Independent Technical Report Updated Feasibility Study Aurora Gold Mine Project, Republic of Guyana

Prepared for

Guyana Goldfields Inc.

Prepared by

srk consulting

SRK Consulting (Canada) Inc.
2CG028-009
February 2017

Effective Date: 31 Dec, 2016
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Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for Guyana Goldfields Inc. ("Guyana Goldfields or the Company") by SRK Consulting (Canada) Inc.. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Guyana Goldfields subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits Guyana Goldfields to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to National Instrument 43-101, Standards of Disclosure for Mineral Projects. The responsibility for this disclosure remains with Guyana Goldfields. 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.

This technical report contains certain non-International Financial Reporting Standards measures. Such measures have non-standardized meaning under International Financial Reporting Standards and may not be comparable to similar measures used by other issuers.

All amounts are expressed in U.S. dollars ($), unless otherwise indicated.

Executive Summary

Introduction

Guyana Goldfields Inc. (Guyana Goldfields, GGI or the Company) owns a 100% interest in the Aurora Gold Mine (Aurora or the Mine), an open pit mine utilizing a standard 5,600 tonnes per day (tpd) CIL processing facility located in Guyana, South America.

The Aurora mine poured its first gold in August 2015 and after a successful ramp-up declared commercial production on January 1, 2016. For the calendar year 2016, the mine produced 151,600 ounces with a total of 1.9 Mt processed at an average head grade of 2.74g/t Au.

This feasibility study assumes a gold price of $1,200 per ounce and reflects an expansion of the current processing facility from 5,600 tonnes per day (tpd) to 8,000 tpd. The proposed expansion of the mill is planned to be implemented in two phases and is expected to increase annual production to above 200,000 ounces beginning in 2018 for the life of the mine with gold production peaking at 302,704 ounces in 2021. Highlights of the study include:

- 16 years mine life and Post-Tax NPV5% of $850 M;
- Average annual production over the life of the mine is 220,000 ounces at an average cash cost of $612 per ounce (including royalties);
- Proven and Probable Mineral Reserves of 36.7 Mt at an average grade of 2.99 g/t Au;
- Measured and Indicated Mineral Resource of 6.3 million ounces of gold at an average grade of 3.25 g/t Au;
- Open pit mining will be ongoing through to 2024. Development of the underground operation will commence in 2022 with first production expected in 2024.
- Phase 1 of the mill expansion will allow processing of a saprolite/rock blend at a rate of 8,000 tpd and is expected to be completed by the end of the first quarter of 2018.
- Phase 2 of the mill expansion will include the addition of a ball mill and will allow the processing of 8,000 tpd hard rock. Phase 2 is expected to be completed by the middle of 2019.

In September 2016 SRK Consulting (Canada) Inc. was retained by the Company to prepare this updated Feasibility Study. The Recovery Methods section of the report is authored by JDS Energy & Mining Inc., Tailings Disposal and Water Management sections are authored by Tetra Tech, Inc.

Property Description and Ownership

The Aurora Gold Mine is located in Guyana, South America, approximately 170 km west of the capital Georgetown and 130 km west north-west of Bartica, a settlement at the junction of the Essequibo and Cuyuni Rivers. Bartica is a regional hub for accessing the interior of north-western

Guyana. The center of the property is located at latitude 6°45′N, longitude 59°45′W. The Project includes the Buckhall Port on the Essequibo River. There is a 150 km road from Buckhall Port to the Aurora Project site, and a ferry crossing of the Cuyuni River at Tapir.

Guyana Goldfields has constructed a runway on the southern bank of the Cuyuni River adjacent to the mining operation, which is suitable for helicopters and short-takeoff-and-landing aircraft. The runway has an approximate length of 1200 m. Guyana Goldfields operates several charter flights per day from Ogle airport to the Project site.

Guyana Goldfields, through its wholly owned subsidiary in Guyana, AGM Inc. (AGM), owns a 100% interest in Aurora. The Company was issued a Mining Licence for Aurora in November 2011. The licence gives Guyana Goldfields the right to build and operate the mine. When the licence was issued, the Company also signed a Mineral Agreement with the Government of Guyana and the Guyana Geology and Mines Commission which sets the fiscal regime, taxation and royalties as they affect the operation of the mine. The licence and mineral agreement were signed by Guyana Goldfields and AGM Inc., and are valid for 20 years and renewable on application for an additional seven-year period. The boundaries of the license form an oblong shape trending approximately southeast from the south bank of the Cuyuni River.

Geology and Mineralization

Mineral Resources at Aurora are confined within an approximately 2 km long corridor, known as the “Golden Square Mile” within the Company’s Mining Licence. The Golden Square Mile area of the Aurora Gold Mine comprises folded metasedimentary and metavolcanic rock of the lower Cuyuni Formation that has been metamorphosed to greenschist assemblages. The Golden Square Mile is located within a broad regional, northwest trending, high strain zone characterized by strong northwest trending and sub-vertical foliation and dip slip shearing (southwest over northeast) and strain partitioning into interconnected network of discrete shear zones.

Gold mineralization at Aurora exhibits features analogous to mesothermal or “orogenic” gold deposits in the West-African Palaeoproterozoic Birimian Supergroup, with all gold mineralization controlled by a series of northwest trending shear zones.

The Aurora Gold Mine area is divided into four major areas of gold mineralization; Rory’s Knoll (includes Walcott Hill East), Aleck Hill (includes Aleck Hill North), Walcott Hill, and Mad Kiss (includes Mad Kiss West).

Exploration Status

After a lengthy hiatus during the financing and development period of the Aurora Gold Mine exploration activity is expected to ramp up significantly in 2017. Exploration efforts will be focused on both near mine targets as well as more greenfield type targets to the north and east of the Mining Licence.

Operations

Open Pit Operation

The open pit has a mine life of eight years based on a total of 19.6 Mt of ore mined at an average grade of 2.87 g/t Au. Average annual gold production over this period is approximately 1.8 M Oz. Saprolite ore makes up 2.9 Mt or ~15% of total open pit ore tonnes and is expected to be largely mined out by the middle of 2019. Approximately 60% of the ore tonnes are sourced from Rory's Knoll, 30% from Aleck Hill, with the remainder coming from the Walcott Hill, Mad Kiss and North Aleck satellite deposits. Mining activity over the near term will be focused on rock ore at Rory's Knoll and saprolitic ore at Aleck Hill..

The average strip ratio over the life of mine is 8.4 to 1. The strip ratio peaks in 2019 and 2020, averaging 14.7 over these two years. As a result of the increasing strip ratio during these years the mining rate is expected to increase from ~35,000 tpd in 2017 to ~95,000 tpd in 2019 and 2020. To accommodate this increase and reduce overall mining costs the Company plans to transition to a larger mining fleet in 2018 consisting of 90-tonne haul trucks. The larger equipment will primarily be used for waste mining activities in the upper benches of the larger Rory's Knoll and Aleck Hill pits. 41-tonne haul trucks will continue to be used to haul waste deeper in the pits and to haul all ore.

Underground Operation

Following a two-year pre-production period, underground mining at Rory's Knoll commences in year 2024 as open pit mining operations wind down. The mining rate at Rory's Knoll averages 5,500 tpd at an average grade of 3.02 g/t Au over an 8-year mine life for a total of 1.4 Moz. Vertical development is assumed to be completed by an underground contractor while all lateral development will be completed by the Company. Rory's Knoll underground will be mined utilizing the open benching and sublevel retreat mining methods via a decline access with truck haulage from a depth of -330 mRL down to -770 mRL. Below the open pit, the Rory's Knoll ore body approximates a sub-vertical pipe with >100 m diameter which, along with results from a detailed hydrogeological and geotechnical model, support the open benching and sublevel retreat mining method approach. The study results show underground mining creates minimal surface subsidence and indicate water inflows are manageable, as has been proven by existing open pit mining activities.

The Aleck Hill underground is expected to contribute 153,000 ounces (1.1 Mt at an average grade of 4.28 g/t Au) to the overall mine plan beginning in 2025 and continuing through to 2030. Aleck Hill will be mined through a combination of transverse and longitudinal LHOS from a depth of about -160 mRL to about -500 mRL.

The Mad Kiss underground contributes 104,000 ounces (0.6 Mt at an average grade of 5.45 g/t Au) to the overall mine plan. Mad Kiss will be developed concurrently with the Rory's Knoll underground with initial production expected in 2024 and continuing through to 2027. Mad Kiss will be mined via longitudinal LHOS from a depth of about -40 mRL to about -380 mRL.

Processing

The existing process circuit has capacity for 5,600 tpd and includes cyanide leach and carbon adsorption process comprising crushing, single-stage grinding, gravity, cyanide leaching, carbon adsorption, carbon elution and regeneration, gold refining, cyanide destruction and tailings disposal.

The proposed plant expansion will be completed in two phases. The first phase will increase the throughput rate from 5,600 tpd to 8,000 tpd assuming that the saprolite portion of the mill feed is between 25% and 50%. The ordering of long lead-time items for the Phase 1 expansion has already commenced and detailed engineering is expected to commence in February 2017. The first phase of the expansion is expected to be completed by the end of the first quarter of 2018. The second phase of the expansion will allow the processing of 8,000 tpd hard rock and is expected to commence in mid-2018 and be completed by mid-2019 when the majority of saprolitic ore has been exhausted.

Gold doré is produced in the on-site refinery and stored in a secure vault prior to transportation off-site.

Mineral Resource Estimates

The mineral resources for the Aurora Gold Mine are reported at a cut-off grade of 0.30, 0.40 and 1.80 g/t Au based on open pit (saprolite and rock) and underground mining scenarios, respectively. The open pit cut-off grades are based on assumptions summarized in ES Table 1, while the underground reporting cut-off grades was determined considering the same price and recovery assumptions in consultation with SRK mine engineers involved in the design of an underground mine for the Aurora Gold Mine.

Mineral resources were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) by Timothy Carew, P.Geo (APEGBC #19706), appropriate independent qualified person for the purpose of National Instrument 43-101.

ES Table 1: Mineral Resource Statement, Aurora Gold Project, Guyana, SRK Consulting, effective December 31, 2016.

Category	Quantity (kt)	Grade (g/t Au)	Contained Metal (k Oz)
Open Pit			
Measured	5,230	3.17	500
Indicated	24,440	2.51	1,970
Measured and Indicated	29,670	2.62	2,440
Inferred	4,770	1.57	230
Underground			
Indicated	30,060	3.91	3,780
Measured and Indicated	30,060	3.91	3,780
Inferred	11,810	4.12	1,570
Combined Mining			
Measured	5,230	2.97	500
Indicated	54,500	3.28	5,750
Measured and Indicated	59,730	3.25	6,250
Inferred	16,580	3.79	1,790

Notes:

1. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates.
2. The cut-off grades are based on a gold price of US$1,300 per ounce and metallurgical recoveries of 97 percent and 94.5 percent for saprolite and rock material respectively.
3. Open pit resources are reported within conceptual optimized open pit shells, whereas underground resources are external to these. Open pit resources are reported at a cut-off grade of 0.30 g/t Au and 0.40 g/t Au for saprolite ore and rock ore respectively, whereas underground resources are reported at a cut-off of 1.8 g/t Au.
4. Open pit mineral resources exclude the mined out areas.
5. Stockpile data based on inventory in RoM, external RoM and Refeed stockpile as of EOY 2016.

Mineral Reserve Estimates

The mineral reserve estimate has been completed to a level appropriate for feasibility studies. The mineral reserve estimate stated herein is consistent with the CIM Definition Standards (2014) for reporting Mineral Resources and Mineral Reserves and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

The Mineral Reserve statement for the Aurora Gold project is presented in ES Table 2.

ES Table 2: Mineral Reserve Statement, Aurora Gold Project, Guyana, SRK Consulting, effective December 31, 2016.

Category	Quantity (kt)	Grade (g/t Au)	Contained Metal (k Oz)
Proven			
OP Saprolite	336	1.60	17
OP Rock	4,864	2.99	468
Total Proven	**5,200**	**2.90**	**485**
Probable			
OP Saprolite	2,934	1.91	180
OP Rock	12,128	3.02	1,178
UG Rock	16,519	3.19	1,694
Total Probable	**31,580**	**3.01**	**3,052**
Total P&P	**36,781**	**2.99**	**3,537**

Notes:

1 Mineral Reserves are based on a gold price of US$1,200 per ounce, 8% royalty and an average metallurgical recovery of 96.0% for saprolite and 94.0% for fresh rock material.
2 Open pit saprolite and rock reserves are reported at a cut-off grade of 0.44 g/t Au and 0.42 g/t Au for vein and upper saprolite material respectively. Open pit rock reserves are reported at a cut-off grade of 0.76 g/t Au and 0.64 g/t Au for vein and Rory’s Knoll rock material respectively.
3 Underground fresh rock reserves are reported at a cut-off grade of 1.5 g/t Au for Rory’s Knoll and 2.41 g/t Au for the satellite deposits.
4 Mineral Reserves are contained within Mineral Resources.
5 Independent qualified persons as defined by National Instrument 43-101 are Robert J. McCarthy P Eng.(# 27309), for open pit reserves and Christopher A. Elliott, FAusIMM (#103277), for underground reserves. SRK is not aware of mining, metallurgical, infrastructure, permitting, or other factors that could materially affect the mineral reserve estimates.

Project Infrastructure and Access

During the mine life a combination of transportation methods, including aircraft, river navigation, and road access will be used to supply the Mine. Overall, the offsite infrastructure includes docking facilities for cargo ships at the Buckhall Port facility on the west side of the Essequibo River. The mine access road is 150 km in length from Buckhall Port to the mine site.

During production, the access road will be mainly used for the supply of food, reagents, spare parts, mining supplies, and diesel fuel. The site airstrip will be used mainly for personnel transportation and emergency situations.

The site entails a series of open pits, waste rock stockpiles, a process facility with associated laboratory and maintenance facilities; maintenance buildings for underground and open pit equipment. Facilities and structures include a warehouse, office, change house facilities, ventilation shaft, mine air cooling process facility, explosives storage area, power generating station, fuel storage tanks, a warehouse and laydown area, a 1,200 m airstrip, and a permanent accommodation complex. The open pit area is protected from potential flooding of the Cuyuni River by a river dike.

Areas for the tailings management, fresh water, and mine water management ponds are provided by several dams at low topographic points. These ponds, and all except one of the waste rock stockpiles, are on the west side of the process facility, away from the accommodation complex. Additional waste is located to the north of the process facility, also away from the accommodation complex.

Conclusions

The results of sensitivity analyses show that the project can withstand a substantial increase in capital and operating costs. A 30% decrease in gold price results in a positive NPV5% of $312 M; at a gold price of $1,500 per ounce the NPV5% results in $1,325 M.

Open Pit Mining

The Aurora Mine has successfully completed its commissioning and as of this writing is in a steady state of mining production. The operations are sufficiently established so as to provide the basis of much of the technical and economic inputs required for this feasibility study update.

This study has demonstrated that AGM would be able to increase the mine production beginning in 2018 to meet a mill throughput rate of 8,000 tonnes per day. Both saprolite and rock ores are mined through 2019 with only rock ore available from 2020 onward. In order to elevate the mill feed grade in the early years, SRK segregated low grade saprolite and rock ores for later feeding to the mill at the end of the mine life.

The increase in mine production rates is enabled by the acquisition of larger loading and hauling equipment starting in 2018. This enables AGM to realize the benefits of lower unit costs as well as limit the size of equipment fleets needed in the mine plan.

Underground Mining

The study has demonstrated that the underground operations Rory’s Knoll and the satellite deposits (Aleck Hill and Mad Kiss) are economically viable in their current configurations.

Processing

The current processing facility operating at AGM is meeting the predicted throughput rates and grind sizes. The proposed expansion of the processing facility in two phases will ultimately result in an expanded throughput rate of 8,000 tpd hard rock.

Environmental

The project has been in operation for one year. There is an ESHS MS System in place and external audits for IFC standards compliance review occur regularly.

There are no non-compliance issues based on verbal information provided by the VP Sustainability and Health & Safety, and confirmed by a general review of public information.

Risks and Opportunities

During the course of the study the following potential risks were identified.

Geotechnical

Open Pit Getechnical

- **Brittle Fault Location and Characterization** – Currently, a model of the brittle fault features and large persistent joints expected behind the pit face in the region of the final pit does not exist. Local structure mapping has been conducted on site, however these observations have not been interpreted to extend behind the pit face to the planned interim or final wall locations. The structural model of the rock at the final pit wall location is an integral part of the overall slope design which must be reviewed against the likelihood of faults to avoid large scale slope failures ranging from multi-bench to inter-ramp scale. An improved understanding of the nature and location of any potential brittle fault features at the final pit wall locations should be undertaken as soon as possible. This is commonly achieved through the development of a 3D structural model. All slope designs and angles are contingent to their review against a fault model and should not be considered final until this reconciliation is complete. Pit slope designs may be governed by major structures.

- **Shear Zone Characterization (Aleck Hill)** – The location and geotechnical nature (thickness, strength, etc) of the shear zones at the Aleck Hill pit are based on limited data. Additional boreholes should be drilled to allow for a more complete characterization of these zones. If the shears are found to be extensive, inter-ramp wall angles may be required to be flattened to accommodate for the foliation and weakened rock mass strength.

- **Wall Control Blasting** – The angles provided within this document are based up best industry blasting practices in which wall control is of primary importance. Excessive blast damage will not allow for the achievement of the designed IRA’s. Bench heights may be reduced to 10m if necessary, however, bench widths should also be evaluated to keep the IRA’s in the design sectors similar.

Underground Geotechnical

The geotechnical design parameters presented within this document are based upon the review of historical data and observations on site. The following risks have been identified to the underground design:

- **Slope Stability** – The current mine design includes open benching to the surface. Instability of the slopes above may lead to excess dilution in the underground mine. Due to their impact on the underground mine, the slopes should be designed to a FOS of 1.3 and above.

- **Brittle Fault Location and Characterization** – Currently, a model of the brittle fault features and large persistent joints expected underground does not exist. Local structure mapping has been conducted on site, however these observations have not been interpreted to extend to the underground locations. An improved understanding of the nature and location of any potential brittle fault features at the underground mine locations should be undertaken as soon as possible. This is commonly achieved through the development of a 3D structural model.

- **Shear Zone Characterization (Aleck Hill)** – The location and geotechnical nature (thickness, strength, etc) of the shear zones at the Aleck Hill pit are based on limited data. Additional boreholes should be drilled to allow for a more complete characterization of these zones. If the shears are found to be extensive, underground development and stope locations may be required to move or be reduced in size to accommodate for the foliation and weakened rock mass strength or an increase in ground support (and associated costs) will be required.

- **Water Inflow** – Any changes in the location and geotechnical nature (thickness, strength, etc) of the shear zones at the Aleck Hill area may affect the local water inflow into the underground excavation. Additional costs may be associated with managing the water inflow.

- **Ore Geometry** – ore geometry which is more complex than expected, may increase damage and dilution during production, and reduce ore recovery.

The following opportunities have been identified to the open pit design:

- **Saprolite Design Angle** – Based on the current behaviour of the saprolite slopes, site specific laboratory testing and specific trial inter-ramp slopes can used to determine in saprolite slopes can be locally steepened. Pore water pressures should be monitored in these areas.

- **Data Reconciliation** – As mining progresses, additional data on both the saprolite and the rockmass (such as joint persistence, joint set orientation, etc) can be collected and used to update the geotechnical domain models. The inter-ramp angles may be able to be increased with increased confidence in the geotechnical data and continued site observations.

The following opportunities have been identified to the underground design:

- **Geological Reconciliation** – As additional geotechnical information is collected on the areas near the underground design, ground conditions found to be more competent than the current model will allow for a reduction in support cost.

Mining

Open Pit Mining

There is risk that the production increase in 2018 may pose logistical and management challenges. A key area to consider is training of new operators and staff to ensure efficiencies envisioned in this study are realized. AGM would be advised to develop a comprehensive execution plan to ensure that milestones are met as new equipment and personnel are brought on line.

Additional resources in the mine proximity would enable the open pit mine life to extend. While such resources have not yet been identified, SRK is aware that brownfield exploration is getting underway to see if such opportunities may exist for the near term.

As previously suggested, the Company may be able to source more used equipment than considered in this study and possibly at better pricing than assumed.

Underground Mining

- **Mudrush** – External mudrush risk exists for the underground mine due to the heavy rainfall and the potential for generating fines and clays from the overlaying saprolite material. This risk will be mitigated by partial pre-stripping of saprolite ore as part of the open pit mining and by implementation of an effective dewatering and water diversion programs, such as perimeter drainage, collection sumps, etc.
- **Labour productivity** – The productivity of the Guyanese workforce in an underground mining environment is not demonstrated. Timely supply of expatriate and skilled local personnel has the potential to be a very significant risk to the success of the project. The ability to adequately train local unskilled labour to the required level is also a key factor for the underground mine. Sufficient time for planning and execution of the contracting strategy must be allowed.
- **Execution risk** – Any mining system is dependent on professional execution and there exists a risk that the planning and procedures are either optimistic or ultimately not well executed. AGM should ensure that both planning and execution capability are well-considered, are not overly optimistic, and that downside risk does not place the underground mining values proposition at undue risk.

The opportunities available for the underground mines are extensive, given that the commitment to commence underground mining is still some years away. Therefore, with the advantage of time and additional data (as it becomes available from operations and deeper exploration drilling) there exists the opportunity to re-evaluate almost every aspect of the underground mining strategy.

As a minimum, opportunities exist in the following aspects of the underground mines:

- **Mining Method**: re-evaluate the range of applicable mining methods in the context of updated resource model and geotechnical data.
- **Ore Handling**: given that Rory's Knoll deposit is open at depth, there is an opportunity to evaluate alternative ore handling strategies prior to committing to the truck haulage method.

Processing

The ability to blend ores using the soft Saprolitic materials may enhance throughputs beyond those estimated here. If so, an opportunity exists to run low grade Saprolite longer into the mine life with the rock ore from underground, thus extending the mine life and improving overall project economics.

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Acronyms and Abbreviations

Distance	
µm	micron (micrometre)
mm	millimetre
cm	centimetre
m	metre
km	km
”	inch
in	inch
’	foot
ft	foot
Area	
m^2	square metre
km^2	square km
ac	acre
Ha	hectare
Volume	
L	litre
m^3	cubic metre
ft^3	cubic foot
usg	US gallon
lcm	loose cubic metre
bcm	bank cubic metre
Mbcm	million bcm
Mass	
kg	kilogram
g	gram
t	metric tonne
kt	kilotonne
lb	pound
Mt	megatonne
oz	troy ounce
wmt	wet metric tonne
dmt	dry metric tonne
Pressure	
psi	pounds per square inch
Pa	pascal
kPa	kilopascal
MPa	megapascal
Elements and Compounds	
Au	gold
Ag	silver
Cu	copper
Fe	iron
S	sulphur
CN	cyanide
NaCN	sodium cyanide

Other	
°C	degree Celsius
°F	degree Fahrenheit
Btu	British Thermal Unit
cfm	cubic feet per minute
elev	elevation
m AMSL	metres elev. above mean sea level
hp	horsepower
hr	hour
kW	kilowatt
kWh	kilowatt hour
M	Million or mega
mph	miles per hour
ppb	parts per billion
ppm	parts per million
s	second
s.g. or SG	specific gravity
usgpm	US gallon per minute
V	volt
W	watt
Ω	ohm
A	ampere
$k	Thousand dollars
$M	Million dollars
tph	tonnes per hour
tpd	tonnes per day
mtpa	million tonnes per annum
Ø	diameter
Acronyms	
SRK	SRK Consulting (Canada) Inc.
CIM	Canadian Institute of Mining
NI 43-101	National Instrument 43-101
ABA	Acid- base accounting
AP	Acid potential
NP	Neutralization potential
NPTIC	Carbonate neutralization potential
ML/ARD	Metal leaching/ acid rock drainage
PAG	Potentially acid generating
non-PAG	Non-potentially acid generating
RC	reverse circulation
IP	induced polarization
COG	cut-off grade
NSR	net smelter return
NPV	net present value
LOM	life of mine
Conversion Factors	
1 tonne	2,204.62 lb
1 oz	31.10348 g

1 Introduction and Terms of Reference

This technical report summarizes the technical information available on the Aurora Gold Mine. This Report is being filed by the Company on a voluntary basis as contemplated under section 4.2(12) of the Companion Policy to National Instrument 43-101 Standards of Disclosure for Mineral Projects. The Report is being filed by Company to provide updated scientific and technical information in respect of the Aurora Gold Mine, and not as a result of a requirement under NI 43-101.

1.1 Introduction

This technical report was prepared for Guyana Goldfields Inc. (GGI) to summarize the results of a feasibility study update (FSU) on the Aurora gold mine project (Aurora or the Mine), located in Guyana, South America.

The contract with SRK Consulting permits GGI 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 GGI. The user of this document should ensure that this is the most recent technical report for the property, as it is not valid if a new technical report has been issued.

Table 1.1: Areas of responsibilities

Company	Area of Responsibility
SRK	Mineral Resource Estimation, Quality Assurance/Quality Control (QA/QC), Geology, Geotechnical Engineering, Mineral Reserve Estimate, Open Pit and Underground Mine Designs, Mine Ventilation and Cooling, Production Schedule, Open Pit Infrastructure, Mining and Closure Costs, Marketing Studies, Economics, Environmental and Permitting
Tetra Tech	Overburden Stockpile and Tailings Disposal, Site Monitoring and Water Management
JDS	Recovery Methods, Processing Capital and Operating Costs

1.2 Responsibility

This report has been prepared by SRK with contributions from Tetra Tech and JDS. SRK does not accept liability for the statements, findings and opinions expressed in the portions of the reports authored by other contributors. This technical report was written by the authors shown in Table 1.2.

Table 1.2: List of authors and responsibilities

Author	Company	Area of Responsibility
Timothy Carew, P. Geo	SRK Consulting	Mineral Resource Estimate, Quality Assurance/Quality Control (QA/QC), Geology
Christopher Elliott, FAusIMM	SRK Consulting	Executive Summary, Introduction, Property Description, Underground Mine Design, Underground Schedule and Costs, Underground Infrastructure, Underground Mineral Reserve Estimate, Conclusions and Recommendations
Eric Olin, RM-SME	SRK Consulting	Mineral Processing and Metallurgical Testing
Robert McCarthy, P. Eng	SRK Consulting	Open Pit Mine Design, Schedule, Reserve Estimate, Open Pit Mining Costs
Jaroslav Jakubec, C.Eng, MIMMM	SRK Consulting	Geotechnical Engineering
Kelly McLeod, P. Eng	JDS Energy & Mining	Recovery Methods, Processing Capital and Operating Costs
Neil Winkelmann, FAusIMM	SRK Consulting	Marketing Studies and Project Economic Analysis
Rich Greenwood, P. Eng	Tetra Tech	Tailings Storage Facility, Waste Management, River Dyke
Cameron Scott, P. Eng	SRK Consulting	Water Management

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

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

This report uses abbreviations and acronyms common to the mineral industry. Definitions have been provided earlier in the report.

1.3 Basis of Technical Report

This report is an update of the feasibility study issued by Metal Mining Consultants Inc. (MMC) on January 18, 2016, an information collected by SRK during a site visit performed between October 24–27, 2016, and on additional information provided by GGI throughout the course of SRK's investigations. Other information was obtained from the public domain. SRK has no reason to doubt the reliability of the information provided by GGI.

This technical report is based on the following sources of information:

- Discussions with GGI personnel;
- Inspection of the Aurora Gold Mine area, including outcrop and drill core;
- Review of exploration data collected by GGI; and
- Additional information from public domain sources.

1.4 Qualifications of SRK and SRK Team

The SRK Group comprises over 1,400 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.

The compilation of this technical report was completed by Mr. Timothy Carew, P.Geo – Qualified Person for mineral resource evaluation, Mr. Robert McCarthy, P.Eng - Qualified Person for open pit mineral reserves estimation, and Mr. Christopher Elliott, FAusIMM - Qualified Person for underground mineral reserves. By virtue of their education, membership to a recognized professional association and relevant work experience, Mr. Carew, Mr. McCarthy and Mr. Elliott are independent Qualified Persons as this term is defined by National Instrument 43-101. Additional contributions were provided by Mr. Grant Carlson, P.Eng, Mr. Eric Olin, RM-SME, Mr. Cameron Scott, P.Eng, Mr. Neil Winkelmann, FAusIMM, and Mrs. Kelly McLeod.

Mr. Elliott, FAusIMM, a Principal Mining Engineer with SRK, reviewed drafts of this technical report prior to their delivery to Guyana Goldfields as per SRK internal quality management procedures. Mr. Elliott did not visit the project.

1.5 Site Visit

In accordance with NI 43-101 guidelines, SRK has visited the Aurora gold project on four occasions to review geology and exploration protocols. The most recent site visit was conducted by Mr. Robert McCarthy, P.Eng, Mr. Timothy Carew, P.Geo, and Mr. Eric Olin, RM-SME, from October 24 to 27, 2016. The purpose of that visit was to review the geology, resource reconciliation, open pit operation, interview field staff, and to gather information required to produce a technical report.

1.6 Acknowledgement

SRK would like to acknowledge the support and collaboration provided by Guyana Goldfields personnel for this assignment. Their collaboration was greatly appreciated and instrumental to the success of this project.

1.7 Declaration

SRK’s opinion contained herein and effective December 31, 2016, is based on information collected by SRK throughout the course of SRK’s investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

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

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

2 Reliance on Other Experts

SRK has not performed an independent verification of land title and tenure information as summarized in Section 3 of this report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties, but have relied on Jonas M. F. Coddett & Associates as expressed in a legal opinion provided to the Company on February 23, 2010 The reliance applies solely to the legal status of the rights disclosed in Sections 3.2 and 3.3 below.

SRK was informed by GGI that there are no known litigations potentially affecting the Aurora Gold Mine Project.

3 Property Description and Location

3.1 Summary

The Aurora Gold Mine is located in Guyana, South America, approximately 170 km west of the capital Georgetown and 130 km west north-west of Bartica, a settlement at the junction of the Essequibo and Cuyuni Rivers. Bartica is a regional hub for accessing the interior of north-western Guyana. The center of the property is located at latitude 6°45′N, longitude 59°45′W (Figure 3.1). The project includes the Buckhall Port on the Essequibo River. There is a 150 km road from Buckhall Port to the Aurora site, and a ferry crossing of the Cuyuni River at Tapir.

The general area of the Aurora Gold Mine has been subject to mineral exploration since the 1940s. This part of Guyana is largely uninhabited with the nearest settlement approximately 50 km away.

Source: SRK, 2017 (Figure provided by Guyana Goldfields)

Figure 3.1: Property location map.

3.2 Mineral Tenure

Guyana Goldfields owns 100% of the Aurora Gold Mine Project in Guyana covering a 5,802 ha area. The former prospecting permit which was granted in 2004, and which was known as the A1 Licence, was replaced by a Mining Licence in November 2011, which gives the company the right to build and operate the mine.

The boundaries of the Mining License prospecting license form trends approximately southeast-northwest, south of the Cuyuni River (Figure 3.2). The northern edge of the shape follows the south bank of the Cuyuni River; all other edges are straight and are defined by six corner points, which are listed in Table 3.1.

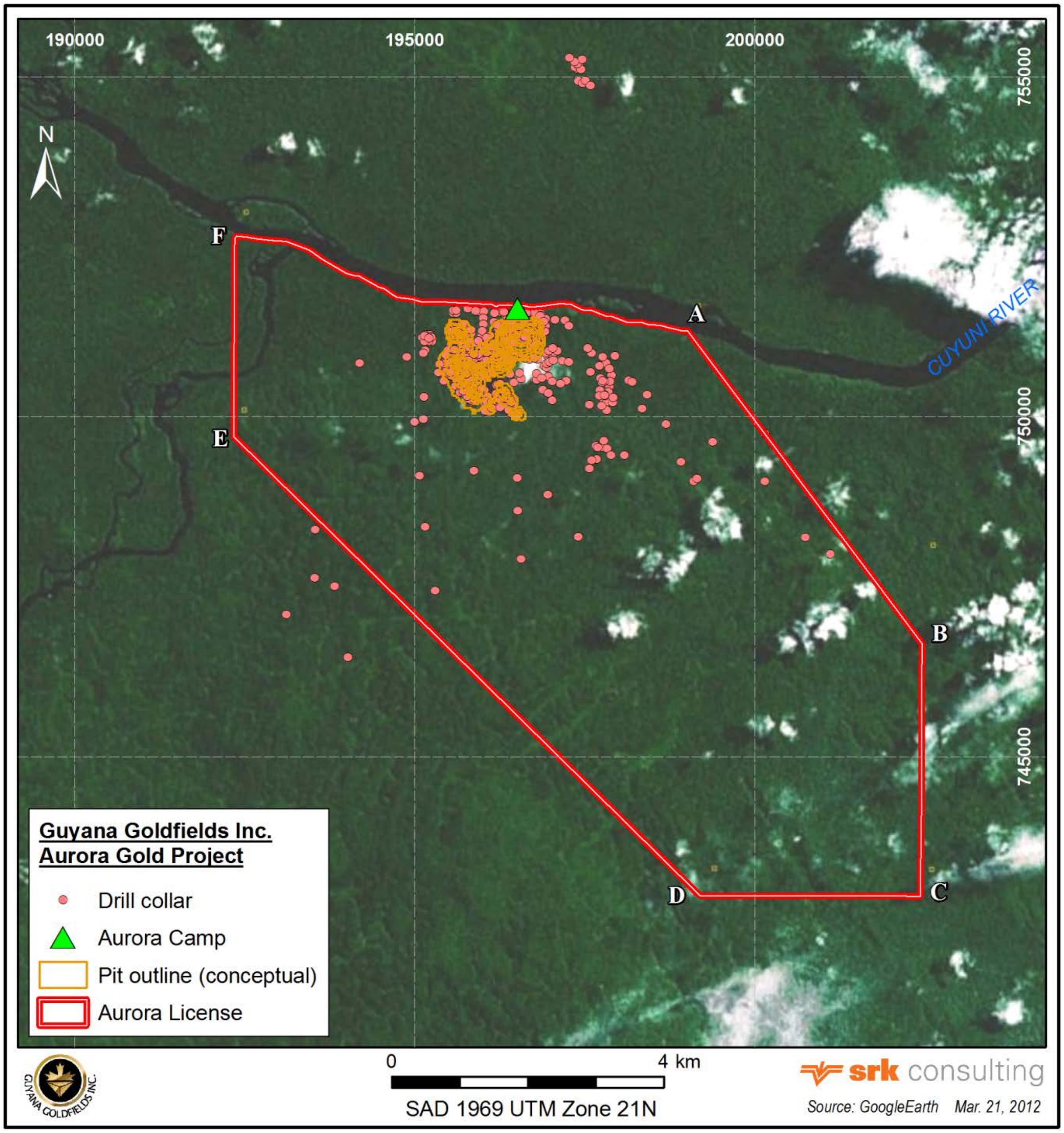
Table 3.1: Corner Points of Prospecting the Mining License

Corner Point ID	Latitude	Longitude
A	6°47'32"N	-59°43'17"W
B	6°45'38"N	-59°41'24"W
C	6°43'03"N	-59°41'24"W
D	6°43'03"N	-59°43'08"W
E	6°46'40"N	-59°46'54"W
F	6°48'15"N	-59°46'54"W

Guyana Goldfields has confirmed that the mineral tenure, surface rights as well as access and permitting issues of the Aurora Gold Mine have been reviewed and were found to be in good standing by independent legal counsel.

The Aurora Gold Mine Project is located in a remote part of the rainforest; hence, a precise description of the property boundaries is difficult. Guyana Goldfields retained Edward Luckhoo (from Montejo, 2009), a registered lawyer, to supply a legal opinion on the land position. Mr. Luckhoo's opinion is:

"From a reference point 'X' located with geographical co-ordinates of latitude 6°47′10″N, longitude 59°42′05″W and situated at the confluence of the Cuyuni River and Gold River, thence going upriver for a distance of approximately 1 mile 704 yards to point of commence 'A,' located with true geographical co-ordinates of latitude 6°47′32″N, longitude 59°43′17″W; thence at a true bearing of 135° for a distance of approximately 3 miles to point 'B,' located with true geographical co-ordinates of latitude 6°45′38″N, longitude 59°41′24″W; thence at a true bearing of 180° for a distance of approximately 3 miles to point 'C,' with geographical co-ordinates of latitude 6°43′03″N, longitude 59°41′24″W; thence at a true bearing of 270° for a distance of approximately 2 miles, crossing the Gold River to point 'D,' located with true geographical co-ordinates of latitude 6°43′03″N, longitude 59°43′08″W; thence at a true bearing of 315° for a distance of approximately 6 miles to point 'E,' located with geographical co-ordinates of latitude 6°46′40″N, longitude 59°46′54″W; thence at a true bearing of 0° for a distance of approximately 1 mile 1,513 yards to point 'F' on the right bank of the Cuyuni River, located with approximate geographical co-ordinates of latitude 6°48′15″N, longitude 59°46′54″W; thence going downriver along the right bank of the Cuyuni River for a distance of approximately 4 miles 633 yards to the point of commence ‘A,’ thus enclosing approximately 14,131.2 acres, save and except all lands lawfully held or occupied.”



Source: SRK, 2017 (Figure provided by Guyana Goldfields)

Figure 3.2: Aurora Gold Mine Project Land Tenure Map on LIDAR and SRTM Data

The mineral resources reported herein occur within an approximately 2 km long corridor, known as the Golden Square Mile, within the Mining Licence. In addition to the Mining Licence, Guyana

Goldfields holds 10 prospecting licences that are contiguous with the Mining Licence. The total combined area of all these licences is approximately 96,093 acres (38,900 hectares).

3.3 Underlying Agreements

Guyana Goldfields acquired its interest in the Aurora Gold Mine in accordance with an agreement dated May 20, 1998 between Guyana Goldfields and Mr. Alfro Alphonso. Guyana Goldfields was originally required to make annual advance royalty payments to Mr. Alphonso in the aggregate of US$225,000 per year during the three year period following the commencement of commercial production, and to pay an additional 2% net smelter royalty (NSR) to Mr. Alphonso thereafter. On March 18, 2004, the original agreement was amended, pursuant to which Guyana Goldfields agreed to pay Mr. Alphonso an annual fee of US$100,000 for as long as Guyana Goldfields maintains an interest in the Aurora Gold Mine, up to a maximum of US$1,500,000.

3.4 Permits and Authorization

All exploration programs to date were conducted under appropriate authorization, license, or equivalent control documents, which were obtained from the appropriate regulatory authority in Guyana.

The Mining Licence, obtained in November 2011, gives the company the right to build and operate the mine. At the same time the company signed a Mineral Agreement (MA) with the Government of Guyana and the Guyana Geology and Mines Commission which sets the fiscal regime, taxation and royalties as they effect the operation of the mine. This licence and the MA were signed by the Company and the Company's wholly owned subsidiary in Guyana, AGM, Inc. and are valid for 20 years and renewable on application for further 7 years periods for as long as mining operations continue on the property.

Significant details among the MA terms include:

- Mining royalty of 5% on gold sales at a price of gold of US$1,000/oz or less;
- Mining royalty of 8% on gold sales at a price of gold over US$1,000/oz;
- No withholding tax on interest payments to lenders;
- Duty and value added tax exemptions on all imports of equipment and materials for all continuing operations at the Aurora Gold Mine, including the construction and operation of a planned port facility, road and power improvements and the construction and operation of the mine; and
- Royalties are deductible from income taxes, and tax losses can be carried forward indefinitely.

Guyana Goldfields received its Environment Permit from the Guyana Environmental Protection Agency on September 27, 2010.

3.5 Environmental Considerations

Guyana Goldfields contracted Ground Structures Engineering Consultants Inc. of Georgetown, Guyana in 2005 to begin baseline environmental and social investigations. In April 2007, an initial Environmental and Social Baseline Report was prepared. As part of the preliminary environmental survey, baseline data was collected for the physical, biological and socio-cultural environments, including field surveys for water quality, flora and fauna. In addition, baseline data was collected in May 2009 and public consultation was conducted as part of an environmental and social report that was submitted to the Guyanese authorities on June 1, 2009 as part of the application process for a mining licence. An Environmental and Social Impact Assessment has been completed in line with the International Finance Corporation's Performance Standards on Social and Environmental Sustainability. To date, no significant social or environmental issues have been identified.

4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

4.1 Accessibility

4.1.1 Accessibility by Air

Guyana has two international airports. Cheddi Jagan International Airport is approximately one hour south of Georgetown, the nation's capital and the airport is serviced by international carriers. A smaller national and limited international airport, Ogle Airport, located 10 km east of Georgetown, provides access to regional Guyana and adjacent countries.

Guyana Goldfields has constructed a runway on the southern bank of the Cuyuni River adjacent to the camp, which is suitable for helicopters and short-takeoff-and-landing aircraft. The runway has an approximate length of 1200 m. The Company operates several charter flights per week from Ogle airport to the project site.

4.1.2 Ground Accessibility

The Aurora Gold Mine is also accessed via a road from the Buckhall Port facility. The road distance between the Buckhall Port facility and the Aurora Gold Mine site is approximately 150 km. The road alignment initially follows the north shore on the Cuyuni River, and crosses over the river at the Tapir Crossing via barge to continue to the mine site. Much of the existing road was constructed by Barama Company Limited for logging. The last 26 km of the road was built and is maintained by the Company.

4.1.3 Access to Buckhall Port

Buckhall Port is the logistics hub for the Aurora Gold Mine and is located on the west bank of the Essequibo River. From Georgetown, Buckhall Port is accessible by driving 42 km west via public highway to the Town of Parika. Parika is located on the east bank of the Essequibo River. From Parika, the Buckhall Port is accessible via boat or barge on the Essequibo River.

4.2 Local Resources and Infrastructure

The project is located in a very remote and uninhabited area of Guyana. Project execution required building all necessary infrastructure including the last 26 km of the access road to the project. The Aurora Gold Mine has an existing mancamp, light maintenance and fuel storage facilities, and an expanded camp to accommodate the upcoming planned expansion activities.

Basic supplies are available in Georgetown, which has a population of approximately 240,000. The city is located approximately 40 km east of Buckhall Port.

Most major items and equipment for the proposed mill expansion will be imported from overseas. Access to the site for project development will be primarily by road or air. Equipment and supplies entering the site will clear customs at Buckhall Port.

Power for the mining operations is generated on-site from diesel generators. Guyana Goldfields is currently undertaking a study to determine the feasibility of alternative energy sources for generating electric power.

Voice and data communication are currently provided by satellite services. The closest national telephone service is located in the town of Bartica, operated by the Guyana Telephone and Telegraph Company.

4.3 Climate

The Aurora Gold Mine is situated in the forested region of Guyana. Guyana lies within the equatorial trough zone and its weather and climate are influenced by the seasonal shifts of this trough and its associated rain bands.

Although one can expect rain every day in the forested region, the seasons are determined primarily by the variation in rainfall patterns.

There are two wet seasons, from April to August and from December to January, and two dry seasons, from February to April and from August to December. Average rainfall in the forest region is 2,124 mm per annum.

Relative humidity is high, ranging from 65% to 100%. Temperatures range from 22°C to 34°C year round.

The humid tropical climate of Guyana is moderated by the north-eastern trade winds. Exploration activities and mining operations will be conducted year-round.

4.4 Physiography

The property extends southeast from the Cuyuni River, which is approximately 50 m above sea level. The area is of moderate relief and covered with dense rainforest. The hills in the area range up to an elevation of 130 m. The low-lying areas result in large swampy areas during the rainy seasons. The landscape of the Aurora Gold Mine project area is shown in Figure 4.1.

Source: SRK, 2017
Looking south towards the open pit areas (Photos provided by Guyana Goldfields Inc.)

Figure 4.1: Landscape in the project area.

There are hills near the river, and to the southeast of the property there is a series of flat-topped hills that rise approximately 200 m above the river level.

Small hills are also present to the southwest of the property and rise approximately 40 m above the river level. These hills are formed of granitic rocks and clay-rich residual deposits that are cut by streams that drain into the Cuyuni River.

5 History

5.1 Summary

Gold mineralization in the project area was first discovered in 1911. Solar Development Company (Solar), a subsidiary of Cominco Ltd., explored the area during 1938 and 1939, in the Mad Kiss, Aleck Hill and Walcott Hill areas. Cuyuni Goldfields Company (Cuyuni) commenced underground mining at Aleck Hill in 1940 and produced an estimated 2,260 to 3,800 kg of gold through 1948. The Geological Survey of Guyana conducted an exploration program for copper in the area in 1963. In 1989, Denison completed a three year exploration program under agreement with South American Goldfields Inc., comprising gridding, soil, rock chip, saprolite, and stream sediment sampling; geological surface and underground mapping; underground sampling, geophysics and drilling. Gold Star Resources Ltd. (Gold Star) bought the rights in 1992 but let the rights lapse within the year. Guyana Goldfields acquired a 100% option on the property in 1998, and has carried out all subsequent exploration activities to date Guyana Goldfields obtained a reconnaissance permit covering approximately 600 km^2 surrounding the original A1 License, and, based on ongoing exploration work, were granted five new prospecting licenses contiguous with the A1 License by the Government of Guyana on June 29, 2004.

5.2 Exploration

Details about legacy exploration programs are limited. Information given in this section is sourced from Cargill and Gow (2003) and Cargill (2005), and is summarized in Table 5.1.

Table 5.1: Summary of Historical Exploration Work Prior to 1998

Period	Company	Activity	Drilling	UG Development	
1911		Discovery of gold.			
1934-1937		Numerous claims staked.			
1938-1939	Solar	Exploration.			
1940-1948	Cuyuni	Systematic development of claims, mining started in 1940.	30 surface (4,809 m) 26 UG (1,600 m)	To depth of ~ 75 m below surface at Aleck Hill	Est. 2,260 - 3,800 kg Au
1963	Geol. Survey of Guyana	Geochemical and geophysical surveys.	19 surface (2,515 m)		
1989-1992	South American / Denison	Gridding, mapping, soil, rock chip, saprolite, stream sediment sampling, airborne and ground geophysics.	56 surface (10,204 m)		
1992	Gold Star	Geochemical sampling.			
Mid-1990s	Mr. Alphonso / Coeur d'Alene	Geochemical survey.			

Gold mineralization in the project area was discovered in 1911 during the Pigeon Island gold rush (Bracewell, 1949). Limited activity is reported after the gold rush until 1934, when numerous mineral claims were staked over a three year period (1934-37). Solar Development Company (Solar), a subsidiary of Cominco Ltd., amalgamated these claims in 1938 and explored the area during 1938 and 1939. Exploration efforts during this time were primarily focused on the Mad Kiss area though gold mineralization at Aleck Hill and Walcott Hill was also identified.

In 1940, Cuyuni Goldfields Company (Cuyuni) acquired the rights to part of the project area and began to develop their mineral claims systematically. In 1945, Cuyuni was able to acquire the remainder of the claims that comprise the current project area. Mining activities commenced in 1940 and continued until 1948, at which point underground development at Aleck Hill had reached a depth of approximately 75 m below surface. Mining records are either missing or lack detail; hence, any production figures that are available are estimates at best. Webber (1952) estimated that approximately 2,260 kg to 3,800 kg of gold were produced by Cuyuni from mineralization with an average head grade of approximately 18 g/t Au.

Cuyuni drilled 26 surface core boreholes (4,321 m) and 26 underground core boreholes (1,600 m) at Aleck Hill and 4 surface core boreholes (488 m) at Mad Kiss.

Cuyuni ceased mining operations in 1948 and the project area lay dormant until 1963, when the Geological Survey of Guyana conducted an exploration program in the Haimaralli Falls area, along the northwest border of the Aurora project area. This program was aimed at identifying copper mineralization. The Geological Survey of Guyana carried out geochemical and geophysical surveys, consisting of Turam electromagnetics and ground magnetics, and completed 19 core boreholes (2,515 m). No significant copper mineralization was intersected.

No exploration work was carried out in or around the project area between 1963 and 1989. In 1989, South American Goldfields Inc. (South American) acquired an Exclusive Exploration Permit covering the Aurora project area. South American did not carry out any exploration work but had an agreement with Denison Mines Ltd. (Denison) to carry out exploration. Commencing in 1989, Denison completed a three year exploration program comprising gridding, soil, rock chip, saprolite, and stream sediment sampling; geological surface and underground mapping; underground sampling; and acquisition of airborne and ground geophysical data. Denison also drilled 56 core boreholes (10,204 m).

The aeromagnetic survey, carried out by Denison in 1990, covered the entire Aurora project area. Initial ground magnetic and induced polarization (IP) surveys were also carried out over the Aleck Hill mine area.

In 1991, the ground magnetic survey was extended northwest and southeast of the Aleck Hill prospect. Interpretation of aeromagnetic data resulted in 15 target areas that were investigated further. Recommendations included deep auger sampling of the target areas, testing 300 m^2 areas covering each target area identified.

An interpretation of the ground IP data by Denison identified 17 targets appropriate for drill testing. A subsequent drilling program by Denison tested only one of these targets and failed to intersect any gold mineralization.

Between 1989 and 1991, Denison drilled 19 core boreholes (2,515, m). Based on this work, Denison prepared a mineral resource estimate that incorporated gold mineralization at the Aleck Hill, Aleck Hill South, Walcott Hill East, Mad Kiss, and Mad Kiss South areas. Denison terminated its participation in the project in 1992.

In 1992, South American sold the rights to the Aurora property to Gold Star Resources Ltd. (Gold Star). Gold Star carried out an unknown amount of geochemical sampling in the same year but subsequently let the rights to the property lapse.

Sometime during the mid-1990s, Mr. Alfro Alphonso acquired the property and subsequently optioned the property to Coeur d'Alene Mines Ltd., who carried out a geochemical exploration program.

5.3 Exploration by Guyana Goldfields (1998 to 2009)

During 1998, Guyana Goldfields acquired a 100% option on the property from Mr. Alfro Alphonso. An unknown amount of geological mapping has been completed on the project area.

In 2002 and 2003, Guyana Goldfields conducted a drill program comprising 39 shallow core boreholes (1,076 m), deep auger sampling, trenching, and channel sampling on the A1 License. In December 2004, Guyana Goldfields obtained a reconnaissance permit covering approximately 600 km^2 surrounding the original A1 License.

Airborne magnetic, radiometric and electromagnetic surveys, and trenching and channel sampling were completed in 2004. Based on this exploration data, Guyana Goldfields applied for five new prospecting licenses contiguous with the original A1 License. These licenses were formally granted by the Government of Guyana on June 29, 2004.

From 2004 to 2009, delineation drilling was completed at the Aleck Hill, Rory's Knoll, Walcott Hill, Aurora and Mad Kiss areas. A total of 851 boreholes were drilled (196,301 m). A petrography study was also completed in 2005 (Kipfel, 2005) and an independent structural study by SRK was completed in April 2007.

A Mineral Resource Estimate was prepared by Micon for Guyana Goldfields in November 2007 (Mukhopadhyay, 2007). During 2008, a Preliminary Economic Assessment (PEA) was prepared by Snowden Associates (Myer, 2008).

In June 2009, AMEC (Montejo et al., 2009) prepared an updated resource model and updated the previous PEA. A Technical Report was completed by AMEC effective as of June 2009, based on drill data up to March 30, 2009. That Technical Report included a description of the exploration work completed by Guyana Goldfields up to March 2009.

Exploration completed since June 2009 is presented in Section 9 below. Details of previous exploration undertaken by Guyana Goldfields mentioned in this subsection are included in a Technical Report prepared by AMEC (Montejo et al., 2009), and are summarized in Table 5.2.

Table 5.2: Summary of exploration work by Guyana Goldfields between 1998 and 2009

Period	Company	Activity	Drilling
1998-2009	Guyana Goldfields	Geological mapping, geophysical surveys, geochemical sampling, trenching, drilling, petrography, independent structural study, mineral resource estimates, preliminary assessments, technical reports.	890 holes (197,377m)

5.4 Previous Mineral Resource Estimates

Micon prepared the first two Mineral Resource Estimates for the Aurora Gold Mine published in November 2007 and December 2008. The third Mineral Resource Estimate was prepared by AMEC in June 2009, and was considered for a preliminary economic assessment, also by AMEC.

The AMEC (Montejo et al, 2009) Mineral Resource Estimate was based on exploration data to March 30, 2009 (508 core boreholes). The estimate considered 173 resource domains generated by Guyana Goldfields.

Open pit mineral resources were constrained by a conceptual pit and underground mineral resources were reported below the conceptual pit. The conceptual pit envelope was designed at a gold price of US$750/oz. The mineral resources were reported at a range of cut-off grades, with the base case reported at cut-off grades of 0.85 g/t Au for open pit and 2.00 g/t Au for underground mineral resources, respectively (Table 5.3).

Table 5.3: Mineral Resource Statement, Aurora Gold Project, AMEC, effective June 2, 2009.

Category	Quantity (kt)	Grade (g/t Au)	Contained Metal (k Oz Au)
Open Pit[1]			
Measured + Indicated	11,136	3.31	1,185
Inferred	5,013	2.44	393
Underground[2]			
Measured + Indicated	13,010	4.85	2,027
Inferred	7,931	3.82	973
Combined Mining			
Measured + Indicated	24,146	4.14	3,212
Inferred	12,944	3.28	1,366

[1] Reported at a cut-off grade of 0.85 g/t Au
[2] Reported at a cut-off grade of 2.00 g/t Au

The fourth resource evaluation for the Aurora Gold Mine was prepared by SRK. This evaluation incorporated an additional 175 core boreholes (63,129 m) since the AMEC (2009) mineral resource evaluation. The effective date of this Statement of Mineral Resource was February 28, 2011. This Mineral Resource is presented in Table 5.4.

Table 5.4: Mineral Resource Statement, Aurora Gold Project, SRK Consulting (Canada) Inc., fffective February 28, 2011.

Category	Quantity (kt)	Grade (g/t Au)	Metal (k Oz Au)
Open Pit			
Measured	5.59	3.44	0.62
Indicated	11.69	3.54	1.33
Measured and Indicated	17.28	3.51	1.95
Inferred	3.53	3.74	0.42
Underground			
Measured			
Indicated	24.89	4.25	3.4
Measured and Indicated	24.89	4.25	3.4
Inferred	6.9	4.1	0.91
Combined Mining			
Measured	5.59	3.44	0.62
Indicated	36.58	4.02	4.73
Measured and Indicated	42.17	3.94	5.35
Inferred	10.43	3.98	1.33

Note: Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates. The cut-off grades are based on a gold price of US$1,045/oz and metallurgical recoveries of 95% for saprolite and Rock material. Open pit mineral resources are reported at a cut-off grade of 0.45 g/t Au inside conceptual pit shells, whereas underground mineral resources are reported at a cut-off of 2.0 g/t Au.

SRK prepared the fifth Mineral Resource Statements, which was published in September 2011. The borehole database contained updated drilling data for the period December 2010 to May 2011. It considered 939 exploration boreholes (291,556 m; excluding geotechnical and metallurgical holes), and 67,843 gold assay intervals.

This Mineral Resource Statement considered 163 new boreholes drilled on the property since the previous statement, which was prepared in February 2011. This Resource Statement is presented in Table 5.5.

Table 5.5: Consolidated Mineral Resource Statement* Aurora Gold Project, Guyana, SRK Consulting (Canada) Inc., effective September 9, 2011.

Category	Quantity	Grade	Contained Metal
	(kt)	(g/t Au)	(k Oz Au)
Open Pit			
Measured	5.75	3.29	0.61
Indicated	14.47	3.31	1.57
Inferred	3.53	3.74	0.42
Underground			
Measured			
Indicated	26.82	4.09	3.52
Inferred	6.49	3.74	0.78
Combined Mining			
Measured	5.75	3.29	0.61
Indicated	41.29	3.83	5.1
Inferred	9.97	3.63	1.17

MMC in conjunction with SRK prepared the sixth Mineral Resource Statement, which was published in January 2016. The mineral resource model was unchanged since the previous report, and was based on drilling data as of April 30, 2012. It was updated on September 30, 2015 to reflect updated topography due to open pit mining at Rory's Knoll. This Resource Statement is presented in Table 5.6.

Table 5.6: Consolidated Mineral Resource Statement* Aurora Gold Project, Guyana, SRK Consulting (Canada) Inc., effective September 30, 2015

Classification	Quantity	Grade	Contained Metal
	(kt)	(g/t Au)	(k Oz Au)
Open Pit Mining			
Measured	5,720	3.24	590
Indicated	26,780	2.51	2,160
Inferred	5,080	1.54	250
Underground Mining			
Measured			
Indicated	30,060	3.91	3,780
Inferred	11,810	4.12	1,560
Combined Mining			
Measured	5,720	3.23	590
Indicated	56,850	3.37	5,940
Inferred	16,890	3.8	1,810

6 Geological Setting and Mineralization

6.1 Summary

Project Mineral Resources are confined within an approximately 2 km long corridor, known as the "Golden Square Mile" within the mining license. The Golden Square Mile area of the Aurora Gold Project comprises folded metasedimentary and metavolcanic rock of the lower Cuyuni Formation that has been metamorphosed to greenschist assemblages. The Golden Square Mile is located within a broad regional, northwest trending, high strain zone characterized by strong northwest trending and sub-vertical foliation and dip slip shearing (southwest over northeast) and strain partitioning into interconnected network of discrete shear zones.

Gold mineralization at the Aurora Gold Project exhibits features analogous to mesothermal or "orogenic" gold deposits in the West-African Palaeoproterozoic Birimian Supergroup, with all gold mineralization controlled by a series of northwest trending shear zones.

6.2 Regional Geology

The Aurora Gold Mine is located in the Archean-Proterozoic Guiana Shield in northeast South America. The Guiana Shield is a palaeo-Proterozoic granite-greenstone terrane and is considered to be the extension of the West-African palaeo-Proterozoic Birimian Supergroup terrane. The Guiana Shield is largely composed of the Barama-Mazaruni Supergroup, a metasedimentary/greenstone terrane intercalated with Archean-Proterozoic gneisses that are intruded by Trans-Amazonian granites, as well as mafic and ultramafic rocks (McConnell and Williams, 1969).

The Barama Group consists of pelitic metasedimentary and metavolcanic rocks. The Mazaruni Group conformably overlies the Barama Group, which also consists of metasedimentary and metavolcanic rocks. The Mazaruni Group is subdivided into the Cuyuni Formation and the Haimaraka Formation.

The Cuyuni Formation consists of pebbly sandstone and intraformational conglomerate, intercalated with felsic to mafic volcanic rock. The Haimaraka Formation conformably overlies the Cuyuni Formation and consists of a thick sequence of mudstone, pelite, and graywacke; significant amounts of volcanic rock are absent from this unit (McConnell and Williams, 1969).

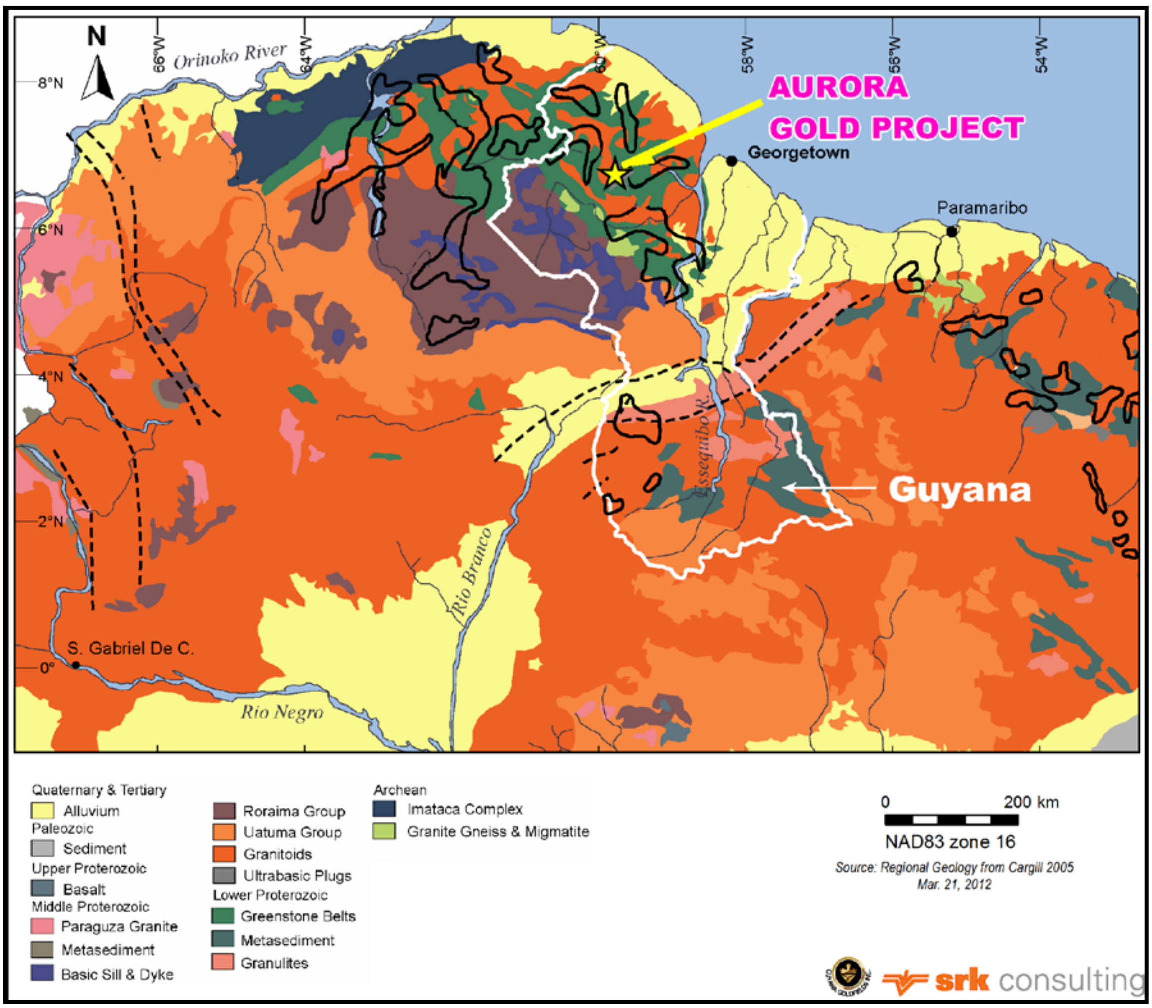
The Barama-Mazaruni Supergoup formed within a geosynclinal basin locally bordered by an Archean continental foreland. The Trans-Amazonian Orogeny, approximately 2,000M years ago, resulted in block faulting, crustal shortening, folding, metamorphism and anatexis of the Barama-Mazaruni Supergroup (Hurley et al., 1967).

The regional metamorphic grade of the Barama-Mazaruni Supergroup is generally lower to middle greenschist facies. Near the contact of some of the larger granitic complexes, the Barama-Mazaruni Supergroup is metamorphosed to upper greenschist to amphibolite facies.

Syn- to late-Tectonic calc-alkaline to intermediate intrusive rocks, collectively known as the Trans-Amazonian Granitoids (Voicu et al., 1999), were emplaced during the Trans-Amazonian Orogeny, between 2,250M and 1,960M years ago (Gibbs and Barron, 1993). They range in composition from granite to granodiorite, diorite, and adamellite.

Intrusive rocks proximal to the Aurora Gold Mine area include the Proterozoic-age Iroma-Ranka, Aurora, and Katruni medium-grained granodiorite and diorite intrusions, and late-stage basic sills and dykes.

Figure 6.1 shows the regional geology of Guyana and the Guiana Shield of northeast South America.



Source: SRK, 2017

Figure 6.1: Regional Geology Setting

6.3 Local and Property Geology

The Aurora Gold Mine is located within a regional northwest-striking sub vertical high strain zone, comprising of metasedimentary and metavolcanic rocks of the lower Proterozoic Cuyuni Formation developed along the northeast margin of an inferred Proterozoic granitic batholith. The distribution of the lithological units, hydrothermal alteration and strain intensity, is not constrained at surface, largely because of poor outcrop exposures and the saprolitic profile. The property geology description provided herein is largely based on drilling information.

SRK assisted Guyana Goldfields in completing a conceptual 3D geological model of the Golden Mile area where the mineral resources are confined (Figure 6.2).

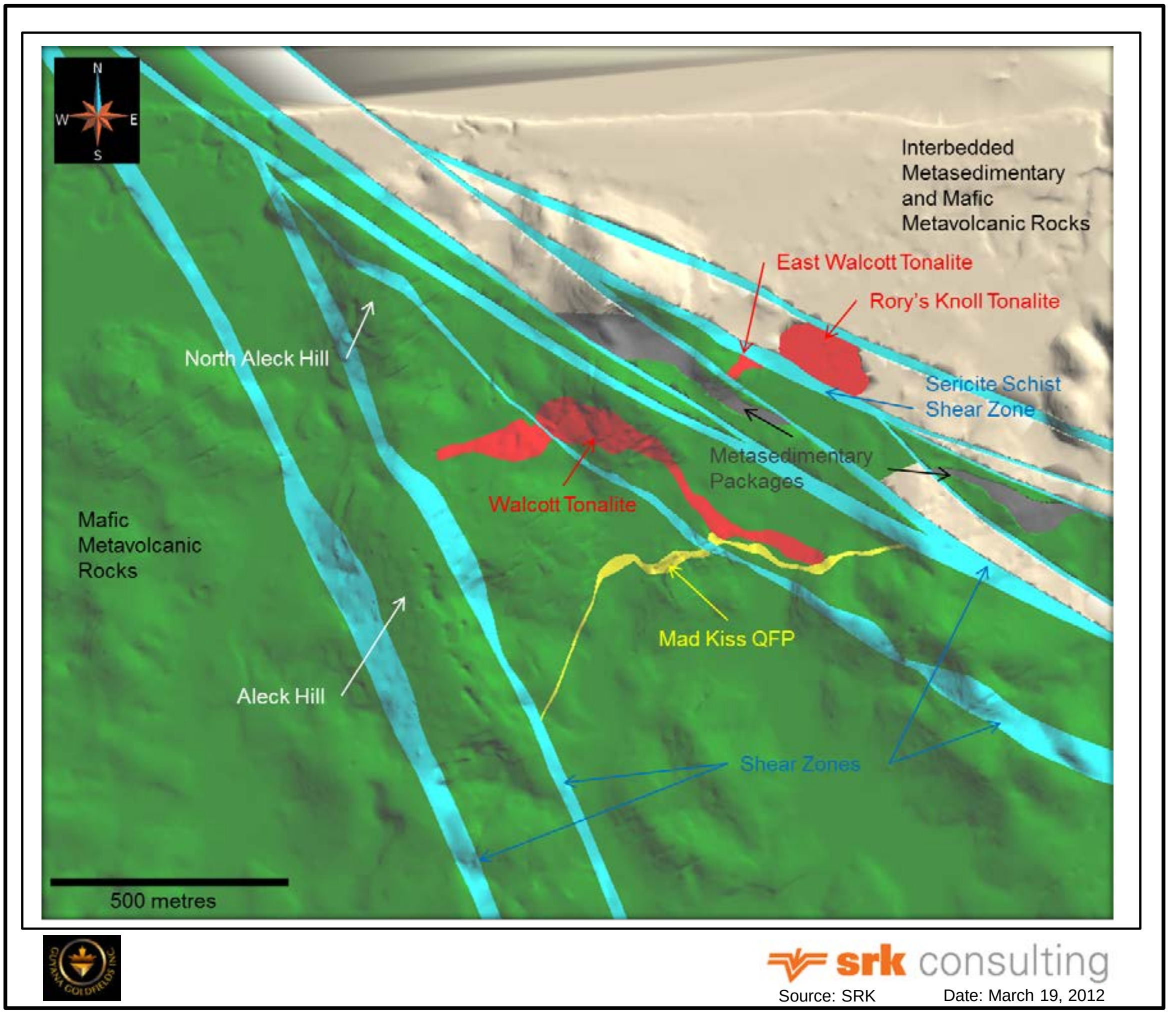


Figure 6.2: Plan View of Schematic 3D Geological Model of the Aurora Gold Mine

The area is affected by a series of 1 m to 50 m wide interconnected shear zones. The shear zones situated between the Rory's Knoll and Mad Kiss area strike northwest (between 290° and 305°) and dip steeply to the northeast (between 70° and 85°). In the Aleck Hill area, the two main shear zones strike to the southeast (approximately 155°) and dip to the southwest steeply (80°). The shear zones are characterized by a strong penetrative fabric locally forming schists in regions affected by the highest strain. Shear zones were modelled as litho-tectonic units and do not take into account the nature of the protolith. For instance, muscovite schists are probably derived from a mafic rock, not necessarily a felsic rock as often recorded in drill logs.

Steeply-plunging mineral lineations are conspicuous on shear fabrics suggesting that the shear zones have a dominant dip-slip component. The asymmetry of pressure shadows locally developed around competent porphyroblasts suggests southwest over northeast movement on the shear zones. Strain is strongly partitioned in the area and rock units adjacent to shear zones are considerably less-strained.

Hydrothermal alteration is spatially related to strain intensity, with the highest strained rocks usually displaying the strongest hydrothermal alteration. Alteration within the modelled shear zones varies between chlorite and muscovite, locally with strong silica replacement. The currently described felsic tuff relates to this muscovite alteration that affects all lithological units.

The locally named tonalite rock at Rory's Knoll is in actuality a mafic rock. It is possibly dioritic in composition that has undergone intense hydrothermal alteration, including silicification. It is unclear if this distinctive rock unit is intrusive. It is a competent lithology in which auriferous quartz-ankerite veins represent dilational sites developed during active deformation and fracturing. The auriferous veins of Walcott Hill, Walcott Hill East, and Rory's Knoll East are hosted within a similar rock type.

The quartz and feldspar porphyry (QFP) dyke(s) modelled in the Mad Kiss area, which could be referred to as a quartz phyric felsic intrusion, is also a competent lithology in which auriferous veins formed in response to dilation of that stiff rock unit.

Metasedimentary rocks logged as either "Ash Tuff" or "Metasediment" in the drill database consist of turbiditic laminated to thickly-bedded argillite and greywacke.

The interbedded sequence of metasedimentary and mafic metavolcanic rocks consists primarily of metasedimentary rock with mafic metavolcanic rock subunits. The interbedded unit is especially prevalent northeast of the Rory's Knoll deposit.

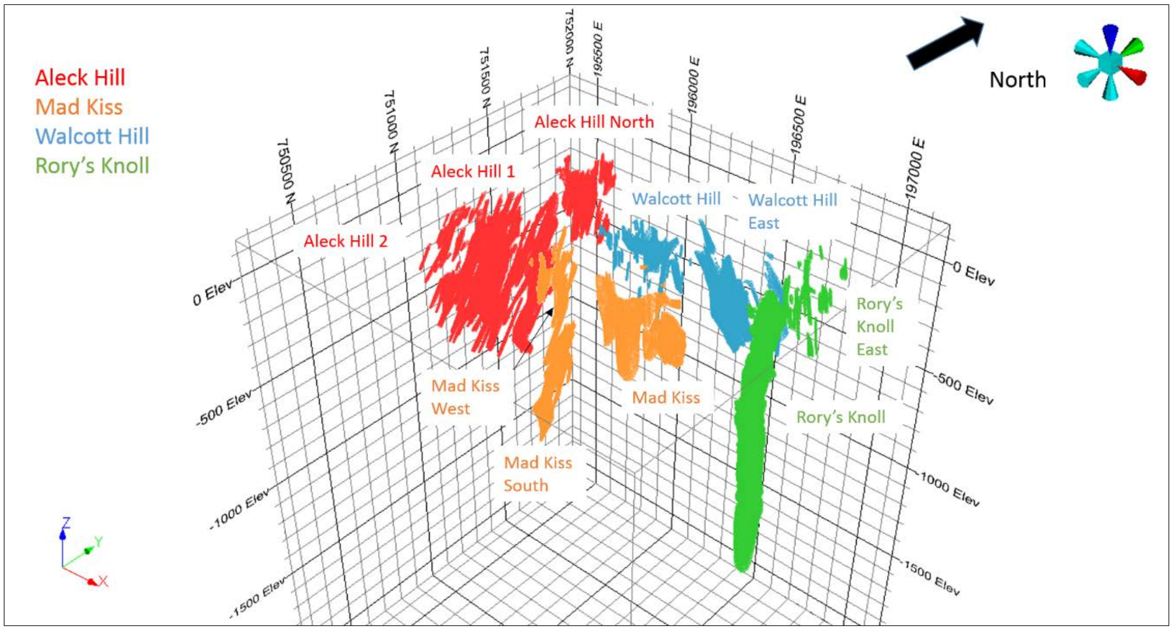
Mafic metavolcanic rocks form the southwest contact of Rory's Knoll, the host rock of the Aleck Hill deposit, and the Mad Kiss Quartz and Feldspar Porphyry. Some mafic rocks are very strongly magnetic, especially in the Mad Kiss and Walcott Hill areas. This is particularly apparent on the aeromagnetic data.

6.4 Mineralization

Gold mineralization in the Golden Mile area is controlled by a series of northwest-southeast trending shear zones. These shear zones are orientated sub parallel to the dominant northwest-southeast structural trend that occurs throughout the Aurora property. The shear zones contain a steep northwest-southeast trending foliation that formed during northeast-southwest shortening.

The gold mineralization in the Golden Mile area is chiefly associated with quartz-ankerite veins containing minor pyrite and associated hydrothermal alteration. The auriferous veins form weak to moderate stockworks preferentially in competent lithologies outside the most intensely deformed and altered segments of the shear zones. The auriferous veins have undergone minor post formation deformation. The geometry of the auriferous stockworks is controlled by the geometry of the competent lithologies and the geometry of shear zones, or a combination of both.

The Aurora Gold mine area is divided into four major gold mineralization zones (Figure 6.3): Rory's Knoll (includes Walcott Hill East), Aleck Hill (includes Aleck Hill North), Walcott Hill, and Mad Kiss (includes Mad Kiss West).



Source: SRK, 2017

Figure 6.3: Oblique Section of the Four Main Gold Mineralization Zones on the Aurora Gold Mine

6.4.1 Rory's Knoll

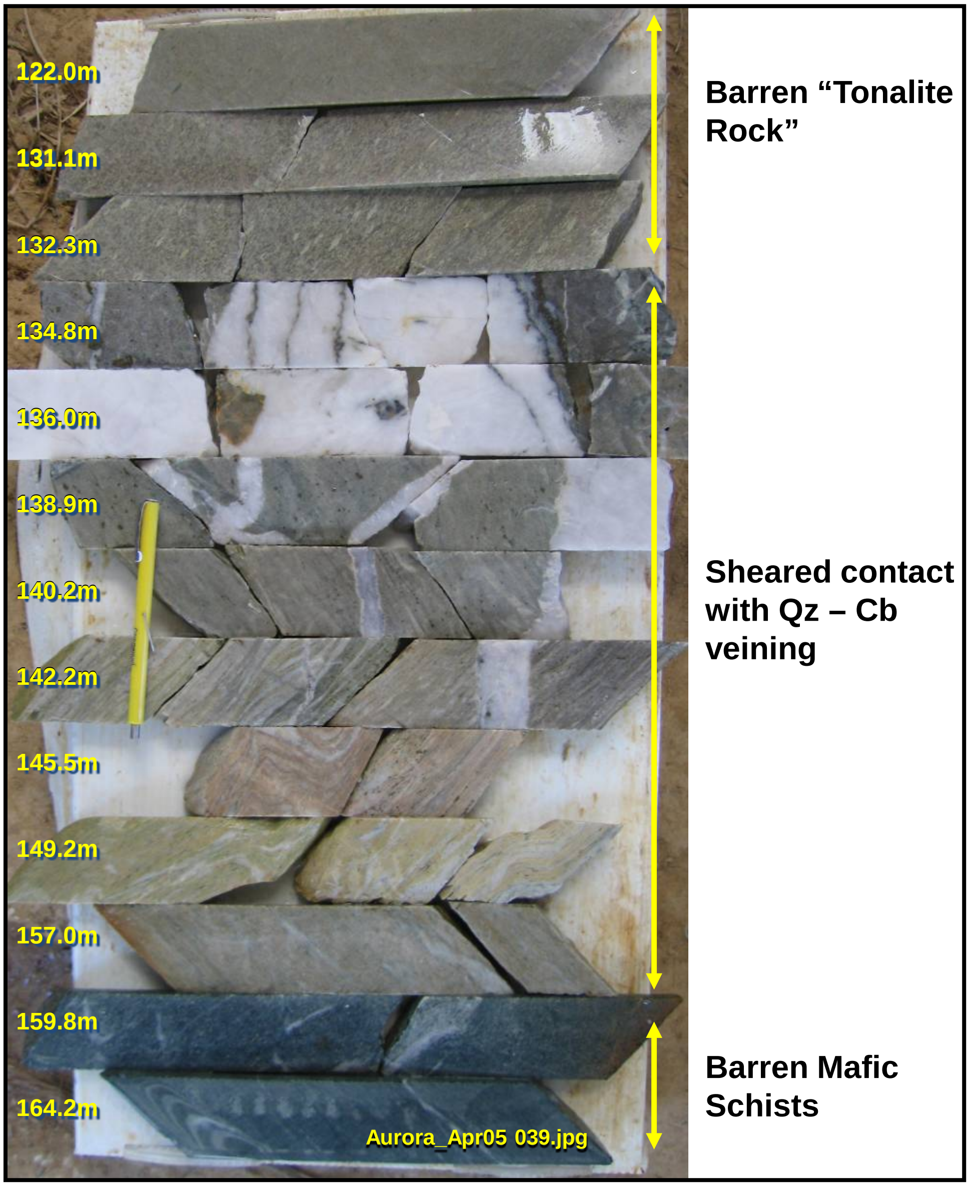
The Rory's Knoll Zone has been intersected by drilling to a depth of approximately 1,600 m below the surface. A total of 37 separate wireframes were interpreted from 266 core boreholes.

The gold mineralization at Rory's Knoll extends about 250 m along a trend of 300° west and plunges approximately 80° to the northwest. It is approximately 100 m thick at the widest point.

At Rory's Knoll the auriferous veins are developed within a distinctive, highly altered porphyritic diorite unit (locally known as "tonalite"). The tonalite is intensely carbonate and possibly albite altered.

A well-developed shear zone is located along the south-western contact of the Rory's Knoll tonalite. This shear zone is typified by strongly foliated muscovite schists with an associated down-dip stretching lineation. Abundant gold-rich quartz-ankerite veins occur in the tonalite near the sheared contact (Figure 6.4). Vein density increases towards the sheared contact.

The steep northerly plunge at Rory's Knoll is interpreted to represent the intersection between a sub vertical northwest trending shear zone and a steeply dipping, possibly west trending stiff lithology (altered diorite). Gold mineralization at Rory's Knoll persists to the west in Walcott Hill East.



Source: SRK 2011

Figure 6.4: Typical Section through Rory's Knoll (Borehole RKD-42).

The Walcott Hill East Zone is adjacent to and located approximately 60 m southwest of the Rory's Knoll Zone (Figure 6.3). Walcott Hill East is approximately 50 m in length with a thickness of approximately 15 m.

At Walcott Hill East, the auriferous zone consists of weak quartz-ankerite stockwork with intense "grey" silica alteration at the contact between a felsic rock and a mafic rock (Figure 6.5A). Auriferous quartz-carbonate veins are preferentially developed in a distinct stiff silica-rich lithological unit at the contact with muscovite schists to the east (Figure 6.5B). No veins occur in the enclosing muscovite schist. The rheology contrast between silica-rich rock and muscovite schist is interpreted to be controlling the formation of auriferous quartz-carbonate veins at Walcott Hill East.

The geometry of Walcott Hill East may also be controlled by the intersection of northwest-trending shear zones with favourable host rocks trending at high angle to the shear zones similar to Rory's Knoll. Thus, the overall plunge of the auriferous zones may be similar to that of Rory's Knoll, which has been better delineated by drilling. The Walcott Hill East Zone has been included in the drilling and wireframes for Rory's Knoll.

Source: SRK 2011

Figure 6.5: Typical Section through Walcott Hill East (A. Borehole EWD-19 and B. Borehole EWD-6).

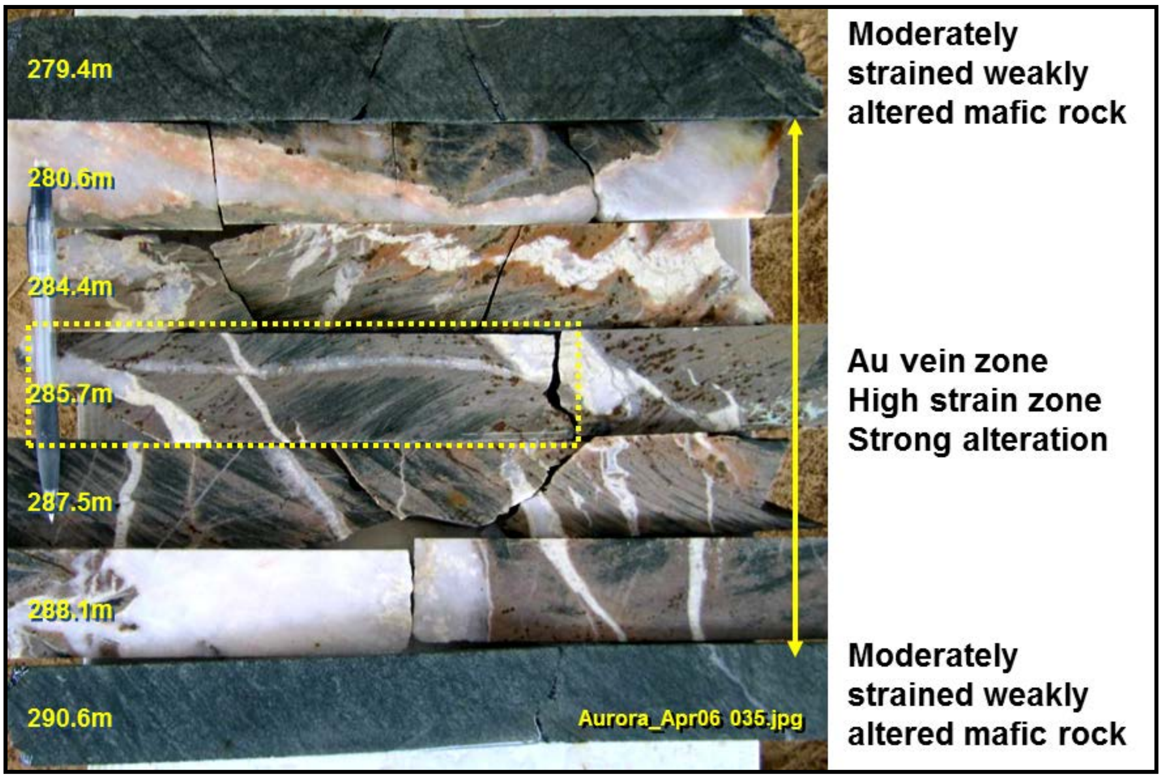
6.4.2 Aleck Hill

Aleck Hill is located approximately 1,000 m southwest of the Rory's Knoll Zone (Figure 6.3). Gold mineralization occurs in two zones: Aleck Hill and Aleck Hill North. The gold zones strike at 150° and dip sub vertically and can be up to 20 m wide. Aleck Hill extends for about 1,000 m along strike with higher-grade material and old underground workings extending about 400 m. The Aleck Hill Zone has been intersected by drilling to a depth of approximately 600 m below surface. A total of 408 boreholes intersect 210 wireframes that have been modelled for Aleck Hill.

Aleck Hill North is located approximately 500 m north northwest of Aleck Hill (Figure 6.3). The Aleck Hill North Zone extends for approximately 250 m along strike, is steeply dipping, and has a maximum thickness of approximately 35 m. The zone is truncated at its northern and southern end, but remains open to depth.

The gold mineralization at Aleck Hill is associated with discrete shear zones developed along lithological contacts within moderately strained, weakly altered mafic volcanic rocks (

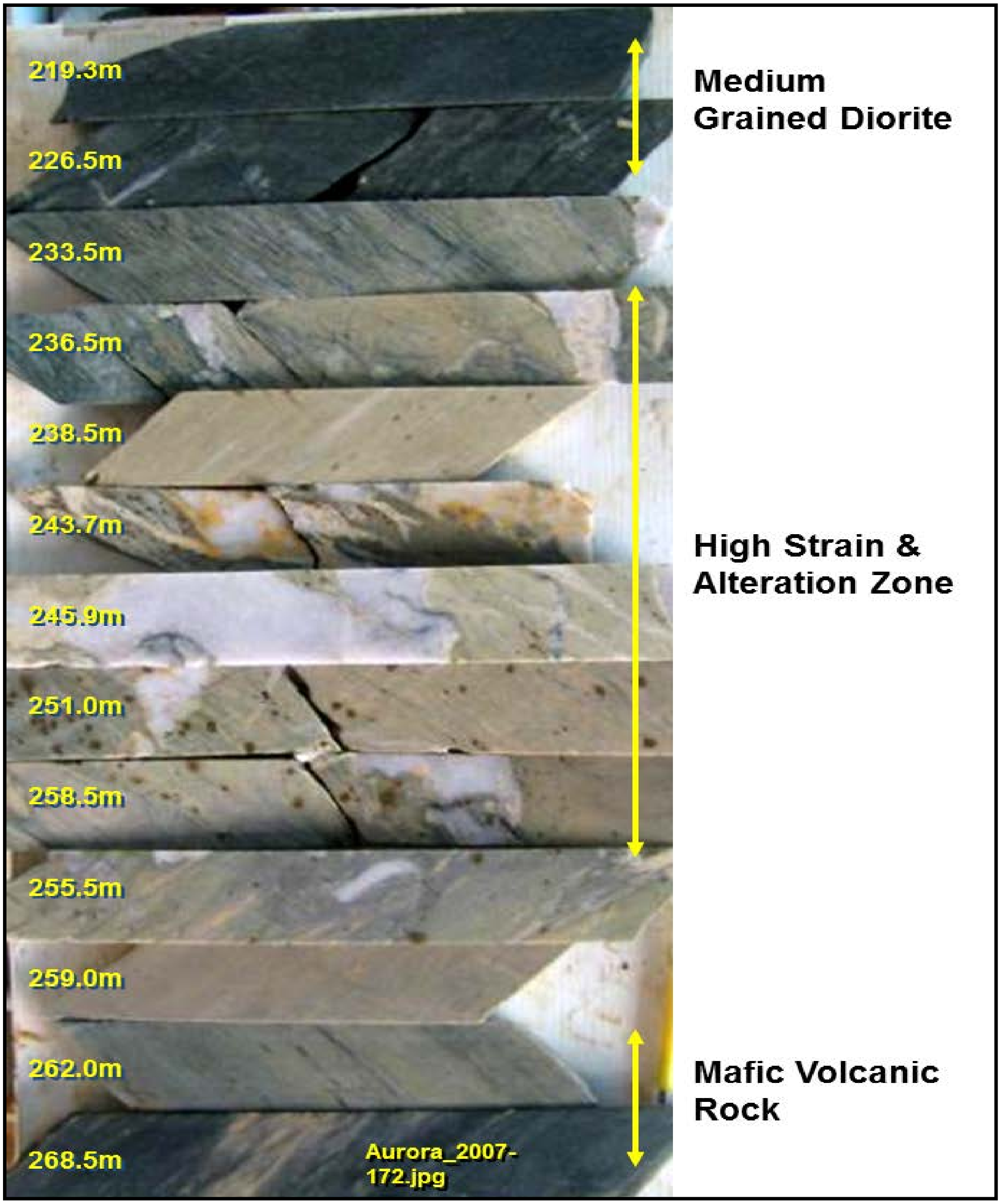
Figure 6.6). The overall trend of the auriferous zones at Aleck Hill is sub parallel to the trend of the northwest-trending regional foliation. Auriferous quartz-carbonate veins are developed inside the discrete shear zone. Auriferous veining formed late relative to the development of the foliation. Quartz-carbonate stockwork veins occur with both flat veins normal to foliation and high angle veins at low angle to foliation. Characteristic “pink” quartz alteration is associated with the quartz-ankerite veins at Aleck Hill. The pink mineral is silica and occurs in veins, in vein selvages, and as pervasive replacement in strong stockwork zones.



Source: SRK 2011

Figure 6.6: Typical Section through Aleck Hill (Borehole AHD-7).

At Aleck Hill North, high strain and alteration zones developed at gradational contacts between diorite and mafic volcanic rocks (Figure 6.7).



Source: SRK 2011

Figure 6.7: Typical Section through Aleck Hill North (Borehole NAHD-06).

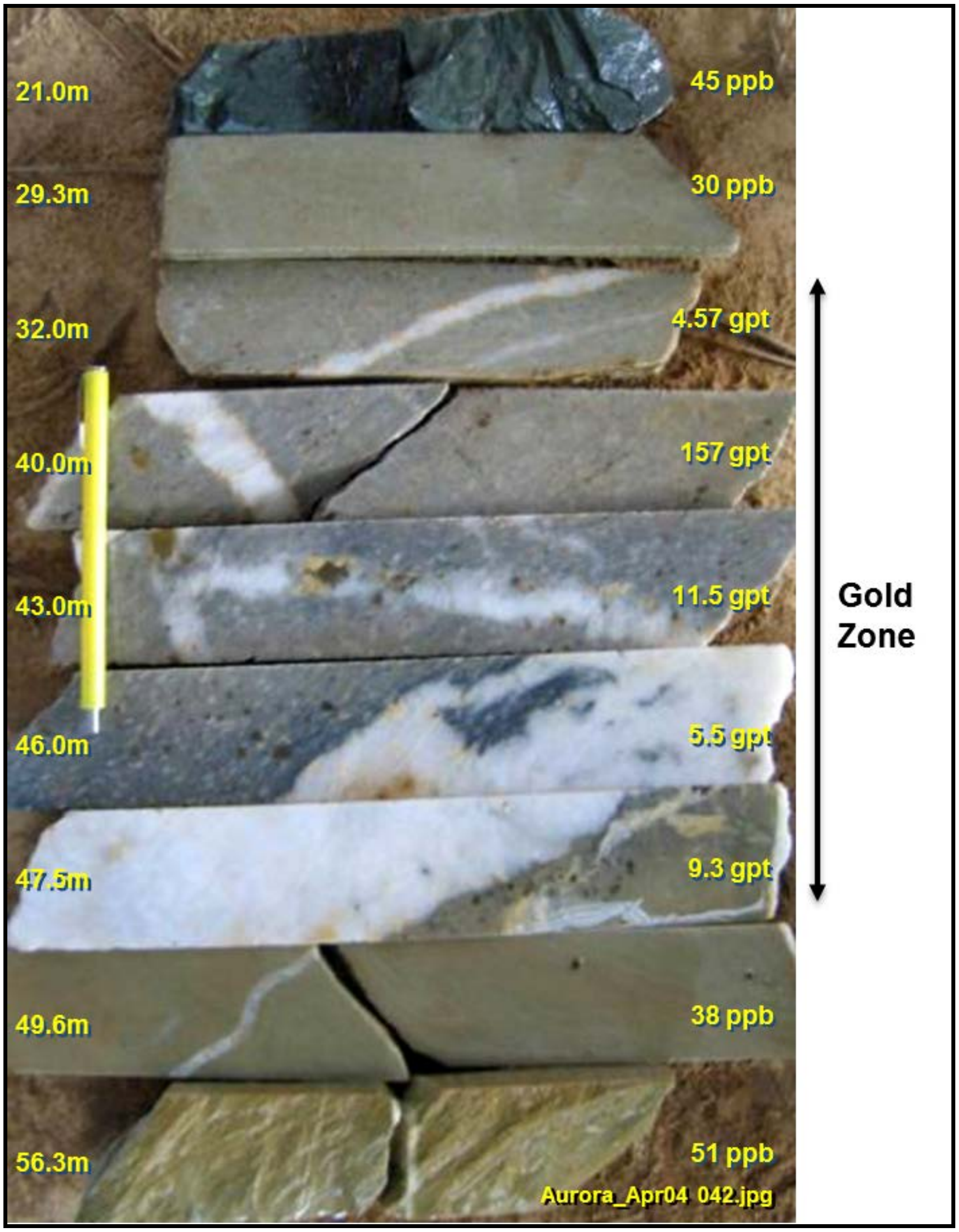
6.4.3 Walcott Hill

Walcott Hill is located approximately 500 m southwest of the Rory's Knoll Zone (Figure 6.3). Unpublished exploration reports from Guyana Goldfields describe the gold mineralization at

Walcott Hill as gold-bearing quartz veins with a thickness of up to 0.6 m striking 320° and dipping sub vertically for about 35 m along strike. Currently, the width is uncertain and the extent along strike and dip is poorly defined. The Walcott Hill Zone has been intersected by drilling to a depth of approximately 650 m below surface. A total of 41 boreholes intersect 47 wireframes that have been modelled for Walcott Hill.

6.4.4 Mad Kiss

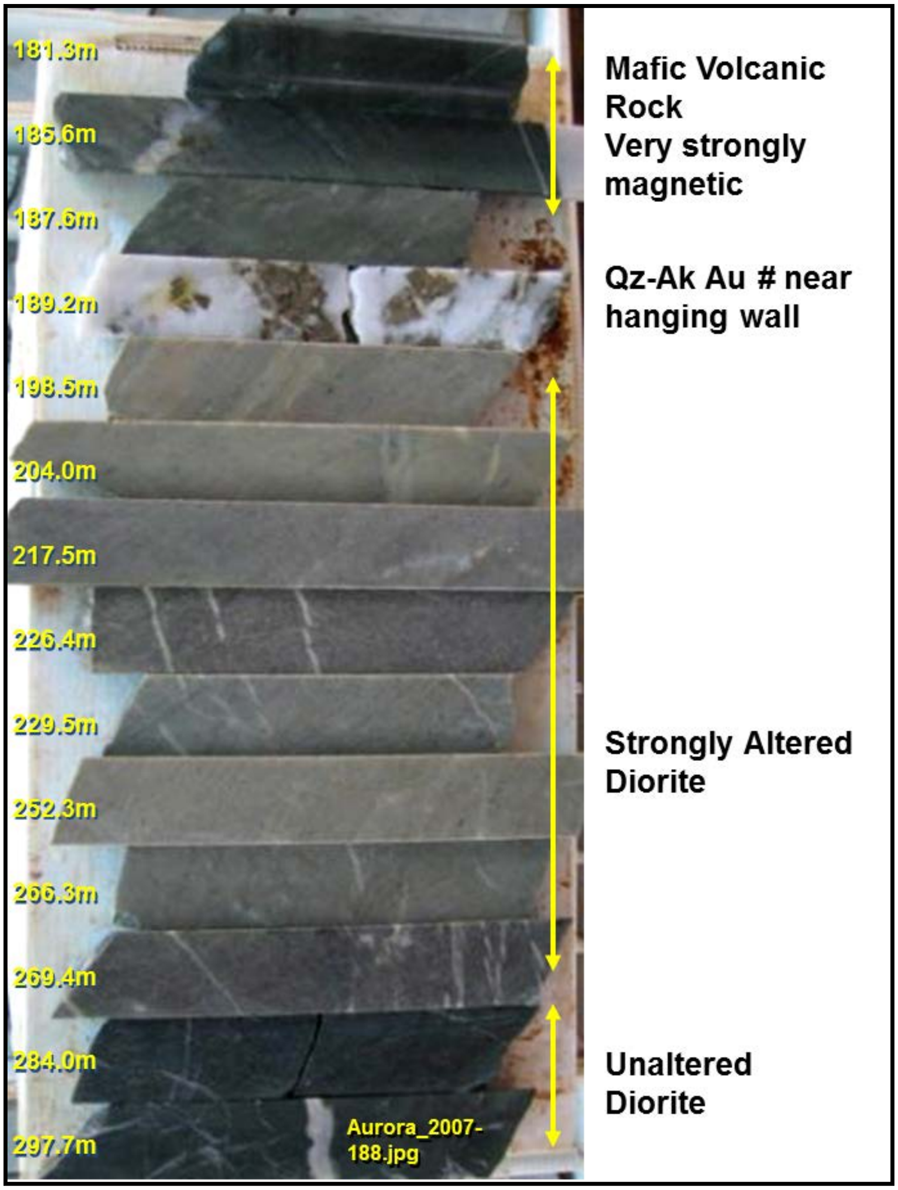
The Mad Kiss Zone is located approximately 750 m south-southwest of the Rory's Knoll Zone (Figure 6.3). In the Mad Kiss Zone, quartz-carbonate veining occurs inside a sheared quartz-feldspar porphyry dyke enclosed in foliated muscovite-rich rock (Figure 6.8). The gold-bearing stockwork system trending about 150° and dipping steeply north and south is also associated with the hanging wall and footwall contacts of the porphyry dyke. The quartz feldspar porphyry dyke is up to 150 m wide. Gold mineralization trends 250° dipping 70° north. Auriferous veins are 2 cm to 5 cm in thickness and occur parallel and normal to the regional foliation. The lower contact of the quartz feldspar porphyry dyke is sometimes marked by a thick quartz-carbonate vein with variable gold grades. There is no veining in muscovite-rich rock.



Source: SRK 2011

Figure 6.8: Typical section through Mad Kiss (borehole MKD-2).

The Mad Kiss Zone has been intersected by drilling to a depth of approximately 700 m below surface. A total of 224 boreholes intersect 52 wireframes that have been modelled for Mad Kiss. The Mad Kiss West Zone is located 200 m northwest of the Mad Kiss Zone, and contains gold mineralization striking between 250° and 290° and dipping sub vertically. At Mad Kiss West, a strong alteration zone is developed at the contact between unaltered strongly magnetic mafic volcanic rock and unaltered diorite (Figure 6.9). Quartz-ankerite gold veins near upper contact (hanging wall) and quartz-ankerite stockwork developed in strongly altered diorite rock of mostly unstrained veins. Available information suggests that the geometry of Mad Kiss, and possibly Mad Kiss West, may be controlled by the intersection of northwest-trending shear zones with favourable host rocks trending at high angle to the shear zones. Thus the overall plunge of the auriferous zones may be similar to that of the Rory's Knoll Zone, which has been better delineated by drilling. This will be dependent on the orientation of the favourable host rock (e.g., Mad Kiss quartz-feldspar porphyry dyke) compared to the orientation of the shear zones.



Source: SRK 2011

Figure 6.9: Typical section through Mad Kiss West (borehole WMKD-16).

7 Deposit Types

The gold mineralization at the Aurora Gold Mine exhibits features analogous to mesothermal or "orogenic" gold deposits typified by Archaean deposits of the Abitibi region, Canada. Features characteristic of the gold mineralization at the Aurora Gold Mine include:

- A strong spatial association to large scale shear zones;
- Relative late timing during active compressional deformation;
- A strong spatial association to large scale shear zones;
- Formed during greenschist metamorphic conditions;
- Association with a propylitic-phyllic alteration assemblages; and
- Is principally hosted in quartz-ankerite-pyrite veining.

8 Exploration

After a lengthy hiatus during the financing and development period of the Aurora Gold Mine exploration activity resumed in 2016. This work mainly consisted of mapping and assessment work at brownfield targets proximal to the Aurora Gold Mine as well as at the Company's Sulphur Rose deposit, located approximately 20km to the northeast of the Aurora Gold Mine. Drilling activities at both brownfield and greenfield targets are expected to ramp up significantly in 2017 once this target generation work has been completed.

9 Drilling

Drilling information prior to 2009 was summarized from Montejo et al. (2009). Information on drilling programs prior to Guyana Goldfields involvement is limited; available data includes 131 core boreholes (19,128 m) drilled by Cuyuni, the Geological Survey of Guyana, and Denison. Table 9.1 shows a summary of those drilling programs. Data from these historical boreholes were not considered for resource estimation.

Table 9.1: Summary of Historical Drilling on Aurora Gold Mine

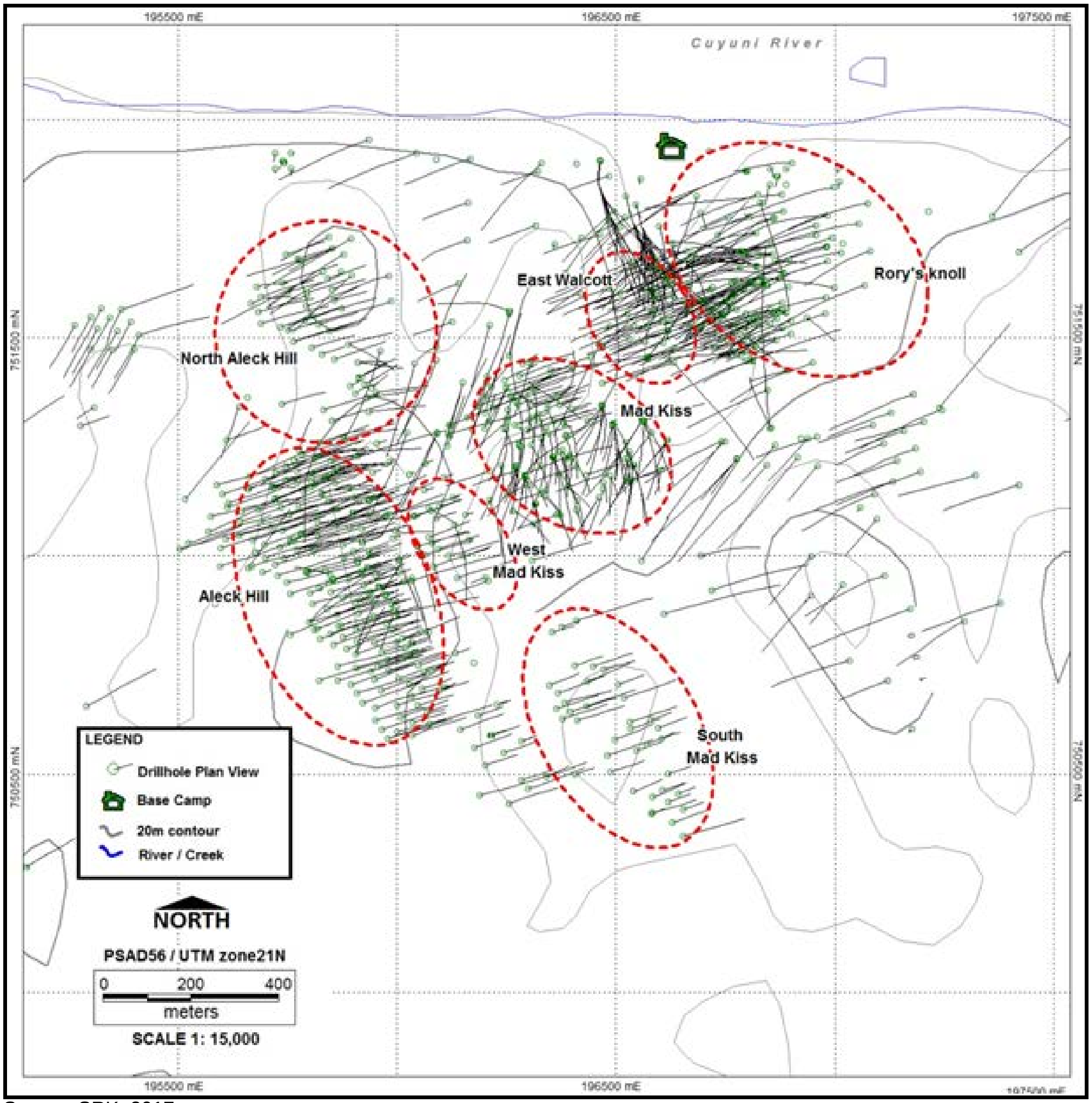
Company	Year	Area	Number of Boreholes	Length (m)
Cuyuni	1940-1948	Aleck Hill Surface	26	4,321
		Aleck Hill U/G	26	1,600
		Mad Kiss	4	488
		Subtotal	56	6,409
Geological Survey of Guyana	1963	Haimaralli Falls	19	2,515
		Subtotal	19	2,515
Denison	1989-1991	Aleck Hill	22	4,550
		Aleck Hill South	2	405
		Mad Kiss	16	2,233
		Mad Kiss South	10	1,850
		Walcott Hill East	3	552
		Walcott Hill	1	286
		Aleck Hill North	1	205
		Mad Kiss West	1	123
		Subtotal	56	10,204
Total			**131**	**19,128**

As of May 31 2011, 1,482 boreholes (totalling 379,853 m) have been completed on the Aurora Gold Mine by Guyana Goldfields (Table 9.2). This also includes boreholes drilled for geotechnical or metallurgical purposes. Table 9.2 presents a summary of drilling carried out by Guyana Goldfields during the period 2002 to 2011.

Core drilling completed since December 2010 focussed on infill existing auriferous zones or expanding their lateral and depth extensions. The primary purpose of the drilling was to increase the confidence in the continuity of the gold mineralization, improve geological modelling, and improve resource classification. As a result of this drilling, the geological interpretation was updated to incorporate the new drilling information for all zones, particularly for Rory's Knoll and Aleck Hill.

Table 9.2: Summary of Drilling by Guyana Goldfields Inc. between 2002 and 2011

Year	Area	Number of boreholes	Length (m)
2002-2003	Aleck Hill	26	738
	Mad Kiss	9	213
	Felice	4	125
	Subtotal	**39**	**1,076**
2004-2005	Aleck Hill Bedrock	16	3,104
	Aleck Hill Saprolite	26	1,933
	Aleck Hill North	5	1,468
	Walcott Hill East	6	1,264
	Walcott Hill	11	1,681
	Mad Kiss South	8	825
	Mad Kiss	14	2,672
	South East Aurora	6	1,532
	Rory's Knoll	52	17,462
	Subtotal	**144**	**31,941**
2006	Aleck Hill Bedrock	26	7,187
	Aleck Hill Saprolite	18	1,266
	Rory's Knoll	27	12,385
	Aleck Hill North	9	1,952
	Walcott Hill East	11	3,744
	Mad Kiss	6	1,450
	Mad Kiss West	18	5,941
	South East Aurora	2	607
	Felice	2	449
	Geophysical Anomalies	11	2,814
	Port-Knockers Workings	11	1,997
	Swamp Vein	6	1,017
	Powis Hill	5	813
	Marupa	13	3,009
	Subtotal	**165**	**44,631**
2007	Aleck Hill Bedrock	11	3,656
	Aleck Hill Saprolite	11	778
	Rory's Knoll	52	20,249
	Walcott Hill East	5	1,713
	Walcott Hill	11	2,500
	Mad Kiss	2	430
	Mad Kiss West	1	251
	South East Aurora	10	3,619
	Geophysical Anomalies	2	755
	Swamp Vein	1	202
	Subtotal	**106**	**34,153**
2008	Aleck Hill Bedrock	5	1,687
	Rory's Knoll	36	12,973
	Aleck Hill North	25	5,054
	Walcott Hill East	16	5,732
	Mad Kiss	7	1,082
	Mad Kiss South	14	2,447
	Mad Kiss West	13	2,460
	Swamp Vein	20	3,127
	Southeast Rory's Knoll	3	565
	Subtotal	**139**	**35,127**
2009	Aleck Hill Saprolite	25	1,626
	Aleck Hill Bedrock	52	10,912
	Rory's Knoll	25	7,688
	Aleck Hill North	26	4,495
	Walcott Hill East	8	1,385
	Walcott Hill	19	2,686
	Mad Kiss	28	6,095
	Mad Kiss South	17	2,387
	Mad Kiss West	5	705
	Swamp Vein	8	1,345
	Rock Mechanics	18	5,221
	Soil Geotechnical	46	2,205
	Metallurgical	7	1,011
	Condemnation	13	2,689
	Subtotal	**297**	**50,450**
2010	Aleck Hill	72	24,010
	Rory's Knoll	14	3,093
	Aleck Hill North	14	3,424
	Walcott Hill East	12	2,402
	Mad Kiss	60	21,301
	Mad Kiss West	4	1,717
	Powis Hill	6	2,153
	Condemnation	25	7,587
	Rock Mechanics	2	549
	Soil Geotechnical	16	3,714
	Subtotal	**225**	**69,950**
2011	Aleck Hill	120	40,086
	Rory's Knoll	20	12,750
	Aleck Hill North	46	11,226
	Walcott Hill East	38	10,878
	Mad Kiss	35	14,185
	Mad Kiss West	32	8,299
	Mad Kiss South	16	3,198
	Marupa	13	1,613
	Condemnation	21	6,950
	Soil Geotechnical	26	3,339
	Subtotal	**367**	**112,526**
	Total	**1482**	**379,853**



Source: SRK, 2017

Figure 9.1: Borehole Location Plan at Aurora Gold Mine. Source: Guyana Goldfields, 2012

9.1 Downhole Surveying

Drill collars were surveyed by Guyana Goldfields personnel using a laser theodolite. Downhole surveys were carried out using a single-shot Reflex survey tool at nominal 50 m intervals.

9.2 Drilling Pattern and Density

During early phases of exploration, Guyana Goldfields located boreholes on a grid with 200 m on centre. Borehole spacing was later decreased to 100 m on centre and finally to 50 m on centre, while still maintaining a general grid pattern.

9.3 Drill Core Sampling

From July 2009 onwards, five geologists were assigned to the project to ensure orderly monitoring of the drilling program. One geologist was assigned to quality assurance and quality control and all core sampling was conducted under his supervision.

The core was placed in plastic core boxes at the drill rig holding 3 m of HQ and/or NQ diameter sized core (6.35 and 4.76 cm diameter, respectively). Core boxes were then transported to the Aurora camp for logging and sampling. Drill core is stored on the property in plastic core boxes.

The core was photographed and rock quality designation (RQD) measured. Logging was carried out by Guyana Goldfields geological personnel recording lithology, alteration, mineralization, and structural features of the core. Once logging was completed, sulphide mineralized, altered, and quartz veined sections were marked for sampling. Both bedrock and saprolite core were sampled. Core recovery is very good, usually approximating 100%, except locally in strongly muscovite-altered rocks.

Sample length is based on geology and sample intervals do not cross lithological contacts. Sample lengths range from 1 m to 3 m.

Unweathered samples were cut in half using a diamond saw and saprolite core was usually cut in half with a knife with fragments of quartz vein material split in a Longyear core splitter.

Assay samples were labelled and placed into a plastic sample bag and sealed for shipment to the Acme Laboratory sample preparation facility in East Coast Demerara, Guyana. The half of the core sample not submitted for assaying was returned to the core box with the sample interval and sample number clearly indicated on the core box with the split core.

Upon collection of a sufficient amount of samples, they were delivered to the Acme preparation laboratory by company personnel in company-owned vehicles.

Sample pulps were shipped from the sample preparation laboratory to Acme's analytical laboratory in Santiago, Chile.

9.4 SRK Comments

In the opinion of SRK, the sampling methodology and procedures used by Guyana Goldfields are appropriate. The core samples were collected by competent personnel using procedures meeting generally accepted industry best practices. SRK concludes that the samples are representative of the source materials and there is no evidence of sample bias.

10 Sample Preparation, Analyses, and Security

Sample preparation, analysis, and security prior to 2009 are described in detail in Montejo et al. (2009) and are summarized below.

Until the 2009 drill program, Guyana Goldfields sent all but umpire samples to Loring Laboratories (Guyana) Ltd. (Loring) for sample preparation and assaying. Loring is a small, unaccredited laboratory with two laboratories, one in Guyana and one Canada. Between 2004 and 2006, Guyana Goldfields submitted an unknown number of samples for check assaying to ALS Chemex Laboratories (ALS) in Santiago, Chile as well as to the Omai gold mine laboratory operated by Cambior Inc. in Guyana for check assaying. ALS operates under a global quality management system that is accredited ISO9001:2000. The laboratory facility at the Omai gold mine was not accredited.

In early 2009 AMEC and subsequently Guyana Goldfields conducted extensive reviews of sample preparation procedures and the analytical performance of the Loring laboratory. Following the reviews, Guyana Goldfields hired a quality assurance and control manager and improved their analytical procedures.

During the 2009 to 2012 period, Guyana Goldfields has used two facilities of Acme Analytical Laboratories Ltd. (Acme), one in Georgetown, Guyana and in Santiago, Chile as their primary preparation and assaying laboratories. The management system of both laboratories is accredited ISO 9001:2000.

10.1 Sample Preparation and Analyses

10.1.1 Sample Preparation and Analyses Prior to 2009

Samples were shipped to Loring in Georgetown, Guyana under the supervision of Guyana Goldfields staff. At the laboratory, samples were dried and crushed to 95% passing a 10 mesh screen. Following crushing, samples were riffled to 300 g and pulverized to 80% passing a 150 mesh screen. Procedures prior to 2009 are described in Montejo et al. (2009).

10.1.2 Sample Preparation and Analyses from 2009 to Present

Samples were shipped to the Acme sample preparation laboratory in East Coast Demerara, Guyana. The assay samples were dried at 60°C followed by crushing to 85% passing a 2 mm screen. An 800 g split was then pulverized to 95% passing a 106 micron screen. A 150 g subsample was taken, placed in a paper envelope and shipped to the Acme analytical laboratory in Santiago, Chile. The remainder of the sample was stored in a plastic bag.

Samples were assayed for gold on 50 g sub-samples using a standard fire assay procedures with an atomic absorption finish (FA/AAS). Samples assaying more than 3.0 g/t Au were re-assayed using gravimetric methods.

10.2 Quality Assurance and Quality Control Programs

Quality control measures are typically set in place to ensure the reliability and trustworthiness of exploration data. These measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management, and database integrity. Appropriate documentation of quality control measures and regular analysis of quality control data are important as a safeguard for project data and form the basis for the quality assurance program implemented during exploration.

Analytical control measures typically involve internal and external laboratory control measures implemented to monitor the precision and accuracy of the sampling, preparation, and assaying. They are also important to prevent sample mix-up and to monitor the voluntary or inadvertent contamination of samples.

Assaying protocols typically involve regularly duplicating and replicating assays in addition to inserting quality control samples to monitor the reliability of assaying results throughout the sampling and assaying process. Check assaying is normally performed as an additional test of the reliability of assaying results; it generally involves re-assaying a set number of sample rejects and pulps at a secondary umpire laboratory.

10.2.1 Quality Assurance and Quality Control Programs Prior to 2009

From 2000 to 2003 Guyana Goldfields did not have formal analytical data quality control procedures, relying on the quality control measures undertaken by the primary laboratories. Starting in 2004, Guyana Goldfields began inserting control samples (blank and certified reference material samples) within samples batches submitted for assaying. Blank material was sourced from crushed coarse granite near Georgetown, Guyana. Guyana Goldfields purchased 12 different reference material standards from Ore Research & Exploration Pty Ltd. (Ore Research), Bayswater North, Victoria, Australia with gold values between 0.046 g/t Au to 9.64 g/t Au. Further details about the analytical quality control programs implemented by Guyana Goldfields are described in Montejo et al. (2009).

10.2.2 Quality Assurance and Quality Control Programs 2009 to Present

For the Aurora Gold Mine, Guyana Goldfields relied partly on the internal analytical quality control measures implemented by Acme Labs. In addition, Guyana Goldfields implemented external analytical control measures on all diamond drill hole sampling. This consisted of using control samples in all sample batches submitted for assaying, including field blanks, certified standard reference material, and field duplicates. Field blanks consist of a material locally sourced from a coarse gravel unit. Field duplicates consist of quarter core. During the period May 2011 to April 2012 seven commercially certified gold standard (GS) reference materials and two commercially certified blank standards (BL) reference materials sourced from CDN Resource Laboratories Ltd. (CDN Resources) in Canada were used on sampling. They are listed in Table 10.1.

Field blanks and field duplicates were inserted approximately every thirty samples and reference materials were inserted approximately every twenty samples.

Table 10.1: Specifications of Control Samples used by Guyana Goldfields on the Aurora Gold Mine between November 30, 2010 and July 31, 2011

Reference Material	Source	Certified Grade (g/t Au)	Standard Deviation	Assay Count
CDN-BL-4	CDN	<1	-	94
CDN-GS-P7B	CDN	0.71	0.035	185
CDN-GS-3G	CDN	2.59	0.09	140
CDN-GS-3F	CDN	3.1	0.12	50
CDN-GS-5F	CDN	5.27	0.17	110
CDN-GS-10C	CDN	9.71	0.325	31
CDN-GS-11A	CDN	11.21	0.435	33

10.3 Sample Security

10.3.1 Sample Security Prior to 2009

SRK has no information regarding sample security procedures during exploration work prior to 2009.

10.3.2 Sample Security: 2009 to Present

Guyana Goldfields maintains a well-documented chain of custody for the assay samples submitted for assaying. Drill core is under the control of drilling contractors who deliver core boxes to the Aurora camp for logging. The logging and sampling areas are secured by a fence. Once samples have been taken, they are securely packed for shipment in sealed rice bags. During shipment, which is carried out by company personnel in company-owned vehicles, sample batches are accompanied by sample submission forms. Samples are delivered directly to the preparation facility in Georgetown. Retained core, pulp, and pulp duplicate samples are stored in a company-owned, secure facility in Georgetown.

10.4 SRK Comments

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

11 Data Verification

11.1 Verifications by Guyana Goldfields

Guyana Goldfields and their independent consultants completed several verification programs for the preparation of previous technical reports including Cargill and Gow (2003), Cargill (2005), Mukhopadhyay (2007), and Montejo et al (2009).

Montejo et al (2009) reviewed the analytical quality control data acquired between 2004 and 2008. The review included independent auditing of the exploration database and the performance of assaying results delivered by the primary laboratories used by Guyana Goldfields. During the review, AMEC identified problems in the assay results delivered by Loring and suggested the implementation of several improvements, including re-assaying of a large percentage of the samples originally assayed by Loring, the use of commercial exploration database software and enhancements to the sample handling practices to avoid mix-ups and mislabelling. In late 2009, Guyana Goldfields retained AMC Mining Consultants (Canada) Ltd (AMC) to review quality control issues and the presence of coarse gold in particular.

In July 2009, Guyana Goldfields made two changes to enhance the integrity of the project data; switching the primary assay laboratory from the non-accredited Loring laboratory to the accredited Acme laboratory and implementing a Century Systems master database, which incorporates stringent data security protocols.

Since late 2009, Guyana Goldfields has implemented all the recommendations expressed by independent consultants and SRK considers that the quality assurance and quality control measures put in place on the Aurora Gold Mine are consistent with generally recognized industry best practices for a pre-development stage exploration property.

11.2 Verifications by SRK

11.2.1 Site Visit

In accordance with NI 43-101 guidelines, SRK has visited the Aurora Gold Mine on four occasions.

For the purpose of this feasibility study update, Mr. Tim Carew, P.Geo, Mr. Bob McCarthy, P.Eng, Mr. Jordan Severin, P.Eng, and Mr. Eric Olin, P.Eng visited the property from October 25th to 28th, 2016 to review geological, mining/production, metallurgical and geotechnical data respectively.

The first visit was conducted by Dr. James Siddorn, P.Geo, and Dr. Jean-François Couture, P.Geo, from April 3 to 10, 2007. The purpose of that visit was to conduct structural geology investigations to ascertain the controls on the distribution of the gold mineralization. In April 2007, SRK examined core intervals from 38 boreholes investigating 10 separate gold zones.

Dr. Jean-François Couture visited the project for a second time on January 21, 2011 to review exploration work completed on the project since April 2007. Mr. Glen Cole, P.Geo, and Ms. Dorota El-Rassi, P.Eng, visited the project from May 24 to 27, 2011 to examine core, interview field staff, and to gather information required to produce a technical report. A focus of this site visit was to devise an appropriate modelling strategy to adequately define gold mineralization at Aurora.

11.2.2 Database

SRK was provided with a GEMS database containing updated borehole data produced during the period of May 2011 to April 2012. SRK was also provided with revised gold mineralization wireframes for all the gold mineralized domains with the exception of Rory's Knoll in DXF exchange and GEMS formats. SRK worked with Guyana Goldfields to define criteria for the definition of these revised gold mineralization domain wireframes.

SRK conducted a series of routine verifications to ensure the reliability of the electronic data provided by Guyana Goldfields. These verifications included checking the borehole data for minimum and maximum values for each field and confirming/editing those outside of the expected ranges; checking for inconsistency in lithological unit terminology and/or gaps in the lithological code; and checking for gaps, overlaps and out of sequence intervals for both assays and lithology tables.

For the wireframes crossovers, duplicate triangles, gaps, and edge boundary joining were verified. SRK found the GEMS database to be in good order and well maintained. SRK considers the database suitable for resource estimation. After review, SRK considers that the gold mineralization wireframes interpreted by Guyana Goldfields represent adequate boundaries for the gold mineralization and suitable for use as resource domains for the purposes of this study.

11.2.3 Verifications of Analytical Quality Control Data

SRK analysed the analytical quality control data accumulated by Guyana Goldfields for the period from May 1, 2011 to May 31, 2012.

Guyana Goldfields provided SRK with external analytical control data containing the assay results for the quality control samples for the Aurora Gold Mine. All data was provided in Microsoft Excel format.

SRK aggregated the assay results of the external analytical control samples for further analysis. Control samples (blanks and standards) were summarized on time series plots to highlight the performance of the control samples. Paired data (field duplicates) were analyzed using bias charts, quantile-quantile, and relative precision plots.

The external analytical quality control data produced for the Aurora Gold Mine are summarized in Table 11.1. The external quality control data produced on this project represents approximately 10.90% of the total number of samples.

Table 11.1: Summary of Analytical Quality Control Data Produced By Guyana Goldfields on the Aurora Gold Mine.

Description	Total	(%)	Comment
Sample Count	27,688		
Blanks	1,017	**3.67%**	
Blank	827		Coarse gravel
CDN-BL-4	113		CDN Resources (<0.01 ppm Au)
CDN-BL-7	77		CDN Resources (<0.01 ppm Au)
Standards	1,014	**3.66%**	
CDN-GS-P7B	304		CDN Resources (0.71 ppm Au)
CDN-GS-P7E	38		CDN Resources (0.766 ppm Au)
CDN-GS-2J	17		CDN Resources (2.36 ppm Au)
CDN-GS-3G	197		CDN Resources (2.59 ppm Au)
CDN-GS-5F	395		CDN Resources (5.3 ppm Au)
CDN-GS-10C	2		CDN Resources (9.71 ppm Au)
CDN-GS-11A	61		CDN Resources (11.21 ppm Au)
Field Duplicates	988	**3.57%**	Quarter core
Total QC Samples	**3,019**	**10.90%**	
Check Assays		**0.00%**	

A number of field blank samples did not return values below detection limit at Acme Labs; assuming a threshold limit of five times the detection limit less than percent of blanks failed. There are a number of blank standard samples above the recommend value of less than 0.01 ppm; approximately 10%of CDN-BL-4 and 30% of CDN-BL-7 failed. These blank failures cannot be explained by mislabelled reference materials. The field blank and standard blank charts may present evidence of sample contamination during the preparation process.

All the gold standard reference materials performed as expected within two standard deviations. There were only two failures (measured as a value exceeding two times the standard deviation of the expected value), however these are likely mislabelled control samples.

Paired (field duplicates) data examined suggest that gold grades are difficult to reproduce by fire assay. Rank half absolute difference (HARD) plots suggest that only 46.3% of sample pairs have HARD below 10%. However, this trend is not uncommon in gold deposits with highly variable grades.

In the opinion of SRK, the results of the analytical quality control data received from Acme Labs from May 31, 2011 to May 31, 2012 are sufficiently reliable for the purpose of resource estimation.

12 Mineral Processing and Metallurgical Testing

This section provides a review of metallurgical studies completed for the Aurora Gold Mine by Lakefield, Ontario-based SGS Mineral Services (SGS) and Resource Development Inc. (RDi). This work was conducted in phases over the period from 2006 - 2013. Outcomes from the relevant metallurgical testwork were included in a new mine plan and process facility expansion design developed by JDS Engineering (Discussed in Section 16) to form the basis of this feasibility study update.

12.1 Metallurgical Process

The Company initiated operations at its Aurora Gold Mine during October 2015. Ore containing about 3.0 g/t Au is processed at the design feed rate of 5,000 tpd through a cyanidation plant that includes primary crushing, single-stage semi-autogenous grinding (SAG) in closed circuit with cyclones to produce a final grind size of 80% passing 75 µm, gravity concentration of coarse gold in a Knelson centrifugal concentrator and a hybrid cyanidation circuit that includes both conventional agitated cyanide leaching and carbon-in-leach (CIL) cyanidation. Solubilized gold is adsorbed onto activated carbon, which is then processed through a carbon elution and electrowinning circuit to ultimately produce a final gold/silver doré product.

Tailings from the cyanidation circuit are detoxified in an industry-standard SO_2/Air cyanide destruction circuit to reduce the weak acid dissociable cyanide (CN_{WAD}) concentration prior to release into the tailings storage facility (TSF). Water is reclaimed from the tailings pond on an as needed basis.

During the period from January-August 2016, the process plant averaged 4,945 tpd at an average grade of 2.72 g/t Au from an ore blend that averaged 86% fresh rock and 14% saprolite. Gold recovery averaged 90.3%.

The Company is planning to expand the capacity of the process plant to 8,000 tpd. Key features of this expansion will be the addition of a ball mill to the grinding circuit, expansion of the gravity circuit, addition of a grinding control thickening, expansion of the leach circuit upgrades to the carbon handling and management systems and upgrades to the tailings detoxification circuit.

12.2 Testwork Program History

Metallurgical studies in support of the Aurora Gold Mine were conducted over the period from 2006 - 2013. Scoping-level metallurgical studies were completed in 2006 and 2008. The Phase 1 metallurgical program was completed in March 2006 and was reported under SGS Lakefield Project 11198-001. The Phase 2 program was completed between December 2008 and April 2009 and was reported under SGS Lakefield Project 12088-001.

Prefeasibility level metallurgical programs were completed in 2009 and reported under SGS Lakefield Project 12088-002. Feasibility level metallurgical programs were conducted between April 2009 and July 2010 and reported under SGS Lakefield Project 12088-005.

Post-feasibility metallurgical studies were conducted by Resource Development, Inc. (RDi) to evaluate process requirements on four saprolite samples. The results of this work are reported in RDi's report, "Metallurgical Testing of Aurora Saprolite Samples", April 3, 2013.

The metallurgical programs included analytical characterization, mineralogy, comminution studies, gravity separation, cyanidation leach studies, solid-liquid separation and rheological studies and cyanide detoxification testwork. Table 12.1 lists the reports completed to date and used for the feasibility study.

Table 12.1: List of metallurgical reports

Laboratory	Report Name	Issue Date
SGS Lakefield Minerals Services, Lakefield, Ontario	An investigation into the Recovery of Gold Project 11198-001 – Report 1	28-Mar-06
SGS Lakefield Minerals Services, Lakefield, Ontario	A pre-feasibility investigation into The Solid-Liquid Separation and Rheology of the Aurora Project Project 12088-002 – Final Report	10-Sep-09
SGS Lakefield Minerals Services, Lakefield, Ontario	An investigation into The Characterization of Samples from the Aurora Project Project 12088-002 – Final Report	14-Sep-09
SGS Lakefield Minerals Services, Lakefield, Ontario	An investigation into The Recovery of Gold from the Aurora Project Samples Project 12088-001 Final Report	24-Nov-09
Contract Support Services Red Bluff, California	Drop Weight Test Report on Three Samples from Aurora	01-Apr-10
Comminution Dynamics Lab-McGill University Montreal, Quebec	Aurora Project Grinding Media Wear	05-May-10
SGS Lakefield Minerals Services, Lakefield, Ontario	An investigation into Grindability Testing of Samples from the Aurora Project Project 12088-005 – Grindability Report	26-Jul-10
SGS Lakefield Minerals Services, Lakefield, Ontario	An investigation of The Characterization of Samples from the Aurora Project – Feasibility Phase Project 12088-005 – Final Report #1	23-Nov-10
SGS Lakefield Minerals Services, Lakefield, Ontario	An investigation into the Liquid-Solid Separation Response of the Aurora Project Samples – Feasibility Phase Project 12088-005 – Report #2	02-Dec-10

12.2.1 Feasibility Study Sample Selection

A total of six HQ boreholes were requested by AMEC in 2009 for the April 9, 2012 feasibility study. The location and the details of the drill holes are presented in Table 12.2 and Figure 12.1.

Particular focus was made on material likely to be mined in the first five years. The location, azimuth, and dip of the boreholes were considered to maximize the number of variability composite samples that could be produced and to also provide an even spatial representation of the different areas of the deposit.

Table 12.2: Boreholes co-ordinates

Borehole	Drill Hole Co-ordinates		Relative Level
	Northing	Easting	(m)
MET-AH-1	750963.494	195915.656	103.621
MET-AH-2	750870.212	195912.516	98.115
MET-EW-1	751540.384	196623.556	55.353
MET-RK-1	751556.557	196796.738	54.604
MET-MK-1	751128.106	196412.323	61.397
MET-MK-2	751103.917	196369.916	59.691
MET-MK-2A	751105.223	196369.456	59.587

Source: SRK, 2017

Figure 12.1: Feasibility Study Boreholes Locations and Details

12.2.2 Head Analyses

Head assays for gold, silver, total sulphur, and sulphide sulphur for the feasibility study metallurgical composites prepared for the SGS metallurgical program (Project 12088-005) are summarized in Table 12.3.

Table 12.3: Head analysis feasibility study metallurgical composites

Composite	Sample Number	Calc. Head Grade Au, g/t	Ag g/t	S_T %	$S^=$ %
Rory's Knoll Saprolite	1	0.47	-	0.02	< 0.05
Rory's Knoll Saprolite 1	-	0.02	-	0.14	< 0.05
Rory's Knoll Upper Volcanics	2	0.26	-	0.18	0.11
Rory's Knoll Upper Volcanics 1	-	0.73	< 0.5	0.1	< 0.05
Rory's Knoll Sericite Chlorite Schist	3	2.01	-	0.39	0.23
Rory's Knoll Tonalite	4	0.08	-	0.16	0.12
Rory's Knoll Quartz Vein	5	2.11	-	0.67	0.56
Rory's Knoll Lower Volcanics	6	0.27	-	0.48	0.26
Mad Kiss Upper Volcanics	7	0.02	-	0.05	< 0.05
Mad Kiss Quartz Feldspar Porphyry	8	3.35	< 0.5	0.79	0.58
Mad Kiss Quartz Felsic Tuff	9	0.03	-	0.06	< 0.05
Mad Kiss Quartz Vein	10	1.34	-	0.54	0.34
Mad Kiss Diorite	11	1.51	-	0.47	0.31
Aleck Hill Saprolite	12	0.67	< 0.5	0.38	< 0.05
Aleck Hill Saprolite 2	-	2.13	< 0.5	0.03	< 0.05
Aleck Hill Saprolite 3	-	0.06	-	0.02	< 0.05
Aleck Hill Upper Volcanics	13	12.4	-	1.15	0.99
Aleck Hill Lower Volcanics	14	0.68	-	0.26	0.17
Aleck Hill Quartz Vein	15	0.3	-	0.12	0.1

Source: SRK, 2017

12.3 Testwork Review

SRK reviewed the results from the various test programs and has identified the relevant test results that were used as the basis for development of the initial process plant design criteria and process flowsheets, as well as revisions to the process design criteria and flowsheets proposed for the process plant expansion. Details related to the existing process plant and proposed plant expansion are presented in Section 16.

12.3.1 Mineralogy

In general, the gold mineralization occurs in the vein systems, whose characteristic mineralization and lithology for the four zones (in the wireframe model) is described in Table 12.4. A mineralogical examination on both Saprolite and Rock ore composites was completed by SGS in 2009 (Project 12088-001). The results of this evaluation are summarized in Table 12.5.

Table 12.4: Lithology and Mineralization by Ore Zone, Aurora Gold Project (Mineral Resource Evaluation, SRK)

Deposit	Lithology and Mineralization
Rory's Knoll	Auriferous veins in highly altered porphyritic diorite, intensely carbonated, albite altered (tonalite). Abundant gold-rich quartz-ankerite veins in tonalite near sheared zone on western contact of tonalite with barren mafic schists. Vein density increases towards contact. Walcott Hill East, auriferous quartz-carbonate veins in silica-rich unit at mafic contact on eastern zone; weak stockwork structure.
Aleck Hill	Auriferous quartz-carbonate veins in shear zone along lithological contact with weakly altered volcanic mafic rocks. Quartz alteration (pink silica) in veins, pervasive replacement in strong stockwork zones. At Aleck Hill North, high strain and alteration at contact between diorite and mafic volcanic rocks.
Walcott Hill	Gold-bearing quartz veins.
Mad Kiss	Quartz-carbonate veining inside sheared quartz-feldspar porphyry dike enclosed in muscovite-rich rock. Stockwork associate with hanging wall and footwall contacts of the porphyry dike. Mad Kiss West, quartz-ankerite gold veins in alteration zone along hanging wall contact between unaltered mafic volcanic and diorite; stockwork in strongly altered diorite rock.

Table 12.5: Crystalline mineral assemblage phases of the saprolite and rock composite samples (12088-001)

	Saprolite Ore Composite Sample	Rock Ore Composite Sample
Major	Quartz	Quartz
Moderate	Mica, plagioclase	Plagioclase
Minor	Kaolinite, potassium feldspar	Dolomite, mica, siderite, potassium feldspar
Trace	Hematite, goethite	Calcite, chlorite, pyrite

12.3.2 Comminution Characteristics

Three series of comminution testing were undertaken to determine the grinding characteristics of the various ore types and lithologies. The first series of comminution tests was conducted at SGS as part of Project 12088-001 to determine Bond Ball Mill Work index (BWi) for the Saprolite ore and Rock ore master composites (100 mesh closing screen). The results are shown in Table 12.6, where a BWi of 14.2 kWh/t was reported for the fresh rock master composite and a BWi of 7.0 kWh/t was reported for the saprolite master composite.

Table 12.6: Bond ball mill grindability test summary (12088-001)

Sample Name	Mesh of Grind	F_{80} (mm)	P_{80} (mm)	Gram per Revolution	BWi (kWh/t)	Hardness Percentile
Saprolite Comp	100	1269	78	3.2	7	1.1
Fresh Rock Comp	100	2129	116	1.69	14.2	47

Source: SGS, 2009 - SGS Project 12088-001

The second series of comminution tests was conducted by SGS as part of Project 12088-002 and included BWi, SAG mill comminution (SMC) and abrasion (Ai) testwork on tonalite, felsic and quartz lithology samples from the Rory's Knoll, Mad Kiss and Aleck Hill deposits. In addition, comminution testing was conducted on a master composite formulated from the three lithologies (55% tonalite, 30% felsic and 15% quartz). The results of this work are summarized in Table 12.7. The BWi tests were conducted with a 90 µm closing screening and resulted in ball mill work indices ranging from 12.7 to 13.9 kWh/t. The SMC Axb test results ranged from 31.9 to 41.9 (smaller Axb values indicate harder ore). Overall, the samples fell into the medium to hard range of hardness, with the felsic sample being the softest. Abrasion indices indicated that the ore samples would be moderate to highly abrasive.

Table 12.7: Comminution test summary (12088-002)

Sample Name	Relative Density	JK Parameters			BW_i	Ai
		Axb	DWi	t_a	(kWh/t)	(g)
Master Comp	2.76	35.3	7.9	0.33	13.9	0.346
Tonalite	2.75	35.7	7.7	0.34	14.3	0.439
Felsic	2.81	41.9	6.7	0.39	12.7	0.258
Quartz	2.82	31.9	8.9	0.29	12.9	0.351

Source: SGS, 2009 - Project 12088-002

Overall, the samples fell into the medium to hard range of hardness, with the Felsic sample being the softest. Overall, the four samples showed moderate to high abrasion indices.

The third series of comminution testing was conducted as part of the feasibility metallurgical program. The main suite of comminution characterization testing was done on composites from the Rory's Knoll, Mad Kiss and Aleck Hill deposits. A smaller set of testing was done on individual lithology samples. The results of this comminution test program are summarized in

Table 12.8. The Rory's Knoll and Mad Kiss composites had similar hardness characteristics, with indices falling in the medium to hard categories, while the Aleck Hill ore was slightly softer. The Bond abrasion indices generally fell in the medium range.

Table 12.8: Comminution test summary (12088-005)

Sample Name	Relative Density	JK Parameters			MacPherson Test		AW_i	CWI	RW_i	BW_i (kWh/t)		A_i
		Axb[1]	Axb[2]	t_a	(kg/h)	(kWh/t)	(kWh/t)	(kWh/t)	(kWh/t)	170M	200M	(g)
Rory's Knoll Comp	2.81	31.2	35.6	0.38	9	8.8	14.8	15.5	-	14.6	-	-
S1-RK Saprolite	-	-	-	-	-	-	-	-	-	-	-	-
S2-RK Upper Volcanics	2.74	-	37.1	0.35	-	-	-	-	16.1	12.9	-	-
S3-RK Sericite Chlorite Sc.	2.86	-	36.5	0.33	-	-	-	-	-	13.6	-	0.16
S4-RK Tonalite	2.84	-	41.9	0.38	-	-	-	-	15.8	12.5	13.2	0.148
S5-RK Quartz Vein	2.76	-	39.8	0.37	-	-	-	-	14.7	13.9	13.7	0.348
S6-RK Lower Volcanics	2.78	-	36.1	0.34	-	-	-	-	-	12.6	-	-
Mad Kiss Comp	2.85	32.3	32.3	0.39	8.6	9.3	14.6	16.8	-	14.5	-	-
S7-MK Upper Volcanics	-	-	-	-	-	-	-	-	-	15.2	-	-
S8-MK Quartz Felspar Por.	2.78	-	46.3	0.43	-	-	-	-	-	16.7	-	-
S9-MK Quartz Felsic Tuff	2.81	-	43.9	0.4	-	-	-	-	-	12.3	-	-
S10-MK Quartz Vein	-	-	-	-	-	-	-	-	-	14.5	-	-
S11-MK Quartz Diorite	2.93	-	45.8	0.41	-	-	-	-	-	12.3	-	-
Aleck Hill Comp	2.85	52.4	50.8	0.66	11.8	6.7	12.7	7.9	-	12	-	-
S12-AH Saprolite	-	-	-	-	-	-	-	-	-	-	-	-
S13-AH Upper Volcanics	2.84	-	59.2	0.54	-	-	-	-	12.8	10.7	-	-
S14-AH Lower Volcanics	2.78	-	51	0.47	-	-	-	-	-	10.9	-	-
S15-AH Quartz Vein	2.77	-	37.9	0.35	-	-	-	-	14.8	1.7	-	0.162
Diorite	-	-	-	-	-	-	-	-	-	-	-	0.142
Quartz Vein	-	-	-	-	-	-	-	-	-	-	-	0.209
Felsic Tuff	-	-	-	-	-	-	-	-	-	-	-	0.119
Upper Volcanics	-	-	-	-	-	-	-	-	-	-	-	0.222
Lower Volcanics	-	-	-	-	-	-	-	-	-	-	-	0.062

Source: SGS, 2010 - SGS Project 12088-005
[1] Axb from DWT

Additional abrasion testwork was conducted on samples of different ore lithologies at the McGill University Comminution Dynamics Laboratory in order to predict the wear of the grinding media. The results are presented in Table 12.9 and showed that for both the ball milling and the previous SAG milling configurations used in the analysis, the quartz vein ore lithology resulted in the greatest predicted overall wear compared to the other ore types. At a throughput rate of 8,000 tpd, the predicted ball wear rate in the SAG mill ranged from 22 – 73 kg/h for the different ore lithologies and the predicted wear rate for 50 mm balls in the ball mill ranged from 99 – 200 kg/h..

Table 12.9: Mill model grinding media wear prediction

Ore Sample	Throughput	Predicted Wear Rate (kg/h)		
		SAG	Ball mill with 25 mm top size	Ball mill with 50 mm top size
Quartz Vein	8000	73	337	200
Felsic Tuff	8000	34	195	111
Lower Volcanics	8000	22	186	99
Upper Volcanics	8000	32	269	144
Diorite	8000	28	196	107

12.3.3 Gravity Concentration

Three series of gravity concentration tests were conducted to evaluate a gravity recovery circuit and the effect of grind size on gravity gold recovery. The first series of gravity concentration tests was conducted by SGS as part of project 12088-001. A gravity recoverable gold (GRG) test was conducted at different grind sizes on both the oxide and fresh rock composites. The GRG values were 51% and 73% gold recovery, respectively, indicating the presence of coarse liberated gold.

Since the GRG tests were positive, a gravity separation test was conducted, using a Knelson MD-3 concentrator as the primary gold recovery unit, followed by upgrading on a Mozley separator. The results are presented in Table 12.10. At the coarse grind size of P_{80} 155 - 190 µm gold recovery into the gravity concentrate averaged about 33%. At the finer grind size of P_{80} 73 - 86 µm gold recovery into the gravity concentrate was higher and averaged about 53%.

Table 12.10: Gravity separation test results (12088-001)

Composite	Test Number	Head Direct	Feed Size K_{80}	Recovery	Gravity Tail
		(g/t Au)	(µm)	(% Au)	(g/t Au)
Saprolite	G1	3.18	190	34.7	2.04
	G3	3.18	73	53.1	1.74
Fresh Rock	G2	5.55	155	32.1	2.93
	G4	5.55	86	52.2	2.19

Source: SGS, 2009 - SGS Project 12088-001

The second series of gravity concentration tests was conducted by SGS as part of Project 12088-002 on the master composite at a grind size of P_{80} 120 µm to produce tailings for downstream cyanidation testwork. The test was conducted with a Knelson MD-3 centrifugal concentrator to produce a rougher gravity concentrate which was further upgraded with a Mozley separator. The results of this test are summarized in Table 12.11, , and resulted in 50.4% gold recovery into the final gravity concentrate representing 0.18 Wt% and containing 1,212 g/t Au.

Table 12.11: Gravity separation test result (12088-002)

Sample	Test Number	Head Direct	Feed Size K_{80}	Recovery	Gravity tail
		(g/t Au)	(µm)	(% Au)	(g/t Au)
Master composite	G-1	5.28	128	50.4	2.16

The third series of gravity concentration tests was conducted by SGS as part of the feasibility study metallurgical program (Project 12088-005). These tests were conducted at grind sizes of about P_{80} 130 - 150 µm with a Knelson MD-3 concentrator, followed by upgrading on a Mozley mineral separator. The results of these tests are summarized in Table 12.12. Gold recovery ranged from 8% for Aleck Hill saprolite to 70% for Mad Kiss quartz feldspar porphyry.

Table 12.12: Gravity separation test results (12088-005)

Test No.	Sample Name	Head Calc. (g/t Au)	Feed Size K_{80} (µm)	Recovery (% Au)	Tailings (g/t Au)	Concentrate (g/t Au)
G1	30% Aleck Hill Saprolite 2/70% Rory's Knoll Quartz Vein Blend	1.53	138	32	1.04	246
G2	30% Aleck Hill Saprolite 2/70% Rory's Knoll Upper Volcanics 1 Blend	0.69	135	24.3	0.52	274
G3	30% Aleck Hill Saprolite 2/70% Rory's Knoll Sericite Chlorite Schist Blend	1.6	132	28.8	1.14	320
G6	30% Aleck Hill Saprolite 2/70% Mad Kiss Diorite Blend	1.59	~150	24.7	1.2	301
G4	Aleck Hill Saprolite 2	1.98	~150	8.4	1.82	47.3
G5	Rory's Knoll Upper Volcanics 1	0.49	~150	38.7	0.3	250
G7	Mad Kiss Quartz Feldspar Porphyry	2.33	~150	70.4	0.69	2113

Source: SGS, 2010 - SGS Project 12088-005

Another series of gravity separation tests using only a Knelson MD-3 centrifugal concentrator (without upgrading on a Mozley Separator) was conducted to produce gravity concentrate for downstream intensive cyanidation testwork. The results are presented in Table 12.13. . Gold recovery into the gravity concentrate ranged from 18% to 54%.

Table 12.13: Gravity separation test results (12088-005 - Knelson only)

Test No.	Sample Name	Head Calc. (g/t Au)	Feed Size K_{80} (µm)	Recovery (% Au)	Tailings (g/t Au)
ILR-1	Rory's Knoll Upper Volcanics	0.73	154	53.8	0.27
ILR-2	Rory's Knoll Quartz Vein	2.11	163	44.7	1.12
ILR-3	Rory's Knoll Saprolite	0.02	124	37	<0.02
ILR-3R	Rory's Knoll Saprolite Sample 1	0.47	39	30.5	0.3
ILR-4	Rory's Knoll Saprolite 2	2.13	<38	18.8	1.36
ILR-5	30% Aleck Hill Saprolite 2/ 70% Rory's Knoll Quartz Vein Blend	--	112	51.6	0.91

12.3.4 Cyanidation Studies

12.3.4.1 Whole Ore Cyanidation

<u>SGS Lakefield</u>

Whole-ore cyanidation tests were completed by SGS for project 12088-001. The tests were performed on the Saprolite ore and Rock ore composites and each sample was tested at three different grind sizes ranging from about P_{80} 62 - 160 µm. The results of these tests are summarized in Table 12.14. For the Saprolite composite, gold extraction ranged from 95% to 97% after 48 hours of leaching. For the Rock ore composite, gold extraction ranged from 91% to 96% after 48 hours of leaching. The Rock ore composite showed greater sensitivity to grind size.

Table 12.14: Whole ore cyanidation results summary (12088-001)

Composite	Test No.	K_{80}	CN Feed Calc	Reagent Cons. (kg/t)		Extraction (% Au)			Residue
		(µm)	(g/t Au)	NaCN	CaO	8h	24h	48h	(g/t Au)
Saprolite	CN1	160	3.47	0.05	2.26	71	94	95.1	0.17
	CN2	97	4.32	0.1	2.3	51	89	96.7	0.15
	CN3	63	3.58	0.1	2.4	54	92	96.6	0.12
Fresh Rock	CN7	147	5.13	0.66	0.44	63	89	91.7	0.43
	CN8	94	4.66	0.75	0.47	71	92	94.4	0.26
	CN9	62	4.46	0.87	0.53	69	94	95.5	0.2

Source: SGS, 2009 - SGS Project 12099-001

A second series of whole-ore cyanidation test was conducted by SGS as part of the feasibility study metallurgical program (Project 12088-005). The results of these tests are summarized in

Table 12.15. Gold extraction from the fresh rock master composite ranged from 91.1% to 94.5% at a grind size of P_{80} 74 µm after 48 hours of leaching. Gold extraction from the saprolite master composite was reported at 97.6% at a grind size of P_{80} 58 µm.

Table 12.15: Summary of Whole-Ore Cyanidation Test Results (SGS Project 12088-005)

Sample	Test No.	K_{80}	CN Feed calc	Reagent Cons. (kg/t)		Extraction (% Au)
		(µm)	(g/t Au)	NaCN	CaO	48h
Fresh Rock Composite	CN1	74	4.20	0.42	0.42	94.5
Fresh Rock Composite	CN2	74	4.39	0.77	0.32	91.1
Fresh Rock – Golder Paste Tech Sample	CN4	58	0.41	0.89	0.23	92.7
Fresh Rock – Golder Paste Tech Sample	CN5	58	0.75	0.9	0.29	96.0
Fresh Rock – Golder Paste Tech Sample	CN6	58	0.69	0.97	0.31	97.1
Saprolite Composite	CN3	58	3.51	0.53	1.83	97.6

Source: SGS, 2010 - SGS Project 12099-005

Resource Development Inc. (RDI)

Resource Development Inc. (RDi) undertook a metallurgical program in 2012 to evaluate gold extraction from four saprolite composites that were formulated from selected drill core intervals. The results of this program are presented in RDi's report, "Metallurgical Testing of Aurora Saprolite Samples, Guyana, April 2013. The test program included cyanidation studies on:

- The minus 28 mesh fraction of the as-received drill core composites;
- The minus 28 mesh fraction ground to P80 150 and 100 µm; and
- Whole-ore cyanidation of the entire saprolite composites ground to P80 150 and 100 µm.

The results of this work concluded that the best overall metallurgical results were obtained by grinding the entire saprolite sample to P_{80} 150 µm. The results of whole-ore carbon-in-leach (CIL) tests conducted on each of the four saprolite composites are presented in Table 12.16

Table 12.16: Summary of whole-ore CIL cyanidation tests on saprolite composites

Composite	Calc Head	Residue	Extraction (%)	Consumption (kg/t)	
	(g/t Au)	(g/t Au)	Au	NaCN	Lime
1	2.22	0.09	95.5	0.26	10.0
2	1.58	0.06	96.3	0.26	9.3
3	2.89	0.27	90.5	0.32	11.7
4	1.68	0.09	94.7	0.26	9.0

Source: RDi, 2013

12.3.4.2 Cyanidation of Gravity Tailings

Three series of leach tests were conducted on gravity tailings to determine the effect of the grind size, the retention time, sodium cyanide concentration and the effectiveness of lead nitrate addition.

The first series of leach tests was conducted by SGS for Project 12088-001. Six tests were completed on gravity tailings produced from both the saprolite and rock master composites to determine the effect of grind size and retention time on gold extraction. The results of these leach tests are summarized in Table 12.24.

For the Saprolite Master composite, the gold extraction from the gravity tailing ranged from 90.5% to 97.0% after 48 hours of leaching as the grind size was reduced from P80 206 - 66 µm. Overall gold recovery (gravity + cyanidation) ranged from 93.8% to 98.0%. For the Fresh Rock Master composite, gold extraction ranged from 85.8% to 93.9% as the grind size was reduced from P80 136 - 59 µm. Overall gold recovery (gravity + cyanidation) ranged from 89.3% to 95.8%.

Table 12.17: Summary of Cyanidation Test Results on the Saprolite and Rock Composite Gravity Tailings

Composite	Test No.	K_{80}	CN Feed Calc.	Recovery, Gravity Conc.	Reagent Cons. (kg/t)		Extraction (% Au)			Overall Recovery
		(µm)	(g/t Au)	(Au %)	NaCN	CaO	8 h	24 h	48 h	(Au %)
Saprolite (G1)	CN4	206	2.27	34.7	0.04	2.03	75	89	90.5	93.8
	CN5	103	2.17	34.7	0.06	2.21	80	96	95.8	97.2
	CN6	66	2.15	34.7	0.16	2.08	77	96	97.0	98.0
Fresh Rock (G2)	CN10	136	2.85	32.1	0.01	0.60	69	82	85.8	89.3
	CN11	90	2.98	32.1	0.34	0.52	79	90	92.1	94.6
	CN12	59	2.72	32.1	0.59	0.54	79	93	93.9	95.8

Source: SGS, 2009 - SGS Project 12099-001

The second series of leach tests on gravity tailings was conducted by SGS for Project 12088-002 on the Master composite that was developed for this program. Nine tests were completed to evaluate the effect of the grind size, retention time and cyanide concentration. The results of these leach tests are summarized in Table 12.25. The gold extraction from the gravity tailing ranged from about 85% to 92%. Overall gold recovery (gravity + cyanidation) ranged from 92.6% to 96.0%. The optimum cyanide leach conditions for the master composite was determined to be a grind size of P80 75 µm, cyanide concentration at 0.75 g/L NaCN and a leach retention time of 24 hours. Under these conditions gold extraction from the gravity tailing was 90.9% and overall gold recovery was 95.5%.

Table 12.18: Gravity Tailings Cyanidation Results (12088-002)

Test No.	K_{80}	CN Feed Calc.	Recovery, Gravity Conc.	Retention Time	NaCN Conc.	Reagent Cons. (kg/t)		Extraction	Overall Recovery
	(µm)	(g/t Au)	(Au %)	(h)	(g/L)	NaCN	CaO	(% Au)	(Au %)
CN1	55	2.25	50.4	48	0.50	0.23	0.49	91.6	95.8
CN2	76	2.30	50.4	48	0.50	0.08	0.44	89.1	94.6
CN3	82	2.42	50.4	8	0.75	0.27	0.21	85.1	92.6
CN4	82	1.98	50.4	24	0.75	0.18	0.31	90.9	95.5
CN5	82	1.94	50.4	32	0.75	0.25	0.33	90.2	95.1
CN6	81	2.12	50.4	48	0.75	0.26	0.36	90.8	95.4
CN7	72	2.19	50.4	24	0.50	0.28	0.30	90.4	95.2
CN4	82	1.98	50.4	24	0.75	0.18	0.31	90.9	95.5
CN8	72	2.40	50.4	24	1.0	0.35	0.23	91.9	96.0
CN9	72	2.02	50.4	24	1.5	0.47	0.22	90.3	95.2

Source: SGS, 2009 - SGS Project 12099-002

The third series of leach tests was conducted by SGS as part of the feasibility study metallurgical program (Project 12088-005). Twelve cyanidation tests were conducted on blended samples to determine the effect of grind size and lead nitrate addition. Each test was run for 24 hours and the cyanide concentration was maintained at 0.75 g/L NaCN. The results of these leach tests are shown in Table 12.19. Gold extraction from the gravity tailing during cyanidation ranged from about 93% to 97%. Overall gold recovery (gravity + cyanidation) gravity ranged from about 95% to 98%. The addition of lead nitrate and the fineness of the grind did not appear to have any significant impact on the gold recovery.

Table 12.19: Summary of Gravity + Cyanidation Test Results on Composite Blends

Composite	Grind Size	Head	Au Recovery	Cyanidation Feed	Lead Nitrate	Au Extraction	Au Recovery
	P_{80} µm	(g/t Au)	Gravity (%)	(g/t Au)	(kg/t)	Cyanidation (%)	Grav + Cyan (%)
30% Aleck Hill Saprolite + 70% Rory's Knoll Quartz	54	1.53	32.0	1.13	0.0	94.3	96.1
30% Aleck Hill Saprolite + 70% Rory's Knoll Quartz	57	1.53	32.0	1.15	0.1	94.8	96.5
30% Aleck Hill Saprolite + 70% Rory's Knoll Quartz	71	1.53	32.0	1.16	0.1	93.5	95.6
30% Aleck Hill Saprolite + 70% Rory's Knoll Quartz	71	1.53	32.0	1.11	0.0	92.8	95.1
30% Aleck Hill Saprolite + 70% Rory's Knoll Upper Volcanics	49	0.69	24.3	0.61	0.0	96.7	97.5
30% Aleck Hill Saprolite + 70% Rory's Knoll Upper Volcanics	49	0.69	24.3	0.57	0.1	96.5	97.4
30% Aleck Hill Saprolite + 70% Rory's Knoll Upper Volcanics	69	0.69	24.3	0.56	0.1	95.5	96.6
30% Aleck Hill Saprolite + 70% Rory's Knoll Upper Volcanics	69	0.69	24.3	0.55	0.0	94.5	95.8
30% Aleck Hill Saprolite + 70% Rory's Knoll Sericite Schist	45	1.60	28.8	1.08	0.0	95.8	97.0
30% Aleck Hill Saprolite + 70% Rory's Knoll Sericite Schist	68	1.60	28.8	1.18	0.1	94.5	96.1
30% Aleck Hill Saprolite + 70% Rory's Knoll Sericite Schist	63	1.60	28.8	1.11	0.1	93.7	95.5
30% Aleck Hill Saprolite + 70% Rory's Knoll Sericite Schist	63	1.60	28.8	1.13	0.0	93.8	95.6

Source: SGS, 2010 - SGS Project 12099-005

The results of gravity + cyanidation testing on three variability samples using optimized test conditions are summarized in Table 12.20. Gold extraction from the gravity tailing ranged from 88% to 93% and appeared to be complete after 24 hours. Overall gold recovery ranged from about 93% to 98%.

Table 12.20: Summary of Gravity + Cyanidation Test Results on Variability Composites

Composite	Grind Size	Head	Au Recovery	Cyanidation Feed	NaCN	Au Extraction	Au Recovery
	P80 µm	(g/t Au)	Gravity (%)	Au (g/t)	(kg/t)	Cyanidation (%)	Grav + Cyan (%)
Rory's Knoll Upper Volcanics	61	0.49	38.7	0.30	0.75	88.3	92.8
Mad Kiss Quartz Feldspar Porphyry	53	2.33	70.4	0.69	0.75	92.7	97.8
30% Aleck Hill Saprolite + 70% Mad Kiss Diorite	76	1.53	24.7	1.20	0.75	90.0	92.5

Source: SGS, 2010 - SGS Project 12099-005

12.3.4.3 Cyanidation of Gravity Concentrates

Intensive cyanidation testwork was conducted on rougher gravity concentrates produced by SGS with a Knelson gravity concentrator. The results of these tests are summarized in Table 12.21 Gold extraction by intensive cyanidation ranged from 88% to 98%, and averaged almost 93% after 48 hours of leaching. Cyanide consumption was high, most likely due to the high peroxide additions required to maintain the oxygen levels above 20 ppm.

Table 12.21: Summary of Intensive Cyanide Leach Results on Knelson Gravity Concentrates

Sample	Gravity Concentrate	Gravity Concentrate		Leach Residue	Intensive Leach	NaCN Cons.
	Au Recovery, %	(g/t Au)	Wt (%)	(g/t Au)	Au Extraction, %	Kg/t conc.
Rory's Knoll Upper Volcanics	53.8	33.5	0.91	4.01	88.0	321
Rory's Knoll Quartz Vein	44.7	52.7	1.68	3.46	93.4	161
Rory's Knoll Saprolite (Sample 1)	30.5	20.0	0.65	0.40	98.0	146
Aleck Hill Saprolite 2	18.8	27.3	1.14	0.96	96.5	14
Aleck Hill Saprolite 2 + Rory Knoll Quartz Vein	51.6	96.3	0.99	11.0	88.6	111
Average	**39.9**				**92.9**	151

Source: SGS, Report 12088-005

12.3.4.4 Leaching and Adsorption Kinetics

Leaching and gold cyanide adsorption kinetic tests were conducted by SGS for project 12088-002 to investigate the effect of pulp density. The results of these tests are presented in Table 12.22 and Source: Ausenco, 2012

Figure 12.2. Adsorption kinetic tests were performed on a pulp generated from a bulk cyanide leach. Results of the test are presented in Table 12.23 and Figure 12.3. Results from these tests indicated that pulp density (in the range of 45% to 55% solids) has no effect on either the kinetics of leaching or the gold cyanide adsorption.

Table 12.22: Gold extraction kinetics at selected slurry densities

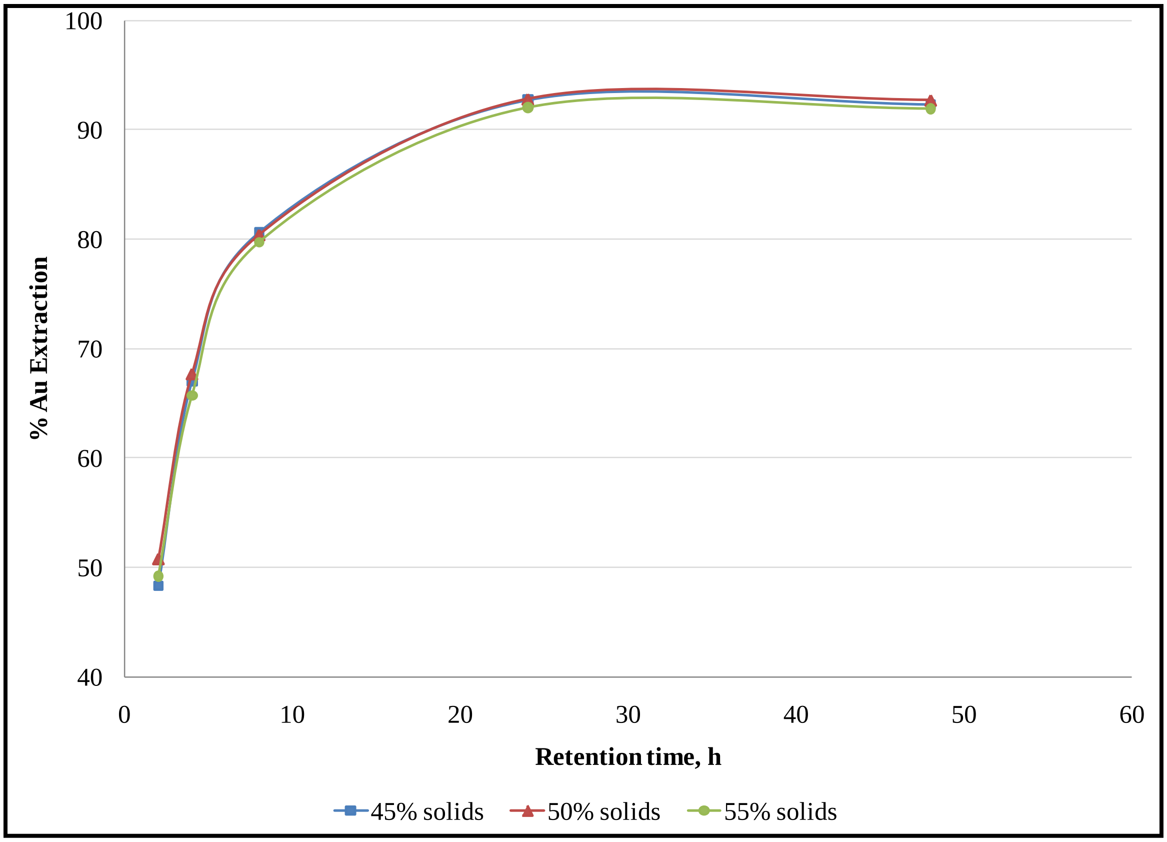
Pulp Density (% solids)	Leach Time	Extraction Au	Solution	Residue
	(h)	(%)	(Au, mg/L)	Au, g/t
45	2	48.3	0.99	1.31
	4	67	1.38	0.81
	8	80.6	1.67	0.49
	24	92.7	1.93	0.185
	48	92.3	1.93	0.196
50	2	50.8	1.3	1.28
	4	67.7	1.74	0.84
	8	80.4	2.08	0.51
	24	92.8	2.41	0.187
	48	92.7	2.42	0.19
55	2	49.2	1.55	1.31
	4	65.7	2.08	0.88
	8	79.7	2.54	0.52
	24	92	2.95	0.206
	48	91.9	2.97	0.209

Source: SGS, 2009 (Project 12088-002)

Table 12.23: Gold adsorption onto carbon kinetic summary

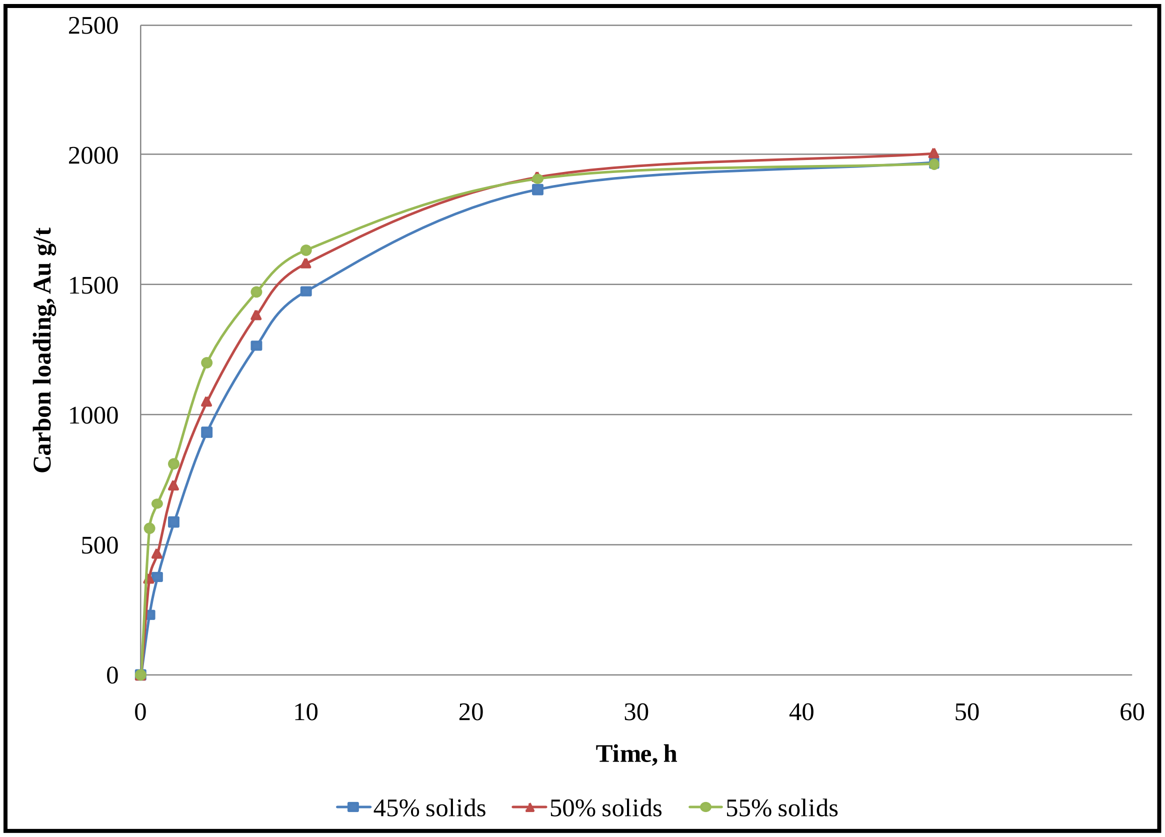
Pulp Density (% solids)	Leach Time	Solution	Loading
	(h)	(Au, mg/L)	Au, g/t
45	0	1.93	0
	0.5	1.73	229
	1	1.6	376
	2	1.41	586
	4	1.09	934
	7	0.78	1265
	10	0.58	1474
	24	0.2	1865
	48	0.1	1969
50	0	2.42	0
	0.5	2.02	372
	1	1.92	468
	2	1.62	730
	4	1.25	1052
	7	0.86	1384
	10	0.62	1583
	24	0.21	1915
	48	0.1	2006
55	0	2.97	0
	0.5	2.21	564
	1	2.08	658
	2	1.86	812
	4	1.29	1201
	7	0.88	1472
	10	0.63	1633
	24	0.19	1907
	48	0.1	1965

Source: SGS, 2009 (Project 12088-002)



Source: Ausenco, 2012

Figure 12.2: Leach kinetic as a function of pulp density plot



Source: Ausenco, 2012

Figure 12.3: Adsorption of gold cyanide kinetic as a function of pulp density plot (12088-002)

12.3.5 Rheology and Settling Thickening

The first series of rheology and thickening testing was completed by SGS for project 12088-002 on Saprolite and Rock composites. The results are summarized in Table 12.24 and Table 12.25. The data indicated that the critical solid density for the master composite thickener underflow was in the range of 68% to 69% solids, at a corresponding yield stress of ~34 Pa. The critical solids density for the Saprolite composite (-20 mesh) was in the range of 54% to 55% solids at a corresponding yield stress of ~20 Pa.

Table 12.24: Settling thickening test results summary (12088-002)

Sample	Flocculent CIBA	Dosage (g/t)	Feed[1] (% wt)	U/F[2] (% wt)	TUFUA[3] (m^2/tpday)	THUA[4] (m^2/tpday)	ISR[5] (m^3/m^2/day)	Supernatant[6] Visual
Master Comp	Magnafloc 455	10	10	67	0.07	0.01	566	Clear
Saprolite Comp	Magnafloc 10	50	10	52	0.04	0.01	1639	Clear

Source: SGS, 2009 (Project 12088-002)

Table 12.25: Rheology test results summary (12088-002)

Sample	Test	Solids %	Unsheared Sample		η_P	Sheared Sample		η_P
			γ	τ_{yB}		γ	τ_{yB}	
			(Range, s^{-1})	(Pa)	(mPa.s)	(Range, s^{-1})	(Pa)	(mPa.s)
Master Comp	T1	71.8	480-600	124	96	300-600	38.9	78
	T2	69.2	360-600	34.2	85	360-600	11.6	75
	T3	66.4	480-600	15.4	35	200-400	5.9	35
	T4	62.4	300-600	3.8	21	120-300	2.2	20
Saprolite Comp - 20mesh	T5	61.7	200-240	124	13	360-600	63.3	49
	T6	57	400-600	49.8	50	240-480	27.8	37
	T7	54.1	360-600	19.9	20	240-400	13.2	26
	T8	50.4	360-600	7.5	15	240-480	4.2	18
	T9	46	120-400	2.2	9	120-360	1.7	8

Source: SGS, 2009 (Project 12088-002)

Additional rheology and thickening testwork was conducted by SGS during the feasibility study metallurgical program on three different samples representing Aleck Hill Saprolite, Rory's Knoll Upper Volcanics and a blend of 30% Aleck Hill Saprolite and 70% Rory's Knoll Upper Volcanics. The results of this work are summarized in The maximum underflow solids densities predicted based on rheologically determined critical solid density were 47% solids for the Aleck Hill Saprolite, 50% solids for the Blend and 60% solids for the Rory's Knoll composite.

Table 12.26 and Table 12.27. The maximum underflow solids densities predicted based on rheologically determined critical solid density were 47% solids for the Aleck Hill Saprolite, 50% solids for the Blend and 60% solids for the Rory's Knoll composite.

Table 12.26: Settling thickening test results summary (12088-005)

Description	Aleck Hill, Saprolite 3, -10 mesh	Blend*		Rory's Knoll
Sampling Test	24	16	25	8
Particle Size, d_{80}, µm	29	31	31	48
Pulp pH	6.5	7.4	10.5	7.2
Initial Solids, % wt	5.8	10	10	10
CIBA Magnafloc 10, g/t dry	77	103	50	45
U/F Solid Density, % wt	40	48	53	53
*Max. U/F % wt Predicted by CSD	47	50	nd	60
Thickener U/F Unit Area, m^2/tpday	0.06	0.08	0.16	0.06
**Above corrected for CSD actual UF	0.07	0.08	nd	0.07
Thickener Hydraulic Unit Area, m^2/tpday	0.01	0.01	0.03	0.01
**Above corrected for CSD actual UF	0.01	0.01	nd	0.01
Initial Settling Rate, m^3/m^2/day	1649	833	265	1567
Supernatant Clarity, 10'-60'-final	Clear	Clear	Clear	Clear

* 30% Aleck Hill Saprolite 3/70% Rory's Knoll Upper volcanics

Table 12.27: Rheology test results summary (12088-005)

Sample	Test	Solids %	Unsheared Sample		η_P	Sheared Sample		η_P
			γ	τ_{yB}		γ	τ_{yB}	
			(range, s^{-1})	(Pa)	(mPa.s)	(Range, s^{-1})	(Pa)	(mPa.s)
Rory's Knoll pH ~ 7.2	T1	63.3	200-250	131	Plug	200-400	51	63
	T2	60.5	200-400	42	42	200-400	27	41
	T3	56.3	200-400	14	28	200-400	11	23
	T4	50.8	200-400	4	15	200-400	3	13
Aleck Hill pH ~ 7.2	T5	50.7	200-400	94	58	200-400	73	56
	T6	47.8	200-400	43	45	200-400	43	34
	T7	43.6	200-400	18	26	200-400	20	21
	T8	38	200-400	6.1	14	200-400	7.8	12
Blend pH ~ 7.2	T9	52.6	100-160	82	Plug	200-400	14	51
	T10	50.9	240-400	43	19	200-400	12	40
	T11	48.1	300-400	14	25	200-400	6.2	26
	T12	43.4	200-400	2.4	16	200-400	1.7	13
Blend pH ~ 10.5	T13A	52.4	200-400	32.8	50	200-400	14.6	22

12.3.6 Cyanide Detoxification

The industry-standard SO2/Air, copper catalyzed system was used to establish test conditions required to detoxify cyanide leach residues. Both batch and continuous detoxification tests were conducted by SGS as part of Project 12088-001 and only batch detoxification tests were conducted by SGS as part of Project 12088-005. It should be noted that the results from continuous detoxification testing provide a more reliable assessment of detoxification requirements than the results from batch testwork. The results of batch and continuous tests conducted by SGS as part of Project 12088-001 are summarized in Table 12.28. In these tests, leach residues containing 260 to 350 mg/L CNWAD were successfully detoxified to <1 mg/L CNWAD. Reagent requirements included the addition of about 4 to 5 g SO_2/g CNWAD and 2.0 to 2.5 g hydrated lime/g CNWAD, with minor addition of copper (from copper sulphate). Tests were conducted at 40% solids and maintained a retention time of 60 minutes.

Table 12.28: Summary of Cyanide Detoxification Tests on Sulfide and Oxide Leach Residues

Test	Mode	Pulp Dens. %	Reactor Vol. L	Reten. Time min	Composition (Solution Phase)						Vol. of Pulp L	Cumulative Reagent Addition*			
												g/g CN_{WAD}		Cu	
					pH	CNr mg/L	CN_{WAD} CN_{picric} mg/L	Cu mg/L	Fe mg/L	Zn mg/L		SO_2 Equiv.	Lime	g/g CN_{WAD}	mg/L Sol'n
Oxide Comp. Pulp from Test CN 13					**10.6**	**356**	**350**	**8.50**	**0.37**	**4.59**					
CND 1-1	Batch	40	1.9	105	8.6		0.6					4.09	2.00	0.071	25
CND 1-2	Continuous	40	1.9	56	8.6	0.50	0.11	2.21	0.20		6.1	4.55	1.90	0.055	19
CND 1-3	Continuous	40	1.9	56	8.6	0.48	0.11	1.49	0.22		4.1	4.84	2.04	0.057	20
CND 1-4	Continuous	40	1.9	61	8.6	0.44	<0.1	3.45	0.25		8.8	5.02	2.21	0.069	24
Composite					8.6	0.50	<0.1	0.20	<0.2	<0.05	18.9	4.83	2.07	0.062	22
Sulphide Comp. - Pulp from Test CN 14					**10.2**	**426**	**260**	**8.26**	**60.3**	**16.2**					
CND 2-1	Batch	40	1.9	60	8.6		<0.1					4.06	6.95	0.60	155
CND 2-2	Continuous	40	1.9	62	8.6	0.54	0.11	0.16	0.14		4.6	4.42	2.58	0.63	163
CND 2-3	Continuous	40	1.9	61	8.6	0.53	0.11	0.15	0.14		3.8	4.13	2.19	0.60	157
CND 2-4	Continuous	40	1.9	57	8.6	0.53	0.27	0.11	0.19		10.5	3.87	2.61	0.57	147
Composite					8.5	0.37	0.21	0.50	0.40	<0.7	18.8	4.06	2.52	0.59	153

12.4 Metallurgical Recoveries

12.4.1 Recovery Estimate from Metallurgical Studies

The Aurora process plant has been designed to process gold ore at an average gold grade of 3.33 g/t Au through a flowsheet that includes grinding to a particle size of P80 75 µm, gravity concentration and cyanidation of the gravity tailing in an agitated leach circuit for 24 hours at a cyanide concentration of 0.50 to 0.75 g/L NaCN. SRK has reviewed the metallurgical testwork that has been conducted and has selected test results that are consistent with the base design criteria for the Aurora process plant. Table 12.29 shows the selected results for tests conducted on both fresh and oxide ore during the first two metallurgical programs conducted at SGS (12088-001 and 12088-002). Based on an assumption that 98% of the gold reporting to the gravity concentrate and 99% of the reporting to the pregnant leach solution (PLS) can be recovered (typical of actual plant practice). SRK estimates that about 93% of the gold in the fresh ore and about 97% of the gold in the oxide ore can be recovered. This represents about 94% gold recovery from an ore blend of 70% fresh and 30% oxide ore.

Table 12.29: Summary of Relevant Prefeasibility Metallurgical Testwork on Oxide and Fresh Ore Composites

SGS Program	Test	Calc Head	Grind	NaCN Conc	Time	Recovery %			Recovery (%)
		g/t Au	P80 µm	g/L	Hours	Gravity	Cyanidation	Grav + CN	Adjusted [1]
Fresh									
12088.001	G2:C11	4.30	90		24	32.1	90.0	93.2	92.0
12088.001	G4:CN14	4.58	86		48	52.2	88.6	94.5	93.1
12088.002	G1:CN2	4.35	76	0.50	48	50.4	89.0	94.5	93.1
12088.002	G1:CN4	4.35	82	0.75	24	50.4	90.9	95.5	94.0
12088.002	G1:CN7	4.35	72	0.50	24	50.4	90.4	95.2	93.8
Average Fresh						47.1	89.8	**94.6**	**93.2**
Oxide									
12088.001	G1:CN6	3.12	66		24	34.7	96.0	97.4	96.1
12088.001	G3:CN13	3.71	73		48	53.1	96.5	98.4	96.8
Average Oxide								**97.9**	**96.5**
Blend 70:30	Average							95.6	94.2

Source: SRK, 2017
[1] Assume 98% recovery from Gravity Concentrate and 99% recovery of soluble Au

Table 12.30 shows the results for selected tests conducted during the feasibility study metallurgical program on blends of saprolite and rock ore (30% saprolite and 70% fresh) as well as composites representing 100% fresh ore. Adjusted gold recoveries ranged from about 90% to 96.5% and averaged 93.4%. On this basis, SRK estimates achievable gold recovery from fresh ore at about 93% and achievable gold recovery from oxide (saprolite) at about 97%.

Table 12.30: Summary of relevant feasibility metallurgical testwork (SGS Report 12088-005)

Test	Blend	Calc Head	Grind	Recovery (%)	Recovery (%)	Grav + CN	Recovery (%)
		g/t Au	P80 µm	Gravity	Cyanidation		Adjusted [1]
AH Sap + RK Quartz Vein	30:70	1.53	71	32.0	92.8	95.1	93.8
AH Sap + RK Upper Vol	30:70	0.69	69	24.3	95.4	95.8	95.3
AH Sap + RK Sericite	30:70	1.60	63	28.8	93.8	94.3	94.3
AH Sap 2 + MK Diorite	30:70	1.59	76	24.7	88.0	92.5	89.8
RK Upper Volcanics	100	0.49	61	38.7	88.0	92.8	91.3
MK Quartz Feldspar	100	2.33	53	70.4	94.0	97.8	96.5
Average				36.5	92.0	94.7	93.5

Source: SGS, 2010 (Project 12088-005)
AH = Aleck Hill
RK = Rory Knoll
MK=Mad Kiss

Conditions:
Retention time: 24 hours
Cyanide Concentration: 0.75 g/L
pH: 10.5 to 11
Slurry density: 40%

[1] Assume 98% Au recovery from gravity concentrate and 99% recovery of cyanidation Au

12.4.2 Plant Performance

Actual plant performance for 2016 (January - August) is summarized in Table 12.31. During this period monthly gold recovery has ranged from 87.4% to 92.8% and has averaged 90.3%. The ore blend has averaged 86% fresh ore and 14% saprolite. Based on relevant metallurgical testwork, an overall gold recovery in the range of 93% to 94% should have been achievable. The lower gold recoveries achieved in the plant can most likely be attributed to the following issues:

- Coarser primary grind than design;
- Inadequate carbon acid washing capability;
- Insufficient carbon elutriation capacity; and
- Inadequate carbon fines management.

AGM is aware of these process deficiencies and expects to address these issues as part of the planned plant expansion described in Section 16.

Table 12.31: 2016 Plant production

Period	tpd	Quantity Tonnes of Ore	Grade g/t Au	Recovery % Au	Metal Recovered Oz Au
1Q16	5,100	462,600	3.07	89.2	41,300
2Q16	4,700	427,700	2.61	91.1	32,000
3Q16	5,300	491,200	2.42	88.7	34,400
4Q16	5,500	507,500	2.94	90.6	43,800
Total	**5,200**	**1,889,000**	**2.74**	**90.2**	**151,600**

12.5 Conclusions and Recommendations

The conclusions and recommendations drawn from the testwork program are presented below:

- Tonalite ore in Rory's Knoll and volcanics at Aleck Hill represent the major mineralized rock types in run of mine (RoM) ore feed. Other rock types represent minor components. Values for design should be weighted according to the major ore types;
- Ores are highly amenable to gold recovery by a process flowsheet that includes gravity concentration and cyanidation of both the gravity concentrate and gravity tailing. Overall gold recoveries in the range of 93% to 94% should be achievable;
- Cyanide consumption from the laboratory tests averaged 0.5 kg/t for fresh rock and is typical for a free milling ore with few deleterious cyanide consumers;
- Leaching and carbon adsorption kinetic tests indicated there should be no effect of increased pulp density (in the range of 45% to 55% solids) on either parameter;
- The ore is amenable to detoxification by the industry-standard Air/SO_2, copper catalyzed process with industry-normal reagent demand and acceptable CNWAD levels for the discharge to a conventional tailings storage facility; and
- Issues related to too coarse of a grind size and poor carbon management must be addressed during the planned plant expansion. Proper consideration of the current plant deficiencies could result in an improvement of in gold recovery of about 3% to 4%.

13 Mineral Resource Estimates

13.1 Summary

Given that no additional exploration data have been collected for the mineralized zones since the completion of mineral resource modelling in 2012, the mineral resource model developed in GEMS has not been updated for this report. SRK has, however, reviewed the relatively limited production and ore control data available from the early stages of production to date, and has concluded that the data is insufficient to inform possible changes to the resource model at this point in time. The Mineral Resource statement has been adjusted using the 2016 End-of-Year (EOY) topographic surface, to account for mining in the Rory's Knoll and Aleck Hill zones, and includes ore stockpile inventories as of December 31, 2016.

13.2 Mineral Resource Estimation Methodology

The evaluation of mineral resources for the Aurora Gold Mine involved the following procedures:

- Database compilation and verification;
- Resource modelling;
- Updating wireframe model for Rory's Knoll by SRK and importing of 3D wireframe models for the other deposits received from Guyana Goldfields;
- Extensive validation of database and the wireframe models prepared by Guyana Goldfields;
- Data processing (compositing and capping) and statistical analysis;
- Selection of estimation strategy and estimation parameters;
- Block modelling and grade interpolation;
- Validation, classification and tabulation;
- Assessment of "reasonable prospects for economic extraction" and selection of reporting cut-off grades and Preparation of Mineral Resource Statement;
- There has been no further drilling in the project area and the current resource model was not updated for this report update

13.3 Database

13.3.1 General

Data used to evaluate the mineral resource were provided by Guyana Goldfields as Microsoft Excel and drawing exchange (DXF) files.

The borehole database contains updated drilling data for the period May 2011 to April 2012. It contains 1,110 exploration boreholes (totalling 366,851 m); excluding geotechnical and metallurgical holes), with collar location, down hole survey data, geological codes, and 86,380 gold assay intervals. The mineral resource model reported herein considers 171 new boreholes drilled on the property since the previous model prepared in September 2011.

Gold mineralization wireframes generated by Guyana Goldfields (all zones with the exception of Rory's Knoll), were supplied to SRK in dxf formats. The multiple zonal wireframes were imported into GEMS software as a single solid per domain.

13.3.2 Data Validation

SRK performed following validation steps on the borehole data and checked:

Minimum and maximum values for each quality value field and confirming/editing those outside of expected ranges;

Inconsistency in lithological unit terminology and/or gaps in the lithological code; and

For gaps, overlaps and out of sequence intervals for both assays and lithology tables.

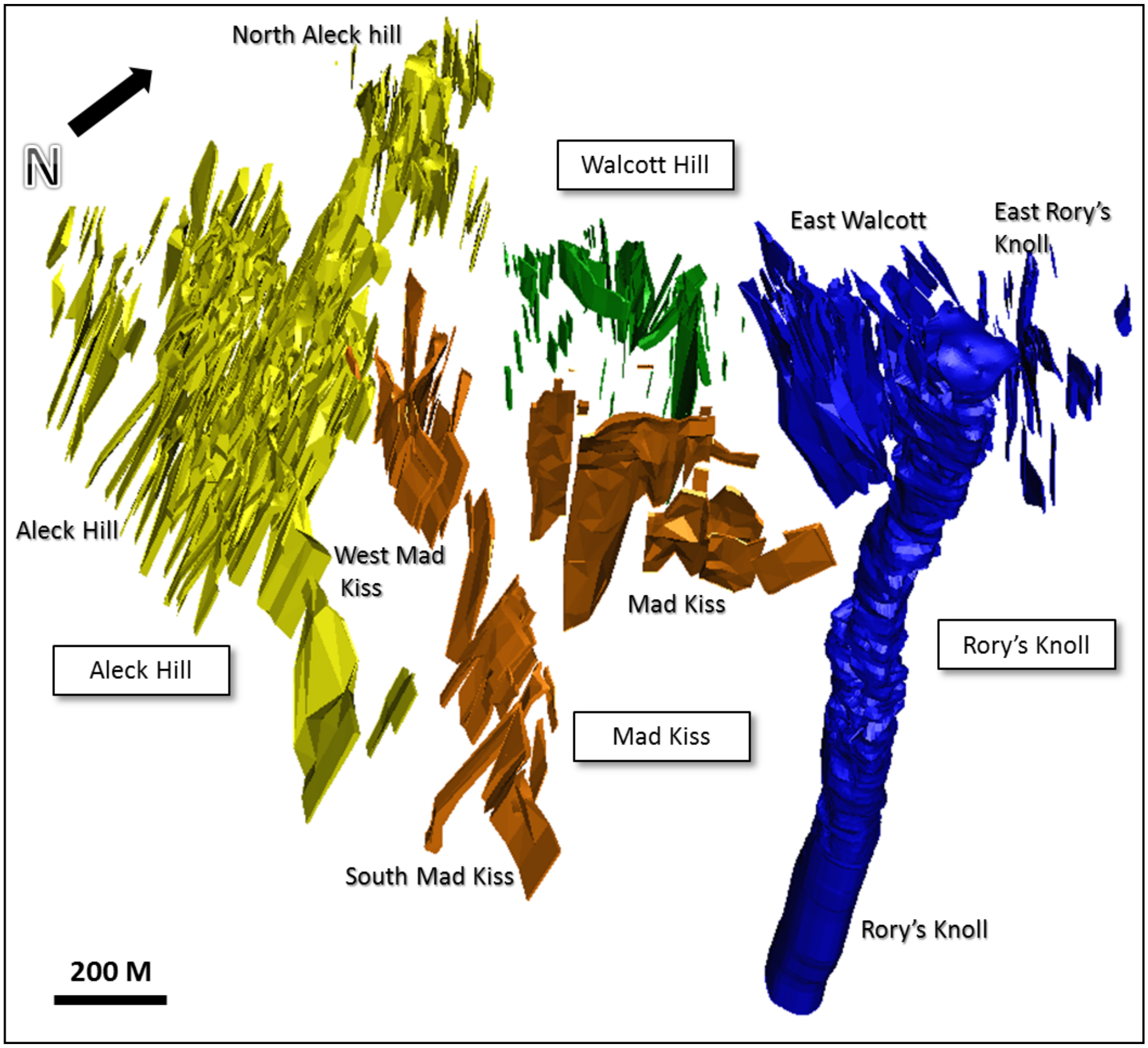
The GEMS database was found to be in good order and well maintained. On completion of the validation procedure, SRK considers the database suitable for resource estimation.

The wireframes were validated by checking for crossovers, duplicate triangles, gaps in the wireframes, and edge boundary joining. SRK accepted wireframe definitions and resource domains defined by Guyana Goldfields for all zones with the exception of Rory's Knoll. Rory's Knoll was modelled by SRK.

13.4 Solid Body Modelling

The Aurora Gold Mine is subdivided into nine distinct auriferous zones: Aleck Hill, North Aleck Hill, Rory's Knoll, East Rory's Knoll, Walcott Hill, East Walcott Hill, Mad Kiss, South Mad Kiss, and West Mad Kiss.

The nine distinct zones are grouped into four main auriferous zones as shown in Figure 13.1. The gold mineralization in Rory's Knoll, and to some extent in East Rory's Knoll and East Walcott, form "carrot" shaped outlines containing a weak to moderate stockwork of quartz-carbonate veins. The gold mineralization at Aleck Hill and Mad Kiss areas forms distinct tabular zones of quartz-carbonate veins.



Source: SRK, 2017

Figure 13.1: Oblique Section looking north showing the main auriferous zones of the Aurora Gold Mine

The Rory's Knoll domain was modelled by SRK. SRK considered the logged lithology and gold grade distribution patterns. In addition to re-defining the outline of this domain, SRK remodelled zones of internal waste located inside the Rory's Knoll domain. In this process, several previously modelled internal waste zones were removed except where demonstrated continuous based on geological data. Those internal waste zones were not considered for grade estimation.

The Rory's Knoll domain was subdivided into a high grade subdomain by constructing wireframes inside the main wireframe around areas of higher grade mineralization. The limits of the higher grade subdomain were initially modelled with Leapfrog software using a threshold of 5.0 g/t gold. Subsequently, the Leapfrog meshes were manually smoothed to define more consistent zones of higher grade gold mineralization. The resulting Rory's Knoll high grade subdomains are entirely contained within the main Rory's Knoll domain.

Additionally, SRK defined the top 10-m layer of saprolite generated using the Leapfrog shells based on the 0.2 g/t gold cut-off. The Guyana Goldfields modelled wireframes representing the vein like structures were subdivided into the fresh and saprolite domain using the bottom surfaced of the saprolite provided to SRK by Guyana Goldfields.

Structural geology investigations, geological modelling, and information from the infill drilling completed during the period of May 2011 to April 2012 support a better definition of the lateral continuity of the structures hosting the gold mineralization within all the gold zones, and consequently improve the confidence in the geological continuity of the gold mineralization.

Infill drilling completed on the other auriferous zones (Aleck Hill, Mad Kiss, and East Walcott Hill, the “satellite deposits”) prompted revision to the geological interpretation. The boundaries of the gold mineralization were revised by Guyana Goldfields to take into account the new drilling information. This infill drilling also improves the confidence in the continuity of the gold mineralization.

Each auriferous zone was assigned a rock code to facilitate identification during the interpolation process (Table 13.1). The Aurora Gold Mine block model coding is based on numeric value, with no waste model included.

Table 13.1: Rock codes in the Aurora gold project block model

Domain/Zone	Block Model Code
Rory’s Knoll	100
Rory’s Knoll High Grade	111
Rory’s Knoll HG Saprolite	112
Rory’s Knoll Saprolite	122
Aleck Hill	200
Aleck Hill Saprolite	222
North Aleck Hill	300
North Aleck Hill Saprolite	322
East Walcott Hill	400
East Walcott Hill Saprolite	422
East Rory’s Knoll	500
East Rory’s Knoll Saprolite	522
Walcott Hill	600
Walcott Hill Saprolite	622
Mad Kiss	700
Mad Kiss Saprolite	722
South Mad Kiss	800
South Mad Kiss Saprolite	822
West Mad Kiss	900
West Mad Kiss Saprolite	922
Saprolite Horizontal (10 m)	2222

13.5 Database Preparation

The domain wireframes were used to code a zone field into the block model (Table 13.2). The geological solids were coded and these values were written into the block model using the wireframe to delineate the auriferous zones. Table 12.2 illustrates the basic sample gold grade and sample length statistics for the borehole data.

Unsampled borehole intervals intersecting geological wireframes were assigned a value of 0.003 g/t. Metallurgical and geotechnical boreholes were excluded from the database prior to the estimation.

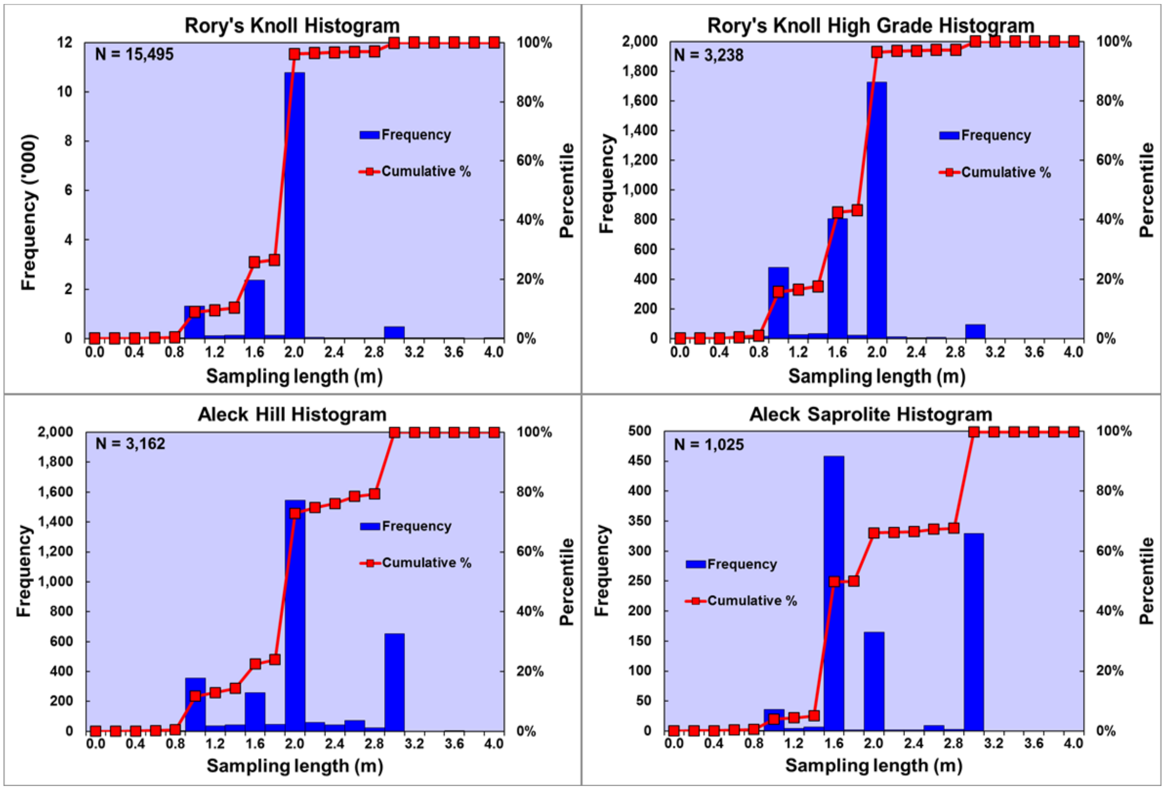
Table 13.2: Basic statistics of raw borehole samples for the Aurora gold project

Domain	Variable	Count	Minimum	Maximum	Mean	Std. Dev.	Variance	COV
Aleck Hill	Au (g/t)	4,187	0	122.44	2.82	6.86	47.12	2.44
Aleck Hill North		1,092	0	45.77	2.1	4.06	16.47	1.93
Rory's Knoll		14,360	0	532.5	2.92	8.35	69.7	2.86
Rory's Knoll HG		3,239	0	532.5	5.32	15.34	235.3	2.88
Rory's Knoll East		401	0	123.47	4.54	8.37	70.1	1.84
Walcott Hill		534	0	80.13	1.89	4.61	21.24	2.44
Walcott Hill East		2,249	0	2343.93	3.95	49.85	2484.97	12.61
Mad Kiss		688	0	157	4.24	11.28	127.25	2.66
Mad Kiss South		218	0.01	41.3	2.46	4.34	18.86	1.77
Mad Kiss West		229	0	150.2	3.37	11.23	126.06	3.33
Saprolite Horizontal		2,001	0	88.41	0.89	3.12	9.73	3.49
Aleck Hill	Length (metres)	4,187	0.15	6.5	2.05	0.64	0.41	0.31
Aleck Hill North		1,092	0.24	3.1	1.75	0.68	0.46	0.39
Rory's Knoll		14,361	0.22	4.5	1.85	0.39	0.15	0.21
Rory's Knoll HG		3,239	0.4	3	1.73	0.44	0.2	0.26
Rory's Knoll East		401	0.35	3.53	1.8	0.59	0.35	0.33
Walcott Hill		534	0.16	4.15	1.76	0.68	0.46	0.38
Walcott Hill East		2,249	0.2	4.1	1.81	0.68	0.46	0.38
Mad Kiss		688	0.5	3.3	1.9	0.5	0.25	0.26
Mad Kiss South		218	0.55	3	1.91	0.68	0.46	0.36
Mad Kiss West		229	0.6	4.5	1.94	0.69	0.47	0.35
Saprolite Horizontal		2,003	0.45	4.5	2.29	0.74	0.55	0.32

13.6 Compositing

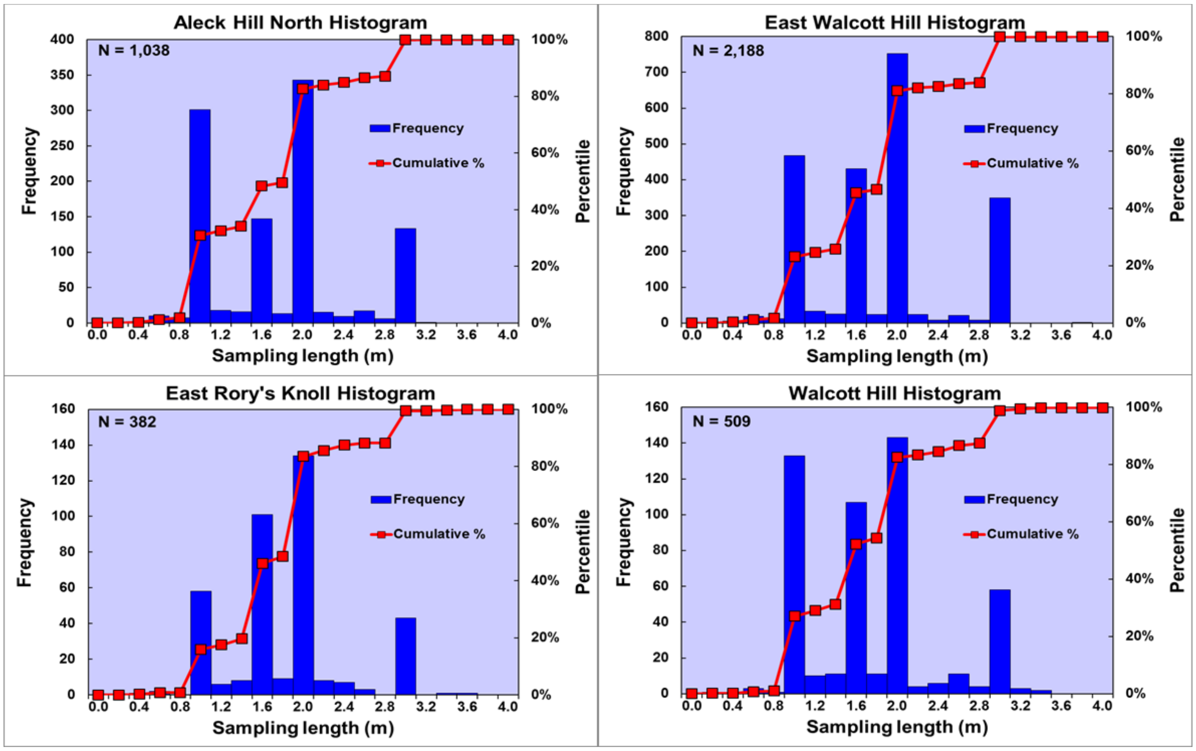
The majority of core assay samples were collected at 2 m intervals (Figure 13.2, Figure 13.3 and Figure 13.4) irrespective of geology contacts.

After a review of sample length histograms for each zone, gold assays were composited to 2.0 m in all the domains except the 10 m saprolite layer where the data was composited to 3.0 m. Composites from each domain were extracted for geostatistical analysis and variography.



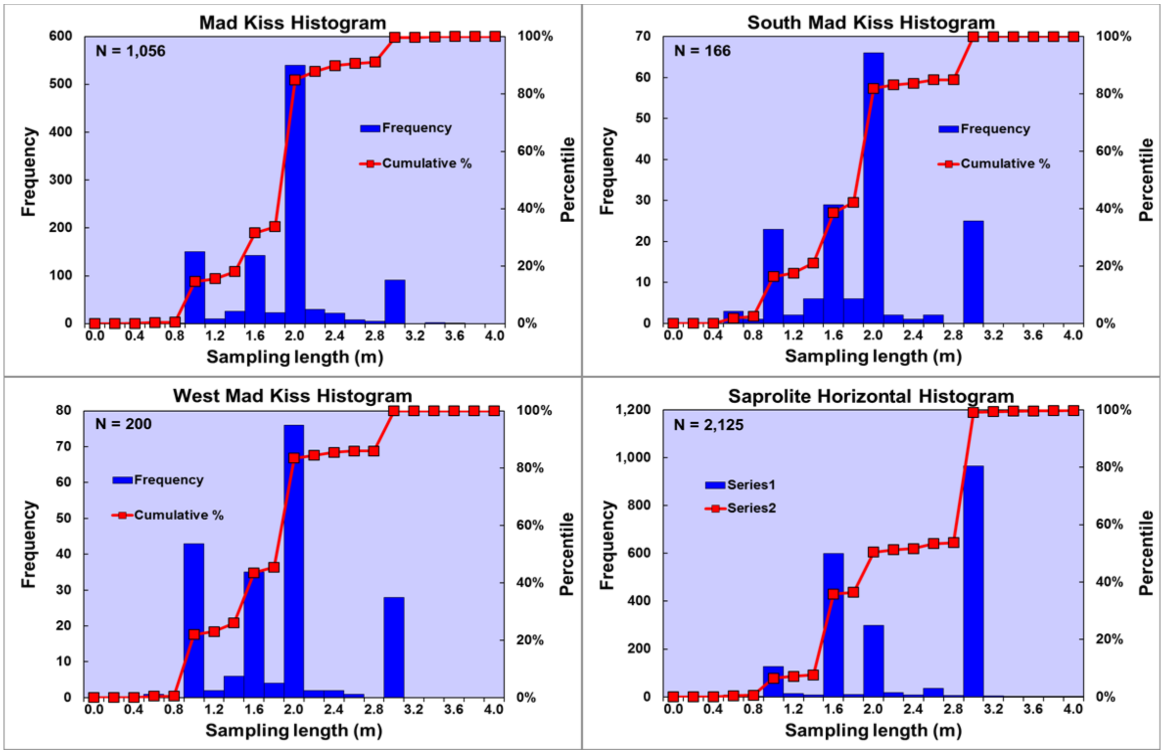
Source: SRK, 2017

Figure 13.2: Sample length histograms for Aleck Hill, Aleck Hill high grade, Rory's Knoll and Rory's Knoll high grade.



Source: SRK, 2017

Figure 13.3: Sample length histograms for Aleck Hill North, Rory's Knoll East, Walcott and Walcott East.

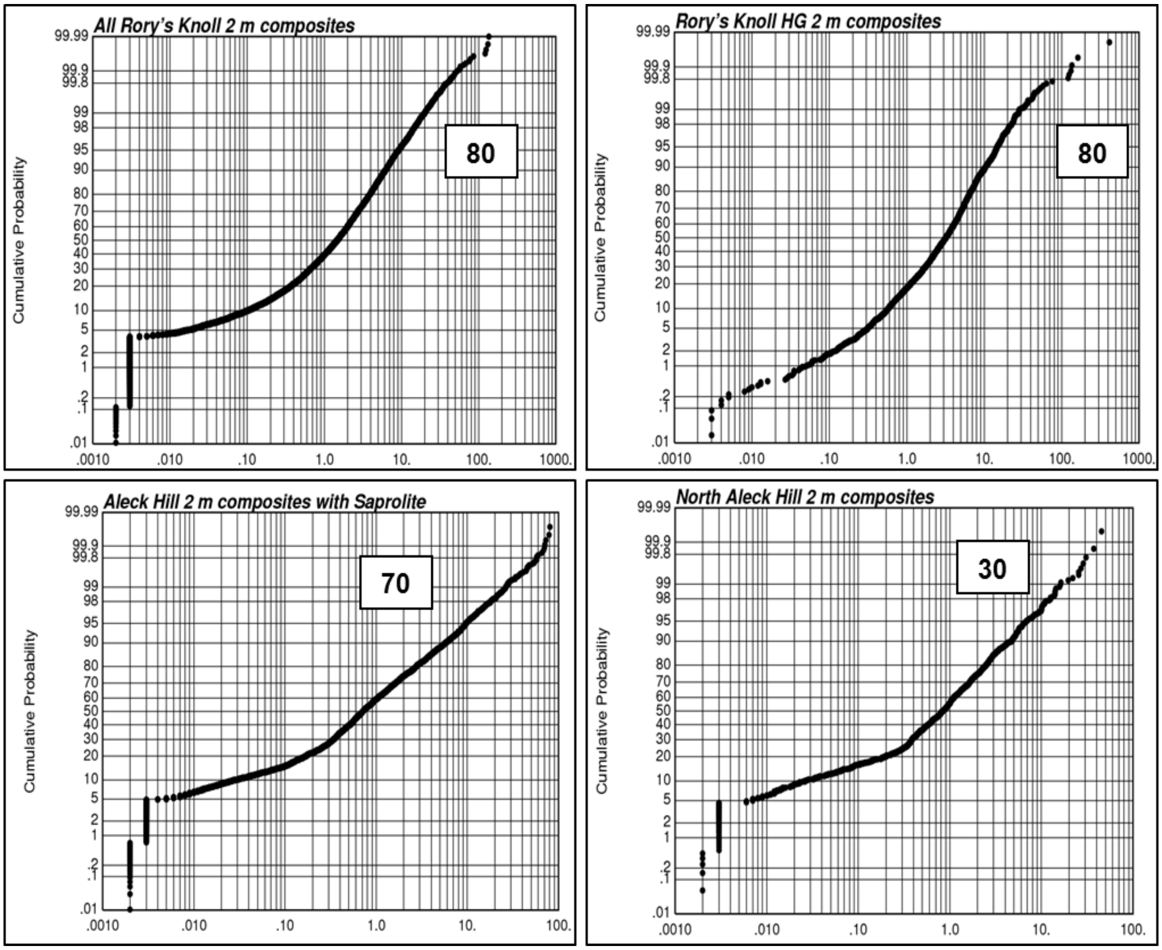


Source: SRK, 2017

Figure 13.4: Sample length histograms for Mad Kiss, Mad Kiss West and Mad Kiss South.

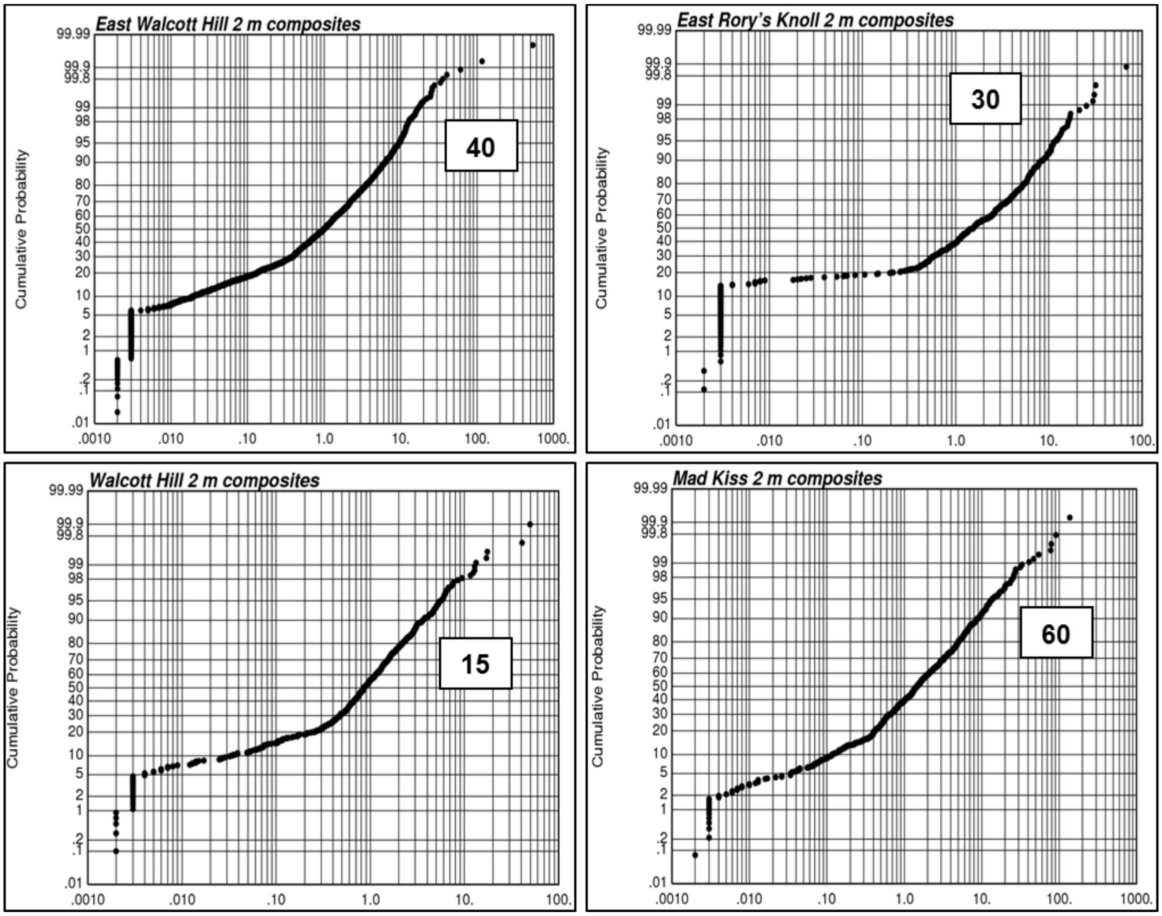
13.7 Evaluation of Outliers

For each domain, a capping value was determined by analyzing histograms and cumulative frequency plots of gold composites (Figure 13.5, Figure 13.6 and Figure 13.7). Capping values were adjusted iteratively by reference to summary statistics to ensure the robustness of statistics to chosen capping values.



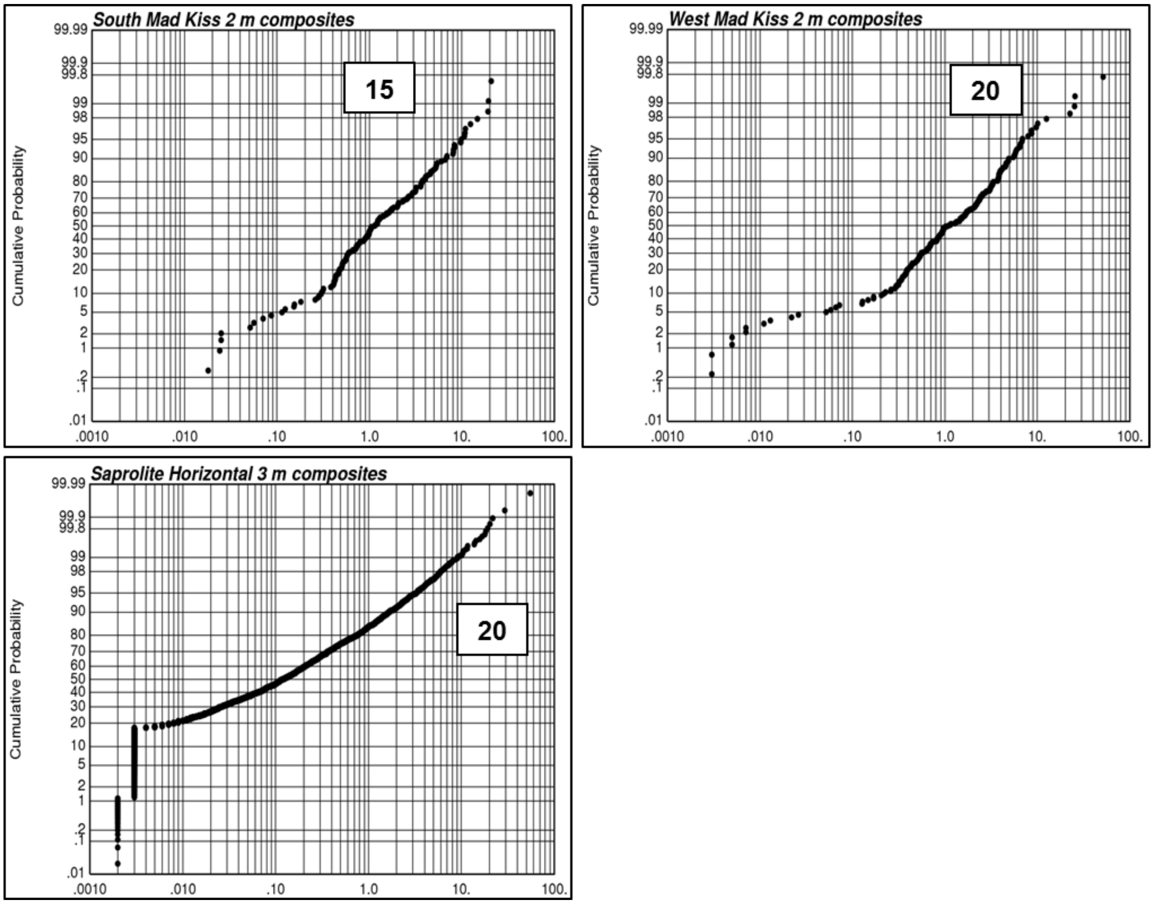
Source: SRK, 2017

Figure 13.5: Cumulative frequency plots for gold composites in Aleck Hill, Aleck Hill high grade, Rory's Knoll and Rory's Knoll high grade - capping grades as indicated



Source: SRK, 2017

Figure 13.6: Cumulative frequency plots for gold composites in Aleck Hill North, Rory's Knoll East, Rory's Knoll High Grade, Walcott and East Walcott - capping levels as indicated.



Source: SRK, 2017

Figure 13.7: Cumulative frequency plots for gold composites in Mad Kiss, Mad Kiss South and Mad Kiss West - capping levels as indicated.

13.8 Specific Gravity

Specific gravity was measured by Guyana Goldfields on 465 pieces of fresh core using a water displacement methodology (Table 13.3). An additional forty-nine measurements from saprolite were generated by AMEC. Specific gravity does not vary much between various auriferous zones. An average specific gravity of 2.80 was assigned to all fresh waste rock blocks, and 1.73 for all saprolite blocks above the saprolite wireframe. SRK recommends that more specific gravity measurements be generated for each zone as the above data was acquired from only a few drill holes.

Table 13.3: Guyana Goldfields specific gravity measurements.

Zone	Fresh			Saprolite*		
	No. Samples	SD	Mean SG	No. Samples	SD	Mean SG
Aleck Hill	32	0.07	2.78	22	0.23	1.73
Aleck Hill North	30	0.07	2.85	1	0	2.48
Rory's Knoll	251	0.05	2.82	7	0.08	1.79
Walcott Hill	29	0.11	2.8			
Walcott Hill East	17	0.06	2.8	8	0.22	1.61
Mad Kiss	53	0.05	2.74	11	0.19	1.71
Mad Kiss South	9	0.08	2.76			
Mad Kiss West	25	0.21	2.74			
Weighted Average	**445**		**2.8**	**49**		**1.73**

*Based on AMEC 2010 saprolite data inclusive of mineralized material

13.9 Statistical Analysis and Variography

Basic statistics for the uncapped and capped gold composites are shown in Table 13.4.

Table 13.4: Statistics for uncapped and capped gold composites.

Category	No.	Min		Max		Mean		Capped		SD		VAR		COV	
	Comp.	Uncap	Cap	Uncap	Cap	Uncap	Cap	# cut	%	Uncap	Cap	Uncap	Cap	Uncap	Cap
Aleck Hill	4,877	0	0	83.32	70	2.39	2.38	4	0.08%	7	6.73	48.99	45.33	2.38	2.3
Aleck Hill North	1,085	0	0	45.54	30	1.91	1.89	3	0.28%	23.31	19.64	543.56	385.89	0.48	0.42
Rory's Knoll	13,637	0	0	412.84	80	2.76	2.72	7	0.05%	4.26	4.11	18.18	16.86	1.79	1.74
Rory's Knoll HG	2,874	0	0	412.84	80	5.11	4.9	6	0.05%	6.74	4.83	45.41	23.36	2.22	1.62
Rory's Knoll East	434	0	0	67.49	30	3.45	3.36	3	0.69%	11.07	6.77	122.64	45.78	2.17	1.38
Walcott Hill	535	0	0	49.6	15	1.62	1.5	4	0.75%	5.65	5.08	31.9	25.77	1.61	1.47
Walcott Hill East	3,369	0	0	542.18	40	2.6	2.33	4	0.18%	3.42	2.07	11.73	4.27	2.13	1.4
Mad Kiss	702	0	0	138.57	60	3.89	3.68	4	0.57%	15.86	4.06	251.39	16.46	4.9	1.42
Mad Kiss South	228	0.02	0.02	20.69	15	2.19	2.13	3	1.32%	9.22	7.47	85.02	55.83	2.41	2.04
Mad Kiss West	272	0	0	150.2	20	3.07	2.22	6	2.21%	2.56	2.2	6.58	4.83	1.28	1.14
Saprolite Hor.	2,197	0	0	54.9	20	0.73	0.71	4	0.18%	4.29	3.17	18.43	10.08	1.37	1.09

SRK calculated and modelled the variograms using capped composites for each domain using Geostatistical Software Library (GSLib). The general methodology to calculate and model variograms consists of calculating omni-directional, directional, and isotropic variograms in the plane of the domain. Principal directions were initially determined by the orientation of the data. Sensitivities were evaluated by varying the direction specification and comparing the resulting experimental variograms.

For each domain, SRK examined four different spatial metrics: (1) traditional semi-variogram, (2) traditional correlogram, (3) normal scores semi-variogram, and (4) normal scores correlogram. In general, the correlogram and normal scores transform facilitate the identification of spatial structure in the composite data, particularly when the traditional variogram shows little continuity. Variogram modelling was performed by assessing the structure(s) apparent from these different spatial measures, and fitting the most reliable measure. Whenever possible, the traditional variogram is the preferred measure to fit a model; however, the correlogram and/or normal scores variogram are often fitted due to the noise apparent in the traditional variogram.

The fitted variogram models were cross-checked against the mineral wireframes within GEMS to ensure consistency in orientation and reasonableness for estimation purposes. Modelled variogram parameters are tabulated in Table 13.5.

Table 13.5: Variogram models for the Aurora Gold Mine

Zone	C0	C1	C2	C3	R1x	R1y	R1z	Mod1*	R2x	R2y	R2z	Mod2*	R3x	R3y	R3z	Mod2
Aleck Hill	0.2	0.7	0.1		13	13	8.5	Exp	65	65	8.5	Sph				
Saprolite Hor.	0.2	0.5	0.3		15	15	5	Sph	40	40	10	Sph				
Aleck Hill North	0.2	0.65	0.15		16	25	8	Exp	80	25	8	Sph				
Rory's Knoll	0.25	0.25	0.28	0.22	6	8	3	Exp	60	30	30	Exp	100	30	30	Sph
Rory's Knoll HG	0.25	0.25	0.5		5	5	20	Sph	34	34	20	Sph				
Rory's Knoll East	0.2	0.6	0.2		70	11	5	Exp	70	70	5	Sph				
Walcott Hill	0.2	0.6	0.2		12	50	6	Exp	50	50	6	Sph				
Walcott Hill East	0.2	0.33	0.47		5	50	4	Sph	50	50	4	Sph				
Mad Kiss	0.2	0.8	-		55	55	6	Exp	90	35	6	Sph				
Mad Kisss South	0.2	0.6	0.2		2	2	2	Sph	-	-	-	-				
Mad Kiss West	0.2	0.5	0.3		12	12	6	Exp	50	50	6	Sph				

*Exp = Exponential and Sph = Spherical

For grade estimation, the search ellipses were rotated to align with each domain. The variograms were fitted using the GEMS "Azimuth-Dip-Azimuth" rotation method (Table 13.6). The methodology to set up this rotation is outlined as follows:

- the first axis rotation (“A”) represents the true Azimuth of the anisotropy X axis (Principal Azimuth - true strike);
- the second rotation (“D”) represents the dip angle of the anisotropy X axis (Principal Dip - negative downwards);
- and the third rotation (“A”) represents the azimuth of the anisotropy Y axis (Intermediate Azimuth).

Table 13.6: Search ellipse rotations

Zone	Rotations of Axes (Gemcom)		
	A	D	A
Aleck Hill	60	70	-30
Saprolite	145	0	55
Aleck Hill North	60	82	-35
Rory’s Knoll	130	75	40
Rory’s Knoll HG	0	0	0
Rory’s Knoll East	50	-85	-40
Walcott Hill East	45	-70	-55
Walcott Hill	55	90	-35
Mad Kiss	-22	-70	68
Mad Kiss South	53	70	-40
Mad Kiss West	60	80	-42

13.10 Block Model and Grade Estimation

A block model was created to cover the entire area of gold mineralization using the same UTM coordinate system as the borehole database (WGS84 Datum Zone 21N). The block model is not rotated and the parent cell size was set at 5 m by 5 m by 5 m. The block model parameters are summarized in Table 13.7.

Table 13.7: Aurora Gold Mine block model parameters.

Direction	Origin*	Size (metres)	Number of Blocks
East-West	195,343	5	365
North-South	750,245	5	350
Vertical	-2,100	5	450
* UTM Coordinates (WGS84 Datum Zone 21N)			

Gold grades were estimated using ordinary kriging as the primary estimator and variogram parameters summarized in Table 13.8. Gold grades were estimated in each domain separately using capped composites from that domain, variogram parameters shown in Table 13.5, estimation parameters presented in Table 13.8, and rotation parameters shown in Table 13.6.

In a previous study, the impact of varying estimation parameters for Aleck Hill and Rory's Knoll was evaluated by SRK in order to select optimal estimation parameters for block grade interpolation. The results of this comparative study indicate that the grade estimation for these domains is insensitive to slight variations of estimation parameters.

Three estimation passes were used to populate the block model with gold grades (with the exception of Rory's Knoll High Grade, East Walcott Hill and Mad Kiss). The first and second estimation passes considered search neighbourhoods adjusted to full and twice variogram ranges, respectively.

For the third estimation pass, the search ellipse was inflated to three times the variogram ranges (in the X and Y directions) to estimate a grade into blocks not estimated after the second estimation pass.

For comparison, gold grades were also estimated using an inverse distance algorithm (power of two) estimator using the same estimation parameters.

For estimation, wireframe boundaries are all considered hard domain boundaries, except for the estimation of the Rory's Knoll domain containing high grade sub-domains. The high grade domain boundaries are considered hard boundaries for the estimation of the high grade sub-domains, but are considered a soft boundary for the estimation of the remaining domain (e.g. lower grade envelope surrounding the high grade sub-domains).

Table 13.8: Grade estimation search parameters

Zone	Aleck Hill	Saprolite	North Aleck Hill	Rory's Knoll	Rory's Knoll HG	East Rory's Knoll	Walcott Hill	East Walcott Hill	Mad Kiss	South Mad Kiss	West Mad Kiss
Code	**200**	**2222**	**300**	**100**	**111**	**500**	**600**	**400**	**700**	**800**	**900**
Pass 1											
No. composites (min/max)	3/8	3/8	3/8	3/8	3/8	3/8	3/8	3/8	3/8	3/8	3/8
Type of search	Octant	Octant	Octant	Octant	Octant	Octant	Octant	Octant	Octant	Octant	Octant
Minimum number of octants	2	2	2	2	2	2	2	2	2	2	2
Max composite per octant	5	5	5	5	5	5	4	4	4	4	4
Max composite per borehole	2	2	2	2	2	2	2	2	2	2	2
Search radius about X	65	40	80	100	35	70	50	50	90	55	50
Search radius about Y	65	40	25	30	35	70	50	50	35	55	50
Search radius about Z	12	10	10	30	20	10	12	12	10	10	10
Pass 2											
No. composites (min/max)	2/12	2/12	2/12	2/12	2/12	2/12	2/12	2/12	2/12	2/12	2/12
Type of search	Ellipse	Ellipse	Ellipse	Ellipse	Ellipse	Ellipse	Ellipse	Ellipse	Ellipse	Ellipse	Ellipse
Minimum number of octants	-	-	-	-	-	-	-	-	-	-	-
Max composite per octant	-	-	-	-	-	-	-	-	-	-	-
Max composite per borehole	-	-	-	-	-	-	-	-	-	-	-
Search radius about X	130	80	160	200	70	140	100	100	180	110	100
Search radius about Y	130	80	50	60	70	140	100	100	70	110	100
Search radius about Z	24	20	20	60	40	20	24	24	20	20	20
Pass 3											
No. composites (min/max)	2/20	2/20	2/20	2/20	-	2/20	2/20	-	-	2/20	2/20
Type of search	Ellipse	Ellipse	Ellipse	Ellipse	-	Ellipse	Ellipse	-	-	Ellipse	Ellipse
Minimum number of octants	-	-	-	-	-	-	-	-	-	-	-
Max composite per octant	-	-	-	-	-	-	-	-	-	-	-
Max composite per borehole	-	-	-	-	-	-	-	-	-	-	-
Search radius about X	195	120	240	300	-	210	150	-	-	165	150
Search radius about Y	195	120	75	90	-	210	150	-	-	165	150
Search radius about Z	36	20	30	90	-	30	36	-	-	30	30

13.11 Model Validation and Sensitivity

SRK completed a detailed visual inspection of the model to check for proper coding of drill-hole intervals and block model cells, in both section and plan. The coding was found to be correct. Grade interpolation was checked relative to drill-hole composites and found to be reasonable.

SRK checked the block model estimates for global bias by checking the mean nearest neighbor (NN) estimate for Au g/t against model OK grade estimates. The results for the main zones for Measured and Indicated fresh rock are shown in Table 13.9.

Table 13.9: Global bias check statistics

Zone	Mean Grade (g/t)		
	Au OK	Au NN	% Change
Aleck Hill	2.304	2.288	0.7
Aleck Hill North	2.106	2.129	-1.1
Rory's Knoll/HG	2.798	2.738	2.2
Walcott Hill East	2.239	2.316	-3.3
Walcott Hill	1.788	1.778	0.6
Mad Kiss South	2.411	2.539	-3.5
Mad Kiss West	1.917	1.987	-5.0

The mean grades were found to be within 5%, which is considered acceptable.

SRK also checked for local trends in the grade estimate by comparing the mean grade estimate from the NN model against the OK model in swaths through the model on easting, northing, and elevation, as illustrated in Figure 13.8.

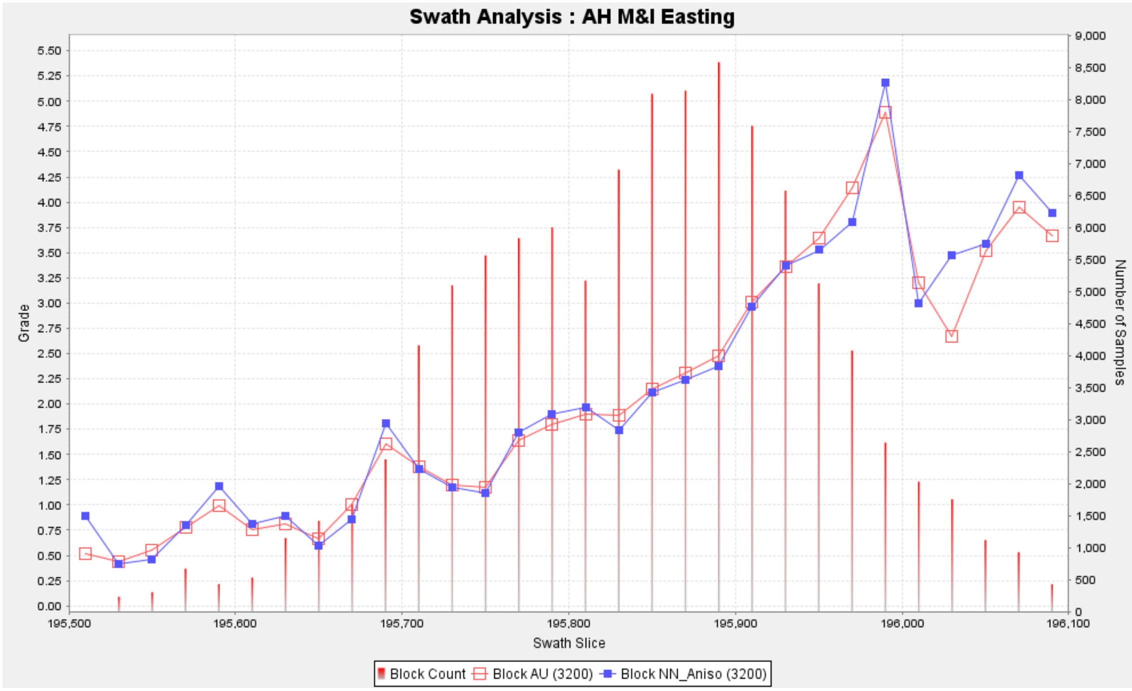


Figure 13.8: Swath Plot - Aleck Hill M&I Easting

The trends behaved as predicted, with major departures occurring only in areas with relatively low block counts.

13.12 Mineral Resource Classification

Block classification involved a two-step process. The first step is an automated classification that considered four main criteria: the number of composites used to code a block, the estimation pass, the average distance to informing composites and the kriging variance. Blocks coded during the first search pass were assigned an Indicated classification. All blocks interpolated during the second and third estimation pass were assigned an Inferred category. Measured blocks were classified only at Rory's Knoll and include only those blocks located within half a variogram range from informing composites and estimation parameters presented in Table 13.10.

For the Rory's Knoll and Rory's Knoll High Grade domains, an Indicated classification was assigned to blocks estimated during the first estimation pass and with kriging variances of less than 10. Those blocks also estimated during the first pass, but with a kriging variance greater than 10 were assigned an Inferred classification, along with all the blocks estimated during the second and third estimation pass.

Table 13.10: Measured classification criteria for Rory's Knoll and Rory's Knoll high grade

	Parameters
No. composites (min/max)	4/8
Type of search	Octant
Minimum number of octants	3
Max composite per octant	4
Max composite per borehole	2
Search radius about X	50
Search radius about Y	16
Search radius about Z	17

In the second step, the automated classification was manually adjusted to remove isolated blocks and to define regular areas at the same resource classification. Isolated blocks were reclassified to that of the surrounding blocks. During the second step, mining factors were also considered, such that isolated clusters of Inferred blocks surrounded by Indicated blocks particularly at Rory's Knoll and Aleck Hill were upgraded to Indicated.

13.13 Mineral Resource Estimate

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

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

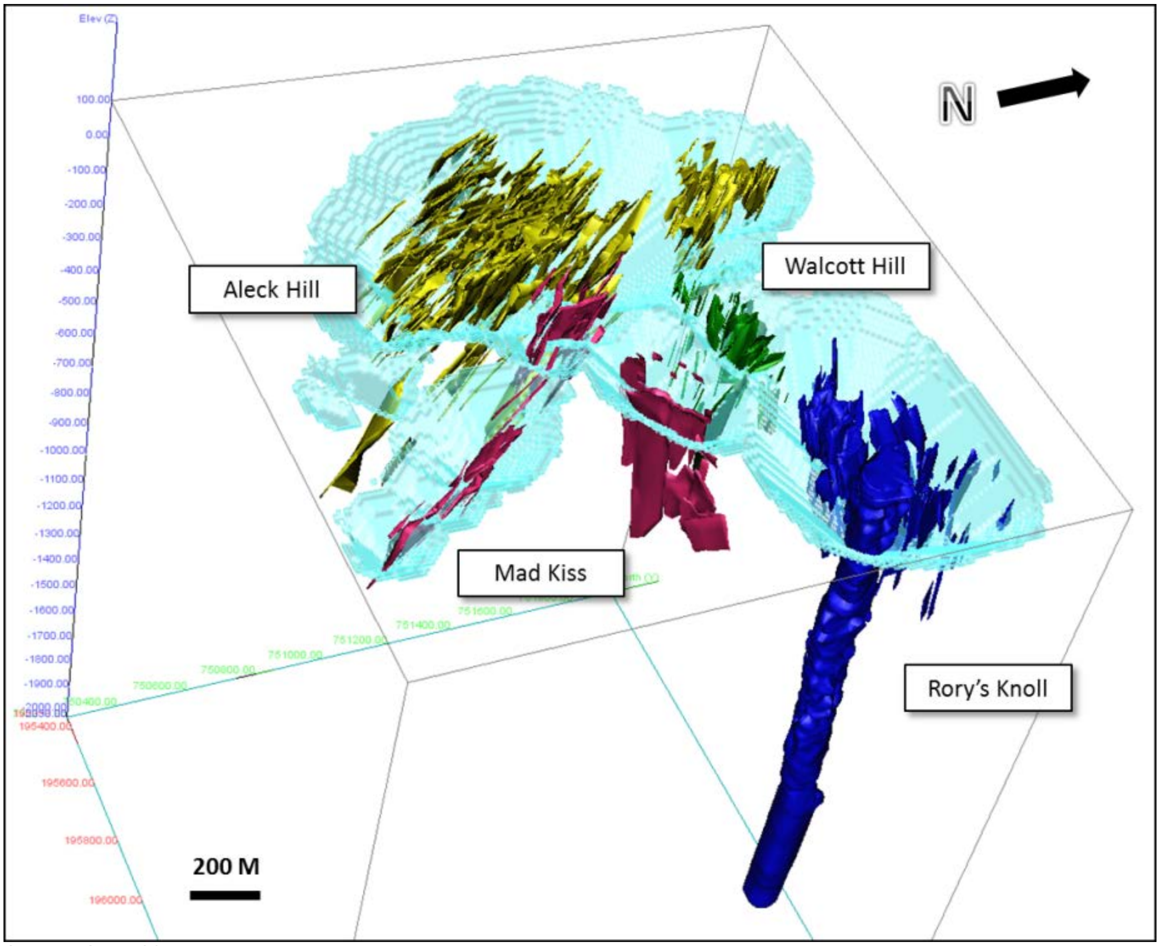
The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade that takes into account extraction scenarios and processing recoveries. SRK considers that parts of the Aurora gold mineralization are amenable for open pit extraction, while other parts could be extracted using an underground mining method. In order to determine the quantities of material offering "reasonable prospects for economic extraction" by an open pit, SRK used Whittle software, which evaluates the profitability of each resource block based on its value. The optimization parameters were based on discussions with Guyana Goldfields and benchmarking against similar projects (Table 13.11).

The reader is cautioned that the results from the pit optimization are used solely for testing the "reasonable prospects for economic extraction" by an open pit and do not represent an economic study as is required to evaluate mineral reserves. SRK considers blocks located within a conceptual pit shell are amenable for open pit extraction (Figure 13.8 and Figure 13.9) and can be reported as an open pit mineral resource. The block model quantities and grade estimates were also reviewed to determine the portions of the Aurora Gold Project deposit having "reasonable prospects for economic extraction" from an underground mine. A schematic long section showing the reserve and resource pits, and the material in the Rory's Knoll zone that are considered to be amenable to extraction by underground mining methods is shown in Figure 13.11.

The mineral resources for the Aurora Gold Mine are reported at a cut-off grade of 0.30, 0.40 and 1.80 g/t gold based on open pit (saprolite and rock) and underground mining scenarios, respectively. The open pit cut-off grades are based on assumptions summarized in Table 13.10, while the underground reporting cut-off grades was determined considering the same price and recovery assumptions in consultation with SRK mine engineers involved in the design of an underground mine for the Aurora Gold Mine.

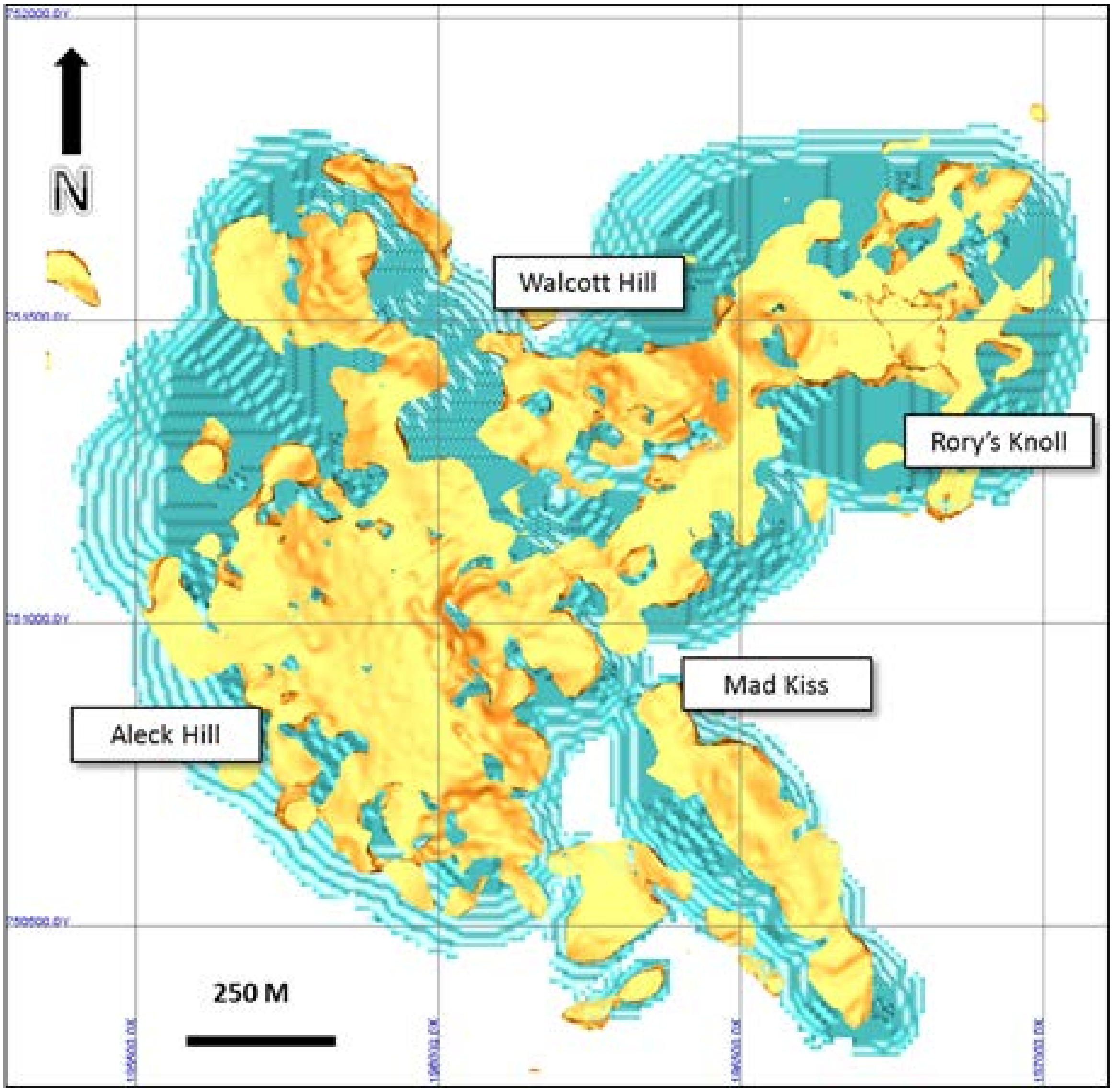
Table 13.11: Conceptual pit optimization assumptions considered for open pit resource reporting.

Parameter	Assumption	
	Saprolite	Rock
Pit Slopes (per geotechnical sector)	23 to 31 degrees	43 to 51 degrees
Mining cost (ore and waste)	US$1.40/t	US$1.75/t
Increment mining cost by 5 m bench	US$0.02/t mined/bench	US$0.02/t mined/bench
Process cost	US$6.00/t feed	US$8.00/t feed
G & A costs	US$3.00/t feed	US$4.00/t feed
Process recovery	97.0 percent	94.7 percent
Assumed process rate	4,000 tpd	8,000 tpd
Gold price	US$1,300 per ounce	US$1,300 per ounce
Mining dilution	16.0 percent	16.0 percent



Source: SRK, 2017

Figure 13.9 Aurora Gold Mine Project modelled domains in relation to the conceptual pit shells



Source: SRK, 2017

Figure 13.10: Aurora 10-m saprolite mineralization in relation to the conceptual pit shell.

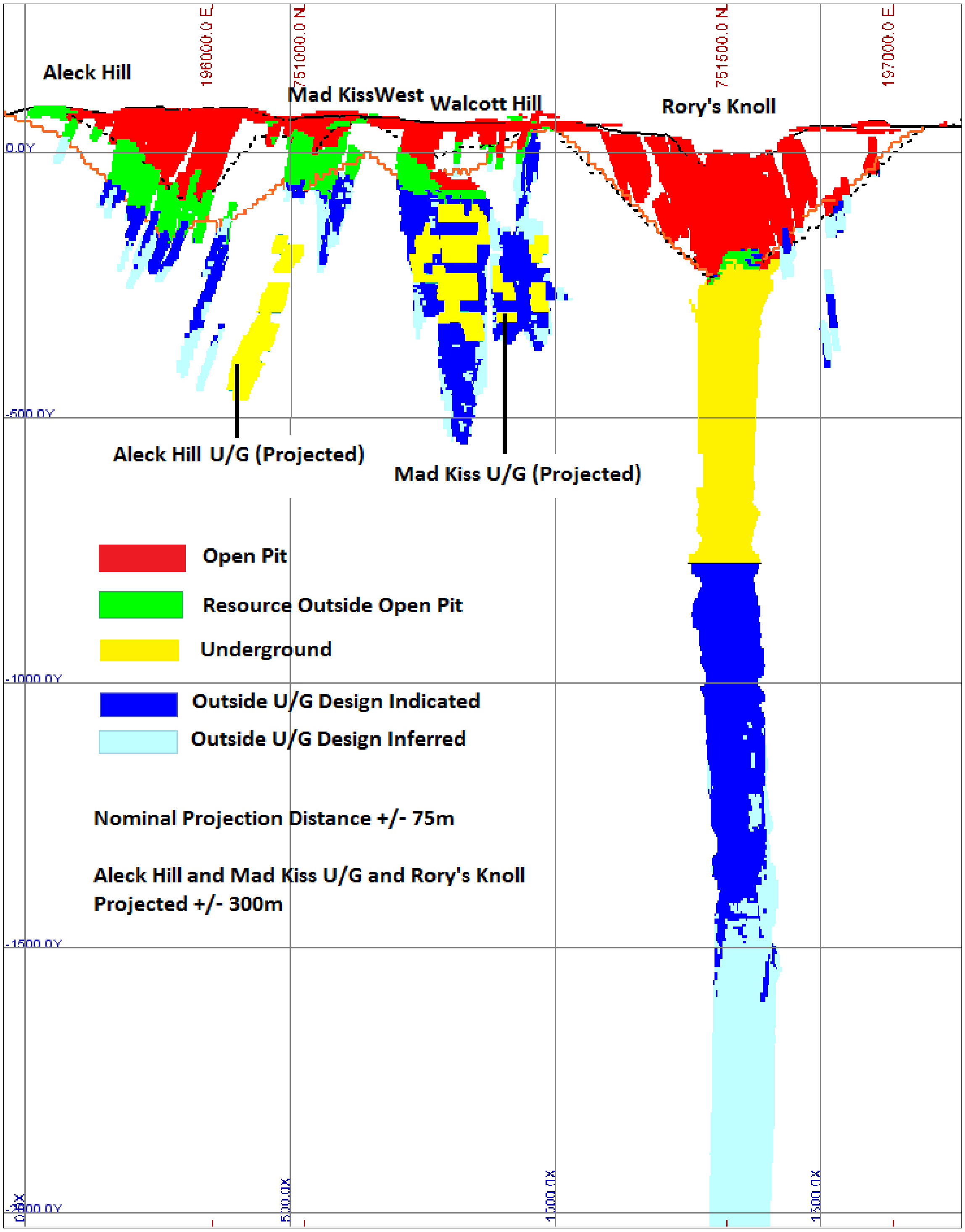


Figure 13.11: Aurora: Schematic long section of material categories showing reserve and resource pits and Rory's Knoll U/G reserves (looking North-West)

Mineral resources were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) by Timothy Carew, P.Geo (APEGBC #19706), appropriate independent qualified person for the purpose of National Instrument 43-101.

Table 13.12: Mineral Resource Statement, Aurora Gold Mine Project, Guyana, SRK Consulting (Canada) Inc., Effective December 31, 2016

Classification	Zone	Quantity (kt)			Grade (g/t Au)			Contained Metal (k Oz)		
		Sap	Rock	Total	Sap	Rock	Total	Sap	Rock	Total
Open Pit Mining										
Measured										
	Rory's Knoll		4,500	4500		3.23	3.23		470	470
	Stockpiles	90	640	730	0.83	1.41	1.34		30	30
Total Measured		**90**	**5,140**	**5,230**	**0.83**	**3.00**	**2.97**		**500**	**500**
Indicated										
	Saprolite	1,670		1,670	0.92		**0.92**	50		50
	Aleck Hill	1,890	11,600	13,490	2.49	2.57	**2.56**	150	960	1,110
	Rory's Knoll	10	6,780	6,790	2.85	2.61	**2.61**		570	570
	Walcott Hill	50	620	670	2.38	2.05	**2.07**		40	40
	Mad Kiss	260	1,560	1,820	2.03	3.61	**3.38**	20	180	200
Total Indicated		**3,880**	**20,560**	**24,440**	**1.78**	**2.65**	**2.51**	**220**	**1,750**	**1,970**
Total M + I		**3,970**	**25,700**	**29,670**	**1.76**	**2.72**	**2.59**	**220**	**2,250**	**2,470**
Inferred										
	Saprolite	2,050		2,050	0.94		**0.94**	60		60
	Aleck Hill	270	1,340	1,610	1.54	1.67	**1.65**	10	70	80
	Rory's Knoll	10	310	320	1.43	2.55	**2.52**		30	30
	Walcott Hill	10	60	70	2.79	1.72	**1.87**			
	Mad Kiss	210	510	720	2.20	3.00	**2.77**	10	50	60
Total Inferred		**2,550**	**2,220**	**4,770**	**1.12**	**2.10**	**1.57**	**80**	**150**	**230**
Underground Mining										
Indicated										
	Aleck Hill		2,700	2,700		3.83	**3.83**		330	330
	Rory's Knoll		25,680	25,680		3.89	**3.89**		3,210	3,210
	Walcott Hill		180	180		2.80	**2.80**		20	20
	Mad Kiss		1,500	1,500		4.60	**4.60**		220	220
Total Indicated			**30,060**	**30,060**		**3.91**	**3.91**		**3,780**	**3,780**
Total M + I			**30,060**	**30,060**		**3.91**	**3.91**		**3,780**	**3,780**
Inferred										
	Aleck Hill		1,620	1,620		3.78	**3.78**		200	200
	Rory's Knoll		9,330	9,330		4.25	**4.25**		1,270	1,270
	Walcott Hill		120	120		2.65	**2.65**		10	10
	Mad Kiss		740	740		3.38	**3.38**		80	80
Total Inferred			**11,810**	**11,810**		**4.12**	**4.12**		**1,560**	**1,560**
Combined Mining										
Measured										
	Rory's Knoll		4500	4500		3.23	3.23		470	470
	Stockpiles	90	640	730	0.83	1.41	**1.34**		30	30
Total Measured		**90**	**5,140**	**5,230**	**0.83**	**3.00**	**2.97**		**500**	**500**
Indicated										
	Saprolite	1,670		1,670	**0.93**		**0.93**	50		50
	Aleck Hill	1,890	14,300	16,190	**2.47**	2.81	**2.77**	150	1,290	1,440
	Rory's Knoll	10	32,460	32,470	**0.00**	3.62	**3.62**		3,780	3,780
	Walcott Hill	50	800	850	**0.00**	2.33	**2.20**		60	60
	Mad Kiss	260	3,060	3,320	**2.39**	4.07	**3.93**	20	400	420
Total Indicated		**3,880**	**50,620**	**54,500**	**1.76**	**3.40**	**3.28**	**220**	**5,530**	**5,750**
Total M + I		**3,970**	**55,760**	**59,730**	**1.72**	**3.36**	**3.25**	**220**	**6,030**	**6,250**
Inferred										
	Saprolite	2,050		2,050	0.91		0.91	60		60
	Aleck Hill	270	2,960	3,230	1.15	2.84	**2.70**	10	270	280
	Rory's Knoll	10	9,640	9,650	0.00	4.19	**4.19**		1300	1,300
	Walcott Hill	10	180	190	0.00	1.73	**1.64**		10	10
	Mad Kiss	210	1,250	1,460	1.48	3.23	**2.98**	10	130	140
Total Inferred		**2,550**	**14,030**	**16,580**	**0.98**	**3.79**	**3.36**	**80**	**1,710**	**1,790**

Notes:

1. Mineral resources are not mineral reserves and do not have demonstrated economic viability. All figures have been rounded to reflect the relative accuracy of the estimates.
2. The cut-off grades are based on a gold price of US$1,300 per ounce and metallurgical recoveries of 97 percent and 94.5 percent for saprolite and rock material respectively.
3. Open pit resources are reported within conceptual optimized open pit shells, whereas underground resources are external to these. Open pit resources are reported at a cut-off grade of 0.30 g/t Au and 0.40 g/t Au for saprolite ore and rock ore respectively, whereas underground resources are reported at a cut-off of 1.8 g/t Au.
4. Open pit mineral resources exclude the mined out areas.
5. Stockpile data based on inventory in RoM, external RoM and Refeed stockpile as of EOY 2016.

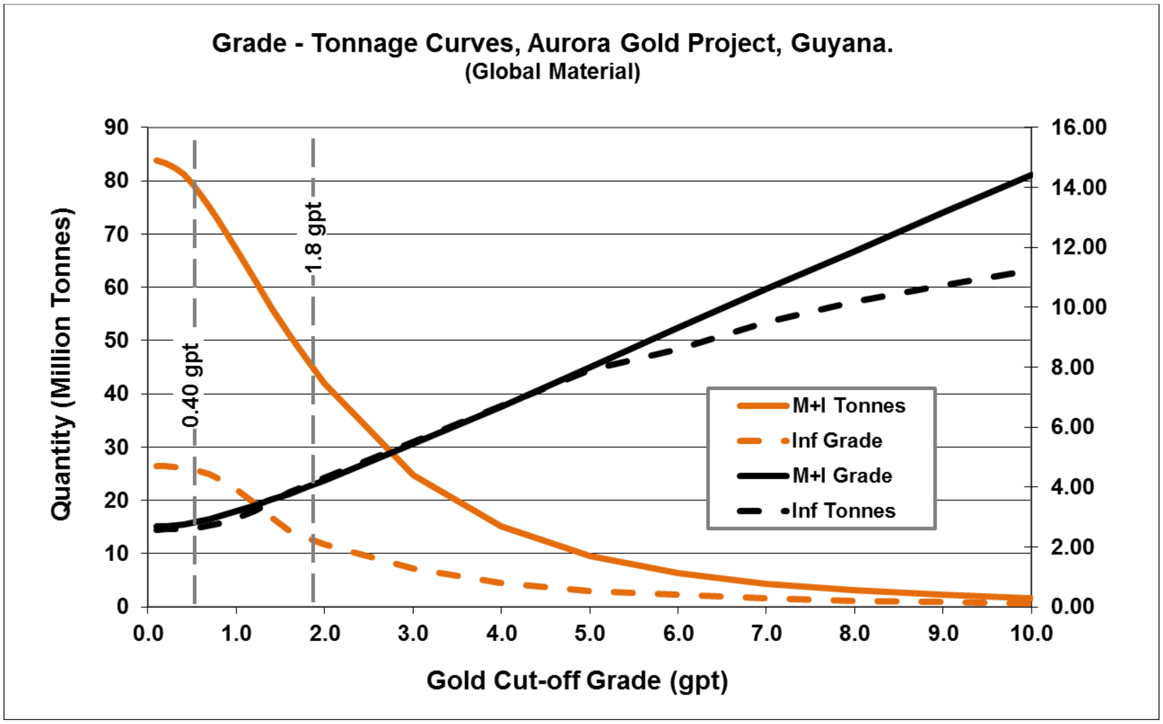
13.14 Grade Sensitivity Analysis

The mineral resources are highly sensitive to reporting cut-off grade. To illustrate this sensitivity, the block model quantities and grade estimates are presented at various cut-off grades in Table 13.13. The reader is cautioned that these figures should not be misconstrued as a mineral resource. The reported quantities and grades are only presented as a sensitivity of the resource model to the selection of cut-off grade. This cut-off grade sensitivity is also illustrated as grade tonnage curves (Figure 13.12).

Table 13.13: Global block model quantity and grades* estimates at various cut-off grades, Aurora Gold Mine Project, Guyana

Cut-off (g/t Au)	Measured & Indicated		Inferred	
	Quantity (Mt)	Grade (g/t Au)	Quantity (Mt)	Grade (g/t Au)
0.1	83,84	2.68	26.41	2.58
0.2	83.34	2.70	26.37	2.59
0.3	82.47	2.72	26.31	2.59
0.4	81.21	2.76	26.14	2.61
0.5	79.46	2.81	25.81	2.64
0.6	77.41	2.87	25.42	2.67
0.7	75.05	2.94	24.88	2.71
0.8	72.47	3.02	24.13	2.77
0.9	69.74	3.10	23.21	2.85
1.0	67.02	3.19	22.04	2.95
1.2	61.53	3.38	19.53	3.19
1.4	56.17	3.58	16.76	3.50
1.6	51.11	3.78	14.3	3.84
1.8	46.33	4.00	12.88	4.08
2.0	41.92	4.22	11.68	4.30
3.0	24.74	5.44	7.21	5.50
4.0	15.18	6.69	4.49	6.70
5.0	9.57	8.01	2.97	7.90
6.0	6.32	9.32	2.31	8.59
7.0	4.36	10.61	1.61	9.49
8.0	3.12	11.86	1.20	10.18
9.0	2.26	13.15	90.00	10.74
10.0	1.67	14.43	64.00	11.25

* The reader is cautioned that the figures presented in this table should not be misconstrued as a Mineral Resource Estimate. The reported quantities and grades are only presented as a sensitivity of the deposit model to the selection of cut-off grade. The above reported quantities may not meet the test of reasonable prospects for economic extraction.



Source: SRK, 2017

Figure 13.12: Aurora Gold Mine Project grade-tonnage curves

14 Mineral Reserve Estimates

14.1 Summary

The Mineral Reserve estimate for the Aurora project abides by the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM) Definition Standards (2014) as required under NI43-101 guidelines. To convert Mineral Resources to Mineral Reserves, SRK applied modifying factors of dilution and ore loss to only the Measured and Indicated categories of the Mineral Resource. Inferred Resources cannot form a basis for Mineral Reserves. The Aurora project consists of both open pit and underground minable Mineral Resources for which Mineral Reserves have been independently estimated.

The Mineral Reserve statement for the Aurora Gold Project is presented in Table 14.1. The estimate is based on the June 2012 resource block model prepared by SRK. This reserve estimate includes the results from open pit mining in Rory's Knoll, Aleck Hill, Walcott Hill, Mad Kiss and North Aleck, and underground mining in Rory's Knoll.

Table 14.1: Mineral reserve statement, Aurora Gold Mine Project, Guyana, December 31, 2016

Category	Quantity (kt)	Grade (g/t Au)	Contained Metal (k Oz)
Proven			
OP Saprolite	336	1.60	17
OP Rock	4,864	2.99	468
Total Proven	**5,200**	**2.90**	**485**
Probable			
OP Saprolite	2,934	1.91	180
OP Rock	12,128	3.02	1,178
UG Rock	16,519	3.19	1,694
Total Probable	**31,580**	**3.01**	**3,052**
Total P&P	**36,781**	**2.99**	**3,537**

Notes:

1. Mineral Reserves are based on a gold price of US$1,200 per ounce, 8% royalty and an average metallurgical recovery of 96.0% for saprolite and 94.0% for rock material.
2. Open pit saprolite rock reserves are reported at a cut-off grade of 0.44 g/t Au and 0.42 g/t Au for vein and upper saprolite material respectively. Open pit rock reserves are reported at a cut-off grade of 0.76 g/t Au and 0.64 g/t Au for vein and Rory's Knoll rock material respectively.
3. Underground rock reserves are reported at a cut-off grade of 1.5 g/t Au.
4. Mineral Reserves are contained within Mineral Resources.
5. Independent qualified persons as defined by National Instrument 43-101 are Robert J. McCarthy P Eng.(#27309), for open pit reserves and Christopher A. Elliott, FAusIMM (#103277), for underground reserves. SRK is not aware of mining, metallurgical, infrastructure, permitting, or other factors that could materially affect the mineral reserve estimates.

14.2 Introduction

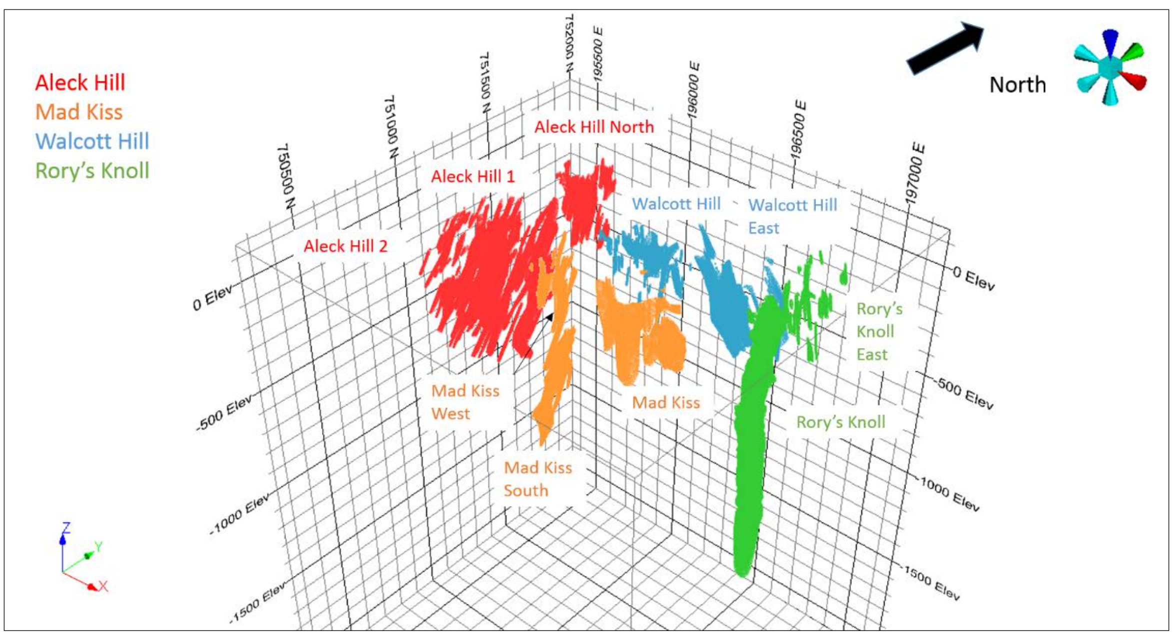
The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM Definition Standards (2014) for reporting Mineral Resources and Mineral Reserves. As such, the Mineral Reserves are based on Measured and Indicated Resources, and do not include any Inferred Resources.

14.2.1 Resource Model

The 2016 Mineral Reserve estimate (SRK, 2017) was developed from a resource model, which was prepared by SRK in June 2012. In July 2015, the resource model was converted by SRK from the GEMS format (percent-fill) to the Datamine format (sub-cells) for underground design purposes. The resource model was not re-estimated during the conversion. The sub-cell boundaries were defined by a wireframe model that defined the lithology boundaries; the sub-cells provide a sharper resolution of the ore/waste contact.

14.2.2 Mining Zones

The Aurora Gold Mine consists of nine distinct auriferous sub-zones, grouped into four main zones, as shown in Figure 14.1: Aleck Hill, Walcott Hill, Rory's Knoll, and Mad Kiss.



Source: SRK, 2017

Figure 14.1: The Main auriferous zones – isometric projection looking North

Rory's Knoll is considered to be the primary mining target for both open pit and underground methods. Aleck Hill is also significant target for open pit mining, while the other deposits, or zones, are considered to be the Satellite deposits.

14.3 Open Pit Mineral Reserves

All of the mining zones noted in Section 14.2.2 are amenable to open pit mining by truck/shovel mining methods. To derive a Mineral Reserve, it is necessary to apply modifying factors of dilution and mining losses to the Mineral Resource and then subject resulting “Reserve Model” to pit optimization analysis to derive economic limits. These processes are described herein.

14.3.1 Dilution and Mining Losses

The approach to establishing dilution and mining losses has not changed since the most recent feasibility study update (MMC, 2016). The following is from the corresponding NI 43-101 report:

Mineralized tonalite, veins in the rock, and some veins in the saprolite are visible in the drill core and it is anticipated that it will be possible to identify the mineralized rock in the mining face. In the upper saprolite mineralization, extensive weathering is expected to make it difficult to visually identify the mineralized material in the dig face.

It is estimated that 0.5 m of dilution will be included along the ore/waste contacts during mining. Based on the geometry of the interpreted veins and tonalite, an estimate of external dilution was made for each of the upper saprolite, saprolite veins, Rory’s Knoll rock, and other rock zones. In addition, the average grade of the anticipated dilution was estimated by SRK resource geologists.

The mining loss of each mineralized zone is estimated to be 5%. This is typical of open pit mining and accounts for misdirected truck loads and losses where excessive dilution reduces gold grades to below the cut-off grade.

The external dilution, dilution grades and mining losses for each mineralized zone are detailed in Table 14.2.

Table 14.2: External dilution, dilution grades and mining losses for each mineralized zone

Mineralized Zone	Mining Losses	External Dilution	Dilution Grade
	(%)	(%)	(g/t Au)
Upper Saprolite	5	17	0.10
Saprolite Veins	5	23	0.06
Rory's Knoll Rock	5	4	0.10
Other Rock	5	22	0.06

14.3.2 Pit Optimization

Pit optimization and strategic scheduling was evaluated using Whittle™ software. This advanced program uses the Lerchs-Grossmann algorithm to generate pit shells providing the maximum operating margin, or cash flow (before capital, taxes or discounting), based on a resource model and a set of economic and technical input parameters.

Pit optimization economic parameters include gold prices and offsite costs, unit mining costs, processing costs, and general and administrative costs,. Pit optimization technical parameters include pit footprint constraints related to the Cuyuni River proximity, estimates of mining dilution, mining loss, process recovery, and pit overall slope angles. Pit overall slope angles are derived from geotechnical criteria, adjusted for the expected haulage ramp layout.

Pit optimization routines use only whole blocks from the resource block model. Only those resource blocks classified as Measured and Indicated Mineral Resources were allowed to drive the pit optimization for the feasibility level study. Inferred Mineral Resource blocks, regardless of grade and recovery, bear no economic value and were treated as waste.

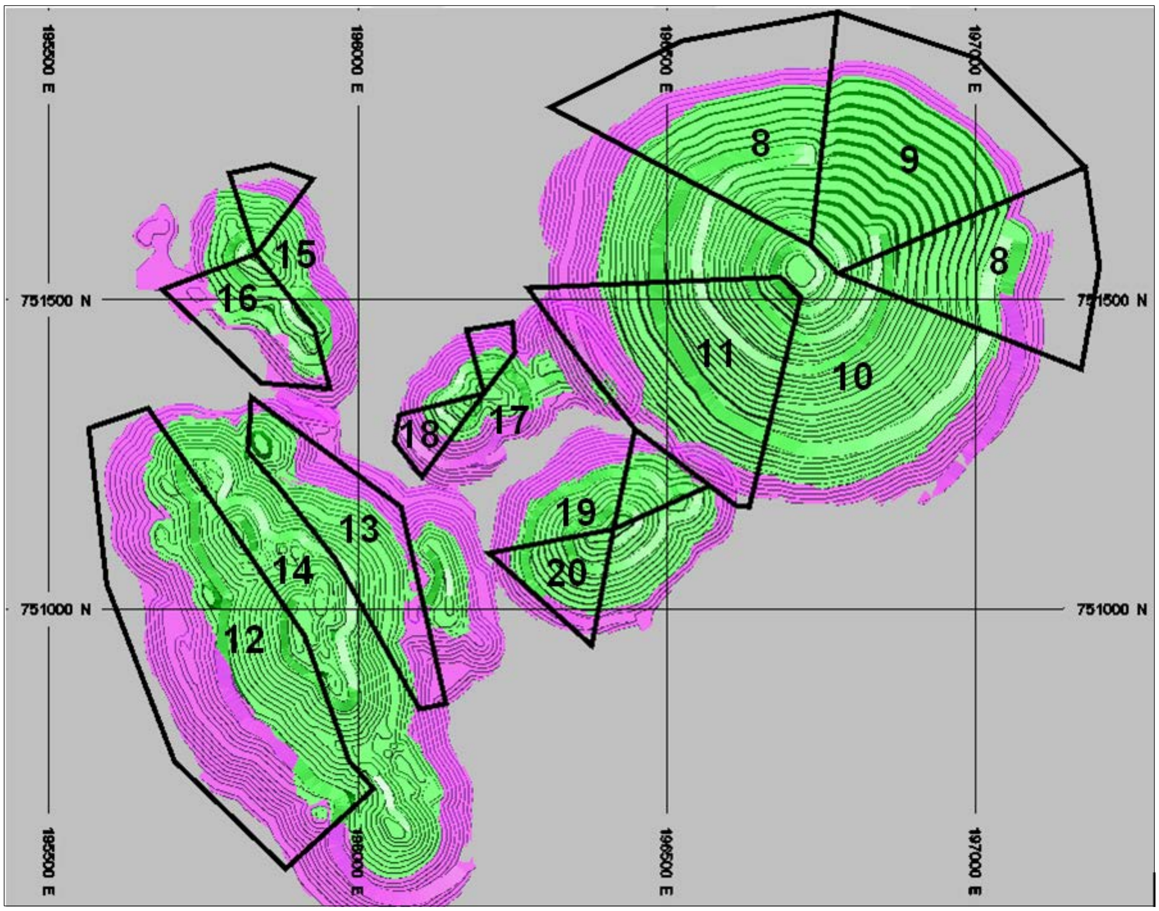
A series of nested pit shells was generated for each of the mining zones. Nested pit shells were evaluated for incremental value, grade and strip ratio. Relevant pit shells were used to guide pit phase and ultimate pit designs. Five areas were evaluated separately to identify the best ultimate pit limits. Rory's Knoll, Aleck Hill, North Aleck, Mad Kiss and Walcott Hill.

The Aurora pit optimization input parameters are summarized in Table 14.3.

Table 14.3: Aurora pit optimization input parameters

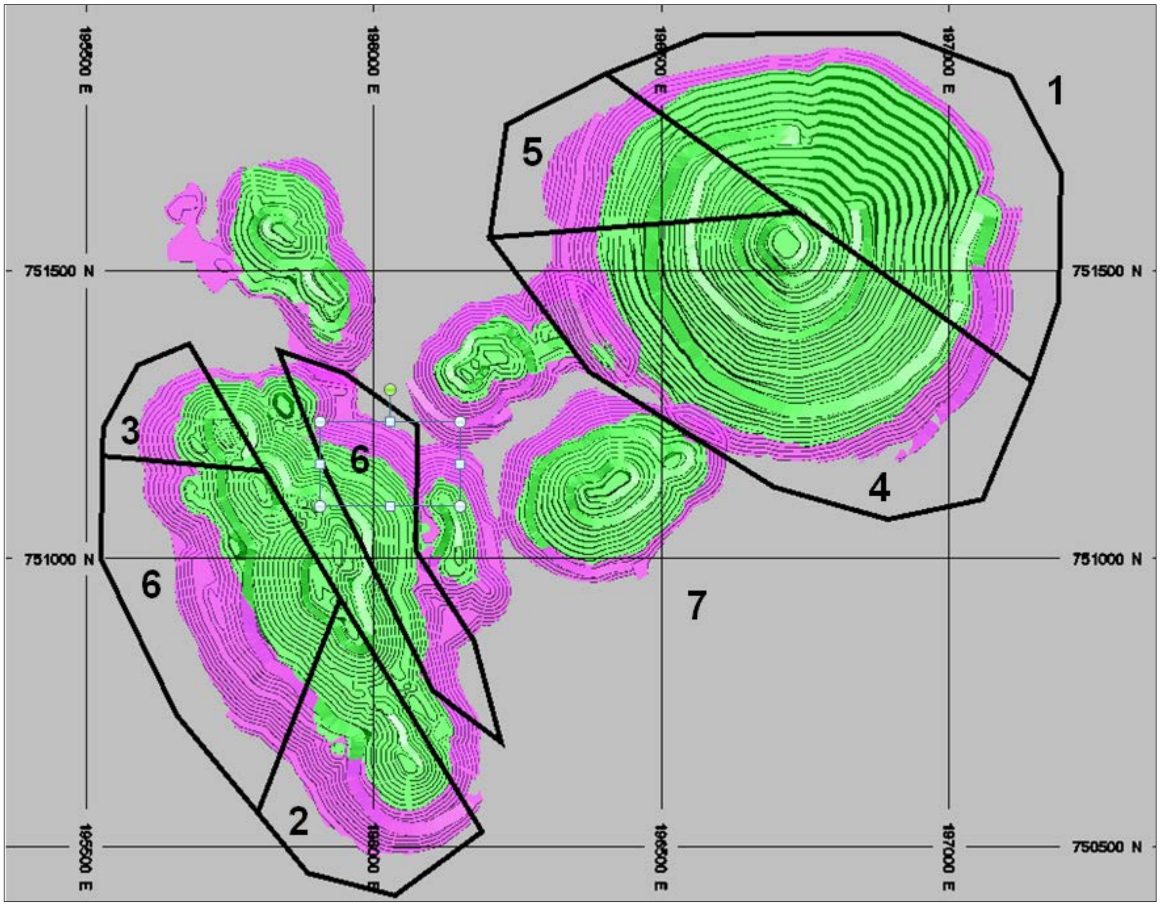
Parameter	Units	Saprolite	Rock
Revenue			
Revenue basis		Measured and Indicated Mineral Resources only	
Au Price	($/oz)	1,200	
Au Payable	(%)	100%	
Refining Charges	($/oz)	2.2	
Royalties	(%)	8%	
Value of Au in Dore	($/oz)	1,102	
Value of Au in Dore	($/g)	35.43	
Mill Recovery	(%)	96%	94%
Estimated Operating Costs			
Mining Cost at Surface	($/t)	1.7	2.5
Incremental Mining Cost	($/t/bench)	0.02	0.02
Process Cost	($/t)	6.29	14.5
G&A Cost	($/t)	6	6
Processing Rates			
Plant Feed Rate	(Mtpa)	2.92	
Discount Rate	(%)	5%	
Mine Design Parameters			
Mining Recovery	(%)	95%	
Dilution		Variable by Rock Type and Area	
Overall Pit Slopes (degrees)		Variable	

The open pit geotechnical design criteria, including the inter-ramp angle, bench face angle and berm widths, are described in Section 15.2.4. The overall slope angles used in the Whittle analysis were calculated from the geotechnical design criteria and the expected wall height and ramp requirements. The geotechnical design sectors for rock and saprolite are shown in Figure 14.2 and Figure 14.3 respectively. The overall slope angles used in pit optimization are derived from these sectors and are listed in Table 14.4.



Source: SRK, 2017

Figure 14.2: Rock geotechnical domains for optimization



Source: SRK, 2017

Figure 14.3: Saprolite geotechnical domains for optimization

Table 14.4: Overall slope angles used in the pit optimization

Slope Code	Rock Type	Domain Name	Bench Face Angle	Bench Height (m)	Berm Width (m)	Inter Ramp Angle	Stacks	Stack Height (m)	Overall Slope Angle
1	Sap	Sap A	70°	5	7	29.5°	1	30	33°
2	Sap (AH<40m)	Sap A1	70°	5	6	32.6°	2	15	27°
3	Sap	Sap A2	70°	5	7	29.5°	1	30	33°
4	Sap	Sap B	70°	5	6	32.6°	2	15	27°
5	Sap (RK HW)	Sap B1	70°	5	6	32.6°	2	30	29°
6	Sap (AH>40m)	Sap B2	70°	5	5	36.2°	2	25	32°
7	Sap	Sap C	70°	5	5	36.2°	2	30	34°
8	Rock	RKN1	80°	20	12	52.2°	4	80	51°
9	Rock	RKN2	70°	20	13	44.6°	4	80	45°
10	Rock	RKS1	80°	10	6.5	50.4°	6	50	45°
11	Rock	RKS2	75°	20	14	45.9°	6	60	45°
12	Rock	AHS1	80°	10	6.5	50.4°	4	60	47°
13	Rock	AHS2	80°	10	5	55.9°	4	60	51°
14	Rock	AHM1	80°	10	6.5	50.4°	3	80	48°
15	Rock	NA1	80°	10	6.5	50.4°	2	40	49°
16	Rock	NA2	75°	10	8	43.1°	2	40	43°
17	Rock	WH1	80°	10	6.5	50.4°	2	40	49°
18	Rock	WH2	75°	10	8	43.1°	2	40	43°
19	Rock	MK1	80°	10	6.5	50.4°	2	40	49°

14.3.3 Whittle Results and Pit Selection

The nested pit shells generated for each of the major mineralization zones were evaluated for tonnes of ore and waste as well as net present value (excluding capital costs). The incremental increase in value, ore tonnes, gold grade and strip ratio were all assessed in selecting pit shells for the ultimate pit limits as well as appropriate step changes for pit phasing.

The detailed pit designs corresponding to the selected pit optimization shells are discussed in Section 15.3.3.

14.3.4 Open Pit Mining Cut-Off Grades

The open pit cut-off grades are calculated separately for each major mineralized zone to account for changes in mining and processing costs, gold recoveries and external dilution. The parameters used to determine the open pit cut-off grades (COG) are the same as those used for pit optimization (Table 14.3). The resulting COGs are listed in Table 14.5.

Table 14.5: Open pit cut-off grade parameters

Description	Units	Upper Saprolite	Vein Saprolite	Rory' Knoll Rock	Other Rock
Cut-off Grade	g/t Au	0.42	0.44	0.64	0.76

14.3.5 Open Pit Mineral Reserve Estimate

Combining the results from the five mining areas and depleting the results to the December 31, 2016 mined out topography surface results in the following quantities for the 2016 year end open pit Mineral Reserve estimate:

Table 14.6: Open Mineral Reserve estimates for Aurora Gold Mine project, effective December 31, 2016

Category	Quantity (kt)	Grade (g/t Au)	Contained Metal (k Oz)
Proven			
OP Saprolite	336	1.60	17
OP Rock	4,864	2.99	468
Total Proven	**5,200**	**2.90**	**485**
Probable			
OP Saprolite	2,934	1.91	180
OP Rock	12,128	3.02	1,178
Total Probable	**15,062**	**2.80**	**1,358**
Total P&P	**20,262**	**2.83**	**1,843**

Notes:

6. Mineral Reserves are based on a gold price of US$1,200 per ounce, 8% royalty and an average metallurgical recovery of 96.0% for saprolite and 94.0% for rock material.
7. Open pit saprolite rock reserves are reported at a cut-off grade of 0.44 g/t Au and 0.42 g/t Au for vein and upper saprolite material respectively. Open pit rock reserves are reported at a cut-off grade of 0.76 g/t Au and 0.64 g/t Au for vein and Rory's Knoll rock material respectively.
8. Underground rock reserves are reported at a cut-off grade of 1.5 g/t Au.
9. Independent qualified person as defined by National Instrument 43-101 for open pit reserves is Robert J. McCarthy, P.Eng.(#27309),. SRK is not aware of mining, metallurgical, infrastructure, permitting, or other factors that could materially affect the mineral reserve estimates.

14.4 Underground Mineral Reserves

14.4.1 Rory's Knoll

Rory's Knoll is the largest underground deposit in this study and continues beneath the Rory's Knoll open pit. The Rory's Knoll underground deposit was addressed in the 2012 feasibility study (SRK, 2012), the 2013 feasibility study (Tetra Tech, 2013), and the 2015 feasibility study (MMC, 2016). For this 2017 feasibility study update the stope shapes and supporting infrastructure were updated to reflect a new open pit design and a cut-off grade based on a lower gold price.

14.4.1.1 Orebody Description

At the cut-off grade of 1.50 g/t Au, the Rory's Knoll orebody shows good continuity along strike and down dip. Below the open pit, the orebody geometry approximates a sub-vertical elliptical pipe of approximately 100 m thick at its widest point and approximately 1,600 m in vertical extent. A small number of barren zones have been identified within the deposit. However, these barren zones have been included in the mine design in order to preserve continuity.

14.4.1.2 Cut-off Grade

A cut-off grade of 1.50 g/t Au was used to define the Mineral Reserves for Rory's Knoll.

In this feasibility study update, a gold price of $1,200/oz is being used:

- The 2016 feasibility study (MMC, 2016),
- GGI supplied inputs,
- External inputs derived from contractor costs, and SRK's experience on similar projects and operating mines.

The output from the cost model was used to develop a preliminary COG estimate that was carried forward for design purposes. A breakeven COG for development material is also included. The COG estimate is based on a break-even operating cost and does not include sustaining capital.

Table 14.7 shows the cut-off grade calculation and the estimated site operating costs used for the Rory's Knoll underground deposit, based on the data available in the 2016 feasibility study update (MMC, 2016) which is summarised in this report.

Table 14.7: Rory's Knoll: cut-off grade estimate

Parameter	Unit	Value
Site operating cost		
Underground mining	$/t	26.10
Processing	$/t	14.50
G&A	$/t	8.00
Total site operating cost	**$/t**	**48.60**
Gold selling price	$/oz	1,200
Au payable	%	100%
Refining charges	$/oz	3.00
Royalty		
Applicable gold price	$/oz	1,200
Royalty rate	%	8%
Value of Au in doré	$/oz	1,101
	$/g	35.41
Processing recovery	%	94%
Value of Au in plant feed	$/g	33.28
Plant feed grade (diluted COG)	**g/t Au**	**1.46**
External dilution at grade	g/t Au	1.13
In-situ Au COG	**g/t Au**	**1.50**
Development COG	**g/t Au**	**0.72**

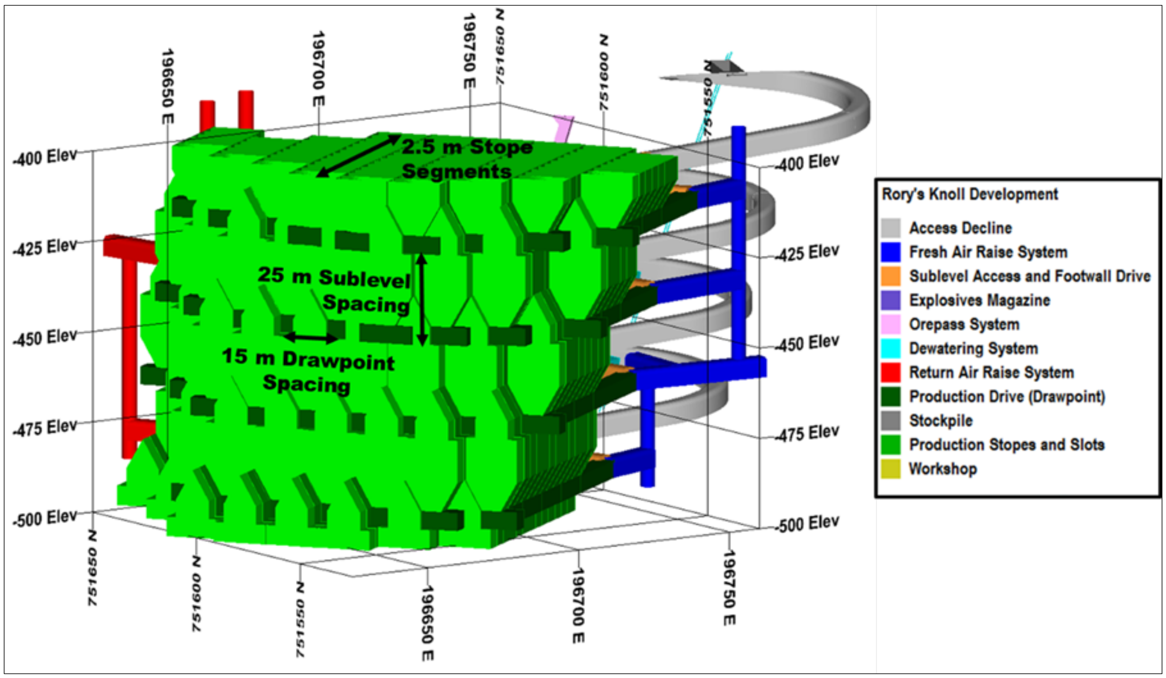
The proportion of dilution is calculated as the ratio of waste/(ore plus waste) and typically expressed as a percentage.

14.4.1.3 Mining Shapes

SRK used Studio5D Planner™ software to layout open benching and Sublevel Retreat (SLR) stopes through the Rory's Knoll deposit. A consistent 25 m sublevel and 15 m drawpoint spacing used by both mining methods allowed for a single stope cross sectional shape to be used throughout the entire deposit. Figure 14.4 shows the typical geometry of the mining shapes.

Stopes were modelled and evaluated in maximum 2.5 m cross section lengths. Where practical, and without compromising the vertical mining continuity required by the open benching and SLR mining methods, stope lengths were adjusted to only include stope segments above the 1.50 g/t Au cut-off. With this methodology, perimeter stopes were only added to the design where there was sufficient stope length to justify the additional develop cost.

Open benching stopes were also clipped where they intersected the open pit. In areas where the uniform stope geometry left ore between the stope and the pit, this remaining material was included in a remnant block model and used in material mixing and dilution estimates. Material mixing and dilution was simulated using GEOVIA's PCSLC software. The software can simulate material flow and mixing for the Sublevel Caving (SLC) mining method. The SLR mining method is similar to SLC except there is no caving. PCSLC software has been used in this study to simulate open benching and SLR mining activities by modeling appropriate sources of dilution and applying appropriate extraction and recovery parameters.



Source: SRK, 2017

Figure 14.4: Rory's Knoll: Geometry of mining shapes (looking Northeast)

14.4.1.4 Dilution and Recovery Estimates

Following is a brief discussion of the approach to estimating dilution and recovery.

The mining evaluation used PCSLC software (refer to Section 15.4.1.1 for more detailed description of the software) to model material mixing, mined dilution and ore recovery for the planned stopes. These results were combined with tonnages and grades of planned development shapes in CAE Enhanced Production Scheduler (EPS) software to generate a total, Life-of-Mine, development and production schedule.

Material contained within the planned stope shapes below the 1.50 g/t Au in-situ cut-off grade, account for the internal dilution sources, but is not considered in the dilution estimate; however, the PCSLC model does report internal dilution for separate consideration. Material outside the ore development and stope shapes provide the sources for the external dilution. External dilution sources have been evaluated and included in the dilution estimate. By definition, external dilution is the material that did not originate from within the designed stope shape and may carry grades that are higher than the cut-off grade; the term “dilution” does not necessarily imply “low grade” or “barren” material.

Low grade and barren material within the planned stope shapes have been assigned a mining recovery of 40% where only the swell material is mined. This strategy is intended to improve the overall grade factor by maintaining a dilution blanket of low-grade material above the production area. This material is considered in the material flow and contributes to the tonnes mined in the LOM plan.

The stopes have been designed within the limits of the defined mineralization after applying the 1.5 g/t Au in-situ cut-off grade (refer to Table 14.7). Stope wall dilution is mainly from the mineralized area surrounding the mine design.

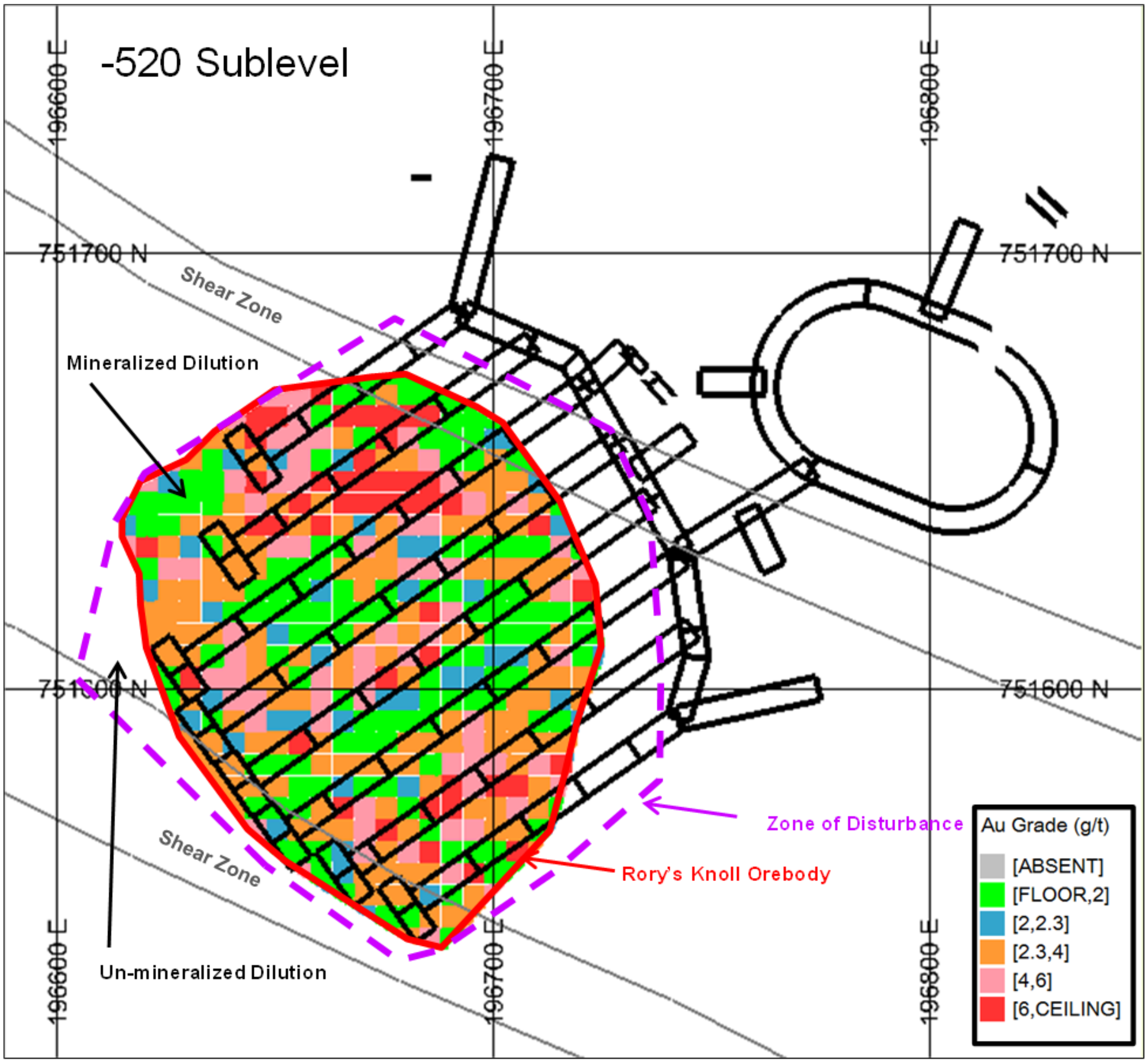
External dilution for the open benching and SLR mining methods includes dilution from mineralized material surrounding the stopes, low and zero grade dilution within the modelled zone of disturbance, and material within the pit walls that may eventually slough into the void opened up by underground mining. The final dilution is determined by the assumed PCSLC mixing parameters and the surrounding dilution model.

External dilution has been calculated from the zone of disturbance (or relaxation) created by mining Rory’s Knoll to a depth of -770 mRL. Numerical modeling of this zone was used to verify the amount of wall dilution. Figure 14.5 illustrates the zone of relaxation created around the underground production stopes, on a representative mining sublevel that was modelled to estimate dilution grades and tonnages.

The PCSLC model tracked three separate sources of dilution:

- model blocks within the zone of disturbance wireframes were flagged as sources of disturbance dilution;
- model blocks between the zones of disturbance designed stope shapes and were flagged as sources of mineralized dilution;
- model blocks within a cone extending between -245 mRL and the -120 m pit bench were flagged as sources of pit wall dilution. This material could potentially subside into the cave as underground extraction removes material confining these walls.

Figure 14.6 shows the selection of block model blocks for dilution color coded by the type of dilution.



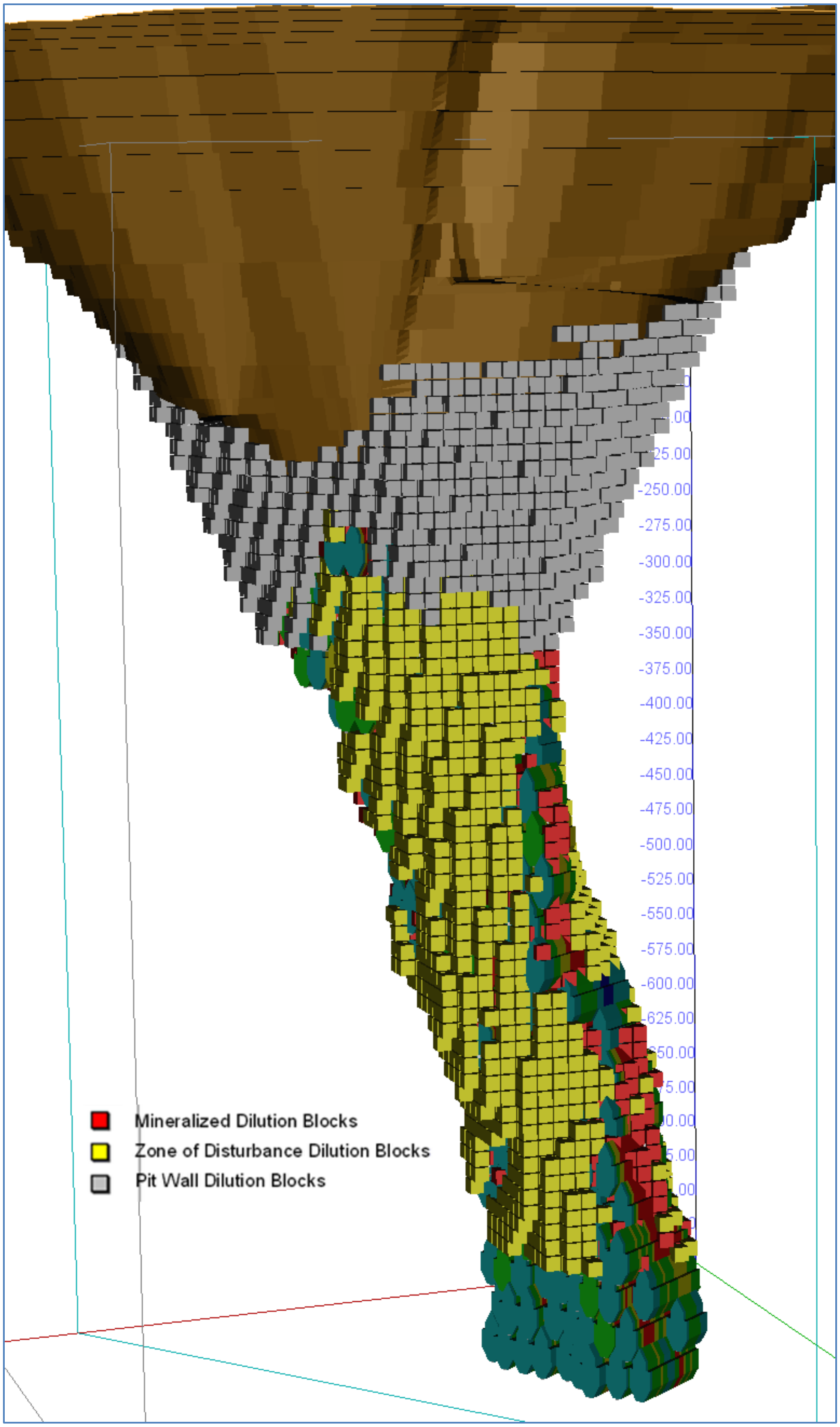
Source: SRK, 2017

Figure 14.5: Rory's Knoll: Typical zone of disturbance (plan view)

The PCSLC model tracked three separate sources of external dilution:

- model blocks within the zone of disturbance wireframes were flagged as sources of disturbance dilution;
- model blocks tangential to the stope shapes and above cut-off were flagged as sources of mineralized dilution;
- model blocks within a cone extending between -320 mRL and the -45 m pit bench were flagged as sources of pit wall dilution. This material could potentially subside into the cave as underground extraction removes material confining these walls.

Figure 14.6 shows the selection of block model blocks for dilution color coded by the type of dilution.



Source: SRK, 2017

Figure 14.6: Rory's Knoll: PCSLC dilution blocks (long section)

These three sources of dilution take their grades and tonnages directly from the PCSLC remnant block model; the remnant block model is a subset of the original block model that has been depleted of the open pit and the underground designs. This approach ensures no material is double counted in both stope ore and dilution material while still respecting the mineralized values within the potential external dilution. The tonnages and grades of the three external dilution sources made available to the PCSLC model are presented in Table 14.8.

Table 14.8: Sources of dilution in PCSLC Model

Source of Dilution	Quantity (Mt)	Grade (g/t Au)	Contained Metal (k oz Au)
Pit Wall	1.56	0.61	30.5
Disturbance Zone	2.35	0.66	50.0
Mineralized	1.22	2.54	99.5
Internal	0.31	1.0	10.0

It is recognized that the mineralized dilution grade is higher than the design cut-off grade of 1.50 g/t Au. This occurs as a result of using a standard stope design geometry across the width of the orebody. Some perimeter stopes were rejected as a result of lacking continuity and size to justify developing the drawpoint, thus, the grade in these rejected stopes drives the grade of the mineralized dilution. In practice, there is opportunity to modify the design of the perimeter stopes to include this material once further infill drilling and ore development has been completed. With the PCSLC model, much of this additional mineralization is still able to be recovered as mineralized dilution.

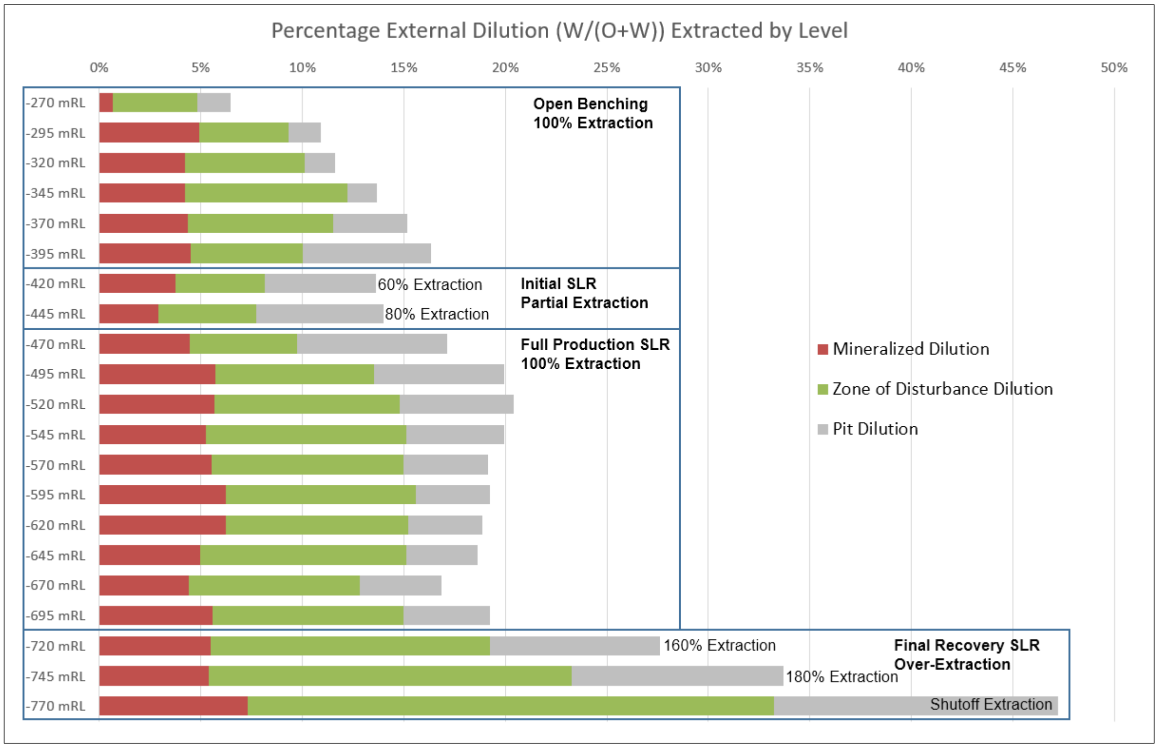
It should be noted that overall dilution is lower on the -420 mRL (60% extraction ratio) and -445 mRL (80% extraction ratio) where the SLR mining commences. The lower ore extraction is required in order to convert from the open benching to SLR mining method where the drawpoints are covered by overlying blasted and sloughed material.

The standard extraction ratio for the open benching and SLR stopes is 100%.

High extraction ratios are used in the lower levels as the mining cycle approaches the bottom of the planned extraction depth. The purpose of high extraction ratio is to gradually recover the mineralised material left behind on previous levels, which contains material above the cut-off grade.

Finally, on the -770 mRL (last production level of the planned mine design), many of the drawpoints are extracted until the cut-off grade is reached or greater than 200% of in-situ tonnes are extracted, to recover additional ore left behind from other levels in the dilution blanket; this results in very high dilution relative to all previous levels. Over-drawing the last production level is a normal procedure in SLR and SLC mines to recover material remaining in the mined out areas above.

Figure 14.7 presents the estimated quantity of each type of dilution occurring at each mining level in the final PCSLC run.



Source: SRK, 2017

Figure 14.7: Rory's Knoll: Dilution modelled in PCSLC

Initially, higher dilution is expected with the completion of the initial open benching sublevels as a result of blast damage to the open pit walls. In addition, uneven open pit highwall and stope geometry may result in waste wedges being produced. Lower dilution entry is estimated during the subsequent phase of open benching and during the initial stages of SLR mining where reduced extraction will result in limited dilution entry. Dilution will increase as SLR mining progresses down to the -770 mRL and the disturbance from stress relaxation increases.

The PCSLC model has been configured for open benching sublevels to achieve higher primary extraction, while SLR levels have reduced primary extraction, on the basis of the dilution blanket contributing dilution during extraction during SLR mining.

The following summarizes the dilution and recovery estimates generated by the PCSLC model:

- Total LOM ore recovery is 89% of the contained metal within the in-situ stope design;
- Average LOM internal and external dilution grade has been estimated to be 1.10 g/t Au;
- Average LOM internal and external dilution tonnes are estimated to be 23%.

When dilution is included in the total, the equivalent of 89% of in-situ stope design ounces are recovered. Table 14.9 shows the calculation for contained metal.

Table 14.9: Rory's Knoll: Contained metal within the total design

Description	Contained Metal (k oz Au)
Insitu Stope Design – No External Dilution	1,330
Recovered from Stope Design (93% of Stope Design)	1,234
External Dilution (8% of stope design)	106
Total Recovered (101% of stope design)	**1,341**

14.4.2 Satellite Deposits

The satellite deposits collectively comprise Aleck Hill, Aleck Hill North, Walcott Hill, Walcott Hill East, Mad Kiss, Mad Kiss South, Mad Kiss West, and Rory's Knoll East. Only Aleck Hill and Mad Kiss were able demonstrate an economically viable mine plan.

14.4.2.1 Orebody Description

The underground extension of Aleck Hill mineralisation is located to the north east of the Aleck Hill open pit. The mineralisation at Aleck Hill consists of series of sub-parallel gold lenses strike southeast at 150° and dip at approximately 120° southwest, while the two main shear zones strike to the southeast (at approximately 155°) and dip to the southwest steeply (80°).

The auriferous zones consist of a series of sub-parallel lenses of variable width (2.5 m to about 20 m, with occasional 30 m widths).

The overall continuity of Aleck Hill mineralization above a cut-off grade of 2.50 g/t Au (target mineralization) is classified as fair, however the continuity improves in the northern extents where underground mining has been planned. Some areas exhibit complete mixing of resource blocks above and below cut-off, while other areas have a clearer separation of mineralization above and below cut-off.

Ground conditions are expected to be good in the mafics, tolerating an open span of up to 20 m. Open spans in the sericite will be limited to about 15 m.

The underground extension of Mad Kiss comprises a series of sub-parallel auriferous lenses, striking southwest at ~250° and dipping at ~70° north while the intercepting shear zone strikes northwest (between 290° and 305°) and dip steeply to the northeast (between 70° and 85°).

The auriferous zones are distributed in a series of sub-parallel lenses of variable width (0.5 m to ~9 m on average).

The continuity of the mineralization above a cut-off grade of 2.50 g/t Au (target mineralization) can be described as good. Some areas are very continuous, especially between -130 mRL and -300 mRL on the west side of the deposit.

14.4.2.2 Cut-off Grade

A cut-off grade of 2.40 g/t Au was used to define the Mineral Reserves for the satellite deposits.

This feasibility study update used a gold price of $1,200/oz to reflect market conditions. SRK prepared a mining cost model based on:

- GGI supplied inputs,
- External inputs derived from contractor costs, and SRK's experience on similar projects and operating mines.

Table 14.10 shows the cut-off grade calculation.

Table 14.10: Satellite Deposits: cut-off grade estimate

Parameter	Unit	Value
Site operating cost		
Underground mining	$/t	43
Processing	$/t	14.5
G&A	$/t	8
Total site operating cost	**$/t**	**66.5**
Gold selling price	$/oz	1,200
Au payable	%	100%
Refining charges	$/oz	3
Royalty		
Applicable gold price	$/oz	1,000
Royalty rate	%	8%
Value of Au in doré	$/oz	1,100
	$/g	35.41
Processing recovery	%	94%
Value of Au in plant feed	$/g	33.28
Plant feed grade (diluted COG)	**g/t Au**	**1.97**
External dilution at grade	g/t Au	1
In-situ Au COG	**g/t Au**	**2.41**
Development COG	**g/t Au**	**0.72**

The proportion of dilution is calculated as the ratio of waste/(ore plus waste) and typically expressed as a percentage.

14.4.2.3 Mining Shapes

SRK used Mineable Shape Optimizer™ software to identify the zones within the satellite deposits that satisfied the cut-off grade, dilution and operational design criteria for each mining block, using the long hole open stoping mining method. Where required, the shapes were manually optimized to further reduce the amount of sub-economic material within the stope shape. Geotechnical recommendations for the stope designs are outlined in Section 15.2.5 (check this). In general, the mining method utilised a 20 m level interval and a 20 m t o 25 m strike length. Stopes were maimed as either transverse or longitudinal, depending on the local orebody geometry.

14.4.2.4 Dilution and Recovery Estimates

Mining dilution is given as the ratio of "waste/(ore + waste)". Mining dilution arising from the footwall and hanging wall of the stopes, was included during in the MSO assessment, as the width of the mineralised zones did not meet the minimum mining width requirement 2.0 m. Mining dilution arising from the backfill and stope end walls was allocated in EPS during the production scheduling. Backfill dilution was ascribed 0.3 m on the adjacent wall and stope end wall dilution was ascribed 0.3 m on the adjacent wall.

Mining recovery was also allocated in EPS during the production scheduling, at 95% transverse stopes and 85% longitudinal stopes.

15 Mining Methods

15.1 Summary

This study is an update of the January 2016 feasibility-level mining evaluation of the Aurora open pit and underground deposits (MMC, 2016) and includes the following scope for the mining aspects:

- Delay Underground Mining as long as feasibly possible;
- $1,200 per ounce gold selling price;
- Mining method selection;
- Cut-off grade estimation;
- Estimate modifying factors for dilution and recovery;
- Development design;
- Stope design;
- Ground support design;
- Design subsidence management strategy;
- Estimate productivity and mining rates;
- Create production schedule;
- Design primary ventilation strategy;
- Estimate mobile equipment fleet requirements and operating hours;
- Estimate personnel requirements; and
- Estimate mine operating and capital costs.

15.1.1 Previous Studies

The initial feasibility study for the Aurora project was completed by SRK in 2012 (SRK, 2012). It concluded with mining Rory's Knoll and Aleck Hill deposits by open pit (with large open pit equipment) and the Rory's Knoll East, Walcott Hill East, Walcott Hill, Mad Kiss and Aleck Hill deposits by underground, via shaft and decline access for Rory's Knoll. The underground mining method was transverse and longitudinal LHOS with paste backfill. The gold price used in 2012 feasibility study was $1,200/oz.

SRK updated the 2013 feasibility study (Tetra Tech, 2013) and concluded on the mining of the Rory's Knoll, Aleck Hill, Aleck Hill North, Mad Kiss and Walcott Hill deposits by open pit and evaluated a mass mining (with no backfill) method for underground at a lower capital case. Only Rory's Knoll was judged amendable to underground mining by mass mining methods. The cross over pit was completed on Rory's Knoll and the underground design chose an open benching/SLR mining approach via decline access only. The gold price used in SRK (Tetra Tech, 2013) was $1,300/oz.

This feasibility study updates the following aspects of SRK (MMC, 2016):

- Reflect the change in the mining strategy, and, on the basis of a higher gold price, to mine larger open pits, compared to the 2016 FS and defer the underground mining. No open pit/underground crossover studies have been undertaken.

- Update the reserves at a higher gold price ($1,200/oz vs. $1,000/oz).

Hence, this study builds on the work carried out in the 2016 feasibility study update (MMC, 2016).

15.2 Geotechnical Assessment

15.2.1 Geotechnical Assessment Update Summary

The geotechnical assessment within this document incorporates the previous geotechnical data (AMEC, 2009; SRK, 2012; Tetra Tech, 2013; SRK, 2013; MMC, 2015) which have been confirmed with observations made during a site visit by SRK in October, 2016. For a full review of the data used in the geotechnical assessment, refer to the 2015 updated Feasibility Study (Metal Mining Consultants, 2016). Site observations from the 2016 visit are limited to rock wall control performance and saprolite stability. No additional laboratory data or core logging was collected.

Additional analyses were conducted on the existing geotechnical data with respect to the open pit design. This was done due to material change in the geotechnical domain model used for the open pit design (as described in Section 15.2.3). No additional analyses were conducted for the underground design section as previous work had already used the mine geotechnical model.

15.2.2 Saprolite Geotechnical Characterization Review

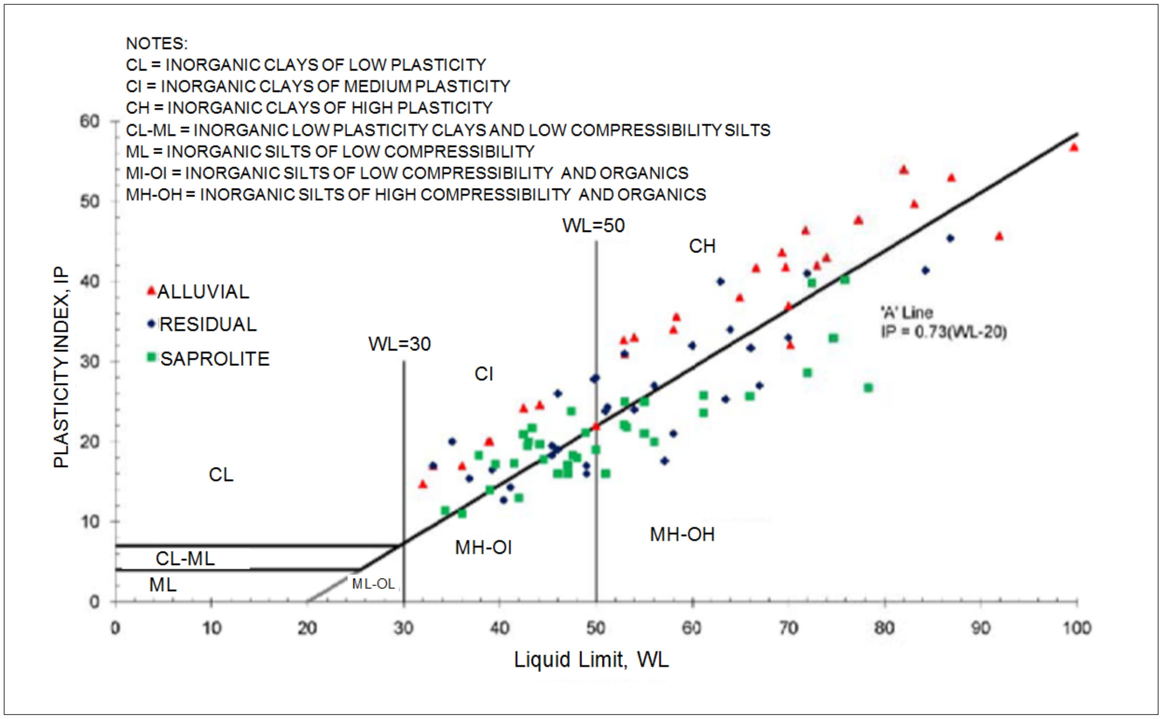
The overburden stratigraphy in the area of the open pits has previously been characterized (AMEC, 2010) as containing 0 m to 3.4 m of Alluvium, 0 m to 6.3 m of Residual Soil, and 7.6 m to 47.5 m of Saprolite. Within this document, only the geotechnical characteristics of saprolite are reviewed as saprolite is expected to have the major influence on stability and design of the open pit within the overburden. Additional background information on the alluvium and residual soil can be located in the 2010 Geotechnical Investigations for Aurora Gold Project (AMEC, 2010).

The saprolite in the open pit areas is summarized as medium to high plastic Clayey-Silt with the following geotechnical characteristics (Figure 15.1 and Table 15.1):

- Natural moisture content 16 to 53%
- Liquid Limit 42 to 72%
- Plastic Limit 22 to 43%
- Plasticity Index 16 to 29%
- Grain size distribution:

- o Gravel 0 to 5%
- o Sand 8 to 88%
- o Silt 4 to 65%
- o Clay 9 to 41%

The SPT N-values ranged generally from 7 to greater than 50 (average 40). The vane tests indicated undrained shear strengths ranging from 124 to 254 kPa. Based on these results, the Saprolite is considered to have a very stiff to hard consistency.



Source: AMEC, 2010

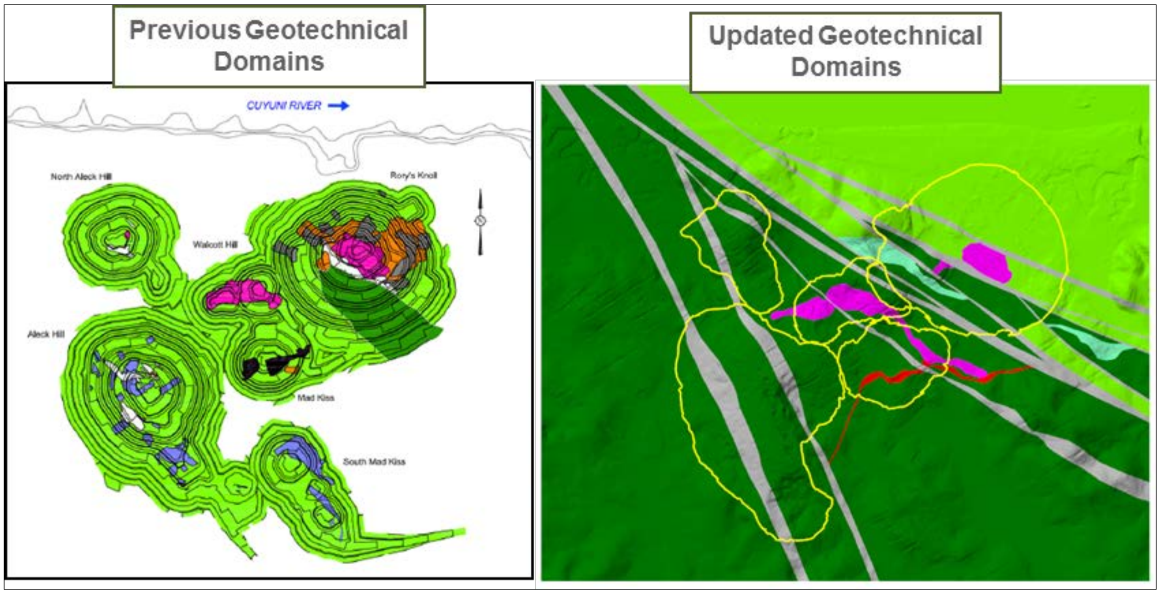
Figure 15.1: Saprolite (green) plasticity chart (AMEC, 2010)

Table 15.1: Vane shear strength and standard penetration test results (Amec, 2010)

Soil type		Vane Shear Strength						Standard Penetration Blow Count ('N-value')		
		Peak (kPa)			Remoulded (kPa)					
		Maximum	Minimum	Average	Maximum	Minimum	Average	Maximum	Minimum	Average
Alluvial	River Dyke	282	59	169	156	14	80	29	0	10
	Open Pit	155	113	134	40	9	26	17	4	8
	Tailings Management Area	-	-	-	-	-	-	15	3	10
	Clarification Pond	-	-	-	-	-	-	19	3	10
	Water Management Pond	-	-	-	-	-	-	20	5	11
Residual	River Dyke	282	28	142	190	14	65	35	8	20
	Open Pit	245	99	161	113	28	57	>50	5	31
	Tailings Management Area	-	-	-	-	-	-	42	4	15
	Clarification Pond	-	-	-	-	-	-	21	7	14
	Water Management Pond	258	54	146	178	24	61	25	4	13
Saprolite	River Dyke	256	256	256	181	181	181	>50	25	>50
	Open Pit	254	124	210	155	56	100	>50	7	40
	Tailings Management Area	-	-	-	-	-	-	>50	7	17
	Clarification Pond	-	-	-	-	-	-	>50	11	25
	Water Management Pond	297	74	184	193	24	85	>50	3	26

15.2.3 Rock Geotechnical Characterization Review

As noted in the 2015 Feasibility Update (MMC, 2015), the open pit geotechnical domains required re-alignment to agree with the accepted lithological model currently used at the mine and as used in the underground design (Tetra Tech, 2013). Previous open pit geotechnical domains were based upon an outdated geological model. As mentioned in Section 15.2.1, no additional geotechnical data was collected as part of this update, however, the data previously collected was reviewed by reorganizing the data into the updated geotechnical domains (Figure 15.2). This reorganization of intact rock and rock mass properties, such as joint and foliation orientation, is the basis for the updated open pit geotechnical recommendations.



Source: SRK, 2017

Figure 15.2: Updated geotechnical domains (right) as compared to previous geotechnical domains (left)

15.2.3.1 Intact Rock Strength

A summary of the material strengths for the geotechnical domains in rock at Rory’s Knoll (RK) and Aleck Hill (AH) based on laboratory testing are given in Table 15.2 and Table 15.3, respectively. No laboratory testing for the geotechnical units defined were available for Mad Kiss, therefore it is assumed that the Aleck Hill and Mad Kiss strengths are similar based on photo evaluation. A full description regarding the determination of the material strength of the geotechnical units is available in the NI 43-101 Technical Report Updated Feasibility Study (Tetra Tech, 2013). Rock strength is given for both the intact (or matrix) of the rock as well as strength along the foliation within the rock. Intact strength is primarily determined by uniaxial compressive strength (UCS) tests. Foliation strength is noted to be roughly half to a third of the intact strength and was determined by review of the samples which had broken along foliation. The foliation strength is particularly important within the Mafics domain (moderately anisotropic) and the shear zone domain (strongly anisotropic).

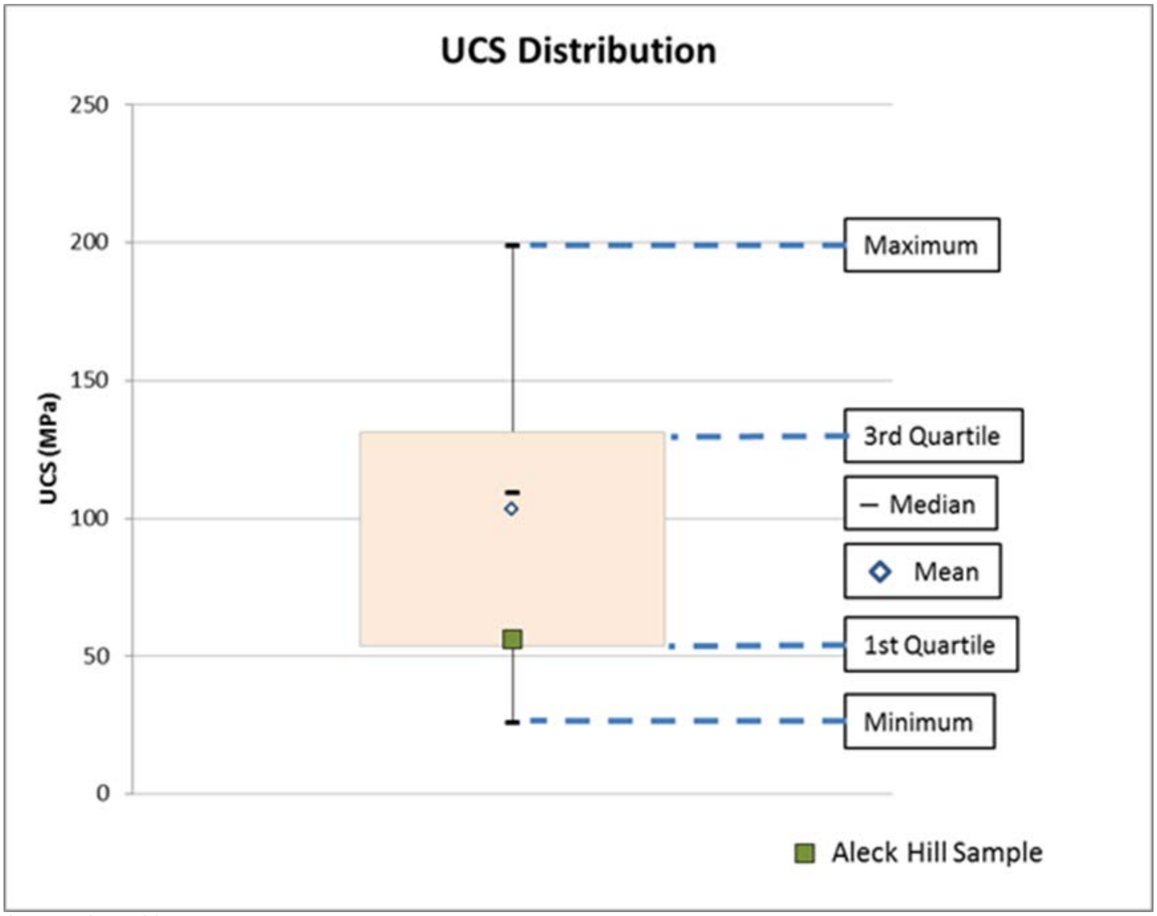
Table 15.2: Rory’s Knoll: Rock parameters for the geotechnical domains

Zone	Unit	Density Avg.	Number of UCS Tests	UCS (Intact) Avg.	UCS (Foliation) Avg.	Young's Mod. E	Poissons Ratio	Tensile Avg.
		(kg/m^3)		(MPa)	(MPa)	(GPa)		(MPa)
RK	TON	2850	7	150	70	74	0.21	10
	SCS	2800	24	105	35	31	0.34	7
	Mafics	2800	28	125	70	65	0.18	10
	Interbedded	2750	21	110	45	66	0.27	12

Table 15.3: Aleck Hill: Rock parameters for the geotechnical domains

Zone	Unit	Density Avg. (kg/m^3)	Number of UCS Tests	UCS (Intact) Avg. (MPa)	UCS (Foliation) Avg. (MPa)	Young's Mod. E (GPa)	Poissons Ratio	Tensile Avg. (MPa)
AH	Mafics	2830	15	118	56	--	--	--
	SCS	2800	1	51	27	--	--	--

The total number of laboratory test results available for the characterization of the satellite deposits is much lower than Rory's Knoll as the primary focus of previous investigations has been the main deposit at Rory's Knoll. Within the Mafics domain, sufficient samples have been tested with strength values at Aleck Hill are similar to those at Rory's Knoll; however, only one sample has been tested within the shear zone domain. With only one sample, it is not possible to determine if the value is an average value for the unit at that location or if it falls within the lower end of the natural distribution of strength established at Rory's Knoll. Figure 15.3 illustrates the uniaxial compressive strength (UCS) distribution of the shear zone domain at Rory's Knoll with the minimum, first quartile, mean, median, third quartile and maximum values of the samples tested. The value from the single sample tested from the Aleck Hill area is shown to be at the lower end of the values of the samples tested at Rory's Knoll, near the first quartile value. The current geotechnical analysis honors the lab test results for the satellite deposits; however, the strength of the shear zone domain in the satellite deposit areas is potentially stronger. Further geotechnical investigations may allow for a less conservative underground design within the shear zone domain unit.



Source: SRK, 2017

Figure 15.3: Comparison of UCS range for the sericite shear at Aleck Hill & Rory's Knoll

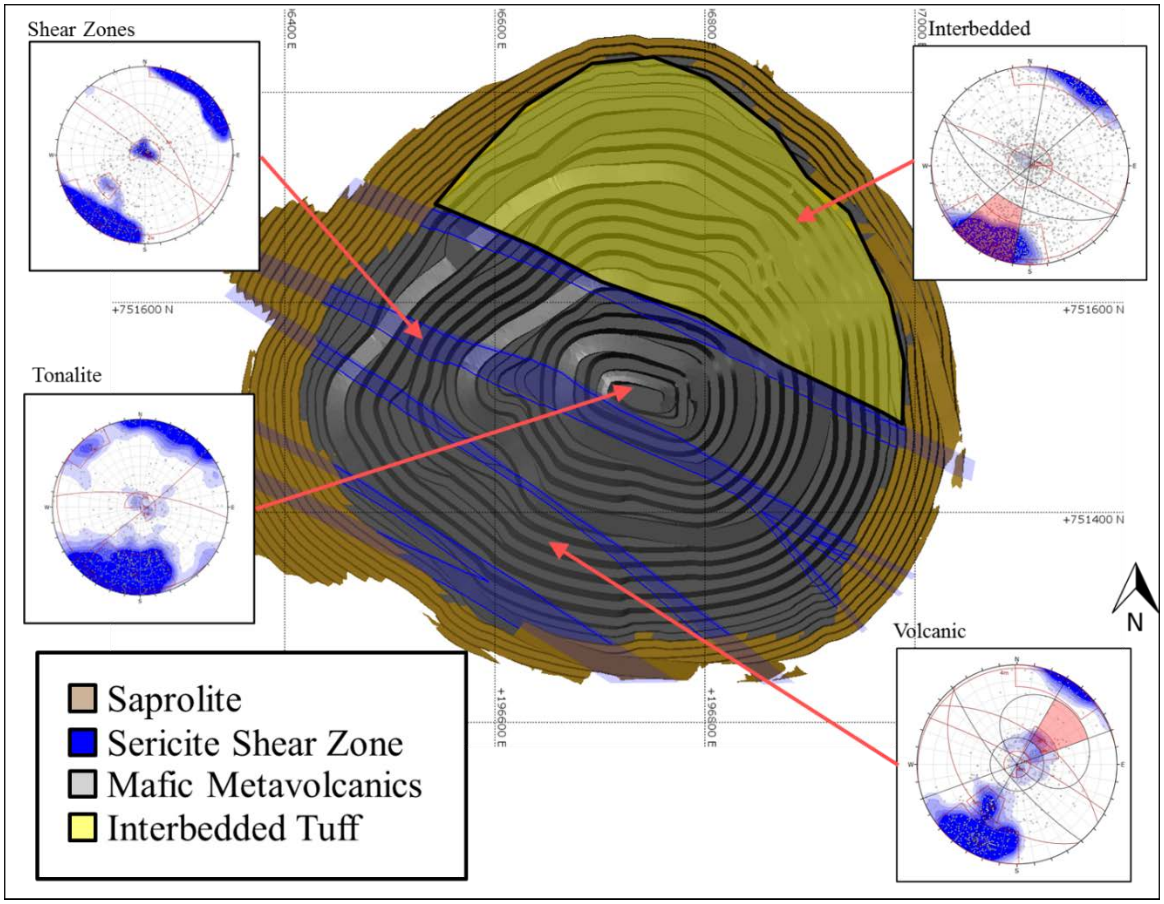
15.2.3.2 Major Joint Sets and Major Structural Features

The ore zone at Rory's Knoll, which is hosted in the tonalite intrusive, is found to dip at around 80° to 85° forming a pipe-shaped structure. Aleck Hill and the other satellite pits are hosted entirely in volcanic sediments (Mafics) located in and around several northwest southeast striking features of strongly foliated sericite shear structures. The shears are located on the northeast and southwest edges of Rory's Knoll and run parallel to sub-parallel to the satellite deposits. As part of the updated geological model, no distinct structural features (i.e. faults) have been identified. The area is interpreted to have undergone primarily ductile deformations although weak structures are expected to run parallel to the main foliation.

Analysis of the oriented joints showed relatively consistent joint orientations (sub-vertical foliation and sub-horizontal joint set) for each geotechnical lithological unit. As such, the major domains from each borehole could be grouped together for each ore zone. The major joints sets, Geologic Strength Index (GSI), and average intact strength of the domains are summarized in Table 15.4 for Rory's Knoll and Aleck Hill. Figure 15.4 and Figure 15.5 illustrate the joint sets identified at the Rory's Knoll and Aleck Hill, respectively.

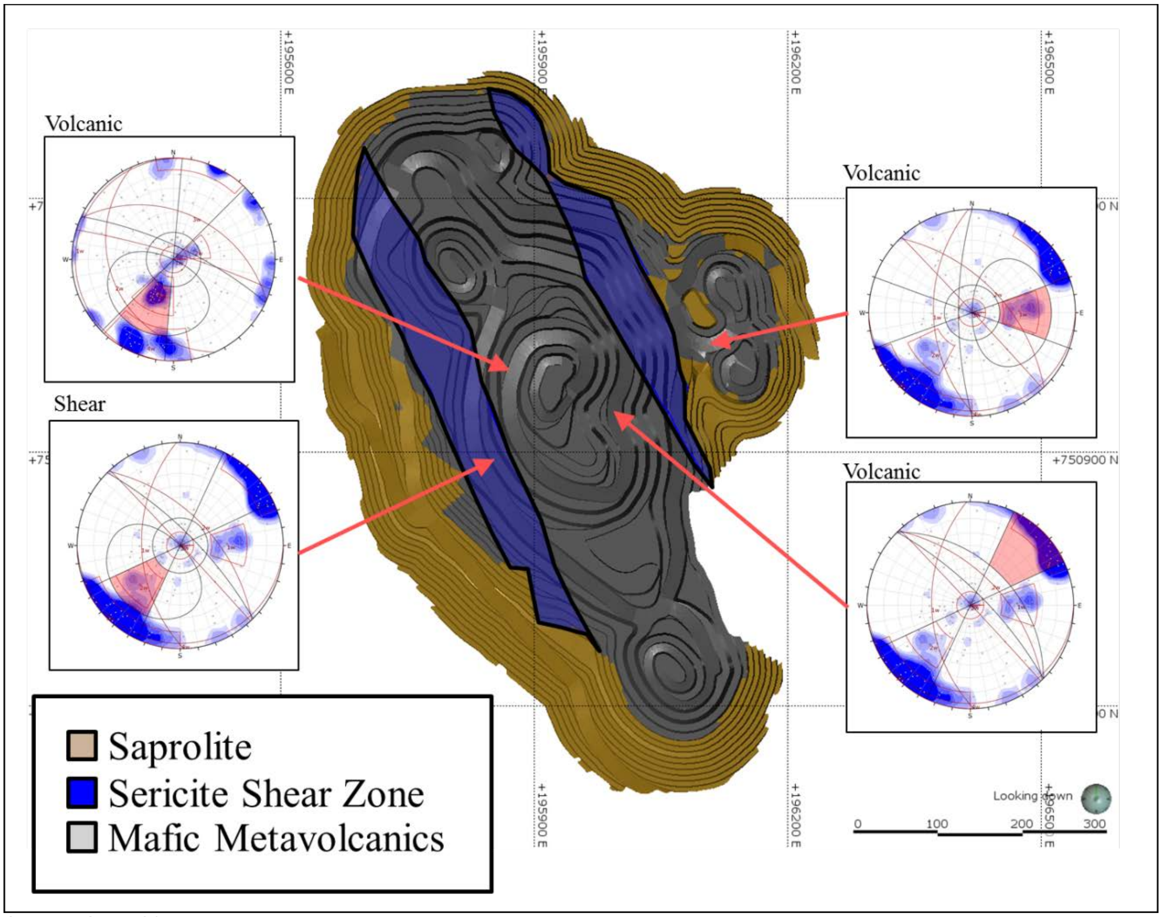
Table 15.4: Summary of rock mass parameters major joint sets per domain

Domain	GSI	Average UCS (MPa)	Joint Set	Dip	Dip Direction
Rory's Knoll (Mafic Metavolcanics)	65	122	**1 (Vertical - Foliation)**	**85**	**30**
			2 (Intermediate)	**50**	**35**
			3 (Horizontal)	5	350
			4 (Intermediate - Minor)	30	230
Rory's Knoll (Interbedded)	55	118	1 (Vertical - Foliation)	80	30
			2 (Horizontal)	5	180
Rory's Knoll (Shear Zones)	45	62	1 (Vertical - Foliation)	80	30
			2 (Horizontal)	5	180
Aleck Hill (Shear Zones)	50	62	**1 (Vertical - Foliation)**	**90**	**25**
			2 (Intermediate)	50	270
			3 (Intermediate, minor)	55	45
			4 (Horizontal)	5	180
Aleck Hill (Mafic Metavolcanics)	65	122	**1 (Vertical - Foliation)**	85	20
			2 (Intermediate)	40	25
			3 (Horizontal)	5	275
			4 (Intermediate - Minor)	25	250



Source: SRK, 2017

Figure 15.4: Distribution of joint sets within Rory's Knoll pit area



Source: SRK, 2017

Figure 15.5: Distribution of joint sets within Aleck Hill pit area

15.2.3.3 Rock Mass Characterization

The rock mass quality of the domains that comprise the open pits were determined from the geotechnical logging of oriented core from the boreholes as well as laboratory testing. The rock mass characterization incorporated the Rock Quality Designation (RQD), NGI-Q classification system (Barton et. al. 1974), the Rock Mass Rating (RMR) classification system (Laubscher and Page 1990), the Mining Rock Mass Rating (MRMR) System (Laubscher and Jakubec 2001), and the Geological Strength Index (GSI) classification system (Hoek, et. al. 1995).

Rock mass parameters for the Mafics and Shear Zone units around the satellite deposits were estimated by re-assigning the available geotechnical borehole logs previously defined geotechnical domains (AMEC, 2009) into the updated geotechnical domains.

The rock mass quality is good with the best quality rock mass being found within the Tonalite domain, followed closely by the Mafics and Interbedded units. Table 15.5 is based on the previously defined geotechnical domains. The updated geotechnical units of Mafics and Interbedded Volcanics include the volcanic sediments, tuffs, and metavolcanics which are all of a similar rock mass quality and are classed as good. The lowest quality rock mass is found in the shear zone domain, which is still classified mainly as fair, but have a tendency for lower RQD values at the contacts. Aleck Hill, which is primarily located in the Mafics appears to have a consistent rock mass quality to Rory's Knoll.

Table 15.5: Statistical analysis of rock mass properties from core logging

Domain	Average Intact UCS (MPa)	Average Foliation UCS (MPa)	RQD (%)	Q (estimated RMR=9lnQ+44)	RMR90	Rock Quality	MRMR	FF/m
Interbedded	110	45	94	7.5 – 18.0	65-70	Good	69	1.4
Sericite	105	35	96	4.0 – 5.9	55-60	Fair	60	1.8
Tonalite	150	70	98	7.5 – 18.0	70-75	Good	68	1.4
Mafic	125	70	97	7.5 – 18.0	65-70	Good	69	0.9

15.2.4 In-situ Stress Estimation

At present, in-situ stress measurements have not been undertaken in Guyana. Based on the world stress map (Heidbach et. al. 2008), the orientation of the maximum principal in-situ stress is expected to be horizontal and in the North-South direction based on moment tensor analysis of regional earthquakes. Assumptions have been made for the regional in-situ stress regime based on experiences in similar geological environments and are used as the basis for open pit and underground numerical modelling and stability analysis. Note, the mean surface elevation is considered to be +80 mRL. In the absence of the stress measurements, a sensitivity analyses was performed to determine the potential impact on the stability of the excavations.

The stress regime developed for the Rory's Knoll open pit and underground mine design is as follows:

- $\sigma 1 = \sigma H = k * \sigma v$; where $k1 = \sigma 1/\sigma 3$ (constant) = 1.5 ; oriented N-S
- $\sigma 2 = \sigma h = k * \sigma v$; where $k2 = \sigma 2/\sigma 3$ (constant) = 1.2 ; oriented E-W
- $\sigma 3 = \sigma v = 0.00275 * 9.81 * \text{Depth}$; oriented vertical

This ratio is mid-way between the typical Canadian stress ratio of 1.9 and the Mexican stress ratio of 1.1, and is a reasonable approximation at this time, as the mining method is less sensitive to stress. However, stress measurements are recommended at the next stage of construction to confirm assumptions.

15.2.5 Open Pit Mining Geotechnical

The open pit design criteria and parameters previously determined (Tetra Tech, 2013; MMC, 2016) were reviewed prior to conducting an update on the open pit design. Pit slope design parameters for this study were based on both on kinematic stability of the bench design using the principal joint sets within the updated geotechnical domains and stability analysis of inter-ramp scale instability of toppling. As in the main dominant rock mass domains. Detailed geomechanical site investigations were performed for the main Aurora deposits of Rory's Knoll, Aleck Hill and Walcott Hill only. However, based on the general homogenous nature of the jointing in the volcanic sediments (the main country wall rock), these design recommendations have been applied to the other smaller satellite pits (Aleck Hill North, Mad Kiss and Mad Kiss South), and should be verified.

15.2.5.1 Saprolite Slope Design Recommendations

Saprolite slope design is based on the plasticity, friction angle, relict structures, and proximity to current drainage which may lead to excess pore water pressure. Often a range of stable bench face angles can be achieved and the design is based on a trade-off between slope stability and increased erosion. Unless distinct relict structures are present, the design should be kept as simple and repeatable as possible. Generally, relict structures at the Rory's Knoll area are believed to mimic the regional foliation which is steeply dipping to the northeast. Table 15.6 summarizes saprolite geotechnical parameters.

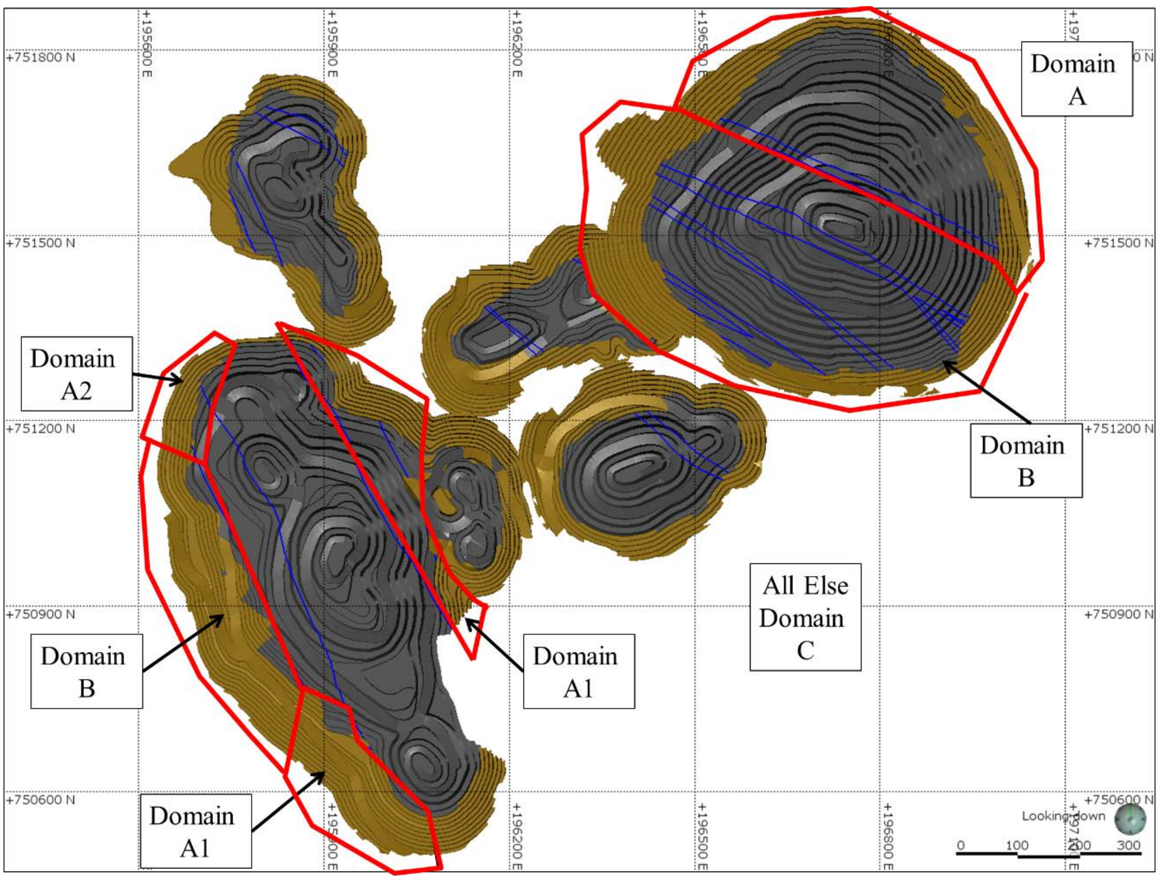


Figure 15.6: Overview of saprolite geotechnical domains

Table 15.6: Saprolite geotechnical design parameters

Pit	Material	Domain	BFA (°)	Bench Height (m)	Berm Width (m)	IRA (°)
Rory's Knoll	Saprolite	A	70	5	7	30
		B	70	5	6	33
Aleck Hill	Saprolite	A1	70	5	7	30
		A2	70	5	8	27
		B	70	5	6	33
		C	70	5	6	33
Walcott Hill	Saprolite	C	70	5	6	33
Mad Kiss	Saprolite	C	70	5	6	33
North Aleck	Saprolite	C	70	5	6	33

Within Rory's Knoll area, saprolite composed of Interbedded volcanic rocks and tuff were observed to erode more easily than saprolite composed of a metavolcanic protolith. The weaker saprolite in combination with the proximity to the Cuyanni River suggests a lower inter-ramp angle is required to ensure stability of the dike located at the crest of the slope.

Within the Aleck Hill area, relict structures within the saprolite in the slopes located directly above the shear zones are expected to create localized bench instability and increased erosion. Domain A1 considers the area of saprolite above the shear zones and near main access ramp. Domain A2 considers the area of the pit that runs in close proximity to the current creek location. It is expected that pore water pressures may be slightly higher in this area. If the pit is redesigned away from the creek location, Domain A2 should be revisited. Domain B considers the saprolite above the main access ramp on the southwest wall.

Maximum stack heights of 40m are recommended. If stack height exceed 40m, a 12m geotechnical berm is recommended.

15.2.5.2 Rock Slope Design Recommendations

Slope design recommendations are based on the current pit slope performance, bench-scale kinematic analyses, and the overall slope stability analyses.

Within Rory's Knoll area, slopes within Domain RKN2 were determined to be potentially susceptible to inter-ramp scale toppling (Figure 15.7). The strong foliation in combination with the proximity to the Cuyanni River suggests a lower inter-ramp angle is required to ensure stability of the dike located at the crest of the slope. Inter-ramp design angles of Domain RKS2 are limited by is joint set (J2 at 50°).

Within the Aleck Hill pit area, geotechnical data is limited to 4 geotechnical boreholes, which do not all intersect the current pit design (Figure 15.8). Current pit design parameters are limited by the presence of intermediate dipping joints at 50 to 56 degrees in the southwest and northeast walls, respectively.

Table 15.7 summarizes geotechnical parameters used for rock.

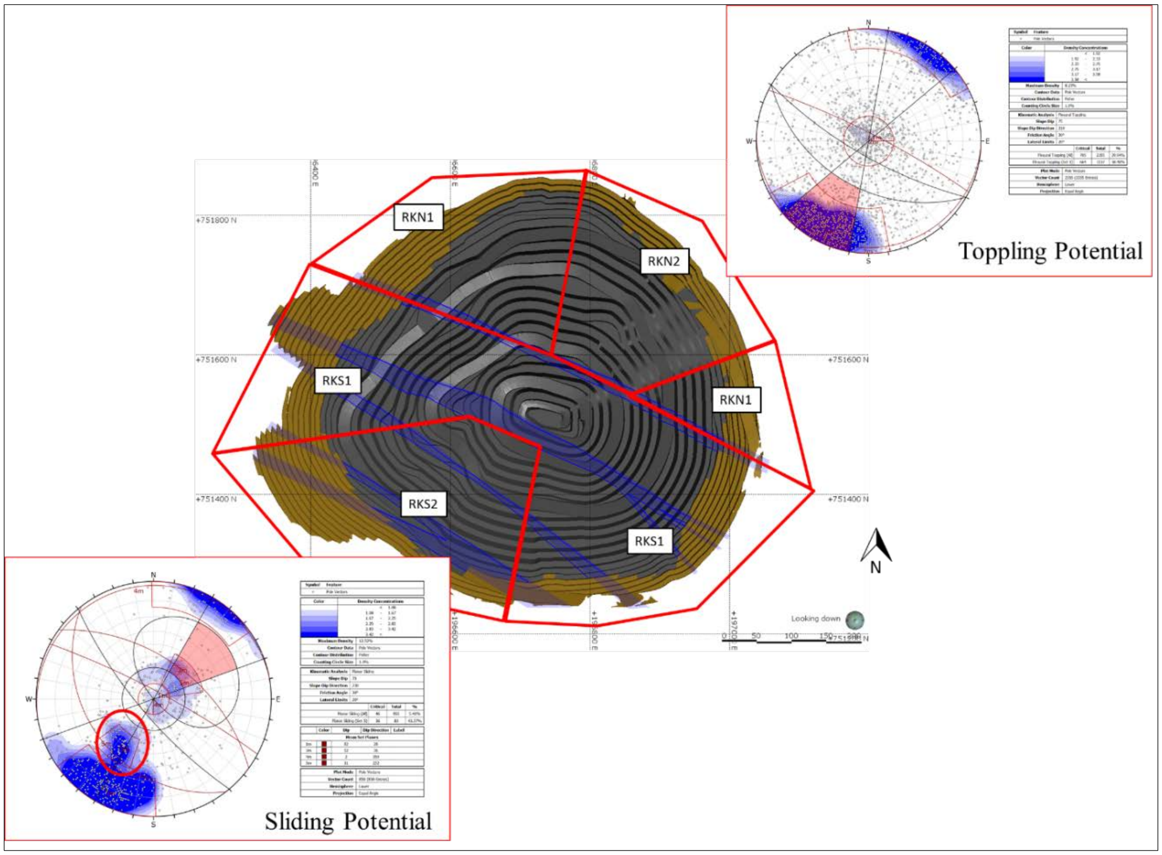
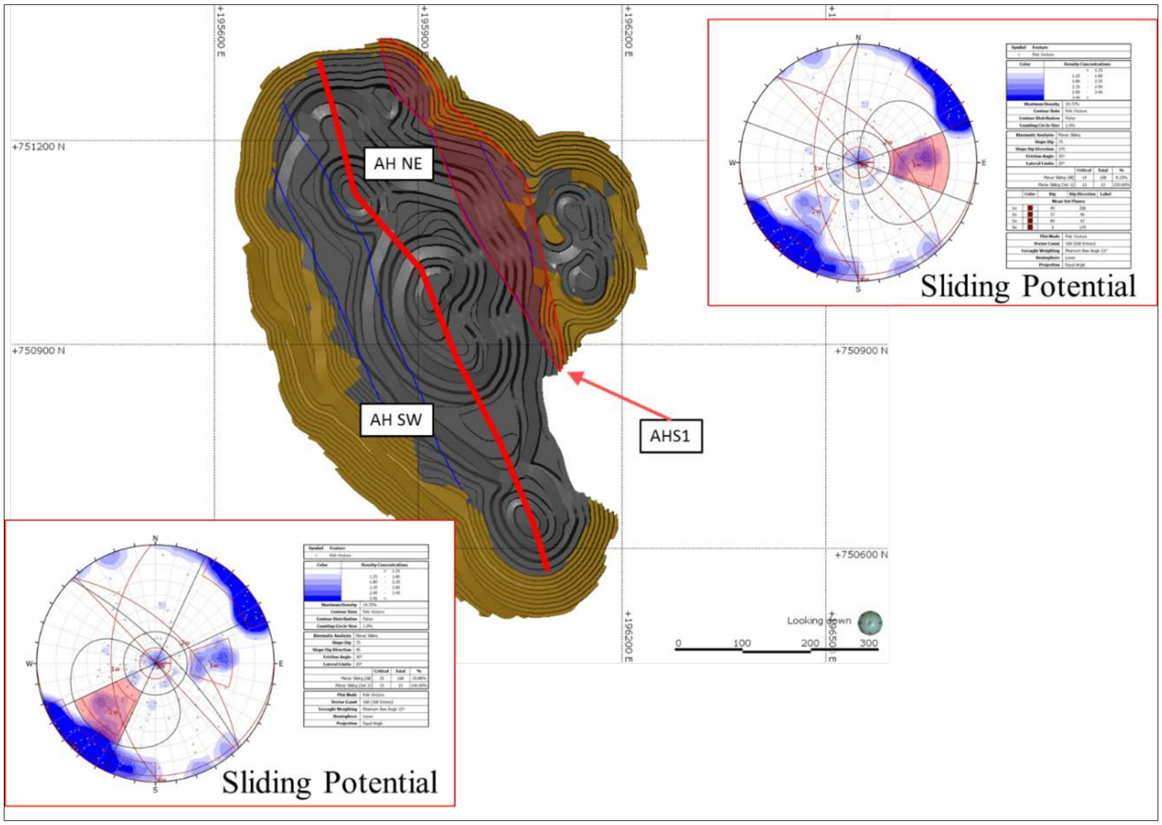


Figure 15.7: Rock geotechnical design domains for Rory's Knoll)



Source: SRK, 2017

Figure 15.8: Rock geotechnical design domains for Aleck Hill

Table 15.7: Rock geotechnical design parameters

Pit	Material	Domain	BFA (°)	Bench Height (m)	Berm Width (m)	IRA (°)
Rory's Knoll	Rock	RKN1	80	20.0	13.0	50
		RKN2	70	20.0	13.0	45
		RKS1	80	10.0	6.5	50
		RKS2	70	20.0	13.0	45
Aleck Hill	Rock	AH NE	80	20.0	10.0	56
		AH SW	70	20.0	14.0	49
		AHS1	75	20.0	14.0	49
Walcott Hill	Rock	RKS1	80	10.0	6.5	50
		RKS2	75	10.0	8.0	43
Mad Kiss	Rock	RKS1	80	10.0	6.5	50
		RKS2	75	10.0	8.0	43
North Aleck	Rock	RKS1	80	10.0	6.5	50
		RKS2	75	10.0	8.0	43

15.2.5.3 Slope Stability Analysis

The primary objectives of the overall stability analyses is to:

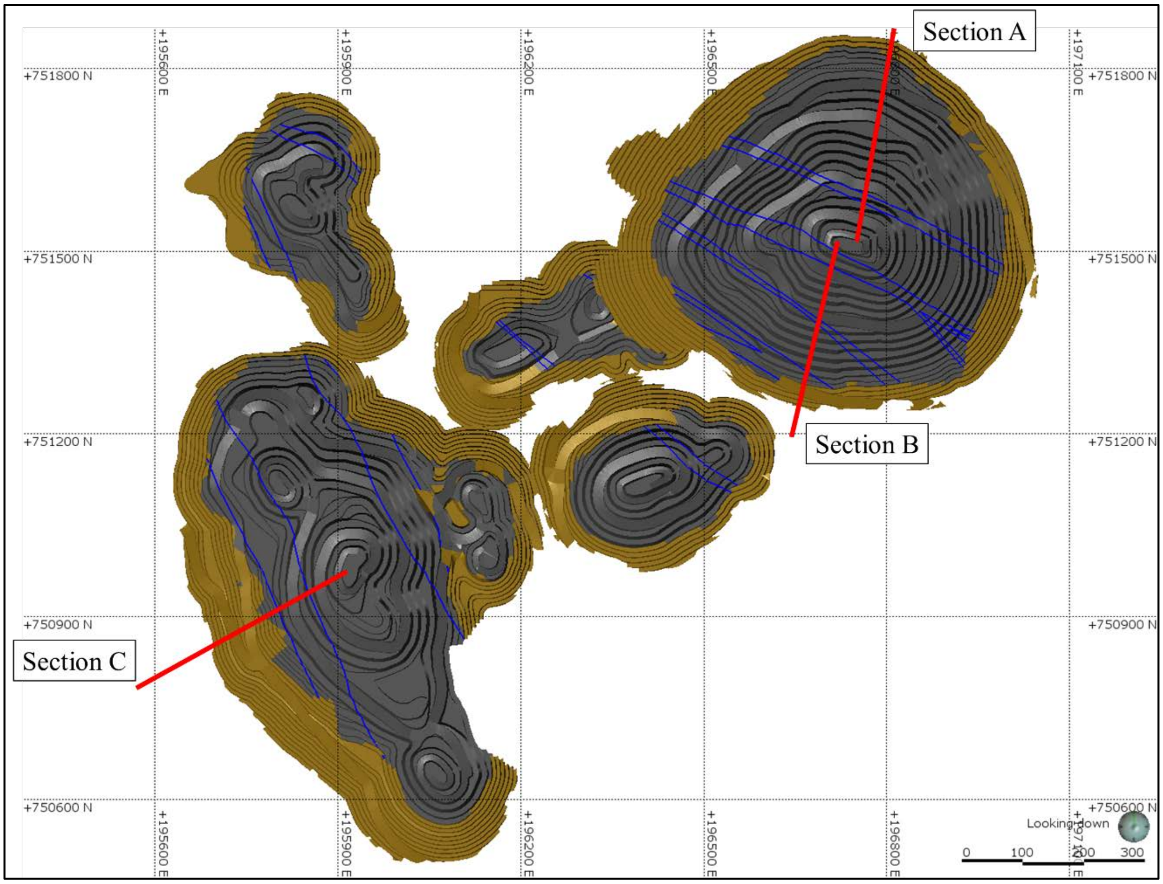
- Assess the sensitivity of each of the rock mass parameters, and its influence on the expected overall stability conditions for the pits;
- Confirm the optimized inter-ramp slope geometries as guided by kinematic analyses and the performance of the existing pit slopes; and
- Assess the impact of groundwater.

The stability modelling has been carried out rigorously to achieve the primary objectives. The modelling approach can be summarised into the following:

1. Evaluation of the sensitivity of the rock mass parameters.
2. Assessment of the expected stability for the existing pit design using the derived rock mass parameters.
3. Performing stability modelling to assess the potential optimization of the pits.

The inter-ramp slope angles were modified from the existing design to target stability conditions appropriate for optimization. The stability modelling was carried out using the Rocscience™ software program Phase2.

A total of three design sections where cut perpendicular to critical wall locations to assess the expected stability conditions. The design sections were selected to encompass the expected geotechnical and hydrogeological conditions, and cover the higher consequence pit walls locations. The locations of the design sections are shown in Figure 15.9.



Source: SRK, 2017

Figure 15.9: Location of the geotechnical design sectors

Typical factor of safety acceptance criteria values for large open pit slopes are summarized in Table 15.8 (Read and Stacey, 2009). Examples of low consequence of failure include individual bench failures and temporary slopes, medium consequences include failure of any slope of permanent or semi-permanent nature, and high consequence include failure of medium to high slopes (50-150m) carrying major haulage roads or underlying permeant mine infrastructure.

Table 15.8: Typical factor of safety and probability of failure acceptance criteria values for large open pit slopes

Slope Scale	Consequence of Failure(1)	Acceptance Criteria	
		Minimum SRF/FoS(2)	Maximum PoF(3), P[FOS ≤ 1]
Bench	Low to high	1.1	25 – 50%
Inter-ramp	Low	1.15 – 1.2	25%
	Medium	1.2	20%
	High	1.2 – 1.3	10%
Overall	Low	1.2 – 1.3	15 – 20%
	Medium	1.3	5 – 10%
	High	1.3 – 1.5	≤ 5%

Notes:

1) Consequence of failure semi-quantitatively evaluated.
2) FOS = factor of safety.
3) PoF = probability of failure.

A minimum acceptable design SRF/FOS between 1.3 and 1.5 has been targeting for overall stability as part of this optimization study. For the pit walls at Rory's Knoll, the consequence of overall stability failure is considered high for the following reasons:

- The close proximity of the proposed mine surface infrastructure (river dyke) to the north east wall;
- Stability requirement of the pits wall for the underground mine, and
- The single access ramp strategy currently adopted for the pits.

However, the risk of slope failure and the corresponding consequence of failure should be independently evaluated at both mine and corporate level to determine what is acceptable.

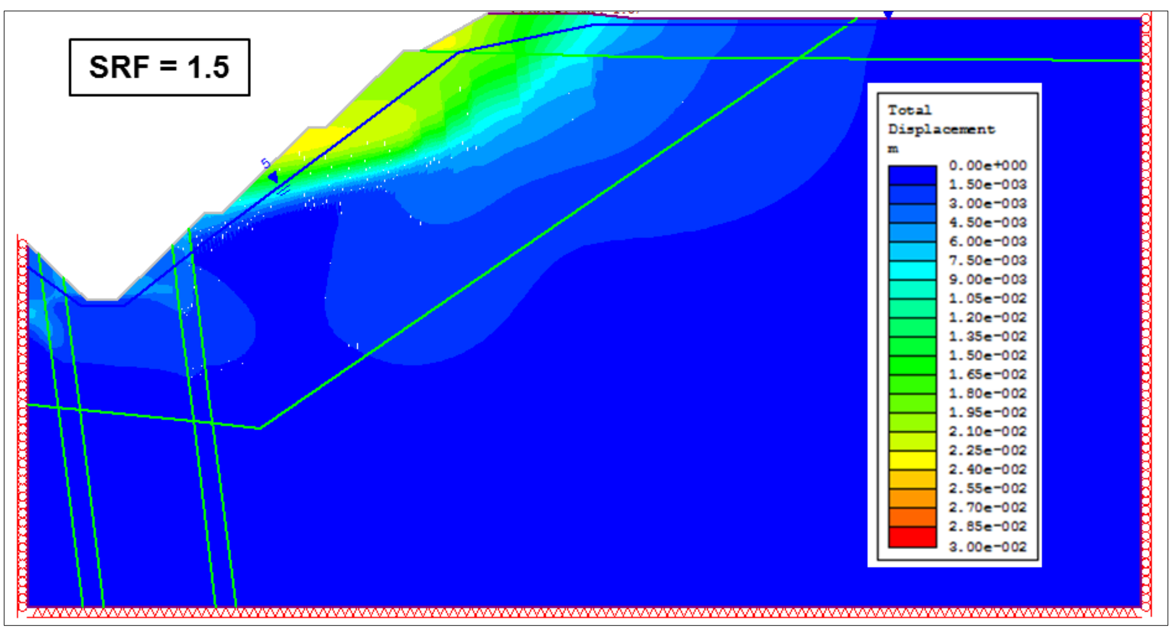
Rock mass strengths were estimated using the Generalized Hoek-Brown strength criterion (Hoek et al. 2002). The inputs summarised in Tables 15.2 and Table 15.3 form the non-linear strength criterion use to assess inter-ramp and overall stability conditions.

The results of the stability analyses for the design sections are presented in Table 15.9, and presented and supported by Figure 15.10, Figure 15.11, Figure 15.12. The main findings from the analyses include:

- The Phase2 analyses indicate that the strength reduction factors are greater than the targeted 1.3 through each design section;
- The Slide analyses indicate that mean deterministic factor of safety's are greater than the targeted 1.3 through each design section;

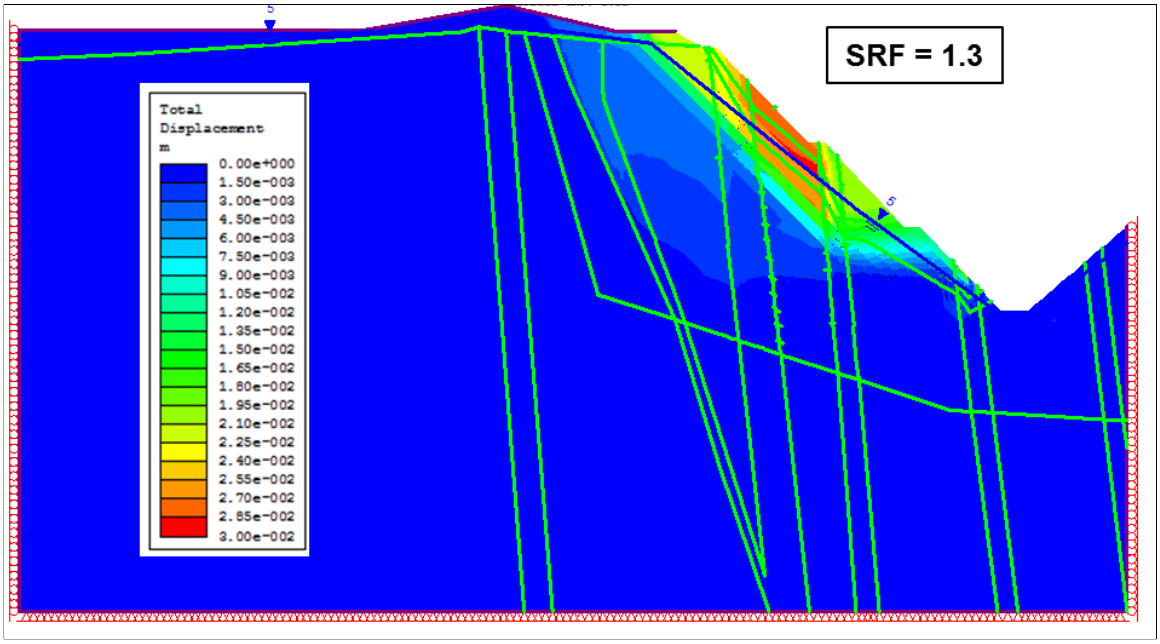
Table 15.9: Results of the stability analyses for the design sections

Design Section	Location	Modelled IRA	Groundwater Conditions[1)]	Phase[2] SRF	Instability Mechanisms	Results Discussion
A	Rory's Pit NE Wall	45°	Minimal Depressurization	1.5	Toppling on Foliation	Acceptable results are achieved utilizing the minimal depressurization strategy.
B	Rory's Knoll S Wall	45°	Minimal Depressurization	1.3	Sliding on J2	Inter-ramp scale sliding may be possible on joint set J2. Further characterization of joint set including persistence and orientation should be undertaken
C	Aleck Hill SW Wall (Saprolite Only)	49°	Minimal Depressurization	1.3	Rotational Failure through Saprolite	Acceptable results are achieved using design parameters



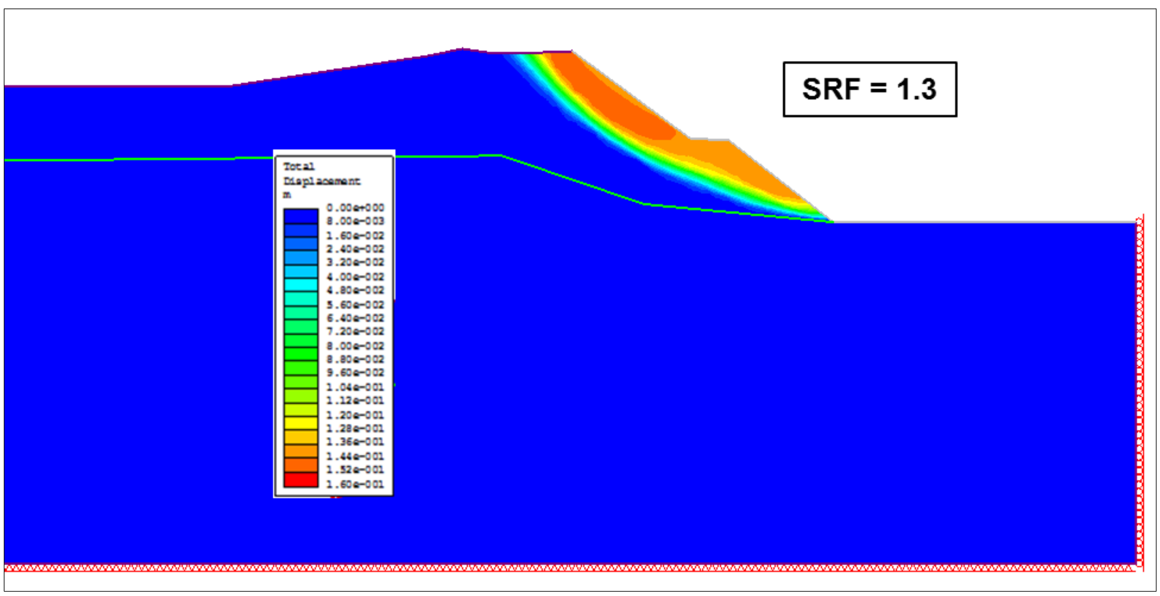
Source: SRK, 2017

Figure 15.10: Stability analysis of section A (Rory's Knoll – NE Wall)



Source: SRK, 2017

Figure 15.11: Stability analysis of section B (Rory's Knoll – South Wall)



Source: SRK, 2017

Figure 15.12: Stability analysis of section C (Aleck Hill – Southwest Wall - Saprolite)

15.2.5.4 Risks and Opportunities Associated with Slope Design Changes

The geotechnical design parameters presented within this document are based upon the review and analysis of historical data and observations on site. The following risks have been identified to the open pit design:

- **<u>Brittle Fault Location and Characterization</u>** – Currently, a model of the brittle fault features and large persistent joints expected behind the pit face in the region of the final pit does not exist. Local structure mapping has been conducted on site, however these observations have not been interpreted to extend behind the pit face to the planned interim or final wall locations. The structural model of the rock at the final pit wall location is an integral part of the overall slope design which must be reviewed against the likelihood of faults to avoid large scale slope failures ranging from multi-bench to inter-ramp scale. An improved understanding of the nature and location of any potential brittle fault features at the final pit wall locations should be undertaken as soon as possible. This is commonly achieved through the development of a 3D structural model. All slope designs and angles are contingent to their review against a fault model and should not be considered final until this reconciliation is complete. Pit slope designs may be governed by major structures.

- **<u>Shear Zone Characterization (Aleck Hill)</u>** – The location and geotechnical nature (thickness, strength, etc) of the shear zones at the Aleck Hill pit are based on limited data. Additional boreholes should be drilled to allow for a more complete characterization of these zones. If the shears are found to be extensive, inter-ramp wall angles may be required to be flattened to accommodate for the foliation and weakened rock mass strength.

- **<u>Wall Control Blasting</u>** – The angles provided within this document are based up best industry blasting practices in which wall control is of primary importance. Excessive blast damage will not allow for the achievement of the designed IRA's. Bench heights may be reduced to 10m if necessary, however, bench widths should also be evaluated to keep the IRA's in the design sectors similar.

The following opportunities have been identified to the open pit design:

- **<u>Saprolite Design Angle</u>** – Based on the current behaviour of the saprolite slopes, site specific laboratory testing and specific trial inter-ramp slopes can used to determine in saprolite slopes can be locally steepened. Pore water pressures should be monitored in these areas.

- **<u>Data Reconciliation</u>** – As mining progresses, additional data on both the saprolite and the rockmass (such as joint persistence, joint set orientation, etc) can be collected and used to update the geotechnical domain models. The inter-ramp angles may be able to be increased with increased confidence in the geotechnical data and continued site observations.

15.2.6 Underground Mining Geotechnical

15.2.6.1 Underground Design Criteria and Parameters

A summary of the key geotechnical parameters of the various structural domains at the Rory's Knoll has been discussed in Section 15.2.4.

As previously noted, the rock mass qualities per geotechnical domain are relatively consistent across the property. However, variation in uniaxial strength was noticed with the sericite schist closer to Aleck Hill being weaker. These factors have been considered in the design; however, the stope lengths maybe modified if strengths are found to be greater than reported. For the purpose of the underground design, average values were used to arrive at a simplified rock mass model, which is summarized in Table 15.10.

- Equasion used to estimate Q: *RMR=9lnQ+44.*

Table 15.10: Underground evaluation rock mass parameters

Domain	Average Intact UCS (MPa)	RQD (%)	RMR90	Q	Rock Quality
Interbedded	110	94	65-70	7.5-18.0	Good
Sericite	105	96	55-60	4.0 – 5.9	Fair
Tonalite	150	98	70-75	7.5-18.0	Good
Mafic	125	97	65-70	7.5-18.0	Good

The in-situ stress used in the numerical modelling for the underground mining geometry is summarised in Section 15.2.4. At this stage, it should be recognized that no stress measurements have been taken in the region and the stress regime applied utilizing a k ratio (maximum horizontal stress to minimum vertical stress) of 1.5 has been assumed.

15.2.6.2 General Infrastructure Headings

General infrastructure ground support recommendations for Rory's Knoll were outlined in the 2013 feasibility study (Tetra Tech, 2013). The ground support design criteria and recommendations have been provided for lateral and vertical development and selected critical infrastructure areas.

The majority of infrastructure will be within ground conditions considered to be of good rock mass quality except development located in the sericite shear which is in category of fair rock quality.

For the majority of the underground development infrastructure, stability assessments and support requirements are based on empirical estimates (based on Grimstad & Barton 1993, Laubscher 1990) and benchmarked with experience in other underground mining operations in a similar context. Figure 15.15 shows the basis for the empirical ground support design.

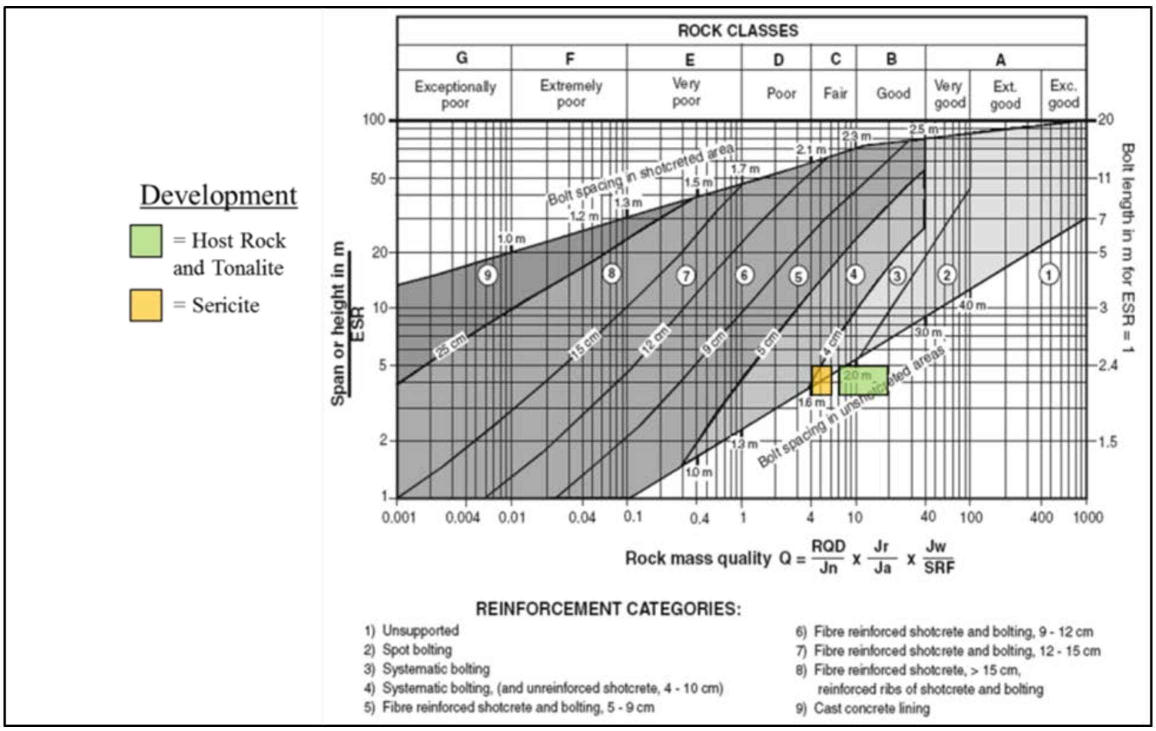
Development

Table 15.9 provides details of the development types included in the empirical support design along with the profile, height, width, and equivalent dimension (De) of the excavation.

Table 15.11: Development types and empirical design input parameters

Development	Profile	Height (m)	Width (m)	De
Decline Development	Arched	6.0	5.5	3.8
Other Capital Development	Arched	5.5	5.5	3.4
Operating Development	Flat	5.0	5.5	3.4
Capital Infrastructure	Arched	7.0	7.0	4.4

Note: Span or Height/ESR where ESR=1.6



Source: Grimstad and Barton, 1993

Figure 15.13: Empirical ground support design

For the infrastructure, where spans are between 5.5 m and 6.0 m, the recommended ground support consists of:

- 2.4 m long fully grouted rebar (22 mm) installed on a 1.5 m x 1.5 m spacing across back and shoulders with 150 mm square plates;
- 100 mm aperture, #6 gauge weld wire mesh (galvanised) across back and shoulders, (to within 1.5 m of the floor in Decline, to within 1.0 m of the floor with other capital infrastructure; and
- 2.4 m long friction anchors (39 mm, galvanised) installed in walls as required.

For the operating development where spans are between 5.0 m and 6.0 m, the recommended ground support consists of:

- 2.4 m long friction anchors (39 mm, galvanized) installed on a 1.5 m x 1.5 m spacing across back and shoulders with 150 mm square plates; and
- 100 mm aperture, #6 gauge weld wire mesh (galvanised) across back and shoulders to within 1.5 m of the floor.

For development in the saprolite (from portal to rock ore contact), additional use of 50 mm fibrecrete, mesh and split set bolts should be applied floor to floor.

For the capital infrastructure, where spans are between 7.0 m and 8.0 m, the recommended ground support consists of:

- 50.0 mm of fibrecrete to within 0.5 m of the floor;
- 2.4 m long fully grouted rebar (22 mm) installed on a 1.5 m x 1.5 m spacing across back and shoulders with 150 mm square plates;
- 7.0 m single Garford cable bolts on a 3 m burden and 2 m spacing (6 m of cable bolt imbedded) with 250 mm square plates; and
- 2.4 m long friction anchors (39 mm, galvanised) installed in walls as required.

Adjustments to standard support for poor ground conditions (estimated at 10% of each decline):

- Pattern rebar spacing reduced to 1.2 m x 1.2 m across back and shoulders;
- Weld wire mesh extended to floor level secured with friction anchors (galvanized); and
- 75 mm shotcrete across back, shoulders and walls down to floor level.

Cablebolts are required on a 2.75 m x 2.75 m spacing for any infrastructure in which the spans are greater than 8.0 m. Six cablebolts should be used for a three way intersection and nine cables should be used for a four way intersection using 6 m long cablebolts.

Cablebolts are required on a 2.0 m x 2.0 m spacing for any infrastructure in which the spans are greater than 7.0 m. In these sections, 18 cables should be used for a three way intersection, and 21 cables should be used for a four way intersection using 6 m long cablebolts.

Cablebolt length should be reviewed on a design span case-basis by the site geotechnical engineer. The development of intersections in zones of poor ground should be avoided. Breakaways should be staggered to limit four-way intersections.

Due to the generally good rock quality and shallow depth of the underground mine, detailed stress analyses are not considered necessary for ground support design at this stage in the project. Stress analyses can be completed to supplement empirical guidelines provided here to evaluate the potential extents of stress-induced damage on extraction levels, and anticipate the level of required support.

Ground Control Management Plan

The purpose of a Ground Control Management Plan (GCMP) is to provide a system for the management of the ground control strategy at the Aurora Gold Mine. A GCMP will be required once underground development commences at Aurora Gold Mine. A GCMP is a live document that is prepared, reviewed and approved by all key stakeholders and is intended to provide a background on the likely ground conditions, required procedures, and policy controls in place to manage the risks related to the rock mass conditions. The GCMP captures key features of the ground control design, implementation, and monitoring and is tailored to suit the complexity of the local geological conditions and mining operations. The GCMP is normally updated annually by on-site rock mechanics engineers as conditions change (with subsequent internal approval), with internal and external reviews completed on a regular basis.

Development of a GCMP is beyond the scope of a Feasibility Study but should be drafted during early development to establish a common understanding of the ground control standards, however, basic guidelines of a GCMP include:

- provision of systems used for evaluating, designing, maintaining, and monitoring the excavation stability to prevent personal injury, damage to equipment or process loss;
- a communication system that defines core responsibilities and accountabilities;
- development and maintenance of a process for hazard identification and risk management with regard to ground control and geotechnical mine design; and
- Introduction of employee control measures to effectively monitor and measure compliance to legislative regulations and corporate policy through audit and review processes.

The GCMP is supported by other procedural documents, such as:

- mine seismicity management plan;
- void management plan;
- subsidence management plan;
- specifications for ground support materials;
- mobile equipment requirements for ground support;
- excavation preventative maintenance plan; and

- management of seismicity following blasting.

The intent of the GCMP is to articulate the higher level strategies aimed at eliminating or minimizing the risk of fatalities, injuries, and incidents resulting from falls of ground and collapse in underground operations, of production loss, and of equipment damage.

Geotechnical Monitoring Program Recommendations

Geotechnical monitoring of open pit stability, stope stability, and the SLR mine is imperative to follow best practices for both production and safety reasons. A properly established monitoring network provides valuable information on mining advance, ground movement and disturbance and infrastructure stability.

Standard open pit slope monitoring using a prism network should be established and developed continuously as open pit mining progresses. If areas of concern are determined through visual observations and/or the analysis of the monitoring equipment, additional slope stability measuring equipment, such as slope stability radar should be implemented to assess the slope stability. This stability monitoring system will continue to be utilized as underground mining progress and will be complemented by additional underground excavation monitoring systems using Lidar or similar. The surveyed underground and SLR excavations can then converted into a 3D model used for on-going geotechnical analysis. Such models aid in further prediction of stability, potential dilution/recovery reconciliation, and risk mitigation and design measures.

Beside the excavation geometry monitoring, the rock mass should be instrumented as decline development progresses with focus on the rock pillar between the SLR stope and production drift and decline as well as the underground infrastructure throughout the site.

SRK recommends that the monitoring program is carried out in the following four phases:

- Phase 1 – Provisional Design and System Selection. Based on the current knowledge of geotechnics and mining, the different alternatives and approaches will be evaluated and an appropriate system will be recommended. The remainder of this section lays the framework and rationale for typical integrated open pit and SLR monitoring systems employed in active mines today.

- Phase 2 – The Final Design would take into consideration the final underground design, updated geology and structures. The detailed costing and implementation schedules are included.

- Phase 3 – Installation and Commissioning Phase. The provisional design will be reviewed and updated if material changes occurred in terms of the mine plan. Typically, the consultant/contractor, together with the instruments manufacturer, would provide supervision of the drilling program, installation of instruments, and commissioning of the systems.

- Phase 4 – Monitoring and Data Analysis. In this phase, mine personnel are provided with appropriate training in order to develop the ability to record, analyse and interpret monitoring results. Typically, an external party would also provide ongoing support and QA/QC.

Where possible, a design should make provision for the reading of a combination of instruments by independent and different processes, e.g., an automated logging system, a stand-alone logger, and manual readings. Instruments and data loggers should be placed with due regard to access in case of failures and protection from exposure to mobile equipment, blasting, dust and moisture.

Decline Portal Evaluation Program

At this stage, detailed geotechnical drilling has not been completed for the proposed portal locations.

SRK would recommend the following studies are completed for the design of a portal box-cut within the saprolite for Rory's Knoll:

- Specific geotechnical drill holes characterizing the talus and overburden materials, depth to bedrock, and bedrock contact rock mass quality.
- Pilot drill holes from the proposed portal location along the decline alignment where insufficient data is available to accurately characterize the rock mass.

15.2.6.3 Risks and Opportunities Associated with Underground Designs

The geotechnical design parameters presented within this document are based upon the review of historical data and observations on site. The following risks have been identified to the underground design:

- **Slope Stability** – The current mine design includes open benching to the surface. Instability of the slopes above may lead to excess dilution in the underground mine. Due to their impact on the underground mine, the slopes should be designed to a FOS of 1.3 and above.
- **Brittle Fault Location and Characterization** – Currently, a model of the brittle fault features and large persistent joints expected underground does not exist. Local structure mapping has been conducted on site, however these observations have not been interpreted to extend to the underground locations. An improved understanding of the nature and location of any potential brittle fault features at the underground mine locations should be undertaken as soon as possible. This is commonly achieved through the development of a 3D structural model.
- **Shear Zone Characterization (Aleck Hill)** – The location and geotechnical nature (thickness, strength, etc) of the shear zones at the Aleck Hill pit are based on limited data. Additional boreholes should be drilled to allow for a more complete characterization of these zones. If the shears are found to be extensive, underground development and stope locations may be required to move or be reduced in size to accommodate for the foliation and weakened rock mass strength or an increase in ground support (and associated costs) will be required.
- **Water Inflow** – Any changes in the location and geotechnical nature (thickness, strength, etc) of the shear zones at the Aleck Hill area may affect the local water inflow into the underground excavation. Additional costs may be associated with managing the water inflow.

- **Ore Geometry** – ore geometry which is more complex than expected, may increase damage and dilution during production, and reduce ore recovery.

The following opportunities have been identified to the underground design:

- **Geological Reconciliation** – As additional geotechnical information is collected on the areas near the underground design, ground conditions found to be more competent than the current model will allow for a reduction in support cost.

15.3 Open Pit Mining

15.3.1 Open Pit Design

The pit shells selected in pit optimization (Section 14.3.2) guide the detailed ultimate and pit phase designs. The designs employ the final geotechnical criteria for bench heights, berm widths, bench face angles, inter-ramp angles and overall slope angles. The objective of the detailed designs is to illustrate practical mining shapes, including bench access for all phases while matching the tonnes of ore and waste from the optimized pit shells as closely as possible.

The strategy adopted for mining the open pits is to start utilizing 90 tonne class haul trucks in addition to the existing fleet of smaller articulated 41 tonne haul trucks. The larger truck size fleet would improve productivity of waste stripping and reduce the unit cost. However, in order to maximize the overall slope angle of the ultimate pit walls and reduce the overall strip ratio, the wide ramp required for the larger trucks is only maintained part way down the pit, at which point the remaining mining must be completed by the smaller truck size fleet using a narrower ramp. In Rory’s Knoll, the wider ramp was designed down to -100 m bench and in Aleck Hill down to 0 m bench. The ultimate ramps in the three smaller pits, North Aleck, Walcott Hill and Mad Kiss, are only the narrower width.

Haulage ramps were designed to have a maximum grade of 10%, except for the bottom benches of each phase where ramps were steepened to 15%. Ramp width was calculated to be 2.5 times the width of the largest haulage unit, with allowance for berms and ditches. Single lane ramps are designed to be 1.5 times the width of the largest haulage unit plus berms and ditches.

The open pit designs were developed for five pits: Rory’s Knoll, Aleck Hill, North Aleck, Mad Kiss and Walcott Hill.

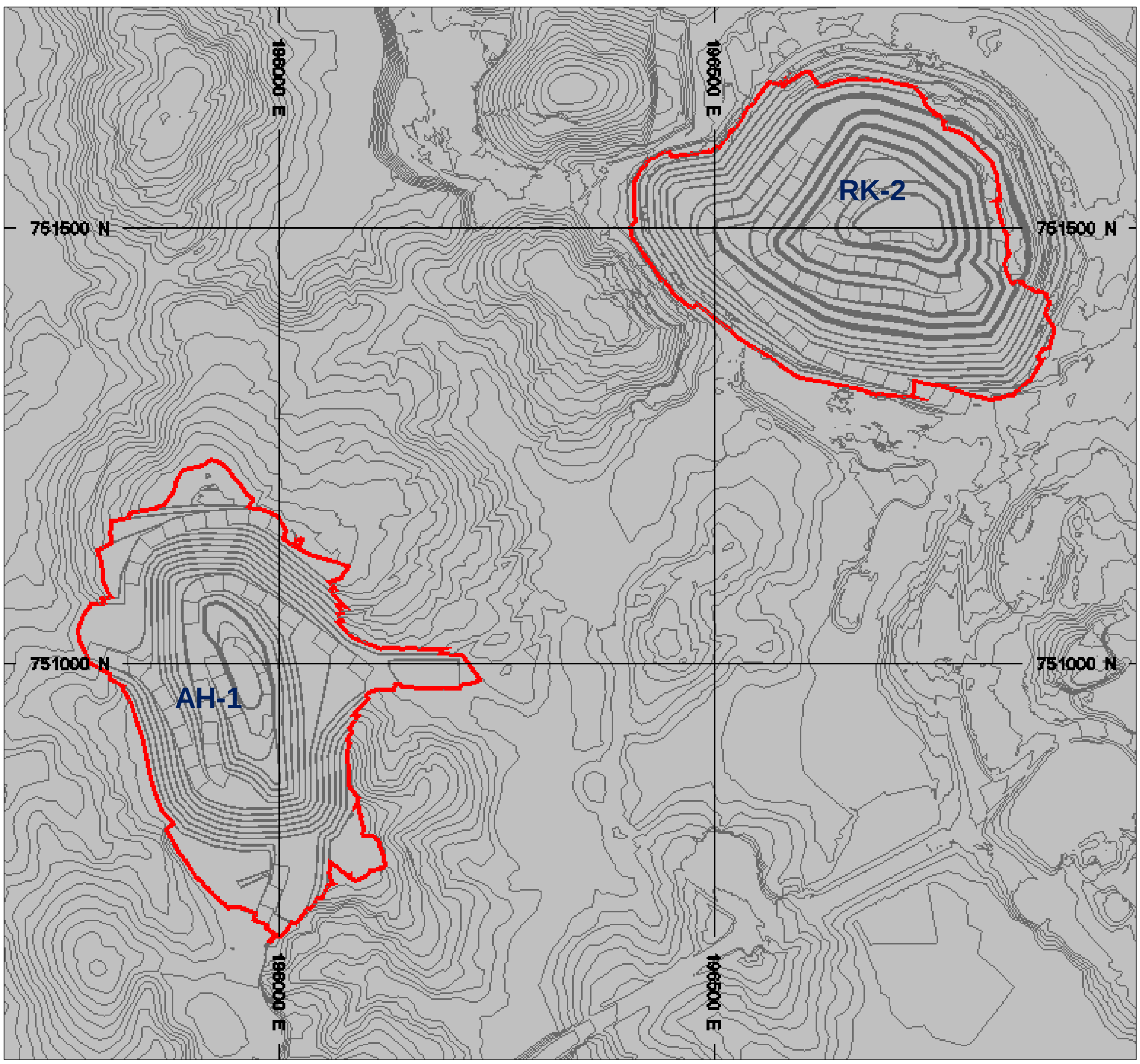
Walcott Hill was designed with only a single phase. The other pits were designed with two to four phases of mining. Phases were designed based on incremental Whittle nested pits at selected gold prices that produced substantial step changes in the net revenue between phases.

Design criteria used in the detailed designs are summarized in Table 15.12.

Table 15.12: Pit design criteria

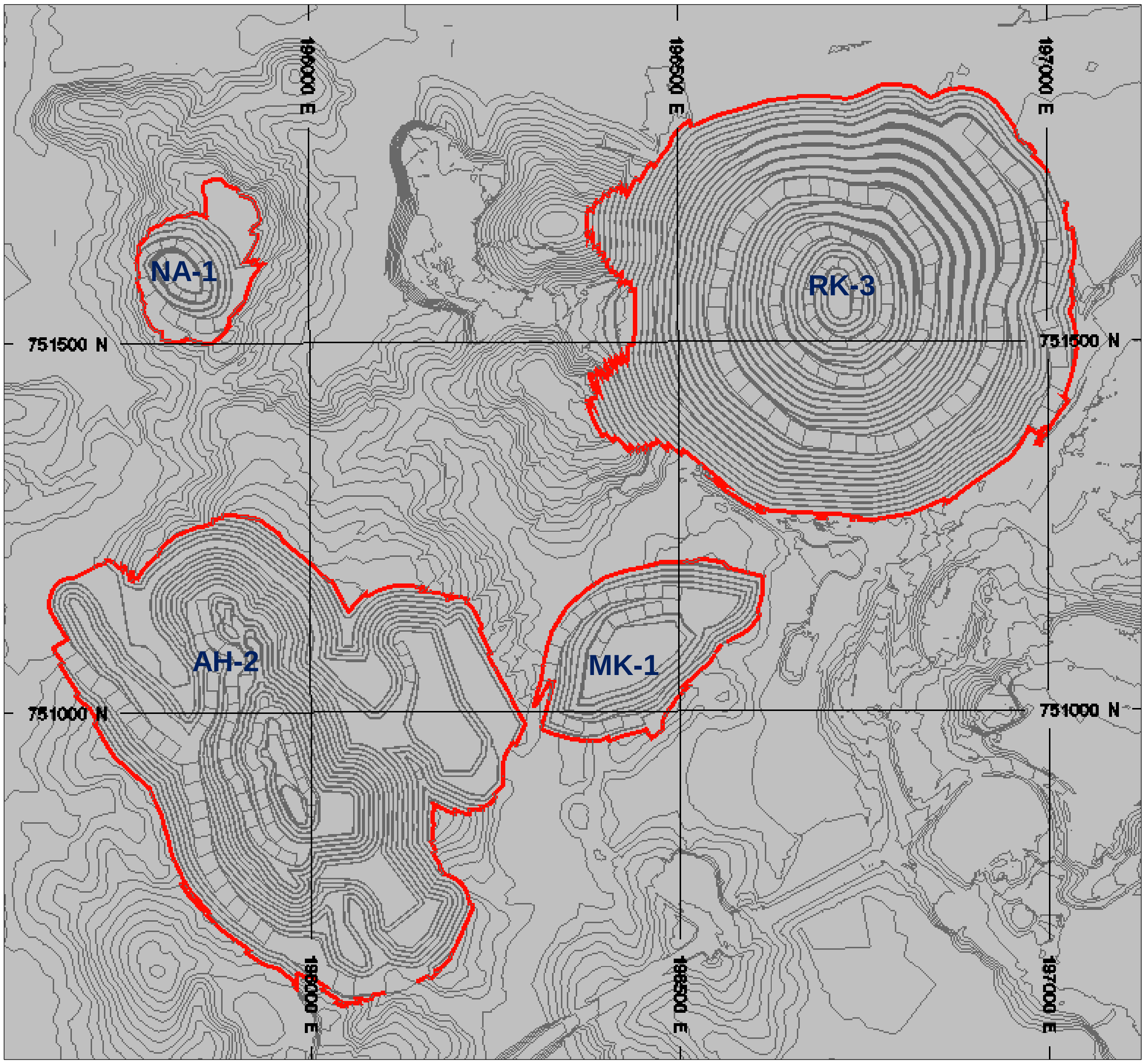
Description	Units	Value
Two-way 90 t Ramp	m	23
Two-way 41 t Ramp	m	16
One-way 41 t Ramp	m	13
Ramp Grade	%	10
Bench height	m	Variable by domain

Figure 15.14, Figure 15.15, and Figure 15.16 illustrate the phase and ultimate pit designs.



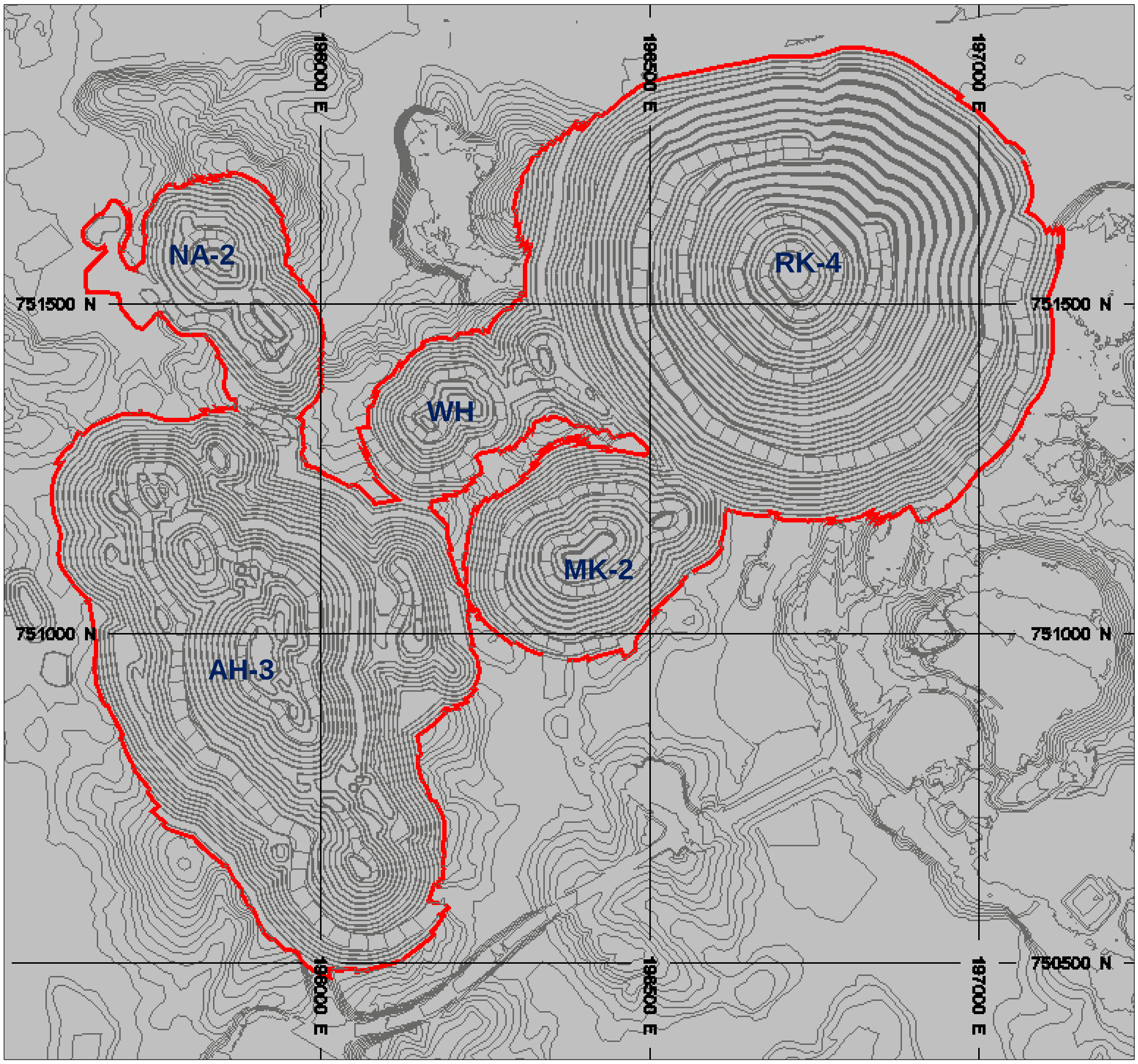
Source: SRK, 2017

Figure 15.14: Current phases in Rory's Knoll and Aleck Hill



Source: SRK, 2017

Figure 15.15: Interim pit designs



Source: SRK, 2017

Figure 15.16: Ultimate pit designs

The tonnes and grade of plant feed and tonnes of waste in the detailed pit designs are summarized and compared to the pit optimizations in Table 15.13.

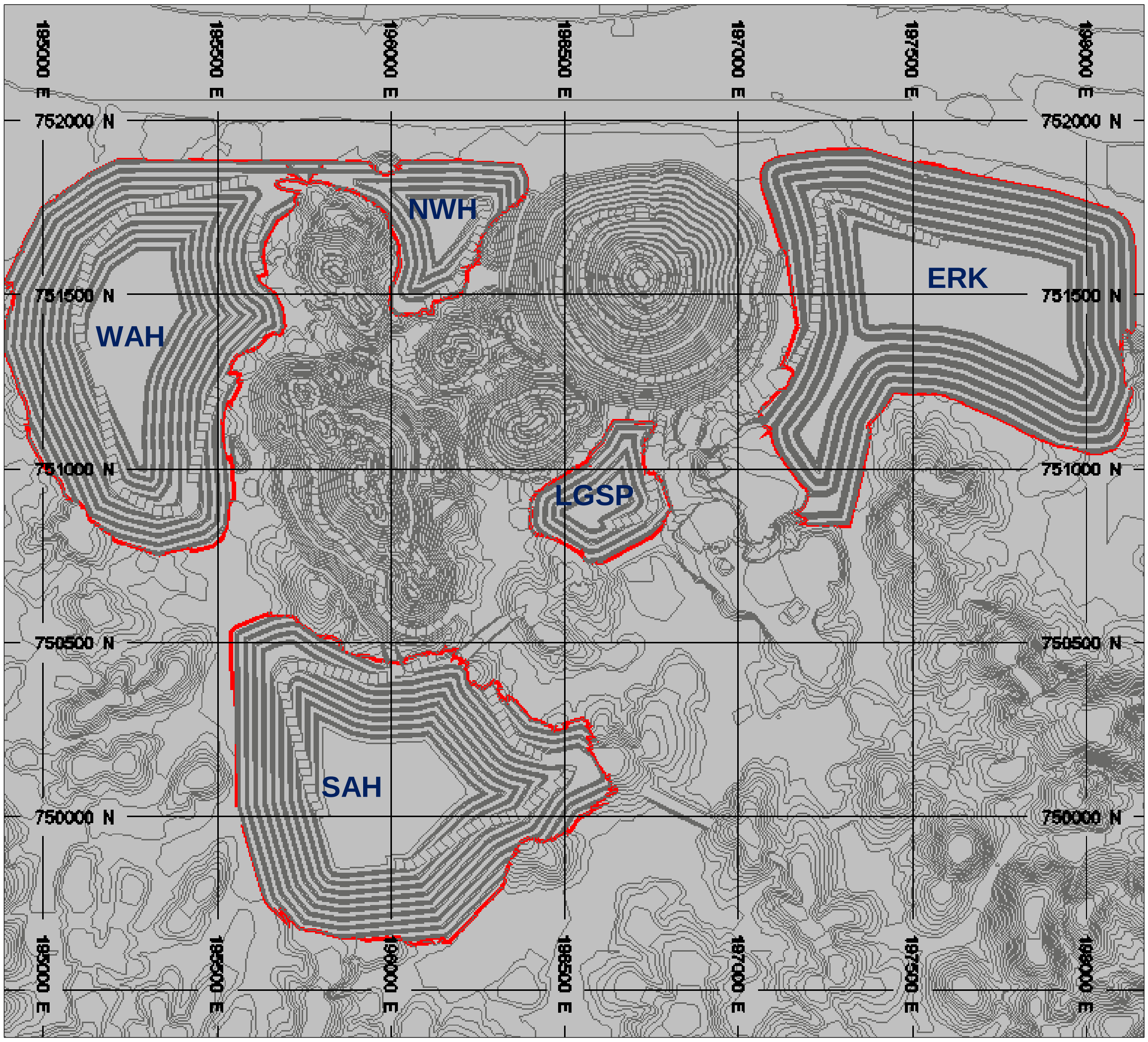
Table 15.13: Pit design quantities

Pit	Phase	Plant Feed	Grade	Contained Gold	Waste	Strip Ratio
		tonnes	g/t	ounces	tonnes	waste:ore
Rory's Knoll	RK1	608,307	3.49	68,231	229,864	0.4
	RK2	2,246,141	2.62	188,860	7,755,416	3.5
	RK3	4,982,309	3.14	502,499	37,701,984	7.6
	RK4	4,413,951	2.88	408,520	56,580,175	12.8
	Subtotal	**12,250,708**	**2.97**	**1,168,111**	**102,267,439**	**8.3**
Aleck Hill	AH1	1,560,323	2.69	134,952	4,467,591	2.9
	AH2	2,157,700	3.1	214,793	13,339,699	6.2
	AH3	2,743,311	2.41	212,557	31,038,467	11.3
	Subtotal	**6,461,333**	**2.71**	**562,302**	**48,845,757**	**7.6**
North Aleck	NA	**958,703**	**1.74**	**53,784**	**4,617,494**	**4.8**
Walcott Hill	WH	**322,272**	**1.62**	**16,826**	**2,772,397**	**8.6**
Mad Kiss	MK	**637,825**	**4.12**	**84,561**	**8,736,101**	**13.7**
Pit Design Total		**20,630,841**	**2.84**	**1,885,583**	**167,239,187**	**8.1**
Pit Optimization Total		21,198,229	2.88	1,961,707	163,483,638	7.7
Difference		-567,388	-0.04	-76,124	3,755,548	0.4
Variance		-3%	-1%	-4%	2%	5%

A variance of 3% on plant feed tonnes and 2% on waste stripping tonnes are within the expected range when moving from optimized pit shells to detailed pit designs. These volumes are not the declared reserve for this feasibility study.

15.3.2 Waste Storage Facilities

Four primary waste storage facilities (WSF) have been designed using similar footprints to the WSFs designed for previous mine plans for the Aurora gold mine, as shown in Figure 15.17. These include a WSF east of the Rory’s Knoll pit (ERK), north of the Walcott Hill pit (NWH), west of Aleck Hill pit (WAH) and south of the Aleck Hill pit (SAH). In addition, a fifth storage facility is designed south of the Mad Kiss pit to be used for long term stockpiling of low grade material. The four primary WSF’s have a total capacity of over 90 Mm^3, which is sufficient for all of the waste stripping in the mine plan. There is also 8 Mm^3 of pit backfill volume identified for the mine plan in the Mad Kiss pit and the west and south lobes of the Aleck Hill pit.



Source: SRK, 2017

Figure 15.17: Waste storage facilities

15.3.3 Open Pit Production Schedule

The open pit production schedule was developed with the aim of delivering the highest ore grades to the mill as early as possible in order to maximize the near term cash flow of the project and maximize the net present value. This was guided by mining the pits and phases in a value sequence per the price sensitivity of the pits and phases from the pit optimization exercise.

The production totals for ore, waste and gold ounces produced in 2017 are derived from the 2017 AGM budget. The benches mined in the 2017 Budget were depleted in the schedule model and then scheduling with the new parameters began for January 2018.

The production rate of 5,600 tpd was used throughout 2017 and until March 2018. In April 2018, debottlenecking activities in the process plant allow for an increase in processing rate up to 8,000 tpd with a maximum of 50% saprolite and a maximum of 75% rock (where saprolite is available).

The production schedule applies a stockpiling strategy in order to elevate the feed grade to the mill early on. Saprolite and rock ores are divided into low grade, medium grade and high grade categories with high and medium grades being prioritized in the mill feed. Low grade saprolite and rock ores are stockpiled separately until required to “fill the mill” when insufficient high grade and medium grade are being mined.

Pit development is generally limited to 12 benches per year in order to have practical mining rates in each phase. This is important for the final phases of Rory’s Knoll and Aleck Hill where the bench advance rate determines the ore release late in the open pit mine life.

The tonnes of ore and waste mined in the production schedule are summarized in Table 15.14.

Table 15.14: Open pit production summary

Description	Units	Total	2017	2018	2019	2020	2021	2022	2023	2024
Saprolite Ore	kt	2,934	1,001	1,417	94	91	-	189	142	-
Rock Ore	kt	16,700	1,469	1,631	2,056	2,229	2,937	2,723	2,556	1,099
Sap Waste	kt	40,360	4,876	22,737	5,029	4,559	1	1,487	1,669	3
Rock Waste	kt	126,687	6,862	5,651	27,684	28,208	27,644	21,926	7,915	797
Ore Processed	kt	20,297	2,044	2,704	2,920	2,928	2,920	2,920	2,762	1,099
Saprolite Ore	kt	3,270	613	1,151	1,083	91	0	189	142	0
Sap Ore Grade	g/t Au	1.88	3.17	2.63	0.72	0.71	0.00	1.00	0.94	0.00
Rock Ore	kt	16,990	1,431	1,553	1,837	2,837	2,920	2,731	2,583	1,099
Rock Ore Grade	g/t Au	3.01	2.72	3.56	3.17	2.49	3.43	2.81	2.88	3.41
Avg Feed Grade	g/t Au	2.84	2.86	3.16	2.32	2.43	3.43	2.69	2.76	3.41
Contained Metal	k oz Au	1,850	188	275	218	229	322	253	245	120

15.3.4 Open Pit Operation

15.3.4.1 Drilling and Blasting

At the Mine, only the rock waste and ore is blasted. Saprolite waste and ore requires no drilling and blasting however it still will require drilling and sampling for grade control. Drilling of all rock is done on five metre benches using current 115 mm (4.5 inch) percussion rock drill and new 165 mm (6.5 inch) down-the-hole drills to be acquired in 2017 and as production levels increase.

Blasting presently is performed using a packaged emulsion product; however, with the transition to the larger diameter blasthole drills, AGM is to commission an explosives delivery truck capable of loading both emulsion and ANFO type bulk explosives. AGM performs all the blasting functions with its own personnel and equipment.

15.3.4.2 Loading

The Mine presently conducts all truck loading with hydraulic excavators (backhoes) ranging from 5.4 m^3 capacity down to 2 m^3 (7.1 yd^3 to 1.6 yd^3). There are also 6.4 m^3 (9 yd^3) front end loaders on site, which are primarily stationed at the Run-of-Mine (ROM) area at the primary crusher, but which occasionally do truck loading in pit.

With the expansion of milling operations in 2018, up to four 10 m^3 (13.1 yd^3) hydraulic front shovels are to be purchased to increase the mine loading capacity. These shovels are intended only to load the larger 90 tonne haul trucks brought on at the same time.

15.3.4.3 Hauling

The Mine presently uses a fleet of 41 tonne (45 ton) articulated haul trucks for mine haulage. This fleet is to be expanded in, but starting in the first quarter of 2018, 90 tonne (100 ton) rear dump haul trucks are to match the 10 m^3 shovels. This equipment will only be deployed to haul waste.

For this feasibility study update, haul profiles were derived for all different combinations of material source (specific pit phase and bench levels) and destination (ROM or specific WSFs and levels). Speed curves were applied against these profiles (including a rolling resistance of 3%) to derive travel cycle times. Full truck cycle times added truck spotting, loading, and dump times to these.

15.3.4.4 Support Equipment

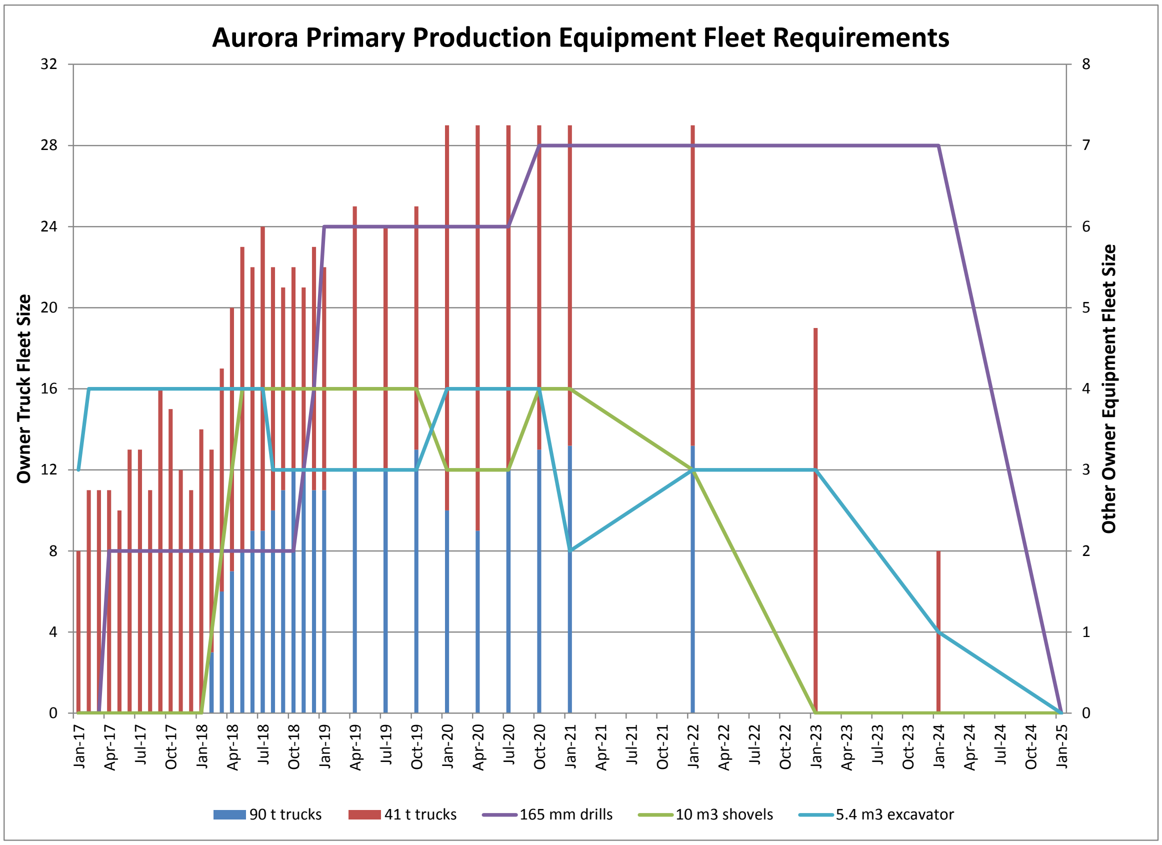
The Mine presently has an assortment of support equipment to maintain roads and dumps. These included tracked dozers (3.8 m/12.7 ft to 4.7 m /15.3 ft blade), graders (4.3 m/14 ft blade), a rubber-tired dozer (4.2 m/13.8 ft blade), and water trucks. This same equipment was adopted in the new mine plan with increasing fleet sizes driven by a combination of increased material moved and greater haul operating hours.

15.3.4.5 Ancillary Equipment

Ancillary equipment at the Mine includes pit pumps and lines, lighting plants, light vehicles, maintenance support vehicles including fuel and lube trucks, mechanics trucks, cranes, etc. The current complement of such equipment is maintained with certain equipment being increased with increasing production areas and levels (e.g. pumps, lighting, light vehicles).

15.3.4.6 Open Pit Equipment Fleet

The primary production equipment (drilling, loading and hauling) are shown in Figure 15.18. The fleet size is indicated as the "owner fleet size". This relates to the capital expenditure of equipment, whereas in some periods there will be additional equipment required which will be accommodated through equipment rentals. This impacts the smaller drills, the 5.4 m^3 excavator and the 41 t trucks.



Note: scheduling periods are monthly 2017-2018; quarterly 2019-2020; annually thereafter

Figure 15.18: Primary production equipment requirements

15.3.5 Open Pit Workforce

The open pit workforce was built up from first principles for mine operations and mine maintenance hourly personnel. The SRK manpower model was calibrated against the AGM 2017 budget. The salaried staff used as its starting point the current complement at Aurora, which was then escalated by 15% for when production levels increased in 2018. Correspondingly, as production tailed off in the later years, the salaried headcount was de-rated. The resulting chart of headcount is shown in Figure 15.19.

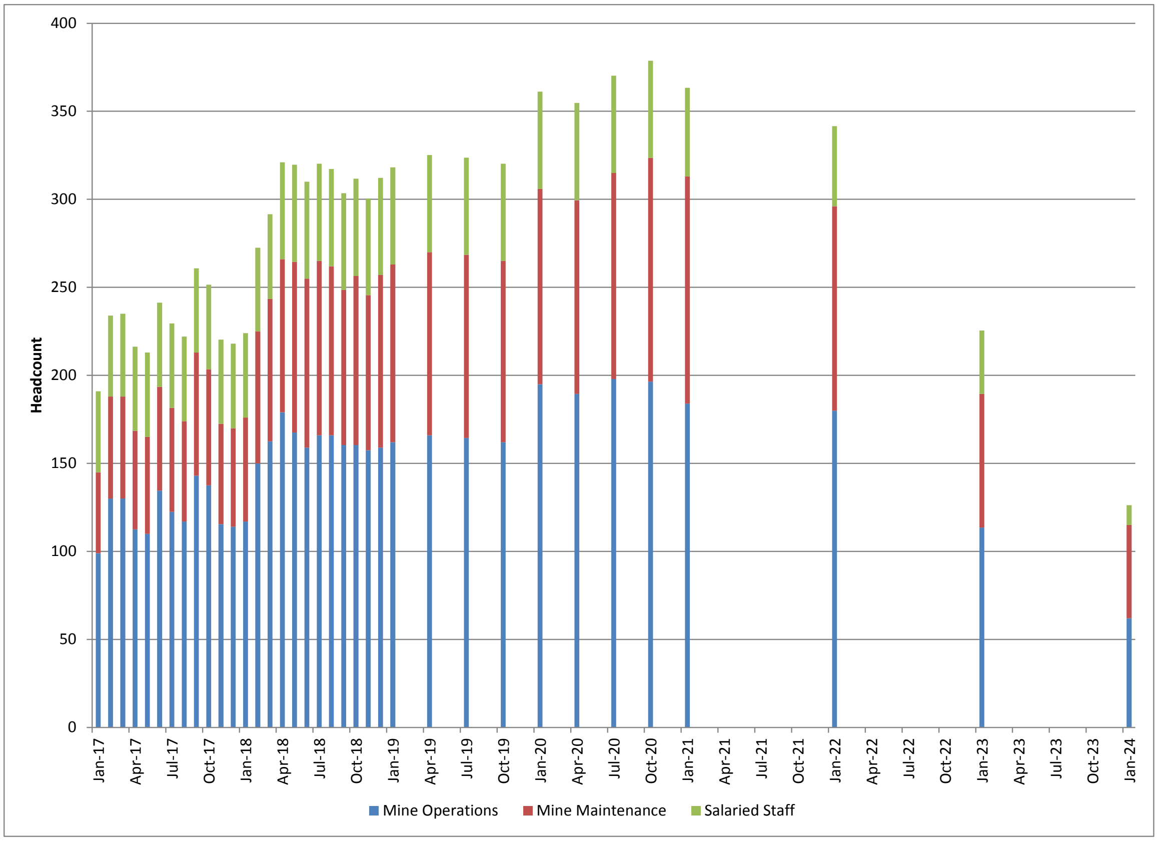


Figure 15.19: Aurora mine open pit head count

15.4 Underground Mining

15.4.1 Mining Methods Selection

The Aurora gold deposit has been defined by surface exploration drilling; there is currently no underground development or exposures of the rock ore mineralisation.

The choice of a mining method is primarily aimed at achieving the lowest cost to finished metal with manageable risk, while maintaining a safe mining environment and achieving optimum production rates and productivities.

The mining method selection for the Aurora deposits was guided by the following:

Geology: continuity and predictability of the lithology that hosts the economic material; whether the ore can be visually identified or whether cut-off is to a grade boundary.

Geometry: actual mining shapes - size, attitude, dip; variability of short and long range geometry that might affect planned dilution and recovery.

Grade Distribution: grades contoured to different cut-offs; geometry and continuity of grade shapes; grade of dilution.

Rock Mass: strength and competence of the rock mass and influence on excavation stability (especially unplanned dilution) and fragmentation from blasting or caving.

Geological Structure: major through-going, significantly weakening, structures that could affect very large excavations or mining directions and sequences.

Disturbances: depth and in situ stresses; presence of water that could decrease stability or cause muck-rushes and/or protection of ground water; high rock temperatures.

External Constraints: production rate required versus tonnes per vertical metre (TVM), rock mass conditions and mining method; protection of surface and avoidance of caving or limiting subsidence.

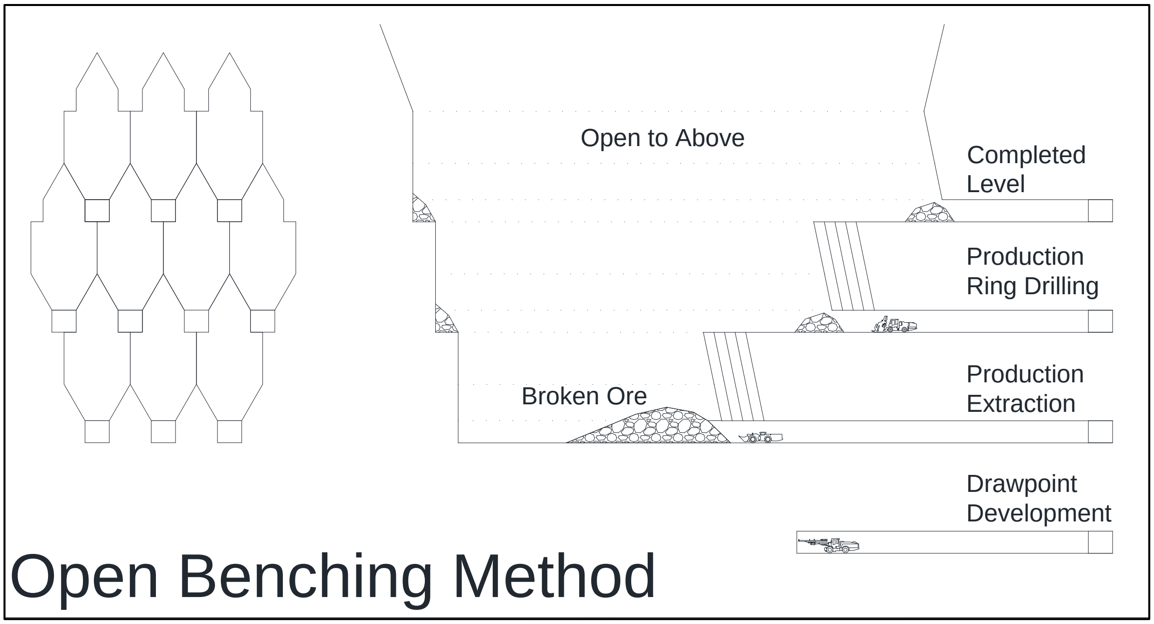
15.4.1.1 Rory's Knoll

The generally competent rock mass, steep dip, and large dimensions make Rory's Knoll conducive to a number of possible unsupported mining methods. Due to the unconfirmed nature of the continuity and regularity of the deposit (no underground development exposure), the over-riding factors in the mining method selection are to identify a method that achieves the following objectives:

- is mined from top down,
- minimizes initial capital,
- ensures mill feed continuity,
- achieves a high extraction ratio,
- can be highly productive,
- can be operated at a low cost, and
- has a repeatable mining sequence to reduce the implementation risk.

SRK (Tetra Tech, 2013) concluded that a combination of open benching and sublevel retreat (SLR) methods was a viable option. Based on the current underground geotechnical knowledge, SRK continues to consider the open benching and Sublevel Retreat (SLR) mining methods appropriate and the most likely methods to deliver an economically viable project with acceptable operational safety standards and productivities.

Open benching will be used for the first three partial and six full sublevels and SLR for the remaining 16 sublevels starting with the -395 mRL. Both open benching and SLR are top down retreating mining methods applied to steeply dipping deposits in strong host rocks The methods do not utilize backfill and are non-caving mining methods. Ore fragmentation is engineered through blast design.

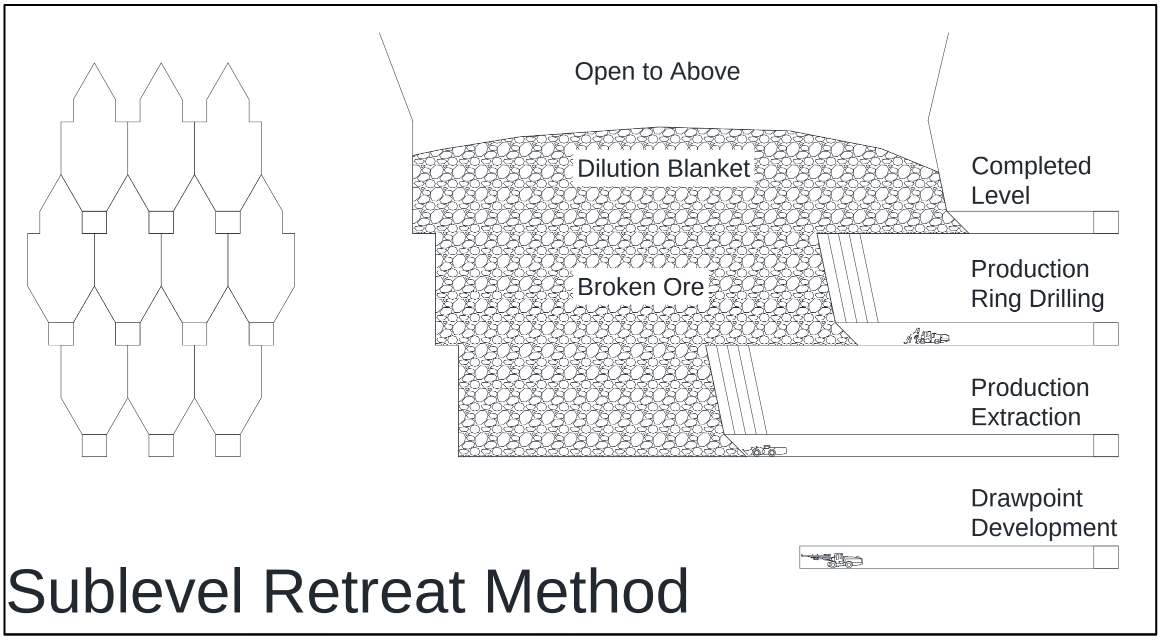


Source: SRK, 2017

Figure 15.20: Rory's Knoll: Open benching mining method

Open benching can be similar in geometry to SLR and can use the exact same sublevel spacing and ring shapes as SLR. The primary difference between the two methods is that open benching blasts are completely mucked leaving the stopes open to above (Figure 15.20) and SLR blasts are confined, blasting against ore that was left behind from levels above (Figure 15.21). There are slight differences in drilling and blasting practices of each to adjust for blasting into an open void or against broken muck.

Modern safe work procedures require that the operator in a manually operated loader does not go beyond the brow of the drawpoint – the loader bucket may certainly go beyond the drawpoint brow, but not the operator. This procedure ensures the loader operator is always situated in a safe working area, under supported ground. Clearly, the open void in an open benching stope is not adequately supported (if at all) and therefore is "off-limits" to the loader operator. Figure 15.21 shows that a greater proportion of the blasted ore will be located beyond the brow of the drawpoint and, therefore, beyond the reach of the manually operated loader. This is overcome by utilizing remotely operated (tele-remote) loaders.



Source: SRK, 2017

Figure 15.21: Rory’s Knoll sublevel retreat mining method

SLR mining leaves a blanket or layer of blasted material over top of the drawpoints to protect personnel and equipment, and also to provide support for the stope walls. The SLR extraction strategy is planned so that material in this blanket is (at least) low grade or (ideally) high grade ore which reduces the flow of barren (waste) material from above into the drawpoints.

SLR is a variation on the better known Sublevel Caving (SLC) mining method. In SLC, caved waste accumulates above the drawn ore, whereas in SLR there is no expectation of a cave developing above the mined stops. This is the case in Rory’s Knoll where the SLR starts beneath the voids created by the open pit and the open benching stopes. SLR is an easy method to mechanize and is normally applied in massive, steeply-dipping orebodies with considerable strike length. SLR typically has high dilution and low mining recovery. A disadvantage of SLR is that the high grade that must be left behind at the top of a column has a reduction in grade as it combines with dilution during mixing before it reaches the draw point. An advantage to SLR over SLC is there is that SLC has a much higher risk of mudrush events occurring as more fines are generated and cave areas above the production levels provide storage for the water.

PCSLC software was used to model the SLR production.

Description of PCSLC

PCSLC was developed through a partnership between Gemcom Software International Inc. (now Geovia Mining Software Solutions) and SRK Consulting to simulate and evaluate the mining of deposits using a Sublevel Caving (SLC) mining method. PCSLC provides users with the ability to simulate and compare multiple mining layouts, mining sequences, and draw scenarios based on predetermined caving parameters. PCSLC moves and mixes material based on defined and calibrated spatial relationships among rings with a process known as Template Mixing. As

PCSLC does not model particle flow or use rules of physics to model material movement, the PCSLC production schedule depends very strongly on calibration of the Template Mixing inputs. PCSLC has default values which were calibrated by SRK using data from marker studies at the Ridgeway mine. For Rory's Knoll, additional changes were made to the calibration to reflect the drawpoint and level spacing and the numerical modelling of the extent of potential dilution.

PCSLC was initially designed for modelling flow within a sublevel caving mine, but it can adequately model flow within open benching and SLR mines with appropriate changes to dilution, extraction and recovery parameters. This model can be further calibrated as project understanding increases with underground development progress.

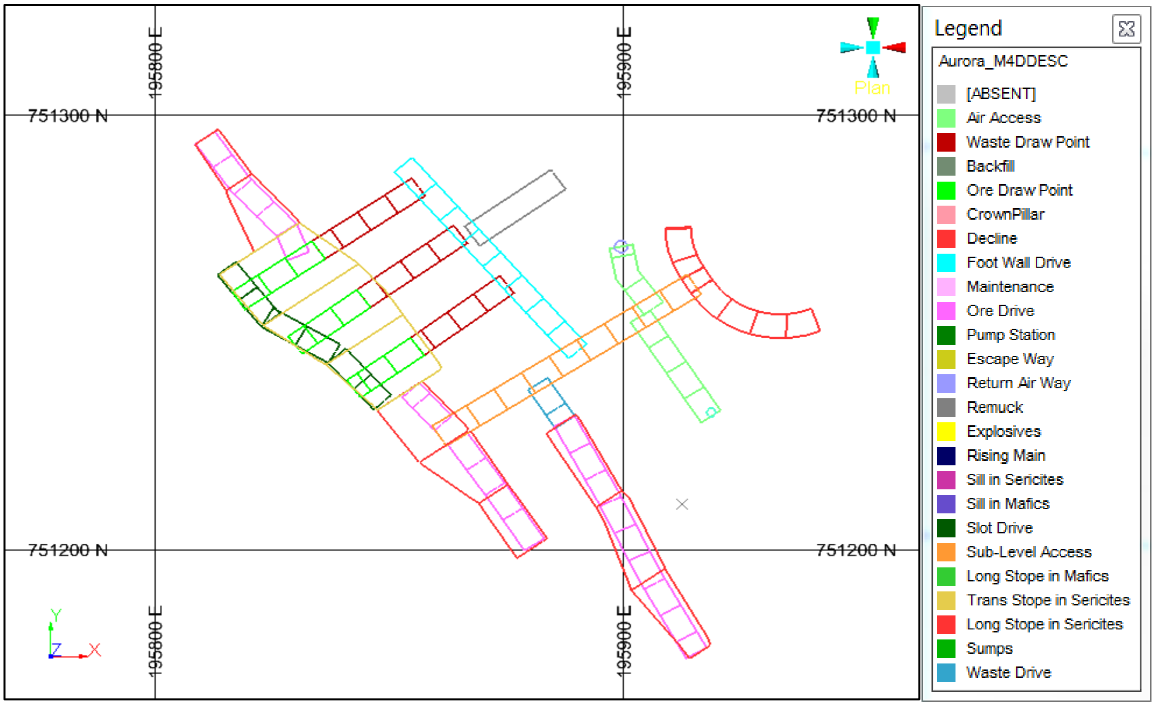
15.4.1.2 Satellite Deposits

Based on the mining context, both satellite deposits (Aleck Hill and Mad Kiss) have the rock mass conditions and geometry to accommodate supported or unsupported LHOS, such as open stoping with pillars (partial extraction) or LHOS with backfill

The supported transverse and longitudinal LHOS mining method was chosen for both deposits because it is better suited for wider orebody such as Aleck Hill and increases the orebody extraction.

Long hole stoping (LHOS) is a highly selective and highly productive mining method and can cater for varying ore thicknesses and dips (0˚ to ~45˚). It differs from manual (man-entry) methods such as timbered and shrinkage, as once the stope has begun the blasting phase it is no longer accessed by personnel. For this reason, the blasted rock is designed to fall into a supported drawpoint or removed with a mechanised loader (LHD).

Transverse and longitudinal longhole mining methods are presented in Figure 15.22, which shows both layouts in plan view. The crosscuts used in transverse stoping are driven perpendicular to the strike of the orebody, whereas in longitudinal development, the ore drives are driven parallel to the orebody strike. The ability for drifts to be driven perpendicular to the orebody in transverse stoping becomes beneficial when stope widths approach 20 m, as development costs can be minimized by decreasing the number of access drifts and cross-cuts in waste.



Source: SRK, 2017

Figure 15.22: Satellite deposits: Transverse and longitudinal LHOS layout

Transverse longhole stoping is a bulk mining method in which the long axis of the stope and access drifts are perpendicular to the strike of the orebody. Typically, draw points are located in undercut access drifts which extend from the footwall access dive, and the free face is mined in a horizontal retreat from the hanging wall to the footwall. In general, transverse longhole stoping is used where the rock mass quality of the hanging wall limits the strike length of the open mining span. This methodology requires more footwall waste development (for footwall drifts and draw points), however, since each stope has an independent access, it has more flexibility with regards to sequencing and scheduling. The transverse methods typically have a higher productivity than the longitudinal method.

Longitudinal longhole stoping methods operate along or parallel to the strike of the orebody. Longitudinal longhole stoping is preferred in areas where ore thickness is diminished, or where narrow veins of ore are present. The orientation of longitudinal mining methods allows the hanging wall and footwall of the orebody to form the sidewalls of the stope. In general, longitudinal methods are used where the rock mass quality of the hanging wall rock is sufficiently competent to accommodate a substantial unsupported exposure along the hanging wall and/or footwall. Longitudinal longhole methods are well-suited to retreat mining, and have less waste development in comparison to transverse stoping, thus reducing development costs. The key limitation of longitudinal longhole stoping methods is the limited ability to depart from a set mining sequence – mining operations start at one end of a mining block, and work sequentially to the other end of the mining block.

In LHOS, the transverse layout has the advantage that a primary and secondary stoping sequence can be used to increase the potential production rate from a mining panel, but the disadvantages is the potential for higher dilution and lower ore recovery of the secondary stopes.

15.4.2 Rory's Knoll Deposit

15.4.2.1 Mining Context

The mining context of the Rory's Knoll deposit can be described as follows.

Geology: A continuous, steeply plunging pipe-like deposit comprised of stockwork mineralization exhibiting variability in gold grade distribution (Figure 15.23);

Geometry: A very thick (>100 m in plan) and large vertical extent (~500 m). (Figure 15.24);

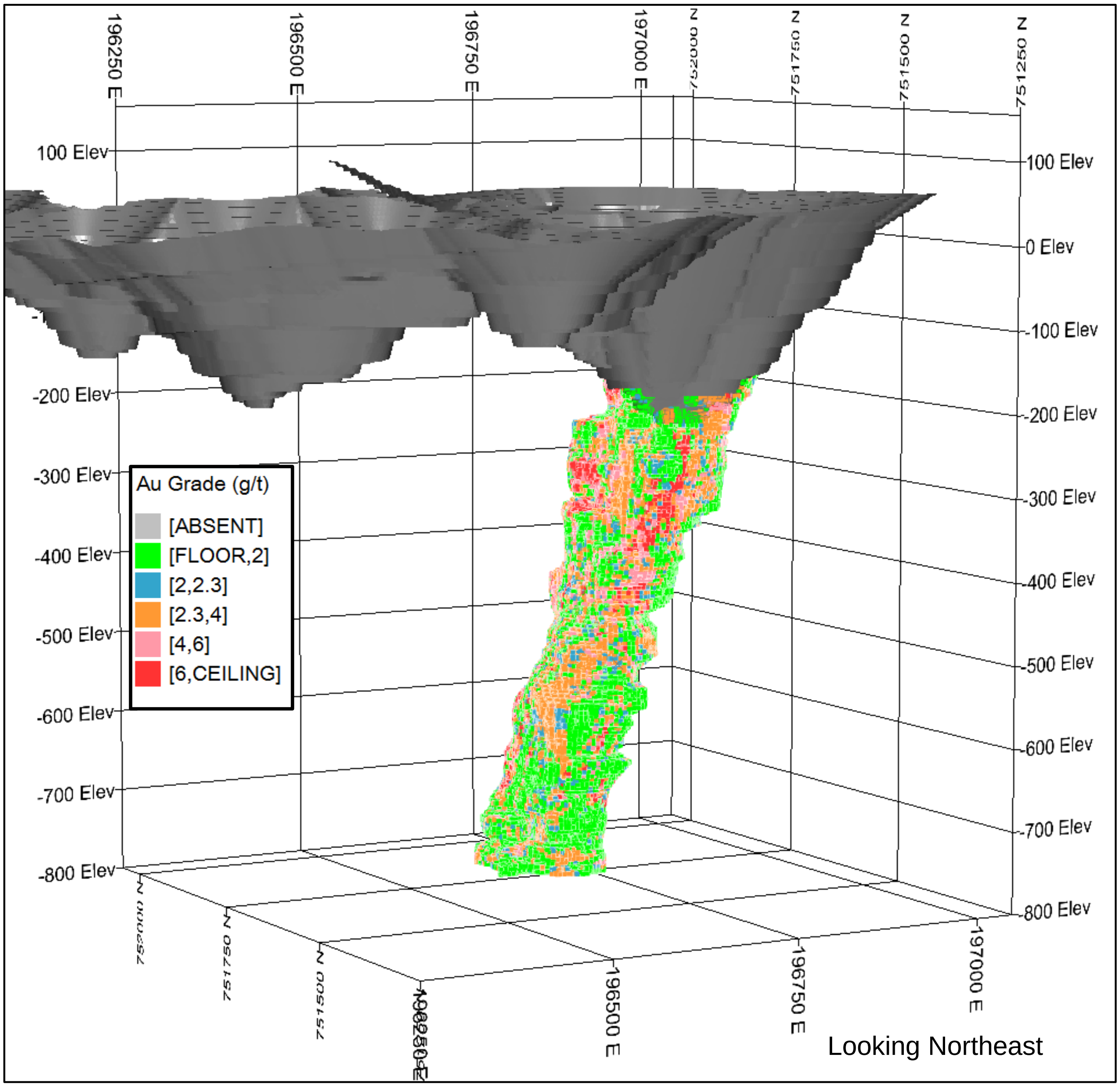
Grade Distribution: At the cut-off grade of 1.50 g/t Au, the Rory's Knoll orebody shows good continuity along strike and down dip. Below the open pit, the orebody geometry approximates a sub-vertical pipe of about >100 m diameter and ~500 m in vertical extent. A small number of barren zones have been identified within the deposit, however these have been readily accommodated in the open benching and SLR mining methods;

Rock Mass: Ground conditions are expected to be good in the tonalite (target mineralization) and surrounding host rock, but expected to be weak in the sericite schist shear zone that is adjacent to Rory's Knoll in some areas. In general, the contact with the shear zone is limited and will have minimal impact on mining;

Geological Structure: Contacts along the hanging wall and footwall (perimeter of Rory's Knoll 3D wireframe) are poorly defined, but it will be possible to visually identify gold mineralization in underground exposures;

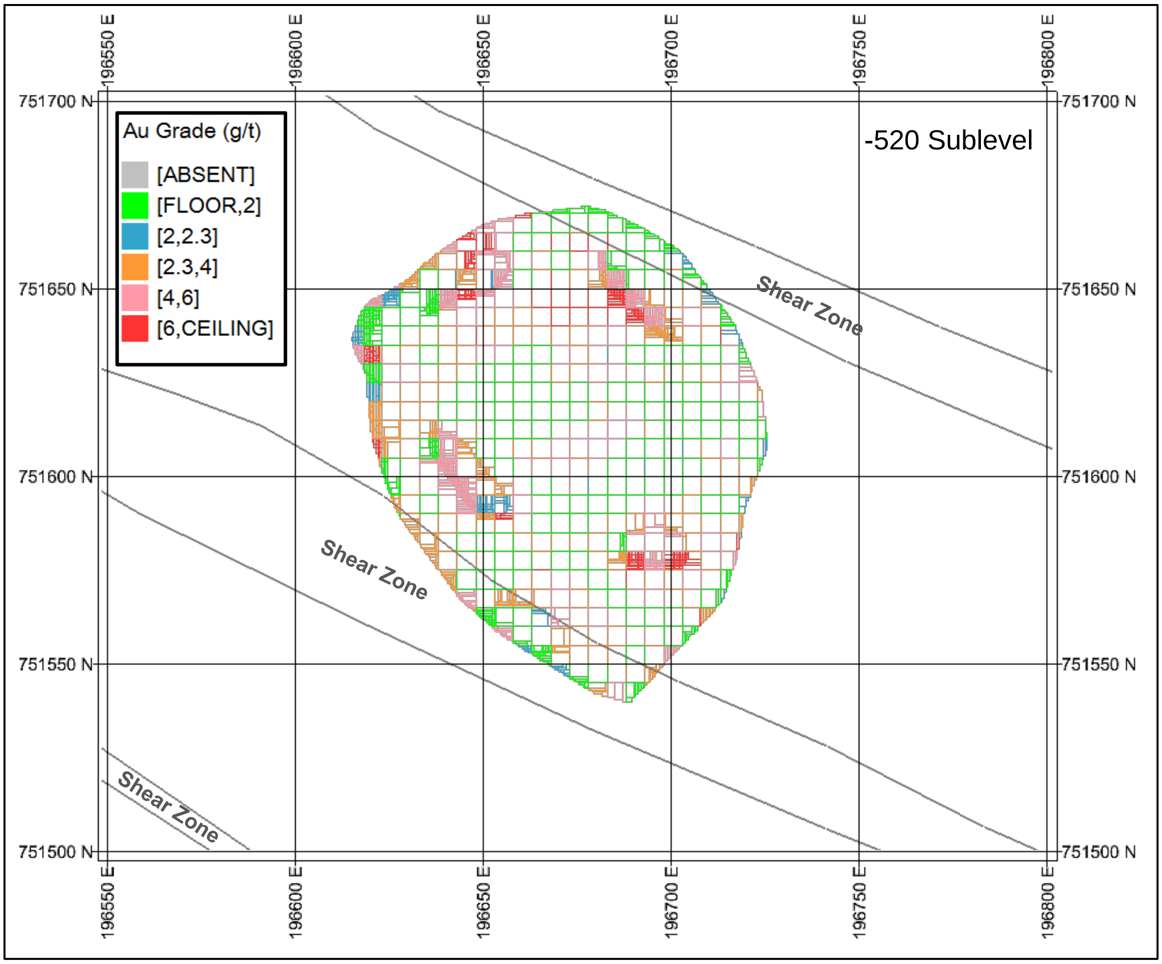
Disturbances It is a deep deposit and stress will be a significant factor when mining below a depth of about 600 m;

External Constraints The high surface temperature and humidity of the tropical climate will require cooled air for mining at depths below about 450 m from surface.



Source: SRK, 2017

Figure 15.23: Rory's Knoll: Grade distribution (looking Northeast)



Source: SRK, 2017

Figure 15.24: Rory's Knoll Plan view at -520 mRL

On the basis of this context, Rory's Knoll will utilise the open benching and Sublevel Retreat (SLR) mining methods.

The Rory's Knoll mine design incorporates Measured and Indicated resources above a cut-off of 0.72 g/t Au, however only material above a 1.50 g/t Au cut-off is considered for immediate mill feed, with the remaining material reported for a low grade stockpile. Cut-off grade calculations are presented in Section 14.4.1.2.

In accordance with the CIM standards for reporting Mineral Reserve estimates, only Measured and Indicated categories were used for the Mineral Reserve estimate. Inferred blocks were assigned zero grade.

15.4.2.2 Mine Design

The mine design accesses Rory's Knoll through a portal at the 75 mRL and exploits the deposit from -270 mRL to a depth of -770 mRL, starting with open benching and transitioning to SLR at the -420 mRL.

Access Ramp and Infrastructure

Decline development and sumps will be developed at gradients of 1:7. Stockpile bays have been included at 150 m intervals along the top section of the decline to facilitate decline development.

The portal will be collared south of Rory's Knoll at the process plant ROM pad. The decline will develop to the north side of Rory's Knoll where all the sublevel accesses and capital infrastructure will be developed from.

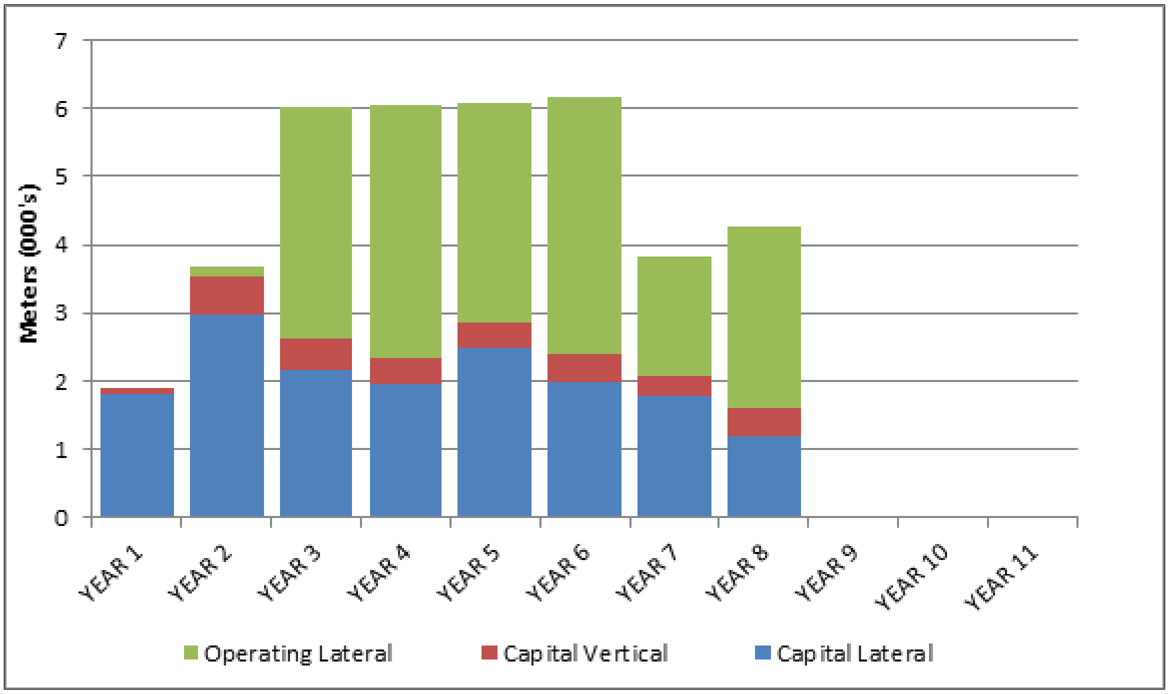
Level Development

All other lateral development will be developed at a gradient of 1:50 to facilitate water drainage off of the level and directed to a sump. Average modeled water inflows into the development areas are not high and it is therefore planned to use ANFO as the primary explosive; emulsion explosive will be used when water is encountered in the blast holes.

Advance rates used for scheduling are considered conservative. A total of 16 km of lateral capital development, 3 km of vertical capital development, and 19 km of operating lateral development will be completed over the life of the mine. The pre-production and LOM physicals and profiles are summarized in Table 15.15 and Figure 15.25.

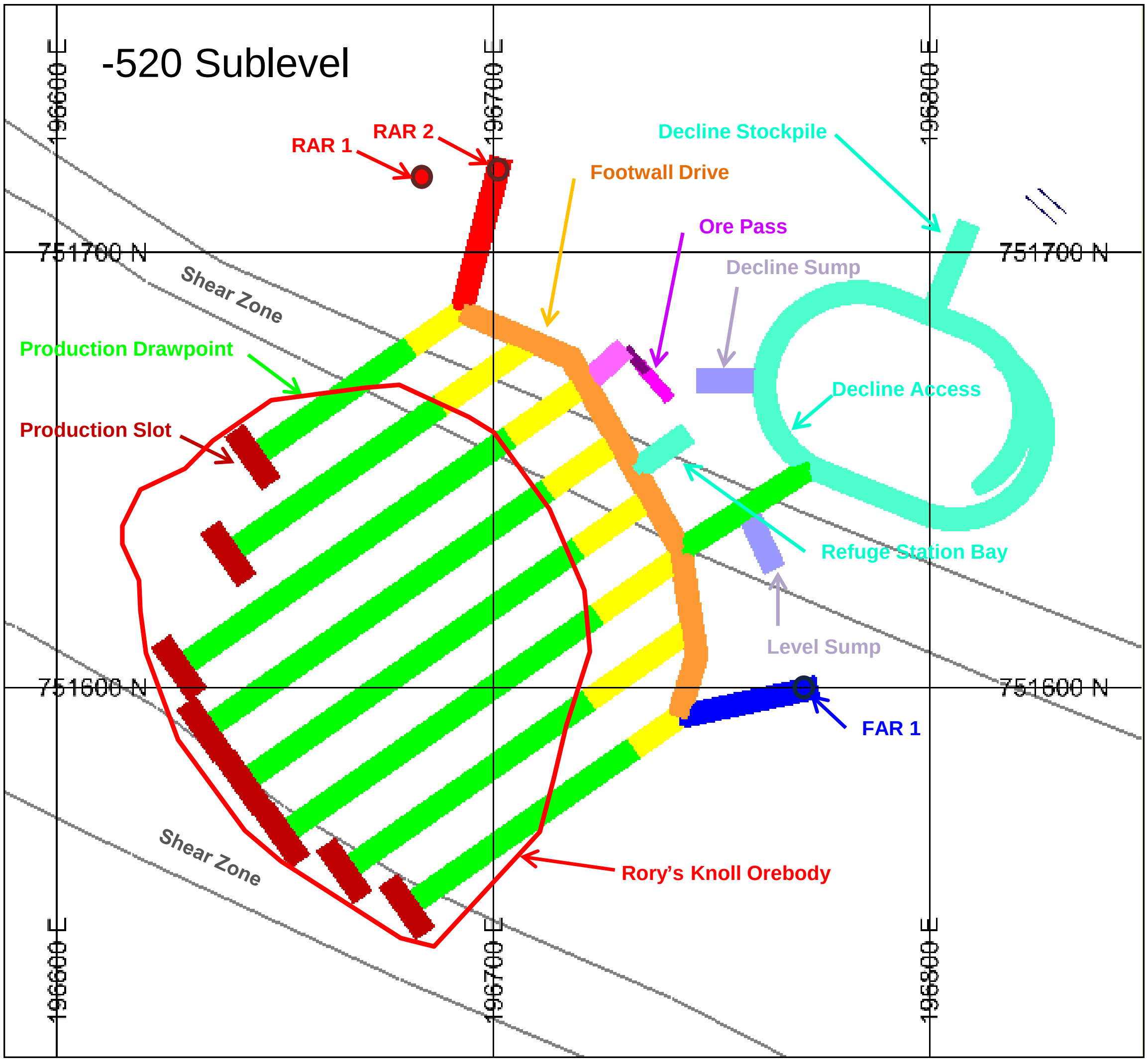
Table 15.15: Capital and operating development physicals

Type	Development Profile	Pre-Production Quantity	LOM Quantity
	(mW x mH)	(m)	(m)
Capital Lateral Development			
Decline	5.5 x 6.0 Arch	2,923	6,249
Sumps	5.5 x 5.0 Flat	195	810
Stockpiles	5.5 x 6.0 Arch	307	1,084
Sublevel Access	5.5 x 6.0 Arch	104	975
Sublevel Footwall Drive	5.5 x 6.0 Arch	210	2,749
Fresh Air Access	5.5 x 5.5 Arch	455	1,290
Return Air Access	5.5 x 5.5 Arch	566	1,770
Orepass Access	5.5 x 5.5 Arch	9	526
Pump Station	5.5 x 6.0 Arch	95	380
Explosives Magazine	5.5 x 5.0 Flat	137	281
Service Bay	5.5 x 6.0 Arch	100	168
Large Workshop Bay	7.1 x 9.5 Flat	0	111
Capital Vertical Development			
Fresh Air Raise	4.0 m Raise Bore	273	745
Return Air Raise	4.25 m Raise Bore	429	1,496
Orepass	3.0 x 3.0 Alimak	0	518
Orepass Drop Raise	2.4 x 2.4 Drop Raise	0	163
Operating Lateral Development			
Waste Drive (Stope Access)	5.5 x 5.0 Flat	106	4,519
Ore Drive (Drawpoint)	5.5x5.0 Flat	261	11,380
Slot Drive	5.5x5.0 Flat	30	2,805



Source: SRK, 2017

Figure 15.25: Rory's Knoll: LOM capital and operating development physicals



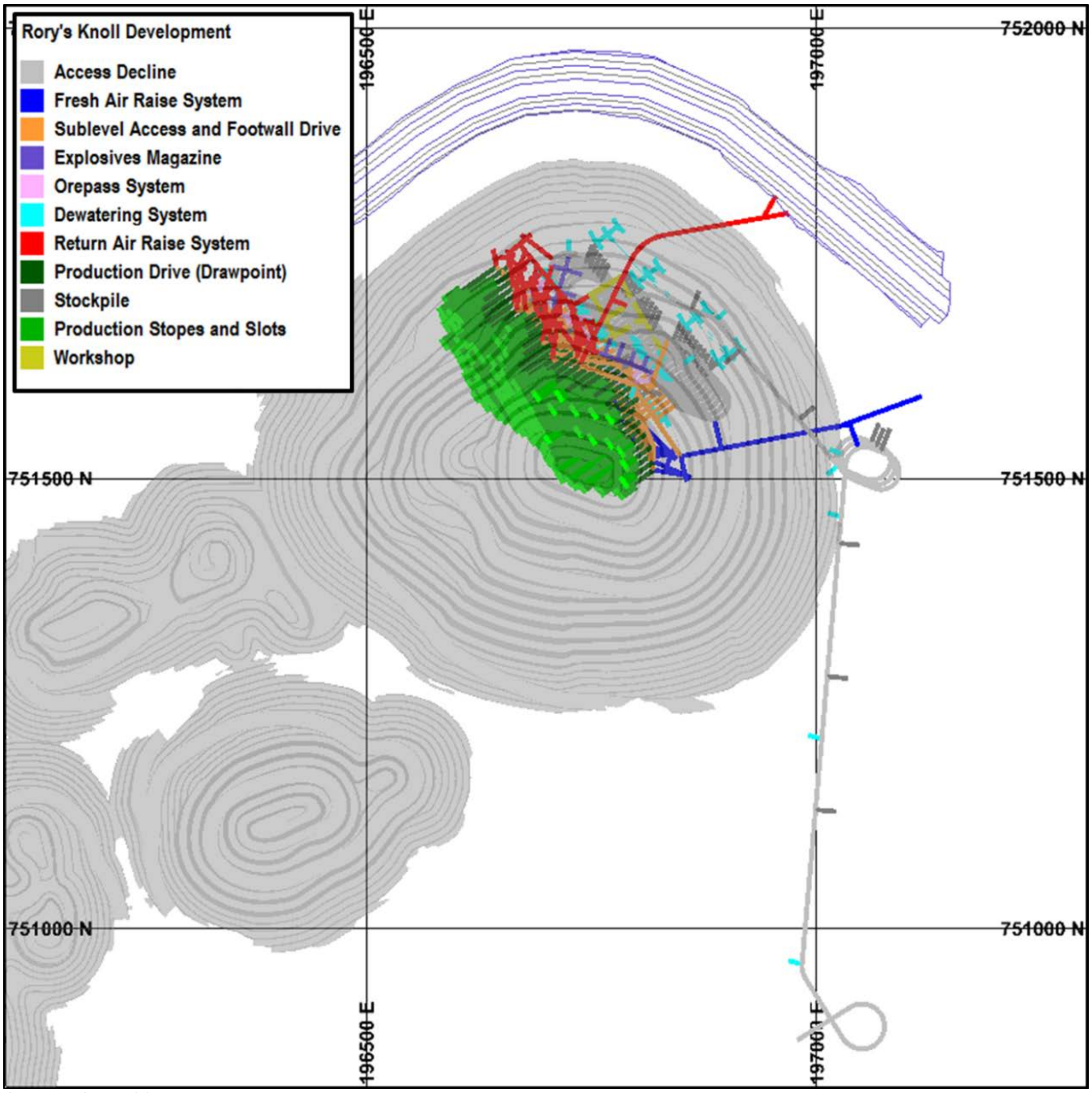
Source: SRK, 2017

Figure 15.26: Rory's Knoll: Layout of a typical sublevel (plan view)

The fresh air raise (FAR) will be advanced in 100 m increments using raise boring techniques. The return air raises (RAR) will be advanced using raise boring techniques in two sets of staggered 100 m legs, with each leg servicing alternating sublevels. The raises will be advanced in conjunction with the decline. A steel, self-supported enclosed egress ladder way will be installed in the fresh air raise on completion of all inter sublevel connections.

Sublevels will be spaced at 25 m vertical intervals and will be developed from the decline to within ~20 m of the orebody. The sublevel access drive will include a sump to collect water from the sublevel and water pumped from the decline face. A footwall drive will be driven along the strike of the orebody and will provide access to the drawpoints, ore passes, the FAR and RAR. Slot drives (each ~15 m long) will be developed at the end of each drawpoint; slot drives may connect adjacent drawpoints where they are of the same length.

Figure 15.27 shows the overall underground development layout plan for Rory's Knoll.



Source: SRK, 2017

Figure 15.27: Rory's Knoll: Underground development layout (plan view)

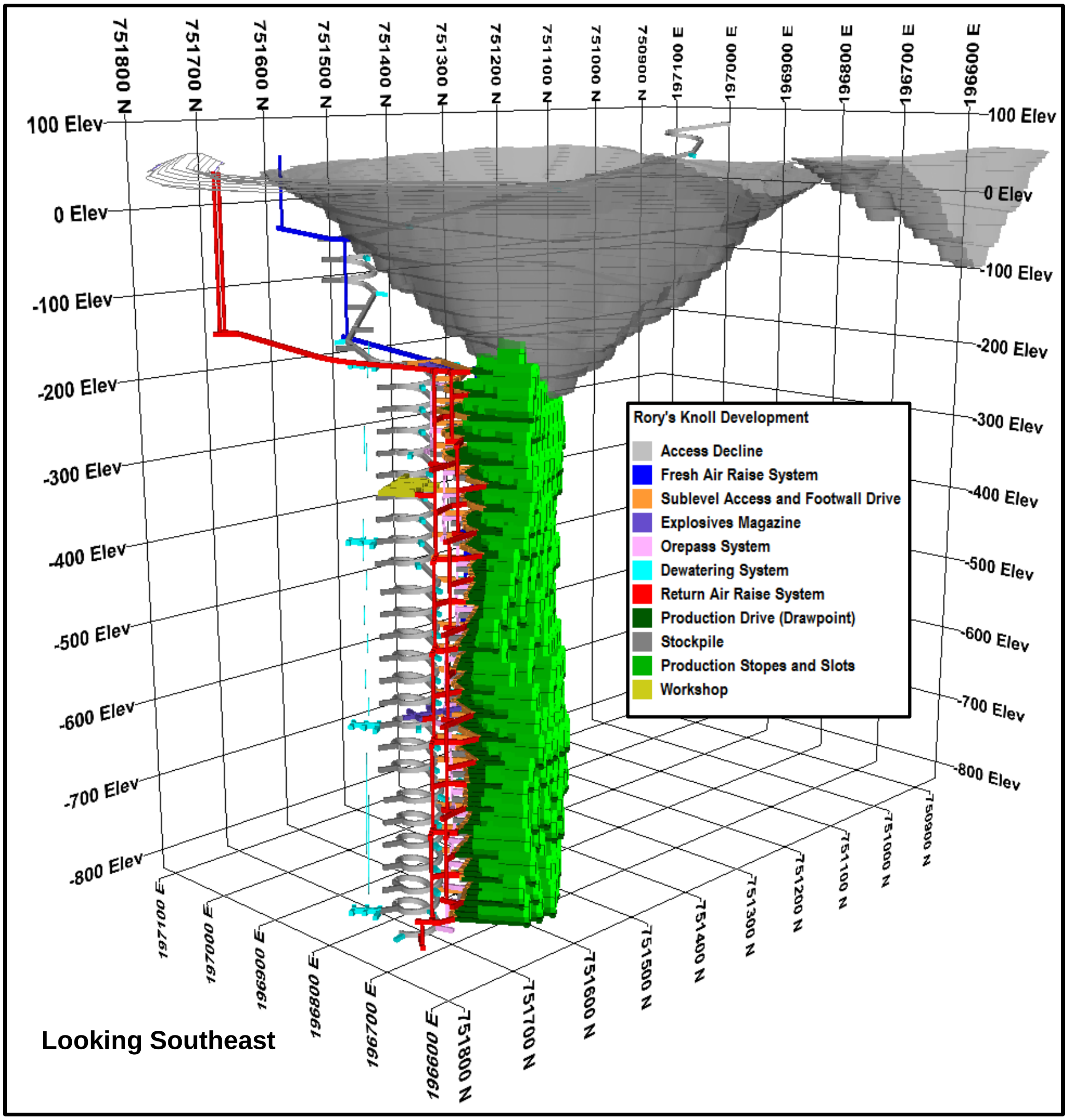
Stope Design

Sublevels for both open benching and sublevel retreat stopes have been designed at 25 m vertically with drawpoints 15 m apart. The stope design parameters have been adopted from modern sublevel caving geometries as they are analogous to those of both the open benching and the SLR mining methods. For this reason there is no difference in the designs of between the open benching and SLR stopes. The open benching and SLR stopes are differentiated in the dilution, extraction and recovery factors applied to them during production scheduling and material mixing estimations. The following stope design criteria has been used:

Production stopes:

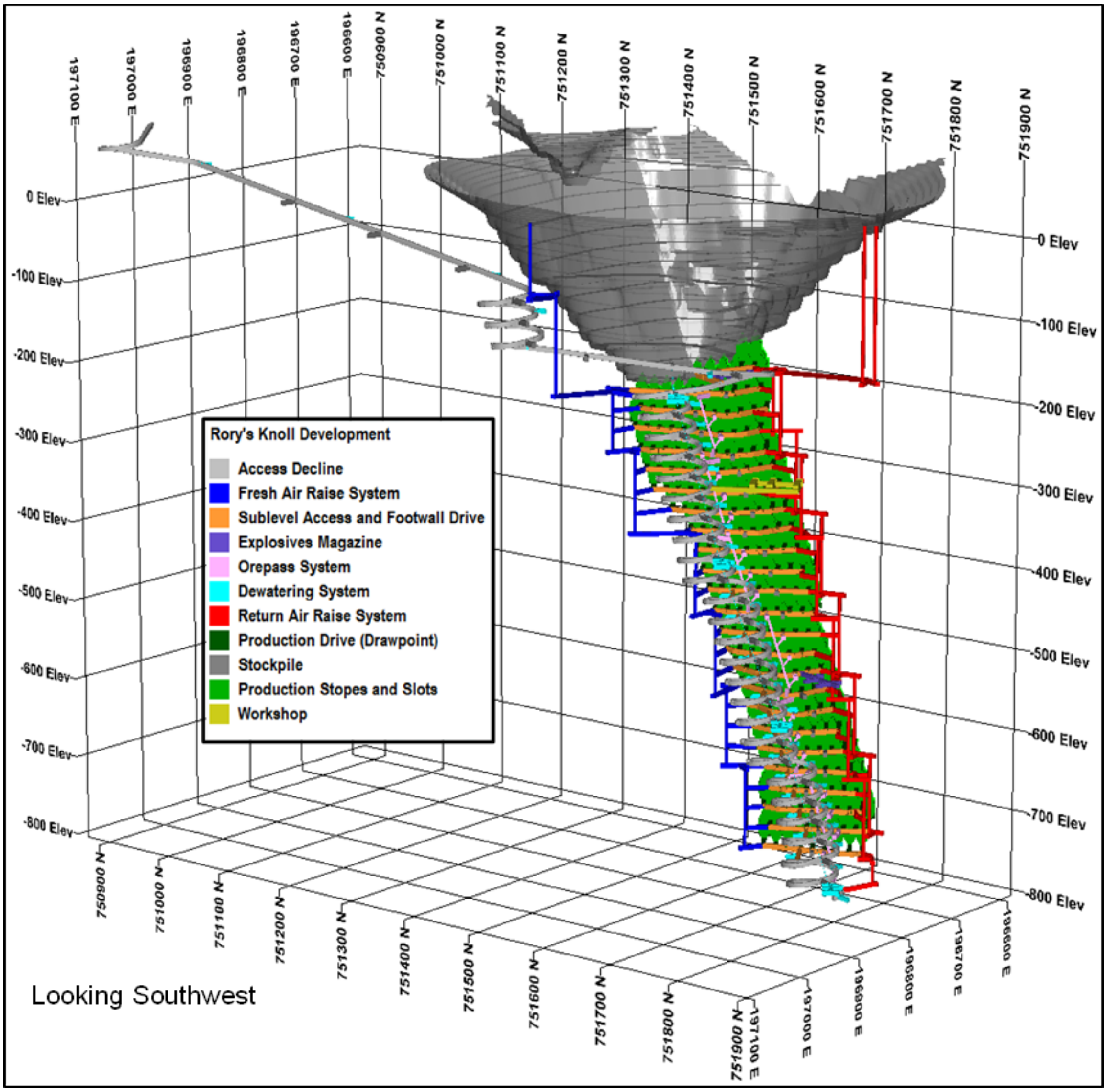
- 33 m stope height (ore drive backs to the top of the apex pillar)
- 15 m stope width
- Strike length equal to the orebody width (15 m to 120 m)
- 60˚apex pillar angle.
- Production slots:
- 25 m sublevel spacing
- 33 m slot height (ore drive backs to the top of the apex pillar)
- 15 m slot width
- 2.5 m burden

Figure 15.28 and Figure 15.29 show annotated views highlighting the key mine design features for Rory’s Knoll.



Source: SRK, 2017

Figure 15.28: Rory's Knoll: Mine design (looking Southeast)

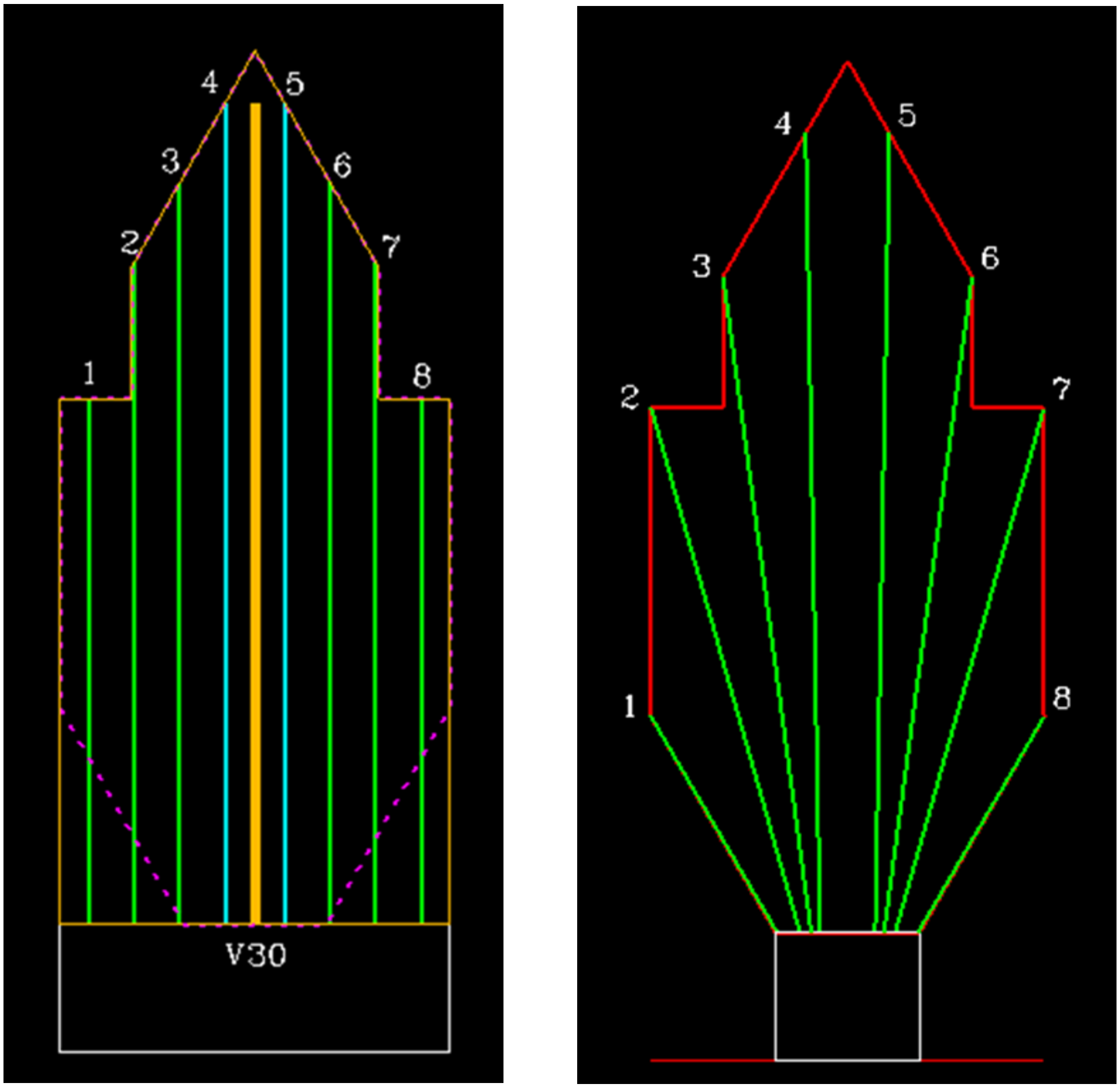


Source: SRK, 2017

Figure 15.29: Rory's Knoll: Mine design (looking Southwest)

Stope Cycle

Blast holes for the production slot will be drilled at 89 mm diameter by an In-The-Holehammer (ITH) drill rig. The slot relief hole will be reamed to 760 mm using a V-30 reaming attachment. Production blast holes will be drilled at 89 mm in diameter by top hammer electric-hydraulic drill rig. A stope drill factor of 10.9 t/m has been calculated and includes slot drilling and re-drills. A 2.5 m burden and 3.2 m toe spacing has been designed for production stopes. A 1.75 m burden and spacing has been designed for production slots. As the mining method is open to surface, wet holes are anticipated, thus production holes will be charged with emulsion explosives.



Source: SRK, 2017

Figure 15.30: Rory's Knoll: Typical production slot and stope ring (cross-section)

Stopes will be mucked with 17-tonne capacity diesel powered loaders (LHD) into an orepass. Some ore will be mined by tele-remotes when the open benching method is employed. The SLR method will employ manual loading of production material as the drawpoints will be chocked off. However, the LHDs will retain their tele-remote capabilities which may be required from time to time to safely remove drawpoint material hang ups.

Stope Sequence

The overall stope production sequence is a top down retreating sequence with the top sublevel leading the sublevels below it. A minimum 25 m lag between adjacent sublevels stope advance criteria is maintained for stability and safety. The orebody width is large enough to accommodate three sublevels in concurrent production.

The stope sequence on the sublevel starts with initiating production from the drawpoint in the middle of the orebody followed by mining adjacent stopes in both directions. A maximum lag of 15 m is maintained between neighbouring stopes.

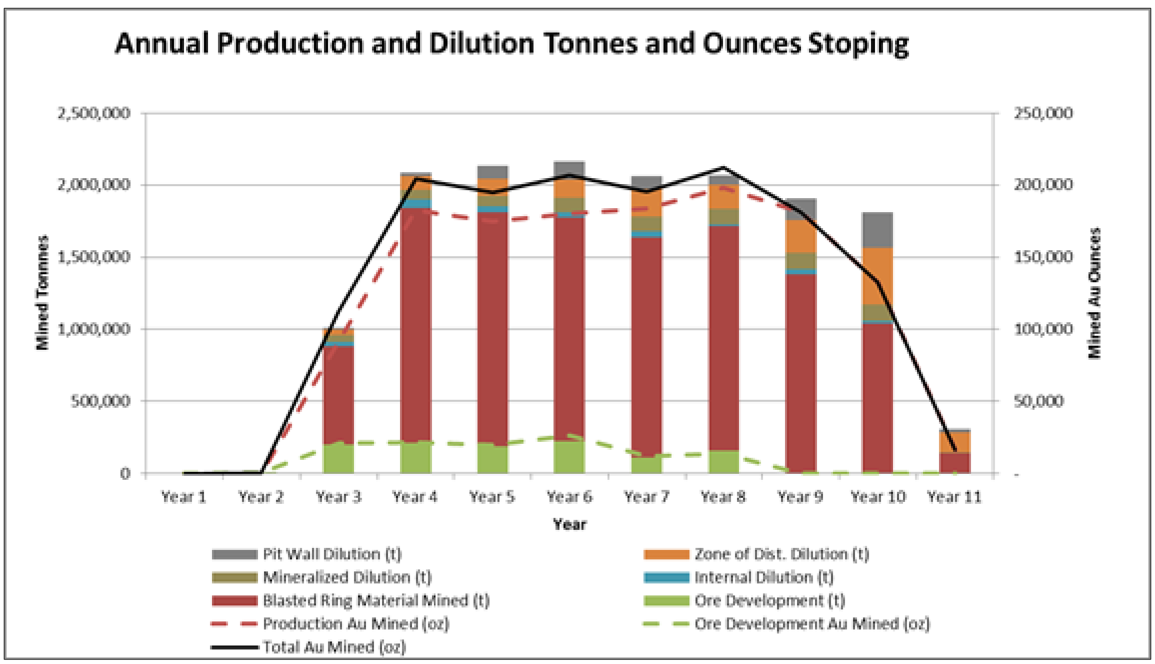
15.4.2.3 Development and Production Schedule

Production Rate

The production rate for the Rory's Knoll underground deposit is a function of the orebody geometry and continuity, the prevailing ground conditions, the number of available stoping areas and the expected productivity for each stope; the production resources were scheduled to achieve a practical production output.

The expected stope productivity is about 500 tonnes per drawpoint per day and based on benchmarking comparable mines. The expected daily productivity of 5,200 tonnes per day is based on a maximum of twelve active drawpoints.

After the initial two year pre-production period, Rory's Knoll will be mined at an average capacity of approximately 1.9 Mtpa over seven years of a nine year mine life. Figure 15.31 presents the stope production schedule.



Source: SRK, 2017

Figure 15.31: Rory's Knoll: Production schedule

Development Schedule

A total of 16 km of lateral capital development, 3 km of vertical capital development, and 19 km of operating lateral development will be completed over the life of the mine.

The development rates used for Rory's Knoll schedule are presented in Table 15.16.

Table 15.16: Rory's Knoll: Development rates

Description	Development Rate (m/d)
Decline	4.0
Sublevel Access	4.0
Ore and Waste Draw Point	3.3
Ventilation Access	4.0
Return Air Raise	2.5
Rising Main	12.0
Escape Way	2.5
Ore and Waste Drive	3.3
Stockpiles	3.3
Sump	3.3
Large Maintenance Bays	1.0
Magazine and Explosives	2.6
Pump Station	2.6
Foot-wall Drive	4.0
Slot Drive	3.3

The LOM lateral development factor (capital and operating) is 360 t of mineralized material per meter of development advance.

A summary of the LOM physicals for pre-production and development of Rory's Knoll is presented in Table 15.17.

Table 15.17: Rory's Knoll: LOM development schedule

Underground Schedule	Units	Total	1	2	3	4	5	6	7	8
Capital Development - Lateral										
Decline	m	6,249	1,456	1,298	522	589	825	648	766	144
	t	559,939	130,478	116,270	46,833	52,805	73,911	58,072	68,640	12,929
Other	m	7,309	134	1,167	1,157	1,145	1,081	1,118	789	718
	t	626,786	10,917	97,967	100,078	99,235	93,560	96,564	68,475	59,989
Capital Infrastructure	m	2,834	216	515	482	233	584	233	223	347
	t	249,127	18,443	41,909	53,647	19,566	47,658	19,568	19,273	29,063
Total Capital Development	m	16,392	1,806	2,980	2,162	1,968	2,490	1,999	1,777	1,209
	t	1,435,852	159,839	256,146	200,559	171,607	215,130	174,203	156,388	101,981
Capital Development - Vertical										
Raisebore	m	2,242	86	565	359	358	284	284	260	47
	t	84,134	2,855	21,236	13,954	13,218	10,655	10,655	9,730	1,832
Drop Raise	m	163	0	0	0	10	0	0	30	123
	t	2,621	0	0	0	161	0	0	484	1,976
Alimak	m	518	0	0	97	0	97	99	0	224
	t	13,045	0	0	2,442	0	2,441	2,507	0	5,655
Total Vertical Capital Development	m	2,922	86	565	456	368	380	383	290	394
	t	99,800	2,855	21,236	16,395	13,380	13,097	13,162	10,213	9,463
Operating Development										
Waste	m	4,488	0	51	825	969	774	916	308	646
	t	345,861	0	3,916	63,552	74,651	59,663	70,559	23,711	49,807
Ore	m	14,216	0	98	2,586	2,738	2,433	2,868	1,467	2,025
	t	1,101,831	0	7,617	200,394	212,159	188,630	222,367	113,754	156,909
Gold Grade	g/t Au	3.26	0	2.51	3.27	3.21	3.26	3.67	3.29	2.75
Contained Au Metal	Oz Au	115,555	0	614	21,072	21,903	19,787	26,252	12,032	13,895
Total Operating Development	m	18,704	0	149	3,411	3,706	3,207	3,784	1,775	2,671
Total Operating Development	t	1,447,691	0	11,534	263,947	286,810	248,293	292,926	137,465	206,716

Production Schedule

The Rory's Knoll production schedule is presented in Table 15.18.

LOM Schedule

The Life of Mine (LOM) Plan quantity and grade estimate for Rory's Knoll is presented in Table 15.19.

Table 15.18: Rory's Knoll: LOM production schedule

Description	Units	Total	1	2	3	4	5	6	7	8	9	10	11
SLR Tonnes Mined	t	13,715,495	0	0	749,711	1,731,078	1,847,848	1,883,442	1,850,688	1,874,518	1,839,570	1,758,555	180,084
SLR Gold Grade mined	g/t Au	3.00	0.00	0.00	3.74	3.21	2.90	2.96	3.05	3.27	3.04	2.32	1.98
SLR Metal Mined	Oz Au	1,321,481	0	0	90,078	178,925	172,413	179,385	181,335	197,019	179,935	130,930	11,459

Note: Year 1 and Year 2 are pre-production years and do not have any stope production.

Table 15.19: Rory's Knoll: LOM plan

Description	Units	Total	1	2	3	4	5	6	7	8	9	10	11
Tonnage Mined	t	14,817,326	0	7,617	950,106	1,943,238	2,036,478	2,105,809	1,964,442	2,031,427	1,839,570	1,758,555	180,084
Gold Grade mined	g/t Au	3.02	0	2.51	3.64	3.21	2.94	3.04	3.06	3.23	3.04	2.32	1.98
Metal Mined	Oz Au	1,437,036	0	614	111,150	200,828	192,200	205,638	193,367	210,914	179,935	130,930	11,459

Note: Year 1 and Year 2 are pre-production years and do not have any stope production.

15.4.2.4 Mobile Equipment Requirements

The mobile equipment fleet required to extract the ore from Rory's Knoll at a nominal rate of 1.9 Mtpa is presented in Table 15.20.

Fleet productivities have been based on first principal calculations, benchmarking, and practical experience. Equipment requirements were based on the estimated required operating hours in each period and the number of units of each piece of equipment needed to meet those hours.

Table 15.20: Underground mobile equipment list for Rory's Knoll

Equipment Type	Model	LOM Quantity	LOM Rebuilds	Max Fleet Size
Jumbo	Sandvik DD421	4	2	2
Bolter	Sandvik DS311	5	3	3
UG loader	Sandvik LH517	7	4	4
UG truck	Sandvik TH551	23	14	13
LH drill	Sandvik DL431	2	2	2
ITH drill	ITH Drilling	1	1	1
Development charge	Normet Charmec MC605 DA	1	1	1
Production charge	Normet Charmec LC605 VE	1	1	1
Services	Normet	14	11	8
Grader	Veekmas FG 15 C	4	4	1
Light vehicle	Toyota Hurth	39	24	12

15.4.2.5 Ventilation

The purpose of the mine ventilation design is to provide sufficient quantity of air for all the mining activities while satisfying the respective legislative requirements.

Ventilation Design Criteria

The common ventilation criteria used for Rory's Knoll are listed below and were governed by:

- legislative requirements,
- number of operating diesel equipment,
- daily produced tonnage,
- mining environment conditions, and
- sound ventilation practice.

Table 15.21 summarizes the criteria used for the Rory's Knoll ventilation design.

Table 15.21: Ventilation design criteria

Criteria	Value	Units of Measurement
General Criteria		
Main Fan Maximum Static Pressure	3500	Pa
Maximum Pressure in Flexible Duct	2500	Pa
Maximum Pressure in Rigid Duct	3800	Pa
Main System Leakage	15	%
Fan One Size Smaller than Duct		
Main Airway Velocities		
Return Air Raises (no personnel)	18-22	m/s
Fresh Air Raises (no personnel)	18-22	m/s
Intake Airways (personnel)	4-6	m/s
Return airways (personnel and tramming)	6-8	m/s
Dedicated Return Airways (no personnel)	8-15	m/s
Declines, ramps	6-8	m/s
Main Airway Friction Factors		
Blasted airways [intake/return] [Irregular sides]	0.011	kg/m^3
Galvanised vent ducting / Layflat ducting	0.002	kg/m^3
Raise bore holes or lined shafts [including losses]	0.007	kg/m^3
Men & material and equipped shafts [concrete lined and streamlined]	0.030	kg/m^3
Alimak Raises	0.013	kg/m^3
Raise bore holes manway equipped	0.012	kg/m^3

The target production rate for Rory's Knoll is 5,200 tonnes per day. On the basis of proposed diesel fleet presented in Table 15.22 it was determined that about 480 m^3/s of fresh air would be sufficient to provide ventilation for all production and development activities for the Rory's Knoll orebody.

Table 15.22: Rory's Knoll: Ventilation allocations

Equipment Type	Model	Rated kW	Utilization	Max. Required	Used kW
Jumbo	Sandvik DD421	110	15%	2	33
Bolter	Sandvik DS411	110	15%	3	50
UG loader	Sandvik LH517	298	56%	4	668
UG truck	Sandvik TH551	579	67%	13	5,043
LH drill	Sandvik DL421-7	110	10%	2	22
ITH drill	ITH Drill (V30 Slot Raises)	130	10%	1	13
Development charge	Normet Charmec MC605 DA	110	45%	1	50
Production charge	Normet Charmec LC605 VE	110	45%	1	50
Services	Normet Utilift MF 540	103	40%	8	330
Grader	Veekmas FG 15 C	129	60%	1	77
Light vehicle	Toyota Hurth	151	35%	12	634
Total For Underground Fleet				**48**	**6,968**
Diesel Required Air Volume m^3/s					418
Leakage (Contingency) 15%					63
Total required volume m^3/s say 440					**481**

In this study, the Rory's Knoll deposit is mined to the -770 mRL. The VUMA-network simulation program was used for the 2012 feasibility study (SRK 2012) mainly due to the mine air cooling. For this study, the Ventsim Visual simulation program was also used to determine the overall pressure drop throughout the mine and to estimate the updated cooling requirements; this software is widely used in the mining industry.

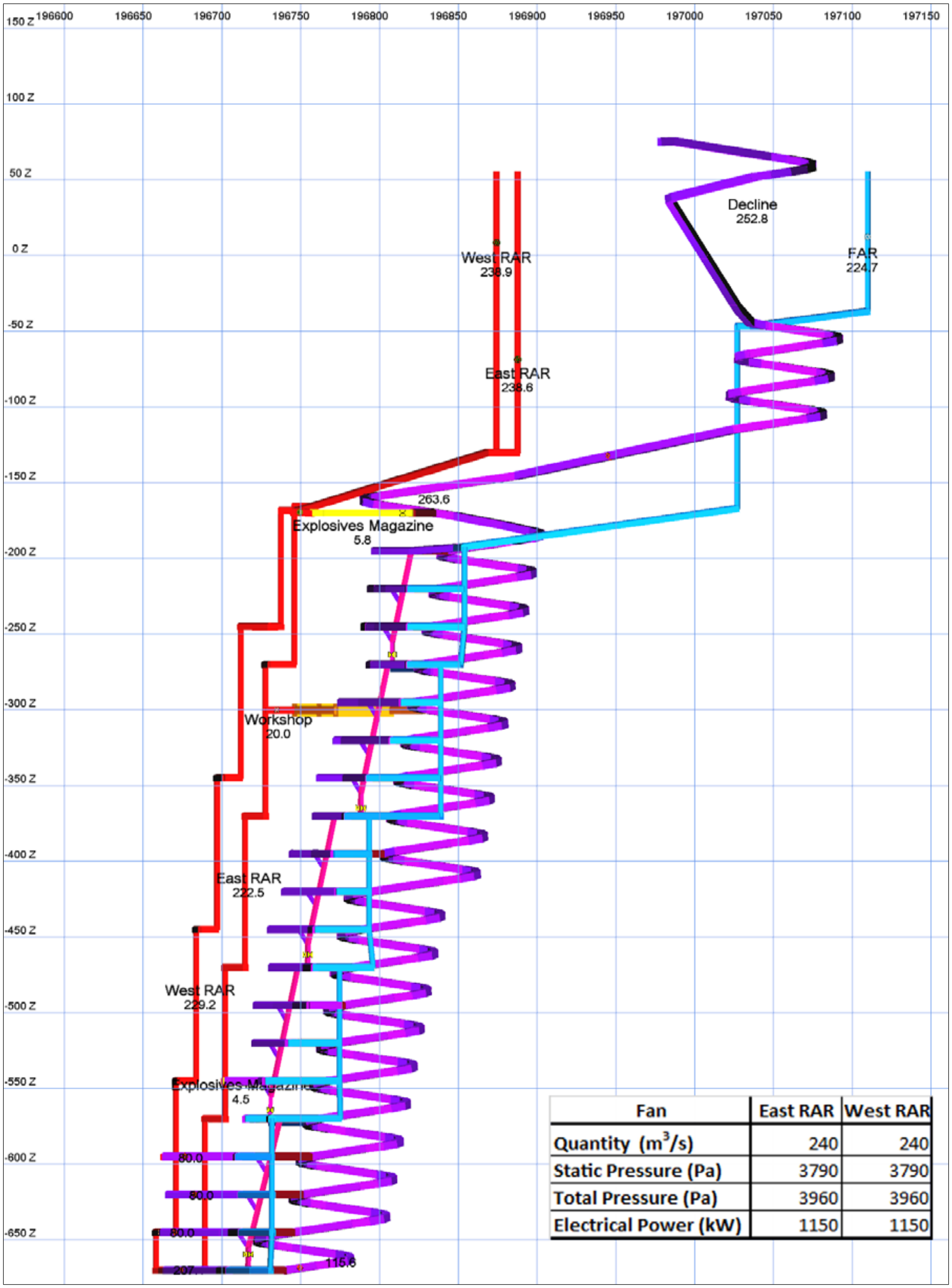
Primary Ventilation

An exhaust ventilation system is proposed for Rory's Knoll deposit with main fans located at the top of the Return Air Raises (RARs). The advantage of this arrangement is that the haulage decline entrance remains unobstructed.

Ventilation regulators will be provided at the RAR connection on each sublevel. Typically these would be roll-up garage type doors, adjustable louvers or guillotine type regulators operated by chain blocks. The flow will be regulated in range of zero to 40 m^3/s.

The supply air volume will gradually increase for each deposit until the full production stage has been reached.

Two independent ventilation circuits are proposed for the Rory's Knoll orebody. One circuit will be dedicated to the 5.5 mW x 6.0 mH decline and the other one to the production sublevels via the Fresh Air Raise (FAR). Figure 15.32 shows the planned primary circuit for Rory's Knoll.



Fan	East RAR	West RAR
Quantity (m^3/s)	240	240
Static Pressure (Pa)	3790	3790
Total Pressure (Pa)	3960	3960
Electrical Power (kW)	1150	1150

Figure 15.32: Rory's Knoll: Peak primary circuit airflow quantities (looking Southwest)

Fan Arrangements

Figure 15.33 shows the general arrangement for the proposed main fan stations. The size of the main fans were based on the expected peak operating capacity during the mine life, according to the Ventsim model.

Each fan will be equipped with inlet box turning vanes and evasé. A floating shaft will be used to connect the drive to the impeller. Two-speed motors, adjustable pitch in motion or Variable Frequency Drive could be used to control the required air volume through the development and production phases of the operation.

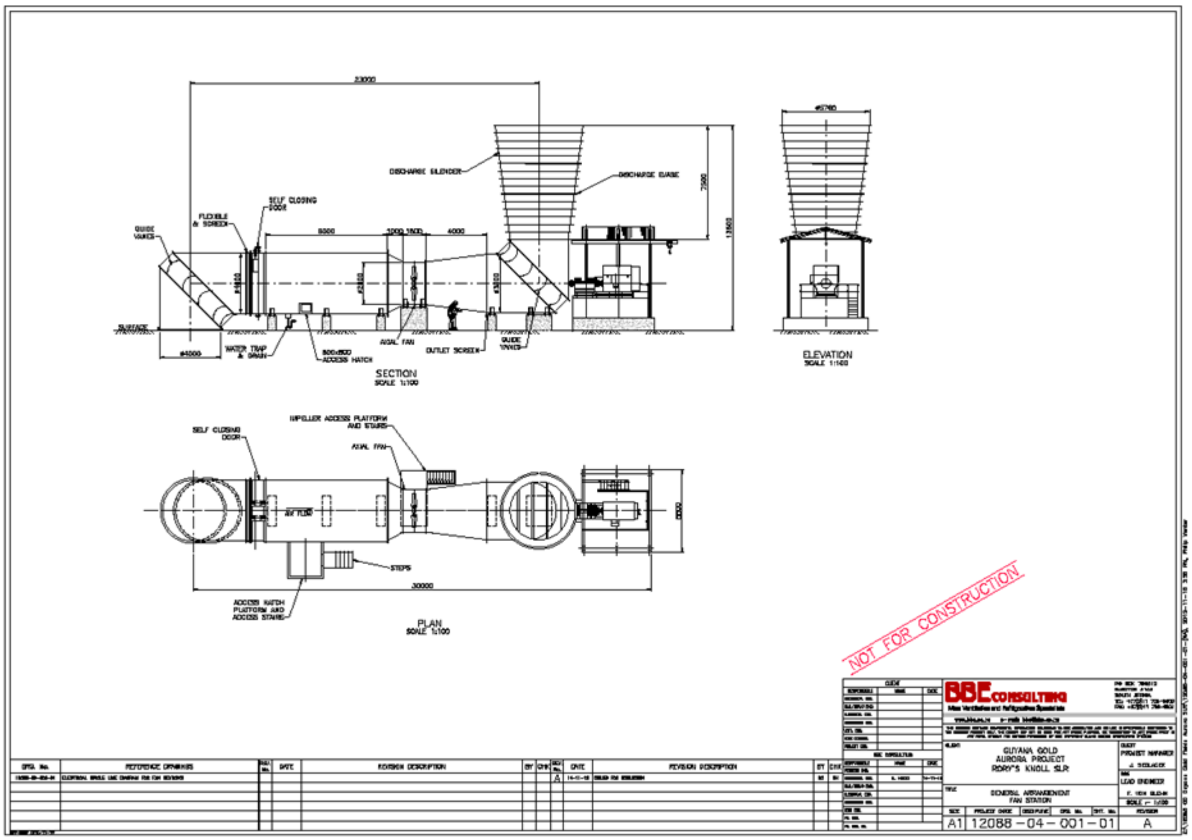


Figure 15.33: Rory's Knoll: Exhaust Fans - General Arrangement

At Rory's Knoll, two 2800 mm diameter axial flow fans are proposed for each ventilation raise. The fans will operate at peak operating duty of 240 m^3/s @ 3790 Pa. Each fan will be equipped with a 1200 kW motor.

Secondary Ventilation

The purpose of the auxiliary ventilation for development or active headings is to take air from the main air circuit and distribute it to the individual workings.

Secondary ventilation to the development and active headings will be provided by auxiliary fans connected to flexible ducting. A 1000 mm to 1800 mm diameter flexible ducting connected to 900 mm to 1400 mm diameter fans respectively will be used depending on the equipment operating in the headings. Ventilation on demand is proposed so only active headings are fully ventilated to reduce the overall ventilation and power demand.

Development and Services Ventilation

In general, there are three types of development encountered on the Aurora Gold Project:

- Long-decline development for accessing the deposit,
- Long-lateral development to access the sublevels, and
- Short lateral development to access and ventilate the production areas.

The decline development in Rory's Knoll will be ventilated with forced air with twin rigid duct in 1000 m segments (approximately 150 m vertically). The forced air mode was selected as it provides more effective ventilation of the face and there is no need for a scavenger vent system to boost the face airflow velocity. The exhaust will free flow back up the decline until the last breakthrough into the RAR or portal is reached. Once the next breakthrough to the RAR is complete the ducting will be moved to develop the next segment of the decline. The whole process will be repeated until the decline is completed.

Sublevel development will initially be ventilated from the decline until the development into the RAR has been established.

Fresh air for the development leading to the production areas will be ducted from the sublevel drives and exhausted free flow into the exhaust raises.

Pump stations, sub-stations and other service ventilation centres will be ventilated in series with the fresh air delivered to the production zones. Maintenance workshops and explosive magazines will be ventilated in parallel to the fresh air and returned directly to the RAR.

Development Fans & Duct Specifications

For long developments, such as the decline, it is proposed to use 1400 mm diameter fans equipped with 110 kW motors and connected to 1800 mm diameter flexible ducting. There will be two rigid duct lines installed: one for the decline development and a second for level development. The fans will provide up to 40 m^3/s @ 2.0 kPa over 1000 m distance.

Ventilation of sublevel development and production areas will be provided by 900 mm diameter fans equipped with 90 kW motors and connected to 1000 mm diameter flexible ducting. This arrangement will be capable of delivering 20 m^3/s up to a distance of 200 m.

Figure 15.23 shows the number of estimated auxiliary fans to be purchased, install and expected usage for Rory's Knoll.

Table 15.23: Rory's Knoll auxiliary fans to be purchased

Property	# Decl. Fans (@ 110 kW)			# Dev. And Prod. Fans (@ 90 kW)			Total Power
	Purchased	Max Installed	Max Utilized	Purchased	Max Installed	Max Utilized	(kW)
Rory's Knoll	3	2	2	50	33	10	1120

Air Cooling

Air Cooling Design Criteria

The VUMA-network simulation program was used to estimate the air cooling requirements in the this study. The following criteria were used to determine the requirement and size of the refrigeration plant for Rory's Knoll:

- surface ambient design condition,
- geothermal properties,
- production face conditions, and
- ramp conditions.

Surface Ambient Design Conditions

The values below are based on hourly averages of measurements made at the project site over the periods 2006 to 2009 and 2011:

- Summer design wet-bulb /dry-bulb temperatures: 26.0°C /31.4°C,
- Barometric pressure: 101.0 kPa,
- Surface air density: 1.14 kg/m^3.

Geothermal Properties

Virgin Rock Temperature (VRT) can be described by:

- VRT (°C) = 23.77 + 0.01169 x depth below collar (m),
- The VRT between -79 mRL and -979 mRL ranges from 24.7°C to 35.2°C.

The recent thermal properties are indicated in Table 15.24.

Table 15.24: Rock thermal properties

Thermal Properties	Units of Measurement	Ore Body	Host Rock
Thermal Diffusivity	M^2/s	1.19×10^{-6}	1.51×10^{-6}
Thermal Conductivity	W/m°C	3.5	4.3
Specific Heat	J/kg°C	1,041	1,027
Bulk Density	kg/m^3	2,820	2,780

Production Face Conditions

The maximum wet-bulb temperature in the production zones will be limited to 28.5°C and the maximum dry-bulb temperature will be 37°C.

Ramp Conditions

The maximum wet-bulb temperature in the ramp will be limited to 32°C provided the trucks in the ramp are fitted with enclosed air-conditioned cabins. This temperature may not be exceeded except in emergency conditions. If a truck breaks down it would be repaired/salvaged by personnel who would have to work for short periods in a high temperature atmosphere but they may be provided with the cooling suits. The maximum dry-bulb temperature will be 37°C.

In the event of vehicle failures/breakdowns personnel should only be allowed to work outside the air- conditioned cabs for short periods in these elevated temperatures.

Estimated Heat Loads

A full interactive computer simulation using Ventsim Visual software was applied to determine air temperatures, flow rates, heat loads and cooling requirements. The simulation considers the following heat load components:

- Surrounding rock – exposed rock in all openings,
- Broken rock – blasted rock,
- Auto-compression – conversion of potential energy into enthalpy, increases with depth,
- Diesel powered equipment – conversion of combusted fuel to heat,
- Auxiliary fans – conversion of electrical energy into heat energy,
- Ground water – when exposed to air and at the temperatures higher than VRT,
- Other sources – such as pumps, electrical sub-stations, workshops, lighting, explosives, strata movement, etc., assigned values from experience based on the size of operation and production rates.

Ventilation Costs

Capital costs reflect the purchase and installation of the main fan stations and air cooling system. Operating costs were based on the estimated number of equipment and production schedules for each deposit.

Summary of the estimated capital costs for the purchase and installation of the two surface fans at Rory's Knoll is presented in Table 15.25.

Table 15.25: Rory's Knoll: Capital cost of surface fan stations

Item	Each ($)	Quantity	Total ($)
Axial Fan-motor set and ducting (240 m3/s @ 4000 Pa, Motor 1200 kW 6P 4.1 kV–60 Hz)	1,608,000	2	3,216,000
Electricals and control	635,866	2	1,271,731
Civils and structural steelwork	304,500	2	609,000
Air Cooling			16,561,119
Subtotal of Direct Costs			**21,657,850**
EPCM, project service and indirects (15% of directs)			3,248,678
Contingency (10% of directs)			2,165,785
Subtotal of Indirect Costs			**5,414,463**
Total Cost			**27,072,313**

Ventsim was used to estimate the operating cost of the main fan at different stages of the mine life.

presents the estimated power costs for main and auxiliary fans over the expected operating life of Rory's Knoll. Auxiliary fan operating costs were prorated based on the expected number of auxiliary fans necessary to support decline and sublevel development as well as production activities for each period.

Table 15.26: Rory's Knoll: Ventilation power and operating cost estimate

Item	YEAR				
	1	3	5	8	10
Fan Operating Costs					
Primary Ventilation Fans					
Power (kW)	-	382	1,819	2,300	702
Yearly Energy Consumption (kWhr)	-	3,346,320	15,934,440	20,148,000	6,149,520
Yearly Energy Cost* ($M)	-	0.76	3.63	4.59	1.40
Development Ventilation Fans					
Power (kW)	220	220	220	220	-
Utilization/day (hrs)	24	24	24	24	-
Yearly Energy Consumption (kWhr)	1,927,200	1,927,200	1,927,200	1,927,200	-
Yearly Energy Cost ($M)	0.44	0.44	0.44	0.44	-
Production Ventilation Fans					
Power (kW)	-	390	900	900	900
Yearly Energy Consumption (kWhr)	-	3,416,400	7,884,000	7,884,000	7,884,000
Yearly Energy Cost ($M)	-	0.78	1.80	1.80	1.80
Maintenance					
Yearly Maintenance Costs ($M)		0.28	0.21	0.21	0.21
Total Fan Cost					
Yearly Operating Costs ($M)	**0.44**	**2.26**	**6.08**	**7.04**	**3.41**
Refrigeration Costs					
Energy					
Required Cooling Capacity (MW)		8	13	15	10
Power (kW)	-	2,084	2,947	3,378	2,516
Yearly Energy Consumption (kWhr)	-	18,260,031	25,811,755	29,587,617	22,035,893
Yearly Energy Cost ($M)		4.16	5.89	6.75	5.02
Maintenance					
Yearly Maintenance Costs ($M)		0.86	0.03	0.05	0.03
Total Refrigeration Cost					
Yearly Operating Costs ($M)	**-**	**5.03**	**5.91**	**6.80**	**5.05**
Total Ventilation Costs					
Yearly Operating Costs ($M)	**0.44**	**7.28**	**12.00**	**13.84**	**8.46**

15.4.2.6 Air Cooling System Descriptions

The underground mine cooling and refrigeration equipment will include a large surface bulk air cooler (BAC) and a central refrigeration system, located on surface near the FAR.

The refrigeration and air cooling facility will operate automatically and will be monitored remotely without the need for permanent on-site operators. The ventilation and refrigeration systems and equipment selection will be optimized with regard to energy efficiency. Further, diurnal and seasonal energy management procedures will be applied to these systems to maximize the power savings.

Bulk Air Cooler

The BAC will cool 270 kg/s of air. The BAC will be in the form of a horizontal spray chamber constructed in concrete in a single cell. Two inlet air cooler fans will force air through the air cooler and drift/duct to the FAR. In the spray chamber, cold water will be sprayed upwards in a flat V configuration into the horizontal airflow. Within the sprays, heat exchange will occur across the large surface area of the water drops. Re-spray systems will be used to achieve high thermal efficiency and this installation will be a two-stage spray chamber.

Refrigeration Machines

The refrigeration machine system will comprise main base-load machines that chill the water flow. Refrigeration machine modules will be standard, packaged, factory assembled R134a packages inclusive of centrifugal compressors, shell-and-tube heat exchangers, lubrication systems, connecting piping and cabling.

Condenser Cooling Towers

The condenser cooling towers (CCTs) will be in the form of mechanical draft, packed, counter-flow type towers in two large adjacent cells. The cooling towers will be constructed in reinforced concrete on top of concrete water basins. Fill-packs will be of the splash-grid type. The cooling tower fans will be 90 kW units. Make-up water supply, piping and control system will be included to provide for evaporation losses.

Water Pump System

The condenser water system will circulate water from the CCT sumps to the plants and back. Provision is made for three off (plus one standby) pump-motor sets of 110 kW rating.

The evaporator water system will circulate water from the BAC sump to the plants. One 160 kW pump motor set and one stand by set are proposed.

The re-spray pump will spray the water in the spray chamber. One 110 kW pump is proposed.

Simulation Results

Size of the complete cooling system was based on the mining depth of -795 mRL.

A hundred percent of the cool air will be delivered to the production zones via the FAR. The decline will be ventilated mainly by uncooled air supplied from the portal and with some cool air as required supplied from the FAR to the decline development headings.

No air cooling will be provided above the -354 mRL. Beyond that the first cooling 5.0 MWR machine will have to be operational at the end of Year 2. Initially it would operate at part load gradually increasing the load to full capacity at Year 7. From Year 8 to the end of the mine life, the second conventional machine will come on stream. There is a possibility that the second refrigeration units may not be installed if the waste heat from power generation could be used and an absorption chiller(s) installed instead.

Air Cooling Costs

A list of capital items and associated costs required for the refrigeration system at Rory's Knoll are presented in Table 15.27. Please refer to Table 16-36 for the incremental cooling capacity and associated operating cost.

Table 15.27: Rory's Knoll: Capital cost of surface air cooling station

Item	Amount ($)
Refrigeration System	6,019,200
Condenser cooling tower mechanical and internals	921,110
Air cooler mechanical and internals	1,476,152
Pumps, valves and piping	1,800,966
Electricals	2,253,038
Civils	4,090,655
Total Direct Costs	**16,561,119**
EPCM, project service and indirects (25% of directs)	4,140,279
Contingency (10% of directs)	1,656,113
Total Indirect Costs	**5,796,392**
Total Capital Cost	**22,357,511**

15.4.2.7 Manpower Requirements

The underground mine development plan calls for all jumbo development and production to be completed by the owner. The raise development and diamond drilling will be done by a contractor. The Aurora technical services manager will be the contract superintendent.

Underground manpower requirements for the mine were estimated from first principles and were checked and finalized by ensuring all equipment was adequately manned. The underground mine is scheduled to operate 360 days per year. An allowance of five days per year has been allocated for weather delays where no mining activities will occur.

The labor estimate assumes a seven-day a week, two 12-hour shifts per day operation. Technical services personnel will work a two weeks on, two weeks off roster and underground operational personnel will work a two weeks on one week off roster or equivalent roster. The estimate includes a heavy reliance on expatriate technical staff, supervision and underground operators. Guyana's lack of underground mining experience will require a comprehensive training effort, which is planned to commence during the pre-production period.

15.4.3 Satellite Deposits

The satellite deposits are Aleck Hill, North Aleck Hill, Walcott Hill, East Walcott Hill, Mad Kiss, South Mad Kiss, West Mad Kiss, and Rory's Knoll East.

The internal AGM mining scoping study (Scoping Study, AGM, 2015) addressed all of the satellite deposits and initially provided underground designs for Aleck Hill (with two mining zones accessed from separate declines), Walcott Hill (mined through Mad Kiss), Mad Kiss, and Mad Kiss South.

The scoping study technical economic model consolidated the two mining zones at Aleck Hill; Walcott Hill, Mad Kiss, and South Mad Kiss were consolidated into Central, At a gold price of $1,250/oz, North Aleck Hill, West Mad Kiss and East Walcott Hill were not viable as underground mining operations.

The scoping study initially included material from the inferred resource category in the LOM plan. The LOM plan was then re-assessed with only measured and indicated material. As a result, only two mining zones remained viable: Aleck Hill and Mad Kiss.

For this feasibility study update, only the Aleck Hill deposit and the Mad Kiss deposit remained viable.

15.4.3.1 Mining Context

The mining contexts of the Aleck Hill deposit and the Mad Kiss deposit are sufficiently different to warrant describing them separately.

Aleck Hill

Geology: Mafic metavolcanic rocks form the host rock of the Aleck Hill deposit. The overall trend of the auriferous zones at Aleck Hill is sub parallel to the trend of the sericite. The series of sub-parallel gold lenses strike southeast at 150° and dip sub-vertically, while the two main shear zones strike to the southeast (at approximately 155°) and dip steeply to the southwest.

Geometry: The gold zones are distributed in a series of sub-parallel lenses of variable width (2.5 m to about 20 m, with occasional 30 m widths) as illustrated in Figure 15.34 and Figure 15.35. The Aleck Hill deposit extends for about 1,000 m along strike with higher-grade material and old underground workings extending to about 400 m below surface.

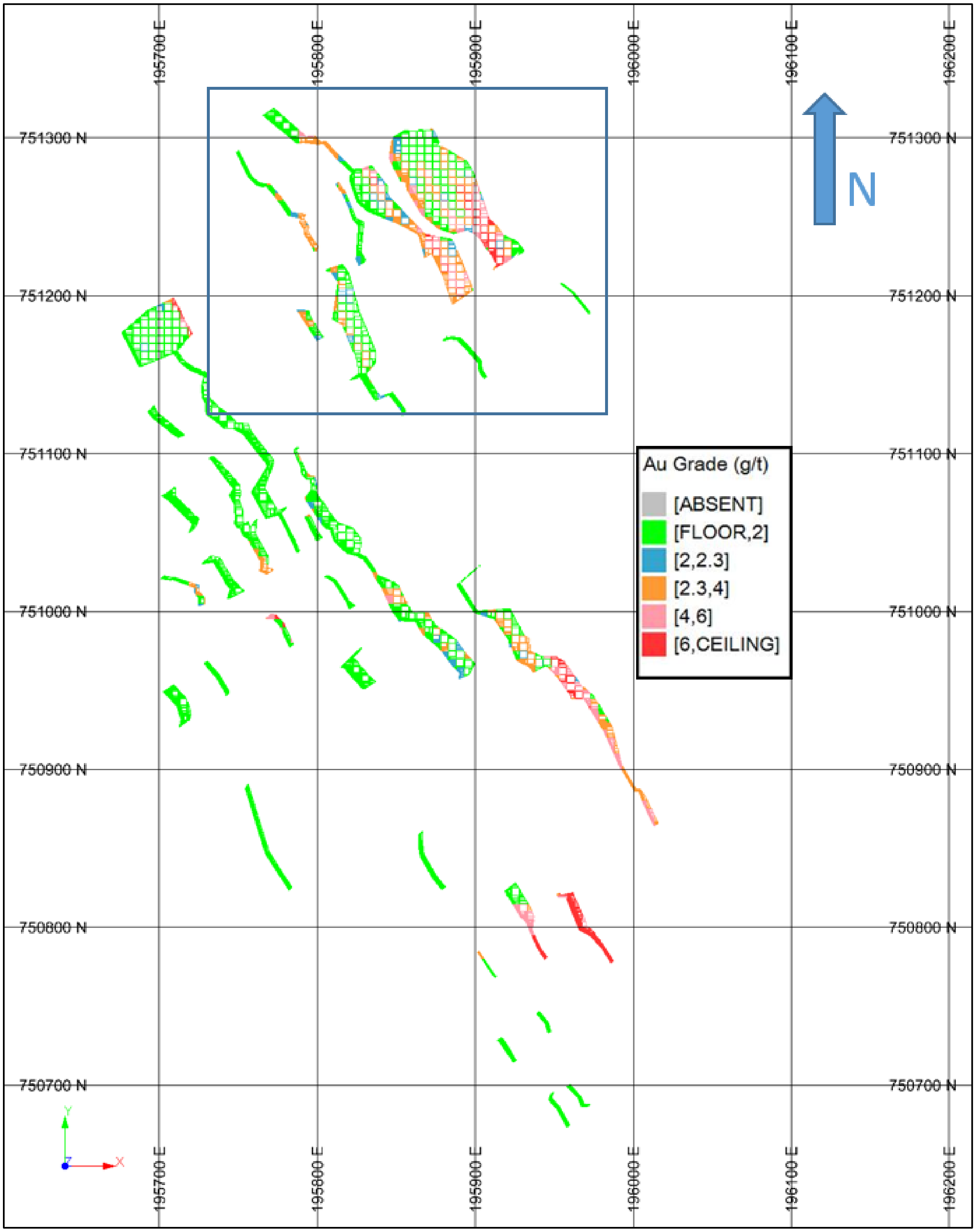
Grade Distribution: The overall continuity of Aleck Hill mineralization above a cut-off grade of 2.50 g/t Au (target mineralization) is classified as fair, however the continuity improves in the northern extents where underground mining has been planned. Some areas exhibit complete mixing of resource blocks above and below cut-off, while other areas have a clearer separation of mineralization above and below cut-off. The mineralized envelopes for Aleck Hill are presented in Figure 15.36 at a cut-off grade of 1.50 g/t Au.

Rock Mass: Ground conditions are expected to be good in the mafics, tolerating an open span of up to 20 m, and mainly fair in the sericite where an open span of 15 m should be observed.

Geological Structure: The shear zones are developed along lithological contacts within mafic volcanic rocks. This means that care will have to be taken while mining in the sericite area. Additional ground support will be required on the back of the stopes and higher dilution is expected.

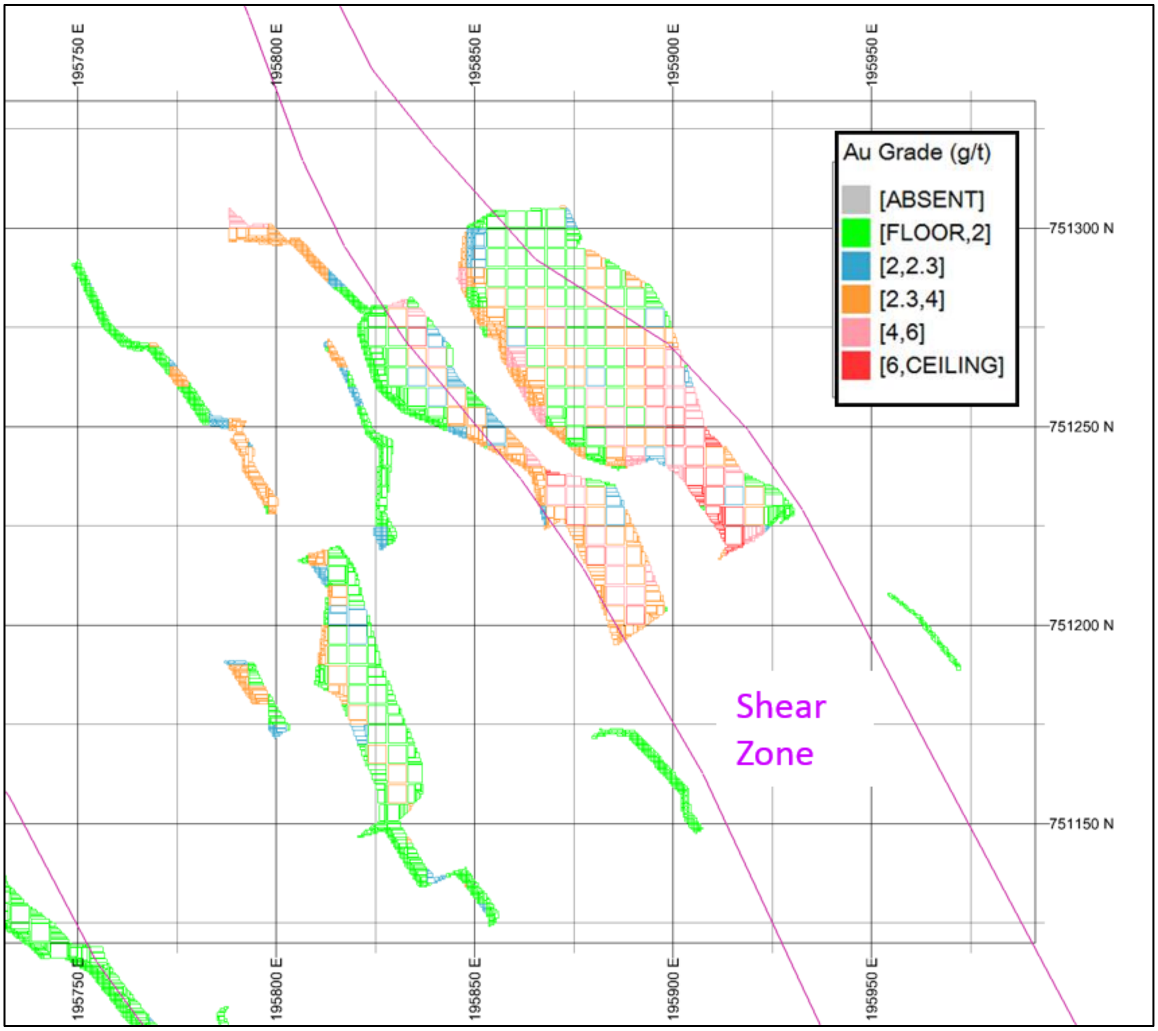
Disturbances It is a shallow-depth deposit (<500 m below surface) and therefore stress is not expected to be a significant factor.

External Constraints The high surface temperature and humidity of the tropical climate will not influence the shallow depth satellite deposits.



Source: SRK, 2017

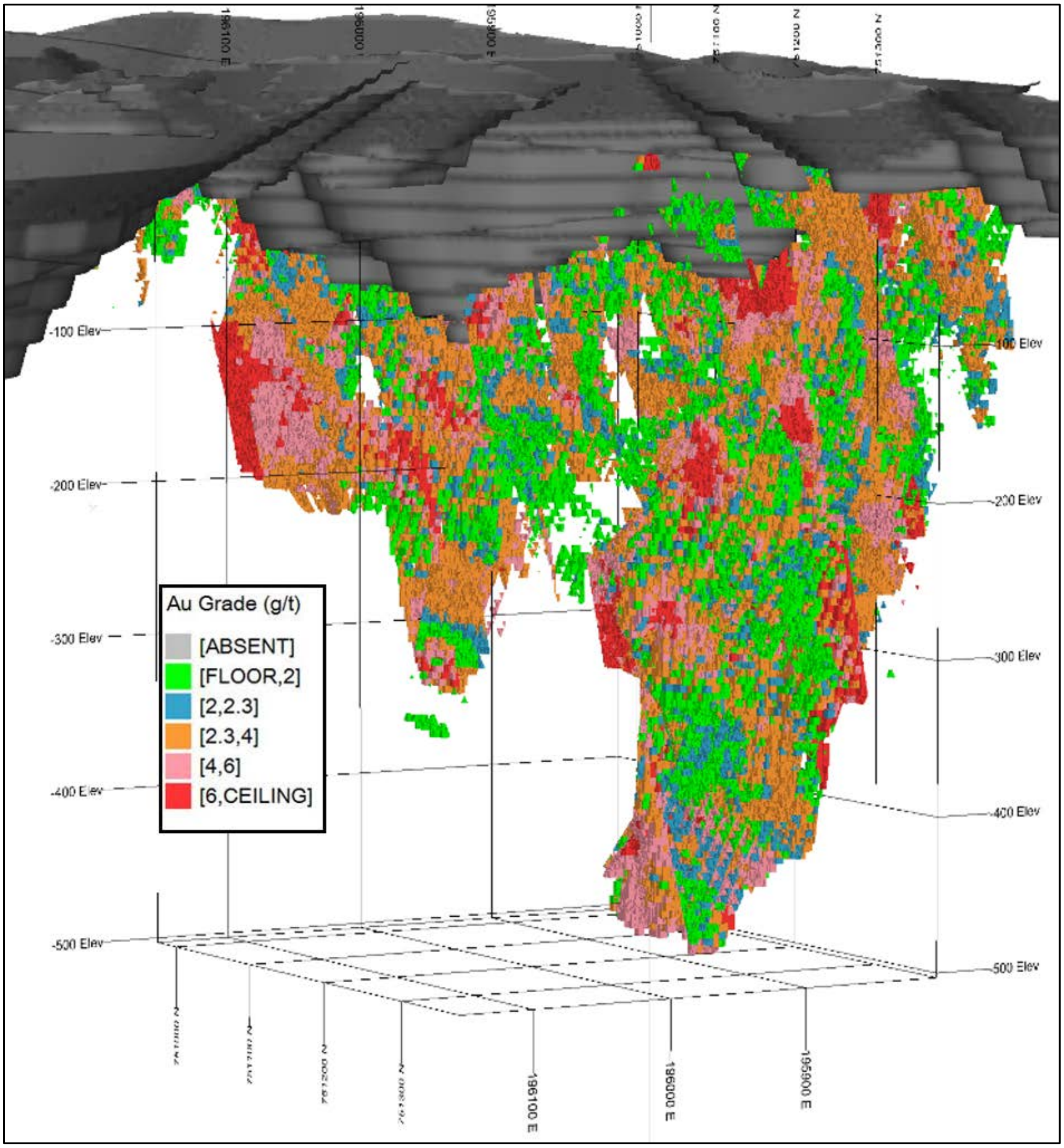
Figure 15.34: Aleck Hill: Mineralized envelopes at –205 mRL (plan view)



Source: SRK, 2017

Note: The inset highlights the area presented in Figure 15 34.

Figure 15.35: Aleck Hill: Mineralised envelopes and serecite shear zone in the stoping area at –205 mRL (plan view)



Source: SRK, 2017

Note: Blocks are shown with a filter of Au>1.50 g/t Au.

Figure 15.36: Aleck Hill: Grade distribution (Southwest view)

Mad Kiss

The Aurora gold deposit has been defined by surface exploration drilling; there is currently no underground development or exposures in the fresh rock mineralisation.

Geology: Mafic metavolcanic rocks form the Mad Kiss deposit. The overall trend of the auriferous zones at Mad Kiss is perpendicular to the trend of the sericite. The series of sub-parallel gold lenses strike southwest at 250° and dip at approximately 70° north while the intercepting shear zone strike northwest (between 290° and 305°) and dip steeply to the northeast (between 70° and 85°).

Geometry: The gold zones are distributed in a series of sub-parallel lenses of variable width (0.5 m to about 9 m in average) as illustrated in Figure 15.37. The veins are generally too narrow for transverse long-hole stoping methods.

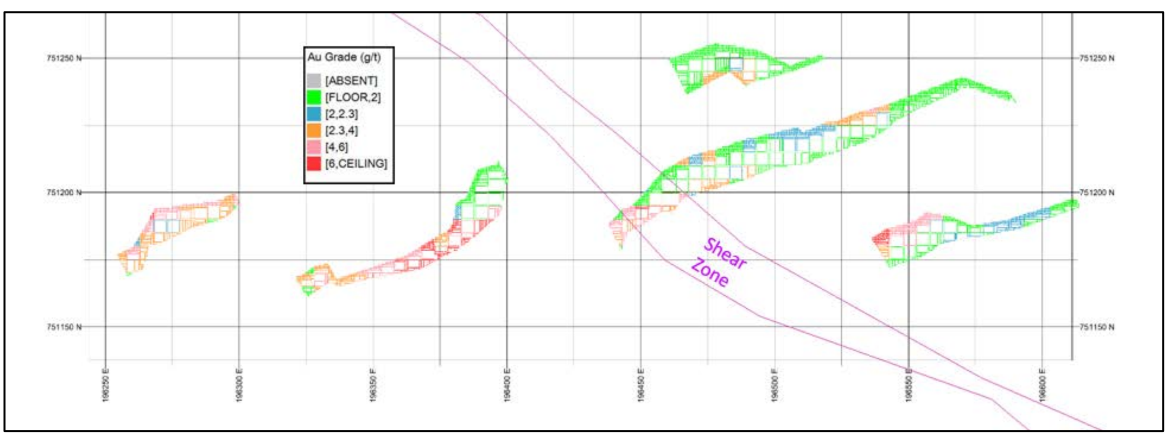
Grade Distribution: The continuity of the mineralization above a cut-off grade of 2.50 g/t Au (target mineralization) can be described as good. Some areas are very continuous, especially between -130 mRL and -300 mRL on the west side of the deposit. The mineralized envelopes for Mad Kiss are presented in Figure 15.38 at a cut-off grade of 2.40 g/ Au.

Rock Mass: Ground conditions are expected to be good in the mafics, tolerating an open span of up to 20 m and mainly fair in the sericite, where an open span of 15 m should be observed.

Geological Structure: The shear zones are developed perpendicular to the lithological contacts within mafic volcanic rocks, implying potentially unstable backs in the sericite in all situations.

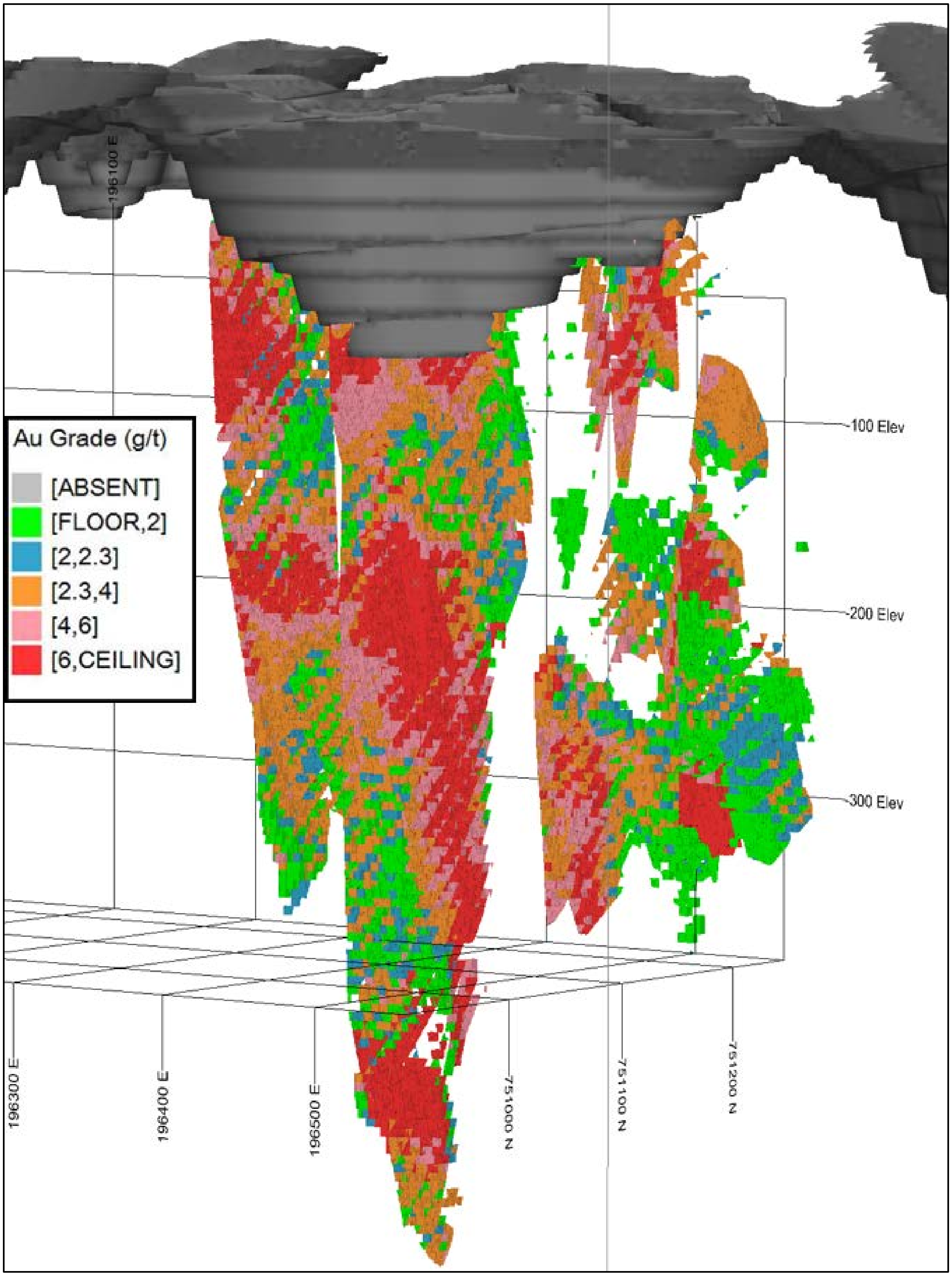
Disturbances: It is a shallow-depth deposit (<400 m below surface) and therefore stress is not expected to be a significant factor.

External Constraints: The high surface temperature and humidity of the tropical climate will not affect the shallow depth satellite deposits.



Source: SRK, 2017

Figure 15.37: Mad Kiss: Grade distribution & sericite shear zone at – 270 mRL (plan view)



Source: SRK, 2017
Note: Blocks are shown with a filter of Au>1.50 g/t.

Figure 15.38: Mad Kiss: Grade distribution (Northwest view)

15.4.3.2 Mine Design

The **Aleck Hill** satellite deposit is located approximately 1,000 m southwest of the Rory's Knoll zone. The mine design exploits Aleck Hill from a depth of about -160 mRL to about -500 mRL.

The **Mad Kiss** satellite deposit is located approximately 750 m south-southwest of the Rory's Knoll zone. The mine design exploits Mad Kiss from a depth of about -40 mRL to about -380 mRL.

The Mad Kiss stopes have a crown pillar of 25 m thickness, located below the Mad Kiss open pit. The level spacing at Mad Kiss is generally 20 m.

The Aleck Hill underground design does not include a crown pillar as the stoping area is located to the northeast of the open pit and not influenced by the open pit geometry.

The mine design for Aleck Hill and Mad Kiss is based on conventional trackless diesel equipment with loader mucking and truck haulage.

Access Ramp and Infrastructure

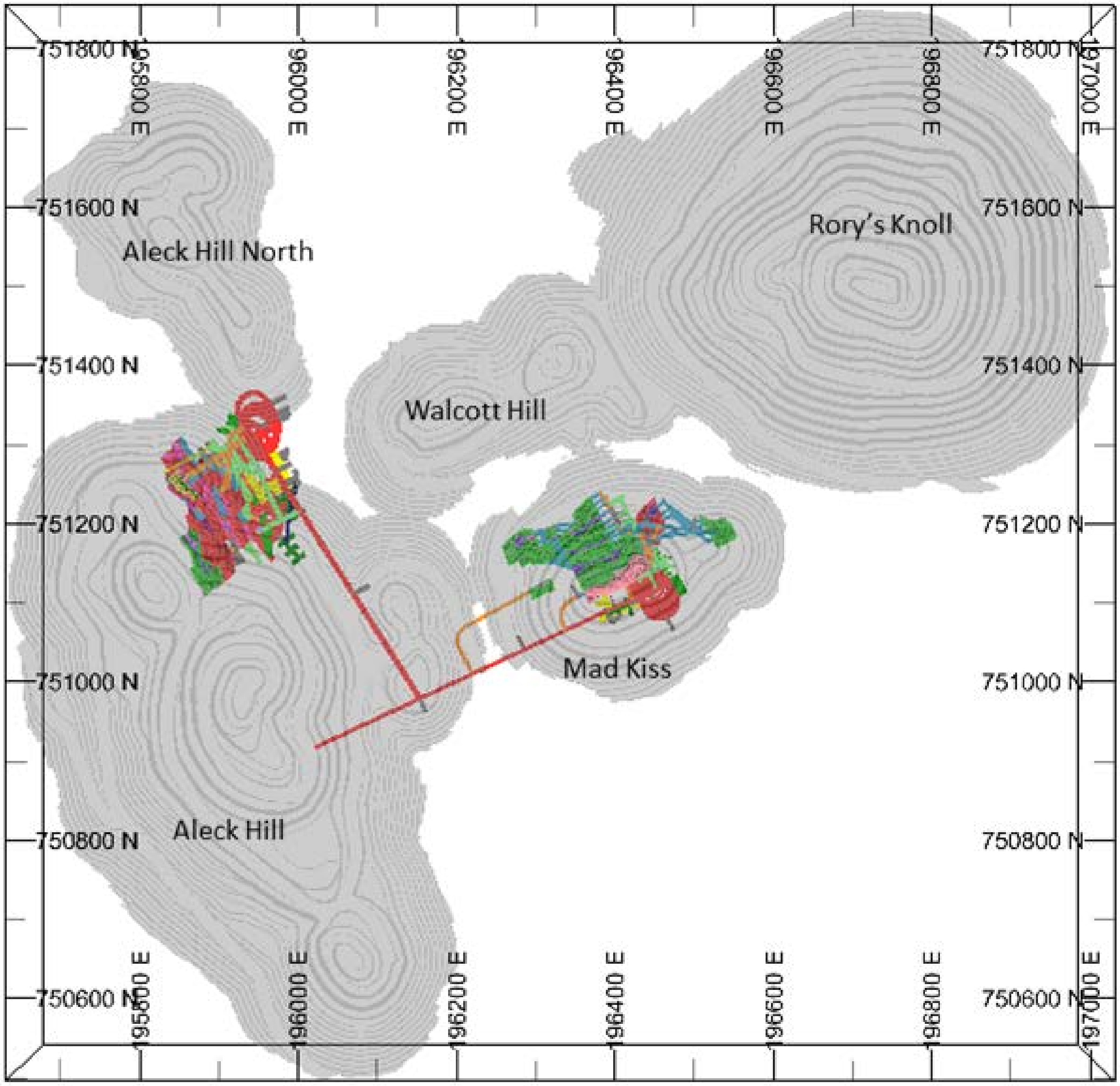
Aleck Hill and Mad Kiss share a common portal and ramp, located in the Aleck Hill pit (Figure 15.39), in the southeast side, at -50 m bench; the ramp splits to access the two deposits at -70 mRL.

The location of the portal has not yet been finalised and may be re-located based on updated geotechnical data.

The access ramp and all life of mine (LOM) capital infrastructure have been designed to generally avoid the known sericite shear zones. The access ramp for each deposit is located on the footwall side of the deposit to take advantage of the favourable ground conditions.

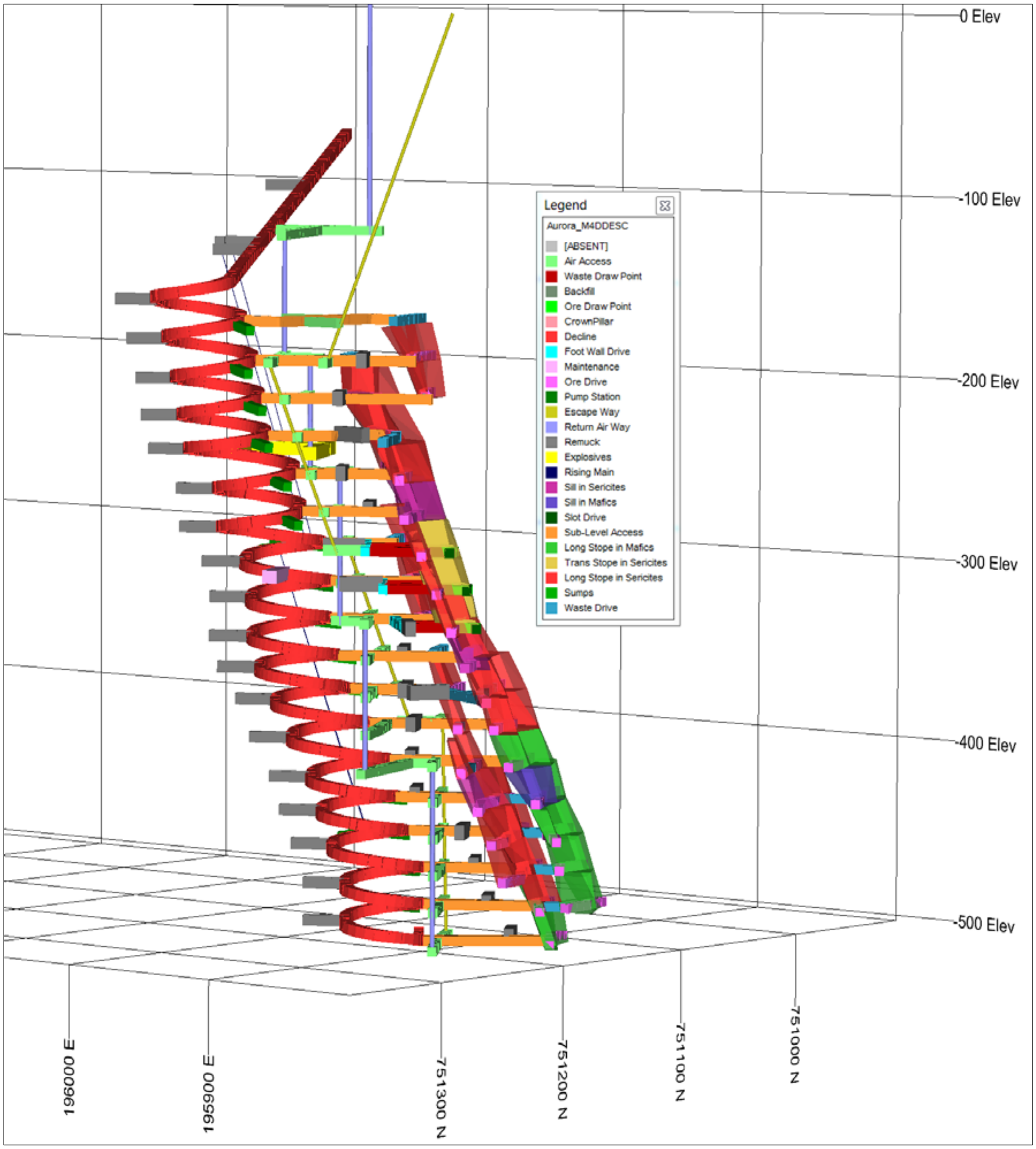
The mine design for Aleck Hill is presented in Figure 15.40 and Figure 15.41.

The mine design for Mad Kiss is presented in Figure 15.42 and Figure 15.43.



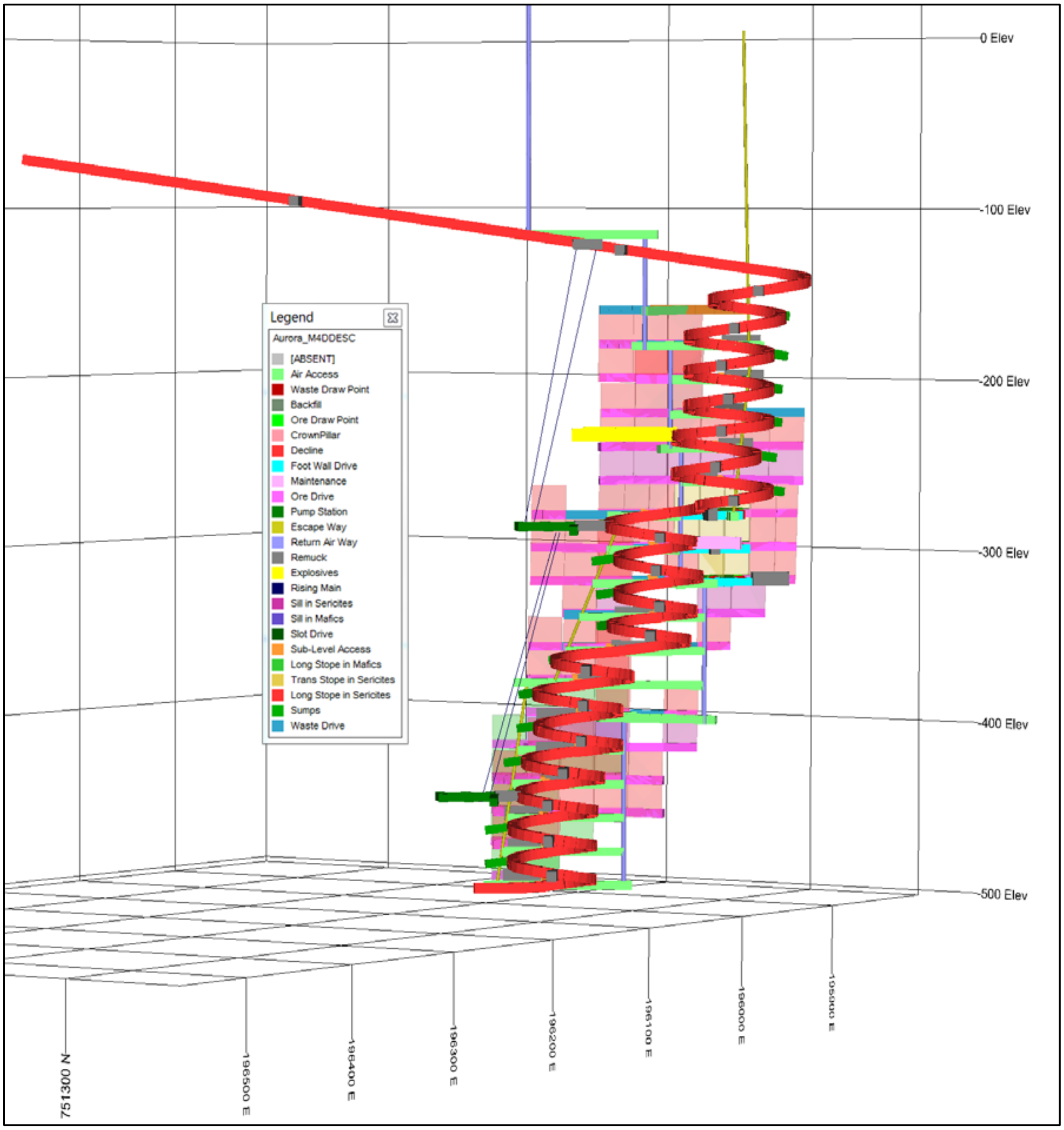
Source: SRK, 2017

Figure 15.39: Location of Aleck Hill & Mad Kiss underground workings (plan view)



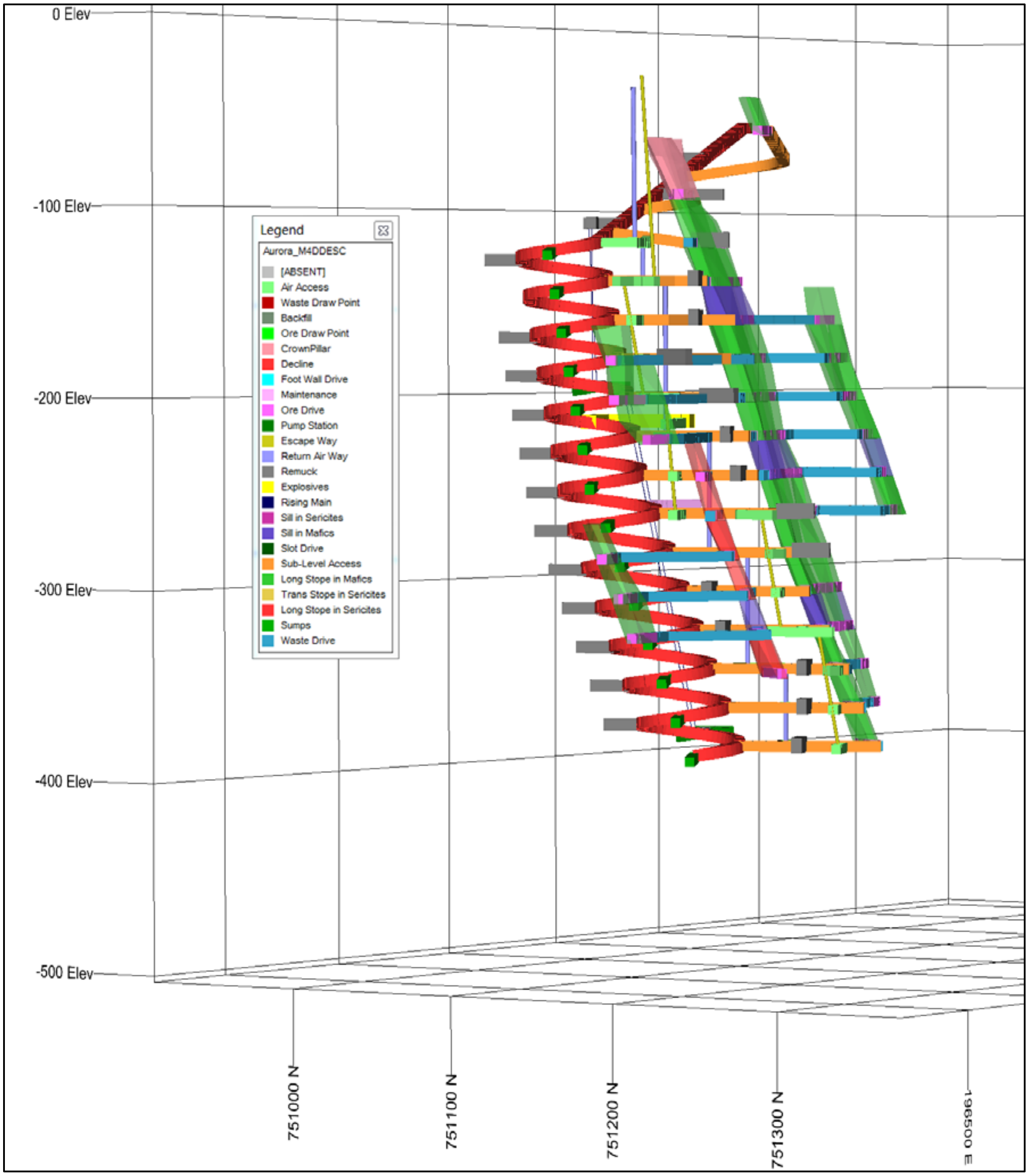
Source: SRK, 2017

Figure 15.40: Aleck Hill: Mine design (looking Southeast)



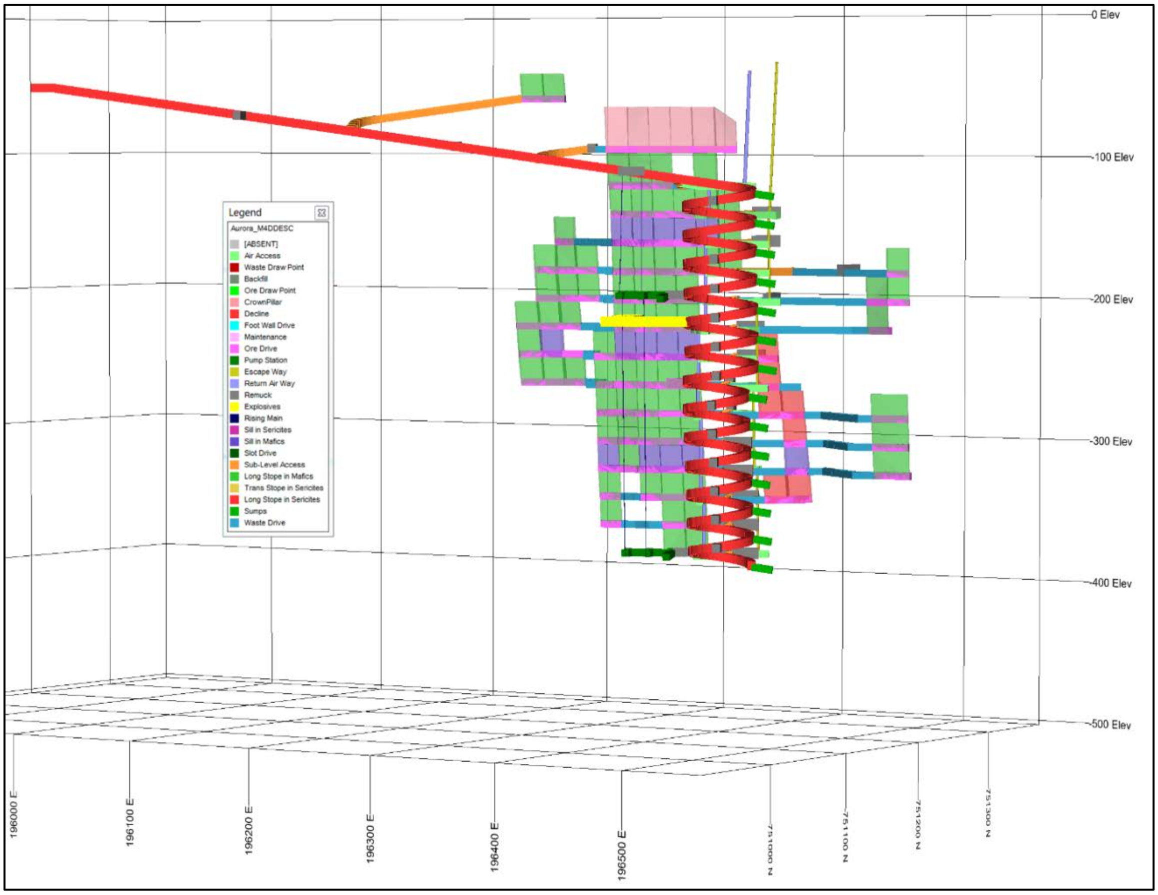
Source: SRK, 2017

Figure 15.41: Aleck Hill: Mine design (looking Northwest)



Source: SRK, 2017

Figure 15.42: Mad Kiss: Mine design (looking Southwest)



Source: SRK, 2017

Figure 15.43: Mad Kiss: Mine design (looking Northwest)

Level Development

The dimensions of the development is generally a compromise between the size of equipment, ground support requirements, and the desired rate of extraction. Larger equipment will provide higher productivity, but will also require larger excavation dimensions and additional ground support.

The decline provides mid-point access to each sublevel. The sublevel access connects to the transverse stopes via a footwall drive in waste and the transverse drawpoints. A slot drive, connected the ends of the transverse drawpoints, is developed on the hanging wall side of the stope.

Longitudinal stopes, typically located at the extreme ends of the sublevels, are accessed from the footwall drive and a cross-cut to the longitudinal drawpoint. The sublevel access may also connect directly to the longitudinal drawpoints where there are no transverse stopes.

The sublevels are ventilated by fresh air raise from the decline, exhausting to a return air raise connected to the sublevel access.

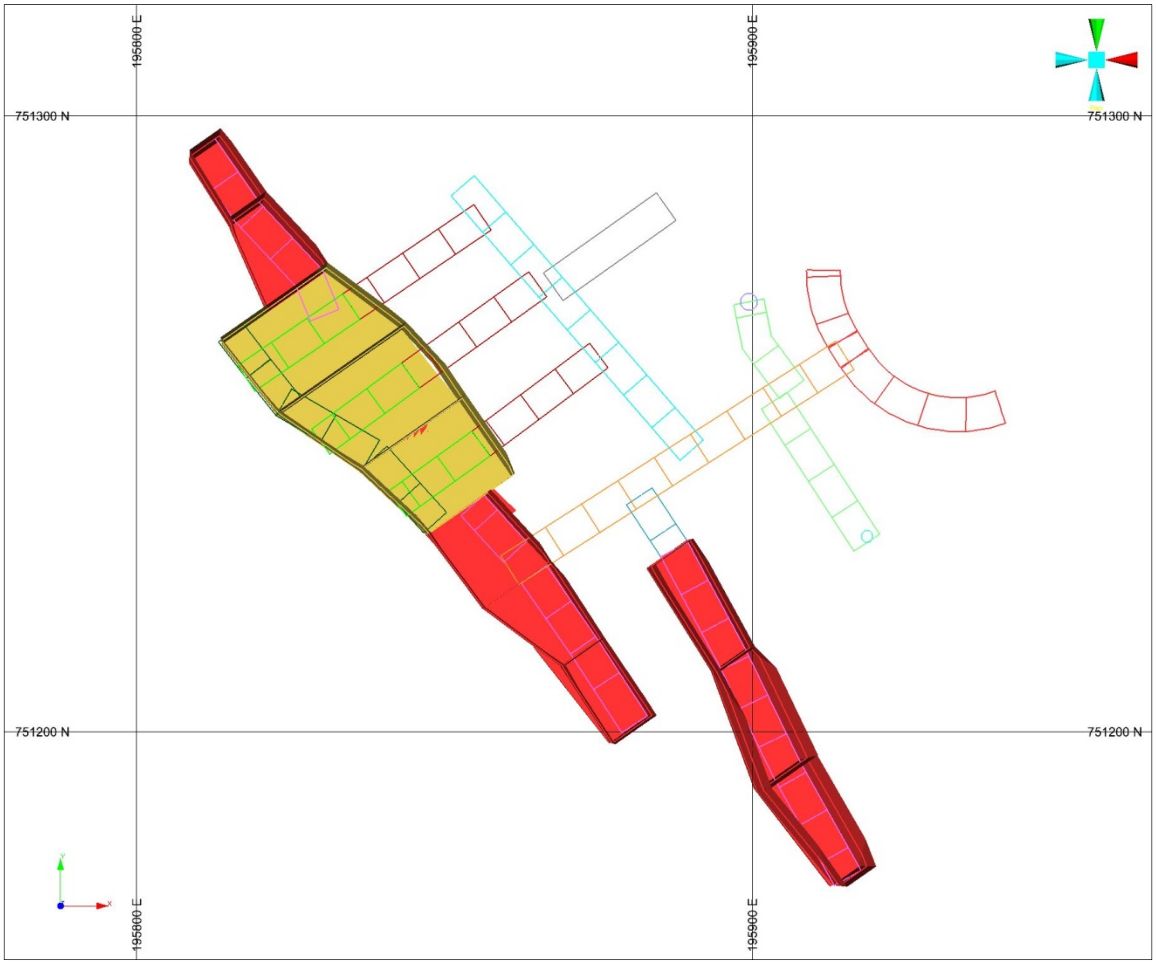
The design parameters for the Satellite deposits are presented in Table 15.28.

Table 15.28: Satellite deposits design parameters

Item	Width (m)	Height (m)
Decline	5.5	6.0
Sublevel Access	5.5	5.5
Ore and Waste Draw Point	5.0	5.0
Ventilation Access	5.0	5.0
Return Air Raise	3.0 m dia	
Rising Main	0.4 m dia	
Escape Way	1.8 m dia	
Ore and Waste Drive	5.0	5.0
Remuck Bay on Decline	5.5	6.0
Remuck Bay on Sublevel Access	5.5	8.0
Sump	5.0	5.0
Maintenance	8.0	7.0
Magazine and Explosives	8.0	7.0
Pump Station	5.0	5.0
Foot-wall Drive	5.0	5.0
Slot Drive	5.0	5.0

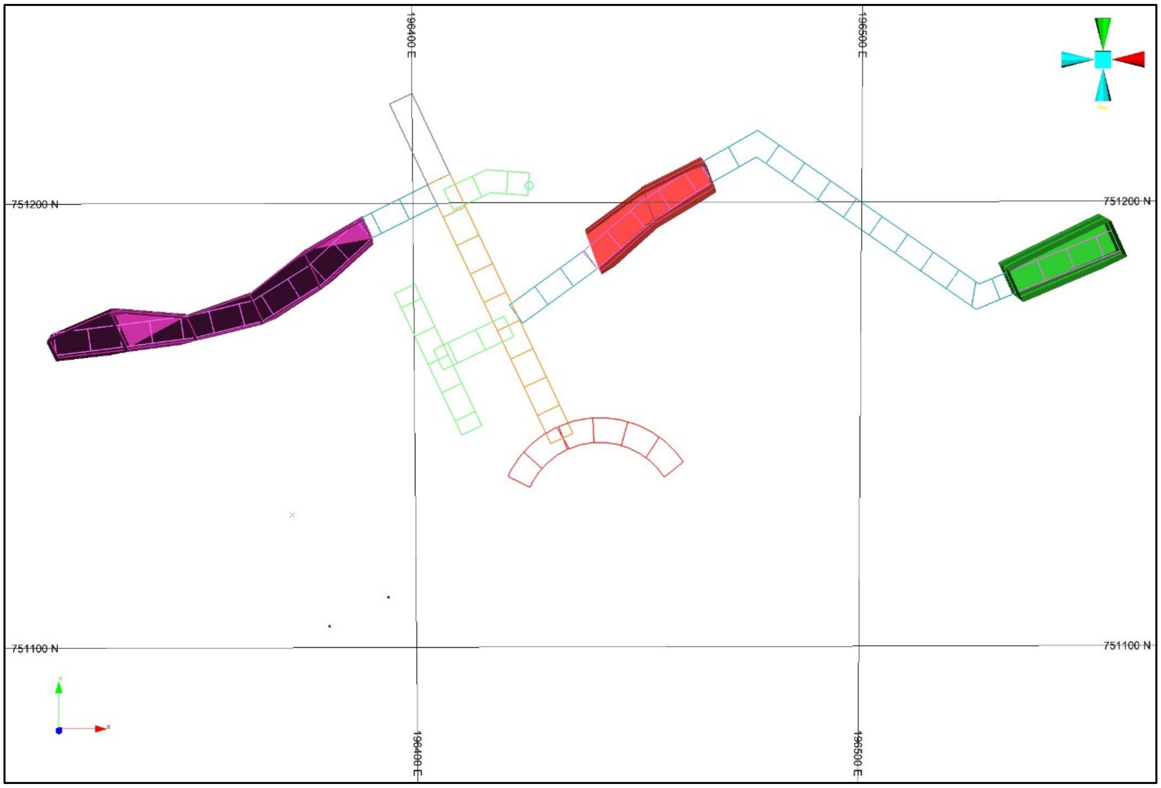
Figure 15.44 shows a typical sublevel layout for Aleck Hill. The design includes provision for transverse stopes (yellow outlines) and longitudinal stopes (red outlines).

Figure 15.45 shows a typical sublevel layout for Mad Kiss. The design includes provision for longitudinal stopes only (red and green outlines).



Source: SRK, 2017

Figure 15.44: Aleck Hill: Typical sublevel layout at –300 mRL (plan view)



Source: SRK, 2017

Figure 15.45: Mad Kiss: Typical sublevel layout –280 mRL (plan view)

Stope Design

Stope design for the satellite deposits is based on geotechnical recommendations outlined in Section 15.2.5.4. The key recommendations are summarised in Table 15.29.

Table 15.29: Satellite Deposits: Geotechnical guidelines (maximum span) for stope design

Location	Maximum Span in Mafics	Maximum Span in Sericite
Stoping Method	(m)	(m)
Aleck Hill		
LHOS Transverse	< 20	< 15
LHOS Longitudinal	< 20	< 20
Mad Kiss		
LHOS Transverse	n/a	n/a
LHOS Longitudinal	< 25	< 15

Note: There are no transverse stopes in the Mad Kiss design.

The Mineable Stopes Optimizer (MSO) tool was used to generate mining shapes that satisfied the cut-off grade, dilution and operational design criteria for each mining block. Where required, the shapes were manually adjusted to minimise the amount of sub-economic material within the stope shape.

The stope shapes at Aleck Hill and Mad Kiss were each designed with a 20 m level interval. The key design factor in MSO was the rock type (mafics or sericite) and the maximum allowable open span along strike for each rock type.

The decision to mine using either transverse or longitudinal techniques was made after the MSO stope shapes had been generated. In general, the maximum design width (across strike) for longitudinal stopes was 17.5 m. Stope shapes greater than 17.5 m wide were generally mined as transverse. The minimum mining width for longitudinal mining is 2 m.

There were a few exceptions where transition from longitudinal to transverse was inefficient, and in these situations, the mining method remained as longitudinal up to about 20 m in width.

This approach to selecting the stoping direction resulted in transverse and longitudinal stoping being utilised at Aleck Hill. The Mad Kiss deposit is generally too narrow for efficient application of transverse and is entirely mined with longitudinal stoping.

Ground support and ground control systems for the stope designs are detailed in Section 15.2.6.

Stope Cycle

Longitudinal longhole stoping methods have limited capacity to depart from a set mining sequence, since the mining operations start at one end of a mining block, and work sequentially to the other end of the mining block. Transverse longhole stoping is more efficient in terms of sequencing since simultaneous production can be achieved in parallel drawpoints.

The Aleck Hill and Mad Kiss deposits have been divided into panels, each consisting of four 20 m high sublevels. The fourth (uppermost) sublevel is the sill pillar, which will be partially mined giving a mining recovery of ~50%.

The overhand mining method will be used for longitudinal and transverse stopes. Mining will start at the lower sublevel and works upwards.

Longitudinal Stopes

Longitudinal stoping is utilised at Mad Kiss.

The stope cycle commences on completion of the production (operating) development, including ground support.

Longitudinal stopes typically consist of one or two ore drives, depending on the width of the stope. The ore drives (5.5 mW x 5.0 mH) are spaced laterally at 15 m intervals, leaving a 9.5 mW pillar between. Lateral spacing of the ore drives may be modified slightly to provide more efficient coverage of the stope.

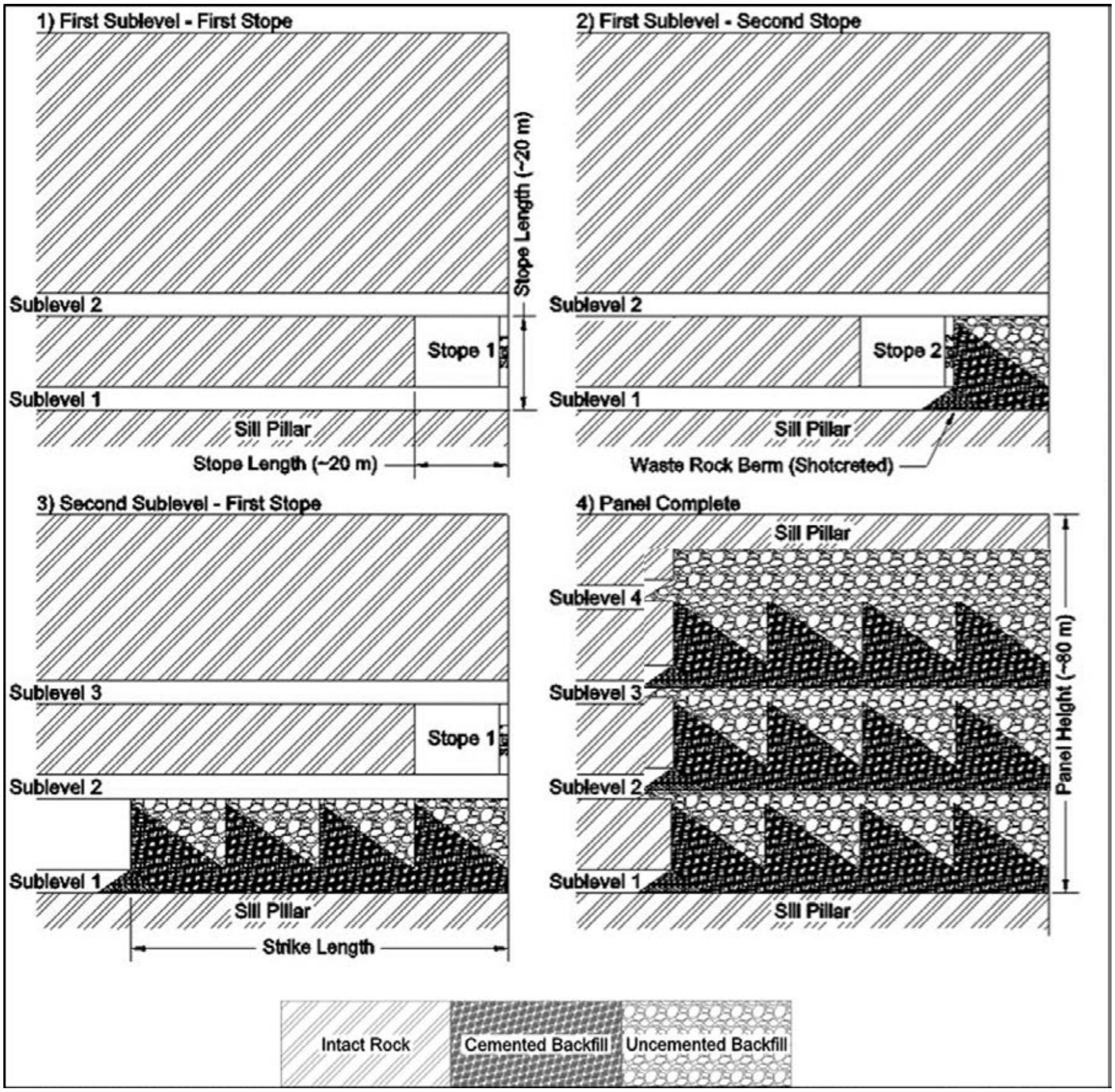
A relief slot is established at the far end of each stope. The slot consists of a raisebored relief hole (760 mm dia.) and a number of easer blast holes used to open the slot to full width. The raiseboring is conventional - the raiseboring rig is located on top of the hole with the reaming head being pulled upwards.

Due to the lack of access to the top of the fourth (uppermost) sublevel, the relief slot for the sublevel is drilled as a blind bored uphole.

All blast holes are drilled upwards, from the extraction level.

Following is the general sequence for mining the longitudinal stopes. Figure 15.46 shows the general longitudinal stoping sequence.

1. Drill the raisebore pilot hole for relief slot
2. Ream pilot hole to full size (V30 reamer)
3. Drill the blast holes (easers) for the slot
4. Drill the blast holes for the remainder of the stope
5. Open (blast) the slot to full width as required.
6. Fire (blast) the production blast hole rings into the slot
7. Open the stope to a maximum hanging wall exposure of 20 m.
8. Muck the broken ore, as required.
9. Backfill the stope.
    (a) Sublevels 1, 2 & 3: The stope void is partially backfilled (~65%) with cemented rock backfill (CRF). The remainder of the void is backfilled with unconsolidated (loose) rock fill (URF).
    (b) Sublevel 4 is not backfilled.
10. The sequence is repeated again for the next stope along strike.
11. On completion of stoping in the ore drive, the adjacent ore drive (if it exists) is mined.
12. Panels may be mined simultaneously to achieve the desired production rate.



Source: SRK, 2017

Figure 15.46: General layout of longitudinal stopes (long section)

Transverse Stopes

Transverse stopes are only mined in the wider zone at Aleck Hill; the longitudinal stoping method is otherwise used.

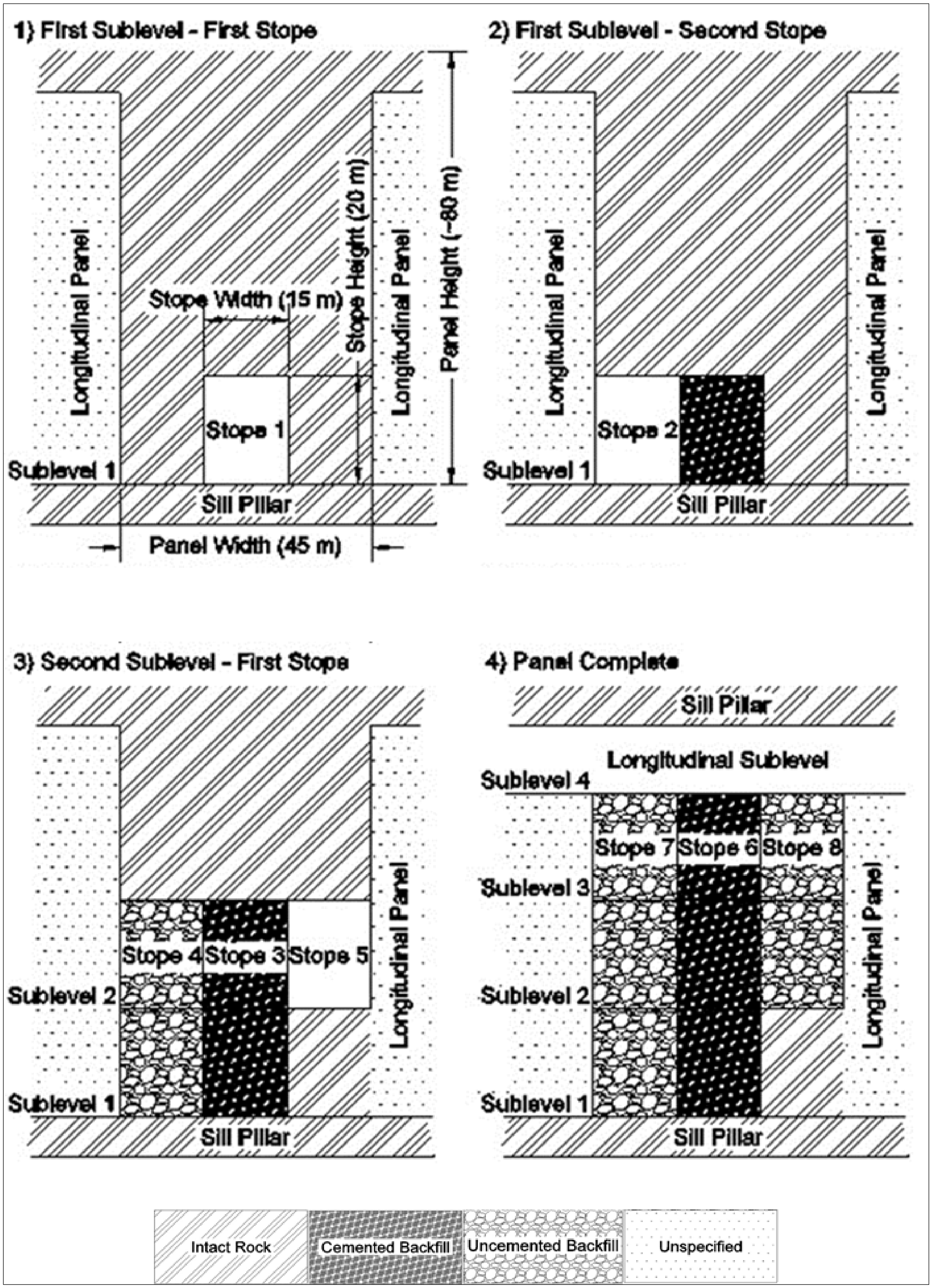
There are no transverse stopes at Mad Kiss.

The stope cycle commences immediately after the production (operating) development, including ground support, has been completed. For transverse stopes, this development is the drawpoints and slot drives.

The mining sequence for the transverse stopes is similar to the longitudinal stopes, i.e. panels of sublevels, starting at the lower sublevel, working upwards, backfilling each stope on completion. However, the transverse stoping area at Aleck Hill only has three sublevels, covering a vertical extent of ~60 m.

Figure 15.47 shows a typical cross-section through the transverse stopes at Aleck Hill. Transverse stoping commences on Sublevel 1 in Stope 1. This is a primary stope which is backfilled with CRF on completion of mining. When the backfill has cured (after ~28 days), the adjacent stope (Stope 2) is mined. Stope 2 is backfilled with URF on completion of mining. Production then moves to Stope 3 on Sublevel 2, followed by Stope 4 and Stope 5 (both secondary stopes, backfilled with URF). If necessary, these two secondary stopes can be mined simultaneously to increase the overall production rate. The sequence is repeated on Sublevel 3 (Stope 6, followed by Stope 7 and Stope 8).

The primary stopes are backfilled with CRF as the backfill will be exposed in the walls of the secondary stopes. The cement binds the rock component of the backfill and prevents it sloughing into the void remaining after the secondary stope has been mined. As there will be further stoping adjacent to the secondary stope, it can be backfilled with URF without fear of the backfill sloughing into an adjacent stope.



Source: SRK, 2017

Figure 15.47: Aleck Hill: Transverse stoping sequence (cross-section)

15.4.3.3 Development and Production Schedule

Production Rate

The production rate selected for the orebodies was determined by the orebody geometry and continuity, ground conditions, number of stopes available, anticipated productivity and scheduling to balance the resources to achieve a practical production output.

The study has used a nominal production rate of 1.1 Mtpa for Aleck Hill and 0.6 Mtpa for Mad Kiss and this compares with benchmarked underground operations utilizing sublevel caving and LHOS mining methods.

A rate of 880 tpd and of 700 tpd has been considered for longitudinal stopes for Aleck Hill and Mad Kiss, respectively. The transverse stopes at Aleck Hill have been scheduled at 950 tpd. A lower rate has been applied to Mad Kiss as the deposit has a smaller footprint and shorter vertical extent than Aleck Hill. The stoping rate includes blasting and mucking time. A specific rate of 150 m/d for production drilling has been applied. A backfill rate of 1000 tpd has been applied.

The production starts in early 2023 for Mad Kiss and in 2025 for Aleck Hill.

Development Schedule

The development rates used for the Satellite deposits are presented in Table 15.30.

Table 15.30: Development rates for Satellite Deposits

Description	Rate (m/d)
Decline	4.5
Sublevel Access	4.5
Ore and Waste Draw Point	4.2
Ventilation Access	4.5
Return Air Raise	2.6
Rising Main	12.0
Escape Way	2.6
Ore and Waste Drive	4.2
Remuck Bay	3.0
Sump	4.2
Maintenance	4.2
Magazine and Explosives	4.2
Pump Station	4.2
Foot-wall Drive	4.2
Slot Drive	4.2

The pre-production development period is 14 months at Aleck Hill and 10 months at Mad Kiss.

The Aleck Hill and Mad Kiss development schedules are presented in Table 15.31 and Table 15.32.

Production Schedule

The Aleck Hill and Mad Kiss production schedules are presented in Table 15.33 and Table 15.34.

LOM Schedule

The Life of Mine (LOM) quantity and grade estimate for the Satellite underground projects are presented in Table 15.35 and Table 15.36.

Table 15.31: Development schedule for Aleck Hill

Underground Schedule	Units	TOTAL	1	2	3	4	5
Capital Development - Lateral							
Decline Capital Development	m	3,146	1,194	581	999	372	0
Decline Capital Development	t	319,532	121,637	59,047	101,210	37,638	0
Other Capital Development	m	4,092	1,046	1,010	852	892	292
Other Capital Development	t	370,896	92,388	91,899	74,943	84,815	26,851
Capital Infrastructure Development	m	389	171	123	0	95	0
Capital Infrastructure Development	t	48,936	29,478	12,143	0	7,315	0
Total Capital Development	**m**	**7,627**	**2,411**	**1,714**	**1,851**	**1,359**	**292**
Total Capital Development	**t**	**739,364**	**243,503**	**163,089**	**176,153**	**129,768**	**26,851**
Capital Development - Vertical							
Vertical Capital - Raisebore	m	1,745	470	531	192	552	0
Vertical Capital - Raisebore	t	13,946	6,421	2,376	2,300	2,848	0
Total Vertical Capital Development	**m**	**1,745**	**470**	**531**	**192**	**552**	**0**
Total Vertical Capital Development	**t**	**13,946**	**6,421**	**2,376**	**2,300**	**2,848**	**0**
Operating Development							
Development Waste	m	832	0	321	185	286	40
Development Waste	t	64,121	0	24,630	14,200	22,216	3,075
Development Ore	m	2,097	0	691	404	625	376
Development Ore	t	145,712	0	47,838	28,223	43,477	26,175
Development Gold Grade	g/t Au	4.53	0.00	4.78	5.16	4.16	3.99
Development Gold Ounces	Oz Au	21,199	0	7,354	4,678	5,812	3,355
Total Operating Development	**m**	**2,928**	**0**	**1,012**	**589**	**911**	**416**
Total Operating Development	**t**	**209,834**	**0**	**72,468**	**42,423**	**65,692**	**29,251**

Table 15.32: Development schedule for Mad Kiss

Underground Schedule	Units	TOTAL	1	2	3	4
Capital Development - Lateral						
Decline Capital Development	m	2,360	1,174	878	308	0
Decline Capital Development	t	240,668	119,701	89,585	31,382	0
Other Capital Development	m	2,902	865	1,167	705	165
Other Capital Development	t	266,617	77,477	101,811	71,983	15,347
Capital Infrastructure Development	m	350	212	43	0	95
Capital Infrastructure Development	t	42,156	27,511	7,330	0	7,315
Total Capital Development	**m**	**5,612**	**2,252**	**2,088**	**1,013**	**260**
Total Capital Development	**t**	**549,441**	**224,689**	**198,726**	**103,365**	**22,662**
Capital Development - Vertical						
Vertical Capital - Raisebore	m	1,181	426	296	91	368
Vertical Capital - Raisebore	t	7,403	2,717	3,607	951	127
Total Vertical Capital Development	**m**	**1,181**	**426**	**296**	**91**	**368**
Total Vertical Capital Development	**t**	**7,403**	**2,717**	**3,607**	**951**	**127**
Operating Development						
Development Waste	m	1,314	54	346	677	237
Development Waste	t	101,343	4,073	26,653	52,427	18,189
Development Ore	m	1,404	184	241	711	268
Development Ore	t	95,094	12,581	16,297	47,921	18,295
Development Gold Grade	g/t Au	6.00	3.48	7.45	6.07	6.28
Development Au Ounces	Oz Au	18,358	1,406	3,904	9,354	3,694
Total Operating Development	**m**	**2,719**	**238**	**587**	**1,388**	**505**
Total Operating Development	**t**	**196,436**	**16,654**	**42,950**	**100,348**	**36,484**

Table 15.33: Aleck Hill production schedule

Description	Units	TOTAL	1	2	3	4	5	6	7
Longitudinal LHOS Production	t	851,029	0	117,093	152,718	165,055	179,116	190,942	46,105
Transverse LHOS Production	t	111,587	0	0	0	0	0	14,079	97,507
Total Ore Production	**t**	**962,616**	**0**	**117,093**	**152,718**	**165,055**	**179,116**	**205,022**	**143,612**
Production Gold Grade	**g/t Au**	**4.25**	**0.00**	**4.69**	**4.54**	**3.83**	**4.09**	**3.77**	**4.93**
Production Gold Ounces	**Oz Au**	**131,441**	0	**17,663**	**22,299**	**20,335**	**23,525**	**24,841**	**22,776**

Table 15.34: Mad Kiss production schedule

Description	Units	TOTAL	1	2	3	4	5
Longitudinal LHOS Production	t	497,830	0	61,174	102,268	168,035	166,353
Production Gold Grade	**g/t Au**	**5.34**	**0.00**	**3.42**	**6.25**	**5.13**	**5.71**
Production Au Ounces	**Oz Au**	**85,346**	**0**	**6,719**	**20,553**	**27,719**	**30,354**

Table 15.35: Aleck Hill LOM plan

Description	Units	TOTAL	1	2	3	4	5	6	7
Mined Ore	t	1,108,328	0	164,931	180,941	208,532	205,291	205,022	143,612
Total Mined Gold Grade	**g/t Au**	**4.28**	**0.00**	**4.72**	**4.64**	**3.90**	**4.07**	**3.77**	**4.93**
Total Mined Au Ounces	**Oz Au**	**152,640**	**0**	**25,018**	**26,977**	**26,147**	**26,880**	**24,841**	**22,776**

Table 15.36: Mad Kiss LOM plan

Description	Units	TOTAL	1	2	3	4	5
Mined Ore	t	592,924	12,581	77,471	150,188	186,330	166,353
Total Mined Gold Grade	**g/t Au**	**5.45**	**3.48**	**4.26**	**6.19**	**5.24**	**5.71**
Total Mined Au Ounces	**Oz Au**	**103,704**	**1,406**	**10,623**	**29,907**	**31,414**	**30,354**

15.4.3.4 Mobile Equipment Requirements

The mobile equipment fleet required for a nominal rate of 0.35 Mtpa at the satellite deposits is presented in Table 15.37.

Mobile equipment requirements was developed based on the scheduled quantities of work and estimated from first principle cycle times and productivities, benchmarking, and practical experience. Equipment purchase, rebuild and replacement schedule was developed based on anticipated equipment life, equipment operating hours and recommendations from manufacturers for equipment rebuilds and replacement.

Table 15.37: Underground mobile equipment list for Satellite Deposits

Equipment Type	Model	LOM Quantity	LOM Rebuilds	Max Fleet Size
Jumbo	Sandvik DD421	2	1	2
Bolter	Sandvik DS311	3	2	3
UG loader	Sandvik LH517	3	0	3
UG truck	Sandvik TH551	4	2	3
LH drill	Sandvik DL431	1	0	1
ITH drill	ITH Drilling	1	0	1
Development charge	Normet Charmec MC605 DA	1	0	1
Production charge	Normet Charmec LC605 VE	1	0	1
Services	Normet	10	5	8
Grader	Veekmas FG 15 C	1	1	1
Light vehicle	Toyota Hurth	6	6	4

15.4.3.5 Ventilation

The ventilation strategy for the satellite deposits follows a similar strategy to Rory's Knoll.

15.4.3.6 Backfill

Rock Fill

Rock fill utilises waste rock, quarried rock or aggregate as a bulking material. Depending on the engineering purpose of the fill, a hydraulic component (cement slurry or cemented tailings) can be combined with the bulking material to reduce void space in the fill or to produce a cemented rock mass. For example, Cemented Rock Fill (CRF) is a blend of cement and waste rock which is widely used in LHOS and in cut and fill mining methods.

Cemented Rock Fill

CRF refers to loose aggregate fill to which ordinary Portland cement is added, often finer materials such as sand or mill tailings are also added to the mix along with other binders, such as pulverised fly ash. The process of mixing unmodified rock fill with cement slurry is very basic and often does not provide a product with a high degree of confidence in the quality of the mix; it is difficult to provide a robust engineering design based on cemented rock fill.

CRF is generally mixed in batches rather than in a continuous process. In its most basic form, batching CRF may consist of pouring cement slurry over a load of loose rock in a loader bucker or haul truck bowl. Larger batches can be made using a loader to mix the cement slurry and loose rock in a sump.

Placement of CRF is usually achieved by hauling each load from the batching site to the dumping site. Due to the variable size of the rock, fill passes are generally not used. The CRF would be dropped into the stopes to be filled. Placement of cemented rock fill needs to be carefully monitored and quality control procedures need to be put in place to ensure that weak zones in the placed fill are not allowed to develop. However, in the proposed context no personnel will be directly exposed to vertical fill walls or backs and only the loader on remotes will ever be exposed to any fill failure hazard.

CRF has been adopted as the backfill method for:

- The Aleck Hill LHOS primary transverse stopes and
- In limited quantities (65%) for the LHOS longitudinal stopes at both Aleck Hill and Mad Kiss, placed at the end of the neighbour stope to stop the waste backfill from rolling back into the active stope being mined.

This type of backfill is widely used in LHOS mines around the world.

SRK has assumed a suitable waste material (strong and not prone to degradation and clay minerals) will be accessed on site (rock from development and the open pit), screened of oversize (if required), and then batch-mixed on surface and transported underground by an ejector truck. Stope delivery would be via loaders or direct ejected from the truck. To achieve the desired strength properties, a cement content could be approximately 5% cement.

Unmodified Rockfill

Unmodified rock fill (URF) is a material (usually waste rock from development or the open pit) that has not undergone modification of particle size grading and contains no binder or cement.

In its natural state, unmodified rock fill is a loose, granular medium and cannot form a vertical face if exposed to an open adjacent void. A dry, single size rock fill may form an angle of repose of about 38°.

URF is not suitable in LHOS primary transverse stopes, but is suitable in the secondary and longitudinal stopes.

URF has been adopted as the backfill method for the Aleck Hill LHOS secondary transverse stopes and at 35% for the LHOS longitudinal stopes.

This type of backfill is widely used in LHOS mines.

15.4.3.7 Manpower Requirements

The underground mine development plan calls for all jumbo development and production to be completed by the owner, raise development will be done by a contractor. The Aurora technical services manager will be the contract superintendent.

15.5 Underground Infrastructure

15.5.1 Mine Dewatering

Rory's Knoll dewatering facility is designed with cascading main dewatering stations located along the main declines and vertically separated by approximately 200 meters (refer to Figure 15.48). Main pump stations will be connected via boreholes except near surface, where the main dewatering line will be installed in the decline and exit the mine via the portal. From the portals, the water will be discharged into the open pit dewatering facilities.

Within the mining levels, water will be collected in sumps and pumped or gravity fed to a main dewatering station.

A dirty water pumping system (rather than a clean water system) was selected to mitigate the risks to the operation as dirty water pumps are designed to deal with the solids introduced into the pumping system by large inflows of water expected during the rainy seasons. Pumps capable of handling dirty water eliminate the need for clarifying very high flow rates of mine water which would require very large and expensive settling sumps to achieve the minimum retention time to produce clear water.

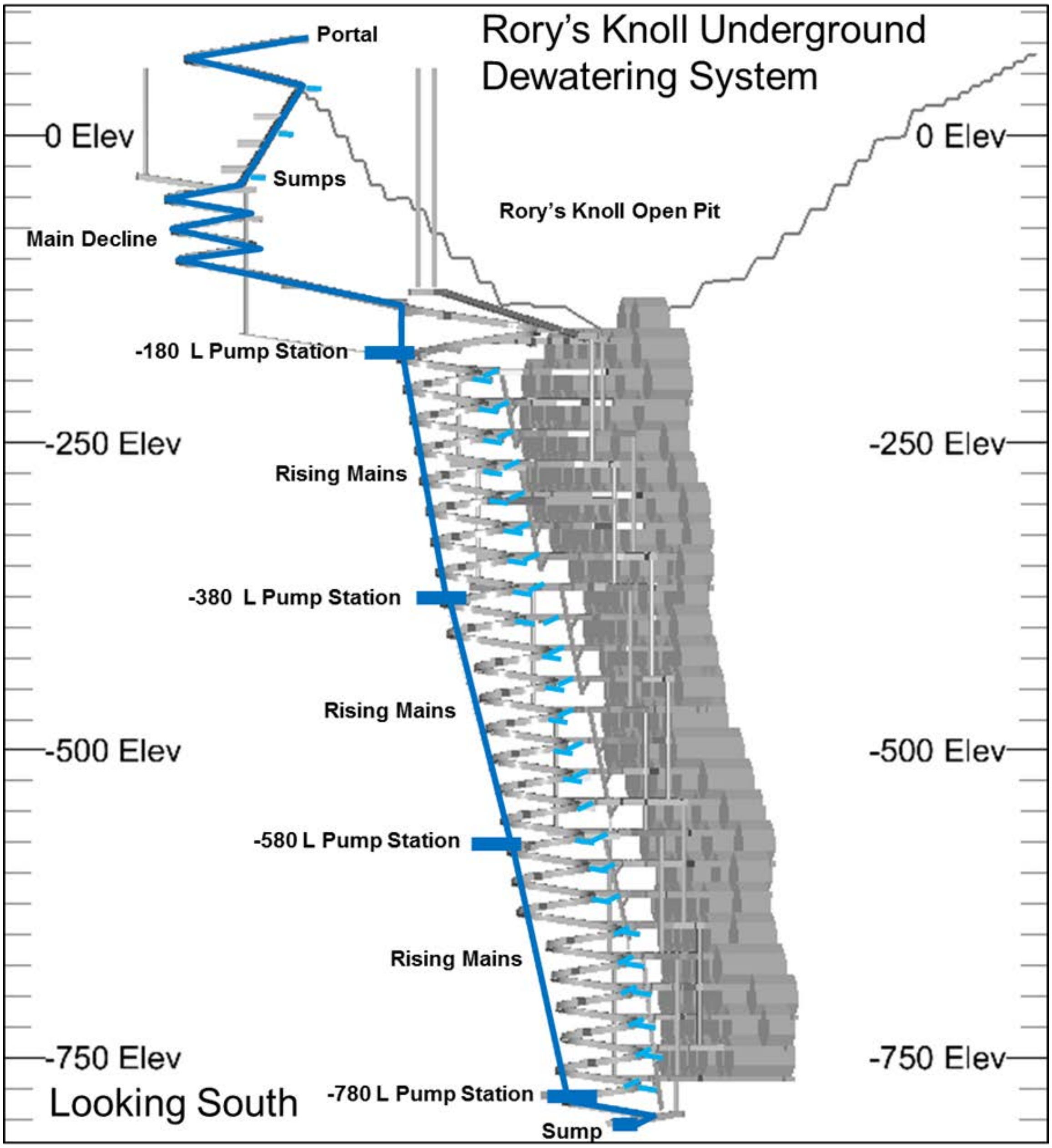
The design capacity for the dewatering facility is provided in Table 15.38. These capacities are based on modelled groundwater inflows with active dewatering from surface (Itasca, 2015) plus anticipated process water for drilling, washing equipment and dust suppression activities.

Table 15.38: Dewatering design capacities

Deposit	Ground Water Inflow (m^3/day)	Process Water (m^3/day)	Design Capacity (m^3/day)
Rory's Knoll	2,300	1,200	3,500

The underground mining breaks through to surface, therefore, the potential inflows during the wet season are more significant. It is estimated that a 25 year storm event would produce ~285 mm of rain over a 24 hour period (Tetra Tech, 2013), or 90,000 cubic meters of water given the footprint of the open pit. This type of event would flood the lower levels of the mine, requiring approximately two weeks to pump out. The dewatering system is designed to have short term emergency capability of double the design capacity in case of this type of event.

Dewatering for the satellite deposits will follow a similar strategy to Rory's Knoll.

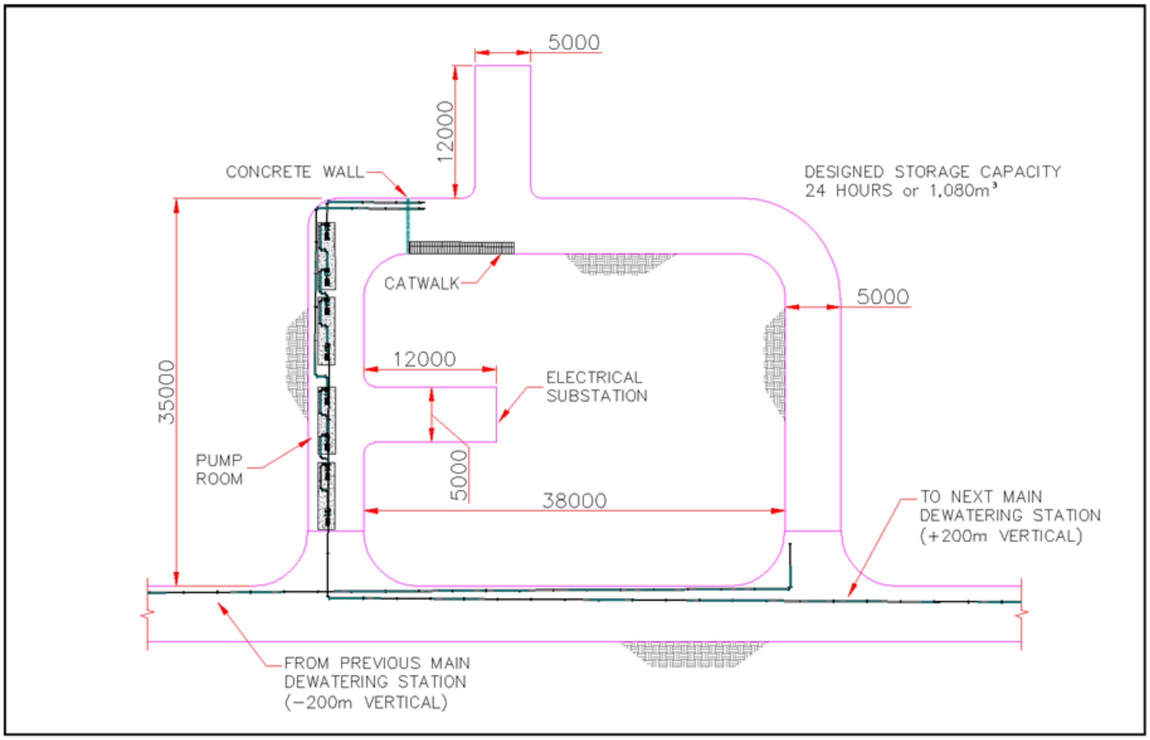


Source: SRK, 2017

Figure 15.48: Rory's Knoll: Dewatering schematic

15.5.1.1 Main Pump Station

The main pump stations (Figure 15.49) include a sump, pumps and motors, and electrical power and control units. The design is modular with all components mounted on transportable steel skids. The system is designed to pump for 16 hours per day to allow for extra flow to catch up following a minor maintenance shutdown. There is one line of active pumps and a second line of back up pumps within each arrangement. This allows for pumping to continue during extended maintenance. The pumps used will be two or three staged centrifugal slurry pumps and gravity fed from an adjacent sump. The sump design includes a retention of 24 hours. The location and design parameters of the main pump stations is summarized in Table 15.39.



Source: SRK, 2017

Figure 15.49: Typical design for main pump station

Table 15.39: Main pump station design parameters

Location (mRL)	Pipe Diameter (mm)	Design Capacity (m^3/day)	Installed Power (kW)
-170	150	3,500	450
-370	150	3,500	300
-570	150	3,500	300
-770	150	3,500	300

15.5.1.2 Level Sumps

Every mining level will have a level sump in close proximity to the mining areas to collect both process water and ground water. These level sumps will consist of a single 12 m long drift driven at a gradient of -15%. The sumps will be connected from level to level by a series of drain holes to allow the water to gravity-flow to the main dewatering stations or pumped from the level sumps to the closest main dewatering stations.

Submersible pumps will be used at the decline face to pump the water to the closest level sump or directly to a main dewater station when in proximity.

15.5.2 Materials Handling

15.5.2.1 Rory's Knoll

Ore Handling

Production ore will be mucked by loaders (LHDs) into ore passes located on each level. The orepass system feeds truck chute loadouts, located on every fourth level (i.e at 100 m vertical intervals).

For open benching stopes, tele-remote loading will be primarily used, as the majority of blasted ore will be situated within the open stops, beyond the drawpoint brow. Conventional (manually operated) loading may be used for brief periods, immediately after a blast, when the broken ore is close to the stope brow. Refer to Section 15.4.3.4 for a description of tele-remote loaders.

For sublevel retreat (SLR) stopes conventional loading will be primarily used as under normal operating conditions the loader will not be required to enter the stope.

Trucks will be loaded from the orepass chutes and will haul the ore to the mill ROM pad.

Underground crushing is not required as the fragmentation from development blasting and the engineered fragmentation of open benching and sublevel retreat blasting will be sufficient to prevent excessive quantities of oversize rocks. Any oversize rocks will be popped either in the drawpoint or at the orepass.

Waste Handling

All waste rock will be hauled to a surface waste dump. Production (stope) ore that does not meet the cut-off grade will be loaded directly into trucks and hauled to a surface waste dump.

15.5.2.2 Satellite Deposits

Ore Handling

Production ore will be mucked by loaders into remuck bays located on each level. Trucks will be loaded at the remuck bays and will haul the ore to the mill ROM pad.

Tele-remote loading will be primarily used, as the majority of blasted ore will be situated within the open stopes, beyond the drawpoint brow. Conventional (manually operated) loading may be used for brief periods, immediately after a blast, when the broken ore is close to the stope brow. Refer to Section 15.4.3.4 for a description of tele-remote loaders.

Underground crushing is not required as the fragmentation from development blasting and the engineered fragmentation of open benching and sublevel retreat blasting will be sufficient to prevent excessive quantities of oversize rocks. Any oversize rocks will be popped either in the drawpoint or at the orepass.

Waste Handling

Where feasible, waste material from underground will be used for the back fill; the waste rock will otherwise be hauled to the in-pit waste dump. When the back fill requirements exceed the waste available from development, waste from the open pit will be back-hauled in the underground trucks to the stopes.

15.5.3 Compressed Air

Compressed air will be supplied by local area compressors. The underground maintenance and service bay area will have a dedicated compressor permanently installed, with air lines from the air receiver routed to convenient locations in the area.

In addition to the permanent compressors, two mobile compressors with a 3,000 cfm capacity will be available.

All mobile drilling equipment, including jumbos, longhole drills, bolters and cable bolters will be equipped with on-board compressors. ITH drilling equipment will have portable adjacent compressors to meet their elevated pressure & flow requirements.

15.5.4 Service Water Supply

Service water for the underground operations is used mainly for drilling, dust control, workshops, washing and fire suppression of class “A” fires. The water will be supplied from a service water tank located in the vicinity of the decline portal. The service water will gravity-flow through a decline pipeline with distribution to the sublevels as required.

Pressure reduction valves will be installed as required.

15.5.5 Potable Water

Potable Water will not be supplied to the underground mine by a separate piping system. Instead, bottled potable water will be delivered to each refuge station and lunchroom. Mine operators will also carry their own supply of potable water.

15.5.6 Fire Prevention

All diesel equipment (light vehicles and heavy duty mobile equipment) will be equipped with automatic fire suppression systems and hand held fire extinguishers. Hand held fire extinguishers will be located also throughout the mine at refueling bays, workshops, explosive and detonator magazines, refuge chambers and lunch rooms. Refueling bays, workshops, explosive and detonator magazines will be equipped with automatic deluge systems.

A mine-wide stench gas warning system will be installed at the two fresh air intakes at Rory's Knoll to alert underground workers in the event of an emergency.

15.5.7 Mine Rescue

Mines rescue teams and a training facility will be established at the mine site. The teams will be comprised of a cross-section of Aurora personnel – i.e. mine operators, mechanical and electrical trades, and technical staff. The team rosters will account for the work cycle to ensure that two full teams are available on site at any time.

Surface and underground training facilities will be necessary for ongoing training and refresher training programs. Mines rescue equipment including a fire tender, ambulance, and all supporting testing and maintenance equipment for mine rescue purposes will be available and specific underground mine rescue equipment would include self-contained breathing apparatuses (e.g. Drager BG4).

15.5.8 Emergency Egress

The fresh air raises will also serve as the emergency egress from the underground mine.

In accordance with best practices, the egress ladder way will be a steel, self-supported enclosed ladder way system with rest platforms. Access to the ladder way will be provided on every production sublevel and access will be established prior to production mining commencing on the sublevel.

15.5.9 Dust Control

The underground mine will have a dedicated water truck for dust suppression in the decline and active sublevels.

Spray nozzles for dust control will be installed at all loading points.

15.5.10 Fueling and Lubrication

Diesel fuel, oil and lubricants will be delivered by a fuel and service truck to mobile equipment underground. Trucks, light vehicles, and support equipment will refuel on surface.

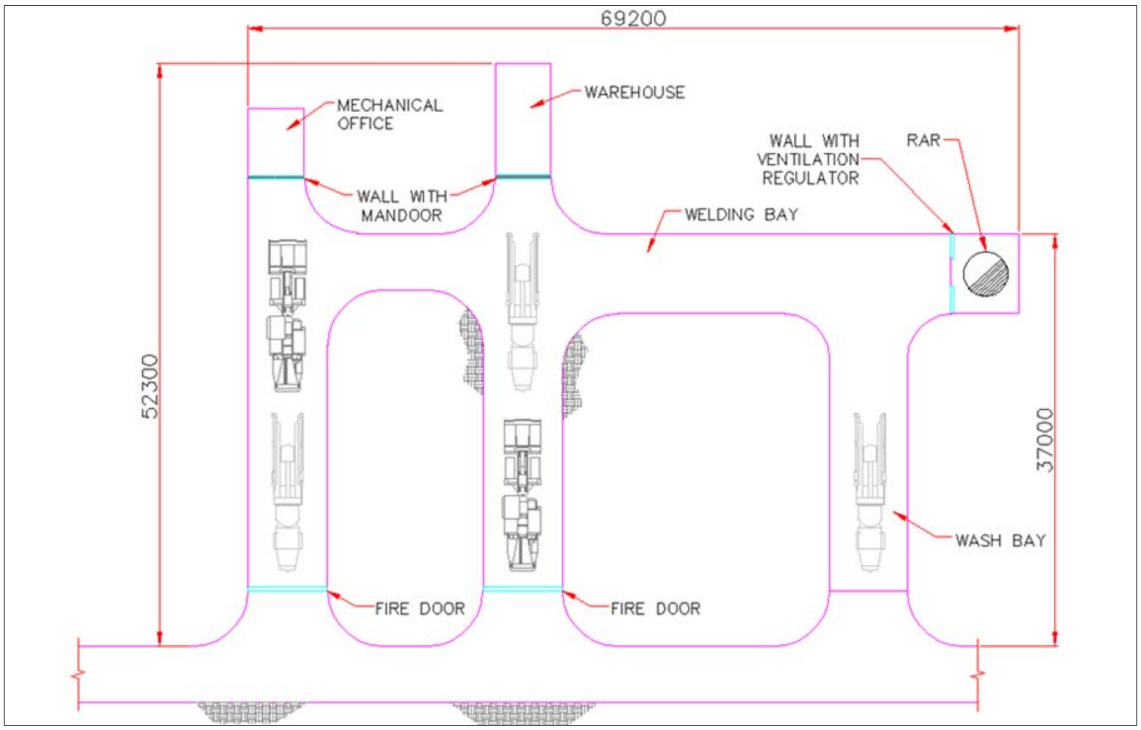
15.5.11 Maintenance Facilities

Maintenance facilities for the underground mobile fleet will consist of a main five-bay underground maintenance workshop and one single bay satellite workshop.

The main workshop is located at -380 mRL and consists of four service bays, a welding bay, an office, a warehouse, and a wash bay. In addition, one satellite workshop will be located at further depth on -580 mRL.

The underground fleet will also have access to the surface open pit truck shop located on surface, ~700 m from the mine access portal. The surface equipment related to underground (forklift, wheel loader, haulage trucks, track dozer, pickup trucks) will also have access to the surface truck shop.

The main workshop will consist of two double bays and one single welding bay, each 7 m wide by 9.5 m high (Figure 15.50). The two larger service bays will be joined by a cross-cut, allowing for forklift and foot traffic to move from one bay to the other without exiting the shop area. One sidewall along the length of each bay will accommodate tool cribs, with the cross-cut having common short-term storage of oil and greases. A drainage trench with covering grating will run the length of each bay to carry water to a nearby sump.



Source: SRK, 2017

Figure 15.50: Main workshop design

Each double bay will be equipped with a 15 t bridge crane spanning the width and running the length of the bay. The welding bay will have a 5 t bridge crane. The service area will be equipped with a stationary compressor and airlines to power air tools and provide compressed air as needed. A welding plug will also be sited in this area. Roll-up doors will separate the maintenance bays from the rest of the mine. An office and a warehouse will be located at the end of one of the double bays.

For the underground project start up, the development contractor will set up a temporary shop near the portal to service the contractor's equipment during the initial development of the decline and production levels.

The surface truck shop has been designed with one bay dedicated to the contractor's underground mobile equipment in place before project commencement. More space in this shop will become available for underground equipment as the equipment requirements for open pit mining are reducing.

The main workshop is located near the main ramp and will be ventilated by an exhaust raise to surface.

15.5.12 Explosives Magazines

The primary explosives storage area (main magazine) will be located on surface. Secondary magazines will be located underground to supply up to seven days of explosive usage.

Two explosives storage facilities are planned underground for Rory's Knoll on the -170 mRL and the -545 mRL to separately store bulk explosives and detonators.

Each explosives magazine will be located off the main decline with direct connection to the return air raise.

Each explosives magazine will be located off of the main decline with direct connection to the return air raise.

Each explosives magazine will be equipped with fire suppression, concrete floors, waste water sumps and secure steel locked gates.

15.5.13 Communications

The communications and tracking systems will consist of:

- Communications Backbone
- Leaky Feeder Network
- Telephone System
- Tracking System
- Personal Emergency Devices (PEDs)
- Operator Control

15.5.13.1 Communications Backbone

The communications backbone in the decline will be a single-mode, multi-pair fiber optic cable connected to the surface control room. The protocol will be TCP/IP over Ethernet. A fiber optic cable will also link all PLC's in the motor drive MCC's with network switching devices (OTN and Ethernet switches) to link the networks.

15.5.13.2 Leaky Feeder Network

The leaky feeder system will allow communications on the levels and will also be used as the transport system for the man and vehicle tracking system.

Two way radios for voice communications will be issued to personnel and fitted to mobile equipment.

Data and voice communication throughout the mine will be achieved through the use of an Ethernet-supported "leaky feeder network". This consists of antenna-cabling being installed throughout the main drives in the mine, with signal repeaters and boosters installed at periodic locations.

The leaky feeder network will include all of the following features:

Ethernet drops at all substations for PLC network connectivity;

Wireless coverage to all underground areas in the mine, including the outdoor areas immediately outside of the portal;

Allows several channels of voice/radio communications for individuals as well as all vehicles;

Allows control-signals to be communicated between substations.

15.5.13.3 Telephone System

A wired telephone system to be used primarily as an emergency communication system will be provided in refuge bays and at strategic points throughout the mine. This system could be augmented by voice over IP (VoIP) system.

15.5.13.4 Tracking System

Personnel tracking will be accomplished using a radio frequency identification (RFID) tag system. An integrated communications cap lamp will contain the RFID tag.

Vehicles will also contain RFID tags and radios. The system will allow operators and mine controllers to monitor, track and allocate personnel and resources.

Movement of personnel, vehicles, and other assets will be monitored throughout the mine. Having the ability to ensure that mine staff are accounted for in an emergency will increase safety and speed the provision of help to any injured personnel. Tracking vehicles and assets can also lead to increased productivity and efficiency by eliminating time wasted looking for equipment underground.

15.5.13.5 Personal Emergency Devices

All underground personnel will be issued with Personal Emergency Devices (PEDs). A separate radio communication system will be installed for this system.

Operator Control

A central control room located on surface is proposed. It will control all the major aspects of the underground operation, such as loading stations, ore passes, ventilation system, service bay and communications.

15.6 Conclusions and Recommendation

15.6.1 Open Pit Mining

15.6.1.1 Conclusions

The Aurora Mine has successfully completed its commissioning and as of this writing is in a steady state of mining production. The operations are sufficiently established so as to provide the basis of much of the technical and economic inputs required for this feasibility study update.

This study has demonstrated that AGM would be able to increase the mine production beginning in 2018 to meet a mill throughput rate of 8,000 tonnes per day. Both saprolite and rock ores are mined through 2019 with only rock ore available from 2020 onward. In order to elevate the mill feed grade in the early years, SRK segregated low grade saprolite and rock ores for later feeding to the mill at the end of the mine life.

The increase in mine production rates is enabled by the acquisition of larger loading and hauling equipment starting in 2018. This enables AGM to realize the benefits of lower unit costs as well as limit the size of equipment fleets needed in the mine plan.

15.6.1.2 Recommendations

SRK recommends that the Company begins considering equipment sources for its mining fleet expansion. In particular, low-hour used equipment should be sourced to ensure optimal economics for the mine expansion.

In preparation for the expansion in 2018, the Company should formalize all procedures and training processes in order to initiate and train new staff. This is to ensure that the efficiencies outlined in this feasibility study update are achieved.

Also, while not adopted in this study, the Company should consider acquiring a mine management system to track and potentially computer dispatch mining equipment. Up to 85 equipment units will be deployed at the Mine at its peak production.

15.6.1.3 Risks

There is risk that the production increase in 2018 may pose logistical and management challenges. A key area to consider is training of new operators and staff to ensure efficiencies envisioned in this study are realized. AGM would be advised to develop a comprehensive execution plan to ensure that milestones are met as new equipment and personnel are brought on line.

15.6.2 Underground Mining

15.6.2.1 Conclusions

The study has demonstrated that the underground operations Rory's Knoll and the satellite deposits (Aleck Hill and Mad Kiss) are economically viable in their current configurations.

15.6.2.2 Recommendations

It is expected that the resource model and the geotechnical model will be updated regularly as new data becomes available from operations and additional drilling. The updated models should be used as a basis for re-evaluating the underground mining methods and production rate to ensure the optimum strategy is advanced.

Given that the Rory's Knoll deposit is open at depth, it would also be appropriate to re-evaluate the ore handling strategy prior to committing to a decline haulage system.

15.6.2.3 Risks

Mudrush – External mudrush risk exists for the underground mine due to the heavy rainfall and the potential for generating fines and clays from the overlaying saprolite material. This risk will be mitigated by partial pre-stripping of saprolite ore as part of the open pit mining and by implementation of an effective dewatering and water diversion programs, such as perimeter drainage, collection sumps, etc.

Labour productivity – The productivity of the Guyanese workforce in an underground mining environment is not demonstrated. Timely supply of expatriate and skilled local personnel has the potential to be a very significant risk to the success of the project. The ability to adequately train local unskilled labour to the required level is also a key factor for the underground mine. Sufficient time for planning and execution of the contracting strategy must be allowed.

Execution risk – Any mining system is dependent on professional execution and there exists a risk that the planning and procedures are either optimistic or ultimately not well executed. AGM should ensure that both planning and execution capability are well-considered, are not overly optimistic, and that downside risk does not place the underground mining values proposition at undue risk.

15.6.2.4 Opportunities

The opportunities available for the underground mines are extensive, given that the commitment to commence underground mining is still some years away. Therefore, with the advantage of time and additional data (as it becomes available from operations and deeper exploration drilling) there exists the opportunity to re-evaluate almost every aspect of the underground mining strategy.

As a minimum, opportunities exist in the following aspects of the underground mines:

Mining Method: re-evaluate the range of applicable mining methods in the context of updated resource model and geotechnical data.

Ore Handling: given that Rory's Knoll deposit is open at depth, there is an opportunity to evaluate alternative ore handling strategies prior to committing to the truck haulage method.

16 Recovery Methods

16.1 Summary

The existing mineral processing plant includes cyanide leach and carbon adsorption process comprising crushing, grinding, gravity recovery, cyanide leaching, carbon adsorption, carbon elution and regeneration, gold refining, cyanide destruction and tailings disposal.

The planned plant expansion will be completed in two phases. The first phase consists of debottlenecking the back end of the circuit and includes the addition of a pre-leach thickener, three leach tanks, upgrades to the carbon management system and the expansion of the elution circuit. The first phase of the expansion is expected to commence in the first quarter of 2017 and be completed by the end of the first quarter of 2018 and will allow a throughput rate of 8,000tpd assuming the saprolite portion of the mill feed is between 25% and 50%. Due to increased retention time, the first phase of the expansion is expected to increase recoveries by approximately 1%.

The second phase of the expansion will include the addition of a ball mill, an expansion of the existing gravity circuit, the addition of one new cyanide detoxification tank and the installation of four new generators with total capacity of 6MW. The second phase of the expansion will allow the processing of hard rock material at a rate of 8,000 tpd and is expected to commence in the middle of 2018 and be completed by the middle of 2019. With the addition of the ball mill the grinding circuit product size is targeted at 80% passing (P80) 75 microns. This finer grind size is expected to translate into increased recoveries of 1% to 2%.

The overall LOM gold recovery based on upgrades to the circuits in both phases of the expansion and the existing operation are predicted to be in the range of 93.0% to 94.0%. The flowsheet upon completion of the second phase of the expansion includes one stage crushing, SAG and ball mill circuit with gravity gold recovery, leaching, carbon in pulp (CIP), carbon acid wash, elution, carbon regeneration, gold refining, cyanide destruction and tailings disposal. The crushing circuit will operate at an availability of 75%. The milling and leaching circuits will operate 24 hours per day, 365 days per year at an availability of 92%.

Gold doré is produced in the on-site refinery and stored in a secure vault prior to transportation off-site.

All of the CIP tailings will continue to be treated using an air/SO_2 cyanide detoxification system prior to tailings disposal. Tailings are then pumped to an engineered tailing management area (TMA) to protect the environment and allow for reuse of decanted tailing water.

16.2 Current Process Strategy

The overall strategy for the gold process facility design is to engineer and construct an efficient, safe, robust process facility to recover the maximum amount of gold using industry proven equipment and processes. Process control will employ standard industry techniques including onsite laboratory and instrumentation. Protection of the public, employees, and the environment is designed into each step of the mining and milling process.

16.3 Design Criteria

The Process Design Criteria and Mass Balance detail the annual ore and product capabilities, major mass flows and capacities, and plant availability. Consumption rates for major operating and maintenance consumables can be found in the operating cost estimate described in Section 20.2.3. Table 16.1 lists the key project and ore-specific criteria for the process facility design and for developing the operating costs.

Table 16.1: Initial Design Criteria

Area	Criteria	Units	Nominal Value
General	Gold Grade	g/t	3.3
	Daily throughput	tpd	8000
	Process plant availability	%	92
	Crushing plant availability/utilities	%	75
Crushing	Number of crushing stages	-	1
	Crushing system product size (P80)	mm	150
	Surge Bin Capacity (live)	t	168
Grinding	Bond Ball mill work index (hard rock)	kWh/t	14.4
	Final mill product size (P80)	µm	75
Gravity	Percent of new feed	%	86
	Number of units	#	3, 2 new
Leaching/CIL	Number of tanks	#	5, 3 new
	Tank dimensions	m dia. x m high	13.2 x 13.2
	Retention time	h	18
CIP	Number of tanks	#	6
	Tank dimensions	m dia. x m high	9.9 x 11.5
	Screens per tank	#	1 modified
	Retention time	h	10
Acid Wash, Elution, Recovery	Carbon plant capacity	tpd	8
	Acid Used	-	Nitric
	No. of Acid Wash Vessels/Elution	-	2
	Acid Wash Batch Size	t	4
	EW recovery	%	99
Electrowinning	EW capacity	-	1 relocated for IL, 1 existing and 1 new for Elution
Cyanide Destruction	Destruction time	h	1.7
	Number of tanks	#	2 existing, 1 new
	Discharge CN(WAD)	mg/L	<5

Source: GGI, Section 12 and JDS 2017

16.4 Process Facility Design Basis

The process facility design for the expanded mill is based on the following parameters:

- Sufficient process facility design flexibility for treatment of all potential ore types and blends;
- Ability to elute up to 8 tonnes of carbon per day as needed;
- Safety built into the design;
- Environmental controls built into the design; and
- Ease of operation and maintenance.

The selection of the equipment and circuit modifications are discussed in the following sections.

16.4.1 Throughput and Availability

The process comminution circuit includes a single stage crushing circuit for the rock ore material and a feeder breaker for initial processing of the Saprolitic materials, followed by one 7.9m dia. x 5.6m effective grinding length (EGL), 5.5MW SAG mill and one 5.0m dia. x 10.7m EGL, 5.0MW ball mill. The grinding mills are sized to achieve the design throughput of 8,000 tpd of rock ore upon completion of the second phase of the expansion with a 15% design factor. The major comminution design parameters are as follows:

- Crusher Work Index (CWI) of 16.2 kWh/t based on the 75th percentile of the samples tested at SGS;
- Bond Rod mill Work Index (RWI) of 16.0 kWh/t based on the 75th percentile of samples tested at SGS;
- Bond Ball mill Work Index (BWI) of 14.4 kWh/t based on the 75th percentile of the samples tested at SGS;
- Bond Abrasion Index (AI) of 0.346 g based on the 75th percentile of the samples tested at SGS;
- Target grind size P80 of 75 µm based on throughput rates and materials processed, and as determined by various leach tests programs.

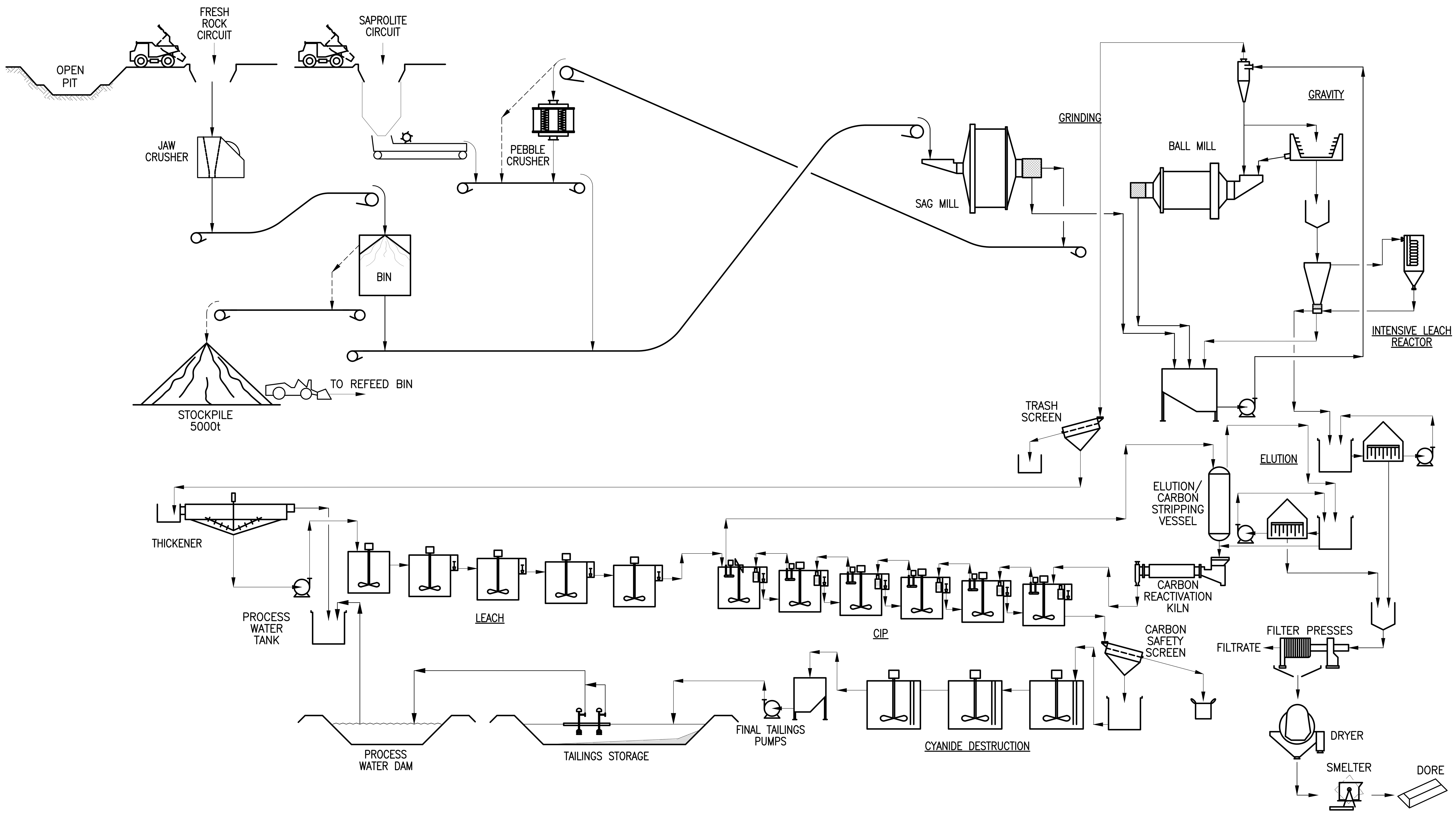
Sizing of the original crushing and grinding circuits was determined through evaluation of comminution Test work performed by SGS Mineral Services (SGS) and commonality of spares between the SAG and ball mills. The ore characterization test work provided data on Crusher Work Index, Bond Ball and Rod Mill indices, Bond Abrasion Index, and SMC and JK (A x b) values as discussed in Section 12.

16.4.2 Head Grade

The process facility is designed to treat various tonnages of ore, based on rock type, with a maximum head grade of 5.2 g/t Au. The average LOM head grade is 3.02 g/t Au, with a design annual gold head grade of 3.33 g/t Au.

16.5 Flowsheet Development

The existing process facility flowsheet design was based on test work results, previous study designs and industry standard practices for handling combinations of Rock ore and Saprolite. A summary of the process flowsheet upon completion of the second phase of the expansion is shown in Figure 16.1



REFERENCE DRAWINGS		
DRAWING NO	DRAWING DESCRIPTION/TITLE	REF
-	-	1

ISSUED FOR
REVIEW
Date: 2017/02/01

REV	YY/MM/DD	DESCRIPTION	DRWN	APVD
B	17/02/01	RE-ISSUED WITH STUDY	MF	KM
A	17/01/13	ISSUED WITH STUDY	MF	KM

CLIENT: GUYANA GOLDFIELDS INC.

CONSULTANT: JDS Energy & Mining Inc.

SUB-CONSULTANT: Allnorth

CLIENT NO:	-	DRWN:	MF	DATE:	16/12/07
PROJECT NO:	16VA0071	DSGN:	KL	DATE:	-
DRAWING SIZE:	ANSI "D"	CHKD:	MM	DATE:	-
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PROJECT:

AURORA MILL OPTIMIZATION PROJECT

TITLE:

PROCESS PLANT SIMPLIFIED PROCESS FLOW DIAGRAM

DWG NO: **16VA0071-2-00-0001** REV: **B**

16.6 Process Description

The process facility design is based on a flowsheet with unit process operations that are proven in the minerals processing industries. Material handling of the saprolite ore can be difficult due to the in-situ moisture, fine particle size, and cohesiveness if not handled properly. To mitigate the risk of downtime when handling this material well known industry engineering practices were incorporated in the design and sizing of equipment. The expanded Aurora Gold Mine gold circuit is expected to include the following unit processes:

- Primary Crushing – Existing Metso C160 jaw crusher;
- Primary Grinding – A SAG mill and new ball mill operating in closed circuit with hydrocyclones producing a final product P80 of 75 µm;
- Gravity – 1 existing QS-30 gravity concentrator and 2 new XD20 gravity concentrators;
- Pre-leach Thickener and Leaching – A new pre-leach thickener and 3 new leach tanks will be added to the existing circuit for a total of 5 leach tanks;
- Carbon Adsorption – Six CIP tanks , each with one Kemix MPS650 inter-stage screen (note: existing MPS550 screens will be converted to MPS650 screen configuration);
- Cyanide Destruction – Destruction of cyanide slurry with sodium metabisulphite for SO2, air and copper sulphate to <5 ppm CNWAD (weak acid dissociable). An additional cyanide destruction tank will be added. Two tanks will be operating with one on standby;
- Carbon Elution and Regeneration – Acid wash of carbon to remove inorganic foulants, elution of carbon to produce a gold rich solution, and thermal regeneration of carbon to remove organic foulants. An additional 4 tonne capacity acid wash/elution vessel will process additional carbon volume;
- Gold Refining – Gold electrowinning (sludge production), filtration, drying, and smelting to produce gold doré. One new larger electrowinning cell will be installed to provide additional capacity for the new elution circuit and one existing electrowinning cell will be relocated for the intensive leach solution from the gravity circuit;
- Tailings Management Area, TMA, for leach residue pumped to the engineered TMA at a slurry density of approximately 45-50% solids.

Process water is reclaimed from the TMA and recycled to the process operations, with raw water being used for make-up as required.

Raw water is reclaimed by a barge pump from the raw water dam and distributed throughout the site.

Potable water is generated on-site by the treatment of a fresh (raw) water source using multimedia filter, chlorine, ultra-filtration and ultraviolet light. Potable water is distributed for use in the process facility and around the site.

Other items included in the process include process instrumentation, control devices, and process facility and instrument air services with associated infrastructure.

16.7 Process Discussion

16.7.1 Reclaim and Crushing

Material from mining operations will feed an apron feeder - primary jaw crusher system, which will produce a crushed product size of approximately P80 of 150 mm.

Feed material is hauled to the run-of-mine (ROM) stockpile (19,000 tonne capacity) adjacent to the primary crushing station.

The feed will be managed from the ROM pad by a front end loader (FEL), which will dump rock ore or saprolite through an 800mm square-grid static grizzly and into a 100t capacity dump pocket. An apron feeder will draw material from the dump pocket at a nominal rate of 444 t/h and discharge directly into the single toggle primary jaw crusher (1600mm x 1200 mm, 160 kW). The crusher discharge will feed the bin feed conveyor. The crusher will operate 18 hours per day.

The crushed ore storage facility will consist of a crushed ore bin with an apron feeder. The apron feeder will transfer ore to the SAG mill feed conveyor.

The coarse re-feed bin will have a 168 t live capacity (or 100m^3 volume) – enough volume to provide a small surge between the crushing plant and the processing facility. Bin overflow material will be conveyed to a coarse stockpile with 5,000 tonnes capacity (equivalent to 16 hours of operations). Material will be reclaimed from the overflow stockpile using a FEL and loaded onto the SAG mill feed conveyor via the refeed bin station as required.

16.7.2 Grinding

The grinding circuit, upon completion of the second phase of the expansion, will consist of a SAG mill, a ball mill and gravity concentrators operating in closed circuit with a hydrocyclone cluster. The grinding circuit will operate at a nominal throughput of 362 t/h (rock ore feed), and produce a final particle size P80 of 75µm. The existing SAG mill is 7.9m in diameter by 5.6m effective grinding length driven by a 5.5MW motor. A new ball mill and two XD-20 gravity concentrators will be added to the grinding circuit. The ball mill will be 5.5 m diameter by 10.7 m effective grinding length driven by a 5.0MW motor.

Water will be added to the SAG mill to maintain the ore charge in the mill at a constant slurry density of 70%. Slurry will overflow from the mill to a vibrating screen, attached to the mill discharge end. The screen oversize material will be recirculated to a 315 kW cone crusher. Crushed critical sized material will then be recycled back to the SAG mill feed conveyor.

For the mill expansion, the existing 300kW cyclone feed pumps will be replaced by larger 375kW slurry pumps. The cyclone feed line will also be increased in size to handle the higher flowrate.

16.7.3 Classification

The hydrocyclone cluster classifies the feed slurry into coarse and fine fractions. The coarse underflow flows to the new ball mill for additional grinding with an off-take to the gravity concentrators. With the expansion, four additional cyclones will be added to accommodate the higher slurry flow and the flow to the gravity recovery circuit. Two cyclones will be dedicated to the current QS30, and one cyclone will be dedicated for each of the two new XD20's. The final mill product with a nominal P80 of 75 µm will flow by gravity to the thickener. The hydrocyclones have been designed with a 300% recirculating load to the ball mill.

16.7.4 Gravity Circuit

The gravity recovery and intensive leach circuits will consist of three gravity concentrators (one 30" and two 20" concentrators) with a feed trash screen, gravity tailings pump box and tailings pump, feeding a concentrate hopper, and a skid-mounted intensive leach reactor. The previous CS1000 intensive leach reactor will be replaced by a larger CS2000 intensive leach reactor.

The scalping screen prior to gravity concentration removes coarse particles and/or metal pieces that would otherwise fill the concentrator with lower grade material, reducing the capacity, and/or damaging the concentrator. Scalping screen oversize will be directed by a launder to the gravity tailings box. Periodically, the centrifugal concentrators will be bypassed and switched to flushing mode to recover the collected concentrate.

Gravity concentrate will be leached to dissolve gold by contacting material with the leach solution in an intensive leach system. The leach solution will include sodium cyanide and caustic solution. The leach solution will be mixed with the gravity concentrate in the reactor feed tank and the resulting pregnant solution will be pumped to the electrowinning circuit.

16.7.5 Pre-leach Thickener

The ball mill hydrocyclone overflow will be fed by gravity to a vibrating trash screen for removal of trash material and then feed a new pre-leach thickener. A cross-cut sampler will be installed to collect a shift composite head grade sample for metallurgical accounting purposes. A flocculent make-up system will be installed adjacent to the thickener and flocculant solution (anionic polyacrylamide) will be added to the thickener feed to promote the settling of the solids. The thickener will have a diameter of 30 m and produce a thickened slurry product of 50% solids in the underflow. The underflow slurry from the thickener will be pumped to the leach circuit (No. 1 leach tank). The thickener overflow will report to the process water tank.

16.7.6 Leaching and Carbon Adsorption

The thickener underflow will be pumped to a pre-leach screen to remove any trash ahead of the leach circuit. The intention is to relocate the existing pre-leach screen to the new first leach tank. . The expanded leach circuit is designed to provide approximately 18hours residence time – three new leach tanks will be installed for a total of five leach tanks, each 13.2 m diameter by 13.2 m high, operating in series. The agitators have down-draft air addition from the leach blowers to provide air to the leach slurry. The new leach tanks will be installed with cone spargers for additional air injection. The slurry from the No.5 leach tank will overflow to the first of six existing 9.85 m diameter by 11.45 m high CIP tanks. The CIP circuit provides approximately 1.7hours per stage (total 10h) residence time at the planned 8,000tpd capacity. Each tank includes an agitator, carbon transfer pump and a new MPS650 interstage screen.

The existing MPS550 interstage screens can be converted to MPS650 units by a simple conversion, thus increasing the screen surface area from 5.5m2 to 6.5m2 to accommodate the higher flowrate.

The average carbon concentration in the CIP circuit is expected to be maintained at approximately 20 g/L. As the leach slurry proceeds through the circuit, metal values in the leach solution will progressively decrease as the gold is adsorbed on to the carbon. Loaded carbon will be transferred from the first CIP tank to the carbon elution circuit, based on a predetermined elution cycle and schedule. The carbon is transferred sequentially, between the CIP tanks, countercurrent to the slurry flow to maximize precious metal recovery.

Loaded carbon will be collected and transferred to the elution columns in discrete batches of up to 4 tonnes per column. The tailings stream from the CIP circuit flows by gravity onto a stationary safety screen to recover any carbon particles that may have escaped from the final CIP tank. Safety screen undersize product is then report to the CIP tailings cyanide destruction circuit.

Lime slurry will be added to the first and second leach tanks to maintain protective alkalinity at a design pH of 10.0 to prevent evolution of hydrogen cyanide gas (HCN). Air will be sparged from the bottom of the three new leach tanks and the new leach tank agitators will have down-draft air injection, similar to the existing tank agitators

Two new blowers will be added to the existing blower facility to provide enough process air to the expanded leach circuit. The new blowers will be sized so that the leach tanks have sufficient air supply with one new blower operating in parallel with the existing blower (and one as standby).

16.7.7 Carbon Acid Wash, Elution and Regeneration

16.7.7.1 Acid Wash

The existing plant consists of one 4 tonne vessel that acts as both the acid wash and elution vessel. In the first phase of the plant expansion, an additional 4 tonne capacity acid wash/elution vessel will be installed to process the additional carbon. Both columns will be operated sequentially to process approximately 6.5 – 7.0 tonnes of carbon, in two batches, per 24hour period.

Loaded carbon, consisting of a batch of approximately 3.7 - 4 tonnes, is treated with a dilute nitric acid solution in one of the two acid wash/elution vessels to remove calcium deposits, magnesium, sodium salts, and fine iron particles. Organic foulants such as oils and fats are largely unaffected by the acid and will be removed after the elution stage by thermal reactivation utilizing a diesel fired rotary kiln.

Nitric acid will be pumped from the dilute acid tank to the acid wash/ strip vessel. Acid will be pumped upward through the carbon bed and exit the column, gravitating to the residue detox tanks. The carbon will then be rinsed and neutralized with the injection of fresh water to remove the acid and any mineral impurities. This wash solution also gravitates to the residue detox tanks.

16.7.7.2 Carbon Stripping (Elution)

The carbon stripping (elution) process utilizes a split-AARl method to produce a pregnant solution, which is pumped through electrowinning cells for gold recovery.

During the strip cycle, solution containing approximately 3% sodium hydroxide and 3% sodium cyanide at a temperature of 115°C (280°F) and 450 kPa (65 psi) is pumped into the column. Subsequently a hot solution of weak electrolyte solution, from the previous elution cycle is pumped through the column to de-adsorb the gold. This is then followed by a cold water wash to remove residual gold and silver in solution. This cold water wash solution becomes the weak electrolyte solution, collected in the weak electrolyte tank, for the next elution cycle.

Solution (pregnant solution) exiting the top of the elution vessel is cooled through the heat recovery heat exchanger. Heat from the outgoing solution is transferred to the incoming cold solution, prior to the cold solution passing through the solution heater. An electric boiler is used as the primary solution heater. The resulting pregnant solution flows to the electrowinning circuit to recover the gold and silver in solution.

Electrowinning will proceed on a batch basis with pregnant solution circulating through the EW cells for a specified time or until a pre-determined solution gold value is attained. The barren solution is then pumped to No.1 leach tank and the next electrowinning cycle begins.

16.7.7.3 Carbon Regeneration

A recessed impeller pump will transfer the stripped carbon from the elution vessel to the kiln feed dewatering screen and hopper. The kiln feed screen doubles as a dewatering screen and a carbon sizing screen, where fine carbon particles will be removed. Oversize carbon from the screen will discharge by gravity to the carbon regeneration kiln feed hopper. Screen undersize carbon, containing carbon fines and water, will drain by gravity into a new carbon fines collection tank. A diesel fired horizontal kiln with residual heat dryer is utilized to regenerate the carbon at a rate of approximately 240kgs per hour. The regenerated carbon gravitates into a small quench tank and then into No.6 CIP tank.

16.7.7.4 Carbon Fines Collection Systems

New ancillary equipment required for the expansion includes a carbon attrition tank and carbon fines collection systems for both new carbon fines (non gold-bearing) and process carbon fines (gold bearing).

New (fresh) carbon is added on a regular basis to the CIP circuit to compensate for the attrition losses occurring throughout the circuit. New carbon will be added to a mechanically agitated attrition tank to remove fines and round off the sharp edges of the carbon granules. These fines will be screened from the carbon, with the oversize being added to the CIP circuit and the fines being discharged to residue.

Regenerated carbon will be screened before being returned to the CIP circuit. This carbon will contain some gold values depending on the elution process stripping efficiency. The screen oversize product will be transferred to No.6 CIP tank and the screen underflow (fines\ collected in a new settling tank, together with the kiln feed hopper screen undersize product, prior to dewatering in a filter press. The filtered product will be processed off-site for gold recovery.

16.7.8 Gold Electrowinning and Refining

The elution electrowinning circuit includes two electrowinning cells. Pregnant solution from the elution vessel will be pumped to the pregnant solution tank at the refinery for electrowinning to produce a gold sludge The solution will be circulated through the electrowinning cells for a predetermined time or until a predetermined solution value is achieved. The electrowinning barren solution is then pumped to No.1 CIP tank.

To accommodate the higher gold production at 8,000tpd one of the EW cells will be replaced by a larger EW cell of approximately twice the capacity.

The gravity electrowinning circuit consists of a single electrowinning cell tank and intensive leach preg tank with internal heater to maintain the temperature of the solution.

Gold rich sludge from the EW cells isbe washed off the steel cathodes using high pressure water into the sludge holding tank. Periodically, the sludge is drained, filtered, dried, mixed with fluxes and smelted in an electric direct-fire induction furnace to produce gold doré. This electrowinning and smelting processes takes place within a secure and supervised area. The gold doré is stored in a vault prior to shipment to a refinery.

16.7.9 Cyanide Destruction

The cyanide destruction of CIP tailings thickener underflow will consists of two existing and one new mechanically agitated tanks, each with a capacity of approximately 465 m^3. Two tanks will be operational and one new tank on standby. Cyanide will be destroyed using the SO_2/Air process. Treated slurry from the cyanide destruction circuit will flow by gravity to the final tailings pump box.

Process air is sparged near the bottom of the 8.2 m dia. by 9.8 m high cyanide destruction tanks, under the agitator impeller, for 1.9 hours total residence time, with two tanks operating. Lime slurry will be added to maintain the optimum pH of 8.5 – 9.0 and copper sulphate (CuSO4) will be added as a catalyst. Sodium metabisulphite (SMBS) will be dosed into the system as a solution as the source of SO2. The detox slurry is transfered to the final tailings pump box.

One additional blower will be added to the existing two blowers to provide process air to the detox tanks. Two blowers will operate simultaneously with the third acting as a standby.

16.7.10 Final Tailings

Cyanide destruction slurry is sent to the final tailings pump box. Final tailings process stream is currently being pumped by two x 90kW tailings pumps (one operating, one standby) to the final Tailings Management Area (TMA). For the expansion, two larger 225kW pumps and a larger tailings pipeline or a duplicate parallel line will be installed to handle the increased flowrate.

16.7.11 Water Supply and Consumption

The following types of water will be used in the process plant.

- Process water: overflow water from the pre-leach thickener will be used as process water. Process water will be used predominantly in the grinding circuit to dilute slurry to the required densities.
- Fresh water: fresh water for the process plant will be used as reagent make-up water, gland water, and for cooling water in the strip circuit boiler. The estimated fresh water consumption in the process plant is about 20 m^3/h.
- Reclaim water: water reclaimed from the catchment impoundments, tailings facility or directly form the water treatment plant will be used as make-up water in the process as required. The estimated reclaim water consumption is about 750 m^3/h.

16.7.12 Reagents

Logistics and security of supply are an important management function throughout the mine life due to the project location. Individual commodities are discussed further down in this section.

16.7.12.1 Sodium Cyanide

Sodium cyanide (NaCN) is delivered in 14 t isotainers. Batches of NaCN are prepared on-site in the NaCN mixing facility. The bulk transport isotainers are commonly used in industry and feature zero personnel exposure to NaCN briquettes.

The isotainer is unloaded at the cyanide mixing area using the designed mixing skid. The water for mixing is circulated through the isotainer to dissolve the dry NaCN product to produce a 28% w/w solution. On completion of mixing, the solution is pumped to the NaCN storage tank using the transfer pump.

The NaCN solution is distributed to each dosing points using dedicated chemical metering pumps specially designed for NaCN duty (with a duty/standby configuration). Line pressure is maintained by the use of an automated diaphragm valve. Dosing to each point is by a flow control valve or variable speed pump and flow meter.

16.7.12.2 pH Modifier – Lime

Quicklime (CaO) is delivered to the site in 20 t bulk containers. Lime is transferred to the lime silo using a hydraulic container unloader and blower. A belt feeder will transfer the lime as required into a slaking mill where it is combined with water to generate slaked lime (milk of lime). A hydrocyclone is used to classify the lime slurry and return excess grit back to the slaking mill for further grinding.

The slaked lime is stored in an agitated storage tank, where it is recirculated around the process facility in a ring main system using the lime ring main pumps (duty/standby arrangement). The lime addition to grinding, leach and detoxification are controlled automatically using pH probes and control valves. Redundant pH monitoring is used in the leaching and detox area.

16.7.12.3 Activated Carbon

Activated carbon is delivered to the site in 500 kg bulk bags.

16.7.12.4 Sodium Metabisulphite

SMBS is supplied in 1000 kg bulk bags as a dry reagent. SMBS is used as a source of SO_2 for the cyanide destruction circuit (air/SO_2 process).

SMBS bulk bags are lifted by an overhead hoist and loaded into the mixing tank by way of a bag splitter. Dilution water is added to produce a solution. The diluted solution is transferred to the SMBS storage tank. Metering pumps (duty/standby configuration) dose SMBS to the detox circuit as required.

16.7.12.5 Copper Sulphate

Copper sulphate is supplied in 1000 kg bulk bags as a dry reagent. Copper sulphate is a chemical used as a catalyst for the cyanide destruction circuit (air/SO_2 process). Copper sulphate bulk bags are lifted by an overhead hoist and loaded into the mixing tank by way of a bag splitter. Dilution water is added to produce a solution. The mixed solution is transferred to the copper sulphate storage tank via the transfer pump.

Metering pumps (duty/standby configuration) dose copper sulphate to the cyanide detoxification circuit as needed.

16.7.12.6 Sodium Hydroxide

Sodium hydroxide (caustic soda) is supplied in 1000 kg bulk bags as a dry reagent. Caustic soda bulk bags are lifted by an overhead hoist and loaded into the mixing tank by way of a bag splitter. Water is added to the agitated tank to produce a solution. A metering pump delivers caustic soda to the process as required.

16.7.12.7 Nitric Acid

Nitric acid is supplied in 550 gallon totes. Nitric acid is diluted with water before being used in the acid wash circuit.

16.7.12.8 Fluxes

Sodium borate, more commonly known as borax, along with silica flour, soda ash and potassium nitrate is delivered to the site on a pallet containing 25 kg bags.

16.7.12.9 Grinding Media

Forged carbon steel grinding media is delivered to the site in bulk bags. SAG mill media is added to the system using a front-end loader and are added via the refeed hopper. A blend of 100mm and 125mm diameter balls is added to the SAG mill.

16.8 Process Control Philosophy

Field instruments will provide inputs to a set of programmable logic controllers (PLCs). Process control cubicles are located in the motor control centres and will contain the PLC hardware, power supplies, and input/output cards for the instrument monitoring and loop control. The PLCs will perform the control functions by:

- Collecting status information of drives, instruments, and packaged equipment;
- Providing drive control and process interlocking;
- Providing proportional-integral-derivative control for process control loops;
- Standard personal computers are located in the main control room and the crusher control room. The computers is networked to the PLCs and operate a supervisory control and data acquisition (SCADA) system that provides an interface to the PLCs for control and monitoring the process facility. The SCADA system is configured to provide outputs to alarms, control the function of the process equipment, and provide logging and trending facilities to assist in analysis of process facility operations;
- Uninterrupted power supplies will provide operating control stations with 20 minutes of standby power.

The general control strategy adopted for the Aurora Gold Mine Project gold process facility is as follows:

- Integrated control via process control system (PCS) for areas where equipment requires sequencing and process interlocking;
- Hardwired interlocks for safety of personnel;
- Motor controls for starting and stopping of drives at local control stations, via the PCS or hardwired depending on the drive classification;
- All drives can be stopped from the local control station at all times. Local and remote starting is dependent on the drive class and the control mode;
- Control loops via the PCS except where exceptional circumstances apply;

- Monitoring of all relevant operating conditions on the PCS and recording select information for data logging or trending; and
- Trip and alarm inputs to the PCS is fail-safe in operation (i.e., the signal reverts to the de-energized state when a fault occurs).

Drives that form part of a vendor package are controlled from the vendor's control panel. At a minimum, "Run" and "Fault" signals from each vendor control panel is made available to the SCADA system via the PLC. Where practical, the PCS will interface with the vendor control panel to provide full operating status, including state of all drives, alarms, and instrument outputs.

16.9 Process Facility Infrastructure and Services

16.9.1 Air

Two rotary screw type air compressors provide high-pressure air for the process facility and instrument air requirements. There is one duty and one standby compressor operating in lead-lag mode.

Air for leaching and detoxification is provided by three low pressure compressors.

16.9.2 Water

Raw water is supplied from the Fresh Water Pond. This water is used for utility purposes including the Fire Water reserve.

Domestic water including safety showers is sourced from the process facilities Raw Water Supply and is treated and filtered prior to distribution. Following treatment the water will meet South American standards for domestic consumption.

16.10 Conclusions and Recommendations

- Utilize the pebble crusher for new feed from the jaw crusher in addition to pebbles recycled from the SAG to increase the capacity in the grinding circuit.
- Evaluate opportunities to reuse existing or used equipment in the mill expansion.
- The FS design involves replacing the existing tailings pumps and pipeline with two larger pumps and a new line. A trade-off study is recommended to add a third pump in parallel to the existing pump. This new pump would serve as a common standby and offer cost savings in terms of increased operating time and reduced maintenance. The trade-off study would evaluate the benefit of adding a third pump.

17 Project Infrastructure

17.1 Project Logistics

During the mine life a combination of transportation methods, including aircraft, river navigation, and road access will be used to supply the project. Overall, the offsite infrastructure includes docking facilities for cargo ships at the Buckhall Port facility on the west side of the Essequibo River. The mine access road is 150 km in length from Buckhall Port to the mine site.

During production, the main road will be mainly used for the supply of food, reagents, spare parts, mining supplies, and diesel fuel. The site airstrip will be used mainly for personnel transportation and emergency situations.

17.2 On-Site Infrastructure

The site entails a series of open pits, waste rock stockpiles, a process facility with associated laboratory and maintenance facilities; maintenance buildings for underground and open pit equipment. Facilities and structures include a warehouse, office, change house facilities, ventilation shaft, mine air cooling process facility, explosives storage area, power generating station, fuel storage tanks, a warehouse and laydown area, a 1,200 m airstrip, and a permanent accommodation complex. The open pit area is protected from potential flooding of the Cuyuni River by a river dike.

Areas for the tailings management, fresh water, and mine water management ponds are provided by several dams at low topographic points. These ponds, and all except one of the waste rock stockpiles, are on the west side of the process facility, away from the accommodation complex. Additional waste is located to the north of the process facility, also away from the accommodation complex.

17.2.1 Power Plant and Distribution

The electrical power supply for the project is a site-based diesel power plant. The power plant consists of eight 1.6 MW generators, based on an N+1 operational philosophy (with 100% power plant availability), for a total installed initial generating capacity of 13 MW. The Company has recently purchased four generators with total installed generating capacity of 6MW. The new generators should be sufficient to meet the power requirements of the expanded processing circuit.

The power plant's generating sets will generate power at 13.8 kV, 60Hz. The process facility's main electrical room is fed with two 13.8 kV lines from the main power plant in order to ensure full redundancy. All other loads of the project are fed at 13.8kV from the power plant through overhead power lines. These power lines are used to deliver power to various locations to support activities during the construction of the project.

The power plant has its own fuel storage facility. Given the remoteness of the Aurora Gold Mine site and its accessibility, a one month on-site fuel storage capacity is provided to accommodate for continuous operation of the power station.

17.2.2 On Site Roads

Project site roads include haul roads suitable for use by mining trucks and service roads for use by smaller vehicles. The site roads are for use by authorized mine personnel and equipment, with access controlled by AGM.

Saprolite covers the entire project area. Roads will, as far as practical, be constructed using cut and fill techniques to achieve design alignment and grade. Placed saprolite fill will require compaction in small lifts in order to provide a competent road foundation. At several locations the compacted saprolite fill has been used as a dike to divert surface water drainage and protect the mining areas from water ingress.

Both haul roads and service roads have been surfaced with a layer of crushed rock fill to facilitate all-season use. Road surfacing material will break down in time, and frequent re-surfacing during the mining operation will be required as part of an ongoing road maintenance program. Dust control on the roads is performed using water trucks, or possibly dust suppressants as needed. Haul roads and service roads have shoulder safety berms equal to half of the height of the largest vehicle tire that traverses that road.

All roads are designed to a maximum gradient of 10%. The vertical alignment of the roads is influenced by the water management plan. The perimeter roads around the mining area are designed with a minimum road surface elevation of 60 m. At a number of locations these roads serve as dikes to protect the mining areas from flooding from the Cuyuni River. Service roads that cross the Cuyuni River flood plain are designed to a minimum road surface elevation of 57 m, so that they can remain in service during a flood. The roads also cross a number of ephemeral stream channels, where large culverts have been installed. At these stream channel crossings the road surfaces are elevated in order to provide adequate cover over the culverts.

The initial roads from the gatehouse location to the Cuyuni River have been constructed and serve as the main haul road providing access to the laydown area, power plant, process facility and ore crusher locations, as well as the mancamp.

Other haul roads connecting the open pits with the ore crusher, ore stockpile site, waste rock stockpile, and the TMA, have also been completed. All haul roads are designed for two-way 40 t truck traffic, with the exception of the haul road from the rock quarry to the TMA location. This haul road is designed for the single lane 40 t articulated truck traffic, with pullouts to permit truck passing.

Approximately a 15 to 20 km network of the service roads have been built providing access to environmental discharge points, airstrip, explosive storage facility TMA and the MWP.

17.2.3 Utilities and Services

17.2.3.1 Fresh Water Supply, Fire Suppression Water and Distribution

Raw water is sourced from water wells drilled into the underlying bedrock, collection of surface water from creeks and rain water harvesting systems. A rain water harvesting area has been constructed at the mancamp to supply fresh potable water and fire suppression water for building services such as dining facilities, showers and toilets. An in-line chlorine metering system will disinfect the water supply. Potable water for the process facility and operations and maintenance facility will be obtained from a roof collection system on the operations and maintenance building.

Fresh raw water supply is obtained from a fresh water pond about 1 km south of the process facility. This water is primarily for fire protection, make-up requirements for the process facility, fluidization and flushing for the gravity concentrators, cooling the drives and lube systems, use in the strip solution heat exchanger, reagent preparation, and gland water distribution.

17.2.3.2 Sewage Collection and Disposal

A sewage treatment process facility has been constructed just east and downhill of the mancamp site. Buried sewer pipes collect sewage from the site to the treatment process facility. The treatment process facility consists of two independent containerized treatment lagoon systems working independently and providing redundancy if one unit must be shut down for maintenance. The proposed system is capable of treating 10,000 L of wastewater per day. Treated effluent will be released to the Cuyuni River via a local tributary.

Domestic sewage from the process facility and operations and maintenance facility will be collected and treated by a sewage lagoon east of the operations and maintenance facility. Treated effluent will be released to the Cuyuni River via a local tributary.

17.2.3.3 Site Security

The principal site entry point on the access road from the Tapir Crossing will be provided with a lighted security gate and vehicle access barrier. A masonry block gatehouse building will provide sanitary facilities, communications equipment and search facilities including metal detection. A weighbridge will be located adjacent to the gatehouse building to enable incoming and outgoing vehicle load monitoring. The site entrance will be monitored by closed circuit television (CCTV) from the operations and maintenance building. CCTV monitoring will be provided at the process facility, gold room and along the Cuyuni River.

17.2.3.4 Communication and IT Systems

Point-to-point satellite communication will be the main communication system between the mine and the outside world. The system includes voice/data/video/fax, internet, and VPN services, including bi-directional links between the mine site and Georgetown.

A backup/emergency satellite system will be provided for redundancy. The backup/emergency system includes voice/data/fax, TV and internet access for a minimum number of users.

VHF/UHF radio communication will be available within a 10 km radius from the process facility. The phone system will be a voice over internet protocol. This will reduce wiring costs and allow voice-messaging integration with e-mail. End-to-end IP video connectivity with business quality transmission will provide video conferencing capabilities. At least three satellite phones installed at strategic areas will be provided for emergency communications.

Satellite TV for entertainment, cellular communication, and FM radio will be provided.

A cellular phone system from Buckhall Port to the site will be installed. This system will be a joint effort of AGM, Barama Logging Company and the government of Guyana.

The IT system will be based at the operations and maintenance building and connected throughout the site by a fiber optic network. The connection between IT devices and end-users will provide high-throughput, secure, reliable and redundant service for data and voice. The network system will be connected to protocol independent multicasts (PIMS) and business networks through routers with firewalls and will provide remote access as required. The system will have security and encryption to prevent unauthorized access.

17.2.3.5 Vehicle Fueling Facility and Mine Equipment Ready Line

The vehicle fueling facility and ready line will be located adjacent to the open pit mine. The fueling facility will store about 400,000 L of diesel and gasoline. Smaller tanks will hold a variety of oils and lubricants. The ready line will be located adjacent to the fueling facility and will be well lighted for 24 hour use.

17.2.4 Site Buildings and Facilities

17.2.4.1 Operations and Maintenance Building

The Operations and Maintenance building is a single story pre-engineered, steel-framed structure with a spread footing foundation. Profiled metal deck roof cladding and open facades with concrete block walls set back 1.5 m to form covered walkways will be included. The building provides offices for administrative and technical staff, including management, training, accounting, safety, and security. It will also include staff support facilities such as a conference room, print room, and lunch room.

The open pit mine operations building, electrical room, and a compressor room will be constructed adjacent to the maintenance shop. Both buildings will be pre-engineered with spread footing foundations.

The mine operations area, approximately 1,200 m^2 in size, will house the following: mine operation staff office, maintenance staff office, heavy equipment/high-rack storage warehouse on the ground floor, low-rack storage warehouse on the mezzanine floor, first aid room, lunch room, locker room and toilets.

The equipment maintenance shop is designed to repair and maintain the mine fleet and other mobile equipment. It will consist of four bays for heavy mobile equipment repairs and maintenance, two bays with two lifting hoists dedicated for heavy vehicle maintenance, two bays allocated for a machine shop, tire servicing, and other major repairs. A light vehicle maintenance building will be located adjacent to the operations and maintenance building.

A 50 t bridge crane will be provided for two bays and storage area. A separate truck wash station, equipped with a washing system with a water/oil separator for heavy mining equipment, will be installed outdoors.

17.2.4.2 Mancamp

The permanent accommodation complex has been constructed on a 10 ha elevated site southeast of the mine complex. The accommodation complex incorporates the following dormitory styles:

Type A dormitories are private, single-occupancy rooms;

Type B facilities are semi-private and have single-occupancy rooms with two rooms sharing one shower and toilet room;

Type C dormitories are double-occupancy rooms with a central shower and toilet facility shared by 30 rooms.

The accommodation complex also includes the following facilities:

Kitchen, dining hall;

Recreation, exercise, and entertainment facility. The building will also have a 50-seat cinema that can also be used for meetings and staff training;

Cricket/soccer field;

Infirmary equipped with trauma treatment facilities as well as life support equipment. The center will be comprised of a waiting/reception area, doctor's office, treatment room/theatre, two bed wards, washroom facilities, electrical/janitorial room, storage room and ambulance parking; and

Emergency power plant. Power for the accommodation facilities will be provided through overhead lines from the central site generating station. An emergency generating station will provide power for essential services.

17.2.4.3 Explosives Magazine

The explosives magazine has been constructed and operated in accordance with Guyanese law. A modular facility is used so it can be relocated during the life of the mine. The site is surrounded by a perimeter security fence with lights.

17.2.4.4 Airstrip

The airstrip will be upgraded prior to dike construction to provide for personnel access, transportation of sensitive equipment, and medical emergencies. The new facility will consist of a runway. hangar and storage for emergency and firefighting materials.

Incoming and outgoing flights will be scheduled for daylight hours only. Temporary lighting will be employed along the airstrip in the event of night time medical emergencies. The runway will be 1,200 m long x 30 m wide with 90 m runway end safety areas at each end. The elevation of the airstrip and related access road will be above the flooding level for continuous serviceability during flood seasons.

Aircraft maintenance and fueling will be performed in Georgetown; thus, no provision for aviation fuel storage facilities will be provided.

Dust suppressants will be used on the runway as required to reduce dust emissions during periods of little or no precipitation.

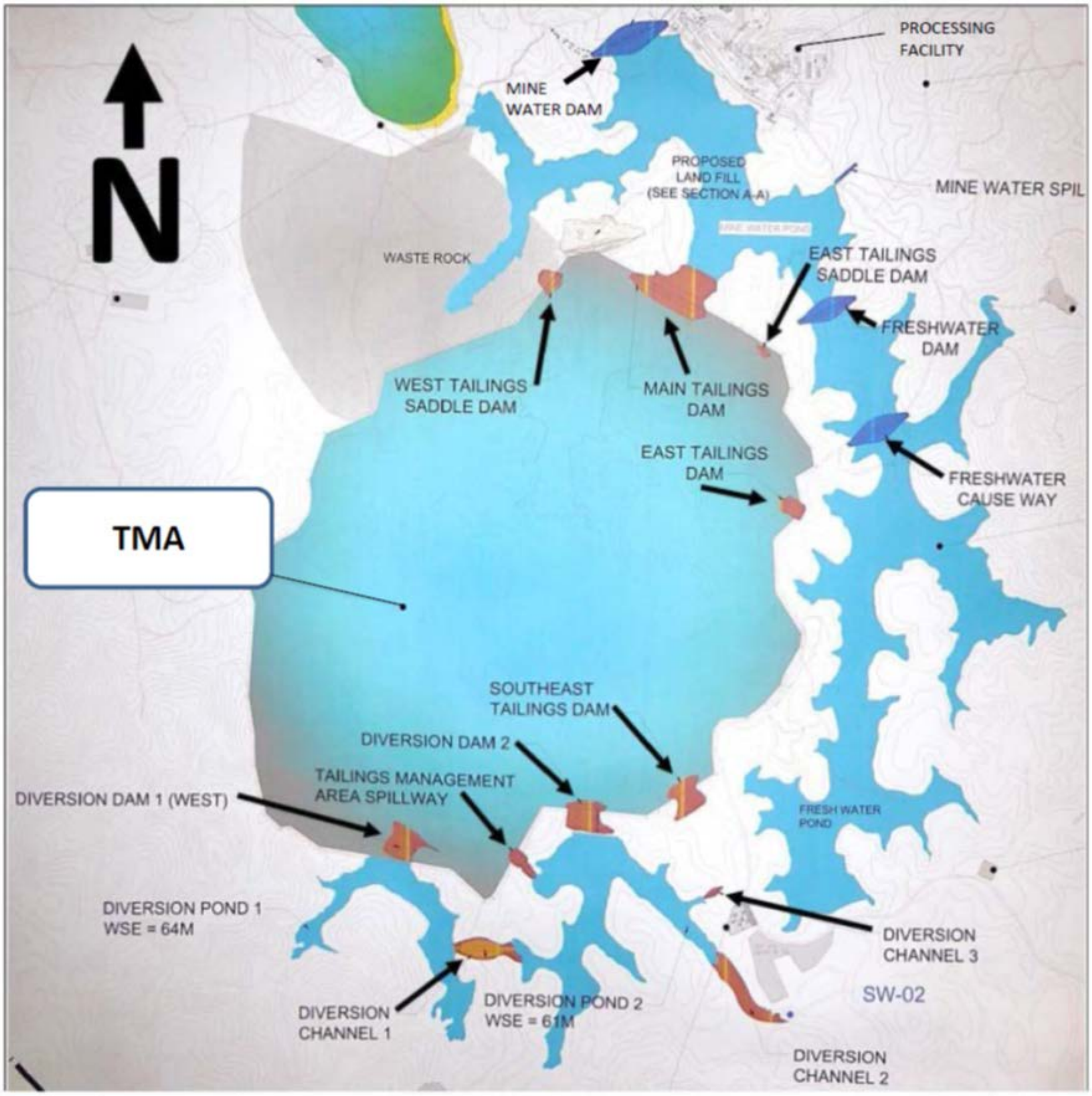
17.2.4.5 Solid Waste Disposal and Recycling Facility

Non-recyclable, non-toxic solid waste are disposed of in an onsite lined landfill. Used tires will be shredded and placed in the landfill.

17.2.5 Tailings Management Area

17.2.5.1 Overview of the TMA and Related Design Criteria

The tailings management area (TMA) is located about 1 km southwest of the process facility. It is nearly circular in shape and covers an area of approximately 240 ha. Tailings are delivered to the TMA through a 2 km long pipeline and discharged through multiple spigot points. Containment at the TMA is based on local topographic highs, complemented by a series of earthen dam segments (Figure 17.1).



Source: SRK, 2017

Figure 17.1: Layout of the TMA and adjacent dams and ponds

Water from runoff and tailings supernatant collects in the TMA pond and a barge-mounted pump sends reclaim water through a pipeline to the processing facility. Excess water is discharged at the southeast corner of the TMA through an overflow spillway into Diversion Pond 2 and then to a local receiving watercourse which flows northeast to the Cuyuni River.

Tetra Tech, the TMA designer, reports that the design of the TMA has been based on best management practice standards, as published by the international mining community, such as:

- CDA

- FEMA
- USACE and
- IFC Environmental Health and Safety Guidelines

The TMA design criteria are summarized in Table 17.1.

Table 17.1: TMA design criteria

Component	Requirement	Source
Embankments		
Hazard Classification	High Consequence Dam	CDA(1)
Minimum Factor of Safety: Operational Condition (long term)	1.5	CDA(1)
Minimum Factor of Safety: End of Construction Condition (short term)	1.3	CDA(1)
Minimum Factor of Safety: Seismic Loading (pseudo-static analyses)	1.0(3)	CDA(2)
Design Earthquake	Use 50% to 100% of Maximum Credible Earthquake	CDA(1)
Hydraulic Structures		
Design Storm: Diversions, Ponds, and Channels	100 year – 24 hour	CAD/USCE
Design Storm: Spillway and Containment	Probable Maximum Precipitation (PMP)	CAD/USCE
Freeboard	3 meters(4)	CAD/USCE
Other Parameters		
Tailings Production	5,000 tpd	-
Required Storage	31 million tonnes	-
Reclaim Flow Rate	TBD	-
Water Storage	Not a critical design parameter	-

From OMS Manual dated January 20, 2016

Notes:

(a) Design Criteria established for high hazard dam classification as specified by CDA
(b) CDA refers to Hynes-Griffin and Franklin (1984) as one of the preferred methods to evaluate slope stability using pseudo-static analysis.
(c) For added conservatism, design considered a Factor of Safety of 1.1.
(d) Freeboard allows for embankment settlement and wave action.

The tailings and diversion dams are designed as homogenous, low permeability earth fill (re-compacted saprolite) structures. A blanket drain will be constructed along the downstream portion of each dam, at the limits of the starter dam.

The TMA dams have been designed to be constructed in multiple stages using the downstream construction method. A crest width of 12 m will be maintained at each construction stage.

Upstream slopes will be constructed at 2H:1V and downstream slopes at 3H:1V. The crest of the starter dam will be at a maximum elevation of 66 m; it is expected that the final stage of the Main Tailings Dam will be at an elevation of 78 m.

The emergency spillway has been designed to discharge storms routed through the TMA up to the probable maximum flood (PMF), while maintaining at least 1 m of residual freeboard below the dam crest. The routing of the 100-year and PMF storms were initiated with the TMA water surface at the spillway invert elevation. The spillway invert will be raised in conjunction with each dam raise.

17.2.5.2 Starter Stage Construction

The starter stage construction included the Main Tailings Dam (MTD), East Tailings Dam (ETD), Southeast Tailings Dam (SETD), Diversion Dam 2 (DD2), East Tailings Saddle Dam (ETSD), West Tailings Saddle Dam (WTSD), Fresh Water Dam (FWD) and Mine Water Dam (MWD). These dams, shown in Figure 17.1, were constructed by AGM's construction Department between September 2014 and March 2015. Quality control during construction was provided by Tetra Tech. The as-built status of these dams, other than the East and West Tailings Saddle Dams, is summarized in Table 17.2.

Table 17.2: As-Built conditions for TMA and water management facilities components

Phase	Maximum Embankment Elevation (m)	Reference Document/Document Number
TMA Starter Stage		Early Works Engineering Interim As-Built Report and Weekly Report
MTD	66	
ETD	66	
SETD	66	
DD2	66	
MWD	58	
FWD	60.5	
Conveyance and Distribution System	N/A	
TMA Emergency Spillway	Invert Elevation N/A	2016 Semi-Annual Site Inspections and Assessments

From OMS Manual dated January 20, 2016

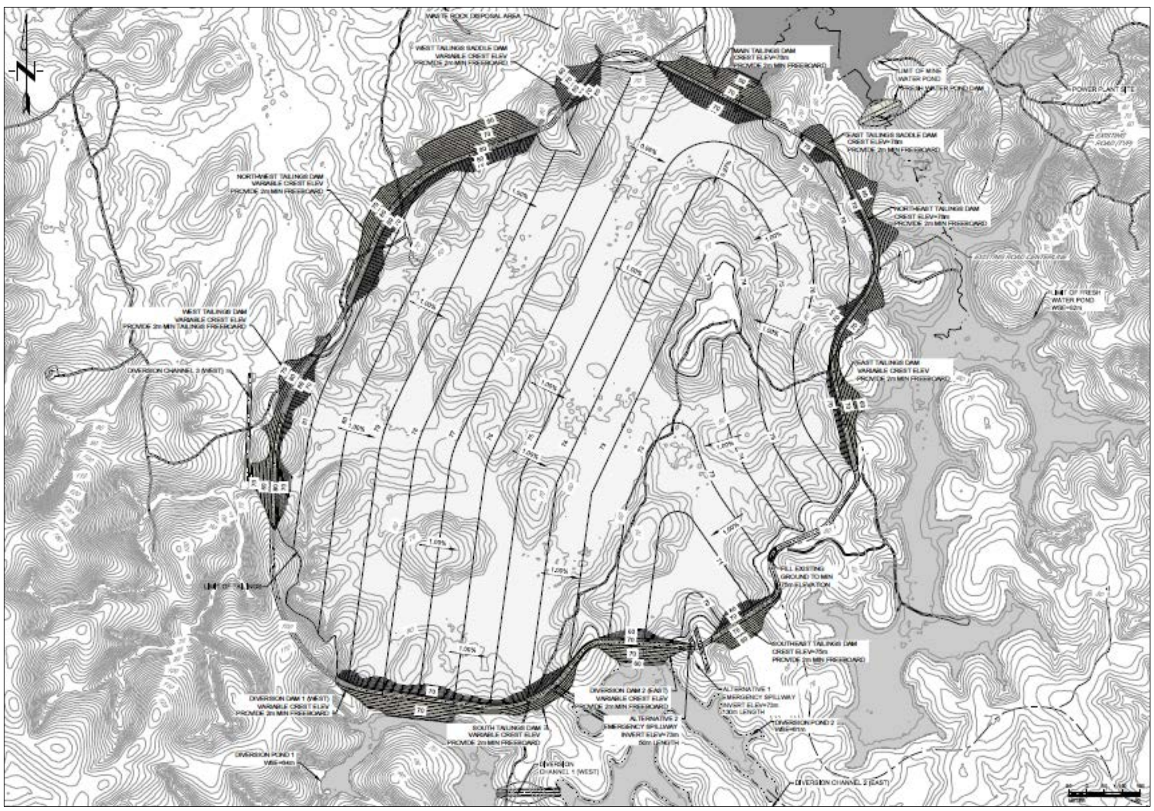
Based on Table 17.2, the TMA dams (MTD, ETD, SETD and DD2) currently have a maximum crest elevation of 66 m. Low ground in the vicinity of these dams corresponds to an elevation of approximately 50 m, so the current height of these structures is about 16 m. The number of construction stages to get to the final dam height (elevation 78 m) has not been established.

The invert elevation of the emergency spillway for the starter dam is not provided in the as-built documentation.

17.2.5.3 TMA Operation

Initial startup of the processing facility and TMA commenced in 2015; commercial production was reached on January 1, 2016. Tailings are pumped from the processing facility through a 2 km long pipeline and spigotted into the TMA at various locations around its perimeter. The discharge locations are moved regularly to limit the frequency at which the reclaim barge must be re-positioned and to ensure that the tailings beaches are sloped in a way that promotes the pooling of excess water near the spillway on the south side of the TMA. As a result, the length of the tailings delivery pipeline will increase over time, ultimately reaching a length of approximately 5 km.

The beach slopes at the end of operations are illustrated at a conceptual level on Figure 17.2. Deviations in these slope directions will vary somewhat during operations, but the general gradients on the tailings surface will need to be shaped in a way that will prevents any of the TMA dams to be overtopped during the design flood event.



Source: SRK, 2017

Figure 17.2: Layout of the TMA at its final stage of operation

Consistent with best practice, a manual describing operations, maintenance and surveillance (OMS) of the TMA has been prepared (Tetra Tech, 2016). A key task within the OMS manual is the requirement that the "Third Party Dam Engineer" complete an annual inspection and safety assessment. Tetra Tech completed such inspections in 2015 and 2016.

17.2.5.4 TMA Closure

As the TMA approaches closure, the discharge of tailings will need to occur in a way that minimizes the accumulation of water on the tailings surface. Therefore, it is planned that the tailings surface will be consistent, to the maximum practical extent, with the tailings beach slopes shown in Figure 17.b.

At closure it is expected that the tailings surface will be capped and re-vegetated. The spillway will remain in place to safely convey flows off the TMA surface, although settling and/or polishing ponds may need to be constructed as part of the closure plan. In addition, the tailings delivery pipeline, barge and reclaim water pipeline will be removed

17.2.5.5 Impacts to the TMA based on Proposed Project Amendments

The current design of the TMA is based on a tailings storage demand of 31 Mt and a production rate of 5,000 tpd. Based on this design, Tetra Tech has estimated the final crest elevation of the Main Tailings Dam to be 78 m.

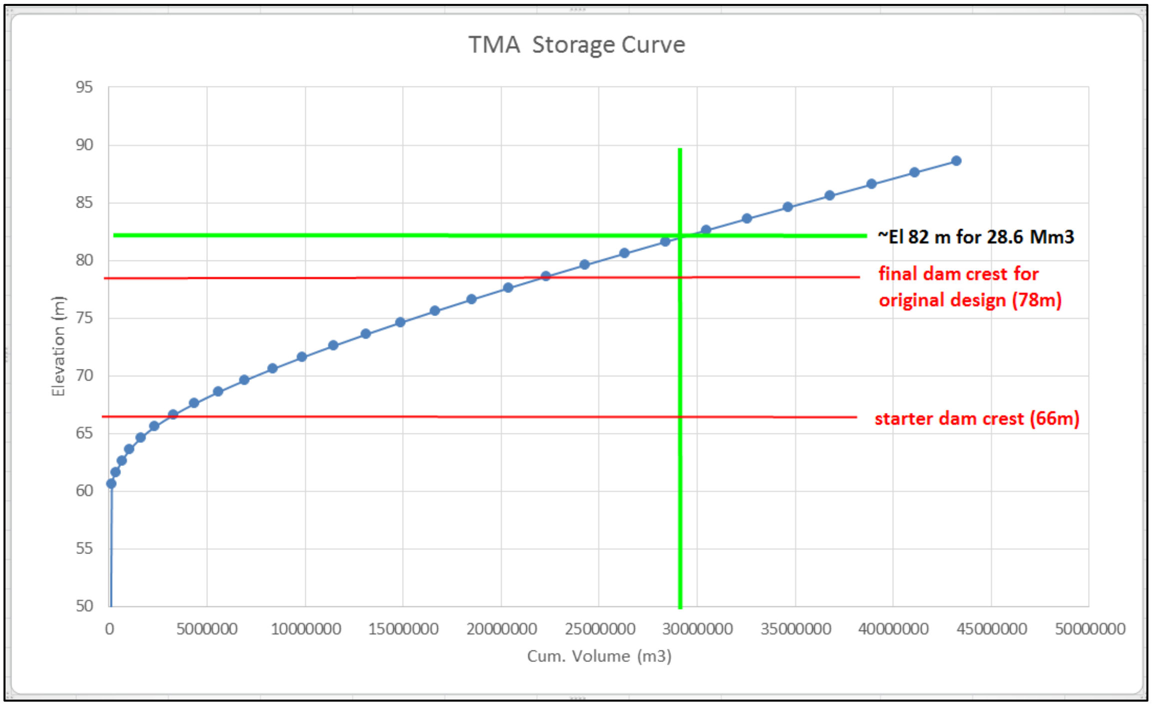
Changes to the total expected quantity of tailings and the production rate are being proposed. In particular, the tailings quantity that will produced between 2017 and 2032 amount to 37.3 Mt. The production rate is targeted to be 8,000 tpd. Allowing for the current design plant throughput of approximately 6 months in 2015 and 12 months in 2016, the tailings tonnage already in the TMA amounts to 2.7 Mt. Coupled with the future tailings noted above, the total tonnage of tailings to be stored at the TMA amounts to 40.0 Mt.

Tetra Tech has indicated that, based on experience on other projects, the average dry density of the tailings is likely to be 1.4 tonnes per cubic metre (tcm) when fully consolidated. Based on this value, 40.0 Mt of tailings corresponds to 28.6 Mm^3.

A storage capacity curve for the TMA was constructed as shown on Figure 17.3. This curve was based on the following:

- The topography is based on the original topographic data with which we were provided;
- The TMA perimeter outline was assumed to provide vertical containment to the tailings;
- The shape of the tailings surface used to calculate the storage volumes was assumed to be consistent with the surface shown on Figure 17.2; and
- The elevation in the storage capacity curve corresponds with the maximum tailings level on the west side of the TMA, i.e. similar to Figure 17.2 (not struck level).

Implicit in the last bullet is the understanding tailings on the south side of the TMA, near the emergency spillway, will be the minimum tailings level.



Source: SRK, 2017

Figure 17.3: Storage capacity curve for the TMA

Figure 17.3 suggests that in order to store the revised tailings quantity, the final dam crest would have to be raised by approximately 4 m, from elevation 78 m to 82 m.

The TMA dam fill volumes for the starter stage amounted to 0.18 Mm^3. Dam fill volumes associated with raises above the starter stage are influenced by the fact that the TMA is a circular facility; the downstream construction method is used for staged dam raises; and the length of each dam typically increases with each raise. Tetra Tech has indicated that the dam fill for the starter stage (elevation 66 m) amounted to 0.18 Mm^3 and an additional 1.93 Mm^3 of dam fill is required to store 31 Mt of tailings (elevation 78 m). In order to store the incremental 9 Mt of tailings associated with the proposed project amendment, an additional 0.8 Mm^3 (elevation 82 m) of dam fill is required. That would bring the total TMA dam fill volume at the end of the LOM to approximately 2.9 Mm^3.

17.2.6 Fresh Water Pond

The fresh water pond (FWP) is located immediately east of the TMA and will hold approximately 600,000 m^3 at the NHWL. The water is obtained from a 140 ha drainage basin. Overflow from the FWP discharges through a spillway into the mine water pond. The water is used for make-up requirements for the process facility, fluidization and flushing for the gravity concentrators, cooling the drives and lube systems, use in the strip solution heat exchanger, reagent preparation, and gland water distribution. The water is pumped from a barge mounted pump through a 1 km pipeline to the process facility.

17.2.7 Mine Water Pond

The mine water pond (MWP) is located immediately south of the open pit mine and has a capacity of 750,000 m^3 at NHWL. The MWP receives water from a 113 ha drainage basin, the 110 ha open pit mine drainage basin and groundwater inflow to the open pit mine. The MWP is designed to detain water for a minimum of 7 days prior to discharge. Discharges from the MWP flow through a concrete box culvert spillway to a tributary of the Cuyuni River.

17.2.8 Emergency Discharge Pond

The emergency discharge pond is located immediately west of the process facility. This pond is a double lined facility used on an emergency basis if a vessel in the process facility needs to be emptied. The pond has a capacity of 2,000 m^3.

17.2.9 River Dike

A river dike has been constructed to mitigate flood risk to open pit and underground operations during high river levels. Breach or overtopping of the dikes could potentially result in extreme losses (e.g., loss of life, and long and costly interruption of mining operations). A new river dyke will be constructed closer to the river in 2018 to accommodate the larger open pit design as per this report.

The current dike and proposed dyke are comprised of a homogenous, low permeability earth fill structure, with a relief well system to control seepage at the dike-foundation contact, a toe drain to control the phreatic surface and keep the downstream shell drained, and revegetated on the river side and land side slopes. A road surface with bedding layer adequate for maintenance vehicle traffic is provided on the dike crest.

The crest elevation of the new river dike will be at an elevation of 60 m. The dike will be about 700 m long, with a 5 m crest width, 3H:1V slope on the river side and a 2H:1V slope on the land side.

The current dike and proposed dyke are constructed of locally available fill materials, including the silty clay to clayey silt (Saprolite), and non-reactive waste rock from the open pit excavation. The sand and gravel, if not available from a local or nearby source, is produced by crushing and screening nonreactive waste rock.

17.3 Off-Site Infrastructure

17.3.1 Buckhall Port

Buckhall is the logistics hub for the Aurora Gold Mine and is located on the west bank of the Essequibo River about 24 km up river from the Atlantic Ocean. The facility includes a wharf, pier, bulk fuel storage, barge slip, customs clearing area, and laydown/staging areas. The facility includes a government-run customs entry port. This site also includes an administration building, two story dormitory buildings, kitchen/dining hall, vehicle maintenance and security. Some equipment is assigned permanently to this facility. There is a light vehicle maintenance facility. A heavy vehicle maintenance shop is constructed for the contract hauling fleet. The site has been fenced and topped with security razor wire to deter unauthorized access.

The pier can accommodate up to 3,000 t sea-going cargo vessels and landing for barge vessels that transship from sea-going vessels too large to travel up the Essequibo River. The fuel depot includes two existing 94,000 L steel diesel fuel storage tanks, one 1-million L tank for diesel and one 2-million L tank for No. 4 fuel. Each fuel type is transferred from incoming fuel barges to the storage area through a series of dedicated pipes and flexible hoses. Both areas include spill containment berms constructed of concrete and masonry with an underlying HDPE liner. The containment volume of the spill containment berm is set no less than 1.5 times the volume of the tanks.

Water is provided from a well and supplemented with rainwater collected from the roofs. Sewage is treated in septic tanks and discharged to leach fields in accordance with Guyanese design regulations. Diesel generators produce electrical power. The communication system includes locally available mobile phone service and satellite dish internet service.

17.3.2 Tapir Crossing

A ferry is located on the Cuyuni River at Tapir Crossing. The ferry has a capacity of 120 t, is self-propelled and operate in forward and reverse. The landings is upgraded and river jetties provided to direct high river flows away from the landing areas.

A barge is currently used to transport vehicles and their cargo directly to the west side of the Cuyuni River. The barge is used as a backup to the ferry.

The site has 24 hour per day, seven day per week security on both sides of the river. Staffing is housed at the Aurora Gold Mine site and transported to Tapir Crossing for each shift.

17.3.3 Access Road

The road distance between the Buckhall Port facility and the Aurora Gold Mine site is approximately 170 km. The road alignment initially follows the north shore of the Cuyuni River, then crosses over at the Tapir Crossing to continue on the west side of the river to the mine site. Much of the existing road was constructed by Barama Company Limited for logging operations.

An improvement plan has been developed that includes a land survey of the road centerline to establish road stationing. The road will require a wearing surface. Drainage structures and log bridges will be replaced and upgraded within the first 5 years of operations.

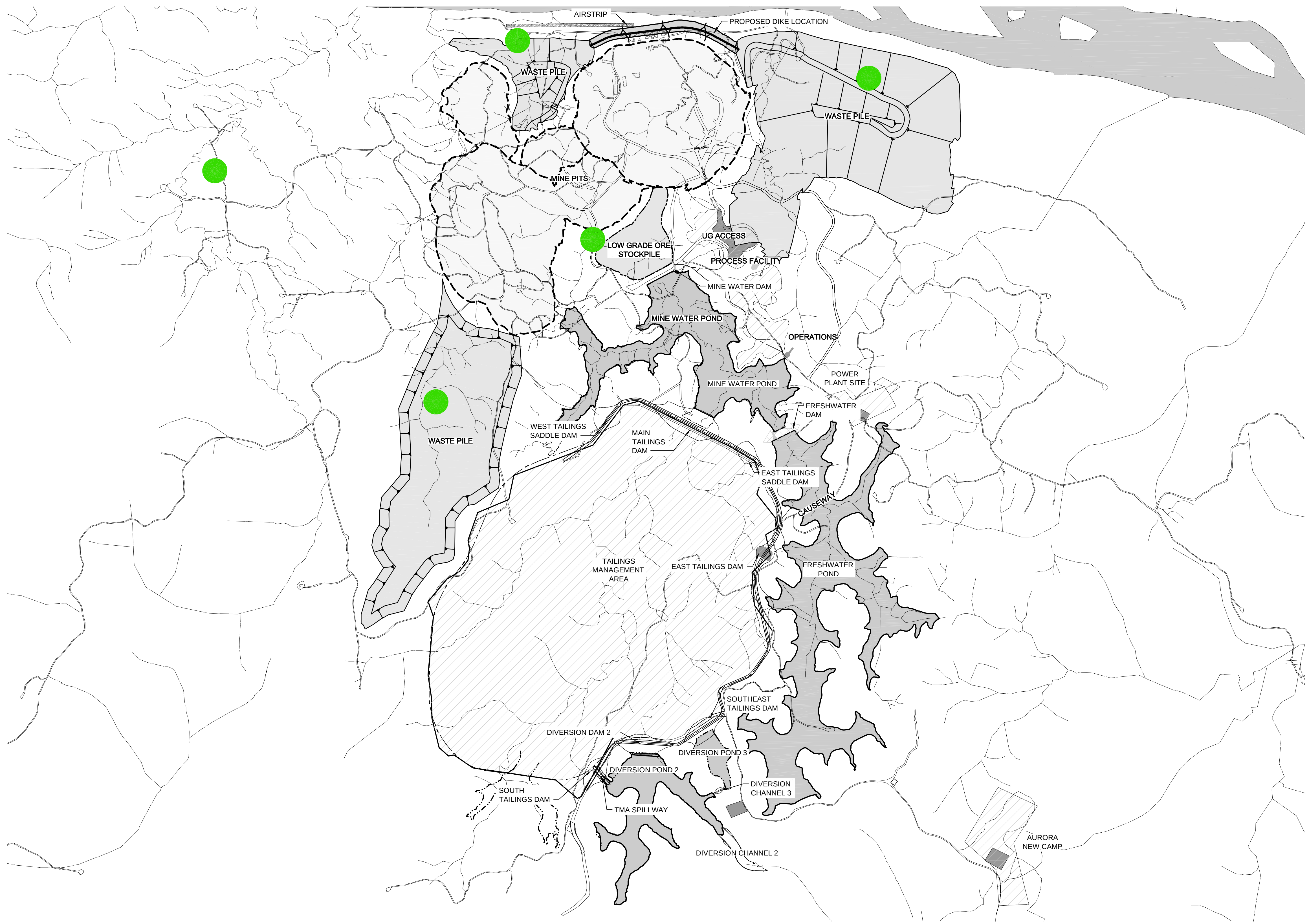
17.4 Overburden Stockpile and Tailings Disposal, Site Monitoring and Water Management

17.4.1 Waste Rock / Overburden Disposal

Overburden and waste rock is placed in four sequentially constructed stockpiles near the perimeter of the open pits; one additional stockpile is constructed for waste saprolite. As noted in

Section 17.4.1, results of static and kinetic humidity cell testing indicate that ARD conditions are not likely to be generated in any of the waste rock/overburden or saprolite stockpiles. The locations of the waste rock/overburden and saprolite stockpiles were selected to reduce the potential for surface water infiltration and to minimize haul distances. Runoff from these stockpiles is and will be directed either through appropriate sediment control measures or into the MWP for settling prior to controlled discharge to the environment (see Sections 17.2 and 17.4.4). The stockpile sedimentation control measures and/or MWP are also being used to reduce sediment load haul road runoff. Sedimentation control measures are designed to provide the required sediment removal for runoff from a 2 year, 24 hour storm.

Additional details on waste rock/overburden and Saprolite stockpile design are presented in Section 17.4.1.

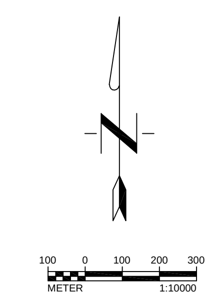


NOTES:

1. ALL ELEVATIONS, GRID COORDINATES AND DIMENSIONS ARE IN METRES.
2. GRID COORDINATES ARE REFERENCED TO UTM, PSAD56-SOUTH AMERICA, ZONE 21.
3. ALL ELEVATIONS ARE ORTHOMETRIC AND WERE DETERMINED BY APPLYING EGM-96 GEOID MODEL.
4. EXISTING GROUND CONTOUR DATA PRODUCED FROM THE LIDAR SURVEY FLOWN JANUARY 13TH – 17TH, 2010,BY AIRBORNE SOLUTIONS INTERNATIONAL LLC.
5. WATERCOURSES (CREEKS) AND DRAINAGE CHANNELS SHOWN ARE INFERRED.

LEGEND

- FREE DRAINING OPEN WATER
- RECLAIM AREA
- REMOVE EMBANKMENT
- OPEN PIT HIGHWALL AND WASTE ROCK SLOPES TO REMAIN AS CONSTRUCTED
- STOCKPILES PROGRESSIVELY CLOSED IN OPERATIONS PHASE



REFERENCE	ENGINEER'S SEAL	Rev	Description	BY	Date
		A	ISSUE FOR REVIEW	JLS	01/09/2017

REVISIONS

Scale: AS SHOWN

Designed by: JLS
Drawn by: JLS
Checked by: JMG
Approved by: RJG

Issued for:

AGM INC.

Issued by:

TETRA TECH
350 Indiana Street, Suite 500
Golden, Colorado 80401
(303) 217-5700 (303) 217-5705 fax

FEASABILITY STUDY
SITE PLAN

REVISION ----

Project: EARLY WORKS PROJECT
Project no.: 114-850053
Location: CUYUNI MAZARUNI, GUYANA
Date: ----
SHEET 1 OF

17.4.2 Tailings Disposal

AGM has adopted a conventional tailings disposal method. Detoxified tailings will be deposited in the TMA via a dedicated tailings slurry pipeline. TMA reclaim water will be pumped back to the process facility for process use. There will be one main tailings dam constructed at the north end of the TMA. Additional smaller dams will be constructed around the perimeter of the TMA (Diversion Dam, Saddle Dams, and additional Tailings Dams). The Diversion Dam was constructed to reduce stream flow entering the TMA from a basin located to the south of the TMA. Runoff along with released tailings water will have an average retention time of 5 months prior to being discharged under the mean annual precipitation conditions. The TMA will be equipped with a two level spillway, which will be raised along with each dam raise. The low flow spillway is designed to provide the required retention time. The high flow spillway is set to provide 1.0 m of freeboard above the Probable Maximum Flood (PMF) level. The final TMA dam crest for the main tailings dam will be at 78 m elevation with the south tailings dams and Diversion Dam at 75 m. The low flow spillway for the final dam configuration will be at 73 m and the high flow spillway at 74 m elevation.

The lining system for the TMA consists of in situ saprolite with naturally very low permeability. Runoff and tailings supernatant water will have a minimum retention time of 5 months prior to being discharged, under mean annual precipitation conditions. Over the first four years of operation, when all tailings production is deposited within the TMA, the mixing ratio for tailings water and precipitation within the TMA capture area is estimated to be between 1:3 and 1:6 under mean annual precipitation conditions.

Water quality in the TMA, MWP, and FWP will be monitored via a surface and groundwater well monitoring program. The water quality monitoring program at the Aurora Gold site is compliant with guidelines set forth by the International Finance Corporation (IFC) for mining and for landfills, the World Health Organization (WHO) drinking water standards, Guyana Environmental Protection Agency (Guy-EPA), and the United States Environmental Protection Agency (USEPA). Water samples will be collected on a quarterly basis.

Grab sampling will be performed at groundwater, stream and creek monitoring locations. Monitoring of groundwater and surface water leaving the project site will also be conducted using a network of monitoring wells and surface water monitoring stations. Up-gradient monitoring wells and at least one up-gradient surface water monitoring station will be established to collect background water quality data.

Sampling from most locations on site would be compared against baseline and a trend analysis would be performed to quantify any changes to water quality.

For locations exhibiting discharge, monitoring data will be compared against established criteria. In the event of an exceedance, the following steps will be undertaken:

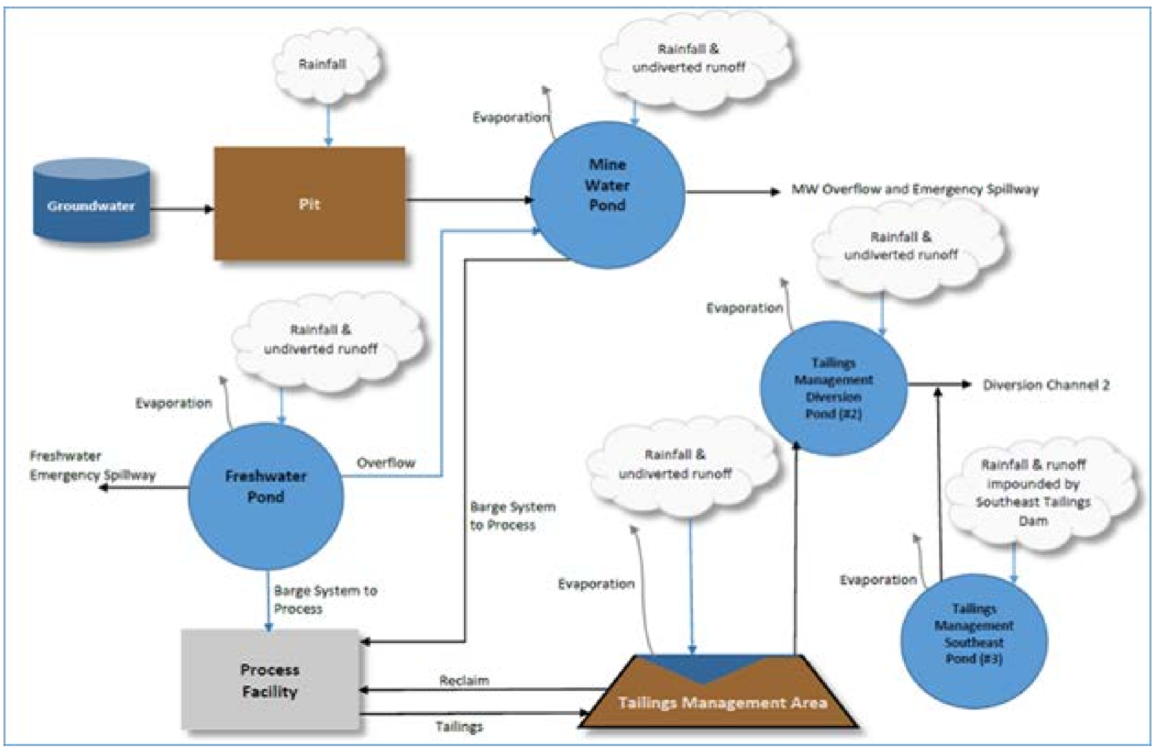
1) Re-sampling will occur within 30 days of receipt of analytical data, and if there are no exceedances, normal monitoring resumes;

2) If exceedances continue, additional data evaluation will include outlier tests, data distribution and trend analyses;

3) If the analyses indicate there is an increasing trend of a particular parameter, site investigations will be undertaken to ascertain the source and sampling frequency will be re-evaluated.

17.4.3 Sitewide Water Balance

An initial water balance for the site was developed using the GoldSim™ software platform. A model was prepared to simulate reservoir and pond operational levels from measured and predicted precipitation and other project inflows and outflows. A conceptual representation of the water balance is shown in Figure 17.5. Site implementation of water management is consistent with the modeled water balance. Excess water will be discharged to the environment from the MWP and will be discharged from the TMA via the spillway. Once the basins are filled, these discharges will occur on a continuing basis. Water discharged from the TMA will first pass through the (dilution) reservoir at Diversion Dam 2 prior to entering the environment. Water from the MWP is diluted by discharges from the FWP prior to discharge to the environment.



Source: Tetra Tech, 2017

Figure 17.5: Conceptual model, site water balance

The water balance model software supports an array of probabilistic functions, and, with the regular input of a wide range of water monitoring data, will support the operation of the TMA, MWP, and FWP to maintain appropriate flows, water quality, and freeboard throughout the TMA starter dam phase of operations in compliance with the Aurora Project Tailings Management Area Management Plan and Cyanide Management Plan. The water balance model will be updated following each tailings embankment raise or as significant new data from operations are obtained.

17.4.4 Water Management

The Mine is located in a tropical setting with substantial rainfall, and the management of water is a necessary operational matter. There are seven primary structures that will be used to manage surface water. These include:

- The Cuyuni River dike
- Overburden and waste rock storage areas,
- The Fresh Water Pond (FWP),
- The Mine Water Pond (MWP),
- TMA Diversion Pond 2,
- an emergency discharge pond for the process facility, and
- TMA.

A water management plan has been developed for the site to provide flood protection, handle water impacted by mining activity, and manage site runoff. The plan forms the technical basis of the Aurora Gold Mine Water Management Plan. Its objectives are to: divert as much water as possible away from the open pit and TMA; minimize the use of raw surface or groundwater and maximize recirculation of process water; reduce the sediment load in runoff from waste rock stockpiles; maintain trafficable site access during storm events; and be compliant with guidelines set forth by the International Finance Corporation (IFC) for mining and for landfills, the World Health Organization (WHO) drinking water standards, Guyana Environmental Protection Agency (Guy-EPA), and the United States Environmental Protection Agency (USEPA). The main features of the site water management plan are discussed further below:

- Cuyuni River Dike - A saprolite earthfill river dike is to be constructed alongside the Cuyuni River in order to mitigate potential flood risks to the project. The dike will be designed for flood and earthquake criteria [i.e., a 1-in-100-year flood level and one-half of the peak horizontal ground acceleration predicted for the maximum credible earthquake (MCE)]. The length of the dike will be approximately 750 m, with a 5 m crest width, 3H:1V slope on the river side, and a 2H:1V slope on the downstream side (pit side). A gravel surfaced road will be located on the crest. The river dike will be anchored into the side slopes of adjacent overburden and waste rock piles.

- Placement of Overburden Stockpile areas (Overburden and Waste Rock Piles) are to be constructed in areas along the Cuyuni River between the River Dike and higher areas of ground to also mitigate potential flood risks. These piles will be designed with the same criteria as the river dike.
- Fresh Water Pond Operation (FWP) - the FWP has be constructed to serve as a makeup water source for the process facility. A dam was constructed at a crest elevation of approximately 60.5 m to form the pond and retain about 454,000 m^3 at the normal high water line (NHWL) of 58 m. During future operational stages the overflow elevation will be raised to the NHWL of 59 m and will hold approximately 600,000 m^3. The pond will fill from precipitation over the 140 ha drainage basin. The excess water from the FWP up to and including the 10-year 24- hour storm flow rate will be discharged through the FWP service spillway into the MWP. Stormwater flow over and above the 10-year 24-hour event will bypass the MWP and flow out the high flow spillway to the Cuyuni River.
- Mine Water Pond (MWP) Operation - water from the open pits, underground operations, and any other site operational water not reporting to the TMA will be pumped to the MWP for clarification, retention, eventual discharge.

The MWP will receive water from its 87 ha drainage basin and the open pit mine's 122 ha drainage basin. The current capacity of the MWP is designed to provide storage capacity for approximately 2.5-days retention of the runoff volume from a 10-year 24-hour rainfall storm. The pond is sized to receive an average pumping rate of about 200 m^3/hr. The average retention time during a normal precipitation year will be about 6 months. The MWP spillway consists of culverts that function both as a low level outlet works and an overflow spillway structure. Water flow into the WMP exceeding the capacity of the MWP will flow out the spillway outlets.

Pit dewatering requirements will be reduced by diverting site runoff away from the open pit. The design concept is to divert runoff away from the open pit in a progressive or staged manner to avoid accumulating large flows and volumes of surface water near the pit perimeter. This will be achieved by diverting site runoff that would normally reach the open pit into diversion channels that will be pumped to the MWP or eventually be discharged into the Cuyuni River.

Clean runoff from any areas of the pit which will not be disturbed by stripping or mining activity will be collected and routed to the environment. In general, runoff will be collected in ditches alongside the site access roads and haul roads, and routed to the diversion channels. Where topography and site layout make ditching impractical, runoff will be collected into small ponds and pumped to discharge points.

- Emergency Discharge Pond Operation; the emergency discharge pond will be located immediately east of the process facility. This pond will be a double lined facility used on an emergency basis if a vessel in the process facility or tailings pipeline needs to be emptied. The pond will have a capacity of 2,000 m^3.

18 Market Studies and Contracts

18.1 Markets

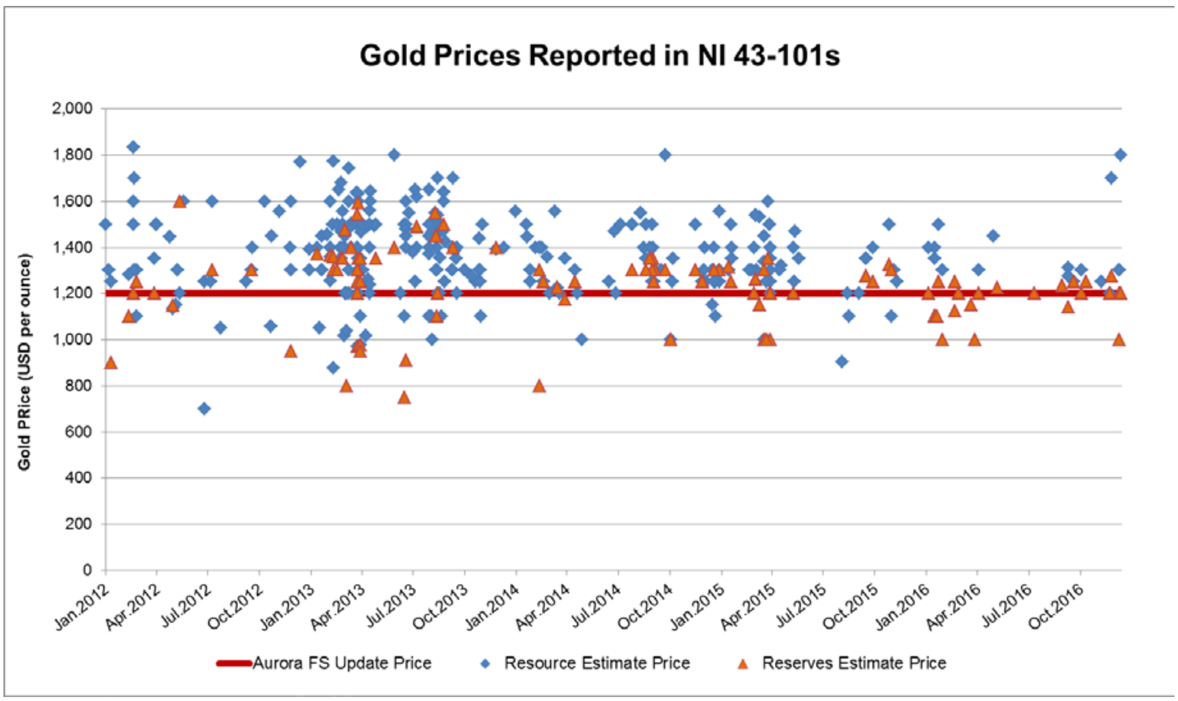
Aurora gold mine produces gold doré bullion, a fungible commodity with an active global market. Gold doré has a high value density meaning:

(a) the realized price of gold contained within doré is not sensitive to customer location and

(b) the refinery and freight costs are negligible in comparison to contained value.

Refinery terms of 99.95% payable gold and refining charges of $0.30 per ounce, as provided by GGI Inc. in November 2016, are typical terms currently being offered for gold doré bullion produced using a carbon-in-leach process.

For mine design and mineral reserve estimation, AGM has set a gold price of $1,200 per ounce. SRK reviewed long-term gold price forecasts and considers the selected price to be reasonable. Figure 18.1 shows a history of gold prices over the last 10 years.

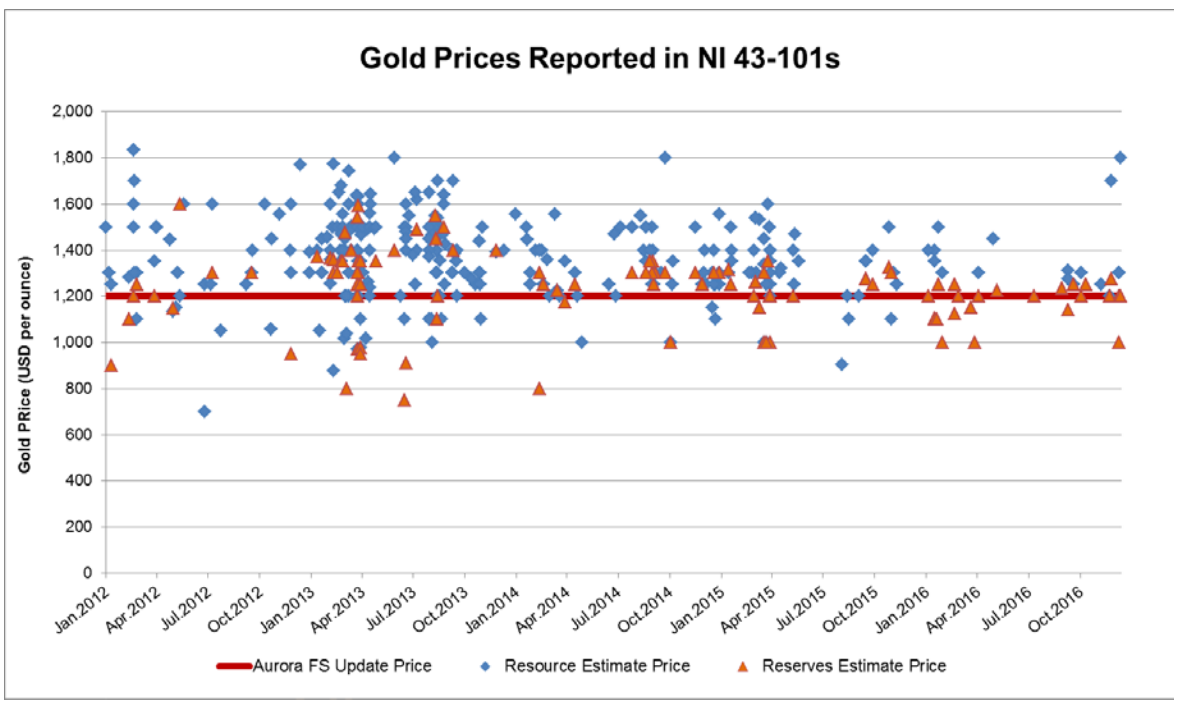


Source: SEDAR, Public Company Documents Search (2016)

Figure 18.2 shows gold prices from 2012 to 2016 for resource and reserves estimates reported in NI 43-101 compliant studies similar to the Aurora Gold Mine.

Source: SNL Financial, Mining and Metals Database (2016).

Figure 18.1: Ten (10) Year average gold price



Source: SEDAR, Public Company Documents Search (2016)

Figure 18.2: Gold prices from NI 43-101 reports

In summary, SRK supports a long-term gold price forecasted at $1,200 per ounce for use in the Aurora project feasibility study.

18.2 Contracts

AGM has entered into a refining contract with Asahi Refining Canada Ltd. AGM will deliver all annual production of doré to Asahi. Asahi will recover 99.95% of the gold and 99.25% of the silver from the doré. Asahi will deliver a credit for recoverable metals notifying AGM of the recoverable metals credits to AGM. Gold and silver will be sold at spot market prices.

19 Environmental Studies, Permitting, and Social or Community Impact

19.1 Summary

This section of the report presents the results of a desktop review of the environmental and permitting issues related to the Aurora Gold Mine.

This review was based on written documentation provided directly to SRK by GGI, a telephone conversation with the GGI's VP Sustainability and Health & Safety, Reed Huppman, and a review of the document obtained from the GGI website entitled. Final Updated Environmental and Social Impact Assessment, Aurora Gold Mine (ENVIRON 2013).

19.2 Permitting

Aurora is subject to a number of regulatory permits and licenses issued by several different Guyanese governmental agencies. The Aurora regulatory register is presented in .

This table indicates 29 permits have been granted. Most of them have expiry dates with renewal requirements. For example, the mining permit requires annual renewal whereas the environmental permit must be renewed every five years, or when there is a significant modification to the site or its operations.

The capacity production increase from 5,000 tpd to 8,000 tpd requires approval by the Environmental Protection Agency (EPA) of Guyana. The process of obtaining this approval requires the submission of an updated ESIA, the draft of which has been submitted to the EPA and is currently undergoing regulatory review. The regulatory review process is relatively straight forward and no delay with the approval is expected.

Table 19.1: AGM Regulatory Register, as at June 30, 2016

Licences/Permits/Lease	Permitting Agency		Applicable Legislation	Status	Date Granted	Expiry Date	Requirements/Actions	Comments/Remarks
Mining License	**LM41: G1**	Guyana Geology and Mines Commission (GGMC)	**a)** Mining Act **b)** Environment Regulations, Mining (Amendment) Regulations **c)** Environmental Protection Act, 1996	Granted/Active	18-Nov-11	17-Nov-16	Renewed annually on or before the Anniversary date	Renewed annually
Environmental Permit	**2009114GGIOO**	Guyana Environmental Protection Agency (EPA)	**a)** Environmental Protection Act (Amendment), 2005 **b)** Environmental Protection Act, 1996	Granted/Active	12-May-16	15-Sep-15		Draft renewal permit issued - awaiting finalisation. Renewal required every five(5) years. Awaiting final approval.
Aerodrome License	**70/2014**	Guyana Civil Aviation Authority (GCAA)	Civil Aviation(Air Navigation) Regulations, 2001	Granted/Active	08-Sep-14	30-Sep-16		**Aurora Aerodrome/Airstrip License** - Renewed annually - Renewal applied for and required inspection scheduled for Sept 29, 2015
Buckhall Port/Wharf Facility - Lease Land	**A-24199**	Guyana Lands and Surveys Commission (GLSC)		Granted/Active	08-Feb-13	NA		50 years Lease **for a portion of State Land used for the Buckhall Port Facility** - Renewed annually, by the 1st of January each year.
Buckhall Port/Wharf Facility Permit	**NA**	Maritime Administration Department (MARAD)		Granted	25-Feb-13	NA		Approval by Letter **to Construct/Operate a Wharf facilty at the Buckhall location**.
Sufferance Wharf (Buckhall)		Guyana Revenue Authority (GRA)	Customs Act chapter 82:01 (section 132)	Granted	18-Dec-14	NA	**As referenced in the GRA approval letter:** 1. Comply with the provisions of Section 132 of the Customs Act chapter 82:01 which states "the master of an aircraft or ship unloading goods at a sufferance wharf shall, if the Commissioner General so requires first enter the goods at the port at which the license has been issued and shall comply with the conditions of the license and such other condition as the Commissioner General may impose." 2. The company or the agent for the company must provide notice at least 24 hours prior to the, an estimated time of arrival and departure of the vessel. This information is to be provided to the GRA's Charity Boathouse, Essequibo for which clearance will be obtained for berthing or departure of vessel. 3. The captain or the agent	**Approval by Letter of request for permission to use the Buckhall Wharf as Sufferance Wharf -** This is an approval issued once to the applicant and requires no renewal.
Buckhall Port/Wharf Facility Extension - Lease Land	**A-24847**	Guyana Lands and Surveys Commission (GLSC)		Granted/Active	15-Apr-14	01-Jan-17		50 years Lease **for a portion of State Land used for the Buckhall Port Facility** - Renewed annually
Buckhall Port/Wharf Facility Foreshore License	**P-25199**	Guyana Lands and Surveys Commission (GLSC)		Granted/Active	08-Oct-14	07-Oct-16		**License Required to occupy the foreshore (Govt. reserve)** - Renewed annually - awaiting reissuance
Main Tapir East - Lease Land	**A-24572**	Guyana Lands and Surveys Commission (GLSC)		Granted/Active	29-Oct-13	01-Jan-17		50 years Lease **for a portion of State Land used for the Tapir Barge Crossing Facility** - Renewed annually
Main Tapir West - Lease Land	**A-24499**	Guyana Lands and Surveys Commission (GLSC)		Granted/Active	17-Sep-13	01-Jan-17		50 years Lease **for a portion of State Land used for the Tapir Barge Crossing Facility** - Renewed annually

Table 19.1: Continued

Licences/Permits/Lease	Permitting Agency		Applicable Legislation	Status	Date Granted	Expiry Date	Requirements/Actions	Comments/Remarks
Well Installation Permit		**a)** GWI **b)** Guy.EPA **c)** Hydro-met Dept (Min. of Agriculture)		Granted	29-Sep-15	NA	**As referenced in the Hydro-Met approval letter**: AGM Inc. w ill provide to the Hydro- met Dept the follow ing: 1. Monthly readings of the static and dynamic w ater levels measured from a fixed point. 2. A copy of the Water Quality results asthey are taken in your monitoring. **As referenced in the EPA approval letter:** Drilling and Well Construction 1. Notify the EPA in w riting of plans to change the facility or operation in any w ay, to allow for the timely approval from the Agency before implementation, including but not limited to the follow ing: Changes in construction, structure, or layout of the facility, plant or building; b. Installation of new and/or changes to equipment, machine, apparatus, mechanism, system or technology serving the facility	**Approval obtained for : a**) GWI - 2 Feb, 2015 **b**) Guy. EPA - 12 Feb, 2015 **c**) Hydro-met Dept (Min. of Agriculture) - Approved Sept 29th, 2015.
Bulk Fuel Storage License/Consumer Installation License	**CI/0348 of2014**	Guyana Energy Authority (GEA)	**a)** Guyana Energy Agency Act 1997 **b)** Guyana Energy Agency (Amendment) Act 2004	Granted/Active	09-Aug-15	08-Aug-35		Renew ed annually - Renew ed
Explosives - Use Permit	**NA**	Guyana Geology and Mines Commission (GGMC)	Blasting Operations Act (Law s of Guyana)	Granted/Active				This is covered under the Blasting Operations Act w hich states that someone, the holder of a Certificate of Competency granted by the GGMC to any person deemed by the commission after examination...conversant w ith the safe use and
Explosives - Import License	**5/2014 A**	Ministry of Home Affairs		Granted/Expired	23-Dec-16	16-Apr-15	License to import is requested w hen an explosives	Valid for six (6) months
Effluent Discharge Approval		Guyana Environmental Protection Agency (EPA)	**a)** Environmental Protection Act (Amendment), 2005 **b)** Environmental Protection Act, 1996		see Comments	NA	EPA Permit Sect 4.1 Adhere to the World Bank Water Quality Standards w ith regards to effluent discharged from the mine site during all operational	There is no specific requirement for an Effluent Discharge Permit; AGM Inc. is how ever required to perform sampling, and provide records and
Use of Cyanide Approval (Permit)		Guyana Geology and Mines Commission (GGMC)	**a)** Mining Act **b)** Environment Regulations, Mining (Amendment) Regulations	Granted	08-Apr-15	NA		Approval by Letter
Electricity Generation Permit (Certificate of Inspection) Aurora,	278238	**a**) Office of the Prime Minister, **b**) Government Electrical Inspectorate (GEI)	Electrical regulations 1907	*Granted*	04-May-15	NA		
Electricity Generation Permit (Certificate of Inspection)	278239	**a**) Office of the Prime Minister, **b**) Government Electrical Inspectorate (GEI)	Electrical regulations 1907	*Granted*	04-May-15	NA		
Electricity Generation Permit (Certificate of Inspection)	278240	**a**) Office of the Prime Minister, **b**) Government Electrical	Electrical regulations 1907	*Granted*	04-May-15	NA		
Electricity Generation Permit (Certificate of Inspection)	278241	**a**) Office of the Prime Minister, **b**) Government Electrical Inspectorate (GEI)	Electrical regulations 1907	*Granted*	04-May-15	NA		
Electricity Generation Permit (Certificate of Inspection)	278242	**a**) Office of the Prime Minister, **b**) Government Electrical Inspectorate (GEI)	Electrical regulations 1907	*Granted*	04-May-15	NA		
Electricity Generation Permit (Certificate of Inspection)	278243	**a**) Office of the Prime Minister, **b**) Government Electrical Inspectorate (GEI)	Electrical regulations 1907	*Granted*	04-May-15	NA		

Table 19.1: Continued

Licences/Permits/Lease	Permitting Agency		Applicable Legislation	Status	Date Granted	Expiry Date	Requirements/Actions	Comments/Remarks
Tower Installation - Buckhall		**a**) Guyana Civil Aviation Authority (**GCAA**), **b**) Guyana Lands and Surveys Commission (**GLSC**)		*Lease presently awaiting approval from the Office of*	*see Comments*	NA		Approval/No Objection by Letter (**GCAA**), 30-June-2014; **GLSC**, *on "Schedule" at OP*
Tower Installation - Repeater #1		**a**) Guyana Civil Aviation Authority (**GCAA**), **b**) Central Housing and Planning Authority (**CHPA**), **c**) Guyana Lands and Surveys Commission (**GLSC**)		*Lease presently awaiting approval from the Office of the President*	*see Comments*	NA		Approval/No Objection by Letter (**GCAA**), 30-June-2014; **CHPA**, 16-April-2015; **GLSC**, *on "Schedule" at OP*
Tower Installation - Repeater #2		**a**) Guyana Civil Aviation Authority (**GCAA**), **b**) Central Housing and Planning Authority (**CHPA**), **c**) Guyana Lands and Surveys Commission (**GLSC**)		*Lease presently awaiting approval from the Office of the President*	*see Comments*	NA		Approval/No Objection by Letter (**GCAA**), 30-June-2014; **CHPA**, 16-April-2015; **GLSC**, *on "Schedule" at OP*
Tower Installation - Repeater #3		**a**) Guyana Civil Aviation Authority (**GCAA**), **b**) Central Housing and Planning Authority (**CHPA**), **c**) Guyana Lands and Surveys Commission (**GLSC**)		*Lease presently awaiting approval from the Office of the President*	*see Comments*	NA		Approval/No Objection by Letter (**GCAA**), 30-June-2014; **CHPA**, 16-April-2015; **GLSC**, *on "Schedule" at OP*
Tower Installation - Aurora		**a**) Guyana Civil Aviation Authority (**GCAA**), **b**) Central Housing and Planning Authority (**CHPA**), **c**) Guyana Lands and Surveys Commission (**GLSC**)		*Lease presently awaiting approval from the Office of the President*	*see Comments*	NA		Approval/No Objection by Letter (**GCAA**), 30-June-2014; **CHPA**, 19-Dec-2014; **GLSC**, *on "Schedule" at OP*
Frequency Allocation Permit(License for the Installation and Operation of Radio Equipment)	**323/31/AU/2013**	National Frequency Management Unit (NFMU)	**a**) Guyana Post and Telegraph Act Chapter 47:01 **b**) Wireless Telegraphy Regulations	Granted/Active	13-Aug-15	14-Aug-16		**Schedule:** Fixed Stations, Repeater Stations, Mobile Stations and Handhelds
Radio Frequency License (Ground to Aircraft communication)	**325F/2/OT/2014**	National Frequency Management Unit (NFMU)	**a**) Guyana Post and Telegraph Act Chapter 47:01 **b**) Wireless Telegraphy Regulations	Granted/Active	21-May-15	21-May-17		Renewed Annually at the on or before the Anniversary
Export Licence - Gold	**121/2016E**	Ministry of Business Invesment, Guyana Gold Board		Granted	16-May-16	15-May-17		

19.3 Environmental and Social Studies and Impacts

The Aurora Gold Mine is located in a remote, forested, and largely uninhabited area of northwestern Guyana.

Guyana Goldfields has partnered with the International Finance Corporation (IFC) of the World Bank Group, involved with Aurora Gold Mine since 2006, providing technical, environmental and social assistance.

Guyana Goldfields, has committed to the establishment of environmental and social practices for the Aurora Gold Mine that comply with the legal requirements established by the nation of Guyana, applicable IFC Performance Standards (IFC, 2012) and IFC "General EHS Guidelines" and the "Environmental, Health, and Safety Guidance for Mining" (IFC, 2007). Compliance with applicable IFC Performance Standards is required by the IFC as well as other lenders to the Aurora Gold Mine.

In keeping with the requirements of IFC Performance Standard 1, "Assessment and Management of Environmental and Social Risks and Impacts" (PS-1), Guyana Goldfields has undertaken a wide range of environmental and social studies in the past years to assess the environmental and social impacts likely to be associated with the project. These studies have included three Environmental and Social Impact Assessments (ESIAs): Final Environmental and Social Impact Assessment Aurora Gold Mine, March 2010; Final Environmental and Social Impact assessment (ESIA) of the Aurora Gold Mine Project in Guyana, May 2010; and Final Updated Environmental and Social Impact Assessment, Aurora Gold Project, July 2013. The third ESIA to IFC standards was prepared in 2013 based on the updated feasibility study (Tetra Tech 2013).

Although small temporary or transient ASM encampments may occasionally be encountered in the project area, there are no formal or established communities or settlements in the immediate vicinity of the site, and only two informal communities or settlements (Aranka Mouth and Buckhall) are located anywhere near the associated project components. As a consequence, the project is not expected to generate the direct socio-economic effects. Most importantly, there were no temporary or permanent communities or residences within the project concession that would have required physical displacement or resettlement.

In addition to the aforementioned ESIA documents, other relevant studies conducted in the area of the project include:

- Rapid Biodiversity Assessment of the 'Golden Square Mile' Area of AGM Aurora Concession in Guyana (World Wildlife Fund (WWF) Guianas, 2006);
- Final Report - Environmental and Social Baseline Aurora Mining Concession for AGM (Ground Structures Engineering Consultants, 2007);
- IFC Public Health Technical Assistance Program for Guyana Goldfields, Phase 1 (Newfields, 2008);

- IFC Public Health Technical Assistance Program for Guyana Goldfields, Phase 2 (Newfields, 2009); and
- Bio-assessment of the Cuyuni River near Aurora: Environmental and Economic Implications (Dr. Nicole Duplaix, October, 2009).

These studies document field work undertaken since 2006 by several teams of national and international biologists, ecologists, and social scientists. Taken collectively, these studies confirm that the environment associated with the Aurora Gold Mine's AOI has been significantly impacted by artisanal and small-scale mining (ASM), logging, hunting, and other human activities for well over a century. The Cuyuni River has likely served as a transportation corridor since the prehistoric arrival of the first indigenous peoples in the region. The immediate area of the Aurora Gold Mine site had modern exploration in the 1930s, modern mining in the 1940s, and has been the site of several exploration programs since that time. Apart from supporting a major logging concession (operated by Barama Company Limited), the construction of the Barama Road has contributed to a significant increase in human activities in the region to the north of the Cuyuni River and to the west of the Essequibo River.

It is noteworthy that the large species of fauna that are otherwise common in pristine habitats along similar types of rivers in this part of South America have not been observed or are rare in the project AOI. The absence of such species is a key indicator of historical human impact, presumably due to the pressures of hunting and the increased turbidity and degradation of river quality from many years of logging and ASM activities far upstream in Venezuela as well as in Guyana, and from the regular boat traffic on the river.

The results of these studies and other ongoing field investigations have contributed to the understanding of the current environmental setting of the Aurora Gold Mine, which is summarized in the following paragraphs.

Climate: the overall climate is tropical with two wet (April-August, and December-February) and two dry (August-December, and February-April) seasons. Based on project site weather station data (which have been available since 2006), the average temperature is about 26° C (79° F), with yearly precipitation averaging approximately 2,124 mm, with major inter-annual variations in rainfall attributable to the "El Niño/La Niña" cycle.

Air quality: there are no developed areas with significant fixed or mobile sources of atmospheric emissions in the area surrounding the project concession. Gaseous emissions in the project area are mostly attributable to rotting vegetation. Some combustion emissions are also related to the operation of the Barama logging operations, as well as ASM dredges, drill rigs, chainsaws, generators, boats, and other motorized equipment. Due to the high humidity and significant level of rainfall, fires are rare in the region, and dust levels on the Aurora Gold Mine roadways, trails, or clearance areas are typically low except on locally heavily trafficked sections of roadway in dry conditions.

Surface water flow and quality: the Aurora Gold Mine and surrounding area are drained by the Cuyuni River, which borders the project Area on the north, along with several associated tributaries, including the Gold River located to the east of the project Area. The Cuyuni River originates in Venezuela and extends some 750 km east to the Essequibo River in Guyana, covering an area of approximately 53,500 km^2 (AMEC, 2009).

Surface water quality in the Cuyuni River and its tributaries has, in general, been impacted historically by upstream ASM activities, particularly in Venezuela, and ASM-associated mercury contamination of surface water, sediments, and fish has been documented as a concern throughout the Guianas. ASM activities have also notably increased surface water turbidity and concentrations of suspended solids in the project AOI. In 2006 and 2007, surface water samples were collected from locations on the Cuyuni River, Gold River, and from an unnamed tributary of the Cuyuni River, both upstream and downstream of the Aurora Gold Mine area. Surface water samples from the unnamed tributary may be considered indicative of background water quality for the many smaller creeks that crisscross the Aurora Gold Mine site. During the 2006-2007 sampling activity, it was observed that total iron was the only parameter that exceeded the guidelines defined in the IFC “Environmental Health and Safety Standard Guidelines for Mining” (IFC, 2007). In 2009, additional surface water and sediment sampling was conducted along the Cuyuni River to assess baseline surface water and sediment quality conditions. Total suspended solids, iron, and oil and grease were detected in surface water samples at concentrations above (IFC, 2007) guidelines. The detection of oil and grease at elevated levels in one of the samples may be attributed to residual petroleum hydrocarbon impacts from previous gold exploration and drilling operations near the Aurora Gold Mine site. In 2011, surface water samples were also recovered from five locations around the site. Sediment samples were also recovered from stream beds at points coincident with surface water sampling locations. Both surface water and sediment samples were tested, and none of the parameters sampled exceeded (IFC, 2007) standards.

Groundwater flow and quality: based on groundwater monitoring conducted by AMEC in 2006 and 2007, shallow groundwater exists within the unconsolidated overburden from approximately 1 to 4 m below grade. In 2011, a well installation program was initiated to fill the data gaps required to better define the regional groundwater flow regime. Results of these investigations indicate that, in the saprolite zone at the site, groundwater flow is primarily northeast and northwest towards the Cuyuni River. The steep gradient in the area adjacent to the Cuyuni River indicates that flow in this area may be highly dependent on (and influenced by) river water levels. Flow in the bedrock zone appears to replicate the saprolite zone flow, primarily to the northeast and northwest towards the Cuyuni River. In 2006 and 2007, quarterly sampling was conducted to assess groundwater conditions prior to the commencement of mining operations, as well as potential seasonal variations in groundwater quality. In 2009, additional groundwater sampling was conducted to assess groundwater quality conditions at locations in the specific vicinity of the proposed mining operations. All samples were unfiltered and were analyzed for total metals and a range of other constituents. For purposes of comparison, none of the parameters measured in any these samples exceeded (IFC, 2007) effluent requirements for mining operations. In the 2011 well installation program, groundwater samples were recovered from selected wells screened in the saprolite zone to develop additional data on groundwater quality and flows on a regional scale. These samples are considered to be representative of groundwater quality at the site since there is an upward gradient from the bedrock into the saprolite at shallow depths. None of the parameters measured in these samples exceeded (IFC, 2007) effluent requirements for mining operations.

Soils: the native soils of the proposed mine site, along the access road, and at the Buckhall Port generally consist of residual material derived from weathered alkalic crystalline rocks (i.e., granite, schist, dolerite, granodiorite, and phyllite), and alluvial sediments derived from stratified and unconsolidated deposits of sand, silts, and clays. In upland areas, soils consist of deep, well-drained, yellow and reddish-brown sandy clay loams and gravelly clays. In riverine and alluvial fan areas, soils are also deep but range from poorly- to excessively well-drained clay loams and sands (Braun and Derting, 1964).

Geochemistry: a testing program was conducted for AGM in 2011 and 2012 by Klohn Crippen Berger (KCB), in order to determine the potential acid rock drainage (ARD)/mineral leaching potential of representative samples of predicted Aurora Gold Mine overburden and waste rock material. Results of static and kinetic humidity cell testing indicate that the majority of the samples tested have very low ARD/metals leaching potential. These results will be used to develop a geochemical block model that identifies areas that are potentially acid generating (PAG), acid generating (AG), and non-acid generating (NAG). This model and associated testing data have played a key role in the development of the project's water quality management strategy, which is discussed in Section 20.3.4. Geochemical data will also be used to adjust estimates of leachate loads to support water quality modeling and the ongoing management of the Mine Water Pond (MWP), Fresh Water Pond (FWP), Tailings Management Area (TMA), and TMA Diversion Ponds 1 and 2, with respect to maintaining effluent/seepage water quality within IFC and International Cyanide Management Code (ICMC) guidelines.

Flora: the Aurora Gold Mine concession, the Buckhall Port, and access road corridors are located in the Mixed Forest Northeast District per the Vegetation Map of Guyana (per Steege 2001), in Region 10. Historically, this region was predominantly forested, but since colonial times, logging and ASM activities have contributed to some deforestation, though the area remains largely forested. Canopy trees are the dominant plant strata, followed by mid and lower story trees, plants. None of the plant species identified are locally endemic, of significance to local communities, or listed as threatened by the International Union for Conservation of Nature and Natural Resources (IUCN) "Red List" or equivalent national or regional assessment.

Terrestrial Fauna: 52 species of amphibians were reported by the 2006, 2007, 2009, and 2011 surveys. All of the species encountered were common to the wider geographic area of Guyana and many are found across the entire Guianan-Amazonian lowlands region. None of the species reported are categorized as endangered or threatened, and only one species is endemic to Guyana. A total of 308 bird species were reported from all combined (2006 through 2011) surveys. Of these species, the only one with an IUCN category other than "least concern" was the harpy eagle; one adult and one juvenile were reported by the 2009 survey but have not been seen since, presumably due to the absence or low numbers of primary prey species (e.g., capuchin and howler monkeys).

Evidence suggests that the presence of small-scale mining, logging, hunting, and other human activities in the project AOI has significantly reduced the natural abundance of large mammals, and has also likely led to the retreat of more sensitive mammalian species due to noise and other human activity. A total of 37 mammalian species were reported by the 2006, 2007, and 2009 surveys. While the 2006/2007 surveys reported four or five species of primates, the 2009 surveys indicated only low numbers of howler and capuchin monkeys. By way of comparison, in undisturbed regions of the Guianas with similar habitats, primates are generally quite abundant.

Aquatic Fauna: a total of 77 fish species were reported from the 2006, 2007, 2009, and 2011 surveys. None of the species that have been identified to date are known to be endemic to Guyana or the Cuyuni river basin. Sampled tributaries were noted to be heavily contaminated from tailings runoff and hydrocarbons from ASM operations further upstream, which resulted in significant sedimentation and likely degraded water quality.

Giving due consideration to the project's environmental setting, the most significant potential environmental (and environmentally-linked social) impacts predicted for the development of the project are summarized in , along with a description of the general strategies that are employed to manage and mitigate such impacts over the life of the project.

Table 19.2: Potential environmental and social impacts and associated management and mitigation strategies

Project Component	Potential Impact	Management/Mitigation Strategies
Mine and mill/ process facility area	Impacts to soils	· Implementation of the Project *Erosion Prevention and Control Plan*; management of drainage and stormwater runoff on haul roads, maintenance of topsoil/organic matter stockpiles, and grading, scarifying, and re-vegetation of road rights-of-way (ROWs), exploratory roads, and other temporarily cleared areas · Implementation of appropriate progressive restoration and erosional stabilization procedures for disturbed areas per the Project *Erosion Prevention and Control Plan* and *Mine Reclamation and Closure Plan* · Implementation of the *Spill Prevention, Control, and Contingency Plan*
	Sediment loading of surface water	· Installation of diversion structures to route un-impacted surface water around mining operations, and to route impacted surface water to the MWP, FWP, and TMA · Construction of sedimentation ponds and implementation of other best management practices (BMPs) in the *Erosion Protection and Control Plan* and *Water Management Plan* to manage stockpile sediment runoff and to detect and mitigate erosion in other disturbed areas · Implementation of progressive restoration and erosional stabilization procedures for areas historically disturbed by ASM, per the *Erosion Prevention and Control Plan* and *Mine Reclamation and Closure Plan*
Mine and mill/ process facility area	Loss of aquatic habitats	· Installation of diversion structures to route un-impacted surface water around mining operations, and to route all impacted water to the Mine Water Pond (MWP), Fresh Water Pond (FWP), and Tailings Management Area (TMA) · Implementation of BMPs in the *Erosion Protection and Control Plan* and *Water Management Plan* to manage topsoil/overburden stockpiles; and to detect and mitigate erosion in other disturbed areas · Implementation of appropriate progressive restoration and erosional stabilization procedures for mined-out areas of the open pits, as well as areas disturbed by ASM, per the *Erosion Prevention and Control Plan* and *Mine Reclamation and Closure Plan* · Implementation of a routine water quality monitoring program in the Cuyuni River and its tributaries as described in the *Water Management Plan* and *ESHS Monitoring Plan*
	Loss of terrestrial habitat and flora	· Minimization of clearance actions/project footprint per the *Erosion Protection and Control Plan, Exploration Management Plan, Early Works Construction Management Plan,* and *Construction Management Plan* · Implementation of a routine biodiversity monitoring program per the *Biodiversity Management Plan* and *ESHS Monitoring Plan* · Implementation of specific mitigation measures for the protection of any identified sensitive species and habitats, per the *Biodiversity Management Plan* · Implementation of BMPs in the *Erosion Protection and Control Plan* and *Water Management Plan* to manage sediment generation from waste rock/topsoil stockpiles; and to detect and mitigate erosional conditions in other disturbed areas. · Implementation of appropriate progressive restoration and erosional stabilization procedures for mined-out areas of the open pits, as well as ASM-disturbed areas, per the *Erosion Prevention and Control Plan* and *Mine Reclamation and Closure Plan*

Table 19.2 Continued

Project Component	Potential Impact	Management/Mitigation Strategies
Mine and mill/ process facility area	Discharges of cyanide and other hazardous chemicals to surface water	· Purchase of sodium cyanide reagent exclusively in solid briquette form, transported to the Project site in dedicated stainless steel ISO delivery/mixing tanks · Sodium cyanide will be transported by truck; delivered in dry briquette form in dedicated stainless steel ISO tanks · Implementation of the secondary containment, engineered spill prevention and control measures, and other BMPs defined by the Project *Cyanide Management Plan,* the Aurora site *Spill Prevention, Control and Contingency Plan,* and the *Emergency Preparedness and Response Plan* · For non-cyanide reagents and fuel, implementation of the secondary containment and engineered spill prevention and control measures, remote fuelling control procedures, and other BMPs defined by the *Hazardous Material Management Plan*, *Buckhall Spill Contingency Plan,* the Aurora site *Spill Prevention, Control and Contingency Plan,* and the *Emergency Preparedness and Response Plan* · *Adherence to the Cyanide Code*
	Slope failure of waste rock and Saprolite stockpiles, disrupting surface flows	· Operator training programs/ compliance with *Overburden Management Plan* · Periodic monitoring of the physical integrity of the waste rock and Saprolite stockpiles, in accordance with the *Overburden Management Plan* and the *ESHS Monitoring Plan,* and regrading and/or strengthening of earthworks or other action as indicated by observed conditions
	Modification of hydrologic flow patterns within streams/creeks due to FWP, MWP, TMA and surface/ underground mining operations	· Installation of diversionary structures/diversion of un-impacted surface water around mining and processing operations in order to maintain biological base flows in Cuyuni River tributaries · Control of discharges from the FWP, MWP, and TMA into tributaries of the Cuyuni in accordance with the *Water Management Plan* and *Tailings Area Management Plan,* in compliance with effluent discharge guidelines and/water quality standards defined by Guyana EPA, (IFC, 2007), and the ICMC
Mine and mill/ process facility area	Breaches and overtopping of the FWP, MWP, and/or TMA	· Provision of sufficient freeboard in the design of the MWP, FWP, and TMA based on the Probable Maximum Precipitation (PMP) event · Inclusion of sufficient contingency in the design of the MWP, FWP, and TMA embankments to withstand PMP events plus an appropriate safety factor · Inclusion of a series of redundant water management features (e.g., spillways, diversion ponds) in the TMA design · Rigorous independent Construction Quality Assurance (CQA) oversight of MWP, FWP, and TMA embankment construction · Development and implementation of probabilistic water balance/monitoring program and other BMPs for the MWP, FWP, and TMA, in accordance with the *Tailings Management Plan, Water Management Plan*, and *Cyanide Management Plan* · Adherence to the Cyanide Code
	Potential runoff of impacted stormwater into surface water	· Diversion of un-impacted surface water around mining and processing operations · Installation of skimmers/oily water separators on mine stormwater discharges, with discharge reporting to MWP for settling/dilution and testing prior to any discharge · Collection (and as necessary treatment) of leachate from onsite solid waste disposal facility prior to any potential controlled discharge · Collection of runoff from mining and process facility processing areas, stockpiles, and roadways, and retention in settling ponds and the MWP (or dedicated settling ponds at individual waste rock stockpiles) for settling and dilution prior to any potential controlled discharge · Regular monitoring of MWP water quality in accordance with the *Water Management Plan* and *ESMS Monitoring Plan* to ensure that controlled discharges meet quality standards defined by the Guyana EPA and (IFC, 2007) · Implementation of the Project *Erosion Prevention and Control Plan*; management of drainage and stormwater runoff on haul roads to minimize sedimentation; maintenance of topsoil/organic matter stockpiles; and grading, scarifying, and re-vegetation of road ROWs, exploratory roads, and other temporarily cleared areas

Table 19.2 Continued

Project Component	Potential Impact	Management/Mitigation Strategies
Mine and mill/ process facility area	Potential runoff or seepage of contaminated water from the TMA into surface water	· Installation of barge and pumpback systems to return TMA reclaim water back to the process facility for industrial use · Installation of embankment seepage interception, collection, and return systems on TMA, as described in Section 20.3. · Construction of TMA in a low-permeability saprolite soil basin, supported by geological evaluation of the basin and local compaction, grouting, or other basin preparation actions during construction as necessary to ensure consistent low-permeability conditions, as described in Section 20.3. · Inclusion of a series of redundant water management features (e.g., spillways, diversion ponds) in the TMA design. · Implementation of probabilistic water balance/water monitoring program for the TMA in the operational phase, in accordance with the *Tailings Management Plan, Water Management Plan*, and *Cyanide Management Plan,* as discussed in Section 20.3. · Regular monitoring of TMA water quality in accordance with the *Tailings Facility Management Plan, Cyanide Management Plan,* and *ESMS Monitoring Plan* to ensure that controlled discharges will be within Guyana EPA and (IFC, 2007) limits, as well as the free cyanide limits recommended by the ICMC for protection of aquatic life
	Diesel oil spill into the Cuyuni River	· Implementation of the secondary containment and engineered spill prevention and control measures, remote fuelling control procedures, oily water separators/treatment systems, and other BMPs per the *Hazardous Material Management Plan* and *Spill Prevention, Control and Contingency Plan* · Implementation of the Project's preventive maintenance (PM) and field inspection programs for the operation of the fuel farm, emergency generator area, and fuelling station · Installation of skimmers/ oily water separators on mine wastewater discharges, reporting to the MWP for settling and dilution prior to any controlled discharge · Regular monitoring of MWP water quality in accordance with the *Water Management Plan* and *ESMS Monitoring Plan* to ensure that controlled discharges will be within the effluent discharge guidelines and/water quality standards defined by the Guyana EPA and (IFC, 2007)
Mine and mill/ process facility area	Cyanide spill into the Cuyuni River or its tributaries	· Purchase of sodium cyanide reagent exclusively in solid briquette form, transported to the Project site in dedicated stainless steel ISO delivery/mixing tanks · Restrict direct delivery of cyanide to the Aurora site using boats or barges on the Cuyuni and Essequibo Rivers · Implementation of the secondary containment, engineered spill prevention and control measures, and other BMPs defined by the Project *Cyanide Management Plan,* and *Emergency Preparedness and Response Plan* · Implementation of operational practices in the process facility that minimize the potential for process upsets, as noted in the Project *Cyanide Management Plan* · Implementation of the Project's PM and field inspection program for the operation of the process facility · Adherence to the Cyanide Code
	Infiltration of potential spills or discharges of cyanide and other chemicals into groundwater	· Purchase of cyanide exclusively in solid briquette form, transported in dedicated stainless steel ISO delivery/mixing tanks · Implementation of the secondary containment, engineered spill prevention and control measures, and other BMPs defined by the Project *Cyanide Management Plan* and *Emergency Preparedness and Response Plan* · Implementation of operational practices in the process facility that minimize the potential for process upsets, as noted in the Project *Cyanide Management Plan* · For other (non-cyanide) reagents and fuel, implementation of the secondary containment and engineered spill prevention and control measures, remote fuelling control procedures, oily water separators/treatment systems, and other BMPs defined by the *Hazardous Material Management Plan*, *Buckhall Spill Contingency Plan,* the Aurora site *Spill Prevention, Control and Contingency Plan,* and the *Emergency Preparedness and Response Plan*

Table 19.2 Continued

Project Component	Potential Impact	Management/Mitigation Strategies
Mine and mill/ process facility area	Potential infiltration of surface water from the Cuyuni River into open pit/ underground mine	· Construction of the Cuyuni River dike system, including seepage collection wells will be installed
	Potential runoff or seepage of leachate from the Solid Waste Landfill	· Collection and periodic testing of leachate from landfill; if testing results indicate effluent quality issues with respect to Guyana EPA or (ICMC, 2007) guidelines, route to MWP for dilution and storage or install local water treatment system
	Potential failure of the TMA and MWP dams after completion of mining activities	· Stabilization, breaching/removal of embankments, closure, and selective re-vegetation of tailings surfaces and embankments per the final *Detailed Mine Reclamation and Closure Plan* · Post-closure monitoring of the stability of the reclaimed MWP, FWP, and TMA areas and periodic monitoring of downgradient surface and groundwater conditions per the final *Mine Reclamation and Closure Plan* and *ESHS Monitoring Plan*
Buckhall-Aurora Access Road	Impacts to soils	· Implementation of the Project *Erosion Prevention and Control Plan*, including maintenance of topsoil/ organic matter stockpiles, and grading, scarifying, and re-vegetation of ROWs and other temporarily cleared areas adjacent to the roadway · Implementation of the *Spill Prevention, Control and Contingency Plan* and *Buckhall Spill Contingency Plan,* placement of spill kits in heavy equipment /passenger vehicles
	Impacts from improper culvert installation at stream crossings	· Design culverts to properly handle natural surface water flows, per the Project *Erosion Prevention and Control Plan;* correct grading, lined with grass and/or riprap, where necessary, to control flow rates/velocities and minimize erosion
Buckhall-Aurora Access Road	Loss/ degradation of aquatic habitats	· Implementation of BMPs in the *Erosion Protection and Control Plan* and *Water Management Plan* to detect and mitigate areas of soil erosion, manage stormwater runoff, and control sedimentation on access road ROWs and other adjacent disturbed areas · Implementation of progressive restoration and erosional stabilization procedures for any excessively wide ROW areas, as well as any ASM-disturbed areas, per the *Mine Reclamation and Closure Plan* · Implementation of routine water quality monitoring program at stream crossings per the *ESHS Monitoring Plan*
	Loss/alteration of terrestrial habitats	· Minimization of clearance actions/project footprint per BMPs in the *Exploration Management Plan, Early Works Construction Management Plan,* and *Construction Management Plan* · Implementation of a routine biodiversity monitoring program within the Aurora concession per the *Biodiversity Management Plan* and *ESHS Monitoring Plan* · Implementation of progressive restoration and stabilization procedures for ROWs and ASM-disturbed areas, per the *Exploration Management Plan, Early Works Construction Management Plan, Construction Management Plan,* and *Mine Reclamation and Closure Plan* · Implementation of specific mitigation measures for the protection of sensitive species and habitats, as directed by the *Biodiversity Management Plan* · Implementation of BMPs in the *Erosion Protection and Control Plan* and *Water Management* Plan to manage stockpiles, and to detect and mitigate erosional issues in other disturbed areas
Buckhall-Aurora Access Road	Impacts to Soils	· Implementation of the Project *Erosion Prevention and Control Plan*, including maintenance of stockpiles, and -re-vegetation of ROWs and other cleared areas adjacent to the road · Implementation of the *Spill Prevention, Control and Contingency Plan* and *Buckhall Spill Contingency Plan;* placement of spill kits on all heavy equipment and passenger vehicles, and installation of emergency response stations at select locations along the access road · Inclusion of appropriate contractual requirements for spill prevention and control and maintenance of spill kits for all delivery contracts
	Spills of fuel and other chemicals during transport between the Aurora site and Buckhall	· Establishment of contractual requirements for the trucking companies transporting other hazardous chemicals to have spill contingency plans and spill kits · Establish emergency response stations at selected locations on access road

Table 19.2 Continued

Project Component	Potential Impact	Management/Mitigation Strategies
Buckhall Port	Impacts to soils	· Implementation of the *Erosion Prevention and Control Plan*, including maintenance of stockpiles and re-vegetation of cleared areas in or near Buckhall compound · Implementation of remote fuelling control procedures and other BMPs per the *Hazardous Material Management Plan* and *Buckhall Spill Contingency Plan* · Placement of spill kits in all heavy vehicles operating within the Buckhall compound
	Potential runoff or seepage of leachate from the Solid Waste Landfill	· Collection and testing of leachate from landfill; if test result so indicate, installation of treatment system prior to controlled discharge
Buckhall Port	Spills of fuel and other chemicals being loaded/ unloaded at the Port	· Implementation of the remote fuelling control procedures and other BMPs per the *Hazardous Material Management Plan* and *Buckhall Spill Contingency Plan* · Spill kits for all heavy vehicles operating within the Buckhall compound · Use of portable floating booms and spill response kits around the Buckhall fuelling terminal and implementation of the secondary containment and other BMPs at the fuel storage facility, per the *Hazardous Material Management Plan* and *Buckhall Spill Contingency Plan*
	Accidental discharges of fuels, oils and grease from equipment and/or from the failure of fuel containment facilities	· Use of spill response kits and implementation of the other BMPs applicable to the fuel storage facility, per the *Hazardous Material Management Plan* and *Buckhall Spill Contingency Plan*
Overall macro-scale impacts of the project	National socio-economic impacts due to closure	· Implement the Project *Community Relations Plan*, *Influx Management Plan*, and final *Detailed Mine Reclamation and Closure Plan;* key actions will include: - diversification of skills/training, and building capacities of former workers and service providers to find economic opportunities with other industries in Guyana; and - Amerindian communities for income generation projects · Implement selected public/community development projects
Overall macro-scale impacts of the project	Potential influx into site, leading to health, security and conflict related risks	· Hiring advertised and controlled through Georgetown office · Control of employee travel to and from site using GGI or GGI contractor vehicles · Prohibition of economic or social contacts by workforce or contractors with transients or local communities · Prohibition of public (transient) access to the mine, process facility, mechanical shops, fuel storage areas, TMA/MWP/FWP areas, man-camp, solid waste landfill, water supply system, airstrip, explosives silos and magazine, and other sensitive locations · Partner with Barama and the Government to monitor and manage influx issues per the *Influx Management Plan* and *Community Relations Management Plan* · Consult with the Government on issues such as promotion of planned regional development · Promote and implement health awareness and disease prevention campaigns among workers and contractors, as well as local communities upstream and downstream of project site
	Potential ASM issues	· Continued consultation and engagement with illegal and artisanal miners in accordance with the *Influx Management Plan* and *Community Relations Management Plan.*

Table 19.2 Continued

Project Component	Potential Impact	Management/Mitigation Strategies
Overall macro-scale impacts of the project	Potential human influx to the mine site and related risks	· Train the security forces on site to monitor potential influx, and to handle influx without creating conflict or security issues, in accordance with the *Influx Management Plan* and applicable sections of the *Community Relations Management Plan* · Partner with the Government and/or Barama to plan for the sustainable development/ growth of the Buckhall community, per the *Influx Management Plan* and applicable sections of the *Community Relations Management Plan* · Consultation with the Government on promotion of planned regional development · Prohibition of onsite hiring. Work opportunities advertised and controlled through GGI's Georgetown office · Control of employee travel to and from site using GGI or GGI contractor vehicles · Prohibit public access to the Solid Waste Landfills · Promote and implement health awareness and disease prevention campaigns, especially for malaria suppression and human immunodeficiency virus/acquired immunodeficiency syndrome (HIV/AIDS) and sexually transmitted diseases (STDs) among workers and contractors, as well as local communities upstream and downstream of the project site
Overall macro-scale impacts of the project	Potential influx into Buckhall due to Port development and related risks	· Train the security forces on site to monitor potential influx, and to handle influx without creating conflict or security issues, in accordance with the *Influx Management Plan* and *Community Relations Management Plan* · Partner with the Government and/or Barama to plan sustainable development/ growth of the Buckhall community, per *Influx Management Plan* requirements. · Consultation with the Government on promotion of planned regional development · Hiring advertised and controlled through Georgetown office · Control of employee travel to and from site using GGI vehicles · Prohibit public access to the Municipal Solid Waste Landfill

The impact management and mitigation measures suggested in are being implemented within the context of a project-specific ESHS Management System designed to comply with IFC PS-1 ("Environmental and Social Assessment and Management System") requirements, and as such be subject to specific change management controls to ensure that mitigation measures are appropriately updated in response to changing environmental considerations, facility modifications or expansions, process changes, or other changes that may occur over the life of the mine.

19.4 Environmental Management

AGM's Environmental, Safety, Health, and Social Management System (ESHS MS) has been developed to support effective ESHS performance associated with its mining activity, while also contributing to overall sustainability. Achieving a balance between environmental, social and economic sub-systems within the global system is considered essential in order to meet the needs of the present without compromising the needs of future generations.

AGM's ESHS Management System has been developed in alignment with requirements of ISO 14001:2015 and OHSAS 18001:2007. These two internationally recognized standards have been implemented by thousands of organizations world-wide and are a best practice approach for management system development. AGM's ESHS MS was also developed to align with the IFC Performance Standards and applicable EHS guidelines.

AGM's ESHS Standards apply to all ESHS activities of AGM, including the Tapir Crossing, Buckhall Port Facility and Georgetown Office.

By taking a "systems" approach, AGM integrates environmental, safety, health and social management disciplines into all of the company's business and management processes and activities.

The AGM ESHS MS consists of Standards, Management Plans or Programs, and Operational Procedures. Key management plans and operational procedures include:

- Overburden, Waste Rock and Ore Stockpile Management Plan will lay out management procedures including requirements for weekly inspections of the geotechnical stability of waste rock/overburden stockpiles, as well as pH in toe drain water in order to detect the potential development of ARD conditions. The Plan will also contain a section that addresses the stockpiling and management of growth media and mulch to support concurrent and final site reclamation activities.
- Tailings Management Area and Water Management Facilities – Operation, Maintenance, and Surveillance Manual (OMSM) defines management and monitoring procedures including weekly, monthly, quarterly, and yearly inspections of the integrity of the tailings and reclaim water return pipelines, as well as the geotechnical stability of all embankments, dams, diversionary and/or spillway structures, and other related earthworks. The goal of operations of the tailings and water management facilities is to construct and operate in such a manner that all structures are stable, all solids and water are managed within the designated areas, and all aspects of tailings and water management are in compliance with regulatory requirements and in conformance with sound engineering practice, company standards, the IFC Environmental, Health, and Safety Guidelines, and commitments to Communities of Interest.
- Reclaim water quality will also be monitored on a routine basis, as well as seepage, and controlled discharge water quality as previously noted. The purpose of such monitoring is to ensure that any seepage or routine discharge from the TMA is compliant with guidelines set

forth by the International Finance Corporation (IFC) for mining and for landfills, the World Health Organization (WHO) drinking water standards, Guyana Environmental Protection Agency (Guy-EPA), and the United States Environmental Protection Agency (USEPA). Any direct discharges from the TMA at closure will also be subject to the same guidelines. Indications of negative water quality trends in seepage or discharge may prompt detoxification process changes, the installation or activation of additional settling/ clarification ponds, treatment and/or polishing systems, or additional biotoxicity testing studies to demonstrate that aquatic species in any receiving waterways are adequately protected.

- Erosion Prevention and Control Management Plan will be applied to all areas that are disturbed by mining operations and that are susceptible to erosion. This will include revegetation and runoff control procedures and measures. activation of additional clarification ponds, treatment and/or polishing systems, or other studies to demonstrate that aquatic species in any receiving waterways are adequately protected.

- Cyanide Management Plan will be based on the ICMC standards of practice that addresses all aspects of cyanide procurement, transportation, and management. In the operational phase of the project, the Cyanide Management Plan will reference applicable operations and maintenance manuals and other operational procedures and inspection and preventive maintenance protocols that will apply to the monitoring of physical integrity and routine operation of: the cyanide receiving/mixing area; cyanide storage and CIL tanks and secondary containment structures; all process and detoxification process facility areas containing cyanide solution >0.5 mg/L CNWAD; and the tailings and reclaim pipelines. The Cyanide Management Plan will cross-reference the Tailings Area Management Plan for the routine physical inspection of the integrity of all TMA earthworks and the proper function of the seepage collection system, reclaim water barge, and reclaim water pumpback system. It will also cross-reference procedures in the Water Management Plan for the ongoing monitoring of water quality and water levels in the TMA reclaim pond, as well as downgradient monitoring well and surface water sampling against the 0.5 mg/L CNWAD limits set by IFC (2007) and the <0.022 mg/L free cyanide limit for protection of aquatic life established by ICMC. The Cyanide Management Plan will also include procedures for monitoring and reporting any wildlife mortalities, the condition of emergency eyewashes/showers and cyanide signage, antidote kit storage temperatures, workforce personal protective equipment (PPE) adequacy and condition, and other specific monitoring issues typical of an ICMC-compliant facility. The Cyanide Code will be strictly followed.

- Hazardous Material Operational Procedure defines handling, storage, disposal and monitoring procedures including weekly inspections of containment integrity, management of accumulated precipitation, segregation of incompatible products and materials, the condition of warning or access control signage, and the functionality or readiness of PPE and fire suppression and other emergency systems.

- Spill Prevention, Control, and Contingency Plan defines preventative and control measures and containment requirements for weekly inspections of containment integrity, management of accumulated precipitation, and the functionality or readiness of PPE, fire suppression systems or equipment, and other emergency systems.

- Air Quality and Noise Management Plan describes procedures for monitoring and controlling air quality. Given the areas lack of anthopogenic emission sources at the remote Aurora site, focus will be on diesel emissions and roadway dust generation during the dry season. Noise monitoring of routine blast operations will provide data for the Mine Manager for improving the efficiency of blasting operations. Ambient workplace noise will be monitored near drilling and loading operations, heavy vehicles, the mill/process facility, power plant, and other noise-generating areas of operation, as the basis for designation of hearing protection zones for.

- Waste Management Plan defines waste handling, accumulation and disposal of non-hazardous, hazardous waste, and medical waste, as well as daily cover of the landfill cells constructed at the Aurora Gold Mine and Buckhall sites. Combustible non-hazardous and hazardous wastes will be incinerated in high temperature incinerators and the ash disposed in separate non-hazardous and hazardous cells with leachate collection. The hazardous waste cell will be lined with impermeable liner. Non-combustible hazardous waste will be stored in containers until it can be transported to Georgetown for recycling. Waste oil will be recycled. Landfill leachate will be monitored and if the leachate does not meet effluent standards it will be incinerated.

- Occupational Health and Safety the ESHS MS includes multiple levels of safety and health measures among Standards, Management Plans, Operational Procedures, Safe Work Plans and extensive training on same. These include: Sustainable Development Policy; Risk Assessment and Control; Job Safety Analysis; Field Level Risk Assessment; ESHS MS awareness; Competence and Awareness; Operational Planning and Control; Fatal Risk Control (Light Vehicles, Heavy Vehicles, Working at Heights, Lifting); Traffic Management; Emergency Preparedness and Response; Crisis Communication; Incident Investigation; Root Cause Analysis; Non Conformity and Corrective action; Spill Prevention, Control and Contingency Plan; Internal Audit, and External Audit.

- Community Relations Management Plan, Influx Management Plan, and a general Grievance Procedure and separate Labor Grievance Plan constitute the main social management plans. Given the remote Aurora site, and lack of any population centers in the area, AGM’s community relations program is largely proactive and regionally focused.

- Final Mine Reclamation, and Closure Plan will be prepared two years before final closure and submitted to Guyana EPA for review and approval. The Mine Reclamation and Closure Plan will include specific requirements for monitoring pit lake filling, as well as the completeness of required demolition or removal actions and the overall effectiveness of the reclamation and revegetation of the reclaimed overburden stockpiles, solid waste landfill areas, the TMA, the MWP and FWP, the Cuyuni dike areas, roadway ROWs, and other reclaimed areas of the project. Specific erosional issues associate with closed areas will be monitored (and if necessary, remediated) in compliance with the Erosion Prevention and Control Management Plan, which will be incorporated by reference. During operations, AGM will periodically update the existing conceptual Mine Reclamation and Closure Plan (AGM ESHS MP 512 Mine Reclamation and Closure Plan).

19.5 Mine Closure

19.5.1 General Description of Mine Reclamation and Closure Plan

AGM has developed a conceptual Mine Reclamation and Closure Plan for the mine that will be issued and implemented within the context of the Aurora ESMS and periodically updated over the life of the mine. The Mine Reclamation and Closure Plan addresses concurrent, interim, and final closure actions; specific actions to minimize the attractiveness of the closed site for any future ASM incursions; workforce retrenchment considerations; and post-closure inspection and monitoring. As previously noted, two years before final closure, a detailed version of the Mine Reclamation and Closure Plan will be prepared submitted to Guyana EPA for review and approval.

Ownership of the airstrip, the road between the site and the end of the Barama Road, the ferry landings at Tapir Crossing, and the Buckhall Port and logistics center is expected to revert to the Government of Guyana at the end of mine life. The Mine Reclamation and Closure Plan therefore concentrates on the decommissioning and closure of primary elements of infrastructure at Aurora and mineral processing operations site and mancamp (see Figure 20-2); these include:

- All open pit mining areas, and the new outlet established for the interconnected pit lakes;
- The Cuyuni River dike;
- The underground mine and associated portal, ventilation shafts, support facilities and underground infrastructure;
- Waste rock/Saprolite stockpiles (to be concurrently closed during the operations phase);
- RoM stockpiles;
- The diesel power plant, power distribution substation, and transmission lines;
- The fuel storage tank farm, secondary containments, and fueling station;
- The mineral processing facility (including ore sorting, crushing, CIL, and cyanide detoxification circuits)
- The TMA and TMA Diversion Ponds 1 and 2;
- The FWP and MWP;
- Laydown areas and warehouses;
- Mechanical and maintenance shops;
- Permitted solid waste landfill;
- Hazardous waste storage facility area;
- Haul and access roads;

- Mancamp and administrative buildings; and,
- Potable water and septic systems.

19.5.2 Summary of Site Closure and Waste Disposal Strategy

Unless other land uses, mixtures of land use, or other beneficial uses of specific elements of mine infrastructure are negotiated with the Government of Guyana and other stakeholders, the overall goals for minedecommissioning and closure will be to return the land to a physically, biologically, and chemically stable and ecologically functional condition that approximates baseline conditions. AGM is also obliged, as a condition of its Environmental Permit (Guyana EPA, 2010), to minimize the potential attractiveness of the decommissioned site for illegal or uncontrolled ASM activities. Concurrent closure options will therefore be sought, wherever possible in the construction and operational phases of mine life, in an effort to minimize the potential for subsidence and erosion damage, to enhance biodiversity and the restoration of natural habitats, and to minimize the potential attractiveness of the site. These options will include:

- Concurrent reclamation and re-vegetation of access and haul road ROWs and construction borrow areas; progressive reclamation and re-vegetation of areas encountered within the miningconcession that may have been damaged by illegal or historical ASM, or other uncontrolled human intrusion. Concurrent placement of growth media covers and re-vegetation of top and bench surfaces of waste rock/Saprolite stockpiles, establishment of stable natural drainage channels, and installation of settling ponds at each stockpile, if required for sediment control; and
- Periodic removal of used, scrap, or surplus mining equipment or materials from the site (e.g., open pit mining equipment at the completion of pit operations; worn-out pumps or piping system components).

In addition, closure actions at the end of mine life will include:

- Placement of final growth media cover, regrading, and re-vegetation of the permitted solid waste landfill at the Aurora mancamp;
- Selective breaching, modification, and re-vegetation of the Cuyuni River dikes, and selective breaching, regrading, and re-vegetation of the FWP, MWP, TMA, and TMA Diversion Pond embankments, as indicated in Figure 20-4;
- Placement of growth media cover and revegetation of the dewatered beach areas of the TMA;
- Construction of effluent settling, dilution, and/or polishing ponds for the FWP, MWP, TMA, and TMA Diversion Ponds if necessary to ensure consistency of discharged water quality with respect to the (IFC, 2007) effluent standards;
- Development of interconnected pit lakes and a stable natural drainage channel to the Cuyuni River (with a security berm and warning signs established throughout the period of pit lake infilling); and

- Controlled closure and abandonment of monitoring wells and piezometers at the end of post-closure monitoring.

It should be noted that Guyana does not currently have any permitted hazardous waste disposal facilities; however, one facility is planned and this facility will be available for use at the time of mine closure. It is also noted that Guyana does not currently have any significant metals or other waste materials recycling capabilities. AGM will monitor for the development of such capabilities over the years of mine operation, and will update the Mine Reclamation and Closure Plan accordingly if viable recycling sources are identified for any of the waste types generated in site decommissioning and closure. However, for the purposes of this Technical Report and the initial version of the Mine Reclamation and Closure Plan it is conservatively assumed that no recycling facilities will be available. With these assumptions in mind, decommissioning wastes will be managed as follows:

Major equipment items are not expected to have any significant resale value at closure are limited to the crushers, the Tapir Crossing barge, the TMA reclaim barge, high-value gold room equipment, and a number of the larger pumps, will be removed at closure and sold if possible. Credit is taken for the anticipated proceeds generated from any sale.

Motorized equipment that is operational will be removed from the site and transferred to other AGM operations or sold on the open market as appropriate; no decommissioning cost credit has been assumed for such equipment, however, as it is expected that most equipment will be near the end of its service life.

Non-operational motorized equipment, tanks, pumps, piping systems, and wood, metal, or concrete structures will be decommissioned and demolished; hazardous materials will be drained or captured and collected in the onsite hazardous waste storage facility until transfer and controlled disposal in a Guyana EPA-approved offsite hazardous waste landfill.

Decontaminated tanks or equipment will be cut or disassembled into manageable sizes and disposed of in the underground mine or in a special inert waste disposal cell constructed in the final waste rock stockpile. Access to the underground mine after disposal actions are complete will be permanently closed by the installation of engineered plugs in the airshafts and main portal. Waste materials that may be disposed of in the waste rock stockpile inert waste cell will be completely covered with at least a meter of waste rock prior to closure.

Concrete foundation areas will be razed to ground level, covered with growth media, graded, and re-vegetated. Concrete demolition rubble will be placed in the waste rock stockpile prior to stockpile closure.

Cyanide facility equipment, tanks, and piping systems will be flushed, with rinsate routed to the detoxification plant prior to disposal in the TMA, after which the rinsed detoxification plant and the tailings pipelines will also be demolished and disposed of in the underground mine or inert waste cell in the waste rock stockpile.

Residual hazardous materials (e.g., unused reagents, fuel, lubricants, paints, insecticides, reagents, or explosives) will be returned to suppliers for credit or otherwise sold to properly licensed or reputable dealers, and strictly for the purposes intended by the manufacturer. Any residual hazardous wastes will be accumulated in the onsite hazardous waste storage facility prior to being routed to an approved offsite hazardous waste landfill; residual medical wastes will be routed to an approved medical waste incinerator in Georgetown.

19.5.3 Cost Estimate

A preliminary estimate of the conceptual costs of closure for the mine at the end of mine life is presented in Table 19.3.

Table 19.3: Estimated end of mine life closure costs

PROJECT TASK AREA	COST ($)
MINE AND WASTE ROCK	**424,539**
Cuyuni River Dike 1	161,985
Cuyuni River Dike 2	97,191
Underground Mine / Facilities	52,000
Open Pit	113,363
Waste Rock Stockpiles	0
PROCESS FACILITY	**518,082**
ONSITE INFRASTRUCTURE	**464,762**
Power Plant and Distribution	438,614
Onsite Roads	25,984
Services and Utilities	0
ANCILLARY BUILDINGS	**1,381,862**
Mine Buildings (Mechanical Shops and Warehouse)	196,110
Permanent Accommodations (Aurora Mancamp)	670,126
Other Facilities (Explosives Magazine, Hazardous Waste Facility, Fuel Farm)	511,928
Air Strip	0
TAILINGS	**1,033,813**
Tailings Management Area (TMA, Diversion Ponds 1 and 2, Tailings Pipeline, Reclaim Pipeline)	825,068
Fresh (Storm) Water Pond	0
Mine (Process) Water Pond	88,745
OFFSITE INFRASTRUCTURE	0
Access Road	0
Buckhall Port	0
Tapir Crossing	0
TOTAL PROJECTED DIRECT COSTS	**3,823,058**
INDIRECT COSTS	
Engineering / Design (5% of directs)	191,153
Contingency (15% of directs)	573,459
Unlisted Items (10% of directs)	382,306
Mobilization / Demobilization / Preparatory Work (10% of directs)	382,306
Construction Engineering / Construction Management	212,846
Monitoring and Maintenance (2 years)	478,514
Equipment Salvage	-345,000
Permitting (1% of directs)	38,231
TOTAL PROJECTED INDIRECT COSTS	**1,953,815**
TOTAL PROJECTED PROJECT COSTS (rounded)	**6,000,000**

These assumptions are integrated into the initial closure cost estimate presented in Table 20-3. In addition, as a condition of AGM's current Environmental Permit (Guyana EPA, 2010), two years prior to cessation of all mining operations and the decommissioning and demolition of ore processing and support facilities, a Detailed Mine Reclamation and Closure Plan (DRMCP) prepared, with appropriate input from affected stakeholders, and presented to the Guyanese EPA for review and approval. Any subsequent updates of this DMRCP shall be issued on a schedule to be negotiated with the Guyana EPA.

19.5.4 Post-Closure Monitoring

For the purposes of this Technical Report, a nominal 2-year post-closure period was defined and is reflected in the initial closure cost estimate. In actual practice, the predicted length of the post-closure monitoring period will be periodically refined in annual reviews of the Mine Reclamation and Closure Plan (and ultimately, the DRMCP) as AGM gains experience with actual progressive closure actions and develops the associated monitoring data that can be used to assess the effectiveness of selected reclamation, revegetation, and erosion prevention strategies. It is expected that these data will permit identification of the most successful reclamation, revegetation, and erosion prevention strategies that can be applied in final closure. The final DMRCP submitted to the Guyana EPA may also include additional negotiated or supplemental post closure monitoring actions or potentially monitoring schedule extensions. These actions will be designed to facilitate final relinquishment of the Aurora Gold Mine property, and may include:

- Physical maintenance and inspection of earthworks, including embankments and any permanent spillways, drains, settling ponds, or diversion channels;, an accordance with the Tailings Management Plan and Erosion Prevention and Control Plan;
- Continuation of surface and groundwater monitoring programs, in accordance with the Project Water Management Plan;
- Continued monitoring of revegetation areas to assess success of vegetative rebound, in accordance with the Erosion Prevention and Control Plan; and

Continued monitoring for the colonization of reclaimed areas by native fauna, in accordance with the Biodiversity Management Plan.

19.6 Social or Community Relations Requirements

Although small temporary or transient ASM encampments may occasionally be encountered in the project area, there are no formal or established communities or settlements in the immediate vicinity of the site, and only two informal communities or settlements (Aranka Mouth and Buckhall) are located anywhere near the associated project components. As a consequence, Aurora is not expected to generate the direct socio-economic effects that often characterize other mining projects. Most importantly, there were no temporary or permanent communities or residences within the Aurora concession that would have required physical displacement or resettlement actions.

The area of the project is very remote, but has been impacted by traditional and largely unregulated ASM, logging, and hunting activities for well over 100 years. As noted in: “Technical Report Identifying the Potential Range of Cultural Resources within the Aurora Gold Mining Project Area, Guyana” (Plew, 2012) there are no known archaeological sites or areas of significant cultural interest within the Aurora concession. Implementation of the Community Relations Management Plan for the project will also provide the means of detecting and appropriately responding to any changing stakeholder views with respect to cultural heritage concerns, as well as employment or contracting opportunities, health and safety, and other social considerations.

The informal communities considered to be in the project AOI are described as follows:

The Buckhall Port is the entrance point to the project area on the Essequibo River, and is the location of a concession area used by the Company for the staging and roadway transportation of materials, equipment, supplies, and employees. The adjacent community is considered to be in the project’s Direct Area of Influence (DAI).

Aranka Mouth is located approximately 16 km downstream from the mine site on the Cuyuni River, and is an informal community serving ASM; along with Buckhall; it is considered to be in Aurora’s DAI.

Bartica has served as a regional hub for providing services to the interior of the country since the 1880s. It is also the home of the Regional Democratic Council office, responsible for legislative and community services in Region 7. Bartica will not be directly impacted by the Project, but is considered to be in the mine’s Indirect Area of Influence (IAI).

It should be emphasized that the mine is not located on lands traditionally claimed or customarily used by indigenous Amerindian peoples. The nearest Amerindian community is Kurutuku, which, due to its remote location some 50 km upstream and on the Venezuela border, is unlikely to be significantly impacted by mine operations. However, as the Cuyuni River is a significant regional transportation corridor, AGM does recognize Amerindian groups as potential Aurora Gold Mine stakeholders. AGM currently employs a number of Amerindian workers, and hiring and contracting opportunities are made known to Amerindian communities. Approximately 15% of the AGM workforce are Amerindians.

Transient miners and other groups are recognized as potential stakeholders in the Community Relations Management Plan, and will have access to the grievance procedures defined therein. Transient miners will also have access to potential employment opportunities, subject to the evaluation of individual qualifications, education, experience, AGM centralized hiring and controlled workforce transportation practices, as well as expectations for strict compliance with the code of conduct established for the AGM workforce and contractor staff.

19.7 Conclusions and Recommendations

- The project has been in operation for one year. There is an ESHS MS System in place and external audits for IFC standards compliance review occur regularly;
- There are no non-compliance issues based on verbal information provided by the VP Sustainability, and confirmed by a general review of public information.

20 Capital and Operating Costs

20.1 Capital Costs

20.1.1 Open Pit Mine

As the Mine is an active operation, the open pit mine capital is largely driven by equipment acquisitions, specifically the primary mine equipment. Primary mine equipment here is defined as production and support equipment. Other capital costs are associated with ancillary equipment (dewatering pumps, pit lighting, light vehicles, maintenance vehicles, etc.), some facilities, and miscellaneous. The latter two categories are largely provided for in the AGM 2017 budget.

Pricing of equipment was based on recent purchases made for the active operations as well as written quotations. In the case of the 90 tonne haul trucks, there are a number of low hour units available on the used equipment market. SRK has assumed that availability will continue at reasonable levels to the end of this year and that pricing at 65% of new would be appropriate.

Included in direct equipment costs are the commissioning costs of the new larger shovels and trucks. Indirect costs of spares is estimated at 3% of the initial purchase of again the larger equipment. Freight costs from Georgetown are not included here, but rather are covered in the site G&A estimate (Logistics).

The summary of all these costs are provided in Table 20.1

Table 20.1: Open pit capital cost estimate

Item	Total Cost ($k)
Primary Equipment	26,122
Ancillary Equipment	2,085
AGM budget 2017	6,213
Subtotal Equipment Purchase	**34,419**
Erection/Commissioning	541
Equipment Indirects	
Freight	0
Spares	131
Total Equipment	35,091
Mine Facilities	0
Miscellaneous	1,010
Contingency (on all but Primary Equipment)	309
TOTAL MINE CAPITAL	**36,411**

The equipment fleet requirements for deriving capital costs are capped for certain equipment types, thus requiring the use of rental units when demand exceeds the owner’s fleet capacity. The smaller drills, 5.4 m^3 excavators and the 41 tonne haul trucks were all treated in this manner.

Finally, there is opportunity to consider salvage value on some of the mine equipment. SRK has assumed that for equipment at less than 50% of their expected life up to 50% of their new value may be achieved, and for equipment less than 80% of life, but more than 50%, only 25% of their new value may be realized in salvage.

20.1.2 Underground Mine

The capital cost estimate for the underground mine includes initial capital costs and sustaining capital costs and is presented in Table 20.2. Initial capital costs were defined as those incurred prior to mine production. Sustaining capital includes costs for ongoing capital development during mine production with an allowance for major equipment replacement

The capital cost estimate is based on the following:

- underground mining equipment list,
- budgetary quotes for the major equipment obtained by SRK from equipment manufacturers,
- project development plan,
- estimated from first principles cost per metre of jumbo development,
- budgetary quotes obtained by SRK from development contractors, and
- in-house database based on SRK experience on similar projects.

The purchase of a permanent mining equipment fleet would be required as the underground mining operation will be performed by the owner. It was assumed that all jumbo development will be done by the owner crews and raise development by a contractor.

AGM provided labor rates, on-site explosives and diesel fuel costs, and electricity prices. The budgetary quotes for drill steel and bits, ground support and blasting supplies were obtained from vendors. Cost estimates for ventilation, air cooling and electrical distribution were provided by other consultants.

Table 20.2 Rory's Knoll underground capital cost estimate

Capital Costs	$M
Capitalized Lateral Development	56.7
Contractor Raise Development	19.1
Contractor Diamond Drilling	5.7
Underground Mobile Fleet	40.3
Ventilation and Cooling	33.9
Mine Services	7.9
Mining Infrastructure	18.2
Safety	2.6
Technical Services	1.3
Direct Costs	**185.6**
EPCM and indirects (15% of directs)	7.6
Indirect Costs	**7.6**
Total	**193.2**

LOM capital lateral development costs for Rory's Knoll average $3,460/m. Vertical capital development rates have been estimated by a contractor and are presented in Table 20.3.

Table 20.3: Vertical capital development unit rates

Cost Item	Unit Cost ($/m)
Contractor capitalized raise bore vertical development	6,877
Contractor capitalized drop raise vertical development	2,167
Contractor capitalized alimak raise vertical development	5,139

Capital costs include estimates for:

- Portal construction and decline;
- Off decline development;
- Lateral development;
- Large diameter raise boring;
- Alimak and drop raising;
- Mobile mining fleet;
- Underground diamond drilling;
- Primary ventilation and secondary ventilation fans;

- Refrigeration plant;
- Mine services and support infrastructure;
- UG power distribution;
- Safety equipment, refuge chambers and egress ladder way system; and
- Technical services equipment and software.

20.1.3 Processing Capital Costs

Processing capital costs for the two phases of the mill expansion are provided in Table 20.4. The ordering of long lead-time items for the Phase 1 expansion has already commenced and detailed engineering is expected to commence in February 2017. The first phase of the expansion is expected to be completed by the end of the first quarter of 2018. The second phase of the expansion is expected to commence in mid-2018 and be completed by mid-2019.

Table 20.4: Processing expansion capital costs

Description	Phase 1 Expansion ($M)	Phase 2 Expansion ($M)	Total ($M)
Crushing	-	-	-
Ball Mill & Grinding	0.3	11.0	11.3
Gravity	-	0.5	0.5
Pre-Leach Thickening & Leaching	9.2	-	9.2
Elution & Carbon Management	2.6	-	2.6
Electro-Winning & Gold Room	0.6	0.3	0.9
Detox & Tailings	-	2.0	2.0
Power Generation & Fuel	-	1.7	1.7
Other	0.1	0.8	0.9
Total Direct Costs	**12.8**	**17.2**	**30.1**
Indirect Costs	6.2	7.3	13.5
Owner's Costs	1.3	2.2	3.6
Contingency (5%)	1.1	1.0	2.1
Total Capex	**21.4**	**27.7**	**48.3**

20.2 Operating Cost

SRK provided Aurora open pit and underground operating cost estimates, while JDS provided operating cost estimates for processing. Table 20.5 shows Project LOM operating cost estimates

Table 20.5: LOM operating cost estimate

Description	LOM Total	Years 2017-2019	Years 2020-2021	Years 2022-2024	Years 2025-2032
	($)	($)	($)	($)	($)
Open Pit Mining	391.3	146.1	143.6	101.6	0.0
Underground Mining	467.6	0.0	0.0	31.5	436.1
Processing (incl. tailings handling)	528.4	108.2	84.4	110.6	225.2
General & Administrative	322.4	64.1	42.8	64.1	151.4
Total Operating Cost	**1,709.7**	**318.4**	**270.8**	**307.8**	**812.7**

20.2.1 Open Pit Mining Costs

Mine operating costs are based on first principles estimation and SRK experience, complemented by existing operating cost data for the same mining equipment currently in use at the Aurora Mine. For new equipment, SRK used available benchmark cost data.

Local labour rates (for operating, maintenance, and salaried personnel) and estimates on diesel fuel pricing ($0.55/l) were taken into consideration for the mining cost estimate.

20.2.1.1 Drilling and Loading

The requirements for drills and shovels are based on productivity factors currently tracked and used for budgeting at the Aurora Mine. New larger drills are acquired starting in 2017 and continue to be added to as production requirement increase in the mine plan. Larger hydraulic shovels (10 m^3 bucket) are added, starting in the first quarter of 2018, to mine waste in the upper benches of Rory's Knoll and Aleck Hill pits.

Costs are similarly derived from the AGM budgeting process. In some periods, operational requirements exceed the owner fleets of drills and excavators (Figure 15.20). In these instances, rental equipment are assumed and currently used rental rates are applied

20.2.1.2 Blasting

Commencing mid-2017, blasting in rock waste and ore is to be performed with bulk ANFO and emulsion assuming a 30%/70% split. In alignment with the AGM 2017 budget, a 4.0 m x 4.6 m pattern is assumed for waste and ore (powder factor of 0.31 kg/t ore and 0.29 kg/t waste). First principle costing of blasting consumables per the AGM 2017 constitutes the blasting costs.

20.2.1.3 Mine Haulage

Truck productivities and fuel consumption rates are derived from the average haul profiles, considering centroids of pit benches and destinations. Starting in the first quarter of 2018, 90 tonne haul trucks are acquired to match the hydraulic shovels mining in waste. The existing 41 tonne haul trucks haul all ore from all the open pits as well as waste from the lower benches of Rory'Knoll and Aleck Hill pits and all waste from the remaining pits.

In some periods toward the end of the LOM, truck requirements exceed the designated owner fleet. In these periods, rental 41 t trucks are assumed to make up the capacity, and current rental rates are applied (derived from the AGM 2017 budget).

20.2.1.4 Support Equipment

The non-production support equipment requirements (dozers, graders, water trucks, etc.) are factored based on the production equipment requirements and the number of active mining areas. The support equipment requirements derived in this way align with the Additional support equipment has been designated for road maintenance of the ore haul.

20.2.1.5 General Mine/Maintenance

General Mine/Maintenance operating costs include those costs associated with ancillary equipment such as light plants and dewatering equipment as well as personnel transport and maintenance service vehicles. Such costs are identified in the AGM 2017 budget as heavy, medium, and light duty equipment. SRK segregated those units that are impacted by higher production, adjusting their costs as required, while using the AGM budget amounts for the remaining equipment.

The labour costs for the mine general labour pool was also included in this cost category.

20.2.1.6 Hourly Operating Labour Requirements

The mine labour requirement was developed from alignment of equipment requirements and operator manning strategies. Modeled operators aligned with the AGM 2017 budget and considered a three crew rotation. The average labour requirement for when the mine is at full production is 212 people with a peak of around 250 people in mid 2020.

20.2.1.7 Mine Maintenance Labour Requirements

Mine maintenance headcount is factored from the equipment count in each period. SRK selected factors that allowed alignment of the modeled headcount with that of the AGM 2017 budget. The average personnel requirement for when the mine is at full production is 104 people, with a peak of about 128 people in mid 2020.

20.2.1.8 Supervision and Technical

The manpower requirements for mine supervision and technical personnel have been estimated based on current AGM headcounts. The costs are however escalated by 15% to account for additional personnel when the mine reaches higher production levels starting mid 2018.

In addition to personnel cost, Mine Supervision and Technical costs account for:

- Training
- Communications/Computers/software,
- Supplies,
- Ex-pat travel,
- External consulting,
- Maintenance G&A
- Freight, and
- Sampling/Assaying.

These costs were all derived from the AGM 2017 budget although SRK did pro-rate the freight and sampling/assaying budgets on a mined tonne basis.

20.2.1.9 Mine Operating Cost Summary

Based on the production schedule, the operating costs were estimated for different units of operation for each period. Overall, the mining operating cost is $2.11 per tonne mined for the life of mine. Table 20.6 presents the summary of the operating costs over the life of mine, from 2017 to 2024.

Table 20.6: LOM Open pit mining cost estimate

Item	Cost
Direct Mining Costs ($k)	
Drilling	43,504
Blasting	109,119
Loading	45,565
Hauling	86,492
Support	40,723
General Mine/Maintenance	22,653
Supervision & Technical	43,281
Total Mining	**391,337**
Mining Cost ($/tonne moved)	
Drilling	0.23
Blasting	0.59
Loading	0.25
Hauling	0.47
Support	0.22
General Mine/Maintenance	0.12
Supervision & Technical	0.23
Total Mining, per tonne moved	**2.11**
Total Mining, per tonne ore	**19.97**

Note that the drilling and blasting costs unit costs are expressed as $/tonne of all material moved. In actuality these costs are only applied against rock moved. The drilling and blasting costs per tonne of rock moved are $0.35/t and $0.80/t respectively.

Table 20.7 shows open pit operating costs by year on a total dollar and unit cost basis.

Table 20.7: LOM open pit mining cost estimate

Description	Total	2017	2018	2019	2020	2021	2022	2023	2024
Production (kt):									
Ore Mined	19,634	2,470	3,048	2,149	2,321	2,937	2,913	2,697	1,099
Waste	165,944	11,738	28,388	32,713	32,767	27,645	23,413	8,928	353
Total Material	185,578	14,208	31,436	34,863	35,087	30,581	26,325	11,625	1,452
Strip Ratio	8.45	4.75	9.31	15.22	14.12	9.41	8.04	3.31	0.32
Direct Mining Costs ($k)									
Drilling	43,504	3,645	7,772	7,993	7,184	5,758	2,643	498	8,011
Blasting	109,119	7,608	23,146	23,642	23,463	19,114	7,952	1,270	2,924
Loading	45,565	8,208	8,701	8,547	7,732	6,752	3,029	627	1,970
Hauling	86,492	10,231	12,744	15,836	17,249	17,384	8,853	2,201	1,993
Support	40,723	6,162	6,369	6,801	5,934	5,623	3,543	1,149	5,143
General Mine/Maintenance	22,653	2,758	2,584	2,568	2,444	2,676	1,664	451	7,508
Supervision & Technical	43,281	7,048	7,400	7,425	6,813	6,059	3,763	559	4,214
Total Mining ($k)	**391,337**	**31,763**	**45,659**	**68,715**	**72,812**	**70,819**	**63,366**	**31,447**	**6,756**
Mining Cost ($/tonne material)									
Drilling	0.23	0.61	0.12	0.23	0.23	0.23	0.22	0.22	0.26
Blasting	0.59	0.22	0.24	0.67	0.67	0.77	0.73	0.65	0.67
Loading	0.25	0.15	0.26	0.25	0.24	0.25	0.26	0.25	0.33
Hauling	0.47	0.15	0.33	0.37	0.45	0.56	0.66	0.72	1.16
Support	0.22	0.39	0.20	0.19	0.19	0.19	0.21	0.29	0.60
General Mine/Maintenance	0.12	0.57	0.09	0.08	0.07	0.08	0.10	0.14	0.24
Supervision & Technical	0.23	0.32	0.22	0.22	0.21	0.22	0.23	0.31	0.29
Total Mining, $/tonne material	**2.11**	**2.41**	**1.45**	**2.00**	**2.08**	**2.32**	**2.41**	**2.56**	**3.56**
Total Mining, $/tonne ore	**19.97**	**12.86**	**14.98**	**32.54**	**31.38**	**24.12**	**21.75**	**11.66**	**6.15**

20.2.2 Underground Mining Costs

20.2.2.1 Rory's Knoll

LOM underground mining costs for Rory's Knoll deposit include $51.5 million for operating development, $280.2 million for production ore mining and $49.5 million for contractor equipment allocation costs.

The underground mining operating costs were calculated for each cost category such as development, production, haulage, maintenance, mine services, and labour. The cost was estimated using a combination of first principles calculations, experience and factored costs. Productivities, equipment operating hours, labour and supply requirements were estimated for each type of underground operation.

LOM operating development costs for Rory's Knoll average $2,752/m. LOM operating cost estimates for Rory's Knoll are presented in Table 20.8.

Table 20.8: Rory's Knoll Underground Operating Costs

Cost Item	LOM Cost ($k)	Unit Cost ($/t ore)
Operating Development	51,481	3.47
Mining Operating Costs		
LH drilling	20,664	1.39
ITH drilling	6,432	0.43
Blasting	25,043	1.69
Bogging	25,066	1.69
Hauling	70,670	4.77
Subtotal	147,876	9.98
Utilities and Labor Operating Costs		
Auxiliary equipment	11,581	0.78
Power	60,210	4.06
Labor	42,242	2.85
Dewatering	15,442	1.04
Ventilation	2,818	0.19
Subtotal	132,293	8.93
Contractor Equipment Allocation		
Equipment Allocation Cost	49,496	3.34
Total Cost of Production	**381,145**	**25.72**

20.2.2.2 Satellite Deposits

Operating cost for Aleck Hill and Mad Kiss deposits is estimated based on the assumption of a shared equipment, underground infrastructure and mine services.

LOM operating cost estimates for Satellites are presented in Table 20.9.

Table 20.9: Satellites Underground Operating Cost

Cost Item	LOM Cost ($k)	Unit Cost ($/t ore)
Operating Development	18,792	11.05
Mining Operating Costs		
LH drilling	3,539	2.08
ITH drilling	1,752	1.03
Blasting	754	0.44
Bogging	2,858	1.68
Hauling	7,204	4.23
Backfill	4,207	2.47
Subtotal	20,314	9.47
Utilities and Labor Operating Costs		
Auxiliary equipment	6,637	3.90
Power	8,285	4.87
Labor	16,017	9.41
Dewatering	8,132	4.78
Ventilation	1,177	0.69
Subtotal	40,249	23.66
Contractor Equipment Allocation		
Equipment Allocation Cost	7,112	4.18
Total Cost of Production	**86,466**	**48.35**

20.2.3 Processing Operating Costs

Processing operating costs were developed from first principals and compared with current operating costs observed at site. Since the AGM process facility is in start-up mode, actual operating costs were focused on reagent consumptions and power costs when the mill was running at design capacity. The power cost was calculated to be $0.14/kWh using a fuel cost of $0.55 per liter. Table 20.10 shows the process operating costs by process function, blend percentage, and milling rate.

Table 20.10: Process operating costs by mill rate

Description	2017 + 1Q18	2Q18 - 2Q19	3Q19 onwards
Saprolite portion of Mill Feed	~30%	25-50%	< 10%
Daily Milling Rate (tpd)	5,600	8,000	8,000
Recovery	91.5%	92.5%	94.0%
Operating Costs by Process Function ($/t)			
Labour	2.56	1.71	1.71
Power	4.83	3.20	3.92
Grinding Media	1.12	1.10	1.63
Liners & Wear	1.02	0.61	0.83
Reagents	4.54	4.20	4.22
Maintenance	2.14	1.84	1.99
Laboratory	0.14	0.09	0.09
Total Processing Opex ($/t)	**16.36**	**12.75**	**14.39**

21 Economic Analysis

21.1 Summary

The economic analysis of the Aurora Gold Mine was undertaken using a Discounted Cash Flow (DCF) model in Microsoft Excel®. The model uses “real” 2017 US dollars as the primary currency. The model uses mid-period annual discounting at a base-case discount rate of 5% and a valuation date of January 1, 2017.

On instruction from GGI, modelling was undertaken exclusively in real terms and no escalation was applied for tax depreciation and working capital calculations. The impact of this assumption is not material for this study.

The model provides key project values for pre-tax and post-tax NPV on a go forward basis. Construction capital costs prior to January 1, 2017 and debt financing are excluded from the analysis. Sunk costs and interest payments are only considered for tax analysis purposes. Values for Aurora Gold Mine IRR and payback are unavailable as the majority of the capital costs have been spent before January 1, 2017 and the project doesn’t generate any negative cash flows. The sensitivity analysis includes fuel and energy cost, overall operating and capital expenditures and commodity price.

Base case gold price used for economic analysis is $1,200 per ounce.

21.2 Financial Model Parameters

21.2.1 Production Schedule

The evaluation of the Aurora Gold Mine is based on a 16 year LOM plan comprised of open pit and underground production. The open pit schedule, developed by SRK, assumes just over seven years of production from Rory’s Knoll, Aleck Hill, Walcott Hill and Mad Kiss deposits. The underground schedule was developed by SRK and assumes approximately eight years of production from the Rory’s Knoll deposit and seven years of production from the underground satellite deposits. The combined mine production schedule is shown in Table 21.1.

The detailed description of open pit, Rory’s Knoll and Satellites underground production schedules can be found in Section 15.3.3, Section 15.4.2.3, and Section 15.4.3.3 respectively.

Table 21.1: LOM combined production schedule

Parameter	Units	Total	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
OPEN PIT																		
Open Pit Material Moved	kt	185,151	13,178	31,436	34,363	35,087	30,581	26,325	12,281	1,899								
Open Pit Waste	kt	165,554	10,708	28,388	32,251	32,767	27,645	23,413	9,584	799								
Saprolite Ore Mined	kt	2,934	1,001	1,417	94	91	0	189	142	0								
Rock Ore Mined	kt	16,662	1,469	1,631	2,018	2,229	2,937	2,723	2,556	1,099								
Total Open Pit Ore Production	kt	19,596	2,470	3,048	2,112	2,321	2,937	2,913	2,697	1,099								
Gold Grade	g/t Au	2.87	2.58	2.63	2.91	2.82	3.42	2.70	2.80	3.41								
Metal Mined from Open Pit	k Oz Au	1,810	205	258	198	210	323	253	243	120								
UNDERGROUND – RORY’S KNOLL																		
Capital Lateral Development	m	16,392						1,806	2,983	2,205	2,109	2,317	2,031	1,880	1,061			
Operating Lateral Development	m	18,704							149	3,346	3,628	3,303	3,682	1,665	2,931			
Ore Mined - Rory's Knoll	kt	14,817							8	879	1,940	2,048	2,091	1,960	2,045	1,851	1,753	242
Gold Grade	g/t Au	3.02							2.51	3.66	3.24	2.93	3.03	3.04	3.25	3.06	2.33	2.01
Metal Mined from Rory's Knoll UG	k Oz Au	1,437							1	103	202	193	204	191	213	182	131	16
UNDERGROUND - SATELLITES																		
Capital Lateral Development	m	13,239							2,252	4,499	2,727	2,111	1,359	292				
Operating Lateral Development	m	2,925							426	766	622	559	552					
Ore Mined - Satellites	Kt	1,701							13	77	315	367	375	205	205	144		
Gold Grade	g/t Au	4.69							3.48	4.26	5.42	4.95	4.70	4.07				
Metal Mined from Satellites UG	k Oz Au	257							1.4	10.6	54.9	58.4	56.7	26.9	24.8	22.8		
Total Gold Mined	**k Oz Au**	**3,503**	**205**	**258**	**198**	**210**	**323**	**253**	**245**	**235**	**257**	**252**	**261**	**218**	**238**	**205**	**131**	**16**

21.2.2 Processing Plant Capacity

The processing plant will operate at different throughput rates during the life of mine. The detailed description of the processing facilities can be found in Section 16.4.

Table 21.2 summarizes the LOM milling schedule.

Table 21.2: LOM tonnage milled

Description	Units	Total/ Average	2017	2018	2019-2032
Throughput Rate	tpd	6,646	5,600	7,400	6,300
Saprolite Mill Feed	kt	3,270	613	1,151	1,505
Fresh Ore Mill Feed	kt	33,509	1,431	1,553	30,525
Total Mill Feed	kt	36,778	2,044	2,704	32,030
Mill Feed Average Grade	g/t Au	3.02	2.65	3.67	2.81
Recovery Grade	%	93.7%	91.2%	92.5%	94.0%
Total Au Metal Recovered	**K Oz**	**3,315**	171	254	2,890

21.3 Cost Estimates Summary

The mine and plant are currently in production. A mill expansion to increase maximum processing rate, particularly as the feed proportion of saprolite ore decreases is planned in this study. Capital cost estimates for open pit, underground mining and processing facilities are discussed in Section 20.1. Operating cost estimates for open pit, underground mining and processing facilities are discussed in Section 20.2.

Project capital cost estimates as of Jan 1, 2017 are presented in Table 21.3 and include expenditures associated with processing plant expansion to handle higher throughput rate, open pit sustaining capital, expenditures required to develop the underground, underground sustaining capital and closure costs. Mine closure capital is scheduled in the cash flow to be spent at the conclusion of mining. For tax purposes, neither loss carry-back nor prior provision was modelled, and therefore the full pre-tax cost is considered.

Table 21.3: Total project capital expenditures

Description	LoM Total ($M)	Years 2017-2019 ($M)	Years 2020-2021 ($M)	Years 2022-2024 ($M)	Years 2025-2032 ($M)
Expansion Capital					
Open Pit Mine	24.3	24.3	0.0	0.0	0.0
Underground Mine	154.5	0.0	0.0	128.9	25.7
Processing Facilities	48.3	48.3	0.0	0.0	0.0
Subtotal	**227.2**	**72.7**	**0.0**	**128.9**	**25.7**
Sustaining Capital					
Open Pit Mine	12.1	8.9	2.9	0.3	0.0
Underground Mine	168.2	0.0	0.0	38.7	129.4
Processing Facilities (incl Tailings Facility)	35.3	6.4	4.2	5.9	18.9
G&A	23.8	10.8	2.0	3.0	8.0
Closure Costs	7.0	0.0	0.0	0.0	7.0
Subtotal	**246.4**	**26.1**	**9.1**	**47.9**	**163.3**
Total Capital Costs	**473.6**	**98.8**	**9.1**	**176.7**	**189.0**

Total project operating cost estimates are $1,710 million as summarized in Table 21.4. LOM cash costs per payable ounce of gold are presented in Table 21.5.

Table 21.4: Total project operating expenditures

Description	LOM Total	Years 2017-2019	Years 2020-2021	Years 2022-2024	Years 2025-2032
	($M)	**($M)**	**($M)**	**($M)**	**($M)**
Open Pit Mining	391	146	144	102	0
Underground Mining	468	0	0	31	436
Processing (incl. tailings handling)	528	108	84	111	225
General & Administrative	322	64	43	64	151
Total Operating Cost	**1,710**	**318**	**271**	**308**	**813**
Unit Operating Cost	**LOM Average ($/$t_{processed}$)**	**($/$t_{processed}$)**	**($/$t_{processed}$)**	**($/$t_{processed}$)**	**($/$t_{processed}$)**
Open Pit Mining	10.64	19.06	24.56	13.16	0.00
Underground Mining	12.71	0.00	0.00	4.08	28.06
Processing (incl. tailings handling)	14.37	14.11	14.43	14.33	14.49
General & Administrative	8.77	8.36	7.32	8.31	9.74
Total Unit Operating Cost	**46.49**	**41.53**	**46.31**	**39.87**	**52.29**

Table 21.5: LOM average cash costs

Description	LOM Average ($/ozAu)	Years 2017-2019 ($/ozAu)	Years 2020-2021 ($/ozAu)	Years 2022-2024 ($/ozAu)	Years 2025-2032 ($/ozAu)
Open Pit Mining	118	234	277	147	0
Underground Mining	141	0	0	46	294
Processing	160	174	163	161	152
General & Administrative	97	103	83	93	102
Total Unit Operating Cost	**516**	**511**	**523**	**447**	**548**
Royalty	96	96	96	96	96
Total Cash Costs including Royalty	**612**	**607**	**619**	**543**	**644**

21.4 Taxation and Royalties

21.4.1 Royalty

A royalty is payable to the government of Guyana and was modelled in accordance with information supplied by AGM. Notably, royalty is applied to contained metal in doré. The applicable rate and the LOM royalty estimate is shown in Table 21.7.

Table 21.6: LOM royalty estimate

Description	Units	Value
Base Gold Price for Royalty Calculations	$/oz	1000
Royalty rate if price is over 1,000 $/oz	%	8.0
Royalty rate if price is less or equal 1,000 $/oz	%	5.0
Total Royalty Estimate	**$M**	**317.3**

21.4.2 Taxes

The taxation liability has been estimated based on guidance provided by GGI. The estimates are simplified for the purpose of this economic analysis.

Corporate income tax rate in Guyana is 27.5% applicable on all earnings before interest (EBIT) after royalty and depreciation. The depreciation rate is assumed to be 20% on a straight line basis.

For taxation purposes, on the direction of GGI, in order to utilize the accumulated tax losses the net operating cash flow is split between Guyana Goldfields and AGM on 25% and 75% respectively.

Existing tax losses (see below), management fees (at 1% of NSR) and loan interest payments (at $10M p.a. for 4 years) were provided by GGI and incorporated into the tax modelling. These adjustments affected only tax and were not otherwise reflected in the project cashflows.

21.4.2.1 Guyana Goldfields Inc.

It is estimated that Guyana Goldfields Inc. (GGI) has an opening balance of tax loss carry forward totalling US$66 Million. The opening balance and any further tax losses are carried forward and deducted from earnings before tax (EBT).

An estimate of LOM corporate income tax for GGI is shown in Table 21.8.

Table 21.7: GGI LOM corporate income tax estimate

Description	Units	Value
EBT (after carried forward tax loss)	$M	232
Corporate Income Tax	**$M**	**64**

21.4.2.2 AGM Inc.

It is estimated that AGM Inc. (AGM) has an opening balance of tax loss carry forward totalling US$37 Million. The opening balance and any further tax losses are carried forward and deducted from earnings before tax (EBT).

An estimate of LOM corporate income tax for AGM is shown in Table 21 9.

Table 21.8: AGM LOM corporate income tax estimate

Description	Units	Value
EBT (after carried forward tax loss)	$M	856
Corporate Income Tax	**$M**	**236**

21.5 Working Capital

Changes in working capital were estimated based on Account Receivable delays (AR), Accounts Payable delays (AP) and Stores Stocks. The assumptions used were twenty (20) days of AR, thirty (30) days of AP, and 20% of total annual operating costs in stores with a 5% annual contraction due to spoilage and obsolescence.

21.6 Revenue

Revenue is determined by applying selected metal price to the annual payable metal contained in doré, minus refining, freight, insurance and royalty payments. Freight and insurance cost for Aurora is estimated at $2.2/oz, and refining cost used is $0.30/oz. Calculation of revenue is presented in Table 21.10.

Table 21.9: LOM revenue calculation summary

Description	Units	Value
Total Ore Milled	ktonnes	36,778
Mill Feed Grade	gpt	3.02
Total Au Recovered	kounces	3,314
Payable Gold Produced	kounces	3,312
Gold Price	US$/oz	1,200
Gross Revenue	$M	3,974.7
Refining and Freight Costs	$M	8.3
Royalty	$M	317.3
Net Revenue	**$M**	**3,649.1**

21.7 Summary of Base Case Input Parameters and Results

A summary of the results of the base case economic analysis is presented in Table 21.11.

Under the base case input estimates, the Aurora Gold Mineis expected to yield an after-tax undiscounted LOM net cash flow of $1,174 million, and an NPV of $850 million at a discount rate of 5% per year.

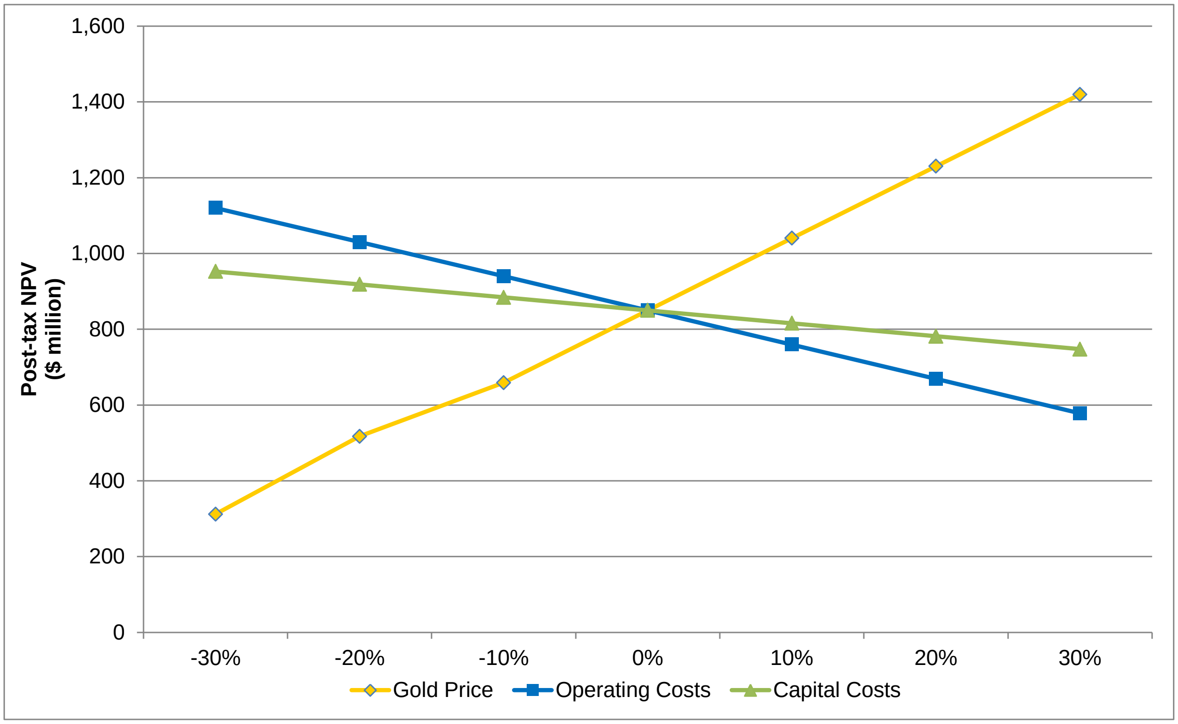
Table 21.10: Project input parameters and results

Description	Units	Value
Open Pit Ore Mined	kt	19,596
Underground Ore Mined	kt	16,519
Total Ore Mined	kt	36,115
Total Ore Milled	kt	36,778
Payable Gold Produced	k Oz	3,312
Gold Price	US$/oz	1,200
Gross Revenue	$M	3,975
Refining and Freight Costs	$M	8
Royalty	$M	317
LOM Operating Costs	$M	1,710
LOM Capital Costs	$M	474
LOM Working Capital	$M	-7
LOM Pre-Tax Cash Flow	$M	1,473
LOM Corporate Tax	$M	299
LOM Post-Tax Cash Flow	$M	1,174
Pre-tax NPV	**$M**	**1,054**
Post-tax NPV	**$M**	**850**

21.8 Sensitivity Analysis

Sensitivity analysis have been conducted to determine the effect on post-tax NPV variations from the base case commodity price as well as the change in overall operating and capital costs. Summary of sensitivity analyses is shown in Figure 22.1.

Figure 21.1: Project sensitivity to changing gold prices, capital and operating costs



22 Adjacent Properties

The authors are not aware of adjacent properties to be included in this technical report.

23 Other Relevant Data and Information

The authors are not aware of any additional information or explanation necessary to make the technical report understandable and not misleading.

24 Interpretation and Conclusions

24.1 Conclusions

Under the base case input estimates, the Aurora gold Mine is expected to yield an after-tax undiscounted LOM net cash flow of $1,174 million, and an NPV of $850 million at a discount rate of 5% per year.

The results of sensitivity analyses show that the project can withstand a substantial increase in capital and operating costs. A 30% decrease in gold price results in a positive NPV5% of $312 million; at a gold price of $1,500 per ounce the NPV5% results in $1,325 million.

24.1.1 Open Pit Mining

The Aurora Mine has successfully completed its commissioning and as of this writing is in a steady state of mining production. The operations are sufficiently established so as to provide the basis of much of the technical and economic inputs required for this feasibility study update.

This study has demonstrated that AGM would be able to increase the mine production beginning in 2018 to meet a mill throughput rate of 8,000 tonnes per day. Both saprolite and rock ores are mined through 2019 with only rock ore available from 2020 onward. In order to elevate the mill feed grade in the early years, SRK segregated low grade saprolite and rock ores for later feeding to the mill at the end of the mine life.

The increase in mine production rates is enabled by the acquisition of larger loading and hauling equipment starting in 2018. This enables AGM to realize the benefits of lower unit costs as well as limit the size of equipment fleets needed in the mine plan.

24.1.2 Processing

The current processing facility operating at AGM is meeting the predicted throughput rates and grind sizes. The proposed expansion of the processing facility in two phases will ultimately result in an expanded throughput rate of 8,000 tpd hard rock.

24.1.3 Underground Mining

The study has demonstrated that the underground operations Rory's Knoll and the satellite deposits (Aleck Hill and Mad Kiss) are economically viable in their current configurations.

24.2 Risks

During the course of the study the following potential risks were identified.

24.2.1 Geotechnical

24.2.1.1 Open Pit Getechnical

- **Brittle Fault Location and Characterization** – Currently, a model of the brittle fault features and large persistent joints expected behind the pit face in the region of the final pit does not exist. Local structure mapping has been conducted on site, however these observations have not been interpreted to extend behind the pit face to the planned interim or final wall locations. The structural model of the rock at the final pit wall location is an integral part of the overall slope design which must be reviewed against the likelihood of faults to avoid large scale slope failures ranging from multi-bench to inter-ramp scale. An improved understanding of the nature and location of any potential brittle fault features at the final pit wall locations should be undertaken as soon as possible. This is commonly achieved through the development of a 3D structural model. All slope designs and angles are contingent to their review against a fault model and should not be considered final until this reconciliation is complete. Pit slope designs may be governed by major structures.
- **Shear Zone Characterization (Aleck Hill)** – The location and geotechnical nature (thickness, strength, etc) of the shear zones at the Aleck Hill pit are based on limited data. Additional boreholes should be drilled to allow for a more complete characterization of these zones. If the shears are found to be extensive, inter-ramp wall angles may be required to be flattened to accommodate for the foliation and weakened rock mass strength.
- **Wall Control Blasting** – The angles provided within this document are based up best industry blasting practices in which wall control is of primary importance. Excessive blast damage will not allow for the achievement of the designed IRA's. Bench heights may be reduced to 10m if necessary, however, bench widths should also be evaluated to keep the IRA's in the design sectors similar.

24.2.1.2 Underground Geotechnical

The geotechnical design parameters presented within this document are based upon the review of historical data and observations on site. The following risks have been identified to the underground design:

- **Slope Stability** – The current mine design includes open benching to the surface. Instability of the slopes above may lead to excess dilution in the underground mine. Due to their impact on the underground mine, the slopes should be designed to a FOS of 1.3 and above.
- **Brittle Fault Location and Characterization** – Currently, a model of the brittle fault features and large persistent joints expected underground does not exist. Local structure mapping has been conducted on site, however these observations have not been interpreted to extend to the underground locations. An improved understanding of the nature and location of any potential brittle fault features at the underground mine locations should be undertaken as soon as possible. This is commonly achieved through the development of a 3D structural model.

- **Shear Zone Characterization (Aleck Hill)** – The location and geotechnical nature (thickness, strength, etc) of the shear zones at the Aleck Hill pit are based on limited data. Additional boreholes should be drilled to allow for a more complete characterization of these zones. If the shears are found to be extensive, underground development and stope locations may be required to move or be reduced in size to accommodate for the foliation and weakened rock mass strength or an increase in ground support (and associated costs) will be required.
- **Water Inflow** – Any changes in the location and geotechnical nature (thickness, strength, etc) of the shear zones at the Aleck Hill area may affect the local water inflow into the underground excavation. Additional costs may be associated with managing the water inflow.
- **Ore Geometry** – ore geometry which is more complex than expected, may increase damage and dilution during production, and reduce ore recovery.

24.2.2 Mining

24.2.2.1 Open Pit Mining

There is risk that the production increase in 2018 may pose logistical and management challenges. A key area to consider is training of new operators and staff to ensure efficiencies envisioned in this study are realized. AGM would be advised to develop a comprehensive execution plan to ensure that milestones are met as new equipment and personnel are brought on line.

24.2.2.2 Underground Mining

- **Mudrush** – External mudrush risk exists for the underground mine due to the heavy rainfall and the potential for generating fines and clays from the overlaying saprolite material. This risk will be mitigated by partial pre-stripping of saprolite ore as part of the open pit mining and by implementation of an effective dewatering and water diversion programs, such as perimeter drainage, collection sumps, etc.
- **Labour productivity** – The productivity of the Guyanese workforce in an underground mining environment is not demonstrated. Timely supply of expatriate and skilled local personnel has the potential to be a very significant risk to the success of the project. The ability to adequately train local unskilled labour to the required level is also a key factor for the underground mine. Sufficient time for planning and execution of the contracting strategy must be allowed.
- **Execution risk** – Any mining system is dependent on professional execution and there exists a risk that the planning and procedures are either optimistic or ultimately not well executed. AGM should ensure that both planning and execution capability are well-considered, are not overly optimistic, and that downside risk does not place the underground mining values proposition at undue risk.

25 Recommendations

25.1 Recommendations

25.1.1 Geotechnical

SRK recommends further investigation into the geotechnical characterization of the sericite schist through either pit wall mapping when exposed or additional borehole investigations. Further to that:

- Geotechnical monitoring of open pit stability, stope stability, and the SLR mine is imperative to follow best practices for both production and safety reasons.
- Standard open pit slope monitoring using a prism network should be established and developed continuously as open pit mining progresses. If areas of concern are determined through visual observations and/or the analysis of the monitoring equipment, additional slope stability measuring equipment, such as slope stability radar should be implemented to assess the slope stability. This stability monitoring system will continue to be utilized as underground mining progress and will be complemented by additional underground excavation monitoring systems using Lidar or similar. The surveyed underground and SLR excavations can then converted into a 3D model used for on-going geotechnical analysis. Such models aid in further prediction of stability, potential dilution/recovery reconciliation, and risk mitigation and design measures.
- A Ground Control Management Plan (GCMP) will be required once underground development commences at Aurora Gold Mine. A GCMP is a live document that is prepared, reviewed and approved by all key stakeholders, and is intended to provide a background on the likely ground conditions, required procedures, and policy controls in place to manage the risks related to the rock mass conditions. The intent of the GCMP is to articulate the higher level strategies aimed at eliminating or minimizing the risk of fatalities, injuries, and incidents resulting from falls of ground and collapse in underground operations, of production loss, and of equipment damage.

25.1.2 Open Pit Mining

SRK recommends that the Company begins considering equipment sources for its mining fleet expansion. In particular, low-hour used equipment should be sourced to ensure optimal economics for the mine expansion.

In preparation for the expansion in 2018, the Company should formalize all procedures and training processes in order to initiate and train new staff. This is to ensure that the efficiencies outlined in this feasibility study update are achieved.

Also, while not adopted in this study, the Company should consider acquiring a mine management system to track and potentially computer dispatch mining equipment. Up to 85 equipment units will be deployed at the Mine at its peak production.

25.1.3 Underground Mining

It is expected that the resource model and the geotechnical model will be updated regularly as new data becomes available from operations and additional drilling. The updated models should be used as a basis for re-evaluating the underground mining methods and production rate to ensure the optimum strategy is advanced.

Given that the Rory's Knoll deposit is open at depth, it would also be appropriate to re-evaluate the ore handling strategy prior to committing to a decline haulage system.

25.1.4 Processing

Further work is required on the potential to utilize the pebble crusher for new feed from the jaw crusher in addition to pebbles recycled from the SAG mill to increase the capacity in the grinding. Further evaluation is also required on the use of existing equipment for the mil expansion to potentially reduce the capex development costs of the proposed expansion.

25.2 Opportunities

25.2.1 Geotechnical

The following opportunities have been identified to the open pit design:

- **Saprolite Design Angle** – Based on the current behaviour of the saprolite slopes, site specific laboratory testing and specific trial inter-ramp slopes can used to determine in saprolite slopes can be locally steepened. Pore water pressures should be monitored in these areas.
- **Data Reconciliation** – As mining progresses, additional data on both the saprolite and the rockmass (such as joint persistence, joint set orientation, etc) can be collected and used to update the geotechnical domain models. The inter-ramp angles may be able to be increased with increased confidence in the geotechnical data and continued site observations.

The following opportunities have been identified to the underground design:

- **Geological Reconciliation** – As additional geotechnical information is collected on the areas near the underground design, ground conditions found to be more competent than the current model will allow for a reduction in support cost.

25.2.2 Open Pit Mining

Additional resources in the mine proximity would enable the open pit mine life to extend. While such resources have not yet been identified, SRK is aware that brownfield exploration is getting underway to see if such opportunities may exist for the near term.

As previously suggested, the Company may be able to source more used equipment than considered in this study and possibly at better pricing than assumed.

25.2.3 Underground Mining

The opportunities available for the underground mines are extensive, given that the commitment to commence underground mining is still some years away. Therefore, with the advantage of time and additional data (as it becomes available from operations and deeper exploration drilling) there exists the opportunity to re-evaluate almost every aspect of the underground mining strategy.

As a minimum, opportunities exist in the following aspects of the underground mines:

- **Mining Method**: re-evaluate the range of applicable mining methods in the context of updated resource model and geotechnical data.
- **Ore Handling**: given that Rory's Knoll deposit is open at depth, there is an opportunity to evaluate alternative ore handling strategies prior to committing to the truck haulage method.

25.2.4 Processing

The ability to blend ores using the soft Saprolitic materials may enhance throughputs beyond those estimated here. If so, an opportunity exists to run low grade Saprolite longer into the mine life with the rock ore from underground, thus extending the mine life and improving overall project economics.

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27 Date and Signature Page

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All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practices.

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