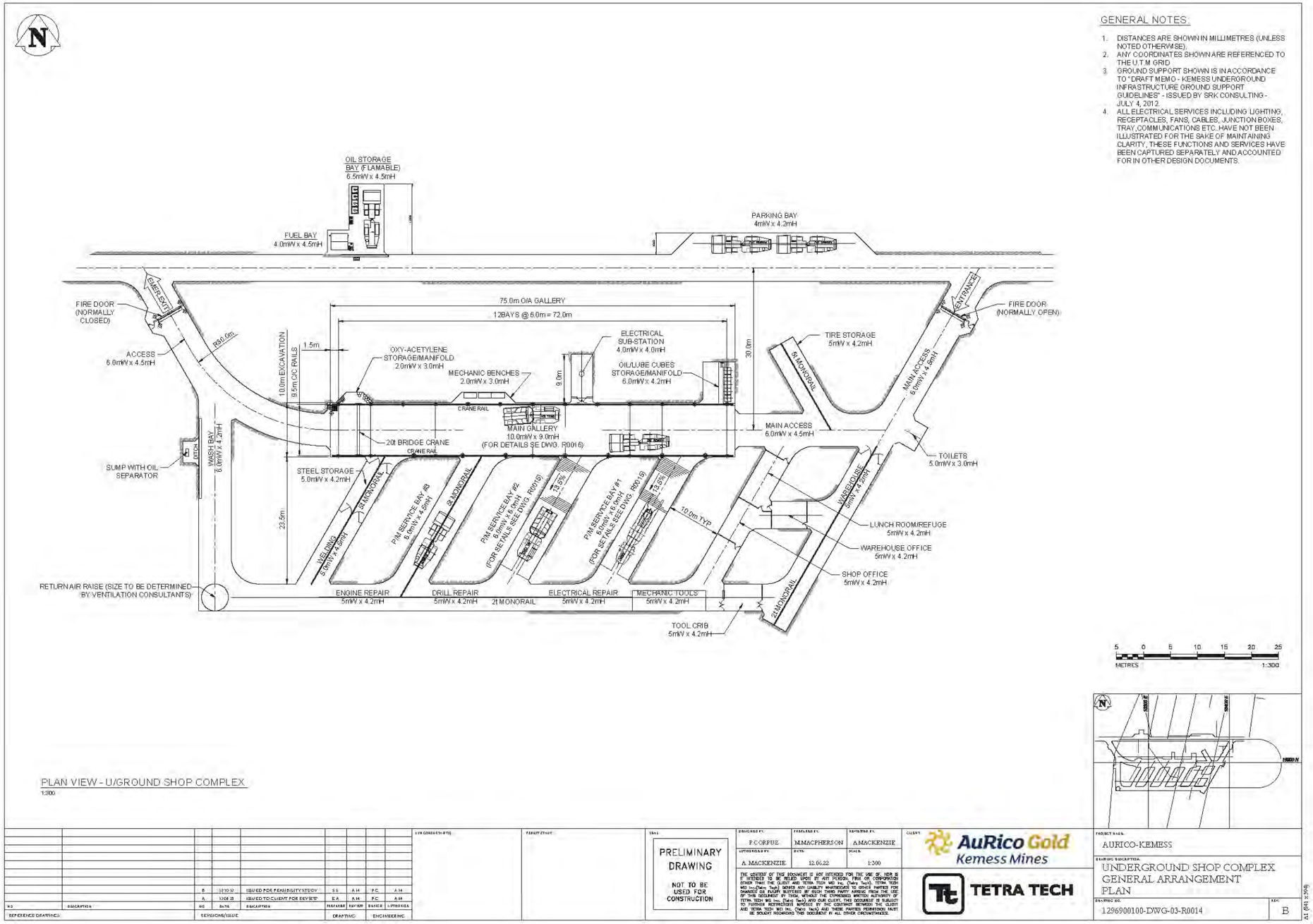


Figure 15.21: Underground workshop (plan view).

15.6.8 Monitoring & Control Systems

In addition to the geotechnical monitoring systems outlined in Section 15.1.6 the following monitoring infrastructure and systems will be required for effective control at KUG.

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 operation will be controlled using a system equivalent to either Modular Mining's dispatch system or Sandvik's manual production management (MPM) system.

15.7 Risk & Opportunities

This section discusses various mining-related strengths-opportunities and weaknesses-threats identified as a result of carrying out the KUG Feasibility Study.

15.7.1 Strengths & Opportunities

There are opportunities to:

- Increase development rates in the first three years to provide additional time for installation of the primary crusher and conveying system. This is considered possible given the relatively conservative advance rates planned. This is further considered possible with suitable incentives for the development contractor.
- Replace the primary intake air raise with an intake air decline, thereby removing the challenges associated with its location. It may be possible to eliminate/reduce intake air heating requirements by employing heat exchangers using return air heat. Replacement of the primary return air raise with a return air decline could also be considered.
- Relocate the primary crusher to the east to better balance tramming distances between the various cave footprint zones. The optimal location is considered to be between extraction drives 10 and 11.
- Stagger shift start and end times to provide periods of reduced production LHD traffic on the extraction level. This should still allow sufficient time for development and production blasting to occur between shifts.
- Re-route the overland conveyor though a tunnel to the Kemess South site, providing a more direct alignment and less exposure to avalanche risk and unknown subsurface conditions. There may also be cost savings associated with this approach.
- Evaluate automation of extraction level production mucking operations. Benefits include reduced operating labour complements and extended operating shifts.

- Evaluate alternative extraction level layouts such as the El Teniente layout.
- Evaluate a truck haulage level below the extraction level in order to reduce production LHD requirements and associated crusher tip congestion.

15.7.2 Weaknesses & Threats

The crusher conveyor system may not be commissioned by Q3 Year –2 due to development and construction delays. Given the ventilation limit on the number of haul trucks that can be employed underground, any delay will directly impact the scheduled production ramp up.

Development of the primary air raises from surface will be challenging over the winter months due to accessibility. These raises may also require ground support which will slow their development and availability. The ongoing supply of propane for the intake air raise heater will also be challenging given the location of the raise collar.

Development into the east side of the cave footprint requires timely establishment of the primary ventilation circuit, including primary raises to surface, intake and return air levels to the southeast corner of the footprint, primary fans underground, and primary production ventilation circuit controls.

Simulation has shown that the greatest risks to achieving the planned 9 Mtpa using the current layout will be the lack of surge/storage in the ground handling system, LHD congestion, and tramming distances (particularly in the east zone).

16 Recovery Methods

16.1 Summary

Primary crusher facilities will be installed in the KUG mine. Ore will be conveyed from the crusher to the underground portal where it will be transferred to an overland conveyor for delivery to the existing mill ore feed stockpile.

One SAG mill/ball mill grinding line was removed from the Kemess South grinding circuit. The targeted mill throughput for the KUG project is estimated to be 24,600 t/d. A simplified flowsheet in Figure 16.1 illustrates the revised mill flowsheet. A copper flotation concentrate containing the gold and silver values will be produced and exported smelters for processing. Tailings will be discharged to the Kemess South pit and this is expected to have sufficient volume capacity for the duration of the KUG operations. Process reclaim water will be pumped to the Kemess South mill facility

16.2 Flowsheet Selection

The process flowsheet, simplified in Figure 16.1, has been designed to utilize as much of the existing equipment as possible to minimize capital expenditures.

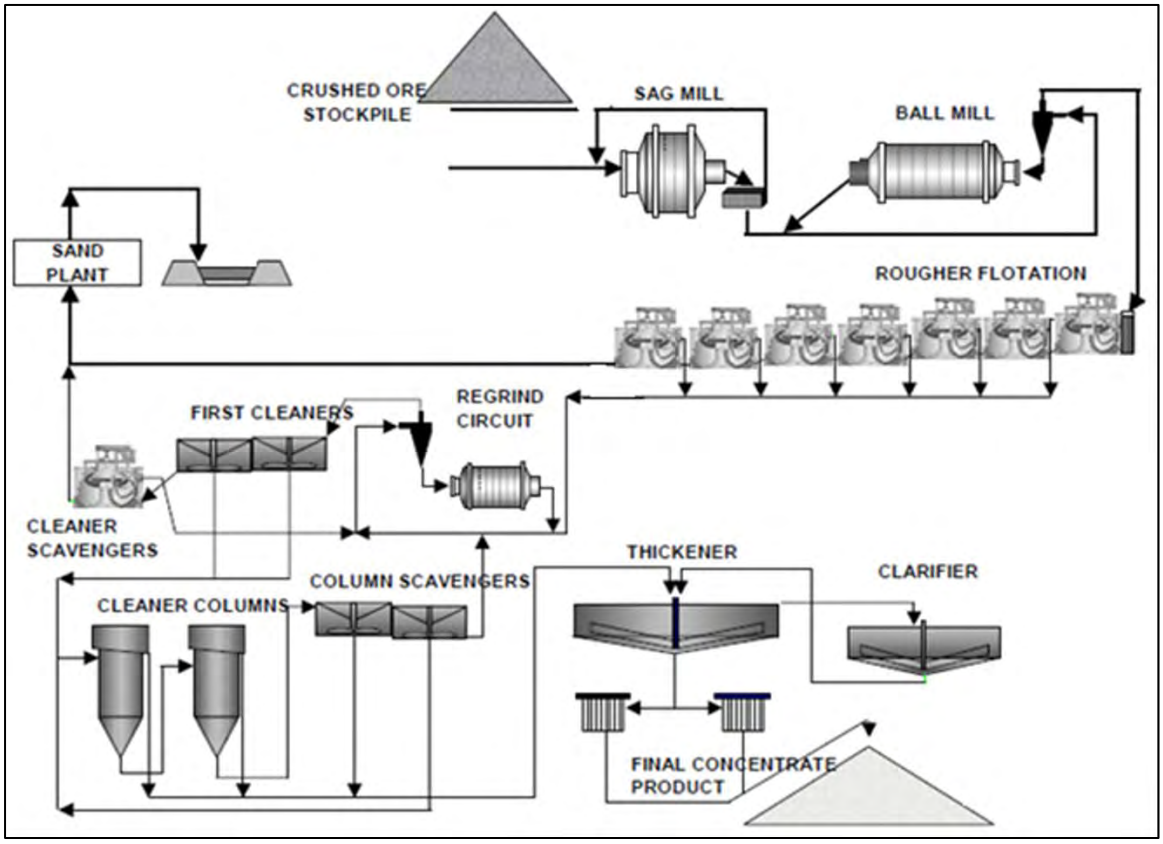


Figure 16.1: KUG and Kemess South simplified flowsheet.

16.3 Process Design Criteria

The process design criteria used for assessing the existing flowsheets was based on metallurgical test work, information from previous operations experience at Kemess, vendor information, and reference to similar operations.

16.4 Primary Crushing

The ore will be crushed underground prior to being conveyed to the surface portal. See section 15.6.2 Crushing.

16.5 Overland Conveyor

Ore will be transferred from the underground conveyor at the surface portal to an overland conveyor system. See section 15.6.3 Conveying.

16.6 Kemess South Mill

16.6.1 Primary Stockpile

Ore will be transferred from the overland conveyor to the mill stockpile using the existing stacker conveyor. The stockpile will have a live storage capacity of 74,200 t. Ore will be reclaimed from the stockpile by the three existing, 6 ft x 20 ft apron feeders delivering ore to a 60" wide mill feed conveyor.

16.6.2 Grinding

The grinding circuit will utilize the existing 34 ft x 15.25 ft SAG mill driven by twin 6,000 hp motors. The SAG mill is a fixed speed mill and is operated in series with a closed circuit 22 ft diameter x 36.5 ft ball mill, also driven by twin 6,000 hp motors. Slurry discharges 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® cyclones prior to the shutdown. The cyclopac consists of 12 cyclones of which nine are operating. It is not expected that there will be any changes to the existing cyclone pumps or cyclopacs for the KUG Project.

16.6.3 Flotation

Cyclone overflow from the grinding circuit will be processed in the 16 existing 127 m3 rougher flotation cells. The circuit layout will have two lines of two parallel rows each containing eight flotation cells. During operation of Kemess South some of the Wemco rougher flotation cells were refurbished with new Outokumpu mechanisms . The refurbishment included the installation of flotation blowers. The distributor installed in the mill permits operation of each rougher flotation bank independently. No additional rougher flotation modifications are envisioned for processing ore from the KUG deposit.

The conversion of the rougher flotation cells was completed with the installation of Outokumpu mechanisms. These new mechanisms were selected to eliminate cell sanding conditions that resulted in a loss of residence time, and to increase mass pull with the positive addition of air. Rougher concentrate will be pumped to the concentrate re-grind circuit to produce an 80% passing 15 µm product for final cleaning. The existing 4,475 kW ball mill re-grind will not be sufficient to handle the re-grinding requirements that result from the increased mass pull and the finer target product size. A two stage re-grind circuit will be configured with the existing ball mill re-grind, operated as the first stage in closed circuit with cyclones. The second re-grind stage will utilize one 1,500 kW IsaMill™ to take advantage of the higher efficiencies offered by stirred mills. The IsaMill™ will operate as a single pass unit.

Re-grind product from the IsaMill™ will be pumped to the first cleaner mechanical flotation cells. These cells were purchased used, and were installed and retrofitted into the original concentrator. They will need to be rebuilt 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 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 17 m^3 cells 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 re-grind and the tails will be sent to the final tailings 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. The original 10.8 m diameter thickener acts as a clarifier for the concentrate thickener overflow solution before it will be returned to the process. Thickened concentrate will be stored in the two 4.88 m diameter x 4 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 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.

16.7 Reclaim Water and Process

Process Water will ultimately be reclaimed from the Kemess South pit. The process water requirements for the mill have been estimated at 60,000 m^3/d.

The reclaim pumping system from the Kemess South pit will use the existing vertical turbine pumps and the reclaim water will be delivered to the mill process water pond.

16.8 Tailings Management

For the KUG project the mill 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 feed from the KUG project; approximately 102 Mt of ore and an additional 2.1 Mt of PAG waste rock over an operating mine life of 13 years

16.8.1 Overview

The waste management plan for the KUG project includes disposal of the mill tailings and 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 16.2). 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 required by Year 7) to an ultimate crest elevation of 1,274 m in order to provide adequate tailings, free water, and flood storage. Water will be reclaimed from the tailings facility for use in processing of ore. Operational constraints dictate that a water treatment facility will need to be built and in operation by Q4 Year –2 when the mill 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.

Figure 16.2: Exhausted open pit with PAG re-handled and tailings deposited into top end.

16.8.2 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 core of the dam. Upon completion, a NAG above water tailings beach and fresh water cap will be developed and an open channel closure spillway with invert at elevation 1,270 m will be constructed through the southern perimiter 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 16.3).

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 elevation 1,260 m.

Stage 2 — Intermediate dam, constructed to elevation 1,265 m, will be raised by the centreline method with 10 m wide compacted till core to elevation 1,265 m.

Stage 3 — Intermediate dam, constructed to elevation 1,270 m, will be raised by the centreline method with 10 m wide compacted till core to elevation 1,268 m, upon which the till core will be tapered with a downstream slope of 1H:1V to elevation 1,270 m.

Stage 4 — Final dam, constructed to elevation 1,274 m, 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 elevation 1,273 m. Upon closure, the till core will be capped with a 1 m thickness of fine NAG rockfill.

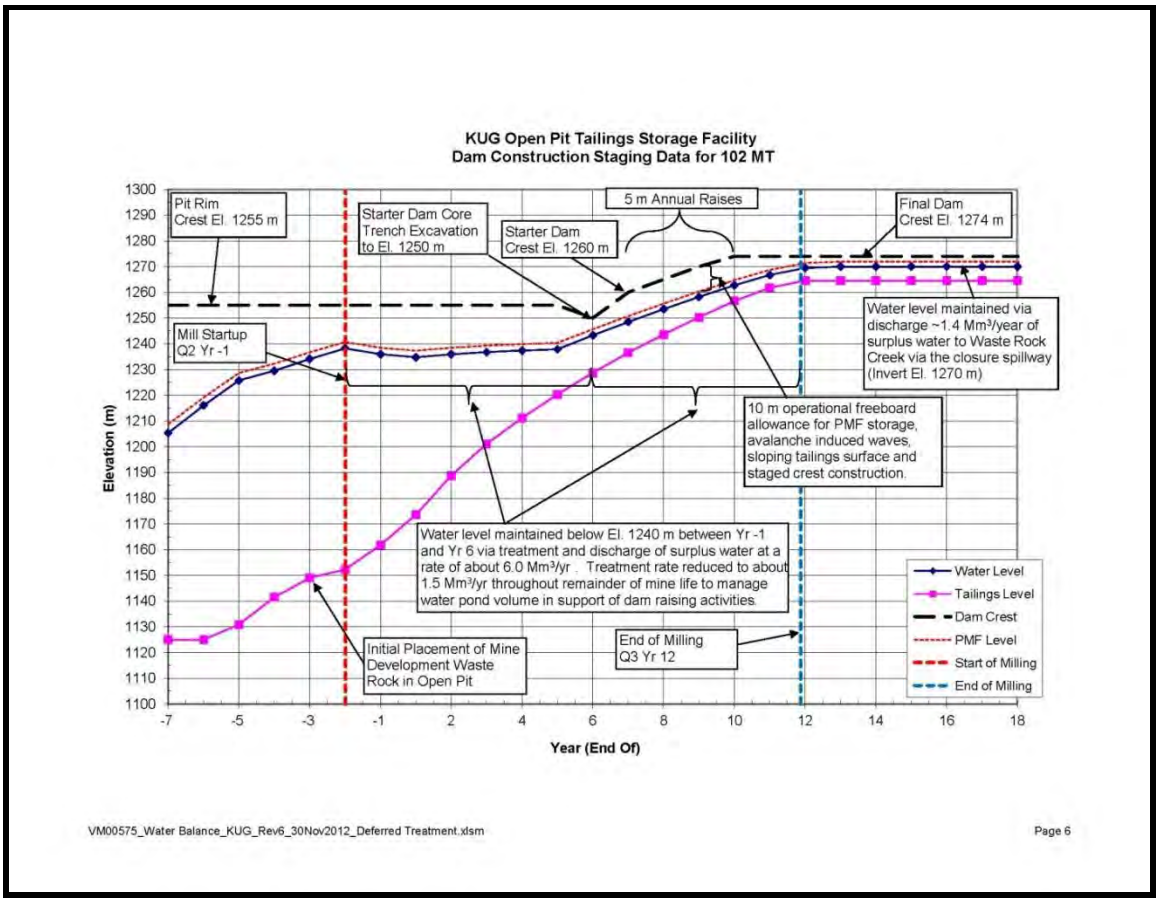


Figure 16.3: Dam Construction phasing and water balance.

16.8.3 Operational Water Management

Underground water management design was by Tetra Tech and is described in Section 15.6.4. In brief, mine water will be collected in sumps in the ventilation/drainage level and pumped via a 200 mm line to the portal, then to the Kemess South pit.

Water management requirements on the surface will be focused on the Kemess South pit, which will also serve as the tailings storage facility (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 diverson water collected in the various sumps and ponds around the 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 40.0 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 start up and that the TMF 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. It assumes that discharges from the Kemess South pit will begin at mill start up and will require treatment for dissolved metals. Assuming mill start up in Year –1, the sequence will be as follows:

- A water treatment plant capable of treating 4,300 m^3/h will be constructed in Year –2, along with a pipeline to carry treated water to the Attichika Creek.
- The diversion ditch along the pit highwall will also be reconnected in Year –2.
- Water will be withdrawn from the pit, treated, and discharged at a rate of 6.0 Mm^3 per year from Year –1 through Year 6.
- Assuming the tailings starter dam is completed in Year 7 and raised in Year 8, Year 9, and Year 10, as per the AMEC tailings design report, the annual rates of water treatment and release will drop to 1.5 Mm^3 per year for the remainder of operations.

16.8.4 Design Considerations

The tailings containment structures were 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 pit, stability analyses were performed and additional provision was made for wave generation in the event of pit wall failure.

16.8.5 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 16 Mt of tailings in the KUG TSF. This would entail an additional lift on the East Dam and susequent extension of the dam toe towards Kemess Creek which contrains further expansion of the TSF to about 16Mt or a 10 m raise of the facility (i.e. East Dam crest to El. 1284 m).

17 Project Infrastructure

17.1 Existing Infrastructure at the Project Site

The Kemess South Mine site previously supported a 50,000 t/d open pit mining and milling 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, milling facility, and related support infrastructure, such as the airstrip, access road, power supply, and concentrate movement infrastructure, are more than adequate for the proposed underground project. The following generalized discussion of facilities is focussed on features relevant to the KUG project.

17.1.1 Camp

The existing camp includes a kitchen, 40-person bunk house units, potable water facility, sewage facility, backup generator sets, site services maintenance shops, and recreational facilities. This camp was used for Kemess South. It was determined that with modest renovations the camp can be upgraded sufficiently to last for the projected life of the KUG operation. Renovation costs were estimated and included as capital costs in the feasibility study.

17.1.2 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 mill and previous mine site. AuRico owns the power line and all related site electrical infrastructure. The power line is currently in use providing power for the care and maintenance phase of the Kemess South infrastructure.

17.1.3 Administration Complex

The administration/surface shop complex is intact and operational. Warehousing and a surface workshop, are established in the complex. All of the accounting, engineering, and administration office spaces from the open pit operation are still available for the proposed KUG operation.

17.1.4 Access Road

The 380 km access road from MacKenzie is being maintained and used on a seasonal basis for ongoing site reclamation.

17.1.5 Airstrip

There is an all-weather airstrip at site that was routinely used by Twin Beech aircraft to ferry personnel on and off site during the operation of the Kemess South Mine. Dash 8 aircraft have used the strip in the past, as well as some heavy haul type STOL aircraft. The strip continues to be maintained and used by the care and maintenance crew as their main site access route.

17.1.6 MacKenzie Rail Loadout Facility

AuRico retains a trans-shipment facility at MacKenzie next to a rail spur for concentrate shipment. Grinding media has also been brought to MacKenzie via rail car from various suppliers. During initial construction 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 to market. There are no improvements required for this facility.

17.2 General and Administration Surface Support

Surface support work has been included in the G&A cost to avoid overlap with the underground operation. Surface support includes:

- Operation of the surface warehouse as well as fuel tank farms, which are more than sufficient for the underground operation (~2 ML storage).
- Accounting, purchasing, HR, administration, and engineering office space.
- Relevant staffing except for engineering staffing which will be accounted for with the underground operation. G&A has been costed to include the environmental group as separate from engineering.
- Emergency surface services include nursing staff, safety professional, ambulance, first aid facilities, surface mine rescue, and fire engine. Underground mine rescue is supported by the underground budget.
- Road and airstrip maintenance including snow removal and grading of the KUG portal access route. Equipment currently on site and scheduled for upgrade/replacement include graders, dozers, sanding truck, pumper truck, float, pickups, 40 t articulated trucks, and buses for crew transportation. Equipment remaining on site from the open pit operation includes graders, dozers, excavators (two Hitachi EX450 backhoes), Tadano 40 t crane, hiab, lube/fuel truck, pumper truck, pickups, loaders, and assorted light service vehicles.
- Tailings line support and general light ditching and pumping support of various catch ponds in conjunction with the mill operations group.
- Operation of a surface mechanical shop to maintain surface equipment.
- Operation and maintenance of the accommodation camp and kitchen and related infrastructure including garbage dump, potable water, and sewage treatment facilities. Potable water is drawn from wells that supported the previous open pit infrastructure and camp. Camp catering is usually contracted out and an updated quote was used from the current provider for the feasibility study costing.
- Provision of site security to control site access and manage wild life risks around the camp, administration, and milling facilities.
- Magazines for explosives used for the open pit are still current and suitable to use for the underground.

18 Market Studies and Contracts

Exen consulting services was contracted by AuRico 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 Kemess concentrate fairly readily marketable and not subject to any penalties. The gold grade is projected to be on the 20–50 g/twhich will result in a good payable metal factor of about 97.5%.

18.2 Metal Price review

The metal market for copper looks to be fairly neutral from the supply and demand side, and while prices are projected to soften somewhat they are still forecast to be slightly above US$3.50/lb into 2015. A longer term price of about $3.00/lb is forecast for copper. Exen did not venture an opinion on gold prices, but AuRico has judged that a longer term price of around US$1,300/oz is a reasonable number to use for a base case.

18.3 Refining Charges

Exen suggest using the following refinery charges:

- Treatment charge: $80.00/dmt
- Copper refining charge: $0.08/lb
- Gold refining charge: $6.00/oz
- Silver refining charge: $0.40/oz

18.4 Marketing Logistics

As done in the past, it is suggested to bulk truck concentrate from the mine 380 km down the forest 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 out east to the Horne Smelter, rail south to Kennecott in the States, and rail south to the port of Vancouver. Rail through Vancouver with ocean freight overseas to Asia appears to be the most economical route and was considered as the base case.

Exen suggests an all up transportation cost of $232/dmt.

19 Environmental Studies, Permitting, and Social or Community Impact

19.1 General

Kemess South Mine is a former producing open pit mine. The Kemess South reserves have been exhausted and a new reserve has been located approximately 6.5 km from the existing plant site. Reclamation and closure preparation of Kemess South has been undertaken annually via the progressive reclamation strategy. AuRico has been recognized for being pro-active in reclamation and closure at the minesite , and has been the recipient of a number of reclamation awards from the British Columbia Technical and Research Committee on Reclamation (TRCT) including:

- 2010 - British Columbia Jake McDonald Annual Mine Reclamation Award, recognition for Outstanding Reclamation Achievements at the Kemess South Mine, and the
- 2001, 2008, & 2009 - Metal Mining Citation - recognition for Outstanding Achievement in Mine Reclamation at Kemess South Mine.

The Kemess South Mine has also been the recipient of the 2010 Mining and Sustainability Award from the Mining Association of British Columbia (MABC). This sustainability award honours industry leadership in responsible mining.

19.2 Environmental Regulatory Setting

The environmental assessment (EA) and permitting framework for metal mining in Canada is well established. The federal and provincial EA processes provide a mechanism to assess potential effects of major projects and expansions or modifications to existing projects. Following successful completion of the EA process, the project enters the permitting phase. The project is then regulated through all phases (construction, operation, closure, and post closure) by both federal and provincial agencies.

Under the *Canadian Environmental Assessment Act* (CEAA 2012), the federal government may delegate any part of an environmental assessment to the province. The Agency may substitute the BC process for a federal EA or recognize the BC process as equivalent to the federal process. Both of these options have the potential to streamline the EA process.

19.2.1 BC Environmental Assessment Process

In British Columbia the Environmental Assessment Office (EAO) manages the assessment of proposed major projects as required by the British Columbia *Environmental Assessment Act* (BC EAA 2002). The EA process provides for an integrated assessment of the potential environmental, economic, social, heritage and health effects that may occur during the lifecycle of a project, and provides for meaningful participation by First Nations, the public, local governments, and federal and provincial agencies.

Mineral mining projects become reviewable in three ways:

1) The BC EAA Reviewable Projects Regulation provides for a mining project or modifications to an existing project to be automatically reviewable if it meets the following thresholds:

    a. Any new mine with 75,000 tonnes or more 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) 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.

3) Proponent "opt-in" whereby a proponent may request that the EAO consider designating its project (that otherwise would not be reviewable) as a reviewable project, and the EAO agrees with and orders such a designation.

19.2.2 Federal Environmental Assessment Process

In the spring of 2012 the Canadian Environmental Assessment Act 1992 was amended and replaced with CEAA 2012. Two significant results of these amendments were the re-definition of what triggers a federal EA and the introduction of legislated time periods within a federal EA if it is required.

Under CEAA 2012, an EA focuses on potential adverse environmental effects that are within federal jurisdiction including:

- fish and fish habitat;
- other aquatic species;
- migratory birds;
- federal lands;
- effects that cross provincial or international boundaries;
- effects that impact on Aboriginal peoples, such as their use of lands and resources for traditional purposes; and
- changes to the environment that are directly linked to or necessarily incidental to any federal decisions about a project.

There are two main methods of trigging a federal EA under CEAA 2012:

1) A project will require an EA if the project is described in the Regulations Designating Physical Activities (2012).

2) Section 14(2) of CEAA 2012 allows the Minister of Environment to designate (by order) a physical activity that is not prescribed by regulation if, in the Minister's opinion, either the carrying out of that physical activity may cause adverse environmental effects or public concerns related to those effects may warrant the designation.

19.2.3 Environmental Assessment Requirements of the Project

The deposit which constitutes the KUG project was previously assessed by a joint federal/provincial EA review panel as the Kemess Mine Expansion – Kemess North Project. This assessment was based on mining the Kemess North deposit as an open pit. The joint review panel recommended that the Kemess North Project not be approved by the provincial and federal governments. The provincial and federal ministers accepted the panel recommendations and did not allow the Project as it was defined to proceed to permitting. In its announcement, the governments indicated *this decision does not preclude the proponent from seeking to modify the project proposal by addressing the factors considered by the panel, and requesting another environmental assessment in the future*.

Substantive and positive changes have been made to the KUG project, which address the environmental and social concerns identified during the joint review panel. In addition, the KUG project is a substantially different proposal than the Kemess North project.

The KUG project may require a federal and/or provincial EA before it may proceed. Completion of a Standard provincial EA, following its initiation, would require approximately 24 to 30 months. Based on current CEAA 2012 guidance documents and the Act's new legislated timelines a standard EA would require 12 to 24 months from the commencement of the federal EA.

19.3 Environmental Permitting Process

The KUG project will likely be required to amend existing provincial authorizations to include the activities associated with the KUG project. Federal authorizations may also be required. If an EA is required, this process can generally be initiated concurrently with the EA and typically adds an additional 60 days to the schedule following a provincial ministerial decision allowing the project to proceed. The following sections contain lists of both federal and provincial licenses and permits that will be required for the Project.

19.3.1 Provincial Authorizations

The main provincial authorizations that would be required for the construction and/or operation of the KUG project include a *Mines Act* permit for the mine plan and reclamation program; environmental management permits for solid refuse disposal, liquid effluent discharge, or air emissions; and a water licence for water management purposes such as water extraction. The following is a list of provincial authorizations, licences, and permits, currently in place for Kemess South, that will need to be amended by the province in order to proceed with KUG.

- Approving Work System & Reclamation Program (M206, Mines Act, MEMNG);
- Main Effluent Permit (PE 15335, Environmental Management Act, MOE) – Tailings Storage Facility and Associated Works, RBC, Mill and Accommodation Site Runoff and Open Pit Water, Waste Rock Dump;
- Refuse to the Ground/Active Waste Rock Dump (PR 14928, Environmental Management Act, MOE);
- Special Waste Consignor Identification Number (BCG07761, Environmental Management Act, MOE);
- Gas Permit (OTH00123, Safety Standards Act, BC Safety Authority);
- Annual Boiler Operator Certificates (Safety Standards Act, BC Safety Authority);
- Road Use Permit (10943, Transportation Act, MOT);
- Road Use Permits (01-7829.01-99 [Finlay Road], 01-7829.08-99 [Finlay – Osilinka Road], 01-9147.01-99 [Thutade Forest Service Road], 01-SS22414-97 [24 small road sections], Forest Act, MFLNR);
- Special Use Permits (S17447 [Minesite Access Roads], S22850 [Cheni Mine Road], Forest Practices Code of BC Act, MFLNR);

- Conditional Water License (East Kemess Creek, Serrated Creek and South Kemess Creek) (118554, Water Act, MOE);
- Conditional Water License Kemess Lake (110454, Water Act, MOE); and
- Mineral and Coal Exploration Activities & Reclamation Permit (MX-13-69 [expired December 31, 2012], Mines Act, MEMNG).

19.3.2 Federal Authorizations

Table 19.1 provides a preliminary list of the federal authorisations that could be required.

Table 19.1: Potential List of Federal Authorizations

Statute	Authorization	Agency	Purpose
Canada Transportation Act	License to operate an airstrip	Ministry of Transport	Authority to operate the Airport
Explosives Act	License No. 682 – the main Magazine storage of explosives and detonators No. 1168 – Magazine storage for avalanche explosives and detonators	Natural Resources Canada	Authority to manufacture and store explosives
Species at Risk Act	Authorization	Environment Canada	Protect species at risk or near risk
Fisheries Act	Authorization of work effecting fish habitat	Fisheries and Oceans	Any work that has the potential to impact waters defined as fish habitat
Radio Communication Act	Radio Licenses	Industry Canada	Provide for the operation of radio systems
Nuclear Safety Control Act	Radioisotope Licence 09-12586-99	Canadian Nuclear Safety Commission	Authorization for Nuclear Density Gauges/ X-ray analyzer

19.4 Environmental Considerations

The KUG project is in essence the addition of an undeveloped ore body to an existing mine operation. 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 project completed an environmental assessment prior to its commissioning in July 1996. 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. All of this data will be available to support the KUG project throughout any required EA and licensing requirements.

The existing environment for the proposed development, including the biological and physical components, has been characterized. The potential environmental impacts of the proposed project will have mitigative measures as necessary to address these impacts.

From a regulatory perspective the most significant potential impacts are associated with the long term management of waste rock, tailings, mine water, and process water. The potential impacts and proposed mitigation associated with these components will be managed as per existing management strategies in accordance with existing authorizations and permits for Kemess South. The physical infrastructure and additional management plans, necessary to support the existing management strategies, for the long term storage and mitigation is addressed in detail elsewhere in this study.

The existing waste management practices associated with the Kemess South project and those proposed for the KUG project represent industry best practices.

From a regulatory perspective there are no activities associated with the KUG project that will result in any potential environmental impacts that cannot be mitigated through the implementation of good engineering practices and management plans.

The KUG project is expected to result in the additional disturbance of approximately 65ha of terrestrial areas. This includes the 35 ha projected subsidence zone. The existing Kemess South mine footprint is 1,900 ha.

The KUG project surface infrastructure would be within the Kemess Creek watershed. Kemess Creek flows from the northeast to the southwest immediately south of the Kemess South minesite 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. Attycelley Creek flows westerly and enters the Finlay River, approximately 200 metres downstream from the Thutade lake outlet.

19.5 Social Setting

The Project is located in the Peace River Regional District of northern British Columbia. The closest communities by road are Germansen Landing (230 km south of the mine, population approximately 44) and Manson Creek (250 km south of the mine, population 40). The closest communities by air are the Aboriginal communities of Kwadacha (or Fort Ware) (approximately 70 km northeast of the mine, population 220) and Tsay Keh Dene (approximately 120 km east of the mine, population 377) and Takla Lake (approximately 180 km south of the mine, population 250).

The majority of the workers and contractors have historically been sourced from the Smithers/Bulkley Valley and Prince George areas. Employees have also lived in the following regions: Kamloops and Caribou District, Okanagan, Kelowna, Yale and Lillooet District; greater Vancouver, Vancouver Island; and Columbia and Kootenay Districts. The distribution of employee locations and its corresponding percentage of the total work force changed over the life of mine operations. Aboriginal people made up 16% of the former mine workforce for any given year.

The development of the KUG project would rejuvenate the economic and social benefits provided by the Kemess South mine. The KUG project is expected to provide employment, procurement, and government revenue benefits on a similar scale to those provided by the Kemess South mine. Between 100 and 475 jobs would be created during the 4 year construction and 13 year production periods. Employment would continue through the closure phase. 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 expected downturn in the forest sector, due to market conditions and the pine beetle infestation. Three Aboriginal traditional territories overlap the project

location – the Kwadacha, Tsay Keh Dene, and Takla Lake. These three groups identify themselves as the Tse Key Nay. In addition, the Gitxsan House of Nii Kyap traditional territory lies adjacent to the Project area along the western boundary.

Joe Bob Patrick family members are registered Takla Lake members, and hold a trapline that encompasses the existing and proposed Kemess mines. The Joe Bob Patrick family has traditionally used the area for hunting, trapping, fishing, and gathering. A trapline compensation agreement was in place during the operation of the Kemess South Mine, and a similar arrangement will be re-established for the Kemess Underground Mine.

19.5.1 Aboriginal Consultations

AuRico and the previous owner of the proposed project have made a concerted effort to work with the Tse Key Nay. Significant dialogue between the Tse Key Nay leadership and community members has taken place through a series of meetings and site visits. Three meetings were held in 2010, nine meetings in 2011, and two meetings in 2012. These efforts resulted in the signing of an Interim Measures Agreement on June 22, 2012. The Interim Measures agreement between the Tse Key Nay and AuRico is a cooperative agreement to cover advanced exploration with respect to the KUG Project. The agreement addresses a variety of topics including project permitting, environmental studies, business opportunities, and employment and training opportunities.

In addition AuRico representatives have supported the Tse Key Nay in their efforts to negotiate revenue sharing of the Provincial Mineral tax with the government of British Columbia.

Discussions between AuRico and the First Nations groups potentially impacted by the project are on-going.

19.5.2 Public Consultations

To date public consultations by AuRico have focused on the First Nations. The consultation efforts have not been limited to the Kemess Underground Project, but have included the Kemess South Mine and in particular it's Reclamation and Closure Plan.

The majority of feedback for the previously proposed Kemess North project coming from mine employees, northern BC businesses, local governments, chambers of commerce, economic development organizations, and mining industry organizations was positive and supportive of the project.

19.6 Closure and Post Closure Water Management

At closure, the mine will be flooded and water-retaining bulkheads established at the portal. The pipeline from the mine will also be re-configured to deliver mine water directly to the water treatment plant (rather than to the pit). The water treatment plant will be re-configured for long-term use in annual seasonal campaigns, projected to be operational during the months of June and July.

After milling and tailings deposition ceases, and with mine water no longer entering the pit, water quality in the South Kemess pit is expected to improve. Once it meets discharge criteria without treatment, pit water will be allowed to flow to Waste Rock Creek via the TSF closure spillway.

It is expected to take roughly 14 years for the underground mine to flood to the portal level. Thereafter, water will be withdrawn from the mine annually and campaigned seasonally through the water treatment plant. According to estimates provided by Lorax, the total annual volume entering

the mine after closure will be 0.9 Mm^3. Roughly that volume will need to be removed and treated each year. It is assumed that the treated water will initially be discharged to Attichika Creek.

If the quality of the treated mine water is demonstrated to be sufficient for discharge via the pit and its closure spillway, it would be routed there and the pipeline to the Attichika Creek decommissioned.

It must be noted that the KUG closure did not consider any site remedial work other than that directly associated with the KUG feasibility study. For example the Kemess South tailings and waste dumps and water runoff are not included. Closure, for the purposes of this report, involves treating water from the underground mine in perpetuity.

20 Capital

20.1 Summary

A total of $683M is estimated to be spent in capital over the life of the project as summarized in Table 20.1 (expressed in un-escalated Q4 2012 Canadian dollars). Full preproduction expenditure including capitalised pre-production operating costs is estimated to be approximately $471M to first ore production, and an additional $325M (maximum net outflow of $38M after considering revenue) to achieve commercial production. Ongoing sustaining capital is approximately $181M, including UG mobile equipment replacements, and ongoing underground development.

20.2 Basis of the Capital Cost Estimate

20.2.1 Currency

The estimates are expressed in Canadian dollars (C$) with an exchange rate to the US dollar of 1:1. No escalation or inflation factors were used in forward estimates. In some cases, historical escalation was applied in order to update some equipment quotations that have been supplied by Tetra Tech.

Table 20.1: Summary capital cost estimates.

Description	Year –6	Year –5	Year –4	Year –3	Year –2	Year –1	Year 1+	Total
Underground capital								
Mine development	4.6	19.3	40.4	17.4	10.9	7.1	7.8	107
Mining labour	0.1	0.3	11.9	18.6	19.3	22.4	25.3	98
Mobile equipment — purchases/rebuilds	1.8	0.0	17.8	20.6	8.6	17.1	72.5	138
Contractor indirects	2.4	9.7	9.7	9.7	4.9	0.0	0.0	36
Conveyor	0.0	10.0	10.0	10.0	0.1	0.1	0.0	30
Other	6.5	5.3	18.3	42.7	25.7	24.8	12.2	136
Processing plant	0.0	0.0	5.9	15.6	0.0	0.0	0.0	21
Pre-construction owner's costs	4.5	0.0	0.0	0.0	0.0	0.0		5
Camp renovation costs	1.9	1.2	1.3	0.0	0.0	0.0	0.0	4
Access road	5.9	8.8	13.2	0.0	0.0	0.0	0.0	28
Tailings storage facility	0.0	0.0	0.0	0.0	0.0	0.0	28.5	29
Surface admin	3.3	1.4	0.0	0.0	0.0	0.0	0.0	5
Power line along conveyor to U/G	1.2	0.0	0.0	0.0	0.0	0.0	0.0	1
Closure and water treatment capital	0.0	0.0	0.0	10.0	0.0	0.0	34.8	45
Total capital cost (excl. capitalised opex)	32	56	129	145	69	71	181	683
Capitalised mining operating costs	0.0	0.0	0.0	8.9	19.5	26.4	0.0	55
Capitalised processing operating costs	0.0	0.0	0.0	17.0	27.7	44.6	0.0	89
Capitalised G&A costs	12.6	15.8	24.5	30.7	32.9	32.5	0.0	149
Less pre-commercial net revenue	0.0	0.0	0.0	(28.1)	(114.8)	(200.6)		
Pre Production Net Pre-tax Expenditure	45	72	153	173	35	(26)		

20.2.2 Responsibility

The major contributors to the various sections are outlined in Table 20.2. The QP's responsible for the various tasks have been identified elsewhere.

Table 20.2: Capital cost estimate preparation responsibility.

Item	Responsibility
Owner's Costs	AuRico
Waste Management Facility	AuRico/AMEC
Site Roads	AuRico/AMEC
Conveyor Installation Surface and Underground	AuRico/Conveyor Dynamics/Tetra Tech/AMEC
Plant and Infrastructure	AuRico/KWM Consulting
Sustaining Capital	Tetra Tech
Mine Closure and Rehabilitation	AuRico/SRK
Environmental Studies/Permitting/Social License	AuRico/SRK/AMEC
Mine Design	SRK
Mine Development and Capital installation	AuRico/Tetra Tech
Mine Equipment	SRK/Tetra Tech
Mine Ventilation	SRK/Tetra Tech
Mine Electrical Reticulation	Tetra Tech
Contingency	AuRico/Tetra Tech/AMEC/Conveyor Dynamics/SRK

20.2.3 Exclusions from the Capital Cost Estimate

The following items were specifically excluded from the capital cost estimate:

- Project finance interest, leasing, off-take agreements, metal strips, reclamation bonding, and other financing arrangements and costs. 100% equity financing is assumed,
- Escalation and currency fluctuations,
- All license and royalty fees,
- Allowances for special incentives (schedule/safety or others),
- Sunk costs,
- Future scope changes and modifications of the proposed FS design,
- Flooding delays cost or resulting construction labour stand down costs, and
- Delays and redesign work associated with any antiquities and sacred sites.

20.3 Underground Mine Capital Cost Estimate

20.3.1 Summary of Assumptions for Estimate

It will take approximately 5 years to develop the KUG project, and access will be via a 4 km road from the existing plant site to the portal entrances.

Approximately 47,000 m of development will be required to access 100 Mt of ore that will be mined from the KUG deposit using block caving mining methods. Access to the mine will be by twin declines: one servicing the mine using trackless equipment and the other devoted to a conveyor

delivering 9 Mt of ore a year to a pre-existing processing plant. Ore will be delivered from the cave directly to a gyratory crusher. The material will be crushed and then placed on a 3.2 km conveyor that will deliver the ore to the surface. The material will then transferred to a 4 km overland conveyor that will deliver the ore to a stockpile near the processing plant.

SRK and AuRico provided all of the designs and scheduling for the development of the block cave orebody. The cost of that development was prepared by Tetra Tech.

AuRico sub-contracted Tetra Tech to provide development, construction, and fixed equipment designs, costs, and schedules. The underground mine capital costs were estimated from vendor quotes for all major equipment. Other costs were estimated based on mining cost service information and/or other factors based on Tetra Tech's experience. 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.

The underground capital costs are shown in Table 20.3.

Table 20.3: UG mine initial capital cost estimate.

Description	Cost to commercial production ($M)
Mine development	99.6
Mining labour	72.6
Mobile equipment — purchases/rebuilds	66.0
Contractor indirects	36.5
Conveyor	30.0
Mining staff	22.5
Maintenance labour	22.1
Mobile equipment — parts/materials	13.1
Crusher	9.7
Electrical power supply and distribution	8.3
Diesel fuel	7.6
Ventilation	5.9
Electricity	5.6
Propane	5.5
Fixed equipment — parts/materials	5.4
Maintenance staff	4.8
Surface waste handling	4.4
Other	8.6
Subtotal	428.1
Capitalised mining operating costs	54.8
Total	482.9

20.3.2 Underground Mine Development

Underground mine development costs were supplied by Tetra Tech and are presented in Table 20.3. These costs were developed from first principles, and AuRico labour rates were taken from the Young Davidson Mine and adjusted for the BC work environment.

The costs were set up as a bid document from a contractor and checked against another bid. Equipment and material costs were from vendor quotes; labour, fuel, and power costs from AuRico;

and other costs were based on Tetra Tech's experience. Fuel was assumed to cost $0.95/L and power was assumed to cost $0.0405/kWh.

20.3.3 Underground Mine Mobile Equipment

Mobile equipment costs were developed from vendor quotations and increased by 5% to cover initial parts stock and a further 4% for freight and on-site assembly. Equipment life cycle operating hours were based on manufacturers' recommendations and on SRK's and Tetra Tech's experience.

20.3.4 Underground Mine Infrastructure

Mine infrastructure costs were developed from first principles. Mine material and installation costs were provided by Tetra Tech, Mine Ventilation Services (MVS), and Conveyor Dynamics, Inc. (CDI). MVS provided the equipment costs for the ventilation and heating systems, and Tetra Tech provided the development/excavation and installation costs. Similarly, CDI provided the equipment costs for the conveyors, and Tetra Tech provided the excavation and installation costs for the underground and surface conveyors. The costs for the other main equipment were provided from vendor quotes, and Tetra Tech provided the development and installation costs.

20.4 Plant and Infrastructure

20.4.1 General

This estimate covers the design and construction of the waste rock treatment facilities, the road to the portal entrance, and modifications to the process plant and surface infrastructure. KWM Consulting Inc. provided estimates for the process plant modifications. The waste rock treatment facility (TSF) and the road to the portal entrance were designed by AMEC. AuRico provided the costs for the other surface infrastructure modifications as well as the G&A.

20.4.2 Process Plant modifications

AuRico subcontracted KWM Consulting Inc. to provide a feasibility level study to determine the requirements to convert the existing plant to process 9 Mt of ore a year from the KUG block cave project.

The estimated cost to convert this facility can be found in Table 20.4.

Table 20.4: Process plant capital modifications.

Mill initial capital costs		Total
Refurbish grinding circuit (including SAG mill & ball mills)	$M	1.7
M10000 IsaMill™	$M	16.2
Flotation rebuild	$M	1.9
Miscellaneous process equipment repairs	$M	1.7
Total (including freight, duty, taxes and 10% contingency)	**$M**	**21.5**

20.4.3 Camp Upgrade and Site-based General and Administration

AuRico has reviewed the status of the present site facilities and has estimated the necessary expenditures for upgrade and repair. General and Administrative pre-operating costs were also

developed by AuRico. Those costs incurred prior to commencement of production are shown in Table 20.5.

Table 20.5: Capitalised general & administrative.

Capitalised site G&A costs		Year –6	Year–5	Year–4	Year–3	Year–2	Year–1	Total pre-production
Flights	$M	2.34	2.60	4.19	6.37	6.64	6.33	28.48
Camp	$M	2.04	2.26	3.66	5.66	5.90	5.63	25.16
Admin costs	$M	1.49	1.98	8.36	10.49	11.81	11.81	45.95
HR/camp costs	$M	6.03	8.05	6.60	6.12	6.25	6.25	39.30
Environment	$M	0.70	0.94	1.68	1.93	1.93	1.93	9.09
Total G&A	$M	12.60	15.83	24.50	30.57	32.53	31.95	148.0

20.4.4 Waste Rock and Water Treatment Facilities

AMEC was subcontracted by AuRico to generate a feasibility level study of the requirements for a tailings and waste storage facility. All of the development waste and tailings will be deposited in the former Kemess South open pit. It has already been permitted and was used to store tailings from the milling of low grade stockpiles. A summary of the capital costs (inclusive of 15% contingency) can be found in Table 20.6.

Table 20.6: Waste rock storage and water treatment facilities.

TSF LOM capital costs		LOM
Final design site investigation	$M	0.9
Diversion channel	$M	0.7
Starter dam to 1,260 m	$M	11.8
Three separate lifts	$M	13.9
Closure spillway	$M	0.7
Borrow and East Dam reclamation	$M	1.3
Total	**$M**	**29.3**

20.4.5 Earthworks and Road/Conveyor Access to Portal Entrance

AMEC was sub-contracted by AuRico to generate a feasibility level study of the cost to create road and conveyor access to the portal entrance (a distance of 4.5 km). The estimated costs (inclusive of 40% contingency) for that facility are shown in Table 20.7.

Table 20.7: Earthworks and road access to portal entrance.

Access road construction		LOM total
Site investigation and design	$M	0.7
Portal access segments 1,2,3,4,5,6	$M	23.2
Vent raise access road	$M	1.8
QA/QC (Year –5 and Year –4)	$M	1.5
Reclamation of portal area/Conveyor alignment	$M	1.4

Access road construction		LOM total
Grand total	**$M**	**28.6**

20.4.6 Indirect Cost Development

EPCM

AuRico has advised SRK that it intends to directly manage all contractors so no external EPCM costs are attributable to this project.

Freight

Tetra Tech and SRK have included a 4% freight allowance for all mobile equipment purchased for the underground. All other costs are inclusive of freight.

Spares

Tetra Tech and SRK have included a 5% allowance for all mobile equipment purchased for underground.

The processing plant is a pre-existing facility and suitable spares already exist for it. New liners for the grinding circuit have been provided for in the process plant modification estimates.

Surface Mobile Equipment

AuRico has a pre-existing fleet of equipment that will provide most of the services required. To upgrade the existing fleet and provide access to the underground, $4.7M in capital was estimated.

Temporary Construction Facilities

Kemess is a former producing mine and has permanent facilities available for construction and development. AuRico has included capital ($4.4 M) for the renovation and upgrade of camp and dry facilities.

Tetra Tech has provided for the construction and installation of temporary facilities at the portal entrance for offices and maintenance facilities.

20.4.7 Escalation

There was no explicit escalation of costs or revenues for this project. All costs and revenues are expressed as 2012 Canadian dollars as the base case.

20.4.8 Owner's Costs

Owner's costs were estimated by AuRico. The capitalised G&A for site costs is estimated to be $149M for the period leading up to commercial production. The owner's cost includes the following items:

- AuRico Management staff and associated equipment & supplies,
- A pre-construction cost of $1.5M to allow for exploration drilling for 2013,
- Hiring and training of crews in advance of commissioning,
- Sustainability commitment expenditures,
- Environmental monitoring and testing during the pre-production period, and
- Operations personnel camp, messing, and transportation during construction and pre-commissioning.

There was no allowance for corporate costs for the first three years of development, but there was an allowance of $9.5M for the last two years of development and pre-production.

Excluded until clarified:

- Legal costs and fees,
- On-going metallurgical test-work,
- Reclamation bonding, and
- Construction insurances.

20.4.9 Contingencies

Contingency provides for the risk of increases in costs due to as-yet explicitly unidentified details.

Contingency does not include allowance for changes in functional specification arising from project design changes or from variation in underlying assumptions that may become apparent during ongoing testing and design.

Other changes that are not expected to be covered by contingency often arise from external or unpredictable circumstances. These include:

- Extreme escalation of engineering and field construction labour above the baseline,
- Extreme abnormalities in industrial relations,
- Extreme change in market conditions and therefore equipment and material prices, and
- Extreme weather or adverse political or regulatory developments.
- KWM Consulting Inc recommended a contingency of 10%, or $1.95M for the modifications to the processing plant.
- AuRico recommended a contingency of 10%, or $0.4M for the surface infrastructure upgrade.
- AMEC recommended a contingency of 15%, or $3.95M for the construction of facilities related to tailings and waste rock disposal.
- AMEC has also recommended a contingency of 40%, or $7.8M for the construction of the access road and conveyor right of way to the portal. This contingency has been applied in the absence of a fully definitive soils analysis that could not be completed because the necessary permits were not in place.
- Conveyor Dynamics recommended a contingency of 15%, or $1.99M for the purchase of the overland and underground conveyors.
- Tetra Tech recommended a contingency of 5%, be applied to the contract mining labour and consumables.
- The amount of contingency is ultimately the client's decision. This decision is based on the client's perceived risk for the project as well as their willingness to accept risk.

20.4.10 Plant Start-up and Commissioning

Pre-production carries all of the development and capital expenses from underground.

The processing plant is not scheduled to start production until sufficient ore is available for limited production. The first 1 Mt of development ore is scheduled to be processed in the last quarter of

Year –2. Operating costs for processing at partial rates were estimated assuming a roster of 2 weeks on – 2 weeks off. During shutdown periods, 55% (rather than 50%) of fixed costs were assumed to apply to reflect the cost of maintenance.

21 Operating Cost Estimate

21.1 Summary and Common Assumptions

Summaries of the KUG LOM and unit operating cost estimates (OPEX) from the commencement of commercial production are shown in Table 21.1.

Table 21.1: Operating costs

Operating costs	LOM ($M)	Unit costs ($/t)
Underground	554.6	$6.07
Concentrator	456.6	$5.00
G&A	319.4	$3.50
Total operating costs	1,330.6	$14.56

The major operating assumptions used for the cost estimation are:

- Diesel fuel price at $0.95/L.
- Power cost at $0.0405/kWh.
- Manpower: AuRico labour rates from the Young Davidson mine were used as the basis for the costs, adjusted for the BC working environment.
    - 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 the site for all employees, is from Smithers or Prince George, with the exception of senior staff.

21.2 Underground Mining Operating Costs

SRK and Tetra Tech jointly developed the mine operating costs. SRK estimated the direct block caving mining costs, which can be found in Table 21.3. Those costs have been developed on a first principle approach and they have been collated under their respective activities.

Tetra Tech estimated indirect underground mining, also with a first principle approach, but they have not been collated by activity. Crushing, conveying, ventilation, pumping, and all maintenance activities are grouped under labour and the various consumables, and can be found in Table 21.2.

Consumables and equipment operating costs were obtained from suppliers for most items and AuRico, SRK, and Tetra Tech experience used for the remaining items.

Mobile operating equipment maintenance costs were increased by 25% over suppliers' estimates to account for damage not anticipated by those suppliers.

The total operating costs from the commencement of commercial production averages $6.07/t.

21.2.1 Indirect Mining Costs

The expense type breakdown of the estimated indirect mining costs from commencement of full commercial production prepared by Tetra Tech is shown in Table 21.2.

Table 21.2: Indirect underground mining costs

Indirect underground mining costs	LOM costs ($M)	Unit costs ($/t)
Indirect labour (hourly & staff)	220.1	$2.40
Electricity/propane	40.2	$0.44
Mobile equipment/fixed equipment — parts/materials/fuel	30.8	$0.34
Mine development/rehab	13.1	$0.15
Total other underground costs	304.2	$3.33

21.3 Block Caving Production Costs

SRK and AuRico designed the KUG block cave. Costs were developed from first principles and SRK's experience, where required. The costs in Table 21.3 reflect the cost to drill and blast the undercut and to remove 50% of the ore. It includes the cost to drill and blast the drawbells. Once the cave has been initiated, it includes the cost to muck the ore from the drawpoints. Every drawpoint has been given a tonnage profile from PCBC These costs reflect the total distance to the gyratory crusher for every tonne of ore based on the tonnage in every drawpoint and the distance of that drawpoint to the crusher.

These costs include all of the labour and materials to drill and blast the drawbells and undercut. It includes all of the labour and parts for the LHDs to tram the material from the undercut, drawbells, and cave. It excludes supervision, maintenance labour, and support labour that are included in indirect costs. These costs from full commercial production are reflected in Table 21.3.

Table 21.3: Direct block cave mining costs.

U/G direct production costs by process	Costs ($M)	Unit costs ($/t)
Ore development and secondary breakage	29.3	$0.32
Production mucking	122.7	$1.34
Operating labour (hourly)	98.4	$1.08
Total underground production costs	250.4	$2.74

21.4 Commentary on Selected Costs

21.4.1 Electrical Costs

Electrical consumption estimates were made based on the installed kilowatts and then factored by usage and load. The average electrical load for the UG mine is estimated to average just less than 6 MW, at a cost of $0.0405/kWh. The main intake and exhaust fans and conveyors will consume 75% of the total power.

21.4.2 Propane Costs

Mine Ventilation Systems (MVS) collected temperature data and used that data to design the heating requirements for the KUG ventilation system. A cost of $0.50/L of propane was given by AuRico to be used as a basis for calculations.

The total annual cost averaged $1.5M per year.

21.4.3 Labour

The labour cost estimate was built on the assumption that no contract mining would take place. The contract labour used for the initial capital development is replaced in the schedule as owner operated crews are trained. Contractors will have been demobilized prior to commercial production commencing.

Development of the manpower productivity is based on the available effective time as shown in Table 21.4.

Table 21.4: Development of underground shift schedule.

Operating factors	Unit	Quantity
Underground production		
Days/year	days/year	365
Crusher total maintenance	days/year	32
Crusher operating days	days/year	333
Mine operating days	days/year	333
Average mining rate	t/yr	9,000,000
Nominal mining rate	t/day	27,027
Shift data		
Working days a week	ea.	7
Shifts per day	ea.	2
Shift length	h	11

21.4.4 Miscellaneous Underground Costs

Other operating costs were relatively minor with the largest being crushing (excluding labour). It is projected to cost $0.75M/yr and 70% of that cost is attributable to power costs.

Dewatering (excluding labour) is projected to cost $0.23M/yr and 80% of that cost is attributable to power costs.

21.5 Process Plant

AuRico developed process plant costs based on actual operating experience when the Kemess South open pit was operational. The historical actual costs have been inflated by 5% to allow for inflation and then appropriate factors were applied to account for lower production. Many of the costs have a fixed component. As such, a straight ratio cannot be applied between the actual and proposed production. AuRico reviewed the material and labour for each section of the process plant under full operating conditions and applied the appropriate factors.

The cost per tonne for full production is not the total LOM cost per tonne, because of the ramping up of production and the fixed costs associated with that ramp up. The total LOM cost per tonne for every tonne of ore (from commencement of commercial production) averages $5.00 rather than $4.96 based on full production at 9 Mtpa (Table 21.5).

Table 21.5: Process operating costs by function.

Mill operating costs by category	Cost per year at 9 Mtpa ($M)	$/t milled
General & administration	9.0	$1.00
Overland conveyor & reclaim	1.8	$0.19
Grinding	25.4	$2.83
Flotation	2.9	$0.32
Other (assay lab/tailings/reclaim/metallurgy)	5.5	$0.62
Total mill summary at 9 Mtpa	**44.6**	**$4.96**

Mill operating costs for LOM	LOM costs ($M)	$/t milled
Total mill opex (LOM)	**456.6**	**$5.00**

21.6 General and Administration (G&A)

AuRico developed costs for general and administration (G&A) based on actual operating experience from Kemess South. These are shown from commencement of full commercial production in Table 21.6.

Table 21.6: General and administration (G&A).

On-site G&A	Costs ($M)	Unit costs ($/t)
Flights & camp	109.3	$1.20
Admin costs	190.0	$2.08
Environmental	20.2	$0.22
Total G&A costs	319.4	$3.50

22 Economic Evaluation

22.1 General

The economic analysis was undertaken using a Microsoft™ Excel® spreadsheet based discounted cash flow (DCF) model that modelled cash flows by quarter (three month periods). The modelling was undertaken in 2012 Canadian dollars (real). Mid-period cash flows and a valuation date of 1 January 2013 were assumed.

A discount rate of 5% (real) was used after discussions with the client.

No currency escalation was used for any of the calculations on instruction from the client.

22.2 Revenues

22.2.1 Production Schedule

Revenues were derived from production schedules developed by AuRico see Table 15.8[1]. The schedules provided tonnes of ore and grades of economic minerals to enable revenues to be derived.

22.2.2 Commodity Prices

Commodity prices used for the base case are shown in Table 22.1.

Table 22.1: Commodity prices

Base case pricing		Long-term
Copper	$/lb	**$3.00**
Gold	$/oz	**$1,300**
Silver	$/oz	**$23.00**

22.2.3 Mill Recoveries

Mill recoveries were estimated by KWM Consulting Inc. and AuRico. The modelled recoveries are shown in Table 22.2.

Table 22.2:Mill recoveries

Modelled recovery to concentrate	
Copper recovery to concentrate	91%
Gold recoveries to concentrate	72%
Silver recoveries to concentrate	65%

22.2.4 Treatment and Refining Costs, Payable Metal Assumptions, and Freight

These costs summarized in Table 22.3 and Table 22.4, were supplied in a marketing study commissioned by AuRico. Various alternate destinations were studied and compared for the most favourable combination of freight costs and economic terms. For the base case, a destination of Japanese, Korean, and Northern Chinese smelters was assumed.

Table 22.3: Treatment and refining costs.

TCRC and freight	Unit costs	LOM costs ($M)
Treatment charge	$80 per dmt	92.9
Copper refining costs	$0.08 per payable lb	43.0
Gold refining costs	$6.00 per payable oz	7.6
Silver refining costs	$0.40 per payable oz	1.5
Total treatment and refining costs		145.1
Freight costs (base case)	$232.16 per dmt	269.7
Total TCRC and freight		414.9

Table 22.4: Payable metals assumptions and net costs.

Payable metals deductions and costs	Effective payable rate	LOM costs ($M)
Copper	*95.5%*	78.1
Gold	*97.5%*	42.9
Silver	*90.0%*	10.1
Total effective payable costs		**131.0**

22.3 Capital and Operating Costs

The capital and operating costs are discussed in section 20 and section 21 respectively.

22.4 Tax and Tax Depreciation

Taxes and tax depreciation were modelled by treating the project as a standalone business. That is, no existing tax losses or credits were considered. Any losses were carried forward to be offset against future revenue.

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%.

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).

Tax depreciation for capital expenditure was estimated in accordance with 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.

The estimate of the LOM tax payable under these categories is shown in Table 22.5

Table 22.5: Federal and provincial taxes.

Federal and provincial taxes		LOM
BC provincial corporate tax	$M	63.8
Federal corporate tax	$M	87.0
BC minerals tax	$M	86.0
Total taxes	**$M**	**236.7**

22.5 First Nations Project Participation

AuRico advised that an interim measures agreement (IMA) with the relevant First Nations groups was 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.

22.6 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 $134M and a post-tax internal rate of return of 10%.

22.6.1 Payback Periods

The cumulative project cash flow becomes positive in real terms 3.5 years after commencement of production. On a discounted basis at 5%, the project has returned the investment 10 years after commencement of construction (Table 22.6).

Table 22.6: Payback periods.

Payback periods (years)	From construction start	From commercial production start
Discounted payback @ 5%	10.0	5.0
Non-discounted payback (real)	8.5	3.5

22.6.2 Sensitivity Analysis

Sensitivity to Price

The following table (Table 22.7) shows sensitivity of the project's post-tax NPV to both gold and copper price assumptions. 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 modeled.

Table 22.7: Sensitivity to revenues.

		Gold price ($/oz)							
		1000	**1100**	**1200**	**1300**	**1400**	**1500**	**1600**	**1700**
Copper price ($/lb)	**2.25**	(178)	(115)	(59)	(11)	35	81	127	173
	2.50	(112)	(57)	(9)	37	83	129	175	220
	2.75	(55)	(7)	39	85	131	177	222	268
	3.00	(5)	42	87	**134**	179	224	270	316
	3.25	44	90	136	182	227	272	318	363
	3.50	92	138	184	229	275	320	365	410
	3.75	140	186	231	277	322	367	412	457
	4.00	189	234	279	325	370	414	459	504

Sensitivity to Costs

The following table (Table 22.8) shows the sensitivity of the project's post-tax NPV to variation in overall costs assumptions.

Table 22.8: Sensitivity to capital costs.

		Capital costs							
		.1.1 30%	**.1.2 20%**	**.1.3 10%**	**0%**	**10%**	**20%**	**30%**	**40%**
Operating costs	**.1.4 30%**	458	422	386	350	314	278	240	204
	.1.5 20%	387	351	314	278	242	205	168	131
	.1.6 10%	316	279	243	206	169	132	94	57
	0%	244	207	171	**134**	96	58	20	(18)
	10%	173	136	99	61	23	(15)	(54)	(95)
	20%	102	65	27	(12)	(51)	(92)	(137)	(182)
	30%	31	(7)	(47)	(89)	(134)	(180)	(234)	(287)
	40%	(42)	(86)	(131)	(183)	(236)	(290)	(344)	(397)

23 Adjacent Properties

There are no properties currently under development or exploitation adjacent to the Kemess project area.

24 Other Relevant Data and Information

There is no other relevant data available about the KUG Project.

25 Interpretation and Conclusions

This section examines the factors that could have an influence on KUG. It generates a positive cash flow with an IRR of 10% based on metal prices of $1,300/oz Au and $3.00/lb Cu. This is the base case, and there is potential for this outcome to change. This section discusses the various factors that could impact on the base case.

25.1 Metal Prices, Operating Costs, and Capital Costs

Revenues may be higher/lower than projected. SRK believes the metal prices that have been chosen are realistic price scenarios reflecting industry norms. They are less than current prices and aligned with (or less than) three year moving averages.

Capital and operating costs may be higher/lower than estimated. The capital and operating costs are based on quotes from suppliers, the experience of AuRico at its Kemess South and Young Davidson Mines, and the experience of its consultants with prior projects.

25.1.1 Reserves

There is the opportunity to significantly increase the reserves beyond that stated in this report. Mineral Resources are available to support a larger mineral reserve. These reserves are constrained by the size of the facility that can receive the tailings. This tailings facility can be optimized to determine how much material can ultimately be placed into it.

25.1.2 Underground Design and Infrastructure

The potential exists to increase production beyond 9 Mtpa based on the number of drawpoints. Production is currently constrained by underground ventilation and tramming limitations, as well as processing plant throughput.

Cavability and fragmentation will vary from base case modelling. There is always a degree of uncertainty scaling up empirical and numerical modeling. Further data gathering for geotechnical analysis as development advances will be necessary, but fragmentation will vary over the life of a cave and that profile cannot be defined with certainty. This may lead to an underground cave ramp up schedule that varies from the base case with associated downside risks.

The ingress of water into the underground is likely to increase when the cave propagates into the broken zone. It is anticipated that the project may experience a production loss of one to two weeks during the life of the mine while the inflow water stored in the ventilation level is removed.

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.

A fully integrated mud rush risk assessment is required prior to commencement of caving. The presence of mud rush risk does not make mining infeasible, but it does lead to requirements for increased monitoring and mitigation plans. The use of autonomous LHDs is a potential option for mitigation.

Given experience gained in the first years of production, it may be possible to increase the planned 15 x 15 m drawpoint spacing thereby reducing the number of drawpoints and associated development costs.

There may be cost, schedule, operational, and ventilation benefits associated with replacing the planned intake air raise with an intake air decline. This issue requires further study.

Due to the cost estimated for the planned portal access road and overland conveyor, the possibility of a tunnel should be evaluated. Schedule impacts should also be considered.

The extraction level layout is considered suitable for use of autonomous LHDs. This technology has the potential to reduce operating labour, LHD maintenance, and LHD capital costs, and should be investigated.

Development is the largest contributor to the capital costs. A detailed review of the proposed infrastructure has the potential to reduce those costs. Internal ramps and portions of the perimeter drift on the undercut level can possibly be reduced

25.1.3 Processing Facility

There is a risk that the planned mill throughput of 9 Mtpa may not be achievable because variable hardness in the orebody may limit SAG mill capacity. It is recommended that more hardness tests be conducted to confirm the representativeness of the assumptions that have been made. KWM Consulting has demonstrated, through its work using the BWI method of analysis, that the hardness of the ore would not create a SAG mill constraint. Nevertheless, it is recommended that further analysis be done using other methods specific to SAG mills to confirm these results.

Provision has been made to add another mill to the re-grind circuit. It is recommended that the manufacturer of that mill be consulted to engineer the final details.

25.1.4 Water Management

Water management will require a water treatment facility. The economic model has taken into account the water treatment facility and annual operating costs.

25.1.5 Surface Infrastructure

Tailings storage capacity is limited to 100 Mt. Additional capacity is potentially available in Kemess South open pit, but this would require raising the East Dam, constructing an additional dam in the saddle area, and reconfiguring the closure spillway.

25.1.6 Environmental Permitting and Closure

Community concerns represent a risk to the project; however, AuRico and its predecessor Northgate Minerals have engaged with the surrounding communities to inform them of plans and gain their support for the project. SRK has been advised that the results to date are positive. Community engagement needs to continue in order to avoid delays in permitting and approvals.

This project will require both provincial and federal approvals, and may require federal and/or provincial EA.

It must be noted that the KUG closure did not consider any site remedial work other than that directly associated with the KUG feasibility study. For example, the Kemess South tailings and waste dumps and water runoff are not included. Closure of these facilities is covered under the existing Kemess closure plan. Closure for the purposes of this report primarily involves treating water from the underground mine in perpetuity.

25.1.7 Other

There is a general shortage of skilled trades and underground miners in British Columbia. There is also a specific lack of personnel with experience in block caving operations. A significant effort will be required to ensure suitably qualified personnel are available for all aspects of the planned operation, particularly the development phase.

Avalanches may affect the mine facility. AuRico has put in place comprehensive mitigation measures to manage any potential risk.

Inadequate care and maintenance may affect the start-up of major infrastructure. AuRico has an ongoing program of care and maintenance executed by a dedicated crew, with this programme planned to be continued through project development

26 Recommendations

This report did not try to optimize this project. There was only one main trade-off study that was done (regarding cave design), and others are warranted. The following recommendations can increase confidence and value in this project. This section identifies those opportunities in point form. Additional details are found in the body of the report.

Mineral Reserves:

- There is the opportunity to significantly increase the reserves beyond that stated in this report. Mineral resources are available to support a larger mineral reserve. These reserves are constrained by the size of the facility that can receive the tailings. A trade-off study to optimize the size of the tailings facility is all that is required to increase mineral reserves. No costs are anticpated to be required other than the cost of the trade off study for the Tailings Management Facility (TMF).

Underground Design and Infrastructure:

- It is recommended that more geotechnical information be gathered and analyzed as development advances (including driving drifts throughout the ore before design parameters are finalized). Cavability and fragmentation will vary from base case because there is always a degree of uncertainty scaling up empirical and numerical modelling. Additional data would reduce that uncertainty and enhance confidence in the representativeness of the given information. The technical staff required are included in the project capital costs.
- A fully integrated mud rush risk assessment is required prior to commencement of caving. The presence of mud rush risk does not make mining infeasible. What it does mean is that programs and policies will have to be put into effect to manage these events. The cost of this additional work is anticipated to be $30,000.
- It is recommended that the use of remotely operated LHDs be considered. This is a proven technology that is routinely used in the mining industry. The extraction level layout is considered suitable for their use. This technology has the potential to reduce operating labour, LHD maintenance, and LHD capital costs, and improve safety.
- It is recommended that a trade-off study be conducted to determine the capital and operating benefits associated with replacing the planned intake air raise with an intake air decline. The potential exists to save time on the development schedule and up to $20M in operating costs if heating costs can be eliminated or reduced. This issue requires further study.
- Development is the largest contributor to the capital costs. It is recommended that a detailed review of the proposed infrastructure be undertaken to see if those costs can be reduced. Internal ramps, portions of the perimeter drifts, and even the maintenance facility are examples of potential cost reductions without compromising the project. Existing in house resources will be used for this trade-off study and so no additional costs are anticipated.

Processing:

- It is recommended by SRK that more hardness tests be conducted on the KUG orebody to confirm the representativeness of the work that has been completed and the appropriateness of the assumptions that have been made.
- KWM Consulting has demonstrated, through their work using the BWI method of analysis, that the hardness of the ore should not be a constraint. Nevertheless, it is recommended by SRK that further analysis be done using other methods specific to SAG mills to confirm these results.
- SRK has noted that the existing re-grind capacity will not be sufficient to handle the greater rougher concentrate mass pull (20–23% of the mill feed) combined with the finer grind size requirements for the underground ore (P80 @ 15 µm or finer). Provision has been made to add another mill to the re-grind circuit. Modifications to the primary grinding circuit are challenging to retrofit and cannot be assured to compensate for a lack of primary mill power. SRK recommends that specific test work be conducted by the manufacturer of that mill to verify the tonnage, feed size, and expected product size.

Surface Infrastructure:

- It is recommended that a trade-off study be completed to determine if additional tailings storage capacity is available and economically viable in the Kemess South open pit. A decision was made in this study to minimize capital. A trade-off study might demonstrate that material can be placed in this facility without major capital cost implications.
- It is recommended that a trade-off study be conducted to determine if a tunnel can replace the planned portal access road and overland conveyor. There are several impacts that need to be considered including the development schedule to production, capital costs, and operating costs. The manpower savings could be significant in terms of the potential savings in travel time to and from the underground workplaces.

Environmental Permitting and Closure:

- Community concerns represent a possible risk to the Project, however AuRico and its predecessor Northgate Minerals have engaged the Tsay Keh Nay to develop a positive relationship that will be mutually beneficial, and thereby gaining support for the advancement of the Kemess Underground Project. When the regulatory review process has been finalized, AuRico will initiate broader community engagement in the region.
- This project will require both provincial and federal approvals, and may require federal and/or provincial EA.
- It must be noted that the KUG closure did not consider any site remedial work other than that directly associated with the KUG feasibility study. For example the Kemess South tailings and waste dumps and water runoff are not included. Closure of these facilities is covered under the existing Kemess closure plan. Closure for the purposes of this report primarily involves treating water from the underground mine in perpetuity.
- SRK recommends that AuRico continue to be proactive in its efforts to engage the community and First Nations so that it can avoid delays in permitting and approvals.

The costs of the studies discussed above are included in Table 26.1.

Table 26.1: Budget for Recommendations

Recommendations Budget Forecast		
Description	**Work Requirements**	**Cost $**
Underground Design And Infrastructure		
Ongoing geotechnical evaluations	This work is ongoing and included in capital costs	$0
Mudrush assessment		$30,000
Remotely operated lhds		$20,000
Trade off study for main ventilation intake raise		$20,000
Trade off study to reduce development	Will be done with existing in house resources.	$0
	Subtotal	$70,000
Processing		
Hardness tests		$25,000
Specific sag mill tests		$50,000
Manufacturer regrind testwork		$25,000
	Subtotal	$100,000
Surface Infrastructure		
Trade off study on tailings storage capacity		$100,000
Trade off study on underground access		$130,000
	Subtotal	$230,000
Environmental Permitting And Closure		
Community engagement	This work is ongoing and included in capital costs	$0
	Subtotal	$0
	Total	$400,000

27 Acronyms and Abbreviations

Area		**Acronyms**	
m^2	square meters	BC EAA	British Columbia Environmental Assessment Office
Mm^2	Million square meters	BHOD	Best height of draw
ha	hectare	BWI	Bond Work Index
		AMSL	Above mean sea level
Compounds and Elements		BC EAA	British Columbia Environmental Assessment Office
Au	Gold	BHOD	Best height of draw
Ag	Silver	BWI	Bond Work Index
Cu	Copper	CDI	Conveyor Dynamics Inc.
Distance		CEAA	Canadian Environmental Assessment Act
µm	micrometer (micron)	DCS	Distributed control system
km	kilometer	DFN	Discrete fracture network
mm	milimeter	EA	Environmental assessment
m	meter	EAO	Environmental assessment office
"	inch	EPCM	Engineering Procurement Construction Management
ft	foot	EPS	Enhanced production scheduler
Mass		G&A	General and administration
kg	kilogram	G&T	G&T Metallurgical Services Ltd.
g	gram	GCMP	Ground control management plan
t	metric tonne	HDS	High density sludge
kt	kilotonne	HOD	Height of draw
lb	pound	HR	Hydraulic radius
Mt	Million tonne	HSSE	Health, Safety, Security, and Environmental
oz	troy ounce	IRMR	In situ rock mass rating
Volume		IRS	Intact rock strength
L	litre	IBA	Impact Benefits Agreement
m^3	cubic meter	LHD	Load-haul-dump
m^3/s	Cubic meter per second	MPM	Manual production management
Other		MRMR	Modified rock mass rating
^oC	degree Celsius	MVS	Mine Ventilation Services
hp	horsepower	NAG	Non-potentially acid generating
h	hour	NSR	Net smelter return
kW	kilowatt	P80	80% passing size
kWh	kilowatt hour	PAG	Potentially acid generating
M	Million	PEA	Preliminary economic analysis
g/t	grams per tonne	PLC	Programmable logic controller
Mtpa	Million tonne per year	QA/QC	Quality Assurance/Quality Control
t/d	tonne per day	RMR	Rock mass rating
ktpd	kilotonne per day	RQD	Rock quality designation
kWh/t	kilowatt hour per tonne	SAG	Semi-autogenous grinding
s	seconds	SRM	Synthetic rock mass
Acronyms		STOL	Short takeoff and landing
AMSL	Above mean sea level	TDR	Time domain reflectometry
ATV	Acoustic televiewer	TMF	Tailings management facility

28 References

AMEC Environment and Infrastructure, a Division of AMEC of Americas Limited, Kemess Underground Project (KUG) Feasibility Study Surface Infrastructure Preliminary Design Report, (AMEC File VM00575.2.800), submitted to AuRico Gold Inc., November 9, 2012.

AMEC Environment and Infrastructure, a Division of AMEC of Americas Limited, Kemess Underground Project (KUG) Feasibility Study Mine Waste and Water Management Design Report, (Reference VM00575.2.700), submitted to AuRico Gold Inc., November 9, 2012.

Amtel Ltd., Kemess North Preliminary Metallurgical Testing, June 2001.

Amdel Ltd., 2002, Characterization of Copper and Gold occurrences in Composites of the Kemess North Deposit, 2002.

Amdel Ltd., Gold Deportment in Copper Cleaner Scavenger Tailings, May 2003.

Atomic Energy of Canada Ltd. (2004). In-situ Rock Stress Determination Project at Kemess Mine South Pit. Draft Report submitted to Northgate Exploration Ltd., February 2004.

AMC Consultants Canada Pty Ltd., Preliminary Economic Assessment of the Kemess Underground Project, Report submitted to Northgate Minerals Corporation, (Report No. 711006), July 22, 2011.

AMC Consultants Ltd., (2011). Preliminary Geotechnical Assessment for Block/Panel Caving of the Kemess Underground Deposit, (Report No. 711006). Report submitted to Northgate Minerals Corporation., September 2011.

Canadian Environmental Assessment Act, 2012 (S.C. 2012, c.19, s.52)

Conveyor Dynamics Inc., Kemess Underground Feeder and Overland Conveyor Design Report, (Report CDI Job #: 520), Report submitted to AuRico Gold Inc., August 11, 2012.

Conveyor Dynamics Inc., Kemess Underground Feeder and Overland Conveyor CAPEX/OPEX Estimate, (Report CDI Job #: 520), Report submitted to AuRico Gold Inc., October 18, 2012.

Exen Consulting Services, October 2012, report titled ~~-K~~emess Copper Concentrates: Marketing & Logistics.

Gemcom Software International Inc., PCBC Modelling Report for the Kemess Underground, August 2012

Government of Canada and British Columbia news release, March 7, 2008

G&T Metallurgical Services, Assessment of Five Composite Samples from the Kemess Underground Project, (Report KM2911), Report submitted to AuRico Gold Inc., September 19, 2011.

G&T Metallurgical Services, Ore Hardness Testing – Kemess Underground Project AuRico Gold Inc., (Report KM3442), Report submitted to AuRico Gold Inc., September 5, 2012.

Itasca Consultants S.A.S, Fragmentation Analysis of Intact and Fractured Rock for the Kemess Underground Project (KUG), (Report 12R-010/A) submitted to SRK, February 15, 2013.

Itasca S.A., Numerical Modeling for Assessing Caveability at Kemess under Block Caving Mining, (Report 12-2722-30D) submitted to SRK, June, 2012.

Itasca Consulting Group, Inc. (2009) FLAC3D – Fast Lagrangian Analysis of Continua in 3 Dimensions, Ver. 4.0, User's Manual. Minneapolis, Itasca.

Itasca Consulting Group, Inc. (2007) PFC3D – Particle Flow Code in 3 Dimensions, Ver. 4.0. Minneapolis, Itasca.

Itasca.Klohn Krippen Ltd., (2005). Kemess Mine Expansion Environment Impact Assessment, Appendix 5 – Waste Characterisation, Report No. M09120A03 02). Report submitted to Northgate Minerals Corp., September, 2005.

JK Tech Pty Ltd., Kemess Underground, SMC Test Report, March 2011.

JK Tech Pty Ltd., Kemess Underground SMC Test Report, July 2012.

Knight Piesold Ltd., (2004). Feasibility Design of the Kemess North Open Pit, (Report No VA101-0004/7-1), Volume 1 of 2. Report submitted to Northgate Exploration Ltd., March 31, 2004.

Knight Piesold Ltd., (2004). Feasibility Design of the Kemess North Open Pit, (Report No VA101-0004/7-1), Volume 2 of 2. Report submitted to Northgate Exploration Ltd., March 31, 2004.

KWM Consulting Inc., Kemess North Underground Project Metallurgy and Process, Submitted to AuRico Gold, September 2012.

KWM Consulting Inc., Kemess North Underground Project Metallurgy and Process, Submitted to AuRico Gold, May 2011.

KVK Consulting Associates Inc., Summary Report on Process Development – Treatment of Ores from the Kemess North Deposit", 28 May 2003.

Lakefield Research, An Update of Metallurgical Testing of Kemess Ores, June 10, 2003.

Laubscher, D.H., (1990). A geomechanics classification system for rating of rock mass in mine design. J. S. Afr. Inst. Mining Metall. 90(10):257-273

Laubscher, D. and Jakubec, J. (2000). The IRMR/MRMR rock mass classification for jointed rock masses, SME 2000. PP. 475-481.

Lorax, Kemess Underground Groundwater Seepage Predictions Final, Report submitted to AuRico Gold Inc, November 2012.

Lorax, e-mail Revision to Maximum Inflow Rate, submitted to AuRico Gold Inc, June 5, 2012.

Lorax, Kemess Underground Project Water Quality Predictions for Mine Workings, submitted to AuRico Gold Inc., January 2013.

McKinley, B.S.M. (2006). Geological Characteristics and Genesis of the Kemess North Porphyry Au-Cu-Mo Deposit, Toodoggone District, North-Central British Columbia, Canada. MSc Thesis, UBC. Unpublished.

Mine Ventilation Services Inc., Ventilation Design for AuRico Gold's Kemess Underground Project, Report submitted to SRK, September 2012.

Ouchi, A.M., Pakalnis, Dr. R., Brady, T.M. (2004). Update of Span Design Curve for Weak Rock Masses. Presented at 2004 AGM-CIM Edmonton.

SGS Lakefield, The Grindability Characteristics of Samples from the Kemess Mine and Grinding Circuit Evaluation (East Zone), August 2008.

SGS Mineral Services, Kemess East Pit Hypogene Plant Trial – May 2009", (ref 12183 – 001), September 15, 2009.

SGS Mineral Services (presentation), Concentrate Grade and Regrind Size Improvement on the East Pit Hypogene Ore at Kemess Mine", Dec 2010.

SMC, Test Report for the Kemess Underground, March 2011.

SRK Consulting Ltd., (2011). Kemess North Project Review, Geotechnical Review and Site Visit. (Report No. 2CN025.000). Report submitted to Northgate Minerals Corp., June 2011.

SRK, Geotechnical Characterization of the Kemess Underground Project, August 2012.

Tetra Tech, Underground Mine Design and Estimation Assistance for the Kemess Feasibility Study, Report submitted to AuRico Gold Inc., December 20, 2012.

29 Date and Signature Page

This technical report was written by the following qualified persons and contributing authors. The effective date of this technical report is December 31, 2012

Qualified Person	Signature	Date
Andrew Jennings	—original signed”	April 1, 2013
Andrew Witte	—original signed”	April 1, 2013
Chris Bostwick	—original signed”	April 1, 2013
Chris Elliott	—original signed”	April 1, 2013
Gordon Skrecky	—original signed”	April 1, 2013
Harold Bent	—original signed”	April 1, 2013
Jarek Jakubec	—original signed”	April 1, 2013
Jeffrey Volk	—original signed”	April 1, 2013
Ken Major	—original signed”	April 1, 2013
Pacifico (Virgil) Corpuz	—original signed”	April 1, 2013

Reviewed by

Original signed
Gilles Arseneau, Ph.D., PGeo.
Project Reviewer

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

APPENDIX A:
Legal Opinions

AURICO GOLD INC.

CERTIFICATE OF OFFICER

TO: SRK Consulting (Canada) Inc.

I, Trent C. A. Mell, General Counsel of AuRico Gold Inc. (the "**Corporation**"), do hereby certify for and on behalf of the Corporation and without personal liability, that in connection with the preparation of the Kemess Underground feasibility study:

1. AuRico Gold Inc. is a successor company of Northgate Minerals Corporation;

2. The Corporation has taken all such necessary steps since 2010 to maintain our claims in good standing;

3. SRK Consulting can reasonably rely on the 2009 opinion for the feasibility study in the preparation of the Kemess Underground feasibility study; and

4. AuRico has no knowledge of any other facts which would materially impact the legal opinion issued in 2009.

This certificate is being given by the Corporation and without personal liability of the undersigned and any party seeking to make a claim with respect to any matter contained in this certificate shall have recourse solely to the Corporation.

DATED at Toronto this 1st day of March, 2013.

Trent C. A. Mell
General Counsel

25th Floor
700 W Georgia St | Vancouver, BC
Canada V7Y 1B3 | Tel 604 684 9151
Fax 604 661 9349 | www.farris.com

September 30, 2009

CIBC World Markets Inc.
Scotia Capital Inc.
Canaccord Capital Corporation
Credit Suisse Securities (Canada), Inc.
Genuity Capital Markets
Merrill Lynch Canada Inc.
Research Capital Corporation
TD Securities Inc.
UBS Securities Canada Inc.
c/o CIBC World Markets Inc.
161 Bay St, BCE Place
P.O. Box 500
Toronto, Ontario M5J 2S8

Osler, Hoskin & Harcourt LLP
Suite 6100, 100 King Street West
1 First Canadian Place
Toronto, Ontario M5X 1B8

Torys LLP
79 Wellington Street West, Suite 3000
Toronto, Ontario M5K 1N2

Dear Sirs/Mesdames:

Re: Northgate Minerals Corporation (the "Corporation")

We have acted as local counsel for the Corporation in the Province of British Columbia in connection with the issue by the Company of 34,300,000 common shares at a price of $2.92 per Common Share, pursuant to the Canadian prospectus and pursuant to the terms and conditions of an agreement (the "Underwriting Agreement") dated September 24, 2009 among the Corporation and CIBC World Markets Inc., Scotia Capital Inc., Canaccord Capital Corporation, Credit Suisse Securities (Canada), Inc., Genuity Capital Markets, Merrill Lynch Canada Inc., Research Capital Corporation, TD Securities Inc. and UBS Securities Canada Inc. (the "Underwriters").

We have been asked to provide this opinion pursuant to section 11(1)(c) of the Underwriting Agreement with respect to those mineral tenures held by the Corporation in British Columbia which are material to the operation of the Kemess mine. In that capacity, we have been instructed to provide you with the following opinion relating to the title of four mining leases, six legacy mineral claims and five mineral cell claims which are further described in Schedule A hereto (collectively, the "Material Claims"). All of the Material Claims are mineral

claims. In rendering this opinion, we have relied upon, without independent investigation, a certificate of an officer of the Corporation dated the date hereof as to the mineral tenures which are material to the operation of the Kemess mine.

We have not been asked to opine on any mineral tenures held by the Corporation outside of British Columbia or that are within British Columbia but not material to the operation of the Kemess mine and have excluded any such mineral tenures from this opinion.

We are qualified to express opinions only with the respect to the laws of the Province of British Columbia and the federal laws of Canada applicable in British Columbia. We express no opinion on the laws of any jurisdiction other than the Province of British Columbia and the federal laws of Canada applicable in British Columbia.

Based upon, and subject to our searches and examinations described above and the qualifications listed below, we are of the opinion that:

1. as at September 25, 2009, the sole recorded holder of the Material Claims is the Corporation, subject to any intervening rights of unrecorded interests of third parties;

2. the mining leases set out in Part A of Schedule A hereto constitute an interest in lands previously covered by those mineral claims described therein as the "Demised Claims", and are valid under the *Mineral Tenure Act* (British Columbia), for terms expiring as set out in Part A of Schedule A, subject to the obligation to pay annual rent;

3. each of the Material Claims is in good standing under the *Mineral Tenure Act* (British Columbia) until the "Good To Date" stated for each Material Claim in Schedule A, subject to the requirements to do annual assessment work as required or to pay related fees as required; and

4. there is no agreement or document recorded in the computer records of Mineral Titles Online BC as of September 25, 2009 granting to or claiming by any other person, firm or corporation, any right in or right to acquire any of the Material Claims or any interest therein which is still in effect other than as disclosed herein and in Schedule A.

This opinion relating to the Material Claims is subject to the following qualifications and assumptions:

1. no investigation was made of the original application for filing of, the location of, the boundaries of any of the Material Claims;

2. no investigation was made of the existence of any interest in any of the Material Claims other than what is noted in the computer records of Mineral Titles Online BC as of September 25, 2009 and the documentation provided by the Vancouver (Coast Laird Mining Region) and Smithers (Omineca Mining Region) Mineral Titles Branch offices in

response to our request for all documentation in effect relating to encumbrances registered against the Material Claims;

3. no review has been made of original documents which are filed at the applicable Mineral Titles Branch office, and only information available online through Mineral Titles Online BC as well as photocopies of documentation provided by the Vancouver (Coast Laird Mining Region) and Smithers (Omineca Mining Region) Mineral Titles Branch offices have been reviewed;

4. that all facts set forth in official public records and other documents supplied by the Mineral Titles Branch or otherwise conveyed to us are complete and accurate, and where such records or other documents are dated as of a date other than the date hereof, we have assumed the content thereof remains accurate as of today's date;

5. that each of the bills of sales or other assignments transferring the Material Claims to the Corporation from various third parties are valid transfers;

6. that the persons who purported to execute documents examined in the course of title examinations were the persons named therein and the persons who purported to execute such documents on behalf of corporations were duly authorized to do so;

7. that copies of documents examined were, in fact, true copies of documents in existence and that the originals of such documents were properly executed;

8. the identity, capacity and authority of any person acting or purporting to act in a representative capacity or as a public official;

9. no search or other correlation was made with respect to tax assessed by any applicable government authority;

10. no examination was made of the ground to determine if any of the Material Claims had been staked or assessment work carried out in accordance with the provisions of the *Mineral Tenure Act* (British Columbia) and regulations and accordingly we assumed compliance with such Act and its regulations as to the staking and assessment work;

11. no examination was made of any free miner's certificate, grouping notice, assessment report or other record to determine its compliance with the provisions of the *Mineral Tenure Act* (British Columbia) and its regulations, nor has any examination been made of any permits issued under the *Mines Act* (British Columbia) to determine compliance with any permit provisions;

12. no representation is made as to the possible effect on the Material Claims, Kemess mine or surrounding property of aboriginal land claims, trap lines, environmentally sensitive areas, unique animal species, parks proposals, protected areas, species at risk federal

legislation, or Land Resource Management Plans, nor have we conducted any search of surface tenures that might be located on the Material Claims; and

13. no representation is made as to whether any registered title exists in respect of any lands in the general location of the Material Claims other than as disclosed herein.

Yours truly,

FARRIS, VAUGHAN, WILLS & MURPHY LLP

SCHEDULE A

MATERIAL CLAIMS

A. MINING LEASES

1. Mining Lease – Kemess South

Tenure No.	District Lot No.	Map No.	Good To Date	Expiry Date	Area (ha.)	Demised Claims (Tenure Nos.)	Notes
354991	7198 7201 7204 7207 7199 7200	094D097 094E007 094D 094E	15 Sep 2010	15 Sep 2027	862.33	238404 243444 243445 350858 242575 - 242578 242581 - 242584	1 – 8, 10

2. Mining Lease – Kemess North

Tenure No.	District Lot No.	Map No.	Good To Date	Expiry Date	Area (ha.)	Demised Claims (Tenure Nos.)	Notes
410732	7327 7328	094E007	29 Sep 2010	29 Sep 2034	950	237800 237801	10 See demised claims

Claims demised unto mining lease 410732 (Tenure No.)	Notes
237800, 237801	5 – 8, 10

3. Mining Lease – Kemess North

Tenure No.	District Lot No.	Map No.	Good To Date	Expiry Date	Area (ha.)	Demised Claims (Tenure Nos.)	Notes
410741	7329	094E007	29 Sep 2010	29 Sep 2034	106	304015 - 304019 405480 405481	10 See demised claims

Claims demised unto mining lease 410741 (Tenure No.)	Notes
304015 - 304019	5 – 8, 10

4. Mining Lease - Midway

Tenure No.	District Lot No.	Map No.	Good To Date	Expiry Date	Area (ha.)	Demised Claims (Tenure Nos.)	Notes
524240	7342	094E	22 Dec 2009	22 Dec 2035	1565	238819 243354 243355 414225 - 414228	10 See demised claims

Claims demised unto mining lease 524240 (Tenure No.)	Notes
238819	1 – 8, 10
243354, 243355	5 – 10

B. LEGACY CLAIMS

Tenure No.	Claim Name	Map No.	Good To Date	Area (ha.)	Notes
241014	SEM #1	094D097 094E007 094D 094E	14 Dec 2019	400	5 – 8, 10
241959	NEK 3	094E007 094E	14 Dec 2019	500	5 – 8, 10
241960	NEW KEMESS 3	094E007 094E	14 Dec 2019	375	5 – 8, 10
242573	DU 2	094E007 094E	14 Dec 2019	500	1 – 8, 10
242574	NEK 4	094E007 094E	14 Dec 2019	350	5 – 8, 10
243440	ALISON 1	094E007 094E	14 Dec 2019	500	5 – 8, 10

C. MINERAL CELL TITLE CLAIMS

1. Nor 2 & 7 cell claim

Tenure No.	Claim Name	Map No.	Good To Date	Area (ha.)	Converted Tenures (Tenure Nos.)	Notes
515686		094D 094E	14 Dec 2019	1427.86	239096 – 239098 301219 303616 350859	10 See converted tenures

Claims converted into mineral cell title 515686 (Tenure No.)	Notes
239096 – 239098, 30219, 303616	1 – 8, 10
350859	1, 3 – 8, 10

2. Nor 5 & 6 cell claim

Tenure No.	Claim Name	Map No.	Good To Date	Area (ha.)	Converted Tenures (Tenure Nos.)	Notes
515693		094D	14 Dec 2019	1534.10	242991 242992 303614 303615 305630 309046 – 309057 310032 – 310037 413267	10 See converted tenures

Claims converted into mineral cell title 515693 (Tenure No.)	Notes
242991, 242992, 303614, 303615	1 – 8, 10
305630	3 – 8, 10
309046 – 309057, 310032 - 310037	5 – 8, 10

3. Tailings cell claim

Tenure No.	Claim Name	Map No.	Good To Date	Area (ha.)	Converted Tenures (Tenure Nos.)	Notes
516786		094E	14 Dec 2019	1391.64	243069 243070 243076 243077 243079 304008 – 304014 305554 305555 311261 – 311268 355405 – 355407 355409 355411 355412 355416 413834	10 See converted tenures

:

Claims converted into mineral cell title 516786 (Tenure No.)	Notes
243069, 243070, 243076, 243077, 243079, 304008 – 304014, 305554, 305555, 311261 – 311268	5 – 8, 10
355405 – 355407, 355409, 355411, 355412, 355416	3 – 10

4. Tailings NW cell claim

Tenure No.	Claim Name	Map No.	Good To Date	Area (ha.)	Converted Tenures (Tenure Nos.)	Notes
516848		094E	14 Dec 2019	105.66	304020 310075	10 See converted tenures

:

Claims converted into mineral cell title 516848 (Tenure No.)	Notes
304020, 310075	5 – 8, 10

5. Tailings N cell claim

Tenure No.	Claim Name	Map No.	Good To Date	Area (ha.)	Converted Tenures (Tenure Nos.)	Notes
516854		094E	14 Dec 2019	1197.16	243068 243441 304021 – 304023 305548 – 305553 310054 – 310056 311291 - 311294	10 See converted tenures

Claims converted into mineral cell title 516854 (Tenure No.)	Notes
243068, 243441, 304021 – 304023, 305548 – 305553, 310054 – 310056	5 – 8, 10
311291 - 311294	5 – 10

NOTES:

1. Claim of Builder's Lien filed by BXL Bulk Explosives Limited, recorded January 29, 1998 (Event **3115575**), as amended by Claim Agreement recorded June 23, 1998 (Event **3120579**)

2. Claim of Builder's Lien filed by Northern Thunderbird Air Ltd., recorded May 15, 1998 (Event **3118760**), as amended by Claim Agreement recorded June 23, 1998 (Event **3120613**)

3. Kemess South Mine Royalty Agreement dated June 22, 1998 between Royal Oak Mines Inc. and Trilon Financial Corporation, recorded June 23, 1998 (Event **3120566**).

4. Royalty Security Debenture dated June 22, 1998 between Royal Oak Mines Inc. and Trilon Financial Corporation, recorded June 23, 1998 (Event **3120567**).

5. Debenture dated June 22, 1998 between Royal Oak Mines Inc. and Trilon Financial Corporation, recorded June 23, 1998 (Event **3120568**).

6. Assignment of Cash Flow Royalty dated February 11, 2000 made by Northgate Resources Limited in favour of Trilon Financial Corporation, recorded February 11, 2000 (Events **3144514** and **3144519**), as reassigned to Northgate Resources Limited, recorded September 11, 2001 (Events **3171013** and **3202537**).

7. Assumption and Novation Agreement dated April 24, 2000 among Kemess Mines Ltd., Royal Oak Ventures Inc. and Trilon Financial Corporation, recorded May 2000 (Event **3147473**).

8. Royalty Security Debenture dated April 24, 2004 between Kemess Mines Inc. and Trilon Financial Corporation, recorded May 2, 2000 (Event **3147474**).

9. Assessment complaint recorded February 23, 1998 (Event **3116124**).

10. Certain of the Material Claims are subject to easements, rights of way, licenses of occupation or similar non-financial encumbrances which we have not specifically enumerated.