Report to:

BRIGUS GOLD CORP.

Black Fox Project
National Instrument 43-101 Technical Report

Document No. 1188930100-REP-R0001-04

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BLACK FOX PROJECT NATIONAL INSTRUMENT 43-101 TECHNICAL REPORT

JANUARY 6, 2011

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1188930100-REP-R0001-04

REVISION HISTORY

REV. NO	ISSUE DATE	PREPARED BY AND DATE	REVIEWED BY AND DATE	APPROVED BY AND DATE	DESCRIPTION OF REVISION
00	10/12/15	Tim Maunula	Charles Tkaczuk	Tim Maunula	Draft to Client
01	10/12/20	Tim Maunula	Charles Tkaczuk	Tim Maunula	Draft to Client
02	10/12/21	Tim Maunula	Charles Tkaczuk	Tim Maunula	Draft to Client
03	10/12/22	Tim Maunula	Charles Tkaczuk	Tim Maunula	Draft to Client
04	11/01/06	Tim Maunula	Charles Tkaczuk	Tim Maunula	Final Report

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GLOSSARY

UNITS OF MEASURE

Above mean sea level	amsl
Acre	ac
Ampere	A
Annum (year)	a
Billion	B
Billion tonnes	Bt
Billion years ago	Ga
British thermal unit	BTU
Centimetre	cm
Cubic centimetre	cm^3
Cubic feet per minute	cfm
Cubic feet per second	ft^3/s
Cubic foot	ft^3
Cubic inch	in^3
Cubic metre	m^3
Cubic yard	yd^3
Coefficients of Variation	CVs
Day	d
Days per week	d/wk
Days per year (annum)	d/a
Dead weight tonnes	DWT
Decibel adjusted	dBa
Decibel	dB
Degree	°
Degrees Celsius	°C
Diameter	ø
Dollar (American)	US$
Dollar (Canadian)	Cdn$
Dry metric ton	dmt
Foot	ft
Gallon	gal
Gallons per minute (US)	gpm
Gigajoule	GJ
Gigapascal	GPa
Gigawatt	GW
Gram	g
Grams per litre	g/L
Grams per tonne	g/t

Greater than	>
Hectare (10,000 m^2)	ha
Hertz	Hz
Horsepower	hp
Hour	h
Hours per day	h/d
Hours per week	h/wk
Hours per year	h/a
Inch	"
Kilo (thousand)	k
Kilogram	kg
Kilograms per cubic metre	kg/m^3
Kilograms per hour	kg/h
Kilograms per square metre	kg/m^2
Kilometre	km
Kilometres per hour	km/h
Kilopascal	kPa
Kilotonne	kt
Kilovolt	kV
Kilovolt-ampere	kVA
Kilovolts	kV
Kilowatt	kW
Kilowatt hour	kWh
Kilowatt hours per tonne (metric ton)	kWh/t
Kilowatt hours per year	kWh/a
Less than	<
Litre	L
Litres per minute	L/m
Megabytes per second	Mb/s
Megapascal	MPa
Megavolt-ampere	MVA
Megawatt	MW
Metre	m
Metres above sea level	masl
Metres Baltic sea level	mbsl
Metres per minute	m/min
Metres per second	m/s
Metric ton (tonne)	t
Microns	µm
Milligram	mg
Milligrams per litre	mg/L
Millilitre	mL
Millimetre	mm
Million	M
Million bank cubic metres	Mbm^3
Million bank cubic metres per annum	Mbm^3/a

Million tonnes	Mt
Minute (plane angle)	'
Minute (time)	min
Month	mo
Ounce	oz
Pascal	Pa
Centipoise	mPa·s
Parts per million	ppm
Parts per billion	ppb
Percent	%
Pound(s)	lb
Pounds per square inch	psi
Revolutions per minute	rpm
Second (plane angle)	"
Second (time)	s
Specific gravity	SG
Square centimetre	cm^2
Square foot	ft^2
Square inch	in^2
Square kilometre	km^2
Square metre	m^2
Thousand tonnes	kt
Three Dimensional	3D
Three Dimensional Model	3DM
Tonne (1,000 kg)	t
Tonnes per day	t/d
Tonnes per hour	t/h
Tonnes per year	t/a
Tonnes seconds per hour metre cubed	ts/hm^3
Volt	V
Week	wk
Weight/weight	w/w
Wet metric ton	wmt
Year (annum)	a

1.0 SUMMARY

1.1 INTRODUCTION

Wardrop, A Tetra Tech Company (Wardrop) was commissioned by Brigus Gold Corp. (Brigus Gold) to prepare a Mineral Resource and Mineral Reserve Estimate and Feasibility Study, compliant with National Instrument 43-101 (NI 43-101), of the Black Fox open pit and underground gold project (Black Fox) in Timmins, Ontario, Canada. Black Fox is located approximately 10 kilometres (km) east of the town of Matheson, Ontario, Canada along the east-west trending 200 km Destor-Porcupine Fault Zone (DPFZ).

This Feasibility Study is intended for the use of Brigus Gold for the further development and expansion of Black Fox open pit and underground mine operation. This report meets the requirements for NI 43-101 and the Resource and Reserves definitions as set forth in the Appendix to Companion Policy 43-101CP, Canadian Institute of Mining, Metallurgy and Petroleum (CIM) – Definitions Adopted by CIM Council, November 2005.

1.2 GEOLOGY

The Black Fox deposit is described by Prenn (2006) as follows:

"Gold mineralization at the Black Fox deposit occurs in several different geological environments within the main ankerite alteration zone, which has an indicated strike length of over 1000 m and a variable true width ranging from 20 to over 100 m . This mineralized envelope occurs primarily within komatiitic ultramafics and lesser mafic volcanics within the outer boundaries of the Destor-Porcupine Fault Zone. The auriferous zones have several modes of occurrence; from concordant zones which follow lithological contacts and which have been subsequently deformed, to slightly discordant ones which are associated with syenitic sills and quartz veins or stockworks."

For this study, the mineralization is subdivided into three main domains based on the continuity and style of the mineralization. The first is called the "Main Zone" and is delineated by the primary domain of shearing and mineralization. It is broader near surface reaching a maximum true width of 150 metres (m) normal to strike and dip and narrows at depth. It averages approximately 80 m normal to strike and dip and has currently been drill tested to 600 m below surface. Within the "Main Zone", the mineralization occurs along both a foliated fabric cut by discrete shear zones and as stockwork carbonate veining. The second mineralization domain is called the "Flow Zones". This mineralization occurs as numerous sigmoid and lens shaped bodies completely hosted within or adjacent to the "Main Zone". The gold mineralization within these bodies has good geologic and grade continuity. The rock is distinctive with strong foliation, pervasive shearing and can be correlated reasonably well between adjacent drillholes. The third mineralization domain is High Grade (HG) Indicator Zone. This was a probabilistic approach to define the zones of mineralization over 2 grams per tonne (g/t) gold (Au). This HG Indicator Zone was constrained within the Main Zone and overlapped at times on the Flow Zone. Each of the three mineralization domains was modeled independently.

1.3 MINERAL RESOURCE

1.3.1 ESTIMATION METHODS

The Black Fox deposit has been estimated using current block modeling techniques in Gemcom© GEMS 6.2.4. This included proper geologic input, appropriate block model cell sizes, grade capping, assay compositing and reasonable interpolation parameters. The results have been verified by visual review and statistical comparisons between the estimated block grades and the composites used to assign them. The ordinary kriging models have also been validated with alternate estimation methods: Nearest Neighbour and Inverse Distance Weighting. No biases have been identified in the model.

1.3.2 RESOURCE CLASSIFICATION

The Mineral Resources are classified under the categories of Measured, Indicated and Inferred Mineral Resources according to CIM guidelines. Classification of the Resources reflects the relative confidence of the grade estimates, as a function of many factors including primarily; assay data quality, QA/QC procedures, quality of density data, and sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization.

No environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to the author that may affect the estimate of mineral resources.

The resource model blocks were classified into Indicated and Inferred categories based on the level of confidence in the grade estimate for each block. This was accomplished with a combination of two main criteria: the number of drill holes (which in part reflects the number of samples used) and the distance to the nearest of the sample points.

Indicated Resources were categorized based on a minimum of three drill holes when the nearest sample point was less than or equal to 20 m. The remaining blocks were classified as Inferred Resources. No blocks were categorized as Measured.

1.3.3 Resource Statement

The mineralization of the Black Fox Mine as of October 31, 2010 is classified as Indicated and Inferred resources. The classified mineral resources are shown in Table 1.1. The mineral resource is reported at a 0.88 g/t Au cut-off grade for the open pit and at 2.54 g/t Au cut-off grade for the underground. These cut-offs have been developed by mine engineering as outlined in Section 19.

All blocks historically mined underground were depleted and not included in the reported resource. The open pit survey provided by Brigus Gold as of October 31, 2010 has been depleted from the resource statement.

Table 1.1 Resource Statement, Black Fox Mine

Mining Potential	Block Model	Cutoff (g/t Au)	Resource Class	Tonnes	Capped Au (g/t)	Contained Au (koz)
Open Pit	2009A	>= 0.88	Indicated	3,164,200	4.445	452.2
		>= 0.88	Inferred	667,100	2.61	56
Underground	2009A_4M	>= 2.54	Indicated	2,504,800	7.182	579.2
		>= 2.54	Inferred	115,200	5.816	21.5
Total Indicated				**5,669,000**	**5.654**	**1,031**
Total Inferred				**782,300**	**3.082**	**78**

The resource contains an Indicated Resource of 1,031.4 koz and Inferred Resource of 77.5 koz of gold.

Table 1.2 Reserve Statement, Black Fox Mine (at Oct 31, 2010)

Zone	Classification	Recovery %	Tonnes	Au g/t	Au oz
Pit	Probable	95%	3,159,800	3.228	327,920
U/G	Probable	95%	2,936,000	5.933	560,008
Stockpile	Proven		352,068	1.630	18,446
Total	Proven/Probable		6,447,868	4.372	906,375

The Proven/Probable reserve contains 906,375 oz of gold as of October 31, 2010 (Table 1.2).

1.4 Mineral Processing and Metallurgical Testing

The Mill is currently treating 2,000 t/d of Black Fox ore (the range of daily mill feed is between 1,500 and 2,200 tonnes), with gold recoveries around 94%. This is confirmed by the gold shipment receipts from Johnson Matthey.

On occasions when the Mill is operated above 2,200 t/d there is a significant drop in recovery, and it has been confirmed using Metsim® (a commercial Computer simulation program) that this decline in recovery could be prevented by replacing the present primary mill with a Semi-autogenous grinding unit.

A review of limited grinding data, also suggests that crowding of the cyclones is causing some fine material to pass the underflow in the secondary grinding circuit. A more thorough review of the grinding and leach circuit to determine the optimum leach size, may improve throughput sufficiently to treat as much ore as the mine produces, without custom milling or new equipment purchases.

The Mill currently includes:

- Primary and secondary crushing
- Two stage ball milling, with one primary mill and two secondary mills, all in closed circuit
- A pre-leach thickener and two parallel lines of three carbon columns in each
- A leach and carbon in leach circuit providing 24 hours of residence time
- A carbon in pulp circuit extracting 3 tonnes of carbon per day
- Two carbon strip and regeneration circuits (one of 1 t/d capacity and the other of 3 t/d)
- A electro-winning circuit and gold bullion (dore) furnace.

1.5 MINING OPERATIONS

1.5.1 SUPPORT SERVICES

Black Fox mining operations will be supported by various infrastructure systems. Electrical power being supplied by Hydro One will serve as the principle source of energy. Propane will provide mine air heating and liquid fuels will be used primarily for mobile equipment. Compressed air, mine de-watering and electrical power distribution as well as communications have been incorporated into the overall plant design. Modifications and upgrades will be made to support the expansion of mining activities as required.

A sufficient number of quality facilities including change houses, offices, warehousing, security, etc. are available to meet the needs of daily mine operations and support personnel.

Underground refuge stations have been outfitted in accordance with provincial and national requirements. A sufficiently staffed and trained mine rescue team will be in place to support mining activities. Evacuation signalling will be primarily by Stench gas (methyl mercaptine) injecting into the ventilation fresh air streams.

1.5.2 HYDROGEOLOGY

Extensive hydrogeologic studies have been completed at the site to evaluate groundwater seepage into the proposed open pit and underground mine workings. From the available information, it can be concluded that:

- The bedrock associated with the open pit is largely (if not completely) dewatered already as a result of underground mine dewatering operations.
- Groundwater seepage may develop along the western and southern walls of the pit at the contact between the more-permeable overburden/weathered upper bedrock and the underlying less-permeable, competent bedrock.
- These groundwater inflows are anticipated to be within or close to the predicted values, but greater than what has been experienced to date.
- Existing monitoring data suggest that Froome Lake has not been measurably impacted by mine dewatering.
- Existing hydrogeologic data support the conclusion that seepage from Froome Lake to the groundwater system is small and that low permeability lake bed materials are limiting the flow to the groundwater system.
  - Regardless, seepage from Froome Lake to the groundwater system could increase as mining progresses and groundwater level lowering in the overburden increases hydraulic gradients between the lake and the overburden groundwater system.
  - The magnitude of the change in seepage will be limited by the low permeability lake bed materials.
- Groundwater level and lake level monitoring programs will continue to monitor for potential impacts to the shallow groundwater system and nearby lakes.
- The current water management strategy for the open pit is effective and should continue to be effective and consistent with operational cost projections.
- Future pit phases may require use of the designed sumps and pumping systems.

1.6 ENVIRONMENTAL CONSIDERATIONS

The environmental permits necessary for development/operation of the mine and mill include:

- Permit to Take Water (PTTW) – for the withdrawal of surface or ground water quantities in excess of 50 cubic metres (m^3) per day – including mine underground or open pit dewatering.

- Certificate of Approval/Air (CofA/Air) – for treatment and discharge of emissions to air, including management of dust and noise in emissions.
- Certificate of Approval/Industrial Sewage Works (CofA/ISW) – for collection, treatment, and discharge of wastewaters.
- Approved Closure Plan including posting of financial assurance for closure.

The Black Fox operation is fully permitted and maintains separate PTTW's, CofA/ISW, and CofA/Air for the mine and the mill. The mine and mill were previously under separate ownership and the facilities were permitted separately. The permits for these facilities were acquired with the properties and have been amended as the operations have evolved, maintaining the separate permitting for the sites.

1.7 FINANCIAL ANALYSIS

Project economics have been evaluated as of Jan. 1, 2011, and have be reconciled against reserves reported as of Oct 31, 2010. Refer to Section 19, Table 19.1

Black Fox Life of Mine (LOM) capital costs totalling US$74.8million are summarized in Table 1.3. Details supporting the Black Fox budgeted capital costs are discussed in Section 23. Capital costs for 2011 are US$34.9million. Ongoing capital accounts for the remaining mine life. Capital cost estimates are in 2011 US constant dollar terms.

Table 1.3 LOM Capital Costs (US$000s)

Description	2011 Capital (US$000s)	Ongoing Capital (US$000s)	LOM Total (US$000s)
Open Pit Mine	$6,233.1	$5,514.3	$11,747.4
Underground Mine	$28,156.9	$33,980.3	$62,137.2
Environment	$91.4	$114.3	$205.7
Mill	$441.9	$285.7	$727.6
Total LOM Capital	$34,923.3	$39,894.6	$74,817.9

Black Fox LOM operating costs are summarized in Table 1.4. Operating cost estimates are in 2011 US constant dollar terms.

Table 1.4 Cash Operating Cost Summary

Description	Unit Cost (US$/tonne milled)	Unit Cost (US$/tonne mined/milled)	Unit Cost (US$/tonne ore mined)	LOM Average (US$/tonne milled)	LOM Total (US$000s)
Open Pit Mining			$2.60	$14.50	$90,594
Underground Mining			$56.48	$26.47	$165,391
Ore Handling	$5.31			$5.31	$33,180
Mill	$13.83			$13.83	$86,419
Assay Lab		$0.98		$1.93	$12,033
Site G&A		$2.50		$4.91	$30,696
Total				$66.94/t	$418,314
				$502/Au oz	

*Mined/Milled is the total tonnes of the pit plus underground mined plus the total milled.

1.7.1 TECHNICAL-ECONOMIC RESULTS

The technical-economic results are based upon work performed by Brigus Gold Black Fox engineers and consultants and has been audited by Wardrop. All costs are in 2011 US constant dollars. The economic model developed by Wardrop is pre-tax and assumes 100% equity to provide a clear picture of the technical merits of the project.

The Wardrop LOM plan and economics are based on the following:

- A gold price of US$1200/oz for 88% of accountable ounces and US$500/oz for 12% of accountable ounces (Sandstorm Agreement).
- Probable reserves, no resources are included.
- A mine life of 8.55 years, at a designed rate of 730,000 t/a milled.
- An overall average metallurgical recovery rate of 94% Au, over the LOM.
- A cash operating cost of US$66.94/t milled, US$502/oz Au.
- LOM capital costs are budgeted to be US$74.8 million being comprised of US$11.7 million for the open pit, US$62.1 million for the underground mine, US$0.2million for environmental and US$0.7 million for the mill.
- No salvage value is modeled.

The base case economic analysis results, shown in Table 1.5, indicate a pre-tax undiscounted cash flow of US$439.0 million and Net Present Value (NPV) of US$359.4million at a 5% discount rate.

Table 1.5 Technical Economic Results ($000s) (at Jan 1, 2011)

Description	Technical Input or Result
Ore	
Open Pit	
Waste	31,742,315 t
Ore	3,101,515 t
Total	34,843,830 t
Grade	3.213 g/t Au
Contained Gold	320,370 oz
Underground	
Total Development	10,166 m
Ore	2,928,318 t
Grade	5.936 g/t Au
Contained Gold	558,849 oz
Mill	
Ore Treated	
Mill tonnes	6,248,669 t
Ore Grade	4.419 g/t Au
Contained Gold	887,754 oz
Recovered Gold @ 94%	834,488 oz
Revenue ($000s)	
Gross Revenue	$933,241
Refining & Transportation Charges	($1,097)
Net Smelter Return	$932,144
Royalty	$0
Gross Income From Mining	$932,144
Realized Price (Gold)	*US$1118/oz Au*
Operating Cost ($000s)	
Open Pit Mine	($90,594)
Underground Mine	($165,391)
Ore Handling	($33,180)
Mill	($86,419)
Assay Lab	($12,033)
G&A	($30,696)
Operating Costs	($418,314)
	US$502/oz Au
	US$66.94/t milled
Cash Operating Margin	$513,829
	US$616/oz Au
	US$82.23/t milled
Capital Cost	
Open Pit	($11,747)
Underground Mine	($62,137)
	table continues…

Environment	($206)
Mill	($728)
Total Capital	($74,818)
Cash Flow	$439,012
($NPV5_{\%}$)	$359,386

A sensitivity analysis was performed for key economic parameters, which are shown below in Table 1.6. This analysis suggests that the project is most sensitive to gold price. Operating costs are slightly more sensitive than capital costs due to the many operating functions associated with the project.

Table 1.6 Project Sensitivity ($NPV_{5\%}$, US$000's)

Description	-20%	-10%	Base Case	+10%	+20%
Gold Price	$210,506	$284,946	**$359,386**	$433,826	$508,265
Operating Costs	$431,395	$395,391	**$359,386**	$323,381	$287,376
Capital Costs	$373,181	$366,283	**$359,386**	$352,488	$345,591

The breakeven gold price was determined to be US$602/oz.

1.8 RECOMMENDATIONS

Black Fox should continue to be developed. The following are Wardrop's recommendations.

1.8.1 MINERAL RESOURCE

The work by Brigus Gold has been found to follow industry accepted practices. Two areas were identified as opportunities to refine the mineral resource:

- Wardrop recommends further specific gravity analysis by rock types and mineralization styles to confirm that one value is appropriate for the whole model.
- Pitard (2005) conclusions on sampling identified that the drillhole data is likely biased and will likely underestimate the contained gold within the deposit. Wardrop agrees with Pitard's work and recommends that a comprehensive grade control procedure be prepared to address the sampling issues be established for the Black Fox Mine. This will facilitate the delineation of ore and waste and the reconciliation between the resource, reserve and milled grade.

1.8.2 GEOTECHNICAL

Pro-active geotechnical monitoring is recommended for all stages of pit and underground development. The monitoring program should be implemented as a staged approach and include detailed geotechnical and tension crack mapping, as well as a suitable combination of surface displacement monitoring (surface prisms) and piezometers.

Sufficient staffing resources should be allocated to collect, process and interpret the geotechnical monitoring data on a weekly basis or as frequently as required. The timely identification of accelerated movements from surface displacement monitoring and tension cracks will be critical.

Up-to-date reports on the status of highwall stability should be compiled and discussed regularly with operations personnel. These reports will also assist mine engineering staff with their efforts to optimize final pit slopes and improve the effectiveness of the controlled blasting program.

All seeps and springs should be inspected, mapped and photographed. Large-scale structures should be characterized and monitored as they have the potential to develop into tension cracks.

1.8.3 MINERAL PROCESSING AND METALLURGICAL TESTING

One option to reduce the mean particle size entering the leach circuit has been presented by DMA. This is to replace the current “primary” mill with a Semi-Autogenous (SAG) mill, with a wrap-around motor.

This is a viable option, but could incur significant costs, and the plant would need to operate on reduced tonnage while new foundations were installed. Wardrop suggests a more thorough review of the existing circuit.

Mill data collected on Sept 17th suggests that too much of the fines are misdirected to the secondary mills, and that minor changes to the cyclones may be an effective solution.

If this is insufficient then there remains the option of re-drilling the primary mill shell bolt holes, This would permit the installation of heavier liners using larger fastening bolts, and would permit the mill to operate using larger (heavier) grinding balls.

Although the mill site tailings facility has been designed for future expansion, it is recommended that the expansion design be reviewed based upon increased ore milling and tails loading to the structure. This should include a detailed evaluation of facility volumetrics and dam structural design capacity. Furthermore, the Phase 5 water management pond design should be evaluated for stability as well as discharge pumping requirements under peak storm/snow melt events. Based upon these analyses recommendations will be provided to upgrade the facility for the additional loading, if needed.

As discussed in Section 25 of this report, Brigus Gold should consider placing a low permeable liner system below the ore stockpile area at the mill site. A costing analysis should be conducted to evaluate increased gold yield from contact water vs. liner installation cost.

1.8.4 MINING OPERATIONS

Given that the surface mining operations are underway and procedures well established there are no major recommendations to improve the operations. It will be important, however, to monitor the procedure for the safe working of the underground old workings to ensure the risks are minimized. This also has an impact on the dilution and recovery of the ore, so close monitoring and reconciliation of tonnages and grade between mine and mill is particularly important.

Presently, the rock stockpiles and soil overburden stockpile appear stable. However, it should be expected that the rock stockpiles will experience additional load due to the underground mine development. Analyses should be conducted to evaluate the stockpile remaining design capacity and required additional capacity if needed based upon projected waste rock generation. In addition, stability analyses will need to be updated based upon added stockpile tonnage. Best management surface water and erosion controls also will also require re-evaluation and updating if stockpile expansions are required. Unless Brigus Gold plans future expansion of the pit, it is expected that the present overburden stockpile configuration will not be significantly altered and therefore should not require any design or operational changes.

UNDERGROUND MINE DESIGN

Wardrop recommends a review to optimize the underground design, specifically the migration of main accesses – from the hangingwall (in the current design) to the footwall. A trade-off study should be carried out to identify the advantages and disadvantages of either location, plus optimization of the location of a single ramp access to serve both the East and West Zones.

1.8.5 ENVIRONMENTAL CONSIDERATIONS

The dirty waste rock stockpile may need to be expanded due to planned underground mining development. This will result in increased contact water inflow to the holding pond. The holding pond capacity should therefore be reviewed, and based upon future peak load conditions expanded if required. Also, the structural capacity of perimeter dams should be evaluated based upon the additional load profile.

2.0 INTRODUCTION

2.1 SCOPE OF WORK AND TERMS OF REFERENCE

Wardrop was commissioned by Brigus Gold to prepare a Mineral Resource and Mineral Reserve Estimate and Feasibility Study, compliant with NI 43-101, of the Black Fox open pit and underground gold project in Timmins, Ontario, Canada. Wardrop's principle underground area of focus was below the 235 Level. Black Fox is located approximately 10 km east of the town of Matheson, Ontario, Canada along the east-west trending 200 km DPFZ.

The Glimmer underground gold mine operated on the Black Fox property over the period 1997-2001, and produced approximately 211,000 oz of gold by contract milling in either the St. Andrew or Macassa mills. Underground mining extended to depths of approximately 200 m to 215 m below the surface before operations were suspended due to low gold prices in May of 2001.

Apollo Gold Corporation (Apollo) purchased the property from the Exall-Glimmer joint venture in 2002 and began exploration of the property in 2003. The Apollo exploration drilling programs intersected significant gold mineralization in both near surface, and down-dip of the area mined by Exall Resources Ltd. (Exall), as well as along strike.

Apollo Gold commenced mining operations in May 2009. In June 2010, Apollo and Linear Gold Corporation (Linear) merged to form Brigus Gold.

This Feasibility Study is intended for the use of Brigus Gold for the further development and expansion of Black Fox open pit and underground mine operation. This report meets the requirements for NI 43-101 and the Resource and Reserves definitions as set forth in the Appendix to Companion Policy 43-101CP, Canadian Institute of Mining, Metallurgy and Petroleum (CIM) – Definitions Adopted by CIM Council, November 2005.

2.2 QUALIFICATIONS OF THE CONSULTANTS

This Feasibility Study has been prepared by a team of consultants sourced principally from Wardrop and Tetra Tech offices in Canada, United Kingdom, and USA (the Consultants) as well as Python Mining Consultants. These consultants are specialists in the fields of geology, exploration, Mineral Resource and Mineral Reserve estimation and classification, open pit mining, underground mining, geotechnical, environmental, permitting, mineral processing and mineral economics disciplines.

Neither the Consultants, their employees, nor associates employed in the preparation of this report, have any beneficial interest in Brigus Gold. The Consultants will be paid a fee for this work in accordance with normal professional consulting practice.

The individuals who have provided input to this Feasibility Study have extensive experience in the mining industry and are members in good standing of appropriate professional institutions. The key project personnel contributing to this report are listed in Section 2.2.1.

2.2.1 PROJECT TEAM AND RESPONSIBILITIES

The Qualified Persons (QPs) for this report are listed in Table 2.1. A brief description of key contributors to the Project is provided below.

Table 2.1 Key Project Personnel

Company/Office	Name	Discipline
Brigus Gold/Black Fox Mine	Dan Battison, P. Eng.	Underground Mining
Wardrop/Toronto	Philip Bridson, P. Eng.	QP, Project Economics
Wardrop/Toronto	Peter Broad, Eur. Ing., P. Eng.	QP, Metallurgy, Processing
Brigus Gold/Black Fox Mine	Jeff Choquette, B.Sc. Mining	Open Pit Mining
Wardrop/Toronto	Virgil Corpuz, P. Eng.	QP, Underground Mining
Tetra Tech/Albuquerque, NM	Michael Gabora, P. Hg., P. Geo.	QP, Hydrogeology
Tetra Tech/Swindon, UK	Richard Hope, C. Eng., MIMMM, ARSM	QP, Open Pit Mining and Reserves
Brigus Gold/Black Fox Mine	Ryan Lougheed, B.Sc., Env.	Environmental
Wardrop/Toronto	Andrew MacKenzie, P. Eng.	QP, Underground Geotechnical
Wardrop/Toronto	Tim Maunula, P. Geo.	QP, Overall Reporting, Resource Estimation, Geology
Wardrop/Toronto	Velo Mehilli, P. Eng.	QP, Underground Reserves
Tetra Tech/Vancouver	Doug Ramsey, M.Sc., R.P. Bio.	QP, Environmental
Tetra Tech/Phoenix, AZ	Marvin Silva PhD, PE, P. Eng.	QP, Open Pit Geotechnical
Tetra Tech/Green Bay, WI	John Starke, P.E.	Tailings
Wardrop/Toronto	Charles Tkaczuk, P. Eng	QP, Infrastructure and Support Systems
Wardrop/Toronto	Karlis Jansons, P. Eng.	QP, Tailings
Python Mining Consultants	Martin Drennan, P. Eng.	Underground Mining

3.0 RELIANCE ON OTHER EXPERTS

Wardrop's opinions contained herein are based on information provided to them by Brigus Gold throughout the course of their investigations. The sources of information include data and reports supplied by Brigus Gold personnel, as well as documents listed in Section 20.

The Qualified Persons preparing and supervising this report have not relied on a report, opinion or statement of a legal or other expert, who is not a Qualified Person for information concerning legal, environmental, political or other issues and factors relevant to this report.

Wardrop used their experience to determine if the information from previous reports was suitable for inclusion in this report and adjusted information that required amending. Revisions to previous data were based on research, recalculations and information of each of the Qualified Persons. The level of detail utilized was appropriate for this level of study.

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

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 PROPERTY LOCATION

Black Fox is located 10 km east of Matheson, Ontario, along Hwy 101 east and approximately 655 km north of Toronto, Ontario. It is located in the Hislop and Beatty townships, District of Cochrane, in the Larder Lake Mining District 90. The project is centered at 48°30' north (N) latitude and 80°21' west (W) longitude. The Glimmer underground mine, formerly operated by Exall, is located within the property boundaries. Figures 4.1 and 4.2 show the property location and the adjacent mines location respectively (Prenn, 2006).

Figure 4.1 Black Fox Location Map

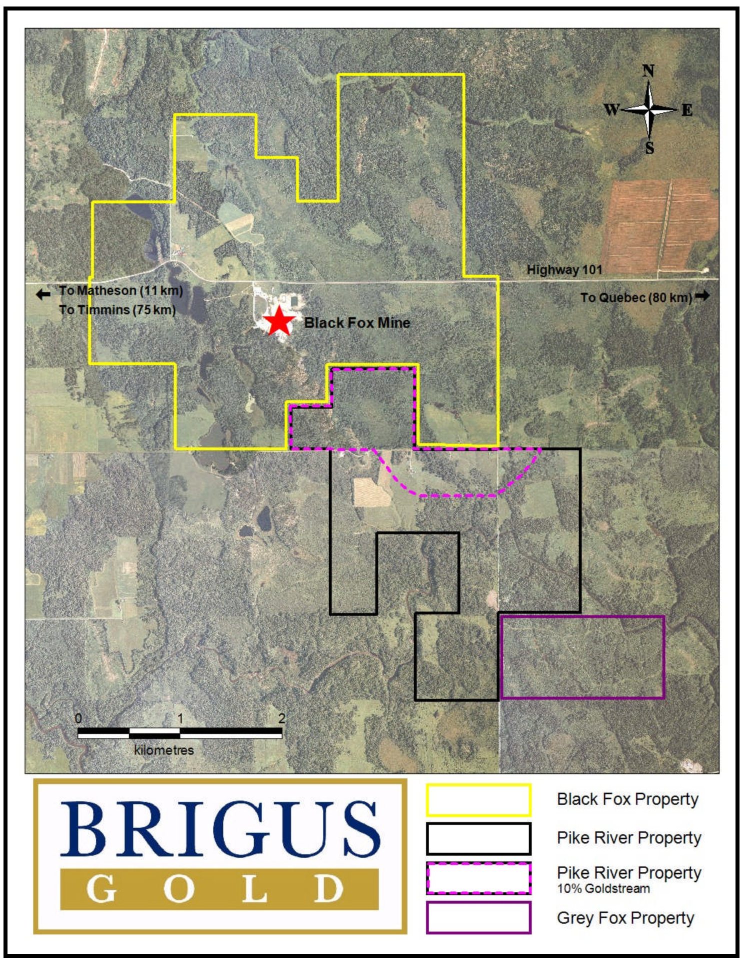


Figure 4.2 Adjacent Mines Location Map

4.2 LAND AREA

The property includes approximately 1,761.41 hectares (ha) of land. Figure 4.3 shows the land tenure for Black Fox and Table 4.1 lists the current land position for the Project.

Figure 4.3 Black Fox Claim Map

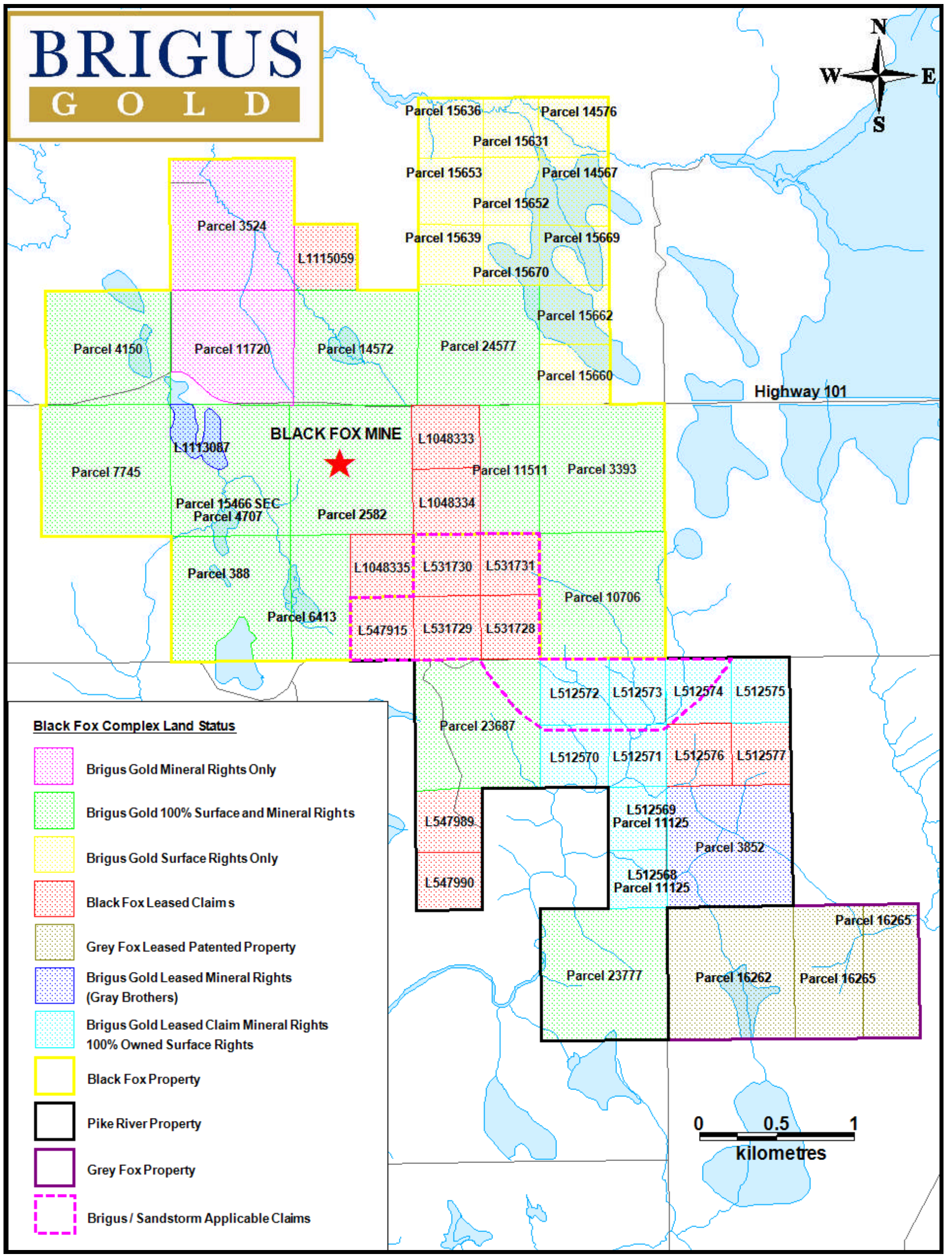


Table 4.1 Current Black Fox Project Property Summary

Township	Pin #	Concession #	Lot #	Parcel	Hectares	Acreage	Status
Surface and Mineral Rights Owned by Brigus							
Beatty	65366-0127	1	6	14572	61.43	151.67	Surf and Mineral owned by Brigus
Beatty	65366-0143	1	8	4150	63.18	156	Surf and Mineral owned by Brigus
Hislop	65380-0531	6	4	10706	68.04	168	Surf and Mineral owned by Brigus
Hislop	65380-0532	6	6	6413	32.60	80.5	Surf and Mineral owned by Brigus
Hislop	65380-0552	6	8	7745	66.83	165	Surf and Mineral owned by Brigus
Hislop	65380-0534	6	7	388	59.33	146.5	Surf and Mineral owned by Brigus
Hislop	65380-0555	6	7	15466	39.95	98.65	Surf and Mineral owned by Brigus
Hislop	65380-0556	6	6	23876	39.89	98.5	Surf and Mineral owned by Brigus
Hislop	65380-0557	6	6	2582	62.95	155.442	Surf and Mineral owned by Brigus
Hislop	65380-0558	6	5	11511	32.62	80.55	Surf and Mineral owned by Brigus
Hislop	65380-0559	6	4	3393	62.87	155.238	Surf and Mineral owned by Brigus
Hislop	65380-0553	6	7	4707	18.63	46	Surf and Mineral owned by Brigus
Beatty	65366-0126	1	5	24577	58.75	145.053	Surf and Mineral owned by Brigus
Beatty	65366-0186	1	6	13005	1.94	4.78	Surf and Mineral owned by Brigus
Hislop	65380-0566	4	4	23777	64.60	159.5	Surf and Mineral owned by Brigus
Total Brigus Owned Surface and Mineral Rights					**733.61**	**1811.383**	
Surface Rights Only Owned by Brigus							
Beatty	65366-0126	1	5	15639	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	15653	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	15636	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	15651	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	15652	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	15670	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	14576	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	14567	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	15669	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	15662	15.92	39.3	Surf. Rights owned by Brigus
Beatty	65366-0126	1	5	15660	15.92	39.3	Surf. Rights owned by Brigus
Hislop	65380-0521	5	4	24023	3.15	7.78	Surf. Rights owned by Brigus
Hislop	65380-0525	5	3	10255	32.60	80.5	Surf. Rights owned by Brigus
Hislop	65380-0499	5	4	11125	32.10	79.25	Surf. Rights owned by Brigus
Hislop	65380-0520	5	4	23687	118.25	292.21	Surf. Rights owned by Brigus
Total Surface Rights Only Owned by Brigus					**360.83**	**599.83**	
Mineral Rights Only Owned by Brigus							
Beatty	65366-0129	1	7	23874	123.27	304.37	Mineral Rights owned by Brigus
Total Mineral Rights Only Owned by Brigus					**123.27**	**304.37**	
Leased Mineral and Surface Rights by Brigus							
Hislop	65380-0489	4	3	16262	64.19	158.5	Leased Mineral and Surface Rights
Hislop	65380-0490	4	2	16265	32.50	80.25	Leased Mineral and Surface Rights
Hislop	65380-0491	4	2	16266	32.50	80.25	Leased Mineral and Surface Rights
Total Leased Mineral and Surface Rights by Brigus					**129.20**	**319**	

Leased Mineral Rights by Brigus							
Hislop	65380-0498	5	3	3852	58.08	143.41	Leased Mineral Rights Only
Total Leased Mineral Rights by Brigus					**58.08**	**143.41**	
Leased Mining Claims by Brigus							
Beatty	65366-0199	1	6	L-1115059	16.22	40.1	Leased MR/SR Claims
Hislop	65380-0636	5	4	L-512572	16.52	40.0	Leased MR Claim
Hislop	65380-0636	5	4	L-512573	16.52	40.0	Leased MR Claim
Hislop	65380-0636	5	4	L-512570	16.52	40.0	Leased MR Claim
Hislop	65380-0636	5	4	L-512571	16.52	40.0	Leased MR Claim
Hislop	65380-0636	5	3	L-512574	16.52	40.0	Leased MR Claim
Hislop	65380-0636	5	3	L-512575	16.52	40.0	Leased MR Claim
Hislop	65380-0636	5	3	L-512576	16.52	40.0	Leased MR Claim
Hislop	65380-0636	5	3	L-512577	16.52	40.0	Leased MR Claim
Hislop	65380-0636	5	4	L-512568	16.52	40.0	Leased MR Claim
Hislop	65380-0636	5	4	L-512569	16.52	40.0	Leased MR Claim
Hislop	65380-0637	5	5	L-547989	16.88	41.69	Leased MR Claim
Hislop	65380-0637	5	5	L-547990	16.88	41.69	Leased MR Claim
Hislop	65380-0638	6	6	L-547915	16.88	41.69	Leased MR/SR Claim
Hislop	65380-0638	6	5	L-531728	16.88	41.69	Leased MR/SR Claim
Hislop	65380-0638	6	5	L-531729	16.88	41.69	Leased MR/SR Claim
Hislop	65380-0638	6	5	L-531730	16.88	41.69	Leased MR/SR Claim
Hislop	65380-0638	6	5	L-531731	16.88	41.69	Leased MR/SR Claim
Hislop	65380-0670	6	5	L-1048334	16.63	41.07	Leased MR/SR Claim
Hislop	65380-0670	6	6	L-1048335	16.63	41.07	Leased MR/SR Claim
Hislop	65380-0671	6	7	L-1113087	9.85	24.33	Leased MR Claim
Hislop	65380-0676	6	5	L-1048333	16.89	41.71	Leased MR/SR Claim
Total Mining Claims Leased by Brigus					**356.42**	**880.11**	

4.3 MINING CLAIM DESCRIPTION

All of the Black Fox claims are current and the required claim fees and work commitments have been completed. All of the claim corners have been surveyed.

4.4 AGREEMENTS AND ENCUMBRANCE

The Project is currently an operating mine. A Closure Plan for Mine Production has been filed for this site. Financial Assurance has been submitted to the MNDM with the Closure Plan.

4.4.1 STAKEHOLDERS AND INTERESTED PARTIES

Stakeholders with authority of some nature at the property will include Brigus, the regulatory agencies, the general public, and non-government organizations (NGOs).

Other Stakeholders include the First Nations of the Abitibi Indian Reserve 70, which is jointly owned by the Abitibiwinn (Québec) and Wahgoshig (Ontario) First Nations, and local private landowners in both Hislop and Beatty Townships. The Abitibi Indian Reserve 70 is located 25 km east of the mine site. Table 4.2 lists the local private landowners described as stakeholders near Black Fox (Dyck, 2007). Any adjoining property not listed in Table 4.2 is crown land.

Table 4.2 Local Stakeholders

Land Description	Stakeholder
Hislop Township	
Parcel # 16617	Paul and Christine Bagordo
Parcel # 4184	Winston and Diana Plant
Parcel # 9385	Winston and Diana Plant
Parcel #1365	Ray Durham
Parcel #10706	Ed Shannon
Resident on Concession 6, Lot 7	John and Gloria Barber
Resident on Concession 6, Lot 6	Winston and Diana Plant
Beatty Township	
Parcel # 3265	1051989 Ont. Inc.
Parcel # 15661	Timmins Forest Products
Residents on Concession 1 Lot 7	George and Evelyn Truax; Joe and Margaret Patterson; Gerald Shannon
Parcel # 23723	Jalbert Logging

Source: Dyck, 2007

4.5 ENVIRONMENTAL LIABILITIES

There are no environmental liabilities at the Black Fox Mine Site.

4.6 PERMITTING AND COMPLETED STUDIES

4.6.1 PERMITTING

A Closure Plan has been developed for the development of the Project, described herein, in compliance with legislation and directives from all pertinent regulatory bodies.

The Black Fox Project currently is permitted under the following approvals:

- Certificate of Approval for Industrial Sewage Works 4-0125-96-006

- Amended Certificate of Approval – Air – (mine heaters and generators) 3505-56R2JP
- Amended Certificate of Approval – Air – (laboratory) 3505-56R2JP
- Permit to Take Water – (mine dewatering) 00-P-6025
- Waste Generator Registration ON2142400.

Upon obtaining the property, Brigus has undertaken to clarify historical permits and obtain new permits required by new or amended legislation.

4.6.2 Completed Engineering and Environmental Studies

Hydrogeological Characterization

A number of investigations have been completed to support the characterization of the groundwater regime in the vicinity of the Black Fox Project. Pump tests on large diameter wells, and monitoring of groundwater levels in exploration holes, were conducted in order to determine the characteristics of the overburden aquifer. Packer testing was also completed on a select number of diamond drill exploration holes to estimate bedrock permeability. The data obtained during these tests has been used to estimate the amount of groundwater that would potentially report to the open pit from the overburden aquifer.

Additional testing has been conducted on selected wells to help approximate in-situ hydraulic conductivity values for each screened interval. A three dimensional, conceptual groundwater model has been developed using the field data obtained to predict the potential effects of mine development activities on the local groundwater and surface waters (e.g., drawdown effects).

Hydrological and Aquatic Habitat Assessments

Hydrological assessments in the past were in large part developed by pro-rating regional flow data to the local watershed areas. Current studies are focusing on developing more accurate estimates of stream flows, runoff volumes and site drainage patterns associated with the existing mine site and future developments. Efforts include detailed watershed mapping initiatives, as well as the development of a stream flow monitoring station on the Pike River and a water level monitoring station in Froome Lake. This information will be crucial in assessing potential adverse environmental effects to the downstream aquatic receiving environment and assisting in storm water management planning activities.

Geotechnical Considerations

A geotechnical investigation program was conducted in support of the project development activities. The program focused on the following areas of development:

1. The open pit perimeter slopes (overburden only).
2. Site buildings and access road foundations.
3. The overburden stockpile.
4. The waste rock stockpiles.
5. The tailings impoundment.

With respect to foundations for buildings and other structures, site services and access roads, geotechnical investigations have been completed for foundation types, bearing capacities at certain founding elevations, excavation conditions, bedding requirements for services, and access road granular thickness design.

Planning for the excavation of the overburden in the pit area (pit stripping) has been considered for the type of overburden, the location of the groundwater table, safe slope configurations, as well as run-off collection and management.

Geotechnical investigations in the vicinity of the waste rock and overburden piles has been assessed and clarified for sub-grade preparations for the placement of material to ensure long term stability. This information has been used to design the existing travel routes, lift heights, and slope configurations.

WASTE CHARACTERIZATION STUDIES

A comprehensive geochemical characterization of all mine waste materials has been completed to support the development of an integrated water and waste management plan for the site. In developing the mine model, waste and host rock materials have undergone a comprehensive geological classification to ascertain the total volumes of materials that will be generated. Representative samples from each type of waste material were selected and tested for their acid generating and metal leaching potential as per the relevant guidance documents.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 ACCESS

Black Fox is located 10km east of Matheson, Ontario and 65 km east of Timmins, Ontario, Canada. Access is via Highway 101 East, which crosses the Black Fox claim block at the properties center from east to west. The mine site and its facilities are located on the south side of Highway 101 East. Supplies and services are available in Matheson or Timmins and can be delivered with a 12-hour turnaround. The primary industries are forestry and mining, and Black Fox is located in a well-established mining camp. Because of this, mining and exploration personnel as well as equipment can be found locally for projects in the area.

5.2 CLIMATE

Temperature ranges from 20 degrees Celsius (°C) to 33°C during the summer months and -30°C to 10°C during the cooler winter months of October to May. The average precipitation is 873.4 millimetres (mm) per year and ranges between 44.5 mm in February to 100.1 mm in July. Rapid melting of accumulated snowfall can produce local flooding on the property for short periods during the spring months. Average monthly wind speeds for the region are 11 to 15k m/hour (hr) (Dyck, 2007). Past operations at the property have not been affected by weather. The surface at Black Fox is mainly agricultural land with secondary growth of poplar and willow shrub.

5.3 PHYSIOGRAPHY

The Black Fox property area is predominantly agricultural land with a mature willow shrub, poplar, black spruce, and white birch forest located to the south and eastern borders of the property. The region is characterized by outwash deposits from continental glaciation including raised beaches, flat clay pans and eskers. Relief includes rock knobs and ridges (Prenn, 2006; Dyck, 2007).

Surface waters include lakes, rivers, and their associated habitats. Lakes include Froome Lake located 0.7 km west of the mine, Leach Lake located 1.4 km northwest of the mine and Lawler Lake located 1.7 km south. Two other lakes, Salve located

5.2 km north and Nickel located 5.9 km north, form the headwaters of the Salve Creek. Salve Lake is designated as a Forest Reserve and Recommended Conservation Reserve (Dyck, 2007).

The property is located on the Salve Creek and Pike River watersheds, which are both tributaries of the Black River. The Black River flows north into to the Abitibi River which in turn flows into the Moose River. The Moose River ultimately flows into James Bay (Dyck, 2007). The Black Fox property has low to moderate topography with elevation ranging from 295 to 330 m above mean sea level (amsl) (Prenn, 2006).

5.4 Local Resources and Infrastructure

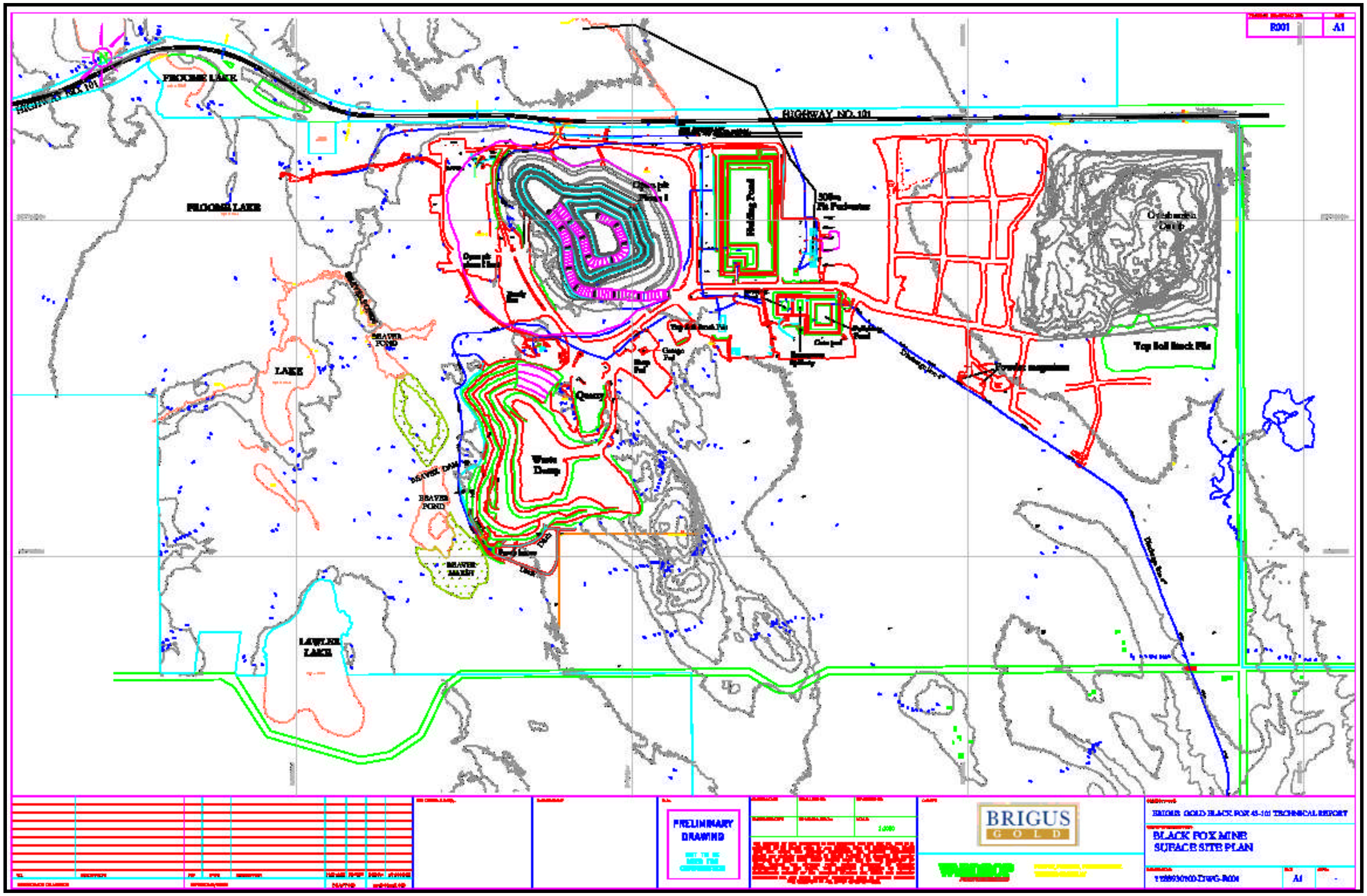
The infrastructure of the Black Fox Project consists of Highway 101 East, which is adjacent to the project site and facilities. The existing surface site facilities, shown in Figure 5.1, consist of the following infrastructure:

- A trailer complex, complete with administration office, mine dry facilities, geological/engineering offices, and shower/washroom facilities.
- Site access road.
- A septic tank and tile field.
- A drilled fresh water well and associated pump house facility located at the east side of Froome Lake, 1500 m long pipeline for the taking of fresh water for showers/toilets and drilling purposes.
- A compressor station.
- A core log shack.
- A maintenance shop.
- Diesel fuel storage tank.
- An approved mine water treatment system, consisting of a holding pond, settling pond, polishing pond, and a spillway pond (for emergency discharge purposes).
- An acid addition building (where sulfuric acid and ferric sulphate are added for pH adjustment and arsenic removal, respectively), associated with the mine water settling pond treatment system.
- A 2.7 km long, 150 mm diameter HDPE pipeline, to Pike River, associated with treated mine water effluent.
- A downcast fan, with mine air heater, and a propane tank.
- A main hydropower line to the on-site 5,000kVA transformer substation and distribution lines.
- A mine portal.

- Waste rock and ore storage pad areas.

Much of the old infrastructure was upgraded in 2010 to facilitate mine development. New infrastructure to support open pit and underground mine operations include buildings and structures, truck shop, laboratory, administration building, the firewater / freshwater pump house, and ancillary buildings.

Figure 5.1 Site Infrastructure



5.4.1 Power Supply

The plant is being fed from an existing 27 kilovolt (kV) power line to the plant site. Power will be distributed at the plant site from this 27 kV power line, a 5 mega volt ampere (MVA), 4,160 volt (V), 3 phase, 60 hertz (Hz) distribution transformer will be installed. The system is being upgrade to adequately handle the new loads.

5.4.2 Water Supply and Distribution

Fresh water is being supplied from a fresh water well.

Fire water will be pumped from the maintenance facility which will source water from the mine polishing pond.

The underground mine water is currently being pumped at the rate of 25 to 35 m^3 per hour Mine run off is anticipated to average above 16 m^3 per hour. Underground mine water reports to the mine holding pond for recycling and or discharge through the Black Fox water treatment facility. Excess water will be treated during the spring and summer months for discharge to the environment through the water treatment system.

5.4.3 Fire Protection

Firewater will be fed by an electric firewater pump with a diesel backup pump in the event of a power failure. Firewater will be delivered to the maintenance shop via buried distribution piping, while the administration complex is supplied with fire extinguishers. The fresh water discharge connection is at an elevation above the tank bottom and ensures the remaining volume will be available for firewater purposes. Municipal fire department is located within 10 km of the Black fox Project.

5.4.4 Access Roads

The Black Fox Project is located 10 km east of Matheson, Ontario, Canada on Highway 101, which runs through the middle of the property north of the ore body.. The property is also contacted by two other roads: Hislop 2 Road to the east and Hislop 6 Conc to the south.

5.4.5 Security

Access to the mine and mill sites, including the pit and tailings impoundments, is limited. Security gates and guardhouses are positioned on the main access roads at the entry points to the project areas. Entry is controlled by a guardhouse operated 24 hours per day, 365 days per year.

The security team's responsibilities include maintaining a constant, 24/7 presence at the site access guardhouse, performing roving patrols around the site, and performing plant security and loss protection.

6.0 HISTORY

6.1 OWNERSHIP HISTORY

The property was first explored by Dominion Gulf in 1952 and then by Hollinger in 1962. In 1988, Glimmer Mine Inc. put together the property package using a combination of crown and private lands. In 1989, Noranda Exploration Company Ltd. (Noranda) entered into a joint venture agreement with Glimmer. As a result of this agreement, Noranda held a 60% interest in the property. During their ownership, Noranda merged with Hemlo Gold Mines Inc. (Hemlo). Exall purchased the property from Hemlo in April 1996, obtaining approximately 60% interest in the property with Glimmer retaining 40%. Apollo Gold acquired a 100% ownership in the fall of 2002 and renamed the property “Black Fox” (Prenn, 2006). In June 2010 Apollo and Linear Gold Corporation (Linear) merged to form Brigus Gold.

6.2 EXPLORATION HISTORY

6.2.1 DRILLING

The first drilling on the property was done by Dominion Gulf in 1952. Hollinger next tested the area in 1962 near the diabase dikes located in the easternmost part of the property. Between 1989 and 1994, Noranda, and later Hemlo, completed eight surface diamond drill programs with a total of 28,014 m of drilling in 143 drillholes. The result of these drilling programs was the definition of an intensive grouping of ore zones in two areas of the property. These ore zones were all within 250 m of the surface. Some high-grade intercepts, including abundant visible gold, were recovered during the drilling program. Between 1995 and 1999, Exall completed another 142 surface diamond drillholes, as well as 708 underground diamond drillholes with mine development (Dyck, 2007).

Eight hundred and ninety-six (896) diamond drillholes were completed by Apollo between 2002 and 2008. Table 6.1 lists the drilling by company and type.

Table 6.1 Black Fox Property Drill Summary

Company	Period	Type (All Core)	Number	Meters
Noranda	1989-1994	Surface	143	28,014
Exall	1995-1999	Surface	142	21,289
Brigus	2002-2007	Surface	500	146,684
Subtotal		*Surface*	*785*	195,987
Exall	1996-2001	Underground	708	61,204
Brigus	2004-2007	Underground	396	78,650
Subtotal		*Underground*	*1,104*	*139,854*
Total		Black Fox	1,889	335,841

6.2.2 Apollo Diamond Drilling and Logging

Norex Drilling International from Porcupine, Ontario, completed most of the surface drilling at Black Fox for Apollo. The holes are typically NQ diameter core unless conditions require a reduction in core size (SRK, 2008).

6.2.3 Mapping and Geophysics

Noranda first performed detailed geological mapping of the property and much of the surrounding area in 1989. This data has provided a very good base of information from which subsequent workers have determine structural trends and location of the most favorable stratigraphic units (Dyck, 2007).

The property has had a number of different geophysical surveys completed by different previous owners in combination with various drilling programs. In conjunction with Noranda's 1989 drilling program, a total field magnetic survey over most of the property was conducted by Exsics Exploration Ltd. Noranda also had Lamontagne Geophysics Ltd. complete an Inductive Source Resistivity survey and R.S. Middleton Exploration Services conduct a conventional IP survey over portions of the property at that time Additional IP surveys were completed in 1997 for Glimmer by JVX Ltd. This later survey was limited to the area adjacent to the mine (Dyck, 2007).

Exploration was also conducted using geological, magnetic and gradiometer surveys conducted by the University of Toronto Electro-Magnetometer (UTEM) survey, and a limited induced polarity (IP) survey (Prenn, 2006).

The highly magnetic anomalies have assisted in the mapping of the basalt and ultramafic units on the property. In addition to this, low magnetic trends may be indicative of hydrothermal alteration that altered the magnetic qualities of the surrounding rocks (Dyck, 2007).

6.3 Historic Resource and Reserve Estimates

The historic resource estimates are summarized in Table 6.2. All of the resource estimates include reserves, except the 1998 estimate, which is resource only and does not include the 1998 reserves listed in Table 6.2. Table 6.3 summarizes the historic reserve estimates made on the Black Fox deposit. The historical reserve and resource estimates performed before the MDA 2006 estimates, pre-dates the development of NI 43-101 reporting guidelines and was not estimated in compliance with NI 43-101 procedures.

Table 6.2 Historic Resource Estimates

	Measured			Indicated			Total Measured and Indicated			Inferred			
Year	kt	Grade g/t Au	koz Au	kt	Grade g/t Au	koz Au	kt	Grade g/t Au	koz Au	kt	Grade g/t Au	Koz Au	Estimator
1994							727	11.3	264.1				Hemlo (Jarvi)
1996							551	11.52	204.1				Roscoe Postle
1996							678	11.3	246.3				Roscoe Postle
1998	44	4.84	6.8	154	5.58	27.6	198	5.42	34.5	382	10.33	126.9	Exall
1999	410	7.27	95.8	796	8.2	209.9	1,205	7.88	305.3	274	5.96	52.5	Exall
2000	586	6.93	130.6	1,022	7.36	241.8	1,608	7.2	372.2	381	6.65	81.5	Exall
2001	268	4.09	35.2	566	4.93	89.7	833	4.66	124.8	353	7	79.4	Exall
2006										7,854	4.89	1,234.8	MDA
2007				5,933	7.04	1,342.7	5,933	7.04	1,342.7	4,185	6.39	859.3	SRK
2008				6,500	6.90	1,441.0	6,500	6.90	1,441.0	3,500	6.62	744.9	SRK

Note: All resources include material reported as reserves except 1998 which is in addition to reserves, Roscoe Postle audited all Exall Estimates
k - kilo

Table 6.3 Historic Reserve Estimates*

Year	Proven			Probable			Total Proven & Probable			Estimator
	kt	Grade g/t Au	Koz Au	kt	Grade g/t Au	Koz Au	kt	Grade g/t Au	Koz Au	
1996				499	11.14	178.7	499	11.14	178.7	Canadian Mine Development (Feasibility)
1996				477	10.7	164.1	477	10.7	164.1	Bharti Engineering Associates
1996				621	11.6	231.6	621	11.6	231.6	Roscoe Postle
1997				665	12.9	275.8	665	12.9	275.8	Roscoe Postle
1998	330	9.88	105	488	10.32	161.9	818	10.14	266.7	Exall
1999	284	8.48	77	553	9.5	168.9	837	9.15	246.2	Exall
2000	422	7.82	106	560	8.93	160.8	981	8.45	266.5	Exall
2001	303	8.45	82	475	9.21	140.7	778	8.92	223.1	Exall
2006				3,063	4.56	449.1	3,063	4.56	449.1	MDA
2007				4,470	6.99	1,004.5	4,470	6.99	1,004.5	SRK
2008				6,460	6.38	1,324.2	6,460	6.38	1,324.2	SRK

*All resource estimates prior to MDA in 2006 are historical and were not reported to NI 43-101 compliance.

6.4 PRODUCTION HISTORY SUMMARY

Ore mined from Black Fox was custom milled from 1997 through September 1999 at the St. Andrew Goldfields Stock Mill located 34 km from the mine. From October 1999 through May 2001, ore was milled at Kinross Gold's Macassa facility in Kirkland Lake, subsequent to mineral tests carried out by Lakefield Research and other metallurgical laboratories. These mills used cyanidation of the whole ore to process the ore. Testwork has indicated that gravity pre-concentration may improve gold recovery (Prenn, 2006).

Black Fox was formally owned and operated by Exall. The previously estimated ore reserves were 3.1 million tonnes (Mt) with a grade of 4.6 g/t Au (449 koz Au) all from open pit mining (Prenn, 2006). The open pit total waste is 47.2 Mt of waste rock and overburden material with an equivalent overall strip ratio of 15.4 waste:1 ore. The underground ore resources (below 9,815 m) were 1.6 Mt with a grade of 8.1 g/t Au.

Table 6.4 summarizes the reported gold production of 210.8 koz from the Black Fox property, with the grades required at 100% recovery.

Table 6.4 Black Fox Project Production Summary*

Year	kt	Grade g/t Au	Koz Au
1997	194	6.79	40
1998	309	6.67	64
1999	259	5.82	48
2000	255	5.82	46
2001	82	4.81	12
Total	**1,099**	**5.97**	**211**

*Actual reported production.

Exall mined portions of the deposit from the bottom of the crown pillar to the 225 m level (measured vertically 225 m below the surface) using conventional underground mining methods including jumbo drills, diesel load haul dump (LHD's) loaders and haul trucks in a random room and pillar method. The limited amount of surface or underground core drilling that was completed by Exall did not allow for detailed mine planning, subsequently the daily mining production planning was determined by management and geological decisions at the face before each round was mined as ore or waste.

Comparing the reserves estimated in Table 6.3 to production in Table 6.4 shows that the grade and tonnage estimates are not very close to the actual production of about 1.1 Mt with and average grade of approximately 6 g/t Au. The estimates between 1996 and 1997 show a range from 162 koz to 275 koz Au. In 2001 the reserve estimate was 140 koz, most of which is still in the ground. All of the historic reserve estimates show higher grades and less tonnes than were actually mined during historic production (Prenn, 2006).

6.5 OPEN PIT MINING SUMMARY

Current mining of the open pit started on March 16, 2009 after five months of extensive overburden stripping. The overburden stripping was performed by Leo Alarie and Sons Contracting and was substantially complete by July 2009. In all 2.1 million m^3 of till overburden material was stripped as part of the Phase 1 open pit (1.9 million m^3 by the contractor) and 1.1 million cubic meters have been stripped year to date October 2010, as part of the Phase 2 open pit by Brigus Gold crews.

Mining in the open pit has progressed ahead of schedule with an expected completion date of December 2010 for the Phase 1 open pit. The final bench of the Phase 1 pit is expected to be the 9907 Elevation.

A summary of mining in the open pit is included in Table 6.5.

Table 6.5 Open Pit Mining to October 2010

	2009	2010	Total
Ore Tonnes	630,514	739,004	1,369,518
Grade (g/t)	3.47	3.22	3.33
Waste Tonnes (No Till)	3,764,541	4,911,729	8,676,270
Mined Tonnes (No Till)	6,072,550	7,119,760	13,192,310
Strip Ratio (No Till)	6.0	6.6	6.3

Note: 2010 tonnes YTD October

6.6 UNDERGROUND MINING SUMMARY

The restart of mining in the underground occurred in June 2010. The mine had been on care and maintenance since diamond drilling was completed in 2006. Mining commenced with the rehabilitation of the existing ramp system and development of the east and new surface ramp system off the existing 235 Level. A new 235 m long 4.5 m diameter ventilation raisebore was completed in August of 2010.

Mining as of October 2010 includes 1,129 m of ramp and access development, 109 m development in ore and 285 m of raise development. As well, 1,095 tonnes of ore have been mined to date.

7.0 GEOLOGICAL SETTING

7.1 REGIONAL GEOLOGY

The Black Fox deposit (formerly the Glimmer Mine) is located east of the city of Timmins in northeastern Ontario located on the DPFZ. The DPFZ has a strike length of about 200 km, and many of Ontario's gold mines are located on or near the DPFZ.

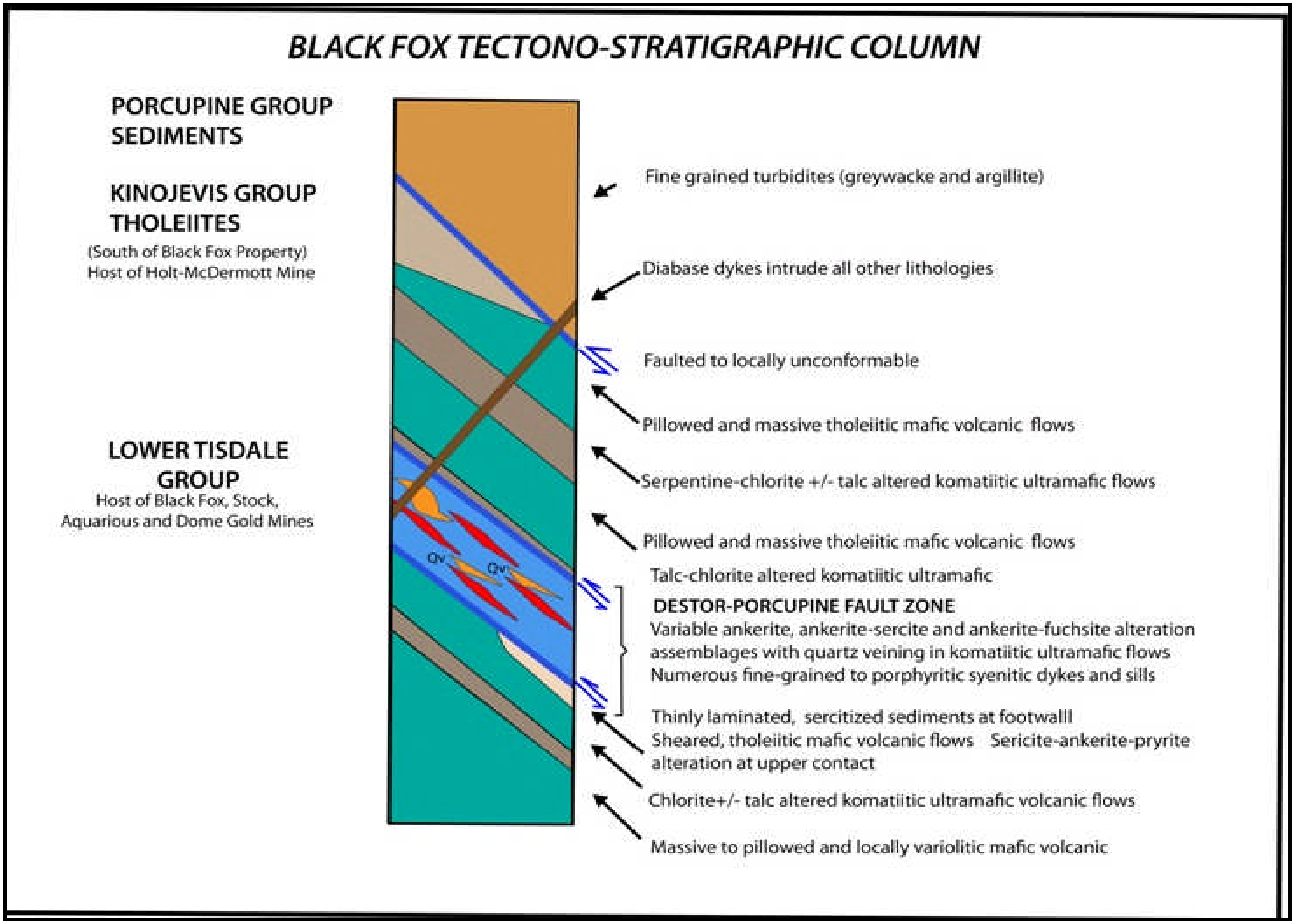
7.1.1 LITHOLOGY

The Black Fox property is located within Precambrian age metavolcanics and metasedimentary rocks of the Abitibi Greenstone Belt. This is one of the world's largest Archean greenstone belts believed to have formed by a complex history of paired arc volcanism and back arc sediments subsequently deformed during continental collision. The area hosts five main rock groups, most of which have tectonic contacts of varying intensity. These include:

- Blake River Group
- Kinojevis Group
- Stoughton-Roquemaure Group (Black Fox Host Units)
- Hunter Mine Group
- Porcupine Group.

The Blake River Group consists of calc-alkalic basalt, andesite, dacite and rhyolite flows and tuffs. It is the youngest of the volcano-sedimentary rocks and stratigraphically overlies the Kinojevis Group. The Kinojevis Group is a sequence of iron rich tholeiitic volcanic rocks that occur on both sides of the Blake River synclinorium. The Stoughton-Roquemaure Group stratigraphically underlies the Kinojevis Group and is a mixture of ultramafic to basaltic komatiite lavas and Mg-rich tholeiitic basalts that host the Black Fox gold zones. This is underlain by calc-alkalic rocks of the Hunter Mine Group. The Hunter Mine Group consists primarily of calc-alkalic pyroclastic and flow rocks in the dacite-rhyolite compositional range. The Porcupine Group of wacke, siltstone and argillite sediments are the youngest in the region. They are separated from the above mentioned volcanic groups by a major fault contact interpreted to have once been a thrust fault. This group lies predominantly north of the Black Fox property. Pre- to syn-kinematic granitic rocks occur throughout the section, cross-cutting all older lithologic units (Hoxha and James, 2007). The tectono-stratigraphic column of the Black Fox area is shown in Figure 7.1.

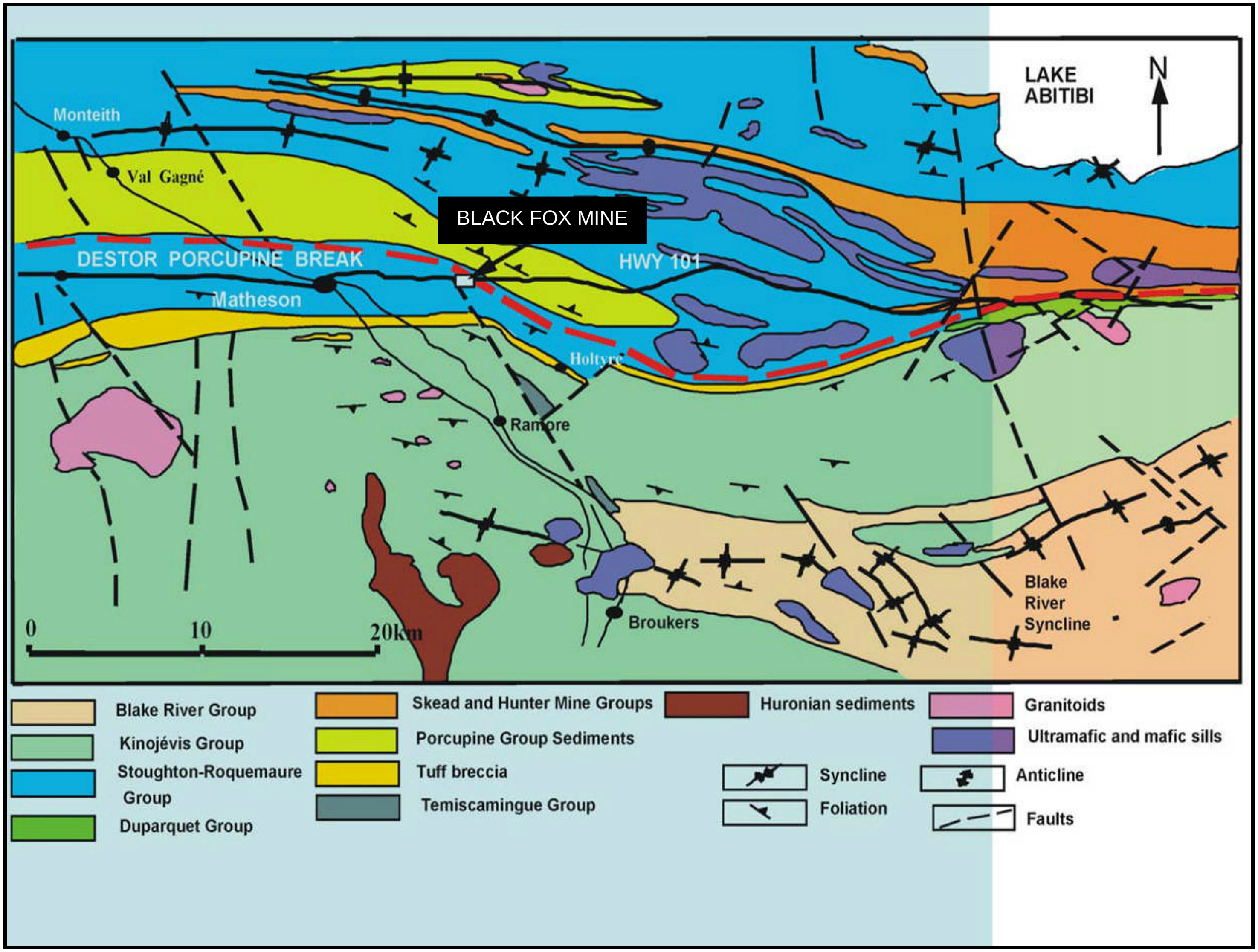
Figure 7.1 Black Fox Tectono-Stratigraphic Column



7.1.2 STRUCTURE

The Black Fox property is situated within a deeply rooted ductile shear zone accompanied by large-scale isoclinal folds. The mineralization is situated on the southern limb of a regional anticline and on the northern limb of the Blake River Syncline. At Black Fox, the axial plane of the syncline strikes roughly NW-SE. The Black Fox deposit is located within the DPFZ. It was first recognized in the early 1900's with the discovery of gold deposits in the Timmins area. The DPFZ extends for over 200 km, from Timmins in the west, to the Duparquet area of Quebec to the east and hosts many of Canada's richest gold mines. The DPFZ hosts gold mineralization comparable to the Cadillac Break to the South and Casa Berardi Fault Zones located to the North. These regional fault fabrics typically strike east to southeast and dip to the south. They are deeply rooted structures that likely penetrate to the mantle, as indicated by the associated ultramafics of the DPFZ and the syenites of the Ross Mine Syenitic Belt (RMSB). Zones of intense hydrothermal alteration measured in thousands of feet are locally associated with these belts. These types of deep-rooted faults are considered to be the main channel way for the upward migration of deep fluids. The main structural feature on the Black Fox property is the intersection of the DPFZ with the RMSB (Hoxha and James, 2007). Figure 7.2 illustrates the regional geology of the area.

Figure 7.2 Regional Geology Map (Hoxha and James, 1998)



7.2 LOCAL GEOLOGY

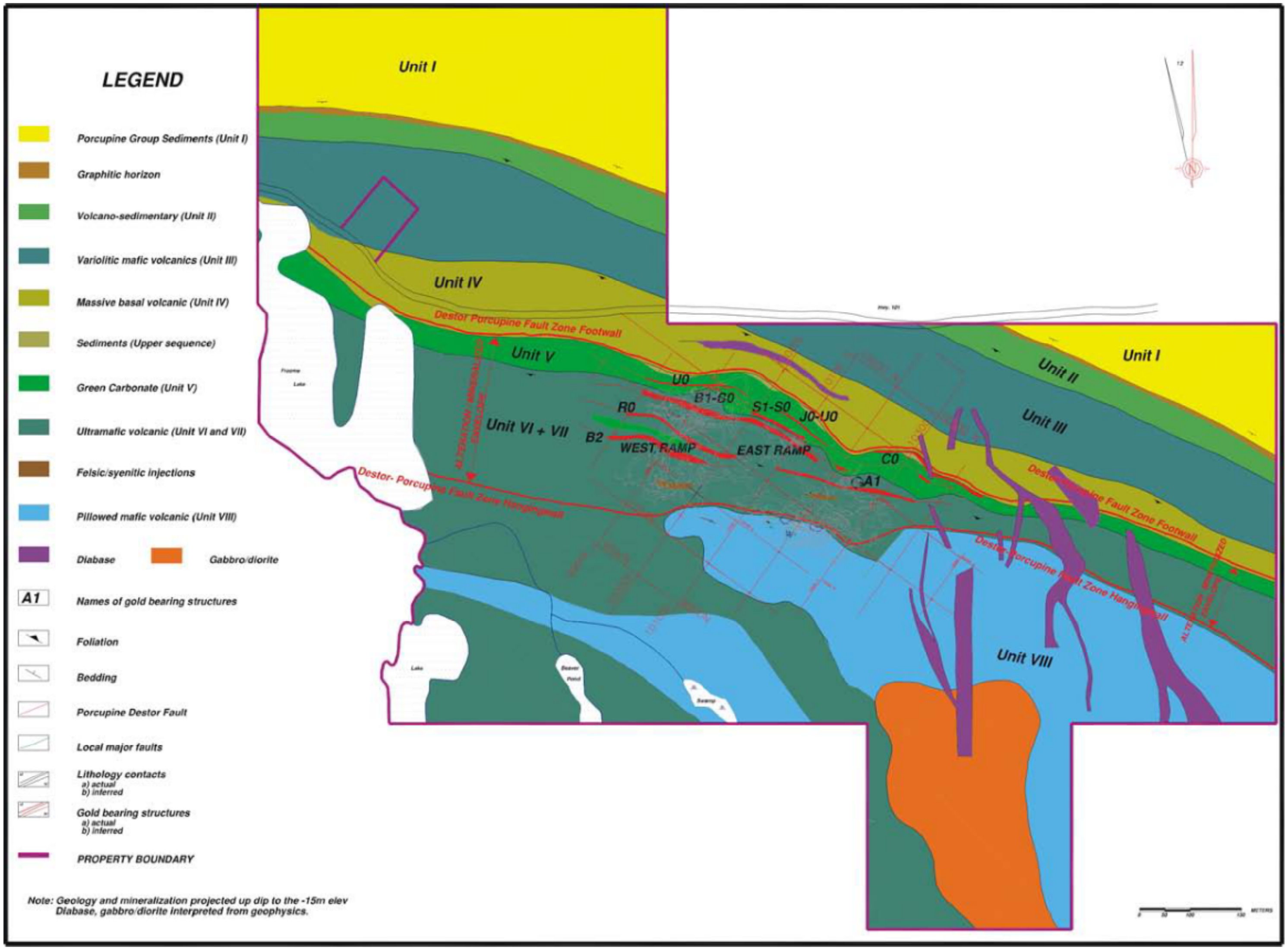
Most of the Black Fox area is rather flat and lacking in outcrops. Pleistocene overburden averages 20 m thick and is composed of lacustrine clay, gravel and till. The main bedrock types consist of variably sheared, faulted, carbonatized and mineralized sequences of komatiitic ultramafic volcanics, belonging to the Stoughton-Roquemaure Group. These strike northwest-southeast across the property, dipping 45 degrees (°) southwest, parallel to the DPFZ. The komatiites are strongly altered to a bleached, light grey-buff color with pervasive ankerite-talc and ankerite-quartz-sericite-fuchsite assemblages. This alteration package is underlain to the north by a thin, fine grained, green greywacke metasedimentary unit, a thick sequence of massive to pillowed tholeiitic mafic volcanic rocks and by the regionally extensive package of argillites and wackes of the Porcupine Group sediments (Hoxha and James, 2007).

To the south, and forming the hanging wall of the main ankerite zone are relatively undeformed very fine-grained, green pillowed tholeiitic mafic volcanics with lesser intercalated black chlorite-serpentine, chlorite and talc-chlorite altered komatiitic ultramafic flows (Hoxha and James, 2007).

Numerous syenitic and feldspar ± quartz porphyry sills and dykes of various ages occur, primarily within the main ankerite alteration zone. They are commonly massive to brecciated, silicified and pyritic with occasional sericite and hematite alteration and a more common black chlorite alteration at the contacts. They vary in color from pink, grey, whitish, yellow, pale green and reddish. Fragments of these dykes frequently occur within the more strongly deformed green carbonate zones and they can contain very high gold grades (Hoxha and James, 2007).

Very narrow to massive, dark green to buff-green mafic dykes and sills commonly occur within the main ankerite zone. They are generally weakly altered and probably post-date much of the alteration and deformation. Diabase dykes are the youngest rocks in the area, occupying very late north-striking crustal fractures. Figure 7.3 illustrates the local geology, and Figures 7.4 illustrate typical cross sections through the deposit.

Figure 7.3 Geological map of Black Fox Mine Property (Hoxha and James, 1998)



7.3 Mine Geology

Surface, underground and exploration drilling has delineated five major rock types in the vicinity of the Black Fox mineralization. These include:

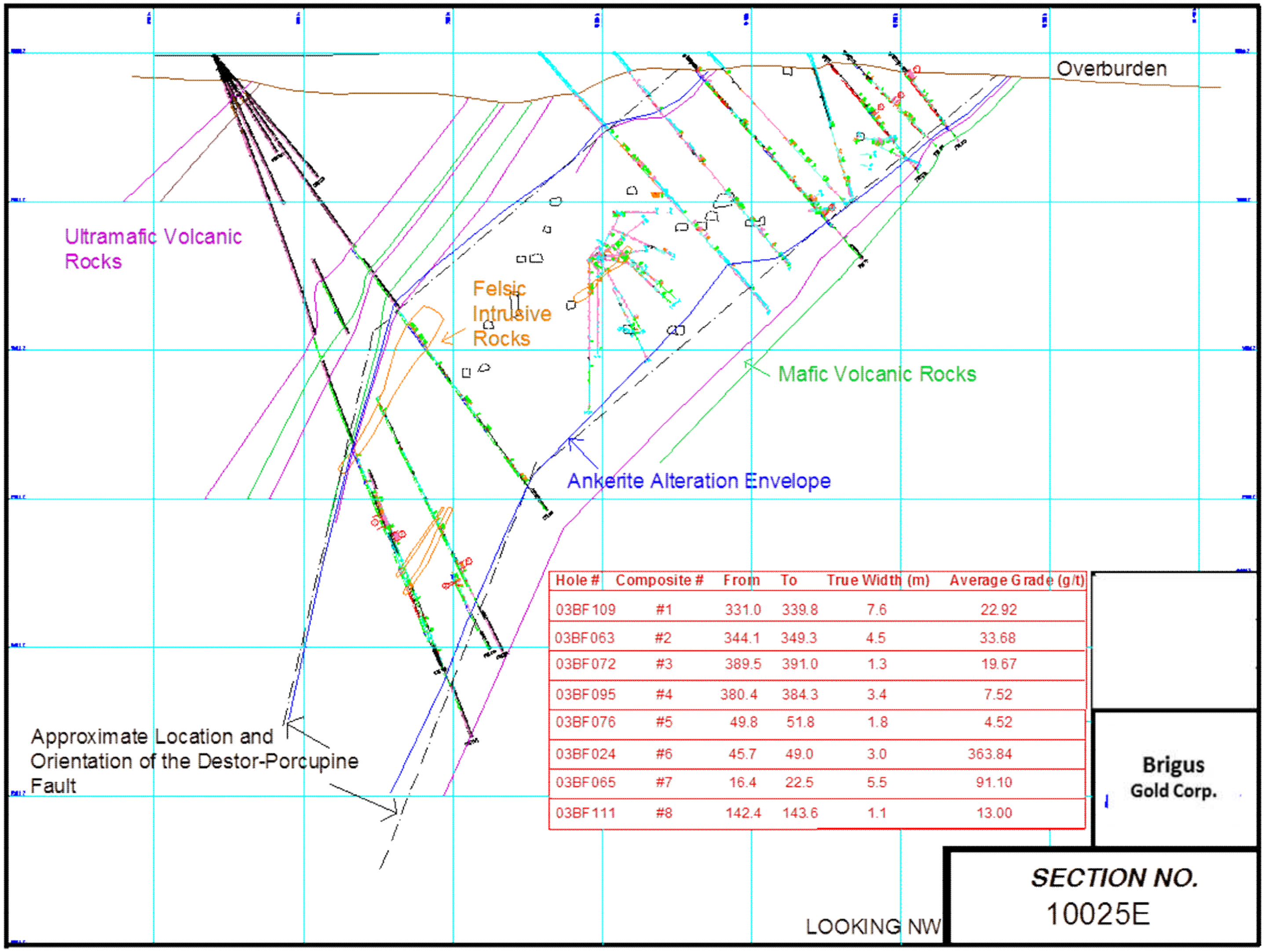
- mafic volcanic units
- metasediments
- green carbonate schist
- ultramafic volcanics
- felsic intrusive units.

7.3.1 Mafic Volcanic Units

The mafic volcanic units are further subdivided into massive mafic volcanics (MV), pillowed mafic volcanics (PMV) and bleached mafic volcanic flows (BMV). The MV and PMV are fine grained typically hosting a significant degree of chlorite alteration. These units occur primarily within the hanging wall of the deposit. In the hanging wall, they are fractured and contain minor amounts of quartz-calcite veins. Where they occur in the footwall, they lack carbonate veining and have more prevalent quartz and chlorite alteration (Hoxha and James, 2007).

The BMV, also known as the “Flow Zones”, is a medium to fined grained, bleached mafic volcanic rock, which is generally located just above the footwall of the mineralization. This unit has weak chlorite and sericite alteration and is associated with fine grained disseminated pyrite. Stronger sericite and pyrite alteration is found near the upper contact of the BMV. Pyrite in this unit is associated with gold. The BMV dips 45° to 55° SW and is moderately foliated. Two quartz vein arrays have been recognized within this unit. The first is a pre-tectonic vein set parallel to the foliation and the second is a series of late veins perpendicular to the foliation (Hoxha and James, 1998).

Figure 7.4 Typical Cross Section



Hole #	Composite #	From	To	True Width (m)	Average Grade (g/t)
03BF109	#1	331.0	339.8	7.6	22.92
03BF063	#2	344.1	349.3	4.5	33.68
03BF072	#3	389.5	391.0	1.3	19.67
03BF095	#4	380.4	384.3	3.4	7.52
03BF076	#5	49.8	51.8	1.8	4.52
03BF024	#6	45.7	49.0	3.0	363.84
03BF065	#7	16.4	22.5	5.5	91.10
03BF111	#8	142.4	143.6	1.1	13.00

7.3.2 METASEDIMENTS

The metasedimentary rocks overlie the BMV and also occur as lens of greywacke (SED) within the green carbonate schists (CGR) as described below. At the top of the BMV, the greywacke layers are interbedded with siltstone. This unit is discontinuous, varies from 0.05 m to 1 m thick and displays graded bedding with stratigraphic tops to the southwest. Its color ranges from pale green to yellowish where well-developed sericite alteration is present. This alteration typically extends over 1 to 2 m wide zone and can be associated with gold mineralization (Hoxha and James, 2007). Greywacke lens occurring within the CGR are yellowish with strong sericite alteration. Generally, they are less than 2 m thick, ranging up to 4 m (Hoxha and James, 2007).

7.3.3 GREEN CARBONATE SCHIST

The CGR, ranges from 15 m to 75 m thick and is continuous along strike and dip across the property. It is characterized by intense ductile and brittle deformation shown by multiple generations of foliation and veining. The CGR host a quartz-ankerite-fuchsite-leucoxene alteration assemblage accompanied by varying levels of retrograde chlorite alteration. This unit contains numerous small bodies and blocks of felsic dikes and sills with a syenitic composition. A complex stockwork of quartz-ankerite veins cross cut the main CGR fuchsite assemblage and the felsic material. This stockwork is accompanied by intense hydrothermal alteration. Locally, grey carbonate fragments, from lapilli sized (~2 mm) to 1 to 2 m angular blocks are found in the CGR. The lapilli sized fragments have been deformed to their current elliptical shape, elongate parallel to foliation. Mineralogy, microscopic texture and structures suggest that the CGR is an ultramafic pyroclastic rock, which has undergone intense ductile deformation. Medium to coarse-grained pyrite is a minor component and is estimated at approximately 1%. Gold occurs as fine-grained free gold located along chlorite slips, as disseminated grains in quartz veins and associated with the felsic dikes (Hoxha and James, 1998; 2007).

7.3.4 ULTRAMAFIC VOLCANICS

The ultramafic volcanic rocks are divided into five units. These include; chlorite-talc ultramafic (CUV), talc ultramafic (TUV), grey carbonate (CGY), silicified grey carbonate (SUV) and ankerite ultramafic (AUV). Generally, the ultramafic volcanics occur stratigraphically above the CGR.

The CUV is dark green, massive, brecciated in places and often magnetic ultramafic rock. This unit does not display pervasive carbonate alteration and carbonate is restricted to late veins and fractures. Tremolite is present, the two primary mineral assemblages are tremolite-talc-chlorite and talc-chlorite-carbonate. Locally, the CUV occurs within the mineralized envelope as a non-brecciated unit and the CUV is not of major economic significance (Hoxha and James, 1998; 2007).

The TUV is pale green-grey, fine grained, marbled with quartz-ankerite fragments and massive ultramafic volcanic rock. It tends to be strongly foliated proximal to shear zones, ranging from 0.3 m to 15 m thick. It is most often associated with the stockwork CGY (Hoxha and James, 1998; 2007).

The CGY is composed of a fine grained, massive matrix composed primarily of magnesite-quartz. Relic outlines of pyroxene and preserved black chromite grains are visible in hand specimen. This unit contains several generations of quartz veining. The CGY is 0.5 m to 2 m thick, generally occurs above the CGR and is bound by talc ultramafic shear zones (Hoxha and James, 1998; 2007).

The intensity of silicification and amount of quartz stockwork is the distinguishing characteristic between the CGY and the SUV. The SUV is very similar in appearance to the CBY, but the SUV is harder due to silicification. Two types of carbonatization-silicification have been observed at Black Fox:

- Impregnation of the original ultramafic volcanic rock by CO_2 and silica-rich fluids throughout the network of micro-fractures and cavities/porosity; and
- Silicification of the altered ultramafic volcanic rock by silica-rich fluid circulating throughout rectilinear centimetre wide extensional fractures associated with shear zones.

The CGY formed by the first process is moderately to strongly fractured, while CGY formed by the second process tends to be massive. Both the CGY and SUV host visible gold and are of economic importance at the Black Fox property (Hoxha and James, 1998; 2007).

The AUV is dark green-brown, fine–medium grained rock composed of a quartz-ankerite-calcite-chlorite assemblage cross cut by quartz-ankerite veining. Chloritization varies throughout this unit with matrix ankerite and calcite alternating downward through the package. Visible gold occurs in highly chloritized area as well as in association with the quartz-ankerite stockwork. The AUV generally occurs above the CGR and is one of the dominant rock types at the Black Fox property (Hoxha and James, 1998; 2007).

7.3.5 *Felsic Intrusive Units*

Many types of felsic intrusive (FI) have been recognized within a number of different lithologies at Black Fox. These range in color from grey to yellowish to reddish brown as a result of different alteration types. Most of the felsic intrusives are fine to medium grained, massive and moderately fractured, but some coarser grained porphyritic bodies have also been observed. Generally, the felsic rocks are discontinuous, lensoidal in shape and aligned with the foliation of the host rock. They are often cross cut by quartz-ankerite stockwork and most are strongly affected by sericite and albite alteration. Varying amounts of fine-grained disseminated pyrite are a strong indication of gold mineralization. Gold occurs as free gold associated with quartz veins. Syenitic pods have been observed in the CGR. These are pink, coarse grained and contain a relatively high concentration of pyrite, at 5-15% they typically have an average gold grade of 15 g/t (Hoxha and James, 1998; 2007).

8.0 DEPOSIT TYPE

The Black Fox mineralization is an Archean age, lode gold deposit located within the Abitibi greenstone belt. The characteristics of this deposit type include; greenstone host rocks and gold-bearing quartz-carbonate veins. The veins occur as two main types. The first are arrays and stockworks along faults and shear-zones with a quartz-carbonate laminated fault-fill. The second are widely distributed extensional veins within carbonatized metamorphosed greenstone rocks. These deposits are typically associated with crustal scale compressional faults with a vertical extent of ≤2 km and limited metallic zoning (Dubé and Geosselin, 2007).

The Black Fox deposit lies along the DPFZ, a major, east-west trending, deep-seated, crustal fault zone. The DPFZ and its numerous splays are associated with many past and current producing gold mines and gold deposits in the Porcupine Camp. The Stock and Aquarius gold deposits are located immediately west of Black Fox and the Holloway and Holt-McDermott Mines are located immediately to the east. Each of these deposits hosts approximately 800 k to 1 million oz gold. The Black Fox deposit is situated midway between two major mines, the Dome-Hoyle Pond and the Holt-Holloway. The Dome-Hoyle Pond deposits located within the same structural regime 65 km west, have shown that gold bearing structures can be traced to 1,600 m below surface where they remain open at depth. The Holt-Holloway Mine, located approximately 45 km to the east has been developed down to 1,200 m below surface.

There are several different styles of mineralization in the deposits associated with the DPFZ. The gold mineralization is structurally controlled, in a variety of geological settings. Alteration types include pyritic ankerite-sericite ± silica-albite altered mafic volcanics, green carbonate fuchsitic altered ultramafic volcanics with quartz stockworks, pyritic, porphyritic to syenitic felsic intrusives and multiple stages of quartz veins with free gold. Much of this variation is found at Black Fox (Prenn, 2006).

9.0 MINERALIZATION

Gold mineralization at Black Fox occurs mainly within an ankerite alteration zone 1 km along strike and 20 m to 100 m wide. This alteration envelope occurs primarily within komatiitic ultramafics and lesser mafic volcanics within the outer boundaries of the DPFZ. In some areas, the auriferous zones are concordant, which follow lithological contacts and have been subsequently deformed to slightly discordant zones that are associated with syenitic sills. Other auriferous zones occur in quartz veins and stockworks discordant to lithology (Hoxha and James, 2007).

The three main styles of gold mineralization observed at Black Fox are:

- Low-sulfide mineralization associated with abundant quartz veining and quartz stockwork within strong ankerite-fuchsite altered ultramafic volcanic rocks.
- Mineralization hosted within mafic volcanic units associated with >5% pyrite and minor to moderate quartz veining.
- Mineralization hosted by silicified felsic dikes.

The first style is low sulfide mineralization occurring within quartz-rich portions of the AUV and CGR rock types. This includes the green carbonate alteration of the “Main Zone”. The typical host is the ankerite-fuchsite altered ultramafic volcanic rocks, commonly found throughout the DPFZ. Quartz veining and quartz stockwork show multiple phases of veining and structural episodes. This is illustrated by cross-cutting veins, chloritic slip surfaces in the quartz veins, and breccia textures. Visible gold is common in high-grade areas (Hoxha and James, 2007).

The second style of mineralization is hosted within mafic volcanic units coded as BMV or MV. This style is referred to as the “Flow Zones”. It is typically associated with >5% fine-grained pyrite, minor to moderate quartz veining and a strong bleaching may be present. The quartz veins are typically parallel to foliation, and visible gold is characteristically absent. This style of mineralization is common in the footwall portion of the DPFZ. It has been tested mainly by the eastern part of the 235 Level underground drilling (Hoxha and James, 2007).

The third style of mineralization is hosted in silicified felsic bodies. These include both quartz-feldspar porphyries and finer grained units which are possibly syenitic in origin. Mineralization in the felsic units is associated with increased silicification, pyrite and some quartz veining all associated with a fracture foliation. In the middle and hanging wall portions of the DPFZ, felsic-hosted mineralization can be correlated from hole to hole over short distances. In the footwall portions, blocks and lenses of felsic material are encountered which do not correlate from hole to hole (Hoxha and James, 2007).

According to Hoxha and James (1998) there have been 15 separate mineralized structures identified within the ankerite envelope. The two main gold-bearing zones of their classification are the A1 at the hanging wall contact and the C0 located at the footwall contact. The other smaller zones located between these two generally have less continuity and width and represent parallel, mineralized shears and faults.

Previous underground mining indicates that sub-horizontal, mineralized bodies located within the "Main Zone", can be up to 15 m thick and very high grade. This suggests that zones of dilation were produced during episodes of structural movements. The majority of the other mineralized zones and quartz veins are 1 to 5 m in width (Hoxha and James, 2007).

At least three generations of structurally controlled quartz veining have been identified in the underground workings. Quartz veins and stockwork zones within the main mineralized envelope are concentrated along shear/fault zones. These structures parallel the main mineralized envelope suggesting they are responsible for the location and formation of the mineralization. The presence of sigmoidal vein structures, multiple quartz injections and re-sheared vein material with chloritic slips indicate complex and repeated structural movements during a cyclic brittle-ductile deformation period. In the quartz stockwork zones, gold mineralization can be erratic possibly related to certain vein sets carrying gold, whereas others are barren (Hoxha and James, 2007).

Prenn (2006) states that "Gold mineralization has been encountered in drill core at depths of 700m below surface to date and, since the host ankerite zone appears to continue further down, it is reasonable to expect that additional mineralization will be encountered with deeper drilling"

10.0 EXPLORATION

Exploration drilling appears to have been first carried out on the Black Fox property in 1952, by Dominion Gulf, and by Hollinger in 1962. Their holes were drilled near diabase dikes located in the eastern most part of the property. In 1988, Glimmer Mine Inc. put together the property package using a combination of crown and private lands. In 1989 Noranda Exploration Company Ltd. entered into a joint venture agreement with Glimmer to earn a 60% interest in the property. Between 1989 and 1994, Noranda, and later Hemlo Gold Mines Inc., completed eight drill programs. In total 27,800 m of drilling was completed, in 142 holes. By 1994 the drilling programs had defined an intensive grouping of ore zones in two areas of the property, within 250 m of the surface. The two areas are approximately 200 m apart, and have a drill indicated resource of 440,000 oz of gold. Some spectacular gold intersections, including abundant visible gold were obtained during the drilling program. In addition to diamond drilling, exploration was conducted by way of geological, magnetic and gradiometer surveys, a UTEM survey, and a limited I.P. survey.

Exall Resources Limited purchased the property from Hemlo Gold Inc. in April 1996, obtaining approximately 60% interest in the property, with Glimmer Resources Inc. holding the remaining portion. Ore from the Black Fox Mine property was custom milled from 1997 through 2001 by either the Saint Andrews or Macassa Mills.

Brigus Gold (formerly Apollo Gold) concluded an option to purchase 100% of the Glimmer Mine (Black Fox Mine) and property title was transferred to Brigus Gold on September 7, 2002. The Black Fox property was purchased from Exall Resources Ltd. and Glimmer Resources. Brigus Gold commenced diamond core exploration drilling from surface in 2003.

This section is partly excerpted from the Technical Report Black Fox Project Matheson, Ontario Canada by N. Prenn of Mine Development Associates, August 14, 2006 and has been standardized to this report.

“The last Brigus exploration drilling in 2007 continued from previous campaigns on 12.5 to 25 m fence lines. The two main emphases included infill delineation of existing mineralization and to explore for areas of new mineralization. In 2004, a 1,250 m long exploratory underground drift (4 m x 4 m) was developed in the hanging wall down to 235 m below the surface, to establish drill stations for an underground drilling program. The underground drilling program consisted of 78,650 m of diamond drilling from 396 core holes. Surface drilling continued and by the end of 2007, Brigus Gold had completed 896 diamond drill holes on the property, totalling 225,334 m. The Brigus Gold drilling consisted of 500 surface drill holes for a total of 146,684 m and 396 underground drill holes for a total of 78,650 m.

During the spring of 2003, Brigus Gold contracted with Quantec Geophysical, Inc., Toronto, Ontario, to complete an IP survey covering the entire property. Lines were spaced every 200 m with 100 m dipole spacing. This survey has shown many chargeability and resistivity anomalies along both the DPFZ and the northwest projection of the Ross Fault. The Ross Fault is the host for the Ross Mine, located approximately 7,500 m southeast of the Black Fox mine. In addition to these, a number of north-south trending anomalies were found. The intersections of these trends are considered to be prime exploration targets. It appears that the data from the earlier Noranda magnetic survey will also be valuable in defining exploration targets. The highly magnetic anomalies may help in mapping the basalt and ultramafic units on the property. In addition to this, low magnetic trends may be indicative of hydrothermal alteration that destroyed the magnetic qualities of the surrounding rocks.

The initial portion of the Brigus Gold surface drilling program concentrated on finding new ore zones below the Black Fox known Resources, along strike and adjacent to the known zones. The targets were the intersection of secondary faults with the DPFZ and also dilation zones within it. The mineralization is so tightly controlled by structures that a hole a few meters away could miss a high-grade zone. Fans of NQ-size drill holes were drilled to test for new ore shoots . The fans were spaced approximately 25m along strike and the intersections of the holes with the DPFZ were planned to be approximately 25m apart. The result of this program was the identification of a number of small, high-grade ore shoots that generally plunge at a 20^o to 40^o angle to the southeast or southwest, along the DPFZ. This is consistent with the intersection of two 45^o to 70^o dipping faults or with a zone of dilation along a fault that has both horizontal and vertical movement. Many of these ore shoots are still open with depth. A near-surface portion of high-grade mineralization was drilled on 12.5 m spacing to improve the definition of this higher-grade mineralization”.

There was no surface or underground exploration drilling in 2008, 2009 and 2010 conducted on the Black Fox deposit by Brigus Gold, except for the completion of 11 NQ-sized condemnation surface diamond drill holes. The condemnation holes have an average depth of 231 m and total to 2,544 m.

In 2010 Brigus Gold contracted Scott Hogg & Associates Ltd. of Toronto, Ontario to carry out a helicopter towed aeromagnetic gradient survey at a 75 m line spacing and contracted Quentec Geosciences Ltd. of Toronto, Ontario to conduct a Titan Deep IP geophysical survey at a 200 m line spacing. The magnetic survey covered the entire Black Fox property and the IP survey filled in areas that were not surveyed by the Quantec IP survey conducted in 2004. The results of both surveys are in the process of being interpreted.

11.0 DRILLING

11.1 DRILLING SUMMARY

A total of 1,889 surface and underground drillholes have been completed on the project by Noranda, Exall and Brigus (formerly Apollo Gold) between 1989 and 2008. Of these drillholes, 896 were completed by Brigus between 2002 and 2008. Table 11.1 lists the drilling by company and type. The Black Fox database includes 185,905 assay intervals.

Table 11.1 Black Fox Property Drill Summary

Company	Period	Type (All Core)	Number	Meters
Noranda	1989-1994	Surface	143	28,014
Exall	1995-1999	Surface	142	21,289
Brigus	2002-2007	Surface	500	146,684
Subtotal		*Surface*	*785*	195,987
Exall	1996-2001	Underground	708	61,204
Brigus	2004-2007	Underground	396	78,650
Subtotal		*Underground*	*1,104*	*139,854*
Total		Black Fox	1,889	335,841

11.2 HISTORIC DIAMOND DRILLING AND LOGGING

Portions of this section were excerpted from Technical Report Black Fox Project Matheson, Ontario Canada by N. Prenn of Mine Development Associates, August 14, 2006 and have been modified and standardized to this report.

11.2.1 NORANDA DRILLING AND LOGGING

Noranda drilled a total of 143 NQ-size diamond core holes between 1989 and 1994. The drillhole have an average depth of 197 m and total to 28,014 m. All holes were surveyed at the collar and had acid etch tests done to measure their dip angle. A Tropari survey was run at the bottom of a few of the deeper holes to measure deviation. The lack of down-hole surveys on many of the deeper holes will influence the accuracy of their location within the zone of mineralization. Core recovery was apparently very good as few recovery problems were listed in the logs. The core was brought to the surface and taken to Noranda's local logging facility. The core was logged for geology and geotechnical parameters.

EXALL DRILLING AND LOGGING

Between 1994 through 1999 Exall drilled 143 NQ-size surface core holes totalling 21,520 m and 707 underground core holes totalling 61,115 m. All of the Exall drill core was NQ-size, unless ground conditions required reduction to BQ. The surface drillholes were down-hole surveyed, however, the underground holes were not surveyed for down-hole deflection, and therefore the bearing and inclination at the collar has to be used for the entire underground drillhole. The core was brought to a surface core area where the geologist logged and sampled it.

Exall resurveyed the collar coordinates of most of the Noranda drillholes, with generally good agreement in the coordinate conversion between the Noranda and Exall data.

11.2.2 BRIGUS DIAMOND DRILLING AND LOGGING

Norex Drilling International from Porcupine, Ontario, has completed most of the surface drilling at Black Fox for Brigus. The holes are typically NQ diameter core unless conditions require a reduction in core size. In general, ground conditions have been very good with average core recovery approximately 95%. The following sections document drilling, chain of custody and logging procedures employed by Brigus. Although no records are available to document the procedure used by the prior operators, there is no reason to suspect they did not follow standard industry practices of the time.

The core is removed from the wire line inner barrel and placed in wooden core boxes. Each box can hold up to 6 m of NQ core. The depth at the end of the core run, along with the length of the run and the amount of core actually recovered, is written on wooden blocks, which are placed in the box at the end of the core run. When the box is full, the drillhole number, along with the beginning and ending depth is written on the outside of the box. A wooden lid is then placed on the box and the box is sealed with wire. The core is stacked at the side of the drill until it is picked up by representatives of Brigus Gold Exploration, Inc. During this time, the core is under the direct supervision of the driller.

The core samples are picked up by Brigus personnel each morning and at various times during the day as necessary. It is loaded into a company truck and taken to the core logging facility on the project site. The core is then unloaded from the truck, the wire ties are removed and the core is inspected for any damage that might have occurred during transport. Each box is then placed in racks within the core logging facility to await logging by Brigus geologists. When the geologist begins logging a hole, a logging form is first computer generated with data regarding the hole ID, depth, date logged, location and the logging geologist. All logging is done electronically with no handwritten data. This eliminates a separate data entry step and the subsequent errors that it can introduce. The geologist moves the boxes of core from the rack to the core logging table. The lids are removed and placed outside for later reuse. The pieces of core are then reassembled, within the box, just as it would have come out of the hole. The core is then measured and that measurement is compared to the core depth markers placed in the box by the drillers. This is documents core recovery and provides a check against any lost or missing core not accounted for by the drillers. All of this data, along with all geological data, are entered into the computer spreadsheet by the geologist. The core is then digitally photographed on the logging bench. This digital record is stored in the computer files for that hole. All of the geological information is backed up on the server daily.

Prior to removing the drill string, the downhole deflection is measured with a Reflex E-Z Shot digital tool (E-Z Shot). Measurements are taken approximately every 50 m down the hole. Occasionally a spurious reading will be obtained near a particularly strongly magnetic rock unit. The geologists review all surveys and any such readings are discarded. As a check, three holes were re-surveyed using a Maxi-bore gyroscopic tool. The Maxi-bore survey duplicated the E-Z Shot survey very well. On average, the E-Z Shot gave readings that were within 3.1% on bearing and 0.4% on dip from the Maxi-bore survey information. All drillholes have their collars located by a licensed surveyor upon completion.

The Brigus drilling program has targeted two main areas of the mineralization. The first is the near-surface area where about half of the surface drillholes were completed. Drilling typically is located along sections oriented 036° azimuth at inclinations of -45° to -50° to provide an alignment oriented nearly perpendicular to the DPFZ.

The second targeted area of mineralization is down dip of the previous drilling. At depth, the DPFZ has the same southeasterly strike, but the dip steepens to an average of -60°. The mineralization still occurs along structural intersections and at dilation zones along the fault. These appear to rake at about -40° to the southeast or southwest. In this area, the shoots tend to be smaller, thinner and less continuous than those encountered near the surface. The drillholes, which test this area, were collared from both the surface and underground. Typically, fans were used so that the structure was tested on 12.5 to 25 m spacing. Eventually, more tightly-spaced drilling from underground platforms will be required to improve the delineation of the mineralization.

11.2.3 *Brigus Diamond Drilling (2008 to 2010)*

Brigus drilled a total of 11 NQ-size condemnation diamond core holes from surface between 2008 and 2010. The drillholes have an average depth of 231 m and total to 2,544 m. Table 11.2 lists the drilling by Brigus.

Table 11.2 Black Fox Drill Summary (2008 to 2010)

Company	Period	Type (All Core)	Number	Meters
Brigus	2008-2010	Surface	11	2544

12.0 SAMPLING METHOD AND APPROACH

This section is partly excerpted from Technical Report Black Fox Project Matheson, Ontario, Canada by Mine Development Associates, August 14, 2006 (Prenn, 2006).

12.1 NORANDA SAMPLING

Little documentation is available describing the details of Noranda's sampling procedures. During the late 1980's it was not a standard component of project reporting to document the sampling procedures. Corporate standards of Noranda have always been to collect a representative sample. The core was logged for geology and geotechnical parameters and then cut in half with a diamond saw. The samples were then sent to either Swastika Labs or Chemex Labs in Rouyn, Quebec.

12.2 EXALL SAMPLING

12.2.1 EXALL DIAMOND DRILLING

The core was brought to the surface where the geologist logged and sampled it. The core was split in half with a diamond saw. Prior to the installation of the mine site laboratory, Techni-Lab provided sample preparation of a 30 grams (g) sample and completed a fire assay of the sample. All samples above 34.3 g/t Au were check assayed, as well as each 20th sample.

When the mine site laboratory was operational, they completed the analysis of the split core. Techni-Lab assayed the occasional overflow that the Exall lab could not handle.

12.3 BRIGUS SAMPLING

The sampling procedure begins with the geologist defining each sample interval and designating such with a sample tag documented in a sample book. They next mark the core with a center line cut mark and replace the core box lids for transfer to the sawing station. In the sawing room technicians saw the core sample in half with a diamond saw and place one half in a bag which is marked with the sample number and includes a sample tag. The half core that remains in the core box has the lid replaced and is placed back in the rack by the technician. Blank and standard samples are inserted approximately every twenty samples and are numbered in sequence with the core samples. The samples are then stored inside the core facility until they are picked up by Swastika Laboratories (Swastika) from Swastika, Ontario. The samples are placed into their truck, with each sample being checked off a list as it is being loaded and then taken directly to the laboratory where they are unloaded into a secure facility. At the logging area, once a truck load of split core has accumulated, the boxes are labelled with hole number and footage on stainless steel tags and then moved to covered storage racks located outdoors.

12.4 BLACK FOX DEPOSIT SAMPLING ISSUES

Prenn (2006) reported what MDA considers, two serious sampling issues at the Black Fox deposit. Both of which are related to coarse gold and sample size resulting in analyses that tend to report less gold than is actually present. The first issue relates to obtaining a large enough sample to represent the area it will influence. The gold at the Black Fox deposit appears to be concentrated in small areas causing drillhole samples to occasionally get too much gold in the sample or more commonly, missing the area of concentration and get too little gold in the sample.

The second issue relates to the particle size and distribution of the gold. When the particles are relatively large and not evenly distributed, the core holes can be too small to obtain a representative sample. This has a similar effect, in some cased it will over estimate the gold content but more typically underestimate it. Some samples may even appear to be waste having not encountered any gold particles that may be located relatively close by. It is likely that holes several meters in diameter would be required to obtain representative samples of the deposit. Prenn (2006) compared the areas that were mined with the drilling present and found many instances of drill indicated waste which were subsequently stoped.

This second issue is accentuated by getting the representative amount of gold in the sample pulp once the core sample has been split, crushed, split again and then pulverized. Gold particles up to 0.15 centimetres (cm) have been observed and particles of 0.06 cm are very common (Pitard, 2005). With gold this coarse, it is easy to create sub-samples that contain too many or too few gold particles if the sample size is not based on the size of the gold particles in the deposit. In order to sample the 0.15 cm gold particles that occur at Black Fox, samples of up to 109 kilograms (kg) must be processed in their entirety (Pitard, 2005). If the sample contains 0.06 cm gold particles, which commonly occur in the deposit, a 7 kg sample must be processed in its entirety (Pitard, 2005). These sample sizes are much larger than the typical 30 g fire assay sample or even the generally larger than the 1,000 g screen metallic assay sample. Once again, the samples result in a few assays containing too much gold, with far more containing less than is actually present in the whole sample.

Without proper size samples the database for the deposit likely contains a few samples that are too high in grade, but far more that are too low in grade. Francis Pitard concluded in his 2005 report on Black Fox mineralization that:

- "The size of the core samples can account for local geology, but cannot account for the local gold content: Relative to the size of the coarse gold, the core mass is too small. The resulting effect is called the In Situ Nugget Effect: It is of the utmost importance for management to understand it.

- As a result, Poisson skewness enters the database, leading to a frequent under-estimation of many ore blocks, and an occasional over-estimation of a few ore blocks.
- Such skewness, if carried too far, as I believe is the case, can under-estimate the gold content of the deposit. However, and this is very important, it is an undeniable fact that the Ore Reserves are under-estimated. This is something to keep in mind: Poisson skewness affects the grade somewhat, but above all, makes a disaster on the estimation of the Ore Reserves, unless you are very lucky by having sharp, natural and obvious ore boundaries (e.g., Midas mine in Nevada).
- By the time the sample is taken to the laboratory sample preparation, you have already lost its main purpose, which is to be reasonably representative of all gold particle size fractions. Then, the preparation and assaying procedure, ignoring the potential presence of coarse gold, makes things even worse, most likely introducing a superimposed secondary Poisson skewness in the database."

SRK (2008) and Prenn (2006) agreed with Pitard's (2005) conclusion, that the drillhole data is likely biased and will likely underestimate the contained gold within the deposit. Wardrop also agrees with the Pitard's work and recommends that a comprehensive grade control and reconciliation program be established for the Black Fox Mine.

13.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

13.1 SAMPLE PREPARATION AND ANALYSES

This section is partly excerpted from Technical Report Black Fox Project Matheson, Ontario Canada by Mine Development Associates, August 14, 2006 (Prenn, 2006).

13.1.1 NORANDA DRILL SAMPLE PREPARATION AND ANALYSIS

The first phase of the Noranda drilling was processed by Min-En Laboratories Ltd. and TSL Laboratories (Holes 1-17). Noranda then used Swastika or Chemex Labs for analysis of the remainder of the samples. Noranda instructed the assay lab to prepare a 15 g sample for analysis, and to re-run samples if the initial analysis was greater than 2 g/t Au using a 30 g sample.

13.1.2 EXALL DRILL SAMPLE PREPARATION AND ANALYSIS

Exall utilized Techni-Lab to complete the assaying of their drillholes until the mine site lab was completed. After completion of the mine site lab in February 1999, most of the assaying for the muck and chip samples was completed at the on-site laboratory, with Techni-Lab used for the drillhole samples, overflow and check assaying.

Techni-Lab dried and crushed the sample to 10 mesh, where a 300 g split was taken. The 300 g sample was pulverized to 80% -200 mesh. A 30 g sample was split from the pulverized material for fire assay with AA finish. Exall requested checks on all assays exceeding 34.3 g/t Au. The Techni-Lab internal checks agreed well with the original sample.

13.1.3 EXALL MINE SITE ASSAY LAB PROCEDURES

Blank samples were introduced with regular samples to verify the accuracy and to see if any contamination was present at the lab. Split assay pulps were sent to an external lab for comparison to verify the accuracy of the Exall mine site laboratory. From January 27 to February 25, 1999 a total of 370 samples were sent to Techni-Lab in Ste. Germaine Boule, Quebec. The difference between the Exall Lab and Techni-Lab was an average of 1.45%.

13.1.4 BRIGUS' DRILL SAMPLE PREPARATION AND ANALYSIS

Brigus sawed the core and shipped ½ of the drill core to either Swastika or SGS Laboratories. The labs prepared a 30 g sample for fire assay with a gravimetric finish. The core was first crushed -10 mesh and a 400 g split then pulverized. As a quality check, the coarse reject sample material from each mineralized zone, over 1.0 g/t Au is sent to the other lab. The rejects are re-split, pulverized and re-assayed using a 30g fire assay with a gravimetric finish. This procedure provided a check on the entire assay process, from sample preparation through to the gravimetric finish. Many of the higher grade samples were run with a screened metallic fire assay. All check data was subjected to a standard quality assurance/quality control (QA/QC) analysis.

Swastika sent certificates of Analysis and electronic data files directly to the Brigus office in Matheson, Ontario. Hard copy results and assay certificate were also faxed to Brigus. The faxed certificates, were marked up with specific hole intervals and cross checked to the digital file for errors. After confirmed to be correct, the faxed copies were stamped complete, and added to the audit file for back referencing. The digital assay file was cut and pasted directly into the electronic core logs. Once the results were pasted in, the sample numbers were cross-referenced to ensure no pasting errors occurred. The completed drilling logs were then saved into a separate file. Once the logs are complete with all assays, they were saved as a "DC" file. The "DC" files were put into a locked folder on the Black Fox database, which can only be accessed as a read-only file. All editing of these files must be done through the Administrator (Project Manager). Once the file has been saved to this folder, the file was sent to Apollo's offices in the USA for modeling and reporting purposes.

All reported assays are final assays, and original certificates of analysis are stored in a separate binder and stored in a fire proof safe at the Black Fox mine site. All assay reporting goes through the Black Fox Project Manager.

Since 2008, Brigus Gold has not completed any further exploration drilling on the Black Fox property, so no additional procedures are reported.

13.2 QA/QC ANALYSES

13.2.1 SUMMARY

During the development of the SRK Pre-Feasibility Study, Analytical Solutions (ASL) of Toronto Canada was contracted to provide an independent QA/QC review of historical and current sampling at Black Fox (Bloom 2006, 2007). The following paragraphs summarize their findings

ASL has been contracted to review documentation related to assay quality control and sampling for the Black Fox mine. The principal objective iwasto justify use of the existing assay database for Resource calculations.

The focus of the studies by ASL was to determine (a) whether there was any evidence of bias in the assay database and (b) the effect of coarse gold on the reliability of the assays.

The Black Fox assay data includes 128,026 assays. The 50th percentile for the dataset is 0.06 g/t Au, the 90th percentile is 0.77 g/t Au and the 95th percentile is 2.23 g/t Au. It is apparent that only the upper 5% of the samples will influence the Resource calculation and the focus of the review should be this relatively small percentage of samples in the database.

No evidence has been found by previous consultants, who have done extensive reviews of procedures and data, of a bias in the gold assays. A systematic bias over a significant amount of time would affect a resource calculation but this problem has not been identified.

Concerns have been raised regarding sample representivity of the Black Fox deposit. Thousands of pulp and reject duplicates confirm that it is difficult to reproduce assays within an arbitrary ± 10% but the assay reproducibility is typical of similar deposits and does not represent a material risk.

The historic check sampling on the project appears to be weak based on current QA/QC requirements for similar styles of gold mineralization. The Noranda check assays appear to be limited to only the same assay pulps. In general, they show reasonable agreement on the mean grade, however individual sample variance is relatively high. The Exall check assay program also was conducted on the same assay pulps. Techni-Lab, who conducted the majority of the exploration assaying for Exall, have been shown in a previous report to produce good reproducibility of the assay pulps.

Brigus has implemented a significantly improved check assay program where there is a check assay on each mineralized interval. In addition to the blank and standard check samples, Swastika runs its own internal check samples. All of the samples are run using a 30 g fire assay. Relatively higher-grade zones are selected from the fire assay results by Brigus personnel and these intervals are re-run with a 1,200 g screened metallic assay. Two of these samples are selected out of each ore zone at random and the rejects are sent to SGS Laboratories in Rouyn, Quebec where they are re-prepped and run for a second screen metallic assay. This is used as the quality check on the first assay set run by Swastika. All of the assay data is sent to Brigus in digital format where it is merged with the geological spreadsheet for that hole.

13.2.2 Noranda Check Assays

The Noranda data includes 196 reruns of 15 g samples of the original 15 g samples. The reruns average 4.6% lower grade than the original samples. The samples over 2 g/t Au were noted to be rerun by a 30 g sample, however most of this data is not in the digital database. Reruns of 80 samples indicate the reruns of 30g are higher in grade by about 5% than the original 15 g sample. Evans (1997) of Roscoe Postle Associates Inc. (RPA) reports that Noranda checked about 10% of their assays.

The Noranda assay sample distribution is missing the high-grade found in all the other drill programs. Prenn (2006) has recommended that check assays should be completed on the Noranda core that remains by metallic assay. These should be completed on intervals inside mineralized zones and just to the outside.

13.2.3 Exall Check Sampling

Techni-Lab batched samples in groups of 24. Each group contained at least one blank sample, one standard sample and duplicate samples. Routine checks were taken on about 5% of the samples and all samples over 34.3 g/t Au, however the check assay data is not present in the assay database. The statistics from past programs however are included in past RPA (1997) audits of the deposit Resources and Reserves for Exall. These indicated very good agreement between the Techni-Lab original assay and the Techni-Lab duplicate on thousands of checks of the same pulp.

13.2.4 Brigus Check Assaying

Metallic Check Assays

Brigus has completed screen metallic assays on 594 samples. Of these, 512 assays can be compared to normal fire assays. The screen metallic assays are 17% higher in grade than the average of the fire assays from these intervals. A total of 289 screen metallic assays are higher in grade to the average of the fire assays, while 223 are equal to or lower in grade. Prenn (2006) believes that screen metallic assays are essential in obtaining a sample assay that is more representative of the gold in the core sampled. Other assay methods will find too much gold on occasion, but the majority will find less than is in the core.

Standards and Blanks

Brigus submitted standards and blanks within each set of samples submitted for assay. Four labs were used with most of the assays completed by Swastika. Several thousand tests were completed and the blanks typically agreed.

A number of sample standards have been run within each group of samples. Swastika has reported reasonable ability to accurately assay the standards.

The following ranges were used to pass or fail the blanks and standards:

- Blank > 0.03 g/t Au = Fail
- Standard 1.422 >1.528 or <1.322 g/t Au = Fail
- Standard 11.27 >12.03 or <10.63 g/t Au = Fail

- Standard 9.62 >10.28 or <9.00 g/t Au = Fail

If the blank or standard failed, then the entire batch (20 samples) would be re-assayed, as well as the failed standard or blank.

Check Assays on Sample Pulps

A total of 8,425 sample pulps have been rerun by the original assayer. These samples indicate good agreement between the original sample and the rerun sample. The check was required to be within ± 10%. If not, the pulp would be re-assayed a second time.

Checks on Sample Rejects

A total of 2,618 assay intervals have been checked by a different lab using splits from the sample rejects. The results indicate that the original sample is higher than the check by about 4%. Of the 2,618 checks, a total of 905 or about 35% have differences of greater than 30%. If the checks were not within 20%, a second pulp would be prepared from the rejects. These relative differences are very significant and point out the need for a more substantial sampling and assaying program.

Mini-Bulk Sample Checks

Large composites averaging about 14 kg in weight were made by combining drillhole core and/or rejects. Typically, nine drillhole intervals were composited into one mini-bulk sample, however the range was 4 to 17 kg. A total of 47 composites were made from mostly ore-grade intervals. Twenty-one of the 47 ore-grade composites contained high-grade. Since these tests use a much larger sample than the assay pulp, one would expect in a coarse gold deposit that the results of the mini-bulk sample gravity tests would be more reliable than the 30 g pulps used for fire assay. The results of the 47 ore-grade mini-bulk gravity tests indicated a 9% lower grade in the mini-bulk samples compared to the individual assays. This is the opposite of what would be expected, and it is likely due to more high-grade material being in the mini-bulk samples than in the deposit as an average. The six waste mini-bulk samples showed an improvement in grade of 1382.3% compared to the individual core assays. One of the waste samples averaged 0.00 g/t Au from the drillhole intervals and 2.82 g/t Au from the mini-bulk composites. The other five mini-bulk samples were not assayed prior to testing. One of these samples averaged 1.38 g/t Au from the mini-bulk test.

13.3 Summary

After the core was logged, the core samples were split by a diamond saw to obtain the assay lab sample. The 50% split was bagged at the site and either picked up by assay lab personnel or shipped to the assay lab. The sample was dried, crushed, split, pulverized, and blended to obtain fire assay pulps. The labs prepared 15 g to

30 g assay ton samples for assay. Most of the assays were completed by fire assay methods with a gravimetric finish.

14.0 DATA VERIFICATION

14.1 HISTORIC

SRK (2008) reported that the historic check assaying conducted by previous operators Exall and Noranda are considered substandard by today's requirements. However, the QA/QC study discussed above indicates that this data set presents no material risk to the current Resource estimation.

Prenn (2006) reviewed Brigus' individual assay verification program and provided the following synopsis.

> *Brigus' program for data verification is a considerable improvement of the past checks, however, while the number of checks have improved, the sampling problems have become more evident. The metallic assays have shown a grade improvement of about 17% over the average of the fire assays for the same intervals. Check assays from pulps have shown good agreement with the original assays, while new pulps prepared from rejects have not shown good agreement with the original assays. MDA believes that the samples from drilling contain less gold than is representative from the area drilled, and that the fire assay samples contain less gold than is in the core sample.*
>
> *MDA recommends that Brigus consider using metallic assays as the only appropriate method to sample the core, and that additional mini-bulk gravity tests and full scale bulk samples be completed for the main types of mineralization in the deposit.*

SRK agreed with the recommendations of Prenn (2006) presented above but recognized that screen metallic assays are quite expensive and typically provide a slow turnaround time. One problem associated with a change in assay procedures at this stage of the project is that it would require a re-assay of as many pulps as are available in order to standardize the database. The benefits of such a program may not outweigh the time and cost associated with it. Model reconciliation with actual mining would provide a valid alternate method to verify the proper usage of the assay data in the estimation technique. Additionally, ASL has reviewed the data set subsequent to the Prenn (2006) review and has concluded that it presents no material risk to the Resource estimation.

The second aspect of the data verification pertains to comparison of the numerical values contained in the electronic database to those reported on the original hard copy assay certificates. This work has been conducted by several reviewers and no problems have been reported to date. The original database was validated by RPA (1997, 1998) and subsequently by Prenn (2006). Further to this work, ASL crosschecked portions of electronic database to original certificates and reported no issues. SRK provided independent verification of the electronic database subsequent to previous reviews in two ways. The first was to evaluate the current procedures used by Brigus to transfer the assay results obtained from the lab to its electronic database. SRK noted that these procedures provide adequate safeguards to the integrity of the assay database and meet or exceed current industry standards. Additionally, SRK was provided with original signed assay certificates from recent drilling and conducted spot checks comparing the certificate values to the electronic database and no errors were found.

14.2 Wardrop Verification

Site visits were conducted from August 31 to September 2, 2010 and again from November 17 to 19, 2010. During these site visits, inspections were conducted of the site, including the open pit, underground and mill location.

14.2.1 *Database Spot Checks.*

Seven drill holes were spot checked by Wardrop. These holes were selected to provide a cross section of geographic location, mineralization intersected and of the various diamond drilling campaigns. The core was reviewed and compared with the paper drill logs on November 19, 2010. No issues were identified with the core logging and transcription.

Each of these drill holes were cross checked with the paper and digital drill logs for the collar coordinates and down hole surveys. The assays for each of the drillholes were crosschecked with their assay certificates. No errors were found. The seven drill holes are listed in Table 14.1 below.

Table 14.1 Drillhole Spot Checks

Drill Hole No.	Location X	Location Y	Location Z	Length (m)	Bearing (°Az)	Dip (°)
04BF324	10449.990	9788.980	10004.150	542.0	0	-67.5
05BF417	10112.484	10030.012	10001.932	190.8	0	-50.0
170-2	10114.469	10018.616	9828.877	62.5	180	18.0
06BF449	9949.910	9915.240	10000.520	176.0	180	-65.0
06BF450	10349.770	10125.290	9999.920	152.0	180	-65.0
06BF453	9813.120	10099.940	9999.250	152.0	0	-65.0
235-351	10275.00	9875.300	9771.200	130.5	0	-42.0

15.0 ADJACENT PROPERTIES

The Black Fox Project is located in the eastern side of the Porcupine District approximately 75 km east of the Timmins Gold Camp. The Project is situated along the DPFZ, which hosts many important properties along the 200 km strike length of this structure (Prenn, 2006). This includes the Dome Mine, now part of the Porcupine Joint Venture, located in South Porcupine near Timmins, Ontario and approximately 65 km west of the Project area.

Properties proximal to the Project area include the Aquarius, Clavos, Hislop, Holloway, Holt and Taylor held by St. Andrew Goldfields. The Hislop is adjacent to the Black Fox Project while the Aquarius, Clavos and Taylor are more proximal to the mill site and the Stock Mine property.

The Ross deposit, which is located south of Hislop, was an underground mine that last operated in 1989. Exploration is currently being conducted by Guardians of Gold Inc. in the Ross Mine area. Guardians of Gold are also evaluating the extraction of gold from the tailings at the Ross Mine

Brigus has an exploration program on the Pike and Grey Fox properties immediately adjacent and to the south of the Black Fox Mine. Recent news releases have returned positive results (Brigus Gold, Brigus Gold Reports Positive Exploration Drilling Results at Black Fox Complex, November 30, 2010).

Brigus has initiated an exploration program at the Stock Mine property to confirm the potential for further gold mineralization (Brigus Gold NR, Brigus Gold Launches Exploration Program at Stock Gold Mine, December 7, 2010).

16.0 MINERAL PROCESSING AND METALLURGICAL TESTING

The Stock Mill is located 31 km west of the Black Fox project. Brigus Gold purchased and upgraded the Stock Mill during a six month period. Refurbishment of the mill was completed in April 2009 and it began operating on May 1st, 2009.

GBM MEC was awarded an EPCM contract to expand and re-commission the facility, which entailed extensive equipment additions and upgrades to increase plant throughput from 1,300 t/d to the current 2,000 t/d. Particularly important areas addressed included the following.

Additions included:

- Sandvik CH430 cone crusher
- 3.7m x 5.5m Osborn ball mill
- Electrical substation and Motor Control Centre (MCC)
- Scalping beneficiation stage (crushing)
- Cyclone packs (grinding)
- Two 345 cubic meter leach tanks
- Inter-tank carbon screens
- Three tonne ADR circuit
- Induction furnance and shipping area (Refinery)
- Reagent storage facility
- Concrete leach containment bund
- Pumps and piping systems
- Programmable logic control system

Upgrades included:

- Material transfer equipment

- High rate thickener
- Tailings management area
- Digital security system

Commercial production was announced in the first week of June 2009.

Current mill operating data presented to Wardrop shows that the mill is treating 2,000 t/d as per expansion target design.

In addition to checking the daily mill summary and monthly stockpile audits Wardrop also checked the procedure for weightometer calibration, and the shipment invoices of bullion sent to Johnson Matthey.

16.1 Metallurgical Testing

This section excerpted from "Technical Report Black Fox Project Matheson, Ontario, Canada" by Mine Development Associates, August 14, 2006 has been standardized to this report.

"The material in this section was developed mainly by Peter Taggart and Associates in conjunction with developing a metallurgical flowsheet and estimated capital and operating costs for a 1,500 t/d processing plant used in the 2004 pre-Feasibility Study completed by MDA for the Black Fox Project.

Canadian Mine Development, commissioned by Exall to prepare a Feasibility Study, retained Mr. Rick Swider, Richard C. Swider Consulting Engineers Limited to direct metallurgical testwork performed by Lakefield Research Limited (Lakefield). The test program, conducted in 1996, was designed to assess the amenability of the Stock mill to treating ore from the surrounding area. The comprehensive program confirmed the suitability of the plant and custom milling operations commenced in 1997.

In 1999, Kinross Gold was holding the Macassa plant in Kirkland Lake, on a "care and maintenance" basis. Exall elected to use this plant, upon the expiry of the three-year custom milling agreement with St. Andrew Goldfields. Operations commenced at the Macassa plant in October 1999 and were terminated in May 2001.

Exall commissioned Richard Swider to oversee additional bench scale and pilot plant test programs in 1999 to examine alternative process options that could enhance process efficiencies.

16.1.1 Mineralization at Black Fox

The Black Fox mineralization is hosted in two zones, the West Zone and the East Zone. The West Zone material principally comprises green carbonate and contains gold in quartz ankerite-veinlets. Minimal amounts of sulphide are present. The East Zone contains up to 5% sulphides, principally as pyrite. While the East Zone mineralization is slightly more refractory than the West Zone material, both exhibit free milling characteristics and yield gold recoveries in excess of 95%.

The mineralization contains finely disseminated visible gold and is amenable to gravity concentration. The host rock contains no graphite or cyanide consuming minerals in quantities sufficient to adversely affect gold recoveries or operating costs.

16.1.2 *INITIAL METALLURGICAL TESTWORK (1996)*

Lakefield Research conducted bench scale test work in 1996 to determine the suitability of the Stock mill to treat mineralization from the Black Fox deposit. The program, designed and directed by Richard Swider, examined recovery of gold by gravity and cyanidation methods. In addition, characterizations of selected samples were performed for environmental purposes.

The work was performed on 67 samples of mineralization, 26 from the East Zone and 39 from the West Zone. The samples were composited into six sample blends, three for each of the two zones, as shown in Table 16.2.

Table 16.1 Gold Head Analyses for the Composite Samples (g/t Au)

	Calculated[1]	General	Low Grade	High Grade
West Zone				
LR Wtd. Average[2]	9.47	7.71	2.76	25.4
LR with Metallics[3]	8.56	8.37	2.56	22.8
LR Test Average[4]	8.38	6.63	2.18	23.1
East Zone				
LR Wtd. Average[2]	20.1	18.1	3.75	41.0
LR with Metallics[3]	22.9	17.7	9.56	40.0
LR Test Average[4]	19.1	17.6	5.99	35.9

1. Calculated from the weighted average heads from composites General, Low, High.
2. The head grade calculated from the weighted average heads from all samples used for compositing.
3. The direct head for each composite using a ±100 mesh pulp metallics procedure.
4. The back calculated average gold head grade from the test program.

Given the presence of visible gold, albeit finally disseminated, reasonable agreement is achieved in most cases. The East Zone Low Grade demonstrates poor reconciliation between the head values shown.

Detailed head analyses of the individual composites failed to identify any elements or compounds that could be environmentally deleterious or that could seriously adversely affect the cyanidation process. Sulphur in the East Zone High Grade was measured at 3.05%. The highest equivalent sulphide content in the West Zone material was 0.48%.

The Bond Work Indices of the East Zone and West Zone General Composites were determined to be 16.6 and 14.9 kWh/t, respectively.

Gravity concentration tests were performed on each composite sample, yielding the results summarized in Table 16.3

Table 16.2 Summary of Gravity Concentration Test Results

Composite	Head (g/t Au)	Wt Recovery (%)	Con. Grade (g/t Au)	Gold Recovery (%)
West Zone				
General	6.63	0.064	5,195	49.9
Low Grade	2.18	0.045	1,531	31.8
High Grade	23.1	0.097	13,132	55.4
East Zone				
General	17.6	0.063	9,580	34.4
Low Grade	5.99	0.069	872	10.1
High Grade	38.0	0.160	15,063	47.6

Lakefield noted that “no coarse (>48 mesh) gold” was observed in any of the gravity tests. Nevertheless, the results suggest that all but the East Zone low-grade mineralization could be amenable to gravity concentration, although free gold commonly observed in the core may be larger.

Cyanidation tests were performed on the gravity circuit tails for each composite to determine the impact that grind and leach time respectively impart on gold dissolution. The results indicated that gold extraction from East Zone mineralization was sensitive to fineness of grind; gold extractions improved as the fineness of grind increased from K80 70 µm through 50 µm to 30 µm. West Zone leach extractions were relatively unaffected by particle size, over the range examined. In addition, the effects of variable leach times within the range of 36 hours to 72 hours were examined. Gold extractions from both East and West Zone mineralization were found to be insensitive to leach times, again over the range examined.

A summary of the key cyanidation data is presented in Table 16.4, based on 48-hour leach times, a K80 50 µm grind for West Zone material and K80 30 µm grind for East Zone mineralization.

Table 16.3 Summary of Cyanidation Test Data

Composite	Reagent Consumption (kg/t) ore		Residue (g/t Au)	Gold Extraction (%)
	Lime	NaCN[1]		
West Zone				
General	0.70	0.32	0.14	95.1
Low Grade	0.89	0.18	0.08	95.0
High Grade	0.86	0.33	0.12	98.9
East Zone				
General	0.76	0.31	0.68	93.1
Low Grade	1.02	0.20	0.33	93.9
High Grade	0.85	0.52	1.38	92.7

[1]Cyanide consumed during leach, not including initial cyanide to 0.5 g/L

The samples examined were very amenable to cyanidation when low dosages of reagents were applied. Grab samples of leach solution taken after 12 hours of leach indicated rapid leach kinetics. Since it was proposed to grind in cyanide solution, Lakefield projected that "a significant proportion of the gold is likely to be recovered in the carbon column circuit".

Gold adsorption test data indicated that no deleterious species were present. It was projected that gold adsorption in a CIP circuit would be rapid and complete after 7.5 hours.

Settling tests were performed on East and West Zone General Composites, at a K80 30 µm grind. A favourable unit area rate was achieved, being less than 0.2 square metres per tonne (m^2/t) per day in all cases in which modest flocculant additions were used.

In conclusion, the favourable Lakefield test results, together with the existing stock mill circuit configuration, supported the concept of milling Black Fox mineralization in the St. Andrew Goldfields' plant.

16.1.3 Stock Mill Operations (1996-1999)

The Stock mill, designed by Leslie Engineering, was constructed in 1988 for a cost of US$17 million. The plant included the conventional unit processes of:

- primary crushing
- closed circuit, single staged fine crushing
- a closed circuit mill
- pre-leach thickener and carbon columns
- leach and CIP circuits
- carbon stripping and electrowinning

- cyanide destruction.

St. Andrew Goldfields' personnel managed and operated the plant, allowing access to Exall's technical representative. Exall paid a processing charge, based on the tonnage milled. In addition, a bonus was paid to the owner, based on gold extractions achieved.

Exall added a 1,075 mm x 1,382 mm Eagle jaw crusher ahead of the Stock mill crusher as the ore was much larger than the previous mill feed. This primary crusher discharged onto the feed belt between the original 614 mm x 922 mm jaw crusher, and the 1,300 mm short head cone crusher, in closed circuit with a screen, prior to being conveyed to the fine ore bin.

Grinding was accomplished in a 2,896 mm x 3,658 mm, 450 kW primary ball mill and a 2,743 mm x 3,353 mm, 337 kW secondary ball mill, in closed circuit with cyclones. Both mills were rubber lined. "Optimum" grinding rates were reported to approximate 43 t/h, subject to the mineralization being processed, with work indices varying within the range 14 to 16 kWh/t. Grinding was performed in cyanide solution.

The cyclone overflow fed a 19.8 m diameter thickener, the overflow from which was flowed to seven carbon columns. The thickener underflow was pumped to four leach tanks to provide a nominal 27 hours retention time, which was airlifted to five CIP tanks that provided 1 hr retention time in each tank.

Carbon from the columns and CIP circuits was stripped at 142°C. One tonne batches of carbon were regenerated on-site in an electrically heated rotary kiln, after washing with 3% nitric acid. Fine "attritted" carbon was recovered and shipped to Noranda. The pregnant strip solution was fed to a 1.0 m^3 electrolytic cell. The sludge produced was dried and charged into an induction furnace to produce doré bars.

Exall's technical representative was present for the monthly estimates of gold inventories and was able to monitor normal operations for about 50% of the time. While operations were satisfactory at the Stock mill, some issues were of concern to Exall. Thus, certain housekeeping issues could have contributed to loss of gold. Further, the Black Fox mineralization was processed in batches, typically of 5,000 t.

Thus, at a nominal 1,000 t/d milling rate, campaigns were generally of five days duration. Ores from other sources were processed between the Black Fox campaigns, rendering precise metallurgical accounting difficult. This problem was exacerbated by scaling in flow-meters, caused by the high lime additions used in the plant. While the plant produced gold recoveries in the mid- to high nineties, Exall elected to ship mineralization to Macassa upon the expiry of the three year custom milling agreement with St. Andrew Goldfields.

During the milling campaign at the Stock plant, a very short plant test was conducted to operate a Falcon concentrator and gravity table. The initial results were not sufficiently encouraging to justify a protracted test. Based on the results of bench scale and pilot plant test programs, this is somewhat surprising. It is conceivable that insufficient time was available to properly fine tune the circuit. It is also possible that the fineness of grind contributed to the poor results.

The Stock mill, operated by St. Andrew's Goldfields as a custom mill, had the capability of a throughput in 1999 of 1,100 t/d (396,000 t/a) on ore to be processed from the Black Fox mineralization. "Metsim"® computer simulations of the grinding circuit have determined that a series circuit, utilizing an existing third grinding mill along with reconfigured existing cyclones, would produce the optimal grind size at the rate necessary to achieve 1,100 t/d. The recovery projections for the Stock mill are 95%, which is considered achievable at the given grind size of P80 = 55 µm and leach times in excess of 24 hours**.**

It was initially designed to process 500 t/d through a conventional fine grinding, cyanide leach gold recovery plant. The plant was expanded in 1993 to 800tpd and additional equipment was added in 1997 to increase the capacity to a maximum of 1,300 t/d depending on the fineness of grind required. The mill was designed to treat ore from the Stock Mine whose head frame is adjacent to the mill. The design also provides for treatment of custom ore delivered by highway trucks to the large storage pad adjacent to the crushing plant.

16.1.4 Historical Processing of Black Fox Ores

Ore from the Black Fox Mine property was custom milled at the Stock mill from April 1997 to September of 1999 operated by St. Andrew Goldfields. The ore was free milling and finely disseminated so cyanidation of the whole ore was undertaken, rather than gravity pre-concentration. Despite visible gold (vg) in some geological samples no coarse gold (>48 mesh) was noted in any of the subsequent Lakefield gravity tests.

Ore was ground to a P75 (75% passing) size of 56 µm in a conventional CIP mill, having 40 hours retention time at 800 t/d, or 30 at 1,100 t/d. Mine production was generally 900 t/d of mill grade ore, unless development was "in ore", which increased output. Run of mine ore was graded between "high grade ore" sent to the mill, and "low grade ore" assaying under 1.6 g/t, which was below the cut-off grade for economical milling (including transport). This low-grade ore was used as foundation for ore piles on unsealed ground, and to purge the mill prior to shutdowns. Ore was transported in 35 t road trucks and weighed over an automated load-bridge at the receiving mill. Prior to the addition of a 42 in x 54 in Eagle Crusher in 1999 the original 24 in x 36 in 60 hp Kemco jaw crusher had been undersized putting a heavy (50 mm: 2 in feed) load on the 4.25 ft (200 hp) Symons SH cone, resulting in frequent maintenance.

Excluding a retrofitted tertiary grinding mill that had surge problems, the Stock mill facility had a 450 kW (600 hp: Allis Chalmers 9.5 Ø x 12 ft) primary mill and 337 kW (450 hp: 9 Ø x 11 ft) secondary mill, both rubber lined. Depending on ore type, optimum grinding was achieved at 43 t/h with ore having a bond index between 14 – 16 kWh/t.

Ore was ground in cyanide to minimize gold entrapment within the mills, and then thickened in a 65 ft (19.8 m) diameter Eimco thickener, leached (4-stage) and mixed with 6-12 mesh granular carbon in 5 (five) 4.27 Ø x 4.27 m (14 Ø x 14 ft) CIP tanks. The thickener overflow was also contacted with carbon in four carbon columns (expanded to seven, of which five were generally on line in 1998). Gold recovery was via a 1 m^3 (35.5 ft^3) electrolytic cell and induction furnace. Tailings disposal was the responsibility of the custom miller. Check assays were conducted at several commercial assay laboratories, normally on random feed grade samples.

Carbon was separated from the pulp using basket screens, and a vibrating (20#) screen. One tonne batches were regenerated on site in a 700°C electrically heated kiln after washing the carbon with 3% nitric acid. Loaded carbon was stripped at 142°C (290°F). Fine, attrited, carbon was decanted and drained into metre cube sacks.

Entrapped gold was recovered by incinerating this product at Noranda Smelter complex in Quebec, which recovered 92% of the contained gold (less minor material penalties and handling costs).

Doré bars were poured under the joint supervision of the custom miller and miner, and shipped (via Brink's) to Johnson Matthey (Canada) in Brampton ON, for refining (95% credit on contained gold). The doré typically contained 5.5% of Silver (range 2.8 – 7.6%), which was paid in full at current metal price. However, no special efforts were made to recover silver in the circuit. (Silver in the ore was not routinely assayed for [generally it was under 5gpt], and no measure of silver recovery was conducted, though some higher silver grades were detected).

Sodium cyanide brickettes were delivered in flow bins by Dupont.

A gravity circuit consisting of a Falcon concentrator and vibrating table were installed soon after start-up but after one week the concentrate in the bowl contained 2,000 g/t material, which was significantly less than the 59,000 g/t found behind the mill liners, and further testing was aborted (Exall project report June 1997).

16.1.5 Macassa Mill Operations (1999-2001)

Operations at the Macassa plant commenced October 1999. A consulting metallurgist, representing Exall's interests, had free access to all operating information at Macassa and was, in effect, the Chief Metallurgist for the operation's. The plant was highly automated and well equipped with security cameras. This degree of operations control and the exclusive use of the plant for Black Fox production, mitigated most of the concerns that were associated with St. Andrew Goldfields mill.

The Macassa plant, designed by Wright Engineers, provided the same basic unit processes as those at the Stock plant, although a two-stage fine crushing plant replaced the single stage fine crushing circuit at the Stock mill. In addition, the Macassa plant included a pre-thickener leach tank. Further, being designed to treat 2,000 t/d, the plant was oversized for the nominal 1,000 t/d Black Fox mine production rate. Accordingly, most of the leach tanks were not required for Exall's purposes. In all respects, the plant was able to satisfy the process requirements.

The crushing circuit comprised a jaw crusher, a standard cone secondary crusher and a tertiary short head crusher, in closed circuit with a vibrating screen. The crushing plant operated 12 hours per day.

Grinding was accomplished in two 600 kW ball mills that operated a 24 hours per day, 5 days per week schedule. The primary mill was steel-lined and charged with 100 mm grinding media. The secondary rubber-lined mill was charged with 25 mm balls. The nominal 1,000 t/d (40 to 45 t/hr) milling rate produced a cyclone overflow grind within the range 70% - 75% passing 53 microns.

The cyclone overflow was directed to a pre-leach tank ahead of the 19.8 m diameter thickener, the former providing a residence time of 30 hours. Given the favourable leach kinetics, gold dissolution was typically 90% complete by the time pulp entered the thickener, and this was recovered in carbon columns.

The thickener underflow passed through four leach tanks, which provided 24 hours of leach time. The other three available leach tanks were simply bypassed. A 5-stage CIP circuit was deployed, prior to treatment of the carbon in a conventional Adsorption-Desorption-Recovery (ADR) circuit.

Sub-standard metallurgical performance and increased cyanide consumption were observed when treating "sulphidic" material from the So zone in August 2000. However, these problems were overcome by blending mineralization prior to milling. Gold recoveries at the Macassa plant are shown in Table 16.5, together with the measured head grade and calculated head grade.

Table 16.4 Summary of Macassa Production Data

Month	Throughput (t ore)	Feed Grade (g/t Au)		Au Recovery (%)
		Measured*	Calculated**	
October 1999	15,628	7.02	6.03	95.06
November	20,562	9.86	7.66	98.14
December	19,217	8.44	5.11	97.43
January 2000	19,385	11.03	6.05	97.75
February	22,864	12.69	8.59	97.87
March	20,779	3.97	3.56	97.43
April	21,377	4.53	4.16	97.38

table continues…

May	24,856	7.05	5.16	97.62
June	22,938	10.87	9.85	98.78
July	19,530	8.24	6.26	97.67
August	20,732	4.90	4.53	86.29
September	23,870	4.84	5.12	97.04
October	23,767	6.01	4.79	96.79
November	23,080	6.60	6.78	97.42
December	12,309	7.73	7.00	98.48
January 2001	20,342	6.51	3.85	97.82
February	17,801	7.67	5.88	98.55
March	21,679	4.41	4.55	98.05
April	17,719	4.73	4.09	97.81
May	4,159	3.56	4.23	97.84

*Assay of composite mill feed sample taken by an automatic sample cutter form the mill feed conveyor discharge.
**Calculated head based on in-plant gold inventory, gold production and tailings losses.

Exall reported three plant feed gold contents; the Measured Head, the Calculated Head grade and the Calculated Cyclone Overflow grade.

- The Measured Head is the assay of a composite mill feed sample, taken by an automatic sample cutter from the mill feed conveyor discharge.
- The Calculated Head is based on the actual amount of refined gold produced, mill circuit gold inventories, the gold contained in the CIP tail residue and solution and the gold associated with the recovered fine carbon; and
- The Calculated Cyclone Overflow grade is the computed total assay of this flow, based on solid and solution assays and the pulp density.

Monthly plant gold recoveries were based on the Calculated Head. A review of plant metallurgical accounting procedures indicates the application of sound protocols. The use of the Calculated Head to determine the overall gold recovery was appropriate, given the quality of the raw data used to calculate this head grade. Further, the Calculated Head grade agreed, within reason, with the Calculated Cyclone Overflow gold grade, reflecting the relative ease with which representative samples of the overflow stream could be taken. However, agreements between the Calculated Head and Measured Head values were frequently poor.

The above data is based on 132 shift Measured Head assays and 73 daily Cyclone Overflow calculated values, all reported over the same three-month period. The Measured Head value is most frequently higher than the other two computed values. Further, the variability of the Measured Head assays is considerably greater than experienced with Cyclone Overflow data. The predominant reason for the higher and more variable Measured Head values is probably related to the difficulty in sampling relatively coarse material in the presence of visible gold.

16.1.6 METALLURGICAL TESTWORK (1999)

Exall commissioned Lakefield to conduct a program of bench scale and pilot plant tests to investigate the potential for gravity pre-concentration, using spirals and vat leaching as means by which toll milling costs could be reduced. The program was conducted under the direction of Richard Swider. The program was expanded to examine other concepts that offered the potential to enhance process economics.

Samples of high and low-grade mineralization were combined to produce six composites, ranging in grade from 2.07 g/t Au to 14.0 g/t Au. Descriptions of the samples are provided in Table 16.6.

Table 16.5 1999 Test Program Sample Description

Composite	Constituents	Grade (g/t Au)
General Composite #1 (GC1)	60% low grade at 5.48 g/t Au 30% high grade at 14.0 g/t Au 10% very low grade at 1.84 g/t Au	10.2
General Composite #2 (GC2)	50% GC1 at 10.2 g/t Au 50% low grade at 5.48 g/t Au	7.67
Low Grade Composite (LG)	100% low grade at 5.48 g/t Au	5.48
Very Low Grade Composite (VLG+A)	75% very low grade at 1.84 g/t Au 25% drum A	2.07
High Grade Composite (HG)	100% high grade at 14.0 g/t Au	14.0
High Sulfide composite (HS+HG)	50% high sulfide at 8.45 g/t Au 50% high grade at 14.0 g/t Au	11.2

The programs involved extensive laboratory and pilot plant work that included gravity concentration, leaching gravity concentrates and tailings, the flotation of gravity circuit tailings, thickening tests and work index determinations. Details of the test programs, and the potential financial implications of the results, are included in the reports issued by Lakefield and Richard Swider.

The conclusions drawn from the 1999 test programs are summarized in point form below.

- The optimum grind for spiral performance was reported to be 166 µm.
- At a feed grind of 150 µm, the spirals produced a gravity concentrate which, at a 15% weight recovery, contained approximately 80% of the gold in the feed.
- Leach residues approximating those at the Stock mill (0.14 g/t Au) were achieved when the 150 µm spiral tailings were subjected to conventional cyanidation.
- Spiral concentrates, reground to 40 µm, were leached to produce leach residues of grades (0.13 g/t Au) similar to those at the Stock mill.

- The High Sulphide Composite proved to be more refractory, reflecting experience in the operating plant. Thus, the 40 µm leach residue graded 1.8 g/t Au, while the overall leach residue (spiral concentrate and tailings) was in the range of 0.40 g/t Au, equivalent to a 97% gold recovery. Knelson concentrators, used to treat spiral tailings, failed to yield any significant benefit.
- Bond Ball Mill work index determinations indicated a value of 17 kWh/t should be used for plant design purposes.
- Flotation was found to be effective in the treatment of sulphide mineralization, but of marginal value when processing low sulphide material.
- Thickener unit area determinations for the leached spiral tailings (158 µm) and the leached spiral concentrate (40 µm) were 0.274 and 0.173 m^2/t per day respectively, at pH values just in excess of 10.0.

The test programs generated a significant amount of useful information that could be used in process trade-off studies during the preparation of a Feasibility Study. It will be important to ensure that the samples tested in this particular program are representative of the mineralization to be processed in accordance with updated mining plans. Additional confirmatory work might be required for feasibility level work.

16.1.7 MINI BULK SAMPLE GRAVITY TESTS (2006)

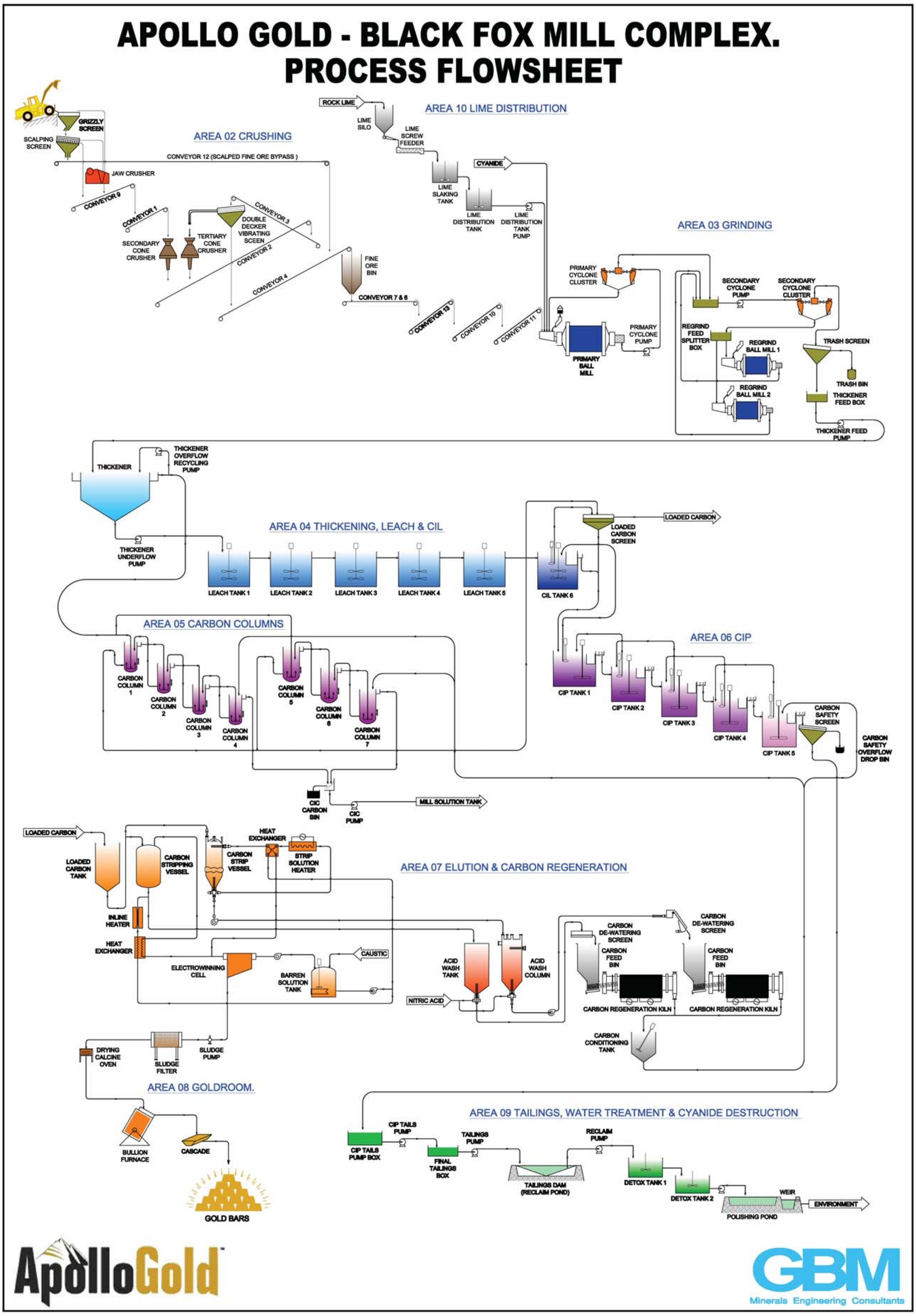
Francis Pitard (2005) recommended 200 mini-gravity tests be completed and compared back to the original sample grades. Apollo started this program with 58 tests completed, averaging about 14 kg per test. Of these tests, 47 were completed on “ore grade” material, six on “waste”, and five were completed on samples that had not been assayed.

The 47 tests completed on “ore-grade” materials indicated an average gravity recovery of about 59%. The average feed size for these tests was K80 114 µm.

16.1.8 CURRENT PROCESS DESCRIPTION

The following plant description outlines the current process facilities flowsheet and layout (Figure 16.1). The complete Mill PLC Control System has been updated to accommodate the expanded operation.

Figure 16.1 Current Stock Mill Process Flowsheet



CRUSHING

From a stockpile on the storage pad ore is reclaimed by front-end loader into a feed hopper equipped with a vibrating pan feeder that discharges on to a 1067 mm wide conveyor belt. This conveyor discharges on to a vibrating grizzly and scalping screen that feeds a relocated 610 mm x 914 mm Kemco jaw crusher. The jaw crusher product falls on to a 610 mm wide conveyor belt where it is joined by the product of a Metso HP 300 cone crusher. This belt discharges on to another 610 mm wide belt that feeds a 1,829 mm x 6,096 mm Metso TS 302 double deck vibrating screen. The oversize from both decks falls into the cone crusher and the undersize falls on to a 610 mm wide conveyor belt that discharges into a cylindrical fine ore bin. The crusher building is equipped with a bag-house dust collector. New conveyors have been installed and electrical substations expanded as necessary.

GRINDING

Crushed ore is fed from the fine ore bin by two 1,067 mm wide belt feeders onto a 610mm wide conveyor that discharges into the feed chute of a 2.9 m x 3.7 m, 450 kW (600 hp) primary "Osborn" ball mill. The mill is equipped with rubber liners and is charged with 70 mm grinding balls. This mill discharges into a pump box from which it is pumped to a 305 mm diameter cyclone and a 381 mm diameter cyclone operated in parallel. The cyclone underflows falls into a splitter box where they can be directed back to the primary mill or to the 2.7 m x 3.4 m 337 kilowatt (kW) (450 hp) regrind mill. The cyclone overflows flow to the regrind mill discharge mill pump box. The main regrind mill is equipped with rubber liners and it is fed with a portion of the primary ball mill cyclone underflow and a portion of the underflow from three 254 mm diameter secondary cyclones.

The main regrind mill discharges into a secondary cyclone feed pump box where it is joined by the overflow from the primary cyclones and the discharge of the second regrind mill. The secondary cyclone underflows fall into a splitter box where they can be sent to either regrind mill. The second regrind mill is a 2.3 m x 2.4 m, 225 kW (300 hp) ball mill also equipped with rubber liners. The discharge from this mill is pumped back to the secondary cyclone feed pump box. The overflows from the secondary cyclones flow to a 914 mm x 2,438 mm vibrating trash screen. The trash screen undersize falls into the thickener feed pump box. Milk of lime and sodium cyanide solution are added to the primary mill feed chute and most of the contained gold is in solution by the time the ore reports to the secondary cyclone overflow.

LEACHING

The ground ore slurry is thickened in a 19.8 m diameter x 3.7 m high Eimco thickener. The thickener overflowed to two trains of carbon in column (CIC) tanks operating in parallel where the gold is adsorbed onto 6–12 mesh granular carbon.

One train of CIC tanks is made up of four 0.9 m diameter x 3 m high columns and the other of three 1.5 m diameter x 3.6 m high columns. The CIC tails is pumped to either the mill solution tank or directly to the primary grinding circuit. The loaded carbon goes to a 1,219 mm diameter Sweco loaded carbon screen and to the loaded carbon tank. The thickener underflow is pumped to a series of four 7.6 m diameter x 7.6 m high agitated leach tanks arranged for gravity flow between tanks through 203 mm diameter pipes.

As part of the mill expansion two new 345 m^3 leach tanks were added. The first of these became the 5^{th} leach tank, and the other is currently operated as a Carbon-in-leach (CIL) tank, The overflow from this CIL tank flows to a series of five CIP tanks, where the gold in solution is adsorbed onto 6-12 mesh granular carbon. Carbon is advanced in the opposite direction to the slurry flow until it reaches the CIL tank. It can then be pumped to two 1.524 m Sweco screens feeding the new 3 t/day strip circuit.

The tails from the last CIP tank passes through a 1,524 x 2,438 mm carbon safety screen to a pump box that used to be the start of a two-stage cycloning circuit for producing underground mine backfill. The backfill cyclones have been removed and the existing sumps and pumps are now used to send the tailings slurry to the tailings pond. The carbon fines from the carbon safety screen are dewatered in one cubic meter bags boxes and sent to a smelter.

CARBON STRIPPING

From the loaded carbon tank the carbon is transferred to a new 3 t batch strip tank where the gold is removed from the carbon by a concentrated solution of caustic and sodium cyanide at 142°C. The pregnant solution flows out of the strip tank through a heat exchanger to an electrowinning cell where the gold is plated onto stainless steel electrodes. Periodically the gold sludge is removed from the electrodes by pressure washing. It is then pumped out of the electrowinning cell by a small diaphragm pump into a small filter press where it is dewatered. Periodically the sludge is removed from the filter press, dried and smelted in an induction furnace before being poured to produce a doré bar. The tails from the electrowinning cell are pumped into a barren solution tank. From there it is pumped through the heat exchanger and an electric inline heater back into the strip vessel. Stripped carbon is transferred from the strip vessel to an acid wash tank where scaling is removed by a 3% nitric acid wash before the carbon is sent through a carbon regeneration kiln. The kiln is operated at 700°C to remove volatile contaminants from the carbon.

The discharge from the kiln falls into a quench tank and from there it is transferred into a carbon conditioning tank. New carbon is also added to this tank and, after attrition agitation, it is transferred to a 1,219 mm diameter Sweco screen where fine carbon is removed. The oversize from that screen falls into the reactivated carbon tank from which it is transferred to the CIC or CIP tanks as required. The original 1 t batch strip circuit remains as an alternative stand-by circuit.

Water Management

Makeup water is pumped from the tailings pond by a barge-mounted pump either to the mill solution tank or into the first of two water treatment tanks. Sodium metabisulfite and copper sulfate solutions are added to the first tank to destroy the residual cyanide and ferric sulfate is added to the second tank to precipitate arsenic. From the second tank the treated water flows into a polishing pond to allow for completion of the cyanide destruction reactions and settling of arsenic particulates before it is discharged into the environment.

17.0 MINERAL RESOURCE ESTIMATES

Wardrop completed the mineral resource estimation using Gemcom© GEMS Version 6.2.4 for the Black Fox Mine Project. Tim Maunula, P.Geo. of Wardrop was the Qualified Person responsible for the resource estimation of these deposits.

17.1 DRILLHOLE DATABASE

The drillhole sample database was compiled by Brigus Gold, reviewed for QA/QC by Analytical Solutions of Toronto Canada and subjected to spot checks and statistical analysis by Wardrop. Wardrop reports the database is acceptable for use in the resource estimation.

The database a Microsoft Access database containing collar locations, drillhole orientations with down hole deviation surveys, assay intervals with results, geologic logs and geotechnical logs. The assay database contains several columns of gold assays representing the original 15 g and 30 g assays as well as numerous repeat assays including some screen metallic analyses. The gold assay data used for this resource estimate was the original assay value unless repeat check analyses had been made. In this case, the average of all the assay results was used in place of the original assay. For samples reported below the assay detection limit, a value of 0 g/t Au was used.

The resource database contains information from 1,889 drillholes totalling 335,841 m of drilling. The maximum drillhole length is 995 m and the average is 178 m. Most of the holes were drilled inclined to the north in order to intercept the south dipping mineralization at a high angle. Drillholes have been collared both on surface and underground.

17.2 MODEL LIMITS

The Black Fox deposit was modeled only for gold content. Two block models were developed for the Black Fox Mine based on the selective mining unit (SMU). For the open pit, the blocks were 3 m x 3 m x 3 m and for the underground model 3 m x 3 m x 4 m. The model boundaries based on local mine grid coordinates are presented in Table 17.1.

Table 17.1 Black Fox Model Limits

Block Model	Direction	Minimum (m)	Maximum (m)
2009A	Easting	9,700	10,600
	Northing	9,500	10,454
	Elevation	9,394	10,027
2009A_4M	Easting	9,700	10,600
	Northing	9,500	10,454
	Elevation	9,393	10,025

17.3 GEOLOGIC MODEL

The Black Fox deposit is described by Prenn (2006) as follows:

"*Gold mineralization at the Black Fox deposit occurs in several different geological environments within the main ankerite alteration zone, which has an indicated strike length of over 1000 m and a variable true width ranging from 20 to over 100m . This mineralized envelope occurs primarily within komatiitic ultramafics and lesser mafic volcanics within the outer boundaries of the Destor-Porcupine Fault Zone. The auriferous zones have several modes of occurrence; from concordant zones which follow lithological contacts and which have been subsequently deformed, to slightly discordant ones which are associated with syenitic sills and quartz veins or stockworks.*"

For this study, the mineralization is subdivided into three main domains based on the continuity and style of the mineralization. The first is called the "Main Zone" and is delineated by the primary domain of shearing and mineralization. It is broader near surface reaching a maximum true width of 150 m normal to strike and dip and narrows at depth. It averages approximately 80 m normal to strike and dip and has currently been drill tested to 600 m below surface. Within the "Main Zone", the mineralization occurs along both a foliated fabric cut by discrete shear zones and as stockwork carbonate veining. The second mineralization domain is called the "Flow Zones". This mineralization occurs as numerous sigmoid and lens shaped bodies completely hosted within or adjacent to the "Main Zone". The gold mineralization within these bodies has good geologic and grade continuity. The rock is distinctive with strong foliation, pervasive shearing and can be correlated reasonably well between adjacent drillholes. The third mineralization domain is High Grade (HG) Indicator Zone. This was a probabilistic approach to define the zones of mineralization over 2 g/t Au. This HG Indicator Zone was constrained within the Main Zone and overlapped at times on the Flow Zone. Each of the three mineralization domains was modeled independently.

17.4 Assay Statistics

The assay data was first evaluated based on the geological domains (as defined in Table 17.2) using boxplots. Figures 17.1 illustrate the grade distribution by geological domain. The BMV and FZ units correspond primarily to the Main Zone and Flow Zones respectively.

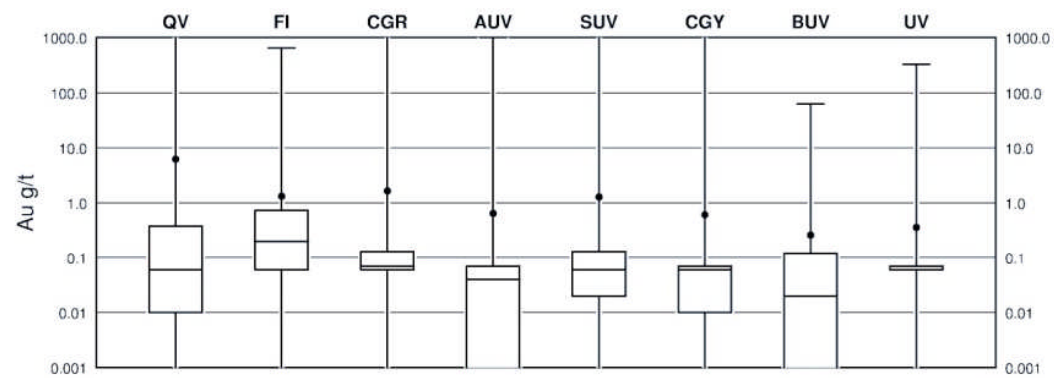
Table 17.2 Geological Domains

Code	Description
LC	Lost Core
MS	Massive Sulfides
QBX	Quartz Breccia
QV	Quartz Veining-(>50%)
FI	Felsic Intrusive
FIJ	Felsic Intrusive- Grey-White Albite Dyke With Clustered Pyrite
CGR	Green Fuchsitic Carbonatized Ultramafic Volcanics
AUV	Ankerite +/- Chlorite+/- Sericite Altered Ultramafic Volcanics
SUV	Silicified Grey Carbonatized Ultramafic Volcanics
CGY	Grey Ankerite-Talc Altered Ultramafic Volcanics
BUV	Bleached Calcite +/- Sericite Altered Ultramafic Volcanics
UV	Ultramafic Volcanics, e.g.., Serpentinite
TUV	Talc-Chlorite Altered Ultramafic Volcanics
CUV	Chloritized Ultramafic Volcanics
MV	Massive Mafic Volcanic Flows
PMV	Pillowed Mafic Volcanic Flows
BMV	Bleached Mafic Volcanic Flows -(Sericite-Ankerite Alteration)
MI	Mafic Intrusives
ARG	Argillite
SED	Sediments
FZ	Fault Zone
OB	Overburden

Figure 17.1 Assay Boxplots by Geological Domain

Wardrop, A Tetra Tech Company ***Brigus Gold Corp.***

Black Fox Mine Assays

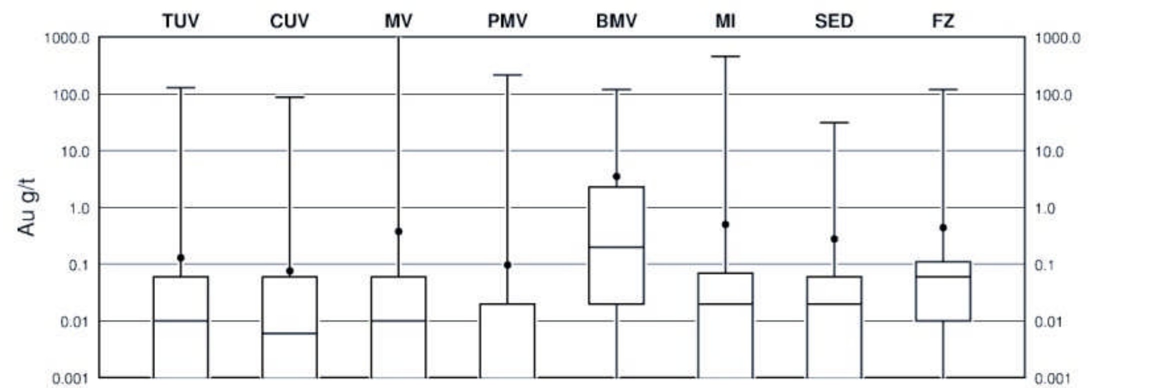


	QV	FI	CGR	AUV	SUV	CGY	BUV	UV	
Number of data	5245	15451	12165	38985	10274	17190	844	8247	Number of data
Mean	6.248	1.323	1.654	0.645	1.283	0.61	0.261	0.358	Mean
Maximum	3448.67	650.0	2516.76	995.92	3884.43	1174.0	63.05	325.0	Maximum
Upper quartile	0.38	0.73	0.13	0.07	0.13	0.07	0.12	0.07	Upper quartile
Median	0.06	0.2	0.07	0.04	0.06	0.06	0.02	0.06	Median
Lower quartile	0.01	0.06	0.06	0.0	0.02	0.01	0.0	0.06	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	Minimum
Variance	4223.47	79.3002	686.953	120.115	1565.37	146.586	4.9239	26.4484	Variance
CV	10.401	6.73	15.85	16.986	30.839	19.834	8.512	14.374	CV
Skewness	35.967	40.213	75.225	49.209	92.58	65.392	26.908	43.751	Skewness

Variable: Au g/t
Acceptable range: 0.0 to 10000.0
Weights: --equal--

QV file: QV0200.txt
FI file: FI0300.txt
CGR file: CGR0400.txt
AUV file: AUV0500.txt
SUV file: SUV0600.txt
CGY file: CGY0700.txt
BUV file: BUV0800.txt
UV file: UV0900.txt

Black Fox Mine Assays



	TUV	CUV	MV	PMV	BMV	MI	SED	FZ
Number of data	15724	9045	24415	13640	2730	4348	2723	3393
Mean	0.13	0.076	0.378	0.097	3.529	0.503	0.279	0.441
Maximum	129.36	88.49	3011.28	215.03	120.91	450.4	31.67	120.0
Upper quartile	0.06	0.06	0.06	0.02	2.318	0.07	0.06	0.11
Median	0.01	0.006	0.01	0.0	0.2	0.02	0.02	0.06
Lower quartile	0.0	0.0	0.0	0.0	0.02	0.0	0.0	0.01
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Variance	3.1596	1.6034	378.96	3.8865	80.6816	51.1075	2.193	10.1697
CV	13.652	16.584	51.543	20.419	2.546	14.226	5.299	7.224
Skewness	48.871	54.35	151.778	96.507	4.869	57.73	11.457	24.597

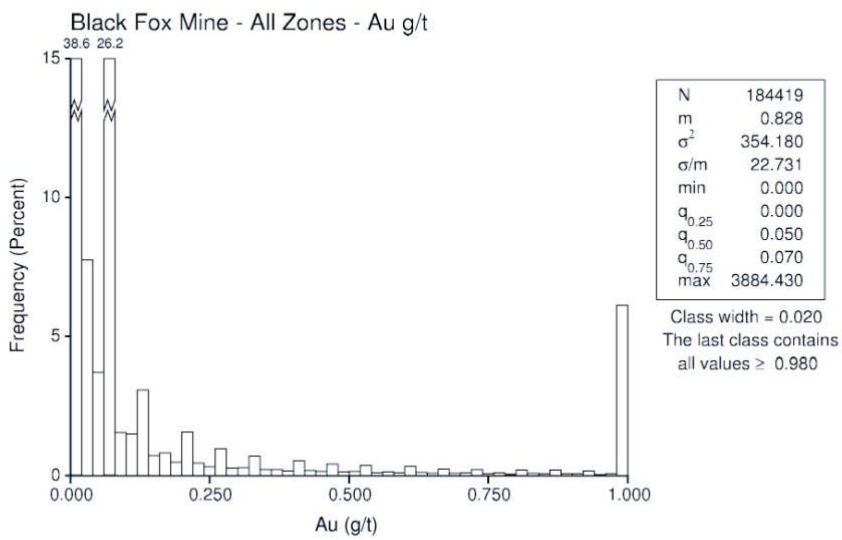
Number of data
Mean
Maximum
Upper quartile
Median
Lower quartile
Minimum
Variance
CV
Skewness

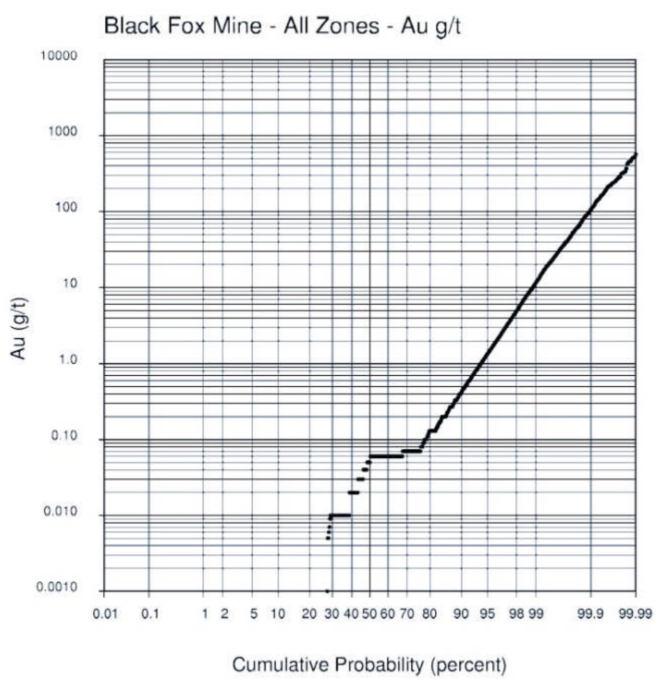
Variable: Au g/t
Acceptable range: 0.0 to 10000.0
Weights: --equal--

TUV file: TUV1000.txt
CUV file: CUV1100.txt
MV file: MV1200.txt
PMV file: PMV1300.txt
BMV file: BMV1400.txt
MI file: MI1500.txt
SED file: SED1600.txt
FZ file: FZ1700.txt

The gold assay data was also plotted on histogram and probability graphs to understand the basic statistical distribution of the raw data. The histogram plots show a strong positive skewness and the probability plot illustrates a continuous population set with no major changes in slope within the main data population. The probability plot does show outlier data values at the upper end of the grade distribution with a small break and change in slope.

Figure 17.2 All Zones, Au Assays





17.5 COMPOSITING

The raw drill data was composited into 2 m and 3 m intervals starting at the collar and continuing to the bottom of the hole. These composite intervals were developed for the corresponding blocks models, underground and open pit respectively. The appropriate codes for missing samples and no recovery were used during the compositing procedures.

The composites were plotted on histogram and probability plots for comparison to the raw assay data.

Figure 17.3 Au 2m Composites

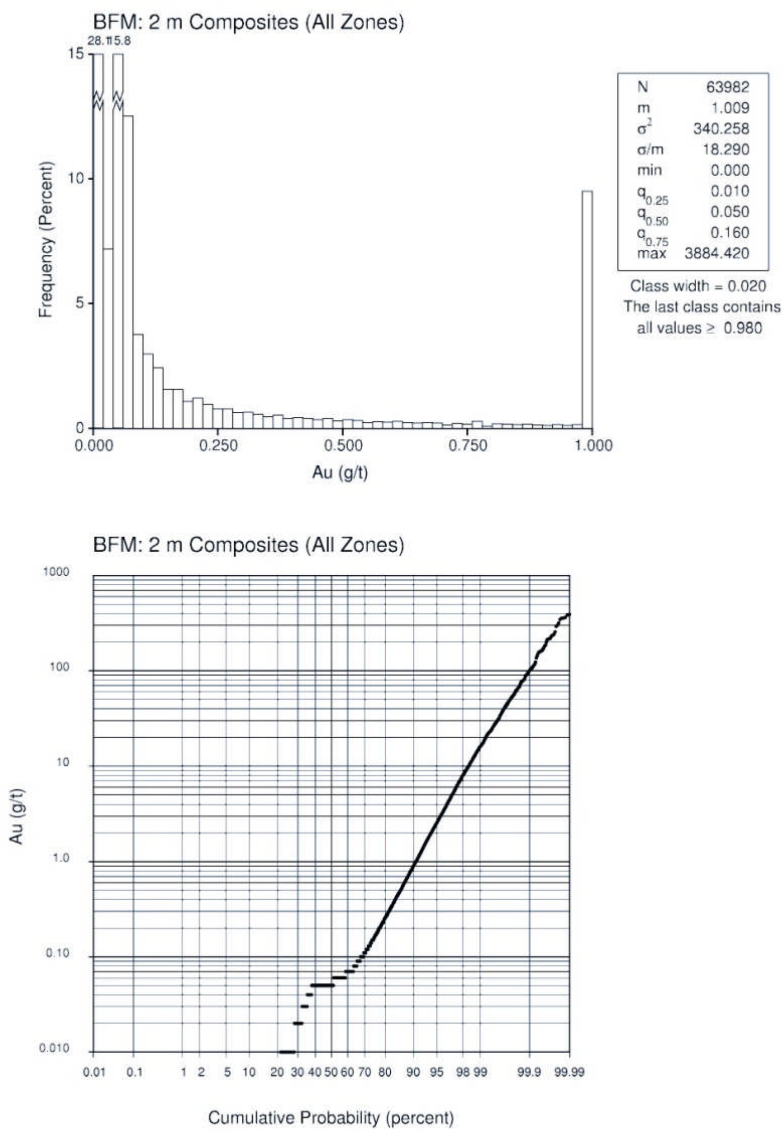
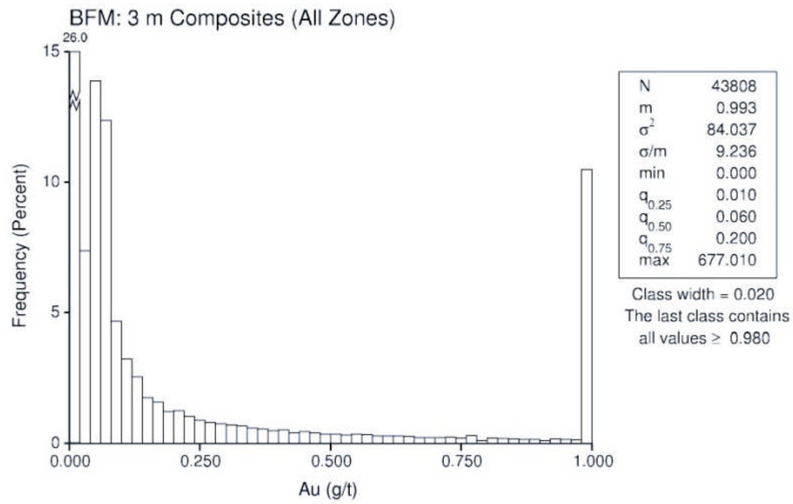
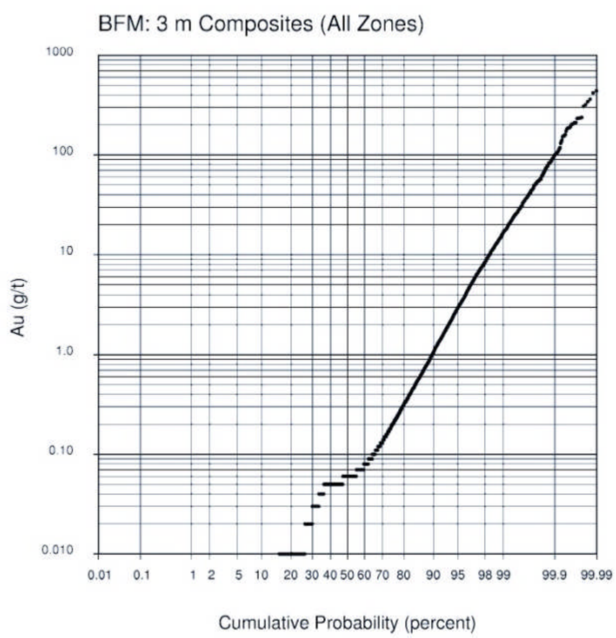


Figure 17.4 Au 3m Composites



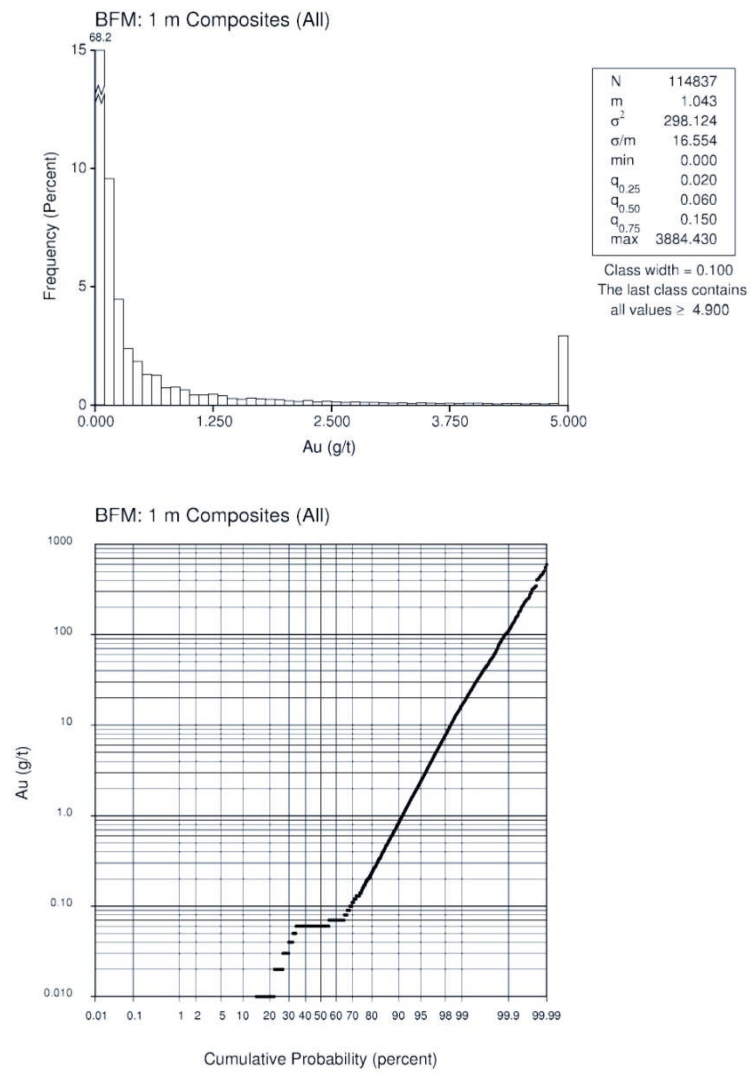


17.6 CAPPING

In mineral deposits having skewed distributions (typically with coefficient of variation greater than 1.0), a few high-grade assays can represent a large portion of the metal content. Often there is little continuity demonstrated by these assays. In other words, it can be assumed they occur at random within the deposit[1].

Review of the probability plot for 1 m composites showed a fairly continuous distribution of Au values up to about the 350 g/t Au level (Figure 17.5).

Figure 17.5 Au 1m Composites



[1] Detailed sampling will undoubtedly show short-scale geological controls which are responsible for the high grades observed and therefore the non-randomness of the high-grade occurrence. At the exploration stage, this short-scale information is not available and there is no apparent pattern to the occurrence of high-grade samples. Hence follows the assumption of a random spatial distribution.

Other methods such as the Parrish (Parrish, 1997) method and a cumulative capping spreadsheet analysis confirm capping is required at Black Fox.

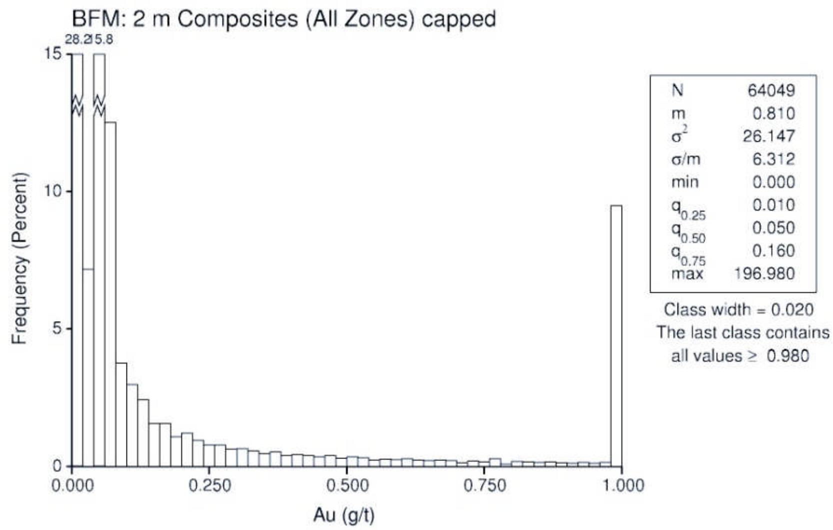
Based on the outlier nature of the composites as evaluated by mineralization and elevation, the Au values were capped at 250 g/t Au before compositing for the HG Indicator Zone above 9820 m elevation and to 180 g/t Au for the remaining areas (Table 17.3). This resulted in 31 assays being capped in the High Grade Indicator zone and 10 assays in the remaining zones.

Table 17.3 Capping Levels

Mining Area	Elevation (m)	Rocktype	Zone	Cap Level (Au g/t)
Open Pit	> 9820	Flow	101	180
		Main	103	180
		High Grade Indicator	107	250
Underground	< 9820	Flow	101	180
		Main	103	180
		High Grade Indicator	107	180

Figures 17.6 and 17.7 illustrate the capped statistics for the 2 m and 3 m composites.

Figure 17.6 Au 2 m Capped Composites



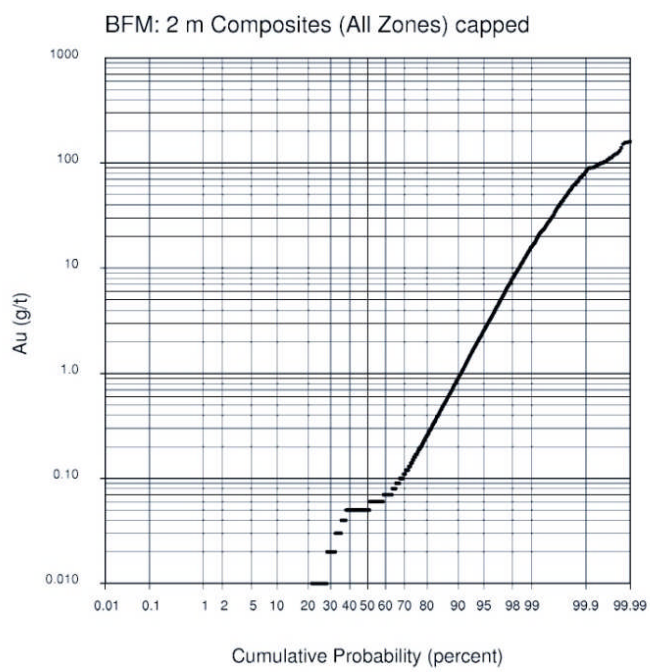
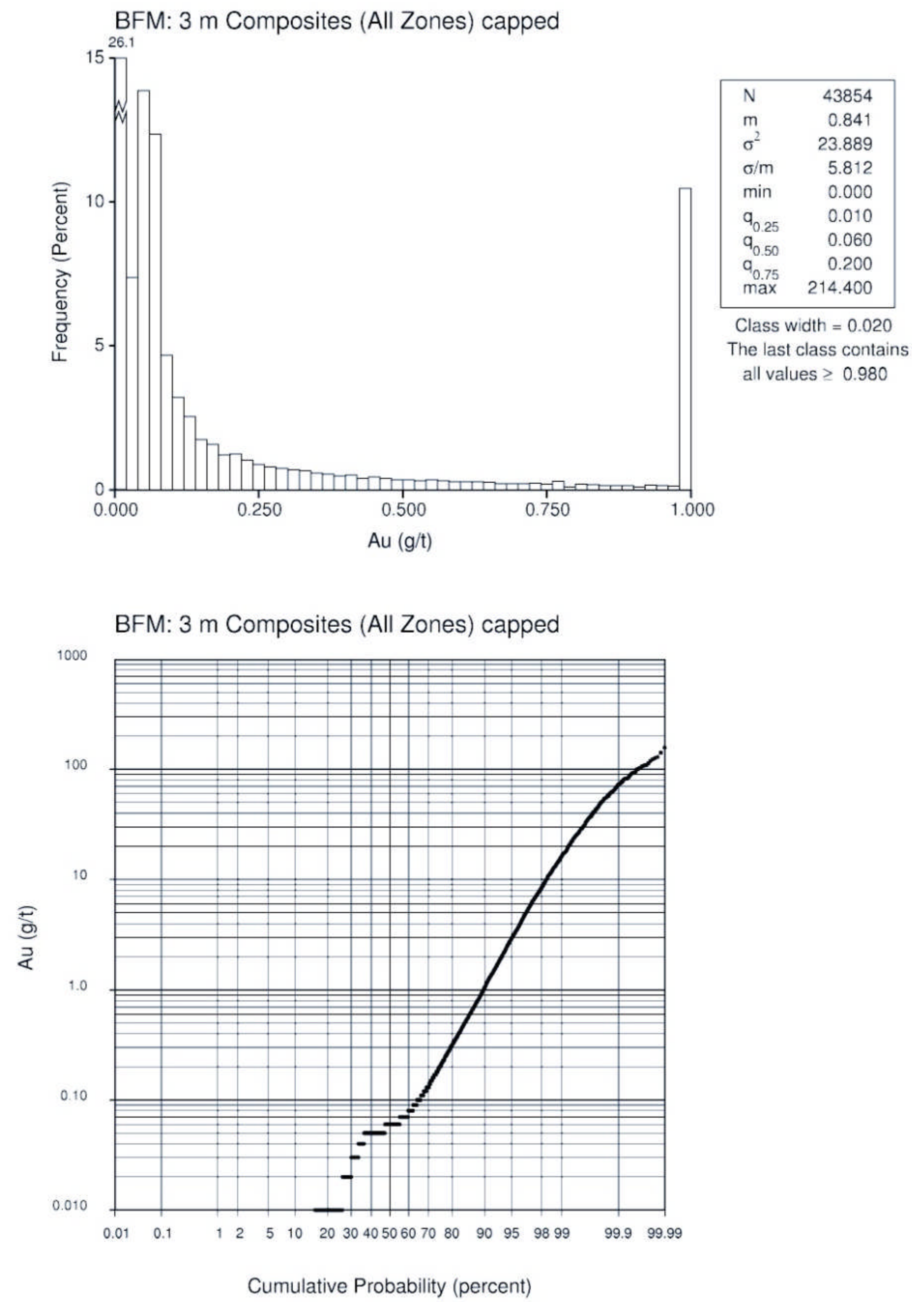


Figure 17.7 Au 3 m Capped Composites



17.7 SPECIFIC GRAVITY

Specific gravity testing has been carried out, in house, by Brigus and described by Prenn (2006) as follows:

"*A total of 1,218 density tests have been completed by Brigus from core intervals. The average density of mineralized material is 2.78 grams per cubic centimetre (g/cm^3), while the average density of unmineralized material is 2.85 g/cm^3.*" For additional confirmation, Brigus further refined this data by sending an additional 107 samples to an outside laboratory for independent analysis. The laboratory results reported an average density of 2.84 g/cm^3 for the ore. This value was used for all material in this resource estimate. Wardrop recommend further analysis by rock types and mineralization styles to confirm that one value is appropriate for the whole model.

17.8 VARIOGRAM ANALYSIS

Geostatisticians use a variety of tools to describe the pattern of spatial continuity or strength of the spatial similarity of a variable with separation distance and direction. One of these is the correlogram, which measures the correlation between data values as a function of their separation distance and direction. If we compare samples that are close together, it is common to observe that their values are quite similar and the correlation coefficient for closely spaced samples is near 1.0. As the separation between samples increases, there is likely to be less similarity in the values and the correlogram tends to decrease toward 0.0. The distance at which the correlogram reaches zero is called the "range of correlation" or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the "range of influence" of a sample; it is the distance over which sample values show some persistence or correlation. The shape of the correlogram describes the pattern of spatial continuity. A very rapid decrease near the origin is indicative of short scale variability. A more gradual decrease moving away from the origin suggests more short scale continuity. A plot of 1-correlation is made so the result looks like the more familiar variogram plot.

Another tool for describing spatial correlation is the indicator variogram. Correlograms show the decrease in the strength of the relationship between grades by distance and direction. Conversely, variograms show the increase in dissimilarity with distance and direction. In the same way that correlograms tend to decrease toward zero, the variogram tends to increase toward a sill or the overall variance of the variable of interest. The indicator is simply a value of 0 or 1 that is assigned to a data location according to whether the grade exceeds some threshold value. The indicator variogram is the variogram calculated using the indicator data. Indicator variograms are useful in describing the spatial continuity of different segments of the grade distribution. They are most useful in examining properties of spatial continuity in the higher-grade samples. Wardrop employed this method to define the High Grade Indicator zone.

The approach used to develop the variogram models used Sage© software.. Directional sample correlograms were calculated along horizontal azimuths of 0, 30, 60, 120, 150, 180, 210, 240, 270, 300, and 330 degrees. For each azimuth, sample correlograms were also calculated at dips of 30 and 60 degrees in addition to horizontally. Lastly, a correlogram was calculated in the vertical direction. Using the thirty-seven sample correlograms an algorithm determined the best-fit model nugget effect and two-nested structure variance contributions. After fitting the variance parameters the algorithm then fitted an ellipsoid to the thirty-seven ranges from the directional models for each structure. The anisotropy of the correlation was given by the range along the major, semi-major, and minor axes of the ellipsoids and the orientations of these axes for each structure.

17.8.1 High Grade Indicator Variograms

Variogram analysis was conducted on 1.0 m drillhole composites. The composites were flagged to differentiate them into data sets to be used for an indicator estimation technique. All composites greater than 2.0 g/t Au were flagged as "High Grade". Variograms were then constructed using Sage software as discussed above. Table 17.4 summarizes the variogram structures, rotations and ranges. The model was spherical and used the ZXZ-RRR Gemcom© rotation convention.

Table 17.4 High Grade Indicator, Correlogram Model

Domain	Nugget	Sill	Rot. Z	Rot. X'	Rot. Z'	Range Z'	Range X'	Range Y'
HG Ind	0.600	0.375	25	50	0	4.80	12.50	3.10
		0.025	25	50	0	18.40	50.50	28.00

17.8.2 Gold Grade Variograms

Variogram analysis for the gold grade was conducted using 2 m and 3m composites. Variograms were constructed using Sage© software as discuss in Section 17.8. Table 17.5 summarizes the variogram structures, rotations and ranges for the 2m composites by mineralization domain. Table 17.6 similarly summarizes the correlogram models for the 3 m composites. The models were all spherical and used the ZXZ-RRR Gemcom© rotation convention.

Table 17.5 2 m Composite Gold Grade Correlogram Model

Domain	Nugget	Sill	Rot. Z	Rot. X'	Rot. Z'	Range Z'	Range X'	Range Y'
101	0.550	0.404	-4	-2	138	5.60	10.70	2.90
		0.046	-59	-83	-9	13.00	125.40	726.90
103	0.300	0.693	-69	75	106	5.40	4.50	66.00
		0.007	56	-8	-58	25.20	94.60	71.00
107	0.300	0.636	-65	-82	142	18.20	4.60	7.80
		0.064	-2	90	35	38.40	11.10	292.50

Table 17.6 3 m Composite Gold Grade Correlogram Model

Domain	Nugget	Sill	Rot. Z	Rot. X'	Rot. Z'	Range Z'	Range X'	Range Y'
101	0.400	0.500	12	44	-10	32.30	56.90	13.40
		0.100	-40	-38	-3	81.10	22.10	40.60
103	0.400	0.500	-32	18	38	8.00	5.00	2.00
		0.100	2	-8	-13	5.00	10.00	30.00
107	0.400	0.450	14	-43	-102	2.00	17.20	10.60
		0.150	-59	19	60	59.60	6.10	63.70

17.9 GRADE INTERPOLATION

Geologic wireframes were used to model the "Main Zone", the "Flow Zone" and the "High Grade Indicator Zone". Table 17.7 summarizes the corresponding block model domain codes for each of these zones.

Table 17.7 Block Model Domain Codes

Code	Description	Resource Class
101	Flow Zone	Inferred
103	Main Zone	Inferred
104	Hanging Wall	
105	Overburden	
106	Old Workings	
107	HG Indicator Zone	Inferred
111	Flow Zone	Indicated
113	Main Zone	Indicated
117	HG Indicator Zone	Indicated

Modelling consisted of grade interpolation by ordinary kriging (OK). Nearest-neighbour (NN) and inversed distance weighted (IDW) grades were also determined for validation purposes. The grade interpolation used search ellipses as defined in Table 17.8. These parameters were based on the geological interpretation and variogram analysis. A three-pass strategy was used in each domain: open pit (Block Model 2009A) and underground (Block Model 2009A_4M).

The number of composites used in estimating a model block grade followed a strategy that matched composite values and model blocks sharing the same ore code or domain during the first and second passes. For the first and second passes, composites from at least three drill holes were used. The third pass allowed composites from a minimum of two drill holes to be used (longer search distances). Estimates used a maximum of fifteen composites per model block. Further details are found in Tables 17.9 and 17.10

Table 17.8 Search Ellipses

Profile	Search Anisotropy	Rotation about Z	Rotation about X	Rotation about Z	Anisotropy X	Anisotropy Y	Anisotropy Z	Search Type	Min. # of Octants	Max. Samples per Octant
OPPPASS1	Rotation ZXZ	20	55	0	17	15	10	Octant	2	3
OPPPASS2	Rotation ZXZ	20	55	0	27	30	10	Octant	2	4
OPPPASS3	Rotation ZXZ	20	55	0	50	60	20	Ellipsoidal	-	-
UGPPASS1	Rotation ZXZ	20	45	0	17	15	10	Octant	2	3
UGPPASS2	Rotation ZXZ	20	45	0	27	30	10	Octant	2	4
UGPPASS3	Rotation ZXZ	20	45	0	50	60	20	Ellipsoidal	-	-

Table 17.9 Interpolation Parameters for Open Pit Model

Block Model	Point Area	Grade Field	Grade Block Model	Interpolation Profiles	Max. Comp/DH	Minimum # of Comp.	Maximum # of Comp.	Search Ellipse	Semi-Variogram	Special Models
2009A	3mComp	GOLD	AUOKC	3MAUOKC1	SVALUE=1	3	15	OPPASS1	OP3M101	NCOMP1 = # of composites used
									OP3M103	
									OP3M107	
			AUIDC	3MAUIDC1		3	15	OPPASS1		
			AUOKC	3MAUOKC2		3	15	OPPASS2	OP3M101	NCOMP2 = # of composites used
									OP3M103	
									OP3M107	
			AUIDC	3MAUIDC2		3	15	OPPASS2		
			AUOKC	3MAUOKC3		2	15	OPPASS3	OP3M101	NCOMP3 = # of composites used
									OP3M103	
									OP3M107	
			AUIDC	3MAUIDC3		2	15	OPPASS3		
			AUNNC	3M_AUNNC	SVALUE=1	1	1	OPPASS3		DIST = Distance to nearest point
		RVALUE	AUOK	3MAUOK1	SVALUE=1	3	15	OPPASS1	OP3M101	
									OP3M103	
									OP3M107	
			AUID	3MAUID1		3	15	OPPASS1		
			AUOK	3MAUOK2		3	15	OPPASS2	OP3M101	
									OP3M103	
									OP3M107	
			AUID	3MAUID2		3	15	OPPASS2		
			AUOK	3MAUOK3		2	15	OPPASS3	OP3M101	
									OP3M103	
									OP3M107	
			AUID	3MAUID3		2	15	OPPASS3		
			AUNN	3M_AUNN	SVALUE=1	1	1	OPPASS3		

Table 17.10 Interpolation Parameters for Underground Model

Block Model	Point Area	Grade Field	Grade Block Model	Interpolation Profiles	Max. Comp/DH	Minimum # of Comp.	Maximum # of Comp.	Search Ellipse	Semi-Variogram	Special Models
2009A_4M	2mComp	GOLD	AUOKC	4MAUOKC1	SVALUE=2	5	15	UGPASS1	UG2M101	NCOMP1 = # of composites used
									UG2M103	
									UG2M107	
			AUIDC	4MAUIDC1		5	15	UGPASS1		
			AUOKC	4MAUOKC2		5	15	UGPASS2	UG2M101	NCOMP2 = # of composites used
									UG2M103	
									UG2M107	
			AUIDC	4MAUIDC2		5	15	UGPASS2		
			AUOKC	4MAUOKC3		3	15	UGPASS3	UG2M101	NCOMP3 = # of composites used
									UG2M103	
									UG2M107	
			AUIDC	4MAUIDC3		3	15	UGPASS3		
			AUNNC	4M_AUNNC	SVALUE=1	1	1	UGPASS3		DIST = Distance to nearest point
		RVALUE	AUOK	4MAUOK1	SVALUE=2	5	15	UGPASS1	UG2M101	
									UG2M103	
									UG2M107	
			AUID	4MAUID1		5	15	UGPASS1		
			AUOK	4MAUOK2		5	15	UGPASS2	UG2M101	
									UG2M103	
									UG2M107	
			AUID	4MAUID2		5	15	UGPASS2		
			AUOK	4MAUOK3		3	15	UGPASS3	UG2M101	
									UG2M103	
									UG2M107	
			AUID	4MAUID3		3	15	UGPASS3		
			AUNN	4M_AUNN	SVALUE=1	1	1	UGPASS3		

17.10 Block Model Validation

Wardrop distinguishes between verification from validation as follows:

- Verification is a manual (e.g. visual inspection) or quasi-manual (e.g. spreadsheet) check of the actual procedure used.
- Validation is a test for reasonableness using a parallel procedure, which may be either manual or (but is usually) a computer-based procedure.

17.10.1 Visual Checks

Interpolated block grades, resource classification, geological interpretation outlines and drill hole composite intersections were verified on screen for plan and section. Based on visual inspection by Wardrop, the block model grades appeared to honour the data well. The estimated grades exhibit a satisfactory consistency with the drillhole composites.

17.10.2 Global Comparison

Wardrop verified the block model estimates for global bias by comparing the average gold grades (with no cut-off) from the model (ordinary kriging) using nearest-neighbour estimates. The nearest-neighbour estimator produces a theoretically unbiased estimate of the average value when no cut-off grade is imposed and is a good basis for checking the performance of different estimation methods. The results (Table 17.11) show no evidence of bias in the estimate.

Table 17.11 Global Comparison

Domain	Grade Item	Au g/t
2009A (OP)	AUOKC	0.578
	AUNNC	0.717
2009A_4M (UG)	AUOKC	0.666
	AUNNC	0.785

Histogram and probability plots were created of the block model data and reviewed with respect to the corresponding composite plots. No apparent bias was noted. Table 17.12 summarizes the block model statistics.

Table 17.12 Composite versus Block Model Statistics

Domain	Grade Item	Mean Grade	Std Deviation	Coeff. Of Var.
2009A (OP)	3m Comp	0.841	4.888	5.812
	AUOKC	0.609	2.262	3.715
2009A_4M (UG)	2m Comp	0.810	5.113	6.313
	AUOKC	0.677	2.330	3.442

Typical block model sections of Au grade with Au composite data are shown in Figures 17.8 and 17.9.

Figure 17.8 Section 10350E Block Model 2009A_4M Cross Section

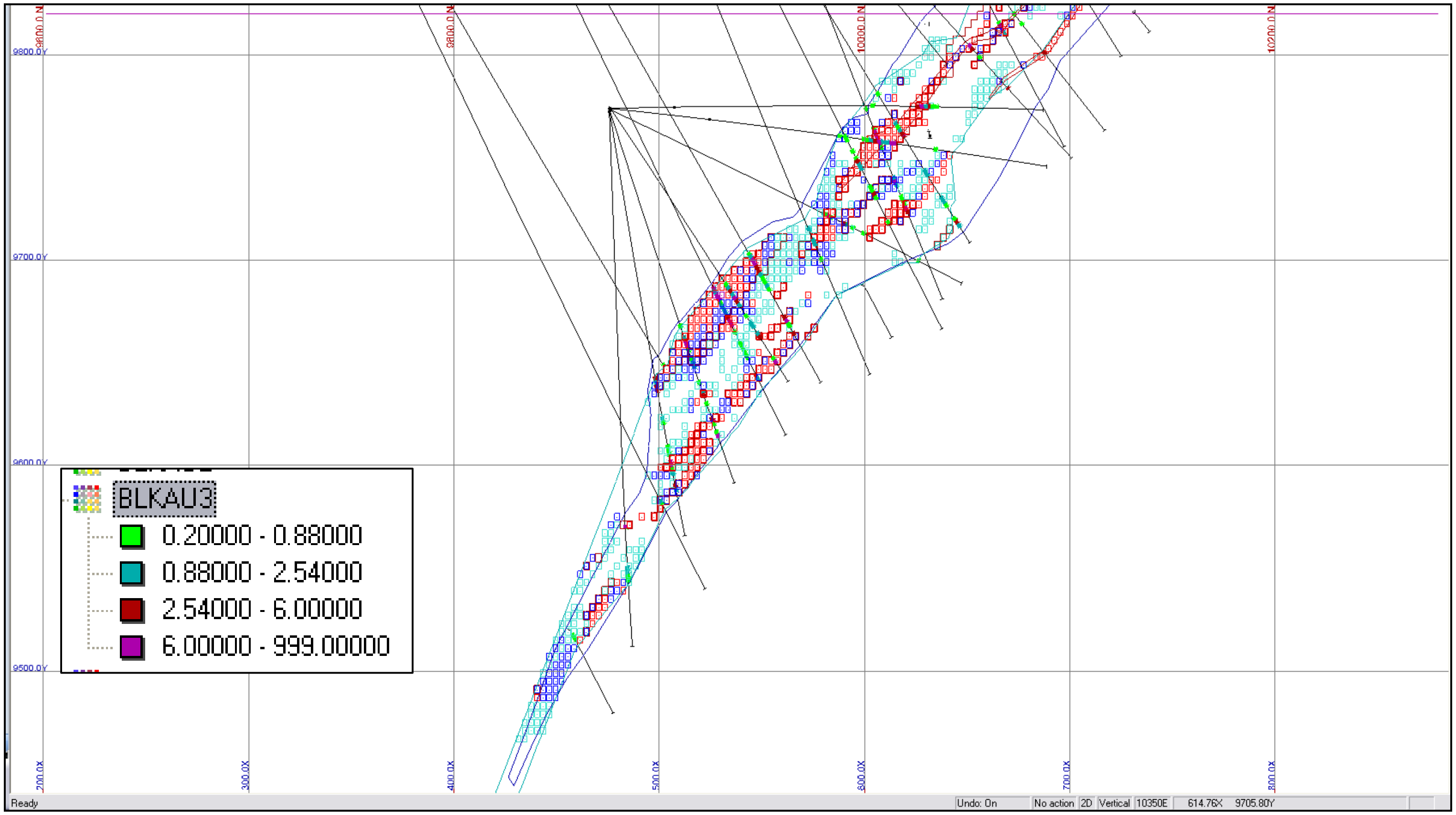
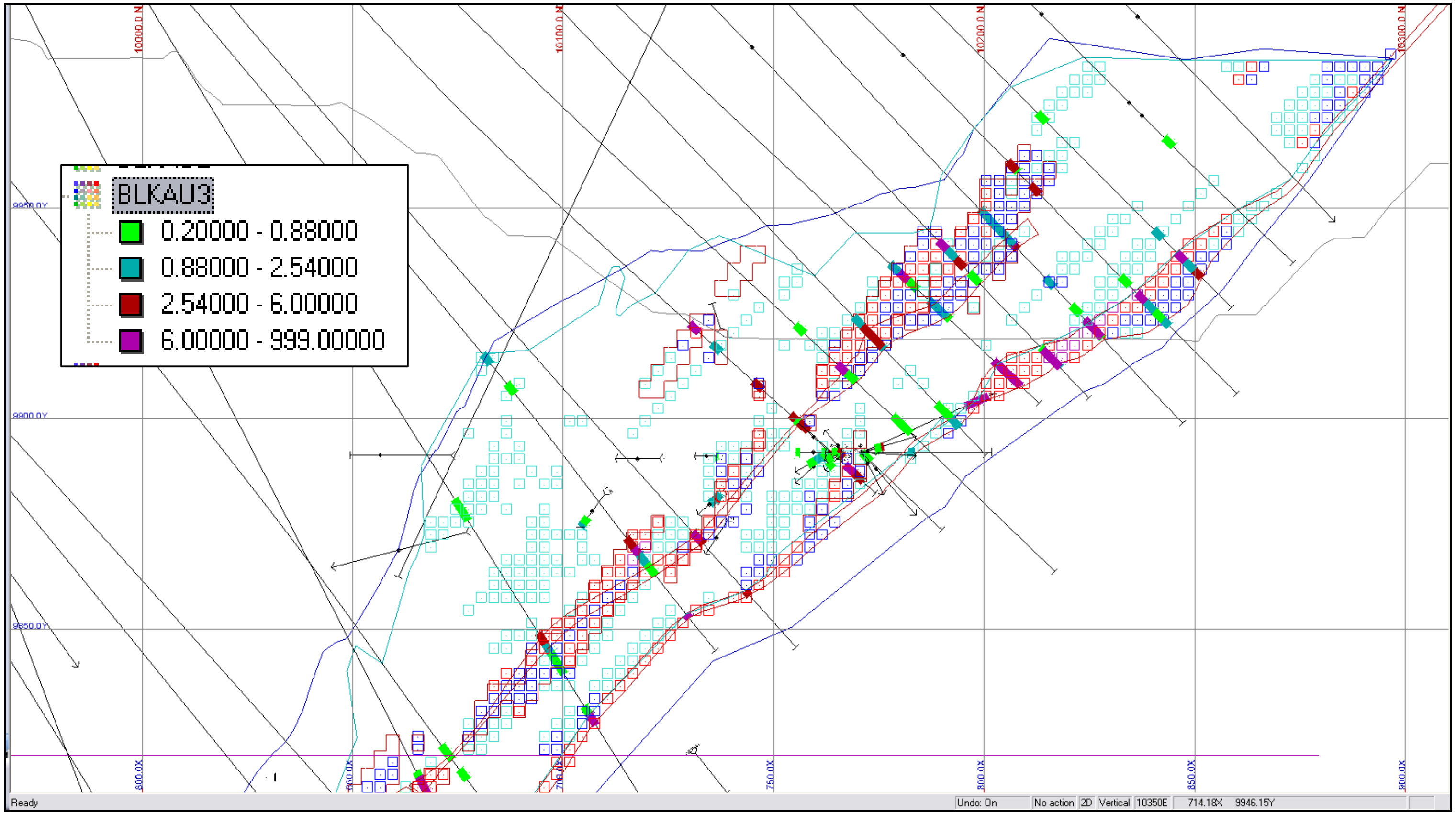


Figure 17.9 Section 10350E Block Model 2009A Cross Section



17.10.3 ADEQUACY OF RESOURCE ESTIMATION METHODS

The Black Fox deposit has been estimated using modern block modeling techniques in Gemcom© GEMS 6.2.4. This included proper geologic input, appropriate block model cell sizes, grade capping, assay compositing and reasonable interpolation parameters. The results have been verified by visual review and statistical comparisons between the estimated block grades and the composites used to assign them. The ordinary kriging models have also been validated with alternate estimation methods: Nearest Neighbour and Inverse Distance Weighting. No biases have been identified in the model.

17.11 MINERAL RESOURCE CLASSIFICATION AND RESOURCE STATEMENT

17.11.1 RESOURCE CLASSIFICATION

The Mineral Resources are classified under the categories of Measured, Indicated and Inferred Mineral Resources according to CIM guidelines. Classification of the Resources reflects the relative confidence of the grade estimates, as a function of many factors including primarily; assay data quality, QA/QC procedures, quality of density data, and sample spacing relative to geological and geo-statistical observations regarding the continuity of mineralization.

No environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to the author that may affect the estimate of mineral resources.

The resource model blocks were classified into Indicated and Inferred categories based on the level of confidence in the grade estimate for each block. This was accomplished with a combination of two main criteria: the number of drill holes (which in part reflects the number of samples used) and the distance to the nearest of the sample points.

Indicated Resources were categorized based on a minimum of three drill holes when the nearest sample point was less than or equal to 20 m. The remaining blocks were classified as Inferred Resources. No blocks were categorized as Measured.

17.11.2 MINERAL RESOURCE STATEMENT

The mineralization of the Black Fox Mine as of October 31, 2010 is classified as Indicated and Inferred resources. The classified mineral resources are shown in Table 17.13. The mineral resource is reported at a 0.88 g/t Au cut-off grade for the open pit and at 2.54 g/t Au cut-off grade for the underground. These cut-offs have been developed by mine engineering as outlined in Section 19.

All blocks historically mined underground were coded as Domain 106 and were not included in the reported resource. The open pit survey was provided by Brigus Gold as of October 31, 2010 so material above this surface is not included in the resource statement.

Table 17.13 Resource Statement, Black Fox Mine

Mining Potential	Block Model	Cutoff (g/t Au)	Resource Class	Tonnes	Capped Au (g/t)	Contained Au (koz)
Open Pit	2009A	>= 0.88	Indicated	3,164,200	4.445	452.2
		>= 0.88	Inferred	667,100	2.61	56
Underground	2009A_4M	>= 2.54	Indicated	2,504,800	7.182	579.2
		>= 2.54	Inferred	115,200	5.816	21.5
Total Indicated				**5,669,000**	**5.654**	**1,031**
Total Inferred				**782,300**	**3.082**	**78**

18.0 OTHER DATA AND INFORMATION

18.1 MINE UNDERGROUND GEOTECHNICAL

The underground portion of the Black Fox Project is expected to extend from below the existing mine (near 200 m depth below surface) to approximately 500 m depth below surface.

Geotechnical data by logging has been completed by Brigus Gold staff and additional investigations are recommended to be completed during mine development.

The mine layout consists of a series of declines and ramps to access a cut and fill type excavation methodology. At present, all primary ramps will be 5 m x 5 m height and width.

Ore access drifts are planned at 4 m wide by 4 m high with stopes up to 6 m wide and 4 m high planned. Where the ore width is larger than 6 m, the mining method selected is a sill drift (6 m wide by 4 m high), and then cemented backfill prior to drifting in adjacent ore. Succeeding overhead cuts will be mined in a similar method but using uncemented rockfill where multiple pass drift and fill is not required.

18.1.1 INFORMATION AVAILABLE

Reports dealing with geotechnical aspects of the Black Fox mine are listed below as well as Referred to the References list at the end of this report.

AMEC Earth & Environmental, W.S. (Stu) Anderson, P. Eng., Associate Geotechnical Engineer 2008 "Black Fox Project, Open Pit and Underground Geotechnical Design"

The AMEC report as summarised below describes geotechnical information gathered by both Apollo and AEE personnel for the Black Fox Project's open pit and underground mine.

Investigations and some analysis were completed in the early winter of 2006. Additional analysis was completed in 2007 and 2008. The report summarises the results of the data analysis and recommends geotechnical designs for the open pit and underground mine. Additional work was recommended to confirm the results and assumptions made in the report.

Underground investigations to confirm the design parameters consisted of the following:

Underground geotechnical mapping of the walls of the main access adit and 235 Drift was reviewed within the current underground workings:

- Review of the geotechnical drillhole data obtained by Apollo Gold prior to 2006 and by Brigus Gold to 2010.
- Lithologic sections of the ore body were reviewed.
- Review of previous geotechnical reports on the underground mine provided by Brigus Gold staff.
- Review of any reportable "falls of ground" reports (two in pit and one underground).

The geotechnical evaluation was conducted with the expectation that it would support an overhand Cut & Fill (C&F) method.

Wardrop audited the following in order to confirm the evaluation:

- review of relevant detailed structural/geotechnical hole data
- review of geotechnical line mapping of underground exposures
- structural considerations of fabric and weak zones within the crown pillar
- evaluation of crown and rib pillar stability – empirical
- interviews with mine staff and ground support installation miners
- site visit and local assessment.

The evaluation includes a rock mass assessment and identifies geotechnical domains. It covers stability assessments for the initial mining concepts that involved open stopes, rib pillars and the proposed crown pillar.

The rock mass characterization (Table 18.1) showed that the Black Fox ore zone is of fair to poor rock mass quality.

The selection of a C&F mining method is supported by this evaluation.

A distribution of the rock mass quality observed within the underground mine is presented in table form below.

Table 18.1 Estimated and Observed Distribution of Rock Quality (RMR)

Ultramafic and Mafic Volcanics				
Very Poor Condition	**Poor Condition**	**Fair Condition**	**Good Condition**	**Very Good Condition**
RMR = 23 to 44%	RMR = 44 to 56%	RMR = 56 to 65%	RMR = 65 to 77%	RMR = 77 to 85%
5%	20%	55%	15%	5%

In summary, 75% of the rock mass within the underground mine is expected and observed to be of fair to good quality.

18.1.2 MAN ENTRY MINING

The mine design was assessed by identifying constraints such as drill access, ore body type and grades and practical mining considerations. The cut and fill mining will require man entry and therefore support requirements will be more demanding for safety reasons than for non-entry stoping methodology. Support will primarily be a function of the span, rock mass strength and characteristics, and presence of adverse geologic structure.

18.2 CRITICAL SPAN

RMR values identified from the analysis of the drillhole data can provide an estimate of span for man entry mining. The Span Design Curve (Wand et al, 2002) is defined as the largest diameter circle which can be drawn within the boundaries of the exposed back and hangingwall. It also assumes local support is installed where necessary (for example, 1.8 m long bolts at a 1.2 m x 1.2 m spacing). The expected RMR values and critical spans expected for the cut and fill stopes (up to 6 m in width) and temporary access drives (4 m x 4 m) are shown in Table 18.2.

Table 18.2 RMR Values vs. Critical Spans

Critical Spans for Ultramafic/Mafic Volcanics			
	Fair/Good	**Poor**	**Very Poor**
RMR	68%	46%	25%
Critical Span	13 m	2.5 m	0 m

The above values assume the following:

- local support is installed where necessary
- stress induced failure does not occur in the back or hangingwall
- “stable” assumes a short term duration (3 months)
- discrete wedges must be supported where identified.

The state of stress within the proposed mine is continuing to be investigated in order to insure critical span applicability in various domains and depths.

Current support in permanent openings include #6 grouted rebar, 1.8 m long and 1.2 m x 1.2 m spacing in back and walls with heavy (#6) gauge mesh. Requirements to escalate support are planned to include 50-100 mm of shotcrete where poor or faulted rock mass conditions are encountered.

Support in the C&F stopes will be accomplished with splitsets, 1.8 m long at 1.2 m x 1.2 m spacing in back with light (#9) gauge mesh.

Brigus Gold supplements its geotech reviews and planning efforts through regular geotech consulting services as provided by Mr. Greg Hunt, P. Eng.

18.2.1 STRUCTURAL STABILITY

Ramps, access drifts, orientations parallel to the strike of the ore body (azimuth of 125°) and perpendicular to the ore body (azimuth of 035°) were assessed for opening dimensions of 5 m wide by 5 m high, 10 m wide by 5 m high and 20 m wide by 5 m high. Structural sets identified in Section 4 were used for the analysis.

Analysis of the factor of safety for the maximum wedge found for each opening dimension was completed. The analysis was carried out using a basic friction angle of 35°, zero cohesion along the joint surfaces and a hydrostatic confinement of 10 tonnes/square metre (m^2). The use of 10 tonnes/m^2 accounts for any stress confinement, or joint bridging that may occur along the discontinuity surfaces.

Standard rock bolt patterns were used in the analysis to see if they were sufficient to support the wedges that were possible. The bolts are assumed to be fully grouted and plated #6 rebar (Grade 690 mega Pascal (MPa)), with breaking strength of 18.5 tonnes. It is assumed that the plate strength will be equal to the bolt strength. The bolts are assumed to be on a 1.2 m by 1.2 m pattern with no offset spacing considered. Bolt lengths were varied from 1.8 m to 3.7 m depending upon the opening span. Screen was not considered in the analysis but it is assumed that screen will be placed to catch any loose wedges that fall within the extents pattern (maximum two tonnes capacity).

The following observations were made from the analysis:

The 5 m by 5 m drift dimension was found to be stable with a standard support pattern. When no confining pressure is applied to the shallow wedges in the 5 m by 5 m configuration, the possible wedge combinations are stable;

The 10 m and 20 m span dimensions where found to possess unstable wedges even with standard support. It should be noted however that the apex height of the worse case wedges (cathedral wedges) was between one and two times longer than the span dimensions, these areas will likely require cable bolts if these wedges are present;

Smaller wedge weights are observed for tunnel with orientations of 035° compared to those of 125°.

A summary of the support requirements are provided in Table 18.3.

Table 18.3 Support Types

Estimated Spans and Support - Man Entry				
Ultramafic Volcanics/Mafic Volcanics – Spans		**Critical failure**		**Support**
		0350 Trend	**1250 Trend**	
Fresh	< 5 m	Wedge	Wedge	Mesh, #6 Rebar, 1.2 x 1.2 m, 1.8 m long in back; 1.5 m x 1.5 m in walls
	5 m to 10 m	Wedge	Wedge	Mesh, #6 Rebar, 1.2 x 1.2 m, 2.4 m long in back; 1.5 m x 1.5 m in walls
	10 m to 18 m	Wedge	Wedge	Mesh, #6 Rebar, 1.2 x 1.2 m, 3.7 m long in back; 2.4 m long and 1.5 m x 1.5 m in walls
Altered	< 5 m	Wedge	Wedge	Mesh, #6 Rebar, 1.2 x 1.2 m, 1.8 m long in back; 1.5 m x 1.5 m in walls
	5 m to 10 m Recommend 5 m. If greater spans than 5 m use local support + 50 mm shotcrete and evaluate. Need to confirm if unfavorable structure exists.	Wedge	Wedge	Mesh, #6 Rebar, 1.2 x 1.2 m, 2.4 m long in back; 1.5 m x 1.5 m in walls + 50 mm shotcrete and evaluate.
	10 m to 20 m these spans not recommended.	Wedge	Wedge	Not recommended - span too great use panel mining ie: drift and fill.
Faulted*	< 5 m	Rock mass/wedge	Rock mass/wedge	Mesh, #6 Rebar, 1.2 x 1.2 m, 1.8 m long in back; 1.5 m x 1.5 m in walls + 100 mm shotcrete in back and walls.

* In faulted zones, short rounds (1 m -2 m) and spilling for advance will need to be considered. If cathedrawl wedges are present, may require cable bolts.

18.2.2 Additional Investigations

The geotechnical information within the proposed underground mine will expand as development continues to depth.

A significant amount of experience has been gathered from the existing underground workings, these workings are presently within 250 m of the surface and the future mine will extend to beyond 500 m below surface.

Modeling based on new drilling to define stresses, confirm rock mass characteristics, groundwater hydrology and structural orientations is recommended to continue on a regular basis throughout the mine life.

19.0 MINING OPERATIONS

The Black Fox orebody is a structurally defined zone of mineralization dipping at approximately 40º to 50º and is defined along strike. Mining will include open pit and underground operations. Ore will be milled using conventional crushing, grinding and CIP recovery technology.

The open pit operation, in combination with underground production, is designed for a 2,000 t/d mill throughput and primarily uses two, 4 m^3 CAT 385CL hydraulic backhoe excavators for ore and one, 12 m^3 Komatsu PC2000 hydraulic backhoe excavator for rock waste, both loading 91 tonne CAT 777F rigid dump trucks. A 6.5 m^3 CAT 988H front-end loader is used as backup to the excavators and will also be used for loading the backfill for the underground operations. This front-end loader is to be replaced in the near future with a larger CAT 992G. An Atlas Copco CM785 and DM45 is being used for primary production drilling. Another CM785 is being used for drilling around the old workings to allow the controlled collapse of these areas to create safe working conditions.

The major support equipment includes two CAT D9T production dozers with rippers, two CAT D8 general purpose dozers, a CAT16H motor grader, a CAT 980H general purpose front end loader, a DX340 rockbreaker excavator and a CAT 10K water truck. Ancillary equipment includes fuel, lube, mechanics, welding and flatbed trucks together with lighting towers, crew van and other small pieces of equipment. The average stripping ratio based on the mine production schedule as of January 1, 2011 is 10:2 (waste to ore) with 3.1 Mt of ore grading at 3.21 g/t gold.

There is considerable overburden in the form of glacial till that must be removed to expose the ore and rock. A fleet of equipment dedicated to this operation includes, two CAT 320CL backhoe excavators loading twelve CAT 740 articulated dump trucks. Support equipment includes two CAT D6 low ground pressure dozers.

Black Fox underground mining will incorporate cut and fill (C&F) mining, utilizing a mining cross-section of 4 m high x 5 to 6 m wide for the cut and a rock fill backfill material with limited cement in specific areas only. C&F was selected due to the versatility of the method to allow the minimal amount of dilution while meeting the production throughput target of 1,000 t/d. Ore from underground will total 2.9 Mt at an average grade of 5.94 g/t.

The Feasibility Study considers continued Black Fox ore treatment via the Stock Mill, milling an average of 2,000 t/d currently owned by Brigus Gold. The refurbishment of the Stock Mill was completed in April of 2009.

The Stock Mill, includes the conventional unit processes of:

- primary crushing
- closed circuit, single staged fine crushing
- two staged grinding
- pre-leach thickener and carbon columns
- leach and Carbon-in-Pulp (CIP) circuits
- carbon stripping and electrowinning
- cyanide destruction.

The production strategy involves mining the open pit at its full capacity of approx.. 30,000 t/d, and processing the highest grade material through the mill in conjunction with underground ore at its nominal 1,000 t/d rate. Excess open pit material will be stockpiled in medium and low grade piles for recovery once the open pit is completed.

Table 19.1 indicates the combined total production from the mine reporting to either the mill (2,000 t/d) or to stockpile.

Table 19.1 Pit and Underground Production Summary (at Jan. 1, 2011)

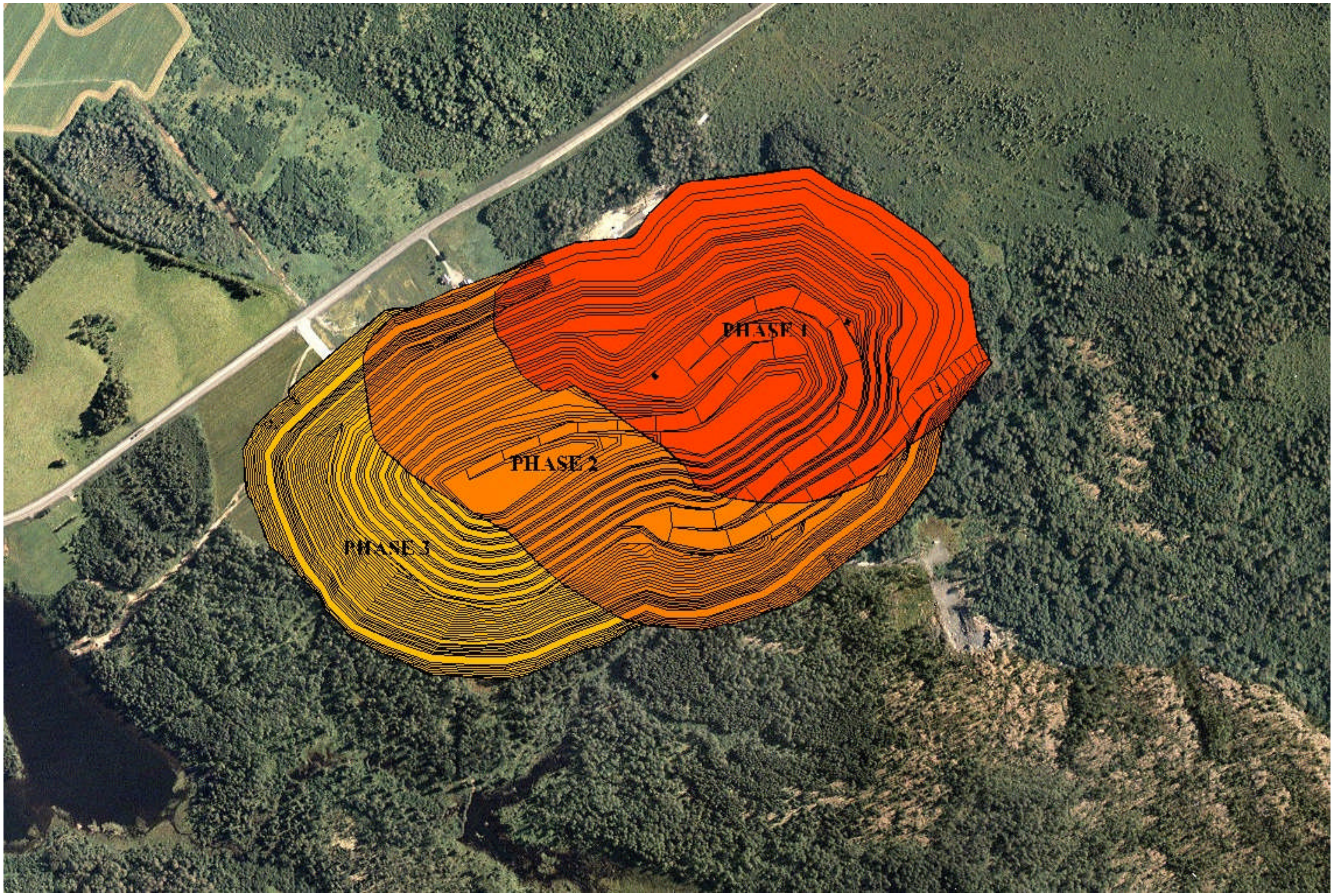
Year	2011	2012	2013	2014	2015	2016	2017	2018	2019	Total
Pit										
Ore Tonnes	432,998	994,889	413,608	973,443	286,577	0	0	0	0	3,101,515
Grade	3.107	3.161	2.649	3.597	3.061	0	0	0	0	3.213
Au (oz)	43,252	101,098	35,232	112,583	28,205	0	0	0	0	320,370
Underground										
Ore Tonnes	297,002	353,238	365,000	360,505	365,000	366,000	365,000	354,728	101,845	2,928,318
Grade	6.387	5.332	6.173	5.862	5.124	6.339	6.870	5.469	5.870	5.936
Au (oz)	60,987	60,553	72,439	67,942	60,129	74,590	80,618	62,371	19,220	558,849
Total										
Ore Tonnes	730,000	1,348,127	778,608	1,333,948	651,577	366,000	365,000	354,728	101,845	6,029,833
Grade	4.441	3.730	4.301	4.209	4.217	6.339	6.870	5.469	5.870	4.535
Au (oz)	104,239	161,651	107,670	180,525	88,334	74,590	80,618	62,371	19,220	879,219

Gold ounces reported in Table 19.1 (projected at January 1, 2011) differ from those reported in Tables 1.2 and 19.17 (actuasl at October 31, 2010) by 18,621 oz. This amount corresponds to the depletion of the reserves during the period of November and December 2010 based on an average combined ore and waste tonnage of 120,000 tonnes with an average grade of 4.81 g/t

19.1 Open Pit Mining

Mining operations have been separated into glacial overburden (Till) and bedrock mining. The mining of bedrock has been separated into three distinct areas for calculating the equipment and costs required for each area. Mining in the waste, ore and old workings areas has been calculated using distinct parameters to account for the various methods and time that will be required for each area. Figure 19.1 shows the pit design and aerial photograph with phase areas for the Black Fox pit design.

Figure 19.1 Phase Layout with Image



Mining commenced with Till removal in August 2008 with ore production beginning in March of 2009 at a rate of 2,000 t/d through the 3rd quarter of 2010. After this time, underground development has started for blending of higher-grade ores for the remainder of the 5-year open pit mine life. When full ore production is achieved, rock and overburden stripping will be maximized at 30,000 t/d in 2011.

Ore is delivered to a primary crusher located near the low-grade stockpile. Run of mine (ROM) ore stockpiles near the primary crusher are planned to provide continuity of ore delivery over short periods (with wheel loaders feeding the crusher). Ore is crushed at the mine site and trucked to Brigus Gold's Stock Mill, which is approximately 31 km from the mine site.

Technical mine planning involved the creation of a mining block model, pit optimization at US$975/oz gold (is elected as optimum pit shell), detailed pit and dump design, production scheduling, haul profile calculation and generation of reserves. Information produced was then used for mine fleet sizing, costing, fleet performance, of which, results were rolled up into the Brigus Gold economic model.

The required mine equipment was calculated based on the open pit production schedule. The open pit equipment requirements were calculated taking into account four distinct areas:

- glacial Till
- waste mining away from mineralization
- ore and waste mining near mineralization
- open pit mining in the old working areas.

19.1.1 *Mining Methods*

Glacial Till

The alluvial Till is being mined by a fleet of owner operated equipment, consisting of two CAT 320CL hydraulic backhoe excavators and twelve CAT D740 articulated dump trucks, Support equipment consists of two CAT D6 Low Ground Pressure (LGP) dozers. Till material is being dumped in separate overburden dumps located to the east of the offices. The overburden is dumped in 3 m lifts in cells created by forming up rock bunds from waste excavation. This method ensures that the free running nature of the overburden is contained. Rock haul roads are constructed to allow the safe passage of the trucks. The LGP dozers are used to form up the rock bunds and dress the overburden.

Open Pit Waste Mining Away from Mineralization

In areas known to contain only waste, the drillhole spacing and drillhole depth is different from areas containing mineralization. About 75% of the waste rock mined will be from areas containing only waste. Blasted waste rock is mined by a 12 m^3 Komatsu PC2000 hydraulic backhoe excavator and 91 tonne CAT 777F rigid dump trucks haul trucks. A 6.5 m^3 CAT 988H front-end loader is used as a backup for the excavator and for general duties.

OPEN PIT MINING IN THE MINERALIZED AREAS

The mineralized zones average 4.75 m wide, but can be as narrow as 1.0 m. The minimization of dilution of the ore is a critical element of the mining operation due to the characteristics of the orebody. For this reason, the ore is mined in 3 m lifts or benches, so that identification of ore blocks can be carried out with the most accuracy and the material mined with the minimum of dilution. The drill cuttings from all blastholes are sampled and assayed in order to provide the basis for ore grade control. The sampling requirements influence the spacing of the blastholes which in turn influences the blasthole diameter.

Mining in the mineralized area is accomplished using two, 4.0 m^3 CAT 385CL hydraulic backhoe excavators to minimize dilution. For this study, all of the ore was scheduled using the smaller excavator but in the larger ore zones the 12.0 m^3 excavator could be used to decrease the loading times. The 91 tonne rigid haul trucks are also used for hauling the ore, despite the fact that this will require 11 to 12 passes. The benefit of this is that the truck fleet is standardized. The loading of the ore zones is restricted to daytime shifts only to provide better control in mining the ore.

OPEN PIT MINING THROUGH EXISTING UNDERGROUND WORKING AREAS

Additional factors have been added to the equipment requirement calculations to allow for the extra time and costs that will be encountered during the mining through the existing underground workings.

Mining in the existing underground workings is scheduled using the smaller 4.0 m^3 excavator. The production rate for mining in these areas is calculated at 65% of the normal production rate to allow for the extra time that is expected in these areas. The 91 tonne haul trucks are used for hauling the ore. As this operation is necessarily slow in these areas the additional number of passes is offset by less trucks and standardization of the truck fleet. The loading of the old working areas is restricted to daytime shifts only for safety concerns.

To ensure that the old workings are safe to work it is necessary to identify the potential area of old workings from existing survey information. These areas are flagged to delineate a safe working zone. Holes are drilled, plugged and blasted to collapse the old workings. Further holes are then drilled to probe the collapsed old workings to ensure they are safe to work.

19.1.2 *PIT SLOPE STUDIES*

A geotechnical stability assessment was conducted on the pit slopes of the Brigus Gold Black Fox mine. In order to carry out the stability assessment, use was made of previous geotechnical investigations and a visit was conducted to the mine site to observe the condition of the existing pit walls.

The available geotechnical data included a geotechnical report prepared by AMEC Earth & Environmental (AEE), dated April 11, 2008 (AMEC 2008). The following is a summary of the findings and recommendations of AMEC's report.

The geotechnical investigation included drilling four boreholes to depths of 149 m to 175 m. All boreholes were logged by AMEC, with Apollo staff completing lithologic logging and point load testing. The boreholes were also surveyed by DGI Geosciences using Acoustic and Optical Televiewers. DGI interpreted the televiewer data in order to determine the orientation of the discontinuities encountered in the boreholes. Geotechnical mapping was also carried out by AMEC in access drives and drifts within the past underground workings. The work was completed in 2006 and included mapping of only primary structures (continuity greater than 1.5 m) as well as identifying the orientation of foliation associated with the various rock types encountered. The mapping data was entered electronically and DIPS Software used to create stereonets and identify trends in the data. AMEC report indicates that the analysis of the underground mapping and acoustic drillhole televiewer data confirmed structural trends of south dipping structure exists within a large brittle to ductile deformation zone called the DPFZ, which strikes generally east-west through the project area and dips between 40° to 50° to the south. Rock types expected to be encountered include ultramafic and mafic volcanics and sediments that have been altered and/or bleached. Faulting also typically follows the dip of the DPFZ.

The ultimate Black Fox Open Pit will be approximately 1 km long along its long axis, which is oriented approximately east-west, parallel to the strike of the ore body and over 500 m wide and 150 m deep. The proposed north wall of the open pit is expected to follow the footwall of the DPFZ, while the south wall will straddle across the hangingwall fault of the fault zone and into the hangingwall rocks.

The geotechnical analysis of the available data indicates the rock mass is primarily competent but will possess relatively weaker rock mass strength and zones associated with faults or chloritised and foliated lithologic groups. Kinematic stability analysis methods were used to determine pit slope bench face, inter-ramp and overall slope angles for the various wall orientations.

Analysis of both the underground mapping and the televiewer results were used to define the domains and primary structural sets for the pit wall slope analysis. Table 19.2 below shows the recommended geotechnical pit design parameters. Achievable designs are considered practical based on current industry experience, given current industry standards including wall control blasting. Optimistic designs are aggressive and will require more expensive mining practices, wall control blast techniques, and careful operational procedures in order for these walls to be successful.

Table 19.2 Geotechnical Information

Domain	Wall Dip Direction	Final Walls Achievable Configuration (18 m high benches)		Stage I Pit Slope Optimistic Configuration*	
		Bench Face Angle/ Berm Width	Inter Ramp Angle	Bench Face Angle/ Berm Width	Inter Ramp Angle
I	190º	55°/8 m	41°	60°/6-10m (alternating)	44°
II	220º	55°/8 m	41°	55°/8m	41°*
III	180º	60°/10 m	41°	60°/10m	41°*
IV	220º	55°/8 m	41°	55°/8m	41°*
V	000º	75°/8 m	54.5°	75°/8m	54.5°
VI	045^0	75°/8 m	54.5°	75°/8m	54.5°
VII	120º	65°/8 m	48º	70º/8m	51º

* Cannot be increased due to a possibility of day-lighting fault structures.

After reviewing the previous geotechnical reports, Wardrop believes that reliable geotechnical data has been utilized to evaluate rock mass characteristics and to develop recommendations for pit slope design. The pit slope design was based on the seven domains that were identified for each phase of the pit development. Design methods used to determine appropriate pit slope angles included a detailed kinematic stability assessment. The currently available data and the corresponding kinematic stability analyses confirm that the recommended pit slope design is reasonable and appropriate. However, it should be recognized that there are inherent risks in any mining development.

A visit to the Brigus Gold Black Fox mine was conducted on November 10, 2010. During this site visit, Wardrop evaluated the stability of the existing pit wall exposed during the development of Phase 1. At the time of the visit the bottom of the pit was at elevation 9,919 and the maximum depth of the pit was approximately 81 m. The pit slope has been constructed according to the recommendations presented by SRK (2008) report, which includes inter-ramp slope angles of 41° for the north (Domain III) and east (Domain IV) walls and 54.5° for the south wall (Domain V). The pit is being developed with 18 m high production benches and 8 to 10 m bench width. The east and south pit walls and the majority of the north wall of Phase1 look fairly stable. However, evidence of a previous localized shallow planar failure was observed in the middle of the north wall (Figure 19.2). The planar failure affected an area about 30 m wide by 16 m long (upper two benches). A planar failure is kinematically possible where a discontinuity plane is inclined less than the slope face (daylights) at an angle steeper than the friction angle. We believe the failure was caused by rock discontinuity daylight and surface water, which probably contributed to the reduction of the friction angle of the fill material within the discontinuity. Brigus Gold's staff took measures to stabilize the affected area and re-designed the pit slope in the area of the failure to an inter-ramp angle of 37°. The berm width at the 9,925 elevation was increased from 8 m to 15 m to achieve the flatter slope angle. The bottom of the pit is currently at elevation 9919 and once the elevation 9,901 is reached, the wall in this area will be buttressed by the ramp. Another action that Brigus Gold's staff are taking to mitigate any future planar failure is that they are drilling angled preshear (60°) in this area, in the past they have used vertical preshear or just a buffer row.

Figure 19.2 North Wall of Phase 1 Showing the Localized Shallow Planar Failure

The geotechnical pit design parameters presented in Table 19.2 are reasonable and appropriate to continue developing the open pit shells and then the slopes optimized to take into consideration the lithologic sequence as well as underground workings that occur behind the pit walls.

It is important to note that a number of large open pit operations have routinely encountered slope stability problems in some areas of the mines. The experiences at most of the large open pits suggest that there is a significant possibility that some area of the pit slope will require flattening during operations in response to slope movement. Therefore, the mine plans should remain flexible so that an extra stepout/buttress can be maintained in critical areas of the pit until the end of the mine life when lower factors of safety can be tolerated. The size and therefore impact of any planar type failures will depend largely on the continuity of the joints forming the planar sliding surfaces. At present, there is no data on the continuity of the joints at the Black Fox mine. Therefore, geotechnical mapping of the exposed bench faces will be required to make an assessment of the likelihood of encountering joints which would be continuous enough to affect the overall wall stability. If continuous joints are noted in the initial pit excavation, then the new data should be used to optimize the current pit slope design.

There will still be a risk of smaller scale wedge and planar failures at a bench scale, however, this is the case for most open pit mines, where there are inclined joints present. These bench scale instabilities would need to be addressed through careful inspections and scaling after blasting. Stability of the final bench faces can be improved through the use of good quality controlled blasting methods and through scaling of loose rock from the bench faces after blasting.

A diversion ditch along the low point of the north pit crest of Phase 1 is recommended to divert surface runoff and snowmelt away from the pit during operations. Shotcrete lining of a low permeability is often recommended for diversion ditches in order to minimize seepage losses and groundwater recharge to underlying pit slopes.

19.1.3 PIT OPTIMIZATION

Pit optimization was carried out using Whittle 4.3™ optimization software using input parameters defined by Brigus Gold and Wardrop and feasibility contributors in January 2008.

Results were evaluated to determine the largest economic or ultimate pit size, optimum pit size and optimum pit size above a mine elevation of RL 9,820.

There were several restrictions placed on optimization runs:

- Waste rock was to be minimized due to possible arsenic leaching from waste dumps and planned storage capacity.
- The definition of open pit and underground mining required space for a crown pillar.
- The requirement that one small pit be permitted and mined before mining to the optimization limits.
- Consideration of underground workings and Till removal costs.
- Pit area restrictions from highway and planned infrastructure.

19.1.4 WHITTLE™ MODEL PARAMETERS

Table 19.3 illustrates the block dimensions and geotechnical zones of the exported Gemcom Surpac™ mine block model into Whittle™. The block model was created using the Gemcom GEMS software. Reblocking converted the geological block model size from 3 m x 3 m x 3 m to 6 m x 6 m x 6 m.

Table 19.3 Whittle Parameters

Whittle Parameter	Type	Value
Base Units		
	Au	g
	Mass	t
Block Model Dimensions		
	Origin (mine grid)	
	x	9700
	y	9500
	z	9400
	Geological	
	X	3
	Y	3
	Z	3
	No. X	317
	No. Y	284
	No. Z	209
	Reblocked	
	X	6
	Y	6
	Z	6
	No. X	159
	No. Y	142
	No. Z	105
Slope		
	Profile 1 (Zone 1)	41°
	Profile 2 (Zone 2)	41°
	Profile 3 (Zone 3)	41°
	Profile 4 (Zone 4)	41°
	Profile 5 (Zone 5)	54.5°
	Profile 6 (Zone 6)	54.5°
	Profile 7 (Zone 7)	48°
	Profile 8 (Zone 10)	20°
	Profile 9 (Zone 45)	45°

19.1.5 Whittle™ Economic Parameters and Cost Adjustment Factors

With the complexity of mining in and around underground workings, a mining cost that reflects this operation has been used.

For optimization purposes, Till was included as an owner operator cost at $1.81/tonne without the need for drilling and blasting. Each economic rock type was used in a mineral processing path with a gold recovery of 94%. Till and Air did not contribute to optimization cash flow results.

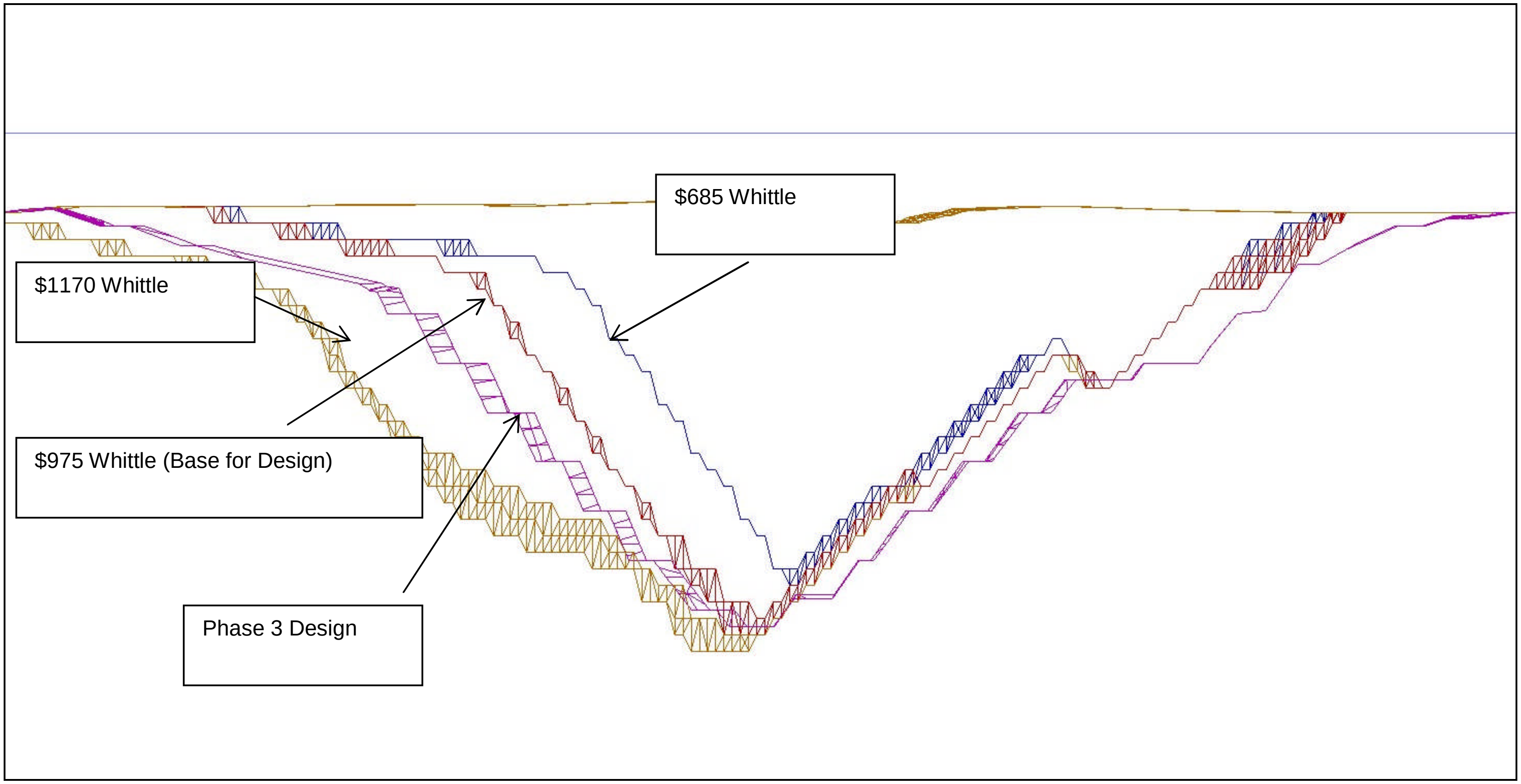
Revenue factors used the default 0.02 step of gold price away from the base US$975/oz gold price. This represented an analysis of pits from US$292/oz to US$1,950/oz for gold.

19.1.6 OPTIMAL PIT SHELL

Figure 19.3 shows the pit shells produced through optimization runs considering all blocks in the block model and those above RL 9,820. The pit shell selected as the base for pit design (Red triangulation in Figure 19.3) achieved the following results:

- Minimal risk to gold price fluctuations given conservative position on open pit value curve.
- Mining cost/ore-t is relatively low.
- Provision for crown pillar does not adversely effect optimization given ability to mine underground.
- Waste rock is kept to a minimum.
- No infringement on highway or lease boundaries.
- Minimal reliance on high-grade pods driving increased stripping.
- No infringement on available dump space or infrastructure.

Figure 19.3 Whittle Pit Shell Results



19.1.7 Pit Design

The Black Fox pit design utilized Gemcom GEMS ™ software for crest, toe and ramp layout with dynamic application of berm and batter angles from block model variables. Using the Whittle™ optimal pit shell, the final pit design aimed to minimize stripping based on application of haul roads and infrastructure limits, and maintaining complex geotechnical constraints.

19.1.8 Pit Design Criteria

During the process of pit construction, several iterations were conducted and compared with pit designs achieved in the Black Fox Pre-feasibility Study. The culmination of this produced a final pit design that would:

- Keep the pit as tight as possible to the pit optimization shell on the (northwest) NW wall.
- Provide at least 46 m clearance from the highway without using retaining walls.
- Minimize ramps in the hanging wall and NW footwall.
- Place ramps on the (northeast) NE footwall where the slope is relatively shallow, hence minimizing stripping.
- Place a 25 m wide ramp to elevation RL 9,877 and then convert to a 15m wide one-way traffic ramp to the pit bottom.
- Incorporate pit wall berms and batter angles based on Table 19.2 and applied from block model zones
- Facilitate an enlarged pit bottom on west side to extract additional waste and provide mining room for crown pillar construction.
- Prevent “Doughnut” mining between mineralized areas.

19.1.9 Bench Configuration

Geotechnical benches were based on an 18 m inter-berm change in elevation. The bench widths are 8 m. Each 6 m lift within the 18 m geotechnical bench will be excavated using two, 3 m “Flitches”.

19.1.10 Haul Road Parameters

Haul roads are 23 m wide for two way traffic and 14 m for one-way traffic. A 25 m wide ramp provides a truck width (6.39 m) to running surface width ratio of about 3.5, which is considered safe. At elevation 9,859, it was necessary to reduce road width to single lane traffic to minimize excessive waste stripping or loss of recoverable ore. Roads have a maximum gradiant of 12% and are designed at 10%, down to the 9,859 Level.

19.1.11 Phase Design

It was necessary to break the final pit design into 3 phases. Phase 1 was defined by a polygon area (In which the phase 1 pit was to stay within) for permitting reasons but would still merge with common walls of the final pit. The Phase 2 pit provided a bridge for scheduling purposes between Phase 1 and the final pit with phase intersections dictated by final pit design. Figure 19.1 shows a plan view with aerial photo of where each phase intersection on topography resides.

Phase 1 pit bottom terminated at elevation RL 9,901 and Phase 2 terminated at elevation RL 9,871 while phase 3 terminated at the final pit floor of 9835. Table 19.4 details the ore and waste tonnages defined by the different phases.

Table 19.4 Phase Tonnage (at Oct 31, 2010)

Phase	Au (g/t)	Ore Mass (t)	Waste Mass (t)	Strip Ratio
Phase 1	4.06	60,319	492,489	8.2:1
Phase 2	3.09	1,449,395	18,252,964	12.6:1
Phase 3	3.32	1,650,120	14,037,690	8.5:1
Total		3,159,834	32,783,143	10.4:1

Till material is mined by owner operated equipment at a rate 16,000 t/d for Phase 2 and 3, the till was mined by a contractor for the Phase 1 pit.

19.1.12 Bedrock Dump Design

The Bedrock dump design target volume was limited to Bedrock material produced in the production schedule that would be mined by Brigus. Till material which is also mined by Brigus was not included in “Bedrock” dump design (separate storage area).

The dump design for Black Fox was broken into two parts. Waste from the first two years of mining was required to stay within a specified polygon used for permitting. Access to the tip head for this dump was via the crusher and extended horizontally from the current outcrop elevation RL 10,028. The initial dump design ultimately lies within the total dump footprint so no benches were included in the design. Table 19.5 illustrates the parameters assigned to the final dump design configuration.

Table 19.5 Dump Design Parameters

Parameter	Units	Value
Berm Width	m	7
Lift Height	m	10
Batter Angle	º	37
Overall Slope	Ratio	1:2
Ramp Width	m	25
Ramp Grade	%	10

The final dump design used the full area set aside for waste dump disposal in the Pre-feasibility Study and was modified to include dual access to the tip head locations. Access to the tip head via the crusher established during Phase 1 mining would necessitate a long haul, so the final dump design had the tip head access as close as possible to the termination point of the ramps for Phase 2 and 3. This allowed two entry points to the dump throughout the mine life and minimized haul distances. The final dump design reached elevation RL 10,078 and contained room for 18.0 million m^3 of fill.

The final dump volume is more than adequate room to accept all pit design waste and has plenty of room for vertical expansion.

The final dump toe generally follows an outcrop boundary between hard rock and loose Till material. Since geotechnical holes are sparse on the SW dump face, careful monitoring is needed if the material dumped on this Till material leads to geotechnical instability.

19.1.13 OPEN PIT MINEABLE RESOURCES

CUT-OFF GRADE CALCULATION

A cut-off grade of 0.88 g/t for the open pit was calculated based on parameters shown in Table 19.14.

19.1.14 PIT PRODUCTION SCHEDULE

Production scheduling was carried out using Gemcom GEMS and an Excel spreadsheet. The schedule was constructed around a 2,000 t/d mill feed from open pit and underground with the open pit contribution ranging from 1000 t/d to 2,000 t/d. The amount of waste stripping was maximized at 30,000 t/d. Ore was defined using a 0.88 g/t cut-off as indicated from pit optimization work and scheduling units were broken into Till and Bedrock blocks within each phase. The production schedule was used to estimate the quantities of waste material produced each year for dump design and as an estimation for annual haul cycle times and distances.

The production schedule was targeted quarterly. Open pit mining terminates in 2015 giving a remaining mine life of five years.

Table 19.6 shows the production schedule results with grade, ore and waste tonnes per day.

Table 19.6 Pit Production Schedule

Year	2011	2012	2013	2014	2015
Tonnes ore	432,998	994,889	413,608	973,443	286,577
Tonnes waste	10,107,224	7,449,458	7,248,201	5,150,170	1,787,262
Au (oz)	43,252	101,098	35,232	112,583	28,205

Small variations in tonnage and strip ratio between Table 19.6 Pit Production Schedule and Table 19.4 Phase Tonnage are attributable to a small difference in pit stripping strategy between October 31, 2010 and January 1, 2011.

19.1.15 *Waste Schedule*

To calculate the required annual dump volume, a loose density of 2.0 tonnes per cubic metre (t/m^3) was applied to the generated waste tonnage produced. For each production year, an incremental solid triangulation was created within the total waste dump design. The centroid of this dump shape was used as a terminating point for the digitized waste haulage profile.

19.1.16 *Haulage Schedule*

Haulage calculations based on the production schedule were estimated using Vulcan™ (v7.5) haul profile software and were used to calculate annual distance and cycle times for ore and waste. Haul profiles were calculated by digitizing ore and waste profiles from the pit haul road to either a dump centroid or crusher location. The schedule was then calculated based on the time intervals for ore or waste from a given bench elevation.

Drilling & Blasting

The design assumptions used to determine drilling and blasting productivity are shown in Table 19.7.

Table 19.7 Drilling and Blasting Assumptions

Production & Wall Control Blast Pattern Data		Production Pattern						
DRILLING & BLASTING PARAMETERS		Void	Ore	Waste	Waste	Wall Control Pattern		
	Units	Areas	Bedrock	Bedrock	Bedrock	Buffer	Buffer	Preshear
Tonnage Factor	dmt/cubic metre	2.840	2.840	2.840	2.840	2.840	2.840	2.840
Blast Pattern Details								
Bench Height	metres	9.00	6.00	6.00	6.00	6.00	6.00	9.00
Sub Drill	metres	1.25	1.00	1.00	1.00	1.00	1.00	0.00
Diameter of Hole	mm	114.00	114.00	114.00	165.00	139.50	139.50	114.00
Staggered Pattern Spacing	metres	3.50	3.00	3.00	4.30	4.00	4.00	1.40
Staggered Pattern Burden	metres	4.00	3.50	3.50	4.60	4.00	4.00	1.40
Drill Equivalent Square Pattern	metres	3.75	3.25	3.25	4.45	4.00	4.00	1.40
Hole Depth	metres	10.25	7.00	7.00	7.00	7.00	7.00	9.00
Height of Stemming or Unloaded Length	metres	4.00	3.40	3.40	3.40	4.00	4.00	
Material Quantity								
Volume Blasted/Hole	cubic metres	127	63	63	119	96	96	18
Tonnes Blasted/Hole	tonnes	359	180	180	337	273	273	50
Powder Factor								
Percent Emulsion		100%	100%	100%	100%	100%	100%	100%
Percent Anfro		0%	0%	0%	0%	0%	0%	0%
Density of Powder	g/cc	1.26	1.26	1.26	1.26	1.26	1.26	1.26
Loading Density	kg/m	12.86	12.86	12.86	26.94	19.26	19.26	12.86
Powder/hole	kg	80.38	46.30	46.30	96.99	57.77	57.77	4.51
Powder Factor	kg/t	0.224	0.257	0.257	0.287	0.212	0.212	0.090
Powder Factor	kg/bcm	**0.635**	**0.731**	**0.731**	**0.816**	**0.602**	**0.602**	**0.256**
Drill Productivities								
Penetration Rate								
	M/hr	25.00	40.00	40.00	50.00	40.00	40.00	25.00
Penetration Rate	M/min	**0.42**	**0.67**	**0.67**	**0.83**	**0.67**	**0.67**	**0.42**
Cycle Time Estimate								
Drilling Time	minutes	24.60	10.50	10.50	8.40	10.50	10.50	21.60
Steel Handling Time	minutes	0.50	0.50	0.50		0.50	0.50	0.50
Set up Time	minutes	2.00	2.00	2.00	2.00	2.00	2.00	2.00
Add Steel	minutes	2.00	1.00	1.00		0.50	0.50	2.00
Pull Rods	minutes	2.00	1.00	1.00		1.00	1.00	2.00
Total	**minutes**	**31.10**	**15.00**	**15.00**	**10.40**	**14.50**	**14.50**	**28.10**
Drilling Factors for Wall Control								
Buffer Holes - 2 Rows								
Wall Control Drill Holes Required	***Perimeter Blast***	Ore	Ore	Bedrock	Bed Waste			
Pre-Shear Holes	holes/metre	**0.00**	**0.00**	**0.71**	**0.71**			
Buffer Holes - 2 Rows	holes/metre	**0.00**	**0.00**	**0.25**	**0.25**			
Material to Remove from Production Blast	tonnes/metre	**0.00**	**0.00**	**103.94**	**12.52**			

OPEN PIT LOADING PRODUCTIVITY

The design assumptions used to determine loading and hauling productivity, including the underground backfill loading and feeding the crusher, are shown in Table 19.8.

Table 19.8 Open Pit Loading Productivity Assumptions

Loading Parameters & Truck Match Calculation			5.0 CM excavator		12.0 cm excavator		6.5 cm Loader	
			Day	Day and Night	Overburden	Rock	Waste	Ore
Schedule Data			90 tonne	90 tonne	90 tonne	90 tonne	90 tonne	90 tonne
(BC)	Bucket Capacity (heaped)	cm	**5.00**	**5.00**	**12.00**	**12.00**	**6.50**	**6.50**
(MW)	Material Weight	kg/bcm dry	**2840**	**2840**	**2000**	**2840**	**2840**	**2840**
MWW	Material Weight Wet	kg/bcm wet	**2850**	**2850**	**2300**	**2850**	**2850**	**2850**
(BF)	Bulk Factor (Swell Factor)		**1.35**	**1.35**	**1.10**	**1.35**	**1.35**	**1.35**
(MW1)	Material Weight = MW / BF	kg/lcm dry	2,103.7	2,103.7	1,818.2	2,103.7	2,103.7	2,103.7
(M)	Moisture		**0.4%**	**0.4%**	**13.0%**	**0.4%**	**0.4%**	**0.4%**
(FF)	Fill Factor		**0.85**	**0.85**	**0.90**	**0.87**	**0.85**	**0.85**
(EBC)	Effective Bucket Capacity = FF x BC	cm	4.25	4.25	10.80	10.44	5.53	5.53
(MW2)	Material Weight = MW1 / (1-M)	wmt/lcm	2.11	2.11	2.09	2.11	2.11	2.11
	Material Weight = MW2 x (1-M)	dmt/lcm	2.10	2.10	1.82	2.10	2.10	2.10
(TP)	Tonnes/Pass	wmt	8.97	8.97	22.58	22.04	11.66	11.66
(TC1)	Truck Size Capacity	cubic m heape	60.1	60.1	60.1	60.1	60.1	60.1
(TC2)	Truck Size Capacity	wmt	90.0	90.0	90.0	90.0	90.0	90.0
(TPV)	Theoretical Passes = TC1/ EBC	passes	14.14	14.14	5.56	5.76	10.88	10.88
(TPT)	Theoretical Passes = TC2 / TP	passes	10.03	10.03	3.99	4.08	7.72	7.72
(AP)	Actual Passes = ROUND TPT	passes	**10.0**	**10.0**	**4.0**	**4.0**	**7.0**	**7.0**
(TL)	Truck Load - Volume = AP x EBC	cm	42.5	42.5	43.2	41.8	38.7	38.7
(TLS)	Truck Load for Simulation = AP x TP	wmt	89.7	89.7	90.3	88.2	81.6	81.6
(TLP)	Truck Load for Productivity	dmt	89.4	89.4	78.5	87.9	81.4	81.4
(TCU)	Truck Capacity Utilized = TLS / TC2	by weight	**99.7%**	**99.7%**	**100.4%**	**98.0%**	**90.7%**	**90.7%**
	Truck Capacity Utilized = TL / TC1	by volume	70.7%	70.7%	71.9%	69.5%	64.4%	64.4%
(AC)	Average Cycle Time	sec	**35**	**35**	**35**	**35**	**39**	**39**
(ST)	Truck Spot Time	sec	**45**	**45**	**45**	**45**	**45**	**45**
(LT)	Load Time per Truck = AP x AC + ST	sec	395	395	185	185	318	318
(LT)	Load Time per Truck = AP x AC + ST	minutes	6.58	6.58	3.08	3.08	5.30	5.30
(MP)	Maximum Productivity = 60 / LT	trucks/hr	9.1	9.1	19.5	19.5	11.3	11.3
	Conversion = MP x TLP/ MW	bcm/hr	286.9	286.9	764.2	601.9	324.3	324.3
	Tonnes/Hour	dmt/hr	814.9	814.9	1,528.5	1,709.5	921.1	921.1

19.1.17 *OPEN PIT MINE EQUIPMENT*

The open pit is being mined using a new fleet of equipment. The open pit equipment required is shown in Table 19.9.

Table 19.9 Mine Equipment

Equipment Type	Description	Units
CAT - 777F Truck	100 tonne truck	5
PC2000 Hydr Excavator	27 tonne bucket	1
CAT - 385CL Excavator	11 tonne bucket	2
CAT - 988H Loader	12 tonne bucket	1
CAT - 980H Loader	10 tonne bucket	2
Komatsu PC300 Excavator	3.2 tonne bucket	2
Komatsu PC270 Excavator	3.0 tonne bucket	1
CAT - 320CL Excavator	3.5 tonne bucket	1
Doosan DX340 - Excavator	4 tonne bucket	1
Tramac V2500	Rock Breaker	1
CAT - D4G Dozer	Utility Dozer	1
CAT - D8T Dozer	Production Dozer	2
CAT – D6TLP Dozer	Production Dozer	3
CAT - D9T Dozer	Production Dozer	1
16H Motor Grader	Production Grader	1
DM 45 Blast Hole Drilll	5 1/4 to 6 1/2" Holes	1
A/COPCO CM785 Drill	4 1/2 to 6 1/2" Holes	2
CAT - IT28G	Integrated Tool Carrier	1
JD 320 Skid Steer		1
CAT 930G Tool Handler	Utility Loader	1
48 x 32 Low Profile Eagle		1

OPEN PIT WATER MANAGEMENT

Based on the 1:25 year storm event modeling, the ultimate open pit area could expect to see approximately 33,000 m^3 of water in a 24hr period. Due to the numerous old workings in the pit floor the water from the open pit drains into the underground operations. Because of this, minimal pumping is required in the open pit. Details on the underground pumping system can be found in the underground section of this report.

19.1.18 OVERBURDEN SLOPES, WASTE AND OVERBURDEN STOCKPILES

AMEC (2008b) carried out a Feasibility Study, which included the stability assessment of the following components: overburden slopes around the open pit, clean waste rock stockpile, dirty waste rock stockpile and the overburden stockpile.

OVERBURDEN SLOPES OF OPEN PIT

In consideration of the nature of the overburden materials and the consequences in the event of a potential large-scale failure (specifically potential for loss of life and mine equipment), the overburden slopes of the open pit are designed with respect to long-term stability (Factor of Safety 1.5) and seismic loading conditions. The following loading cases have been analyzed:

- Steady State Static Loading: Long-term static loading condition; using drained shear strength parameters for granular soil deposits and peak undrained shear strength for the cohesive deposits.
- Post Earthquake Conditions: Temporary weakening of susceptible silt layer due to partial liquefaction or excess pore pressure generation (use of residual strength parameters, where applicable).
- Based on the stability assessments, a slope inclination of 3H:1V is considered adequate for the individual slope benches, which should not exceed 8 m in height. The recommended overall inclination for the pit perimeter slope is 5H:1V. Also note the following conditions that should be respected to maintain safe slopes:
    - No water ponding is to be permitted at the slope crest surfaces (through satisfactory drainage control).
    - The pit overburden slope should be provided with erosion protection as the slope safety could be compromised by erosion, which could be extensive in the silty and fine-grained sand materials.

Based upon site observation conducted on October 28, 2010 the pit overburden excavation was conducted in substantial conformance with design parameters described above. The following observations were noted during the site visit.

- Overburden slope to the rock interface appears to be stable without evidence of slope displacement, cracking or bulging at the face.
- Surface water best management procedures appears to be satisfactorily implemented, controlling surface water run-off on the overburden cut slope.
- Ponding of water was not observed at the overburden toe of slope.
- Erosion control best management practices appeared to have been adequate installed reducing soil erosion from the slope face.
- Site personnel noted no operational concerns regarding the overburden slope stability or surface water management and soil erosion.

Based upon site observations and discussion with site personnel, the overburden slope appears presently to be stable. Noted operational surface water and erosion controls best management practices have been implemented proving reasonable surface water and erosion control.

DIRTY WASTE ROCK STOCKPILE

Geochemical investigations of waste rock were carried out by AMEC in conjunction with the geotechnical investigations and design studies. Tasks completed included geochemical characterization of 26 specimens of existing waste rock collected from the surface, 132 core samples from the ore zone, and 80 core samples of waste rock selected from the open pit drilling program. Static testing for selected samples from the various sources was conducted including analyses for: ABA (acid base accounting), concentration of total metals, and British Columbia Ministry of Energy and Mines (BC MEM) leachate extraction. Semi-quantitative mineralogical analyses of some samples were also conducted by Rietveld X ray diffraction (XRD):

- Most waste rock at the Black Fox property will be net non-acid generating with a safety factor much greater than the conservative screening criterion of 4:1 for the neutralization potential (NP) to acid generation potential (AP) ratio (NP/AP or NPR) for waste rock (Price, 1997). Values for neutralizing potential (determined by analytical titration) are in excess of 100 kg aCO_3/t; and
- All waste rock types identified on the Black Fox Property contain elevated concentrations of total arsenic (As), nickel (Ni) and chromium (Cr) relative to average crustal abundances, however, the leachability of these elements is variable and dependent on rock type.

Preliminary analyses conducted by AMEC suggested that the potential exists for leaching of both arsenic and nickel from waste rock produced at the Black Fox Mine. The rock types with such potential will be stored in the Dirty Waste Rock Stockpile. Some temporary storage may be required to account for scheduling lag between waste rock production and use for construction.

The proposed dirty waste rock stockpile will be located south of the open pit, with a rock outcrop hill to the east and a beaver pond to the west. The ground surface on the west side continues to slope down to Lawler Lake. The ground surface elevation within the proposed footprint area of the dirty waste rock stockpile is generally sloping towards west from about elevation 298 to about elevation 292.

The total quantity of dirty waste rock that will be excavated from the open pit development is estimated to be about 43.1 Mt. The footprint area of the stockpile is roughly 55 ha. The stockpile top is at elevation 358. The overall exterior slope of the stockpile, which is based on slope stability analysis results, is about 2H:1V (comprising 10 to 12 m high benches with minimum 10 m bench width).

Based upon site observation conducted on October 28, 2010 the dirty rock waste stockpile appears to have been constructed in substantial conformance with design parameters described above. The following observations were noted during the site visit.

- *The rock slope appeared stable, no presence of slope displacement, was observed.*
- *Ponding of water was not observed on rock overburden stockpile.*
- *Surface water best management practices appear to have satisfactorily implemented surrounding the rock stockpile area.*

- *Contact water from the dirty rock stockpiles appears to be satisfactorily managed via adjacent holding pond.*
- *Site personnel noted no operational concerns regarding the rock slope stability or contact water collection.*

Based upon site observations and discussion with site personnel, the dirty rock stockpile appears presently to be stable and operated in accordance with best management practices.

CLEAN WASTE ROCK STOCKPILE

The total quantity of the clean waste rock that will be produced is estimated to be about 2.92 Mt, or 1.54 million m^3 (based on 1.90 t/m^3 dry density for the dumped material). The site designated for the temporary clean waste rock stockpile is located east of the open pit. The existing ground surface of the generally flat lying area is at about elevation 293.

Based upon site observation conducted on October 28, 2010 the clean waste rock stockpile was constructed in substantial conformance with design parameters described above. The following observations were noted during the site visit.

- *The clean rock slope appears presently stable, no presence of slope displacement was observed.*
- *Ponding of water was not observed at the rock overburden pile toe of slope.*
- *Surface water and erosion control best management practise appear to have satisfactorily implemented surrounding the rock stockpile area.*
- *Site personnel noted no operational concerns regarding the rock slope stability.*

Based upon site observations and discussion with site personnel, the clean rock stockpile slope appears to be presently stable and operated in accordance with best management practices.

OVERBURDEN STOCKPILE

The total quantity of overburden to be excavated from the open pit area is estimated to be about 31.7 Mt. The overburden material at the open pit area is variable. The most predominant deposits at the open pit site are the silty sand to sandy silt, silt and till deposits, with some silty clay layers on the east side. The upper part of the silty sand to sandy silt stratum is mostly drained, although the level is relatively high on the east side within the silty clay deposit. This suggests that except for the saturated silty clay deposit, the other deposits will be suitable for excavating, handling and stockpiling.

The overburden material will be stocked in three areas referred to as the East Overburden Stockpile, the North Overburden Stockpile (located north of Highway

101), and the West Overburden Stockpile (located west of Froome Lake and south of Highway 101).

Table 19.10 presents estimated stockpile capacities and the design features of the three stockpiles. The stockpile capacities take into consideration the volume that will have to be stocked prior to the dam construction and the volume that will be used for dam construction.

Table 19.10 Principal Features of Overburden Stockpiles

Design Item	East Overburden Stockpile	North Overburden Stockpile	West Overburden Stockpile
Design storage capacity of the stockpile	0.69 x 106 m^3	0.98 x 106 m^3	4.96 x 106 m^3
Estimated final (long-term) storage volume	0.23 x 106 m^3	0.27 x 106 m^3	4.96 x 106 m^3
Footprint area of stockpile	10 ha	21 ha	42 ha
Existing ground surface elevation	299 to 300	290 to 297	289 to 294
Final elevation of the top of stockpile	302	294	310
Maximum height (initial or final stage)	14 to 16 m	2 m (northeast) to 9 m (northwest)	20 m
Perimeter slope (H:V) based on slope stability analyses	6H:1V	9H:1V (north, west, and south) to 4:1 (east)	6H:1V (north, west, and south) to 9:1 (east)

Based upon site observation conducted on October 28, 2010 the overburden material was placed in substantial conformance with design parameters described above. The following observations were noted during the site visit.

- *The overburden stockpile appeared stable; no presence of slope displacement, budging was observed.*
- *Ponding of water was not observed at the overburden stockpile toe of slope.*
- *Surface water nest management practise appear to have satisfactorily implement.*
- *Erosion on long slopes can be a problem. Present best management practices appeared adequate to reduce slope soil erosion.*
- *Site personnel noted no operational concerns regarding the overburden stockpile.*

Based upon site observations and discussion with site personnel, the overburden slope appears to be presently stable and operated in accordance with surface water erosion best management practices.

19.2 HYDROGEOLOGY

A summary of the hydrogeology of the Black Fox Mine is provided based on information gathered during a site visit on November 10, 2010, monitoring data provided by Brigus Gold and the numerous hydrogeologic studies that have improved the understanding of the site hydrogeology and groundwater flow systems. These studies include:

- Hydrogeological Report Apollo Gold Black Fox Open Pit Project (AMEC 2005a).
- Technical Memorandum titled Black Fox Project – Groundwater Flow Model for a Small Open Pit (AMEC 2005b).
- Technical Memorandum titled Modified Black Fox Groundwater Flow Model - Phase 2 Open Pit and Underground Workings Seepage (AMEC 2009a).
- Technical Memorandum titled Apollo Gold Corporation – Black Fox Mine 2009 Lakebed Sediment Sampling at Froome Lake (AMEC 2009b).
- Permit to Take Water Amendment Application – Pit and Underground Workings Dewatering (Phase 2 Pit Expansion) (AMEC 2010).
- Surface Water Monitoring Report for 2009, as per Conditions 4.1 to 4.3 and 4.5.3 of Permit to Take Water #7460-7K8RAB (AMEC 2010).

The Black Fox Open Pit (ultimate) is anticipated to be approximately 1 km long along its long axis which is oriented approximately east-west, parallel to the strike of the ore body and over 500 m wide and 150 m deep. The proposed north wall of the Open Pit is expected to follow the footwall of the DPFZ, while the south wall will straddle across the hangingwall of the fault zone and into the hangingwall rocks. The intent of the hydrogeologic summary is to provide an overview of the major findings from these hydrogeologic studies as they relate to the key issues of open pit expansion / operations. The key hydrogeologic concerns are:

- Groundwater seepage rates into the open pit and underground workings.
- The potential for lowering lake water levels in Froome Lake.
- The effect of the larger thickness of overburden west and south of the open pit during pit expansion.
- Management of groundwater and surface waters in the pit.

19.2.1 HYDROGEOLOGIC CONDITIONS

HYDROSTRATIGRAPHY

The geology of the site has been described extensively in section Section 7– Geological Setting. The overburden at the site is characterized by Pleistocene glaciofluvial and glaciolacustrine deposits of varying thickness (approximately 6 to 30 m). In the vicinity of the proposed open pit, these sediments typically include silty clay/clayey silt ranging from 6 m thick near the western margin of the proposed pit to 15 m thick at the eastern margin of the proposed pit and, underlying the silty clay, a sand unit containing varying amounts of gravel and silt and ranging in thickness from 0.3 to 15 m (AMEC 2005a). West of the proposed pit there exists a glaciofluvial deposit, interpreted to be an esker, which trends north/south for approximately 6 km. The existing geologic information (AMEC 2005a) indicates that the esker is primarily sand and silt and is underlain by coarse-grained sand and gravel interpreted to be a basal till. The esker is overlain by a clay layer ranging in thickness from 0 to 12 m (AMEC 2008). Where present, this clay layer limits recharge into the esker. The coarser sand of the esker is isolated from the proposed pit wall by a silty sand unit (AMEC 2005a), which is expected to limit groundwater seepage into the pit.

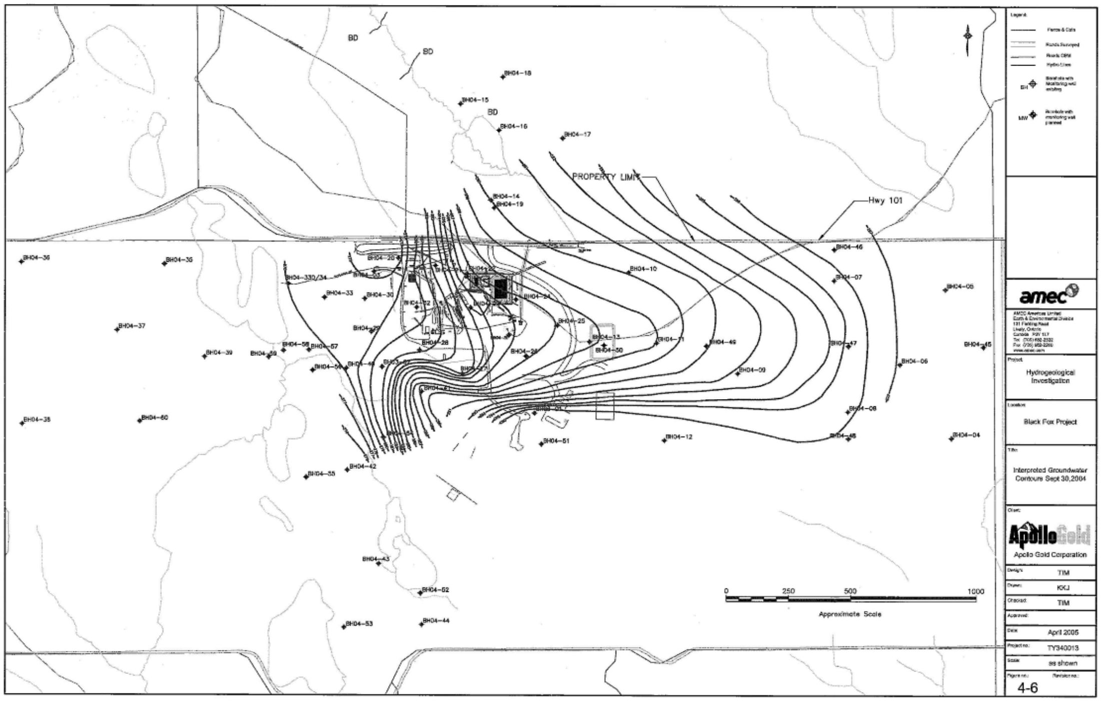
The underlying bedrock is comprised of a variably sheared, faulted, carbonatized and mineralized sequence of komatitic ultramafic volcanics belonging to the Stoughton-Roquemaure Group (SRK 2008). The DPFZ is a major geologic structure that strikes generally east-west through the project area and dips between 40º to 50º to the south. The upper 5 to 15 m of the bedrock is described as a shallow fractured bedrock aquifer, beneath which bedrock is characterized as unweathered, with low fracture densities and consisting of conglomerates, quartzite, ultramafics and volcanics. The depth to bedrock increases with distance from the proposed Open Pit area towards Froome Lake.

Where present, the basal till and upper weathered bedrock would be expected to act as a contact zone aquifer which may yield higher seepage rates to the pit and dominate shallow drawdown propagation as a result of the higher hydraulic conductivity of this combined unit, relative to surrounding geologic units.

GROUNDWATER FLOW

The depth to groundwater in the overburden is highly variable across the site. Typically, groundwater is within a few meters of the ground surface north and west of the site and is deeper (approximately 10 to 15 m below ground surface (bgs)) south and west of the Project Site where the overburden is thicker. The groundwater gradients around the Project Site have been affected by historical dewatering of the underground mine from 1997 to 2001 and by the current dewatering operations, which restarted in January 2004. The horizontal hydraulic gradients reported by AMEC (2005a) for September 2004 (Figure 19.4) and February 2005 at the site indicates that groundwater flow is primarily towards the underground mine workings. AMEC (2005a) suggests that the broader hydraulic gradients are likely controlled by two small tributaries to Salve Creek, resulting in groundwater flowing to the north.

Figure 19.4 Interpreted Overburden Groundwater Elevation Contours



The overburden is underlain by approximately a 5 m to 15 m thick zone of weathered bedrock. The weathered bedrock has a higher fracture density and is more permeable than the deeper, more competent rocks below. Permeability in the bedrock is primarily due to secondary fractures since the bulk rock is typically metamorphosed or highly consolidated and has minimal porosity. Limited potentiometric data are available for groundwater within the deeper bedrock. This is in part because the underground workings are dewatered to approximately 235 m bgs, and thus the overlying bedrock is unsaturated. Groundwater inflow into the existing workings has not been reported to be highly controlled by large-scale geologic features (e.g., the DPFZ) (R.Loughead, Personal Communication). As a result, the groundwater fluxes associated with this zone are likely small and may potentially decrease with time depending on the connectivity of the fracture networks on a sub-regional scale.

SURFACE WATER – GROUNDWATER INTERACTIONS

The hydraulic connection between Froome Lake and the regional groundwater flow system was been investigated through the installation of a monitoring well network (AMEC 2005a), lakebed sediment sampling (AMEC 2009a) and lake level and shallow groundwater level monitoring (AMEC 2010b). The lake bed sampling concluded that the lake bed deposits are comprised of fine grained materials such as sandy silt, clayey silt, sand and silt, and silty clay. The low permeability of these sediments was interpreted by AMEC (2009b) to be perching surface water and making it independent of the groundwater regime. Further, a comparison between the bottom elevation of Froome Lake and the elevations of monitoring wells completed west and east of the lake indicates that the lake stage (~286 m amsl) is 7 to 10 m higher than the groundwater elevations in nearby monitoring wells (AMEC 2009a). The lowest reported lake bed elevation was 276.5 m amsl, which is less than the highest reported groundwater elevations in nearby monitoring wells (279 to 280 m amsl) (AMEC 2009b). From this information in can be concluded that there is a hydraulic gradient from the lake to the shallow groundwater. The data do not prove that Froome Lake is hydraulically disconnected from the shallow groundwater flow system, although the large head difference suggests that the lake bed permeabilities are low. The monitoring wells used to evaluate lake-groundwater interactions are not screened across the water table, although vertical gradients in the local groundwater system are small enough that this approach is expected to be valid.

The existing underground workings are within approximately 700 m of Froome Lake and dewatered to 235 m bgs (dewatering recommenced in 2004), with no apparent effect on lake levels (R. Loughead, Personal Communication). Furthermore, monitoring of lake levels since 2009 suggest that pit excavation has not affected lake levels or trends in the area lakes, including Froome Lake (AMEC 2010b). This suggests that the hydraulic resistance of the lake bed may be low enough that the flux generated by lower groundwater levels is minimal, and thus lake levels are not impacted. Additional groundwater level lowering as a result of pit expansion could result in higher hydraulic gradients from Froome Lake to the groundwater system. However, the magnitude of this flux will be limited to by the reportedly low permeability lake bed materials (AMEC 2009b).

HYDRAULIC TESTING AND CHARACTERIZATION

Two test wells and multiple observation wells were installed as part of the hydrogeologic studies at the Project Site (AMEC 2005a) for the purpose of testing the overburden aquifer. Meaningful data were obtained from a step test and a constant-rate discharge test completed on a single well (PW-2) south of the proposed open pit. The well was completed with 7.3 m of 10-slot, 150 mm diameter stainless steel well screen into sand with trace clay. No meaningful responses were measured in observation wells as a result of the 46 m^3/day (7 Imperial gallons per minute) stress, which was constrained by the limited available drawdown and saturated thickness (approximately 13 m). As a result, only recovery data in PW-2 were used to estimate a hydraulic conductivity of approximately 1×10^{-5} centimetres per second (cm/s). Based on professional experience, this value is consistent with the typical range for silty sand, albeit on the low end of the range.

Packer testing completed on bedrock coreholes by AMEC (2005a) suggests that the hydraulic conductivity of the bedrock decreases with increasing depth. This concept is consistent with the geologic logging of the rock core and rock quality designation data, which suggest the bedrock is more competent at depth. The hydraulic conductivity estimates ranged from approximately 2×10^{-4} cm/s in the upper weathered bedrock to 2×10^{-6} cm/s is the deeper bedrock. The geometric mean of all values was 1×10^{-5} cm/s.

Slug testing was completed on 15 monitoring wells with screened intervals in various geologic materials encountered at the site. These estimates were used to constrain hydraulic conductivity estimates for finer-grained units and the overburden-bedrock contact zone. The values, which ranged from 2.7 x 10^{-3} to 4.6 x 10^{-7} cm/s, generally were within expected ranges and were largely consistent with the results from the other hydraulic testing programs and literature values for similar geologic media.

MINE WATER CHEMISTRY

Mine water from the underground workings is collected and pumped to the surface and reports to the Holding Pond. The water is treated for suspended solids (via settling), arsenic (via ferric sulphate addition) and ammonia (via aeration) in the Holding Pond and water treatment system (AMEC 2010a). Water samples were collected from the mine water sump in 2004 and 2008 to characterize the untreated mine water. Results from both sampling events are provided in Table 19.11 (reproduced from AMEC 2010a). The iron concentration in the 2004 sample is likely attributed to suspended solids (AMEC 2010a).

Table 19.11 Mine Water Chemistry

Parameter	2004 Sample (mg/L)	2007 Sample (mg/L)
Total suspended solids	Not analyzed	7
Antimony	<0.10	0.0035
Arsenic	0.18	0.329
Copper	0.008	0.003
Iron	3.13	0.22
Lead	0.12	<0.001
Molybdenum	0.027	0.016
Nickel	0.066	0.077
Zinc	0.016	0.011

Note: "<" denotes less than method detection limit
mg/L – milligrams per Litre

OPEN PIT DEWATERING

The Phase I pit does not currently require active dewatering, with the exception of capturing small volumes of seepage along the overburden-bedrock interface on the east side of the pit; that water is collected in a sump and pumped to the Holding Pond. Berms divert surface water away from the Open Pit. Precipitation falling on the pit floor and pit walls flows to the low points in the pit, where it rapidly infiltrates into the underground workings (Chris Frank, Personal Communication), aided by unplugged exploration holes that penetrate the underground workings (AMEC 2009a). Although designed sumps are present in the Phase I pit, it has not been necessary to use them. Currently, there is very little seepage into the Phase I Open Pit, and what is present appears to occur along the overburden-bedrock contact (Figure 19.5). The dry conditions in the pit are a result of favourable weather conditions and the fact that the pit area was already drained as a result of continued dewatering of the existing underground workings (conducted under PTTW No. 00-P-6025). With the water level in the bedrock already maintained below the pit floor, active dewatering has been largely unnecessary.

Figure 19.5 Seepage Along Overburden-bedrock Contact

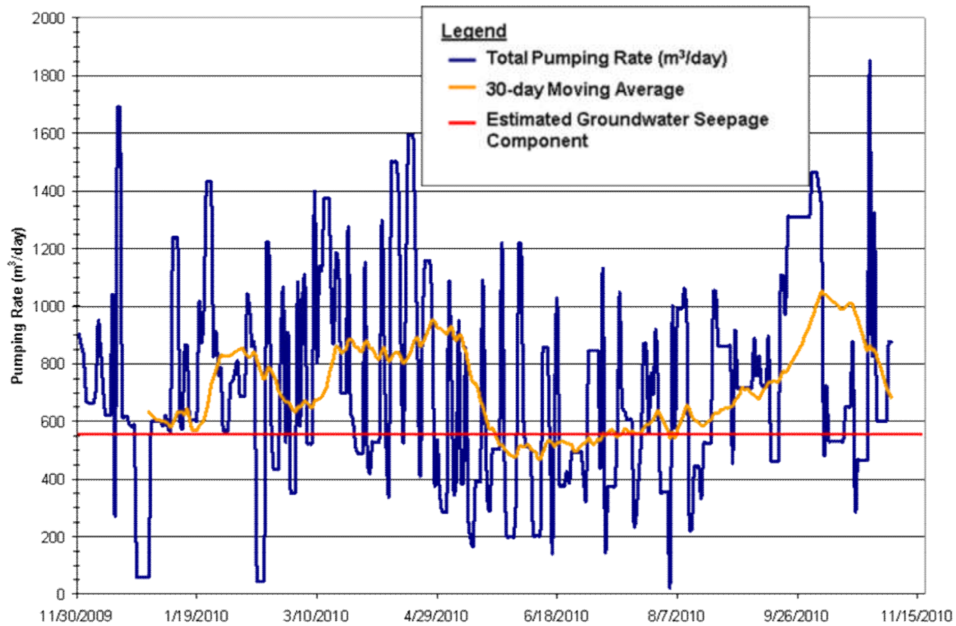
OPEN PIT WATER MANAGEMENT

SRK (2008) determined that, based on the 1:25 year storm event modeling, the ultimate Open Pit area could expect to see approximately 33,000 m^3 of water in a 24-hour period (SRK 2008). The open pit sump design allows for this quantity, with the pump system designed to pump this volume over a 48-hour period. Two pumps will feed two, 20.3 cm discharge lines that report to the Holding Pond. Based on the planned pit configuration, these required designs have not changed.

UNDERGROUND MINE WORKINGS DEWATERING

Dewatering of the underground mine workings was restarted in January 2004 and continues to the date of this report. During that time, water stored within the workings and the drainable porosity of the surrounding bedrock was removed. The dewatering rates from 2010 (through October) averaged 716 m^3/day (versus 626 m^3/day in 2009), with maximum daily rates as high as 1849 m^3/day (Figure 19.4). The underground workings are dewatered via a sump in the 235 level exploration drift, which is pumped via a series of three sump areas to the Holding Pond. Based on the dewatering rates (Figure 19.6) the groundwater seepage component is estimated to be approximately 550 m^3/day, which is consistent with previous estimates (AMEC 2009a).

Figure 19.6 Mine Dewatering Discharge Rates



Note: Data provided by Brigus Gold

19.2.2 Predictive Analysis

Groundwater Modeling

Based on the hydrogeologic investigations conducted in 2004, 2005 and 2009, the Modular Finite-Difference Flow Model (MODFLOW) developed by McDonald and Harbaugh (1988) for the United States Geologic Survey (USGS), was used to simulate groundwater flow in the vicinity of the Black Fox Site (AMEC 2005a, AMEC 2005b, AMEC 2009a). The most recent flow modeling completed for the Project Site considered the "Phase II" pit (this term relates to the two phases of permitting and includes Pit Phases I through III) and underground workings (AMEC 2009a). This model incorporated the larger Phase II pit and the lower permeability layers associated with lake bed sediments analysis (AMEC 2009b). Froome Lake was simulated using a constant head boundary is this version of the model, and thus lake stage cannot be affected by simulated declines in groundwater elevations beneath or adjacent to the lake. A steady-state calibration was performed using water levels measured in 40 site monitoring wells and the reported groundwater seepage rate of approximately 500 m^3/day into the existing underground mine workings. The calibrated model parameters (AMEC 2009b) are included in Table 19.12.

Table 19.12 Calibrated and Tested Hydraulic Conductivity Values

Geologic Media	Hydraulic Conductivity (cm/s)	Range of Tested Hydraulic Conductivity Estimates(3) (cm/s)
Sand	$1x10^{-2}/5x10^{-3}$	$2.7x10^{-3}$ to $9.6x10^{-6}$ (4)
Silty Sand / Sandy Silt	$5x10^{-5}$	$4.2x10^{-4}$ to $3.2x10^{-5}$
Silty Clay	$3x10^{-6}$ (2)	$1.9x10^{-6}$ to $4.6x10^{-7}$
Glacial Till	$3x10^{-4}/9x10^{-5}$	$2.5x10^{-4}$ (5) to $1.7x10^{-5}$
Weathered Bedrock(1)	$5x10^{-5}$	$2.4x10^{-4}$ to 7.7 x 10^{-5}
Bedrock	$3x10^{-6}/1x10^{-6}$	$1.5x10^{-5}$ to $2.4x10^{-6}$

Note:(1) Upper 5 m to 15 m thick zone of the upper bedrock;
(2) 1x10-6 cm/s in the areas beneath Froome Lake, Middle Pond and adjacent ponds;
(3) Estimates based on the values reported in AMEC (2005a);
(4) Upper value is from a sand and silt and lower value is from sand. Simulated values are more reasonable estimates.
(5) Value reported from the till/bedrock contact

The calibrated flow model was subsequently verified by simulating the gradual excavation of the Phase I Open Pit from January 2009 to the end of July 2009 using a transient simulation (no storage properties were reported). The model results show two monitoring wells (BH04-14 and BH04-19) between the 0.5 m and 0.1 m predicted drawdown contours the remaining 10 site monitoring wells beyond the 0.1 m drawdown contour. Water-level measurements made between March 2009 and August 2009 show that the depth to water in BH04-14 decreased by 1.33 m and the depth to water in BH04-19 increased by 1.7 m during that time period. The remaining site wells had generally shallower depths to water in August 2009 than historically (June 2004 – February 2005). The monitoring data largely support the assertion that the excavation of the pit between March 2009 and August 2009 did not have a significant effect on the overburden aquifer and that the drawdown predicted by the model may be conservative.

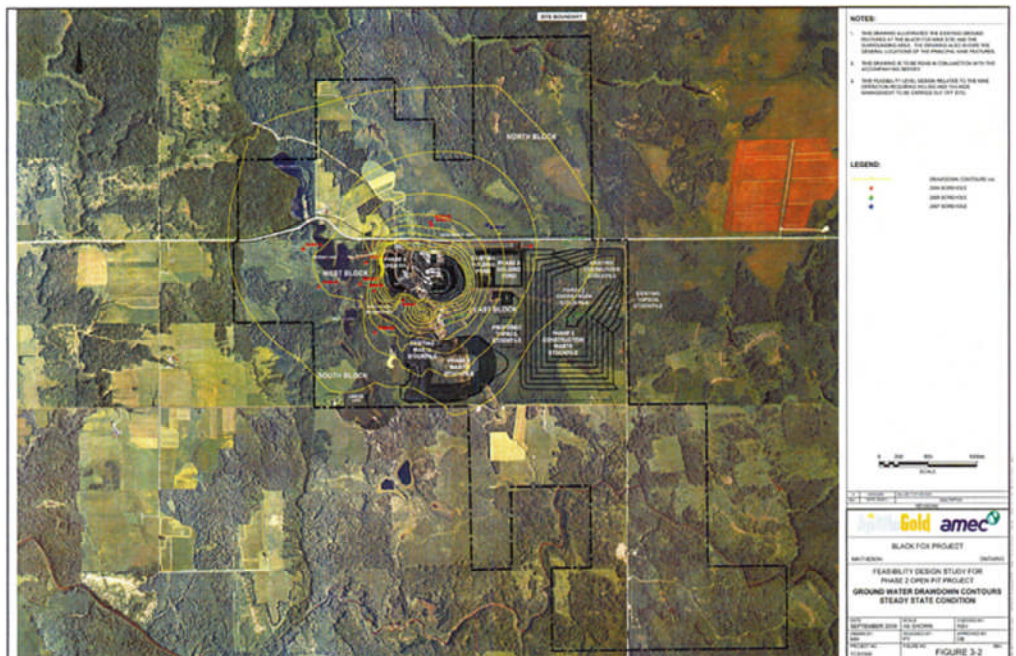
Given that the underground workings were dewatered to 235 m and the workings underlie the pit, little new hydraulic stress was exerted on the system by the pit excavation. The groundwater level impacts would largely be related to seepage into the pit along the overburden-bedrock contact rather than the dewatering of the bedrock, since this condition already existed in 2009.

The groundwater flow model predicts that the steady-state groundwater seepage rates into the pit and underground workings will be 653 m^3/day and 162 m^3/day, respectively. The total of 815 m^3/day is approximately 200-300 m^3/day higher than the current estimated groundwater seepage rate of approximately 550 m^3/day. Groundwater seepage rates during initial pit excavation were estimated to be 1,000 m^3/day (AMEC 2009a), to account for water released from storage.

IMPACT ASSESSMENT

The steady-state overburden groundwater level drawdown associated with the Phase II pit and underground workings dewatering extends a maximum of 1750 m to the northwest (as defined by the 1 m drawdown contour relative to calibrated flow model when only the underground mine workings were being dewatered) from the pit (Figure 19.7). Overburden groundwater level drawdown relative to initial hydrologic conditions prior to any mine operations or dewatering and groundwater level drawdown in the bedrock were not presented in AMEC (2009a). Changes in Froome Lake stage elevations resulting from the predicted groundwater level drawdown was not simulated as a result of using a constant head boundary condition. Results of the groundwater flow modeling were included in the PTTW Amendment Application (PTTW No. 7460-7K8RAB), which is being reviewed by the Ministry of Environment.

Figure 19.7 Overburden Groundwater Drawdown Steady-State Conditions



Reproduced from AMEC (2010a), Figure 3-2.

A sensitivity analysis on the initial model runs (AMEC 2005a) determined that groundwater seepage rates were most sensitive to increases in the hydraulic conductivity of the glacial till. A one hundred percent increase in the modeled till hydraulic conductivity value of 3.0×10^{-4} to 6.0×10^{-4} cm/s, results in a twenty percent increase in groundwater seepage to the pit. No sensitivity analysis was completed for the revised groundwater flow model (AMEC 2009a). However, given that the geologic model which forms the foundation of the groundwater flow model is largely unchanged the relative change in groundwater inflows from the earlier sensitivity analysis (AMEC 2005a) is a reasonable first approximation of relative changes that could be expected for the revised inflow estimates and pit configuration. The Phase II pit may be more sensitive to changes in overburden

The footprint of the proposed pit is largely within the existing cone of depression associated with the dewatering of underground mine to 235 m bgs. Therefore, additional groundwater level drawdown is likely constrained to potential effects in the overburden as a result of groundwater seepage associated with the western and southern extents of the pit. The maximum drawdown at the seepage face cannot exceed the saturated thickness of the overburden/upper weathered bedrock, which is on the order of 10 to 15 m. Further, based on the groundwater modeling results and historical groundwater level monitoring records, effects to local residential wells is expected to be negligible (AMEC 2010a).

Both a groundwater level monitoring program and water takings monitoring program are in place as part of existing conditions in the PTTW (PTTW No. 7460-7K8RAB). Continual assessment of the monitoring results is the most effective manner in which to evaluate impacts associated with ongoing pit development.

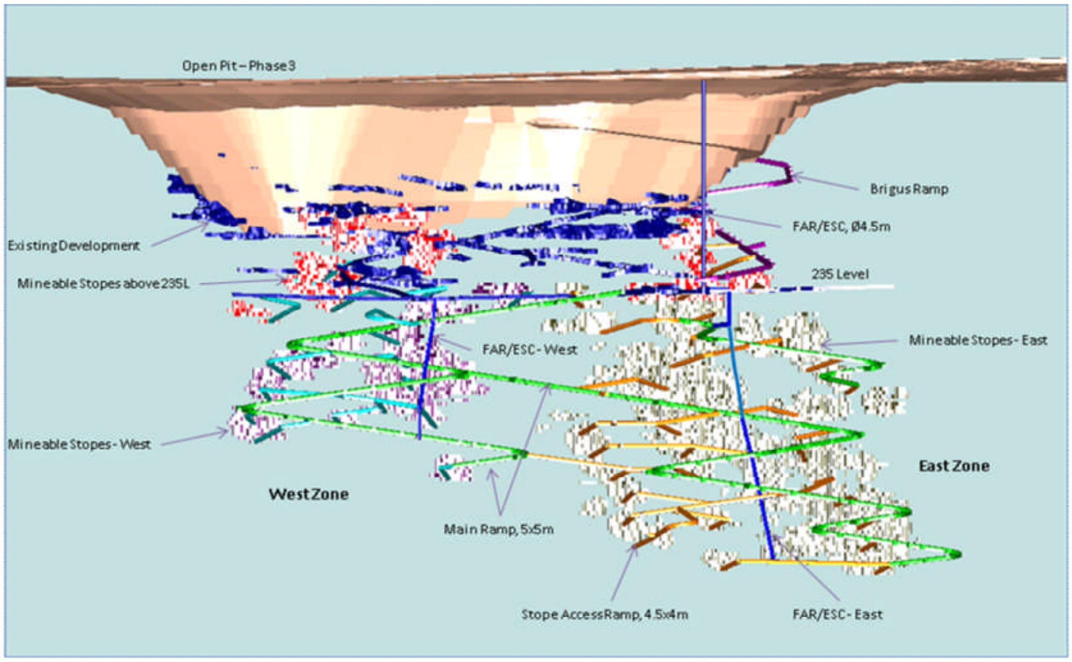
OPERATING COSTS

Build out of the planned Open Pit will result in intersecting a greater thickness of saturated overburden in the west and south walls of the pit. This may result in a sustained groundwater inflow along the overburden-bedrock contact. Assuming the geology is as projected (AMEC 2005a; AMEC 2009a; AMEC 2009b), then the groundwater inflows should be within the range of estimates provided (AMEC 2005a; AMEC 2009a). The current dewatering strategy is considered appropriate, although the possibility exists that the designed sumps and pumps may have to be utilized as the pit footprint is expanded. No major changes in the groundwater inflow rates or dewatering strategies are anticipated so the estimated dewatering costs are considered appropriate.

19.3 UNDERGROUND MINING

Ore reserves around and below the final open pit limits will be mined by underground methods. Underground reserves include remnants from the old workings, located mostly above elevation 9770 (235 Level), and new ore blocks below that elevation, grouped into two main zones – east and west (see Figure 19.8).

Figure 19.8 Overall Mine Section Showing Open Pit & Underground



Current underground activities include mining of the remnants via existing openings, development of the access ramp (an extension of the future Brigus Ramp) to the East Zone and West Zone starting from the 235 Level, and installation of manway and services in the recently bored main ventilation raise. Access from surface is via the existing Exall ramp whose portal is now at the brink of the open pit. Fresh air is forced down an existing raise; used air is returned to surface via the ramp.

Enclosed in the mass of the ultimate pit plan, both the Exall ramp and the vent raise will eventually be mined through. Future access to underground will be provided by a new Brigus Ramp collared from a bench near the bottom of the final pit excavation. A new fresh air raise has recently been bored 4.5 m in diameter from surface down to the 235 Level. The raise is now being equipped with a manway and service lines. When completed, it will provide an underground second egress. Excavation and equipping of the fresh air raise will proceed to lower levels in progress with mining horizon.

Production from the underground operations at a nominal rate of 1,000 t/d will complement the open pit production to sustain the required mill feed of 2,000 t/d. After the open pit production ceases in 2015, underground production will be supplemented by stockpile open pit material.

Table 19.13 Underground Highlights

Deposit Type	Archean age, lode gold deposit located within the Abitibi greenstone belt
Mine Plan Tonnage	2.936 Mt @ 6.85 g/t Au below 235 Level (Elev. 9770)
In-situ Rock Density	2.83 t/m^3
Mining Rate	1,000 t/d
Mining Method	Mechanized Cut & Fill with rockfill, cemented at sill cuts only and specific areas as required.
Mine Life	8.25 years

19.3.1 SELECTION OF MINING METHOD

Cut and fill is the most suitable method based mainly on the wide spatial distribution of values, the complex nature of mineralization, the low dip (generally 50°) and geomechanical properties of the orebodies.

Alternative underground mining methods, such as longhole stoping, are generally not applicable due to the potential for greater dilution and ground control issues in shallow dipping stopes.

19.3.2 UNDERGROUND MINEABLE RESOURCES

CUT-OFF GRADE CALCULATION

Calculation of mineable resources below 235 Level (elevation 9770) involves the determination of the applicable mining method, and economic ore blocks based on cut-off grade and stope design. A cut-off grade of 2.54 g/t for underground and 0.88 g/t for the open pit was calculated based on parameters shown in Table 19.14.

Table 19.14 Underground Cut-off Grade

Description	Terms	Unit	Amount
Refinery Terms:			
Payable Accountability	JM Contract	%	99.91%
Refining	JM contract	$Cdn/oz	$0.55
Transportation	Brinks	$Cdn850/$500K	
		$Cdn0.65/$1000/$501-$2M	
		$Cdn0.47/$1000/>$2.1M	
NSR* Calc:			
Au price	Brigus	US$/oz	$1150
Au price	Sandstorm	US$/oz	$500
Au price accountability	Brigus	oz	88%
Au price accountability	Sandstorm	Oz	12%
F/X		$1Cdn=$1US	$0.9625
Mill recovery	Brigus	%	94%
Grams to oz	Factor		0.03215
Resource		tonnes	1,000
In situ Au		grams	2,540
In situ Au		oz	81.66
Recovered Au		oz	76.76
Refinery payable		oz	0.07
Brigus accountable Au		oz	76.69
Brigus payable		oz	67.49
Sandstorm payable		oz	9.20
Brigus NSR gross revenue		US$	$82,214
Refinery	JM	US$	-$41
Transportation	Brinks	US$	$0
Brigus NSR net revenue		US$	$82,173
Brigus NSR net revenue		US$/oz	$1,071.47
Brigus NSR net revenue		US$/t ore	$82.17
C&F mining cost		US$/tonne	-$56.48
Ore handling cost		US$/tonne	-$5.46
Mill cost		US$/tonne	-$13.83
Assay lab		US$/tonne	-$1.64
G&A cost		US$/tonne	-$4.87
Brigus margin		US$/tonne	-$0.10
Resource COG	C&F	grams/tonne	2.54

*NSR – Net Smelter Return, G&A – general and administrative

Table 19.15 Open Pit Cut-off Grade

Description	Terms	Unit	Amount
Refinery Terms:			
Payable Accountability	JM Contract	%	99.91%
Refining	JM contract	$Cdn/oz	$0.55
Transportation	Brinks	$Cdn850/$500K	
		$Cdn0.65/$1000/$501-$2M	
		$Cdn0.47/$1000/>$2.1M	
NSR Calc:			
Au price	Brigus	US$/oz	$1150
Au price	Sandstorm	US$/oz	$500
Au price accountability	Brigus	oz	88%
Au price accountability	Sandstorm	Oz	12%
F/X		$1Cdn=$1US	$0.9625
Mill recovery	Brigus	%	94%
Grams to oz	Factor		0.03215
Resource		tonnes	1,000
In situ Au		grams	880
In situ Au		ozs	28.29
Recovered Au		ozs	26.59
Refinery payable		ozs	0.02
Brigus accountable Au		ozs	26.57
Brigus payable		ozs	23.38
Sandstorm payable		ozs	3.19
Brigus NSR gross revenue		US$	$28,484
Refinery	JM	US$	-$14
Transportation	Brinks	US$	$0
Brigus NSR net revenue		US$	$28,470
Brigus NSR net revenue		US$/oz	$1,071.47
Brigus NSR net revenue		US$/t ore	$28.47
C&F mining cost		US$/tonne	-$2.65
Ore handling cost		US$/tonne	-$5.46
Mill cost		US$/tonne	-$13.83
Assay lab		US$/tonne	-$1.64
G&A cost		US$/tonne	-$4.87
Brigus margin		US$/tonne	-$0.02
Resource COG	Open Pit	grams/tonne	0.88

A summary Resource Statement – Ordinary Kriging Model for the region below 9820 m elevation is shown in Table 19.16.

Table 19.16 Resource Statement Below 9820 m Elevation

Cut-off 2.54 g/t Au	Indicated	2,504,800 tonnes, 7.192 g/t capped Au
	Inferred	115,200 tonnes, 5.816 g/t capped Au

The underground stope design includes only the indicated resource blocks with higher than cut-off grade. The reserve estimate is based on 95% mining recovery and applying a total dilution factor of 25%, 15% planned with 1 g/t grade and 10% unplanned with 0 g/t grade. The 15% planned dilution consists of hanging wall and footwall low grade material included in the stope shape. Backfill material removed with the ore on floors comprises the 10%. The 100% mining recovery is optimistic from an operations perspective. To establish a confidence limit of +/- 5% a 95% recovery factor was applied to the reserve estimate, which resulted in a difference of 24,414 oz.

Analysis of ore zones for potential stoping yielded the results in Table 19.17.

The underground portion of the Black Fox Project is expected to extend from below the existing mine (near 200 m depth below surface) to approximately 500 m depth below surface. Geotechnical data has been limited to the logging completed by Apollo Gold staff and additional investigations are recommended to be completed during mine development to confirm assumptions made in this study.

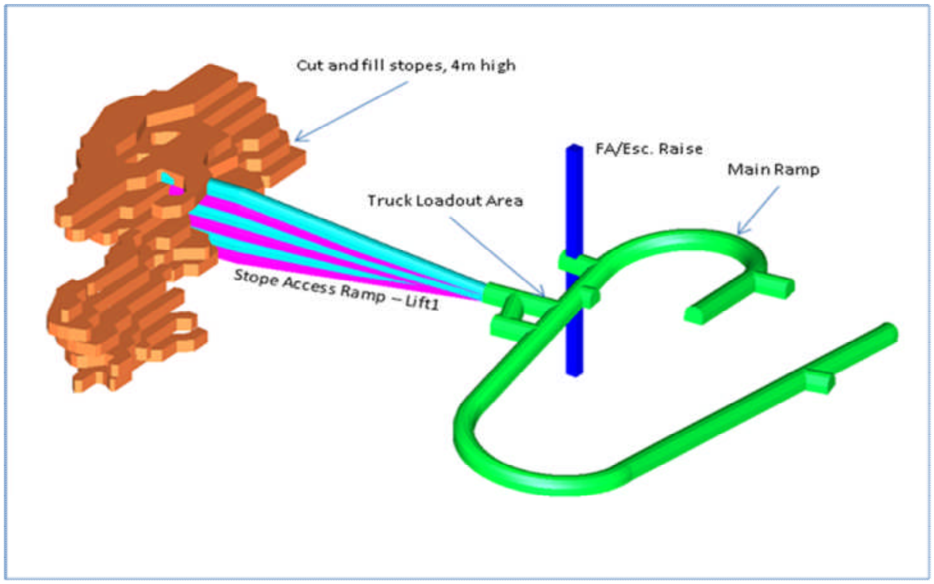
Table 19.17 Reserve Statement, Black Fox Mine (at Oct 31, 2010)

Zone	Classification	Recovery %	Tonnes	Au g/t	Au oz
Pit	Probable	95%	3,159,800	3.228	327,920
U/G	Probable	95%	2,936,000	5.933	560,008
Stockpile	Proven		352,068	1.630	18,446
Total	Proven/Probable		6,447,868	4.372	906,375

19.3.3 *Stope Design*

The underground stopes were designed by slicing the resource model on 4 m plan views. The design was constructed on only the indicated resource blocks above the 2.54 g/t cut-off grade (diluted). The 1 g/t to 2 g/t indicated blocks were displayed during the design process but were only included in the design if higher grade material could be added by including them. The inferred blocks were not displayed during the design process. Polygons were digitized to define the stope outlines on the 4 m plan views. The polygons were then extruded 4m vertically to define the three dimensional (3D) stope shape. Figure 19.9 shows an example of an isometric view for a typical stope with the stope outline and the indicated resource blocks.

Figure 19.9 Isometric View of a Typical Stope



STOPE DESIGN PARAMETERS (CUT AND FILL)

Factors taken into account when selecting the mining method included:

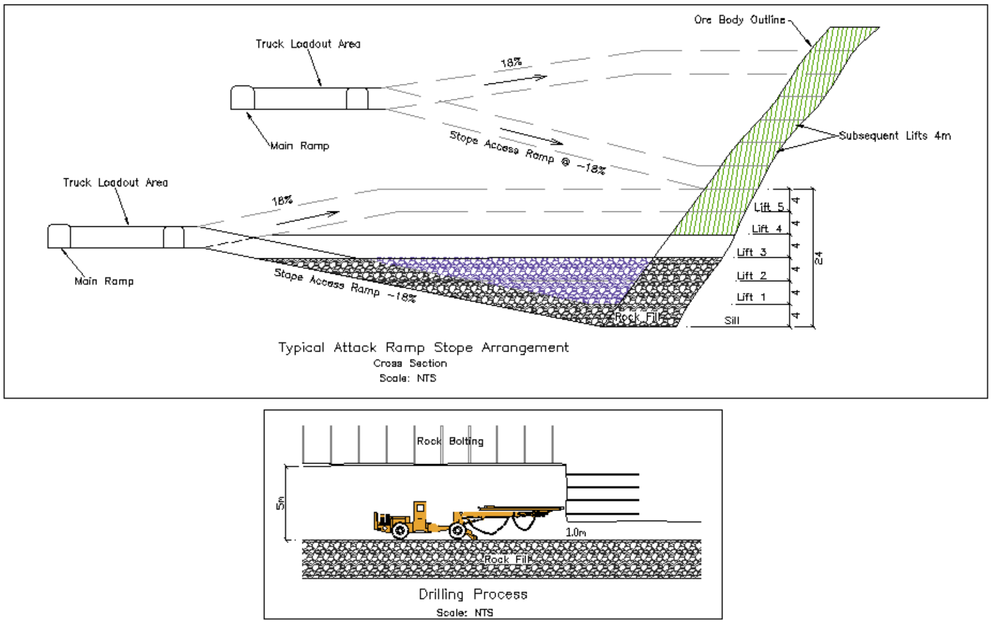
- Geometry - continuity, size and shape - of the orebody.
- Dip angle and plunge of the orebody.
- Geomechanical properties of the orebody.
- Value of in-situ ore, mining dilution and recovery.

C&F was determined to be the most suitable method for the ore below the 235 Level, for the following reasons:

- shallow 50° dip of the orebody
- spatial and complex distribution of economic blocks will require high degree of grade control and stoping flexibility
- competence of ground.

Figure 19.10 shows an example of the mining sequence for a typical cut and fill lift.

Figure 19.10 Typical C&F Mining Method



A stope ramp will be driven down to the sill cut of the stope where mining begins. Depending on the width of the orebody, mining can be done with a full face jumbo round and slashing. The height of the cut will be limited to 4 m, primarily for grade control, and for ground stability. When the cut has been mined, development waste rock will be hauled into the stope for backfill and to establish a floor from which to mine the next cut. Access to the next cut will be created by slashing the back of the stope ramp. The sequence will be repeated for a total of 6 cuts or “lifts”, covering a vertical interval of 24 m. The second and subsequent cuts will also be mined by using a jumbo where a horizontal face is drilled and blasted (breasting) to the opening above the fill floor below. In areas where the ore width is greater than stable spans, a lift will be mined by drift-and-fill method, starting from the hangingwall far end and retreating to the access ramp.

Explosives will consist of a combination of ANFO and packaged emulsion products.

BACKFILL DESIGN

The primary consideration for backfilling is to achieve a consistent tight fill throughout the wider cut and fill areas. This is necessary where multiple drift-cuts will be mined adjacent to each other on the same cut. If tight fill is not achieved the apparent span could get very large resulting in a potential for stope back instability.

INTERACTION BETWEEN UNDERGROUND AND OPEN PIT MINING

The open pit will mine out the current underground access portal and ramp. The new surface access ramp will be completed by this time. The underground mining will extract the orebody in a top down sequence. Open pit mining and operational underground stoping are never in proximity to each other, thus the question of interaction from a blasting and ground control point of view does not arise. There will be a hydrological connection in that any of the surface water reporting to the open pit mine operations and not pumped out will ultimately report to the underground mine operations. Upgrades to the underground water pumping system have been incorporated into the underground mine design.

The open pit benches will be mining through old backfilled and open stopes in the upper levels of the old mine only after confirming the location of these stopes through test drilling.

19.3.4 DEVELOPMENT DESIGN

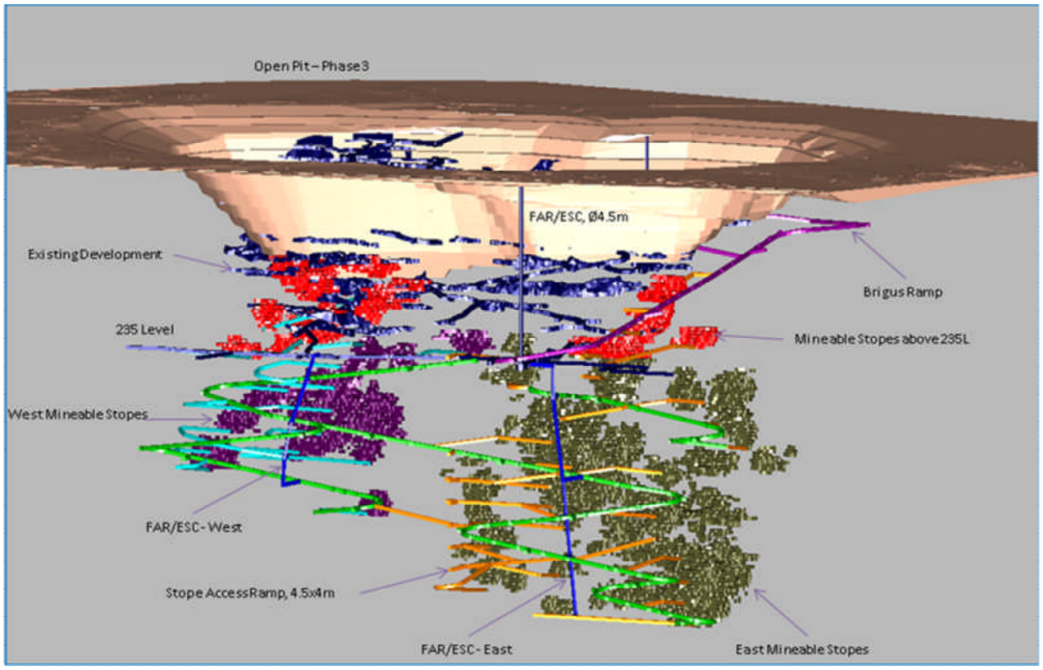
The underground development is based on the stope design work described above.

EXISTING MINE WORKINGS

Underground openings exist from surface down to the 235 Level (measured vertical 235 m below the surface). These include the Exall access ramp, ventilation raise, and exploration drift on 235 Level, stopes and cross-cuts to access the stope areas in Figure 19.9 . Developed in 1997 and mined subsequently using conventional trackless equipment, the drifts and cross-cuts are sized to accommodate such equipment. Operations ceased in 2001. Historical production came from development headings and random room and pillar stopes. In conversation with Brigus staff, most of the stopes were backfilled with rock.

With the resumption of mining operations in March 2009, new internal ramps, drifts, cross-cuts and stopes have been created. The most important recent opening is the 4.5 m diameter fresh air raise from surface to 235 Level.

Figure 19.11 Existing Underground Openings



Surface Access Development

The future underground main access will be a new ramp with a portal on a bench (approximate 9913 elevation) close to the bottom of the final open pit extents (Figure 19.12). Skirting the final pit walls, the ramp will break through the 235 Level exploration drift, at a spot close to the bottom of existing new 4.5m Ø fresh air raise (Figure 19.13). From this level, the ramp will be located optimally central between the East Zone and West Zone down to the horizon which the West Zone bottoms out, approximately elevation 9600. From this elevation down, the ramp will be located closer to the East Zone.

An initial portal associated with the "Phase 2" open pit sequence has been developed from the "Phase 2" pit ramp. Anticipated in the transition from "Phase 2" pit to "Phase 3", the east wall of the pit will be excavated. The "Phase 2" portal excavation is anticipated to be lost during the "Phase 3" preparation. As such, a "Phase 3" portal is planned and has been incorporated in the schedule to account for this potential.

Figure 19.12 Overall Section Showing Brigus Ramp

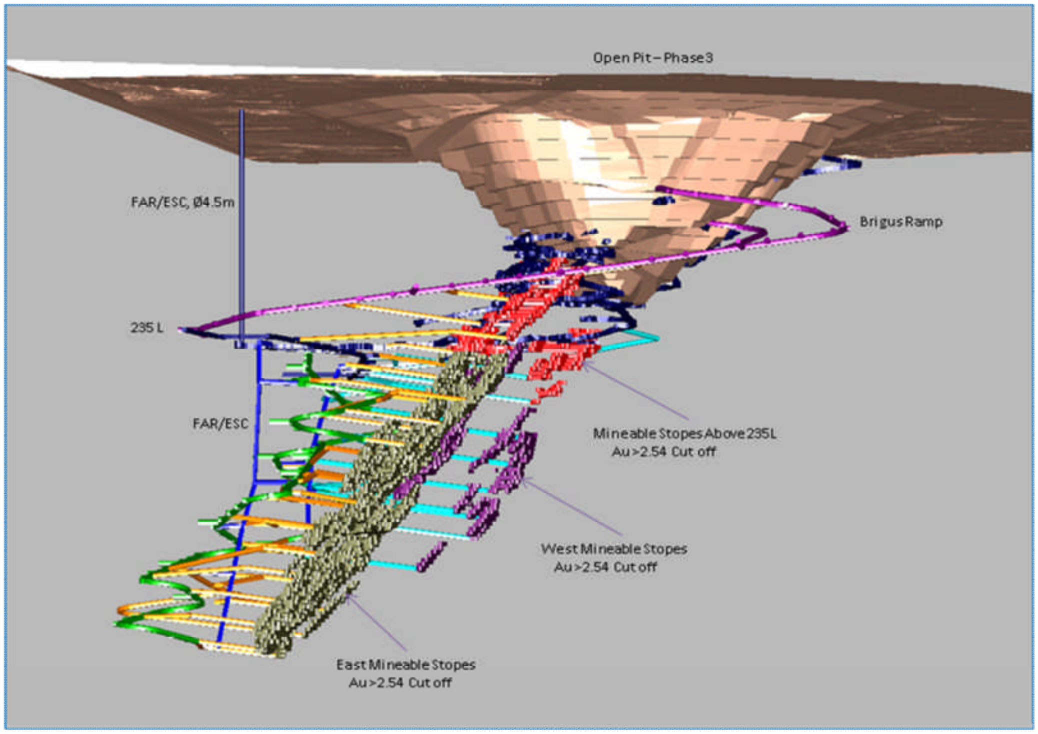
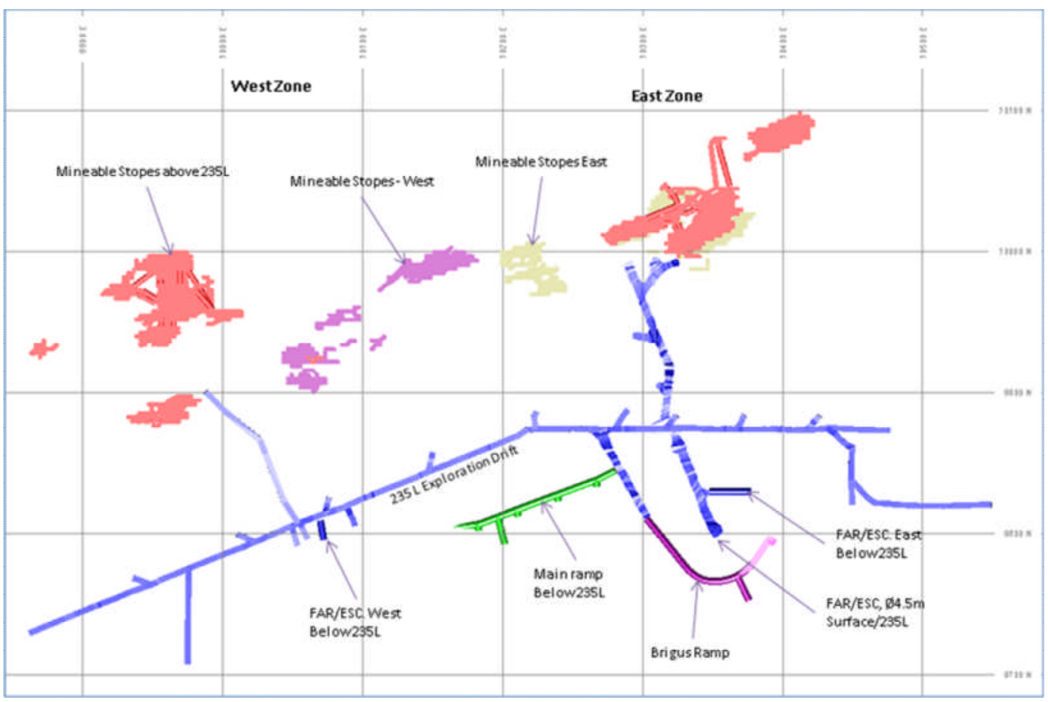


Figure 19.13 Plan View 235 Level Showing FAR and Ramp Location



With cross-sectional nominal dimensions of 5 m wide x 5 m high, the ramp can accommodate the largest unit of the trackless mobile equipment fleet. Passing bays will be established by widening strategic portions of the ramp. During ramp development, remuck chambers will be excavated at a maximum spacing of 150 meters. As ramp development progresses, these chambers will cease as remuck bays and thus can be used as sumps, electrical equipment station and storage rooms. Collars of work level drifts can also be used initially as remuck chambers. Where there are no suitable openings along the ramp, safety stations will be excavated at no more than 30 m apart.

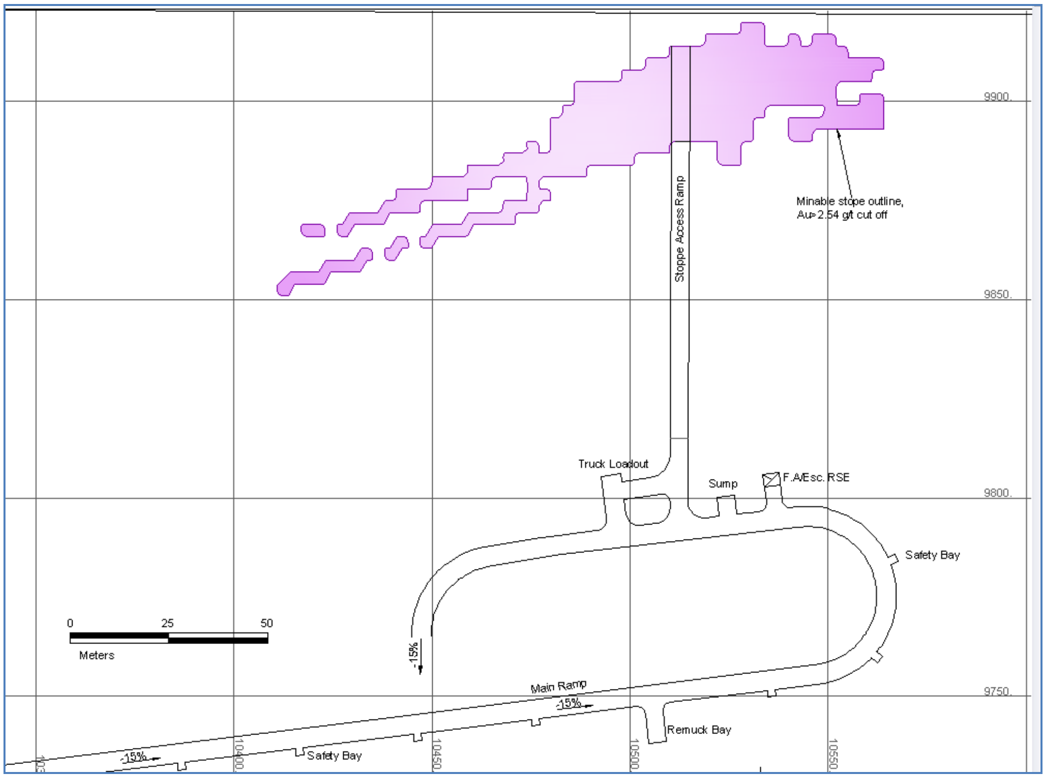
STOPE ACCESS DEVELOPMENT

Once the main ramp reaches a work level elevation, lateral development will commence. Design of lateral development is geared to the stoping method described in the following section. With six, 4 m high stope floors (one sill cut and five lifts) accessed from the same cross-cut horizon, level interval will be 24 meters. A typical level (Figure 19.14) will generally include the following:

- All openings where 50-tonne trucks travel must be 5 m wide x 5 m high minimum.
- A cross-cut off the main ramp towards the ore zone terminating at the truck loadout station.
- A remuck/truck loadout station established approximately 75 m from the stope wall.
- A 4 m x 4 m drift to connect to the fresh air raise.
- A stope access ramp, 4.5 m wide x 4 m high, driven from the truck loadout station to the sill cut of the stope at -20% grade. When the sill cut is completed, this ramp will be backslashed to access the next floor. This procedure will be repeated until the top floor is mined (Figure 19.14). Depending on the contiguous strike length of the ore block, this stope access ramp arrangement will be replicated along strike at nominal 200 meter spacing.

In a case where there is an isolated ore block askew to a general level elevation, a chase ramp or drift will be driven to access and mine it.

Figure 19.14 Typical Level Plan



19.3.5 *Production and Development Schedule*

The underground mine development and production schedule was prepared by Python Mining Consultants.

Development Productivity

The development schedule is based on Brigus crews with the following performance:

Ramp and Lateral Development:

- 2-boom Jumbo crew = 5 m per day per jumbo (single or multiple heading 5 m total maximum per day).
- There are two 2-boom jumbo crews.

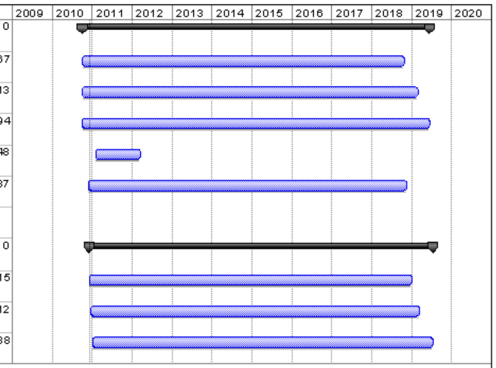
Raise development consists mainly of the Fresh Air Raise (FAR) which will be driven by Alimak raise climber at an average rate of 3 m per day.

Development Schedule

The development and production summary schedule Gantt Chart is shown in Figure 19.15

Figure 19.15 Development and Production Summary Gantt Chart Schedule

ID		Task Name	Start	Finish	Unit	Qty
1		Development	10/1/10	6/12/19		0
2		Above 235mL	10/1/10	10/25/18	m	3269.67
3		East Ramp below 235mL	10/1/10	2/25/19	m	10665.13
4		West Ramp below 235mL	10/1/10	6/12/19	m	4627.94
5		East Vent Raise	2/1/11	3/12/12	m	287.48
6		West Vent Raise	11/25/10	11/14/18	m	152.87
7						
8		Production	12/3/10	7/13/19		0
9		Stopes Above 235mL	12/3/10	12/31/18	t	439659.15
10		East Stopes Below 235mL	12/20/10	3/10/19	t	1870137.12
11		West Stopes Below 235mL	1/1/11	7/13/19	t	626271.88



STOPING PRODUCTIVITY

A round 5 m wide x 4 m high x 3.7 m long yields 210 tonnes. For an average total advance of 5 m per day, a face will produce 284 tonnes. There must be five stopes in the production/backfill cycle plus 2 spare stopes.

PRODUCTION SCHEDULE

Production from below 235 Level starts in December 2010 at 250 t/d ramping up to the target steady state average of 850 t/d by May 2011. Production peaks at 1,000 t/d in November 2011 and sustained at this rate for the life of mine. Production ceases in the 3rd quarter of 2019 when all the existing reserves have been exhausted. The production schedule is shown in Table 19.18.

Table 19.19 summarizes the transition of resources to reserves as of October 31, 2010.

Table 19.18 Underground Production Schedule (at Jan. 1, 2010)

All tonnes = diluted	2011	2012	2013	2014	2015	2016	2017	2018	2019	Total
East Zone										
East Sill Tonnes	34,645	53,272	26,142	76,161	22,868	15,791	34,214	59,655	2,422	331,935
East Sill Grade	5.74	4.38	7.06	5.10	4.39	4.41	4.56	5.04	5.56	5.05
East MCF Stoping Tonnes	153,138	210,375	226,061	218,445	306,340	189,116	101,582	232,660	50,021	1,688,723
East MCF Stoping Grade	5.81	5.14	5.27	5.70	5.10	4.53	4.56	5.58	5.39	5.25
West Zone										
West Sill Tonnes	7,837	17,064	22,858		1,084	27,563	43,163	10,871	8,525	138,965
West Sill Grade	10.23	5.67	7.06		4.27	8.91	8.69	4.05	5.79	7.60
West MCF Stoping Tonnes	101,383	72,527	89,938	65,899	34,708	133,530	186,041	51,542	40,877	776,446
West MCF Stoping Grade	7.18	6.51	7.96	7.30	5.85	8.60	8.14	5.77	6.49	7.50
Total Sill & MFC										
Total Sill Tonnes	42,481	70,336	49,001	76,161	23,953	43,354	77,377	70,526	10,947	470,899
Total Sill Grade	6.57	4.69	7.06	5.10	4.38	7.27	6.86	4.89	5.74	5.81
Total MCF Stoping Tonnes	254,522	282,902	315,999	284,344	341,047	322,646	287,623	284,202	90,898	2,465,169
Total MCF Stoping Grade	6.36	5.49	6.04	6.07	5.18	6.21	6.87	5.61	5.89	5.96
Total										
Total Tonnes	**297,002**	**353,238**	**365,000**	**360,505**	**365,000**	**366,000**	**365,000**	**354,728**	**101,845**	**2,928,318**
Total Grade	**6.39**	**5.33**	**6.17**	**5.86**	**5.12**	**6.34**	**6.87**	**5.47**	**5.87**	**5.94**
Total Oz	**60,987**	**60,553**	**72,439**	**67,942**	**60,129**	**75,590**	**80,618**	**62,371**	**19,220**	**558,849**

Table 19.19 Summary of Underground Diluted Minable Reserves

Mineral Resource @ October 31, 2010					Mining Resource			Mineral Reserve @ October 31, 2010				
Zone	**Classification**	**Tonnes**	**Au g/t**	**Au oz**	**Dilution %**	**Tonnes**	**Au g/t**	**Classification**	**Recovery %**	**Tonnes**	**Au g/t**	**Au oz**
Pit	Indicated	2,328,274	4.196	314,113	30%	3,326,105	3.228	Probable	95%	3,159,800	3.228	327,920
	Inferred	667,100	2.610	55,977								
U/G	Indicated	2,446,667	7.416	583,342	25%	3,262,222	5.933	Probable	95%	2,936,000	5.933	560,008
	Inferred	115,200	5.816	21,541								
Stockpile								Proven		352,068	1.630	18,446
Pit	Indicated	4,774,940	5.846	897,455				Probable		6,095,800	4.531	887,928
U/G	Inferred	667,100	2.610	55,977								
Stockpile								Proven		352,068	1.630	18,446
Total		5,442,040	5.449	953,432				Proven/Probable		6,447,868	4.372	906,375

SCHEDULING PROCESS

The design and scheduling process does not address the detail within each cut and fill stope cut, instead it is assumed that as soon as the stope access ramps reach the ore, production will commence at a rate of 430 t/d until the complete cut tonnage was mined out. When the cut ore is mined out, backfill is placed at a rate of 650 t/d until the stope is filled. When the cut is filled the stope access ramp backslashing for the next cut starts. Over the quarterly timeframe used for the representation of the scheduling results in the economic model, this lack of in-stope scheduling detail is acceptable.

The schedule is built up by linking together stopes to mimic a crew moving from stope to stope over time. Because each stope produces development waste and requires backfill, in addition to mining ore, each area on average produces only 200 t/d of ore. It was therefore necessary to schedule five areas mining together to produce 1,000 t/d of ore consistently over the life of the mine. It is anticipated that only four mining crews (combined production and development) per shift will be required to keep these five areas producing consistently.

Internal ramp, stope access and raise development is scheduled to follow the progression of the stope mining.

DEVELOPMENT AND PRODUCTION SCHEDULE

During the startup production period between December 2010 and February 2011, the development priority will be the development of the new ventilation circuit. Two underground development crews will produce approximately 400 m per month during this period. In addition to the development work, there will be some stoping in the areas above 235 Level that will be mined from the current access ramp.

The new surface access ramp from 235 Level (5 m x 5 m) was started in July 2010 and advances at a rate of 180 m per month to be completed in February 2011 with a completed ramp with a portal accessing the pit ramp and connected to the existing 235 Level. This ramp is scheduled to be completed by February 2011 because the current surface access ramp will be mined out by the open pit in early 2011.

The 400 m per month mining rate for the ventilation system drifts and the internal ramps (underground drift/ramp 5 m x 5 m) increases to an average of 450 m per month from February 2011. By this time the drifts for the ventilation system are substantially complete and the focus is on the internal ramps and stope access drift preparation. In 2011, the production rate on the internal ramps and stope access drifts reduces to a steady state of 300 m per month until it is completed in mid-2014. The manpower from this development reduction will be moved to cut and fill production stoping. The underground drift/ramp (4.5 m wide x 4.0 m high) relates to the development of the ore access ramps into the ore. This work is carried out by the stope crews in addition to the backslashing, cut and fill ore mining and the backfilling.

Internal 3 m x 3 m Alimak raise development is as required to support the deepening of the internal ramps and stope blocks.

19.3.6 MINING METHOD

DRILLING AND BLASTING

For ramp and level development, 2-boom electric hydraulic (EH) jumbo drills will be used to drill blastholes. Blastholes will be nominal 1 3/4 inch (45 mm) diameter x 12 feet (3.7 m) average length for development headings.

Single boom EH jumbo will be used in MCF production drilling. Holes are 1 1/4 inch (32 mm) x 11 feet (3.4 m).

Overall explosive consumption has been based on the use of ANFO and packaged emulsions (13%) at a powder factor of 1.63 lbs/ton for mine development. It is assumed that emulsions are not needed in the MCF stopes, resulting in powder factor of 1.00.

WASTE HAULAGE

During ramp and lateral development, blasted waste rock will be mucked by an 8 cubic yard (yd^3) (LHD) unit and hauled to a remuck bay, or ideally directly into a 30-t truck fitted with an ejector box which will deliver to an empty stope for backfill. As a last resort, waste must be hauled to surface when all underground openings suitable for storage have been filled. Waste that is brought to surface will be piled in a designated area in the open pit, preferably in close proximity to the ramp portal. It will be retrieved, backhauled to underground by ore trucks once a stope is ready for backfilling.

ORE HAULAGE

Blasted ore will be mucked and hauled by an 8 yd^3 LHD to a truck loadout station. A typical station will be adjacent to a remuck bay, with the back slashed to a height suitable for an LHD/truck loading operation. Trucks with nominal 50- tonne payload will haul the ore all the way to the primary crusher in the processing plant on surface.

GROUND CONTROL

Current support in permanent openings includes #6 grouted rebar, 1.8 m long and 1.2 m x 1.2 m spacing in back and walls with heavy (#6) gauge mesh. Requirements to escalate support are planned to include 50-100 mm of shotcrete where poor or faulted rock mass conditions are encountered.

Support in the Cut and Fill stopes will be accomplished with splitsets, 1.8 m long at 1.2 m x 1.2 m spacing in back with light (#9) gauge mesh.

BACKFILL CYCLE

The primary backfill material will be the waste rock produced from underground development. If necessary, supplemental material will be sourced from the open pit waste dump.

Rockfill will be delivered to the stope using a dedicated fleet of 30-tonne trucks with ejector box. A rammer will push up the rock "tight" to the back, leaving only enough space for the swell of the ore blasted in the next cut above.

A cement binder will be added to the fill in the sill cut. A slurry tank will be set up in the vicinity of the stope, equipped with a mixer and a pump. The pump will deliver the slurry to the stope via a pipeline. The pipeline will fork into two or three perforated branch lines strung to the back of the stope. The slurry will spray and percolate the rockfill. Cement slurry will be placed at a ratio of five parts cement to 95 parts (w/w%) or 5% by weight.

19.3.7 UNDERGROUND MINE VENTILATION

The underground mine ventilation circuit design is based on the use of a new main ventilation raise (4.5 m diameter raisebore) from 235 Level to surface. This raise will be used to intake fresh air into the mine with the exhaust returning up the new ramp. The main ventilation raise will be equipped with a propane heater 4.1 MW (14.0M btu) which is adequate for use in the winter months to preheat fresh air to above freezing. The primary escape way will be the new ramp and the secondary egress will be in the new ventilation raise, which will be equipped with ladders and landings.

Table 19.20 summarizes the maximum ventilation requirements for production and development needs for diesel-powered equipment.

Table 19.20 Equipment Fleet and Ventilation Requirements

Equipment No.	Description	Hp	Kw	Utilization Factor	Ventilation Diesel Hp	Ventilation Air Required	
						Cfm	m^3/s
UG-J1	2 boom jumbo	148	110	25%	37	3,699	1.7
UG-J2	2 boom jumbo	148	110	25%	37	3,699	1.7
UG-J3	1 boom jumbo	74	55	25%	18	1,849	0.9
UG-L1	8 yd Scoop	327	243	70%	229	22,878	10.8
UG-L2	8 yd Scoop	327	243	70%	229	22,878	10.8
UG-L3	6 yd Scoop	296	220	70%	207	20,713	9.8
UG-L4	6 yd Scoop	296	220	70%	207	20,713	9.8
UG-L5	3 yd Scoop	202	150	70%	141	14,122	6.7
UG-H1	Haul Truck 50-Tonne	525	390	70%	367	36,718	17.3
UG-H2	Haul Truck 50-Tonne	525	390	70%	367	36,718	17.3
UG-H3	Haul Truck 30-Tonne	389	289	70%	272	27,209	12.8
UG-H4	Haul Truck 30-Tonne	389	289	70%	272	27,209	12.8
UG-S1	Scissorlift	174	129	70%	121	12,145	5.7
UG-S2	Scissorlift	174	129	70%	121	12,145	5.7
UG-S3	Scissorlift	174	129	70%	121	12,145	5.7
UG-U1	Grader	155	116	70%	109	10,850	5.1
UG-U2	Anfo Loader	174	129	70%	121	12,145	5.7
UG-U3	Minecat	99	74	70%	69	6,930	3.3
UG-U4	Minecat	100	75	70%	70	7,000	3.3
UG-U5	Toyota	128	96	70%	90	8,960	4.2
UG-U6	Toyota	129	96.75	70%	90	9,030	4.3
UG-U7	Mancarrier	174	129	70%	121	12,145	5.7
UG-U8	Fuel Truck	174	129	70%	121	12,145	5.7
UG-U9	Boom truck	174	129	70%	121	12,145	5.7
UG-U10	Tractor-Minecat	99	74	70%	69	6,930	3.3
UG-U11	Tractor-Minecat	99	74	70%	69	6,930	3.3
UG-U12	Tractor-JD	99	74	70%	69	6,930	3.3
UG-U13	UPC Shifters	30	23	70%	21	2,100	1.0
UG-U14	UPC Shifters	30	23	70%	21	2,100	1.0
					Totals	391,183	184.6

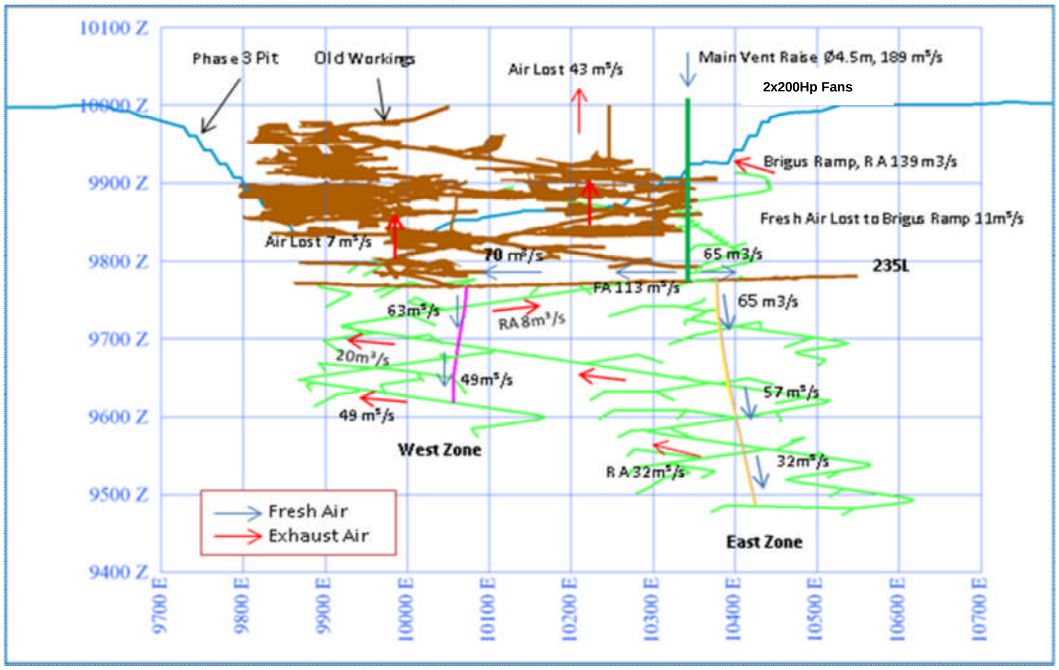
The primary airflow is down the main ventilation raise to 235 Level where a series of air doors and auxiliary ventilation fans direct the fresh air to the operating development and production faces. As sections of the mine are phased out and access is no longer required, bulkheads will be installed in strategic areas to divert air flows to operating areas from the ventilation circuit and thereby significantly reducing the future ventilation requirements for these areas.

The west portion of the mine will receive fresh air supplied by a distribution system on the 235 Level carrying fresh air to the west vent raise system.

The ventilation requirements as idenitified in Table 19.20 are based on 0.06 m^3/s/kW, which meets the Ontario regulations. The new main ventilation raise will be equipped with two new 200 hp fans to deliver the necessary air volume. Similar fans will be required for the internal east and west vent raises as well as the new internal exhaust booster fan.

Figure 19.16 shows flow rates and distribution. Smaller stope fans and vent duct are used to direct the ventilation air to the working faces. The return airflow is directed up the access ramps to surface.

Figure 19.16 Ventilation Air Flow Schematic



19.3.8 UNDERGROUND MINE EQUIPMENT

MOBILE EQUIPMENT

The fully trackless mining operation below the 235 Level requires the mobile equipment fleet, as listed in Table 19.21, for ramp and level development and production from MCF stopes. All equipment will be purchased new, and for the LOM for about eight years, no future purchases will be necessary to replace any unit.

Table 19.21 List of Mobile Equipment

Quantity	Description	Hp
2	2 boom jumbo	148
1	1 boom jumbo	74
2	8 yd Scoop	327
2	6 yd Scoop	296
1	3 yd Scoop	202
2	Haul Truck 50-Tonne	525
2	Haul Truck 30-Tonne	389
3	Scissorlift	174
1	Grader	155
1	Anfo Loader	174
2	Minecat	99
2	Toyota	128
1	Mancarrier	174
1	Fuel Truck	174
1	Boom truck	174
2	Tractor-Minecat	99
1	Tractor-JD	99
2	UPC Shifters	30

STAFFING

The Black Fox Project will require highly trained underground miners, support staff, diesel mechanics, and a technical management group that is familiar with mechanized cut and fill mining techniques. The project is located in an active open pit and underground mining area where such individuals with the required training and education are readily available. Demand for such individuals, in the mining field, has dramatically increased with a subsequent substantial increase in wages and benefits that will be paid to attract and retain these experienced individuals. An on-the-job training program has been designed and is being implemented to ensure that properly trained and experienced miners are available for the mine operation.

A rigorous safety training program has been implemented to create a continuous improvement program required for a modern mining operation. Critical safe working procedures have been established and are enforced.

Many of the salaried level positions are shared between the open pit and underground operations to achieve greater efficiencies in the Black Fox operations. These positions include engineering and geology staff, environmental, human resources and safety.

MANPOWER REQUIREMENTS

The mine personnel required for the underground operations consists of the crews and staff listed in Table 19.22. These requirements ramp up to a maximum total of 71 persons at work in July 2011 – when production reaches a steady state.

Table 19.22 Mine Personnel and Crews

Personnel Per Shift	Max. At work
Underground Management	
Mine Manager	0.5
Mine Superintendent	1
Mine Supervisor	4
Safety Coordinator	1
Mechanical Supervisor	1
UG Mgmt	8
Development Personnel	
Jumbo Miner	4
Bolters	8
Loader	4
Development Personnel	16
Production Personnel	
Truck Operator	8
Scoop Operator	4
Jumbo Miner	4
Utility Labourer	4
UG Production	20
Maintenance Personnel	
Lead Mechanic	2
Mechanic	4
Lead Electrician	2
Electrician	2
Construction Leader	2
Construction Miner	4
Grader Operator	1
	table continues…
Dry Man	1
UG Maintenance	18
Engineering Personnel	
Engineering Coordinator	1
Mine Designer	1
Mine Planner	1
Mine Technologist	2
Chief Surveyor	1
U/G Surveyor	1

Clerks	2
Engineering Personnel	9
Geology Personnel	
Senior Geologist	0
Beat Geologist	0
Geology Personnel	0
Summary	
Total Management	8
Total Development	16
Total Production	20
Total Maintenance	18
Total Engineering	9
Total Geology	0
Total Personnel	71

19.3.9 Support Services

Supplies and Debris Handling

Mining supplies, rockbolts, screens, pipes, vent ducts, etc., will be brought down to the mining areas using utility vehicles. Unused excavations, such as remuck bays, diamond drill stations, abandoned development headings, etc. if required will be converted to a laydown/storage bay. The same vehicles will backhaul lunchroom refuse and other debris to surface.

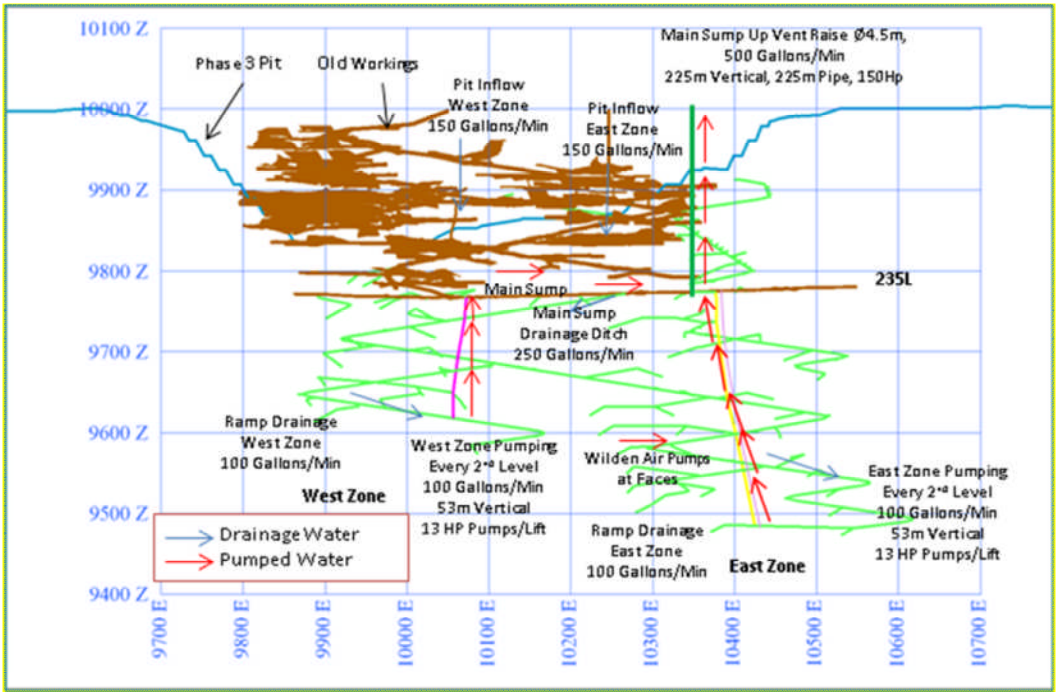
Compressed Air

Compressed air will be required in the development and production headings to operate the handheld jackleg and stoper drills required for drilling holes for ground control bolts, the Alimak raise contractors and utility requirements throughout the mine. The existing compressor equipment will be used to supply the compressed air requirements for the initial phase of the reopening of the Black Fox underground mine. Upon completion of the new fresh air raise and utilities services buildings a temporary compressor will be used to supply underground while the two existing units are relocated.

Water Management

Water management for the Black Fox underground mine was designed to handle approximately 815 m^3/day (124 gpm) on average of excess ground water. Most of the new sump and pump design capacity (located on 235 Level) is due to the excess water reporting from the ultimate open pit and will be designed to handle up to 3260 m^3/day (500 gpm). The underground water system upgrades have been phased with the open pit expansion schedule. The underground headings will have small pumps that pump water to the intermediate sumps located in the new east and west haulage ramps. The intermediate sumps will subsequently pump water to 235 Level main sump. Excess water from the mine will require total pumping capacity of 150 hp. Pumps will feed twin 10.2 cm (4 in) water lines installed in the new Main Vent Raise. Normal operating conditions will require the use of one pump and line at a time. The twin system allows for routine maintenance and using both pumps as required during a 1:25 year storm event on the surface. The underground excess water will report to surface water distribution system to be treated and discharged. Refer to Figure 19.17

Figure 19.17 Dewatering Schematic



Electrical Systems

Three new feeders are planned to be installed (one, 5 kV 3C-350 MCM riser teck cable,two, 5 kV 3C-4/0 riser teck) in the new Main Vent Raise and used to supply 5kV-600V transformer substations.

Electrical Power Consumption

Electrical power is being supplied to the mine site from the Hydro One network. Peak power demand is currently 1.5 MW resulting in a peak energy consumption of 950 MWh per month (figures are approximate). From information supplied by Brigus engineering personnel the site is limited to drawing a maximum of 10 MW from Hydro One. There is sufficient line capacity to meet projected power requirements in order to support current expansion plans.

COMMUNICATIONS

Underground communications will be an important part of managing the safe development and production requirements of the mine operation. The mine currently has hard line underground mine phones located at strategic locations throughout the mine. As the mine expands the mine phone system will be expanded. New phones will be installed at the new permanent underground ventilation and electrical facilities, underground shop and refuge stations as required. The existing leaky feeder communication system will also be upgraded as required to allow better communication within the mine. Offsite communications for phones and data is a supply and service contract provided by a local telecommunications provider.

UNDERGROUND GARAGE

A small garage will be constructed underground on 400 Level (elevation 9600) for running repairs. Major repairs and scheduled preventive maintenance work will be performed in the main shop on surface.

EQUIPMENT MAINTENANCE AND SUPPORT

The surface shop will be used to perform most of the maintenance requirements on the underground equipment fleet. Routine tire repairs, oil changes and preventive maintenance procedures will be performed at this facility. The facility will be equipped with an overhead crane, compressed air, lighting, welding equipment, equipment wash down area, and a small secure warehouse area.

HAULAGE ROUTE MAINTENANCE

Haulage routes both surface and underground are maintained on a regular basis to maintain optimal haul route performance. A CAT-Toromont 135H performs this function underground and a CAT-Toromont 16H performs this on surface.

EXPLOSIVES STORAGE

No surface explosive storage facilities are planned. Explosives will be delivered by local suppliers direct to site where they will be transferred to a mine utility vehicle and delivered directly to underground powder and cap magazines for storage.

19.3.10 HEALTH AND SAFETY CONSIDERATIONS

REFUGE STATIONS

Refuge stations will be installed in accordance with provincial and national requirements for underground mines. The stations will be constructed in muckbays that are no longer needed at strategic locations around the mine. There will be one in the east side production area and one in the west side. Each station will be supplied with air (from the compressed air line), water, and communication equipment including mine phones and leaky feeder connections.

MINE RESCUE

Brigus has prepared a Memorandum of Understanding with the local MASHA authority that establishes the responsibilities for the underground mine. The number of miners underground at any one time will require that 15 trained mine rescue personnel be onsite. The mine will use MASHA supplied equipment and training as part of the workplace safety insurance board premiums that are paid by the mine.

In the event of a mine incident that requires the immediate evacuation of the mine stench gas (methyl mercaptine) will be introduced to the compressed air system as well as the main ventilation fans on the surface. Miners will be trained to make their way quickly to the closest refuge station along well marked escape routes.

EMERGENCY RESPONSE

The Black Fox Mine Emergency Response Plan (ERP) establishes procedures for responding to emergencies at the Brigus property that may jeopardize the health or safety of personnel, the natural environment, or property. The ERP addresses potential emergencies identified through site risk assessments.

The ERP has been developed by Brigus personnel, and is updated as required by changes in activities, legislation or policies. At a minimum, the procedures related to spills are reviewed annually and updated as required by the Metal Mining Effluent Regulations (MMER).

The ERP is designed to promote effective response with minimal confusion and disruption of activities at the Black Fox Mine site. It is constructed so that it can be initiated and operated by on duty personnel until such time as additional personnel support is required. The plan is also intended to define responsibilities and to establish priorities for essential activities.

SECOND EGRESS

The Main Vent Raise will be outfitted with a number of different services including an escape manway which will serve as the principal secondary means of egress. Use of secondary egress is governed by the Mine Emergency Response Plan.

19.3.11 *Surface Facilities for Underground Mine*

Office and Dry

Lockers, change room and shower (dry) faculties for 200 personnel are available and include facilities for both a women's and visitors dry. Office areas are made up of seven fully serviced modular buildings. These are configured to support a dry, main office, underground engineering & geology, pit operations, pit geology, surface pit supervision and exploration.

Maintenance Shop, Warehouse and Cold Storage

The surface maintenance shop will be used to perform most of the maintenance requirements on the entire equipment fleet. Routine tire repairs, oil changes and preventive maintenance procedures will be performed at this facility. The facility will be equipped with an overhead crane, compressed air, lighting, welding equipment, equipment wash down area, and a small secure warehouse area. Sufficient outside and covered cold storage areas exist for regular mine consumables

Compressed Air Plant

The current compressed air plant will be relocated to the Main Vent Raise once it is complete. The final configuration will include dual Sullair 25-200 compressors feeding a 1000 gallon air receiver located on 235 Level via a 8 inch line. Distribution from 235 Level will be via 4 inch air lines.

Backfill

Surface backfill facilities will not be required to support the cemented backfilling of underground cut and fill stopes.

19.4 Tailings Storage Facility at Stock Mill

19.4.1 *Tailings Management*

The ore from Black Fox site will be processed from the Stock Mine mill.

On October 28, 2010 Wardrop conducted a site visit at Black Fox Milling site to observe the existing tailings facility and review operational practices. In addition, Wardrop reviewed construction as-built documents for the Phase 4 tailings management area and Phase 5 water management area to assess conformance with design parameters and evaluate best engineering practices.

Site Observations

Wardrop was present at the above referenced site on October 28, 2010 and noted the following conditions;

- *Observations of the perimeter dam showed no lateral displacement or distress of slopes.*
- *Water seepage through the perimeter dam was not observed.*
- *Ponding of water outside of the perimeter dam was not observed.*
- *Operations surface water and erosion control practices appear to be effective.*
- *Tailings decant water appears to be managed satisfactory by site personnel. No presence of water beyond the facility perimeter was observed*
- *The Phase 5 water management pump station has been operating without problems and appeared to be functioned as designed.*
- *Site operation personnel noted no operational concerns.*

Review of As-Built Drawings

Wardrop reviewed the Phase 4 Tailings Management Area and Phase 5 Water management as-built reports prepared by AMEC (Apollo Gold Milling Site , Phase 4 Tailings Management Area Perimeter Dam Upgrade, As Built Report, March 9, 2010, Apollo Gold Phase 5 Water Management Pond As-Built Report, February 2010) to assess compliance with design parameters and best industry practices.

The Phase 4 Tailings Management Area and Phase 5 Water Management Pond were constructed and operational in early 2010. Review of the above reference as-built drawings shows that the tailings management structures constructed in substantial conformance with design parameters. Furthermore design and construction have been accomplished with current best engineering practices for such structures.

Based upon site observations, discussion with site personnel and review of as-built drawings the tailings management facility presently provides adequate containment for tailings as designed.

19.5 MINE SITE WATER MANAGEMENT

AMEC carried out an overall site water management study for the Black Fox site, with a Holding Pond as the water management centre. The following sections describe in details of the concept and design features.

DESIGN CRITERIA AND CONSIDERATIONS

1. The Holding Pond will serve as the central water management facility for the entire site. Contaminated water from the Open Pit, Underground Operations and Dirty Waste Rock Stockpile will be pumped to the Holding Pond and from there sent to treatment plant for treatment. The Holding Pond was designed to provide sufficient storage capacity for a minimum 5-day retention time for the 1:25-year rainfall storm volume Environmental Design Flood (EDF). The pond water will be allowed to spill for flood events exceeding the EDF.

2. Surface runoff from the Open Pit will be continuously pumped to the Holding Pond with a capacity that prevents substantial accumulation of water in the pit (leading to production disruption) for hydrologic events up to 1:25 year event.

3. Runoff from the dirty waste rock stockpile will be collected for temporary storage and pumped to the Holding Pond for treatment. The collection system was designed for the 1:25 year event. The pump was sized to effectively draw down the collection pond to the minimum operating level within 5 days after the design event.

4. No runoff collection or treatment system is required for the clean waste rock stockpile and the overburden stockpile. Surface runoff shall be discharged to the environment with sediment control measures in place, if required.

The overall site water management schematic and water balance under average year runoff condition is shown on Figure 19.15.

SUMMARY OF DESIGN PARAMETERS

The principal design features are summarized below.

OPEN PIT

The required pumping capacity was designed at 33,020 m^3/d (about 6,060 gpm) to be able to pump the runoff resulting from a 1:25 year 24hr storm together with the seepage from the underground workings. The inflow design parameters used to deriving the required pumping rate are summarized in Table 19.23.

Table 19.23 Open Pit Pump Capacity Design Parameters

Description	Parameter
Total footprint area for the Open Pit	33.0 ha
The rainfall depth corresponding to the design event (1 in 25 year 24 hr rainfall)	98.3 mm
Total runoff volume corresponding to the design event	32,440 m^3
Seepage from Open Pit and underground workings to the design duration	580 m^3
Total flow volume corresponding to the design event	33,020 m^3

DIRTY WASTE ROCK STOCKPILE

The seepage and runoff emerging along the perimeter toe of the Dirty Waste Rock Stockpile will be isolated from the clean runoff from the surrounding areas by a containment berm and will be directed to a pump sump at the low lying area at the northwest corner of the stockpile. The collected water from the sump will be pumped to the Holding Pond. The storage capacity of the pump sump is designed for 1 in 25 year 24 hr rainfall event as in Table 19.24.

Table 19.24 Dirty Waste Rock Stockpile Pump Sump Design Parameters

Description	Parameter
Total footprint area for the Dirty Waste Rock Stockpile (within the diversion berm)	55.0 ha
Runoff coefficient used for the waste rock stockpile	0.8
The rainfall depth corresponding to the design event (1 in 25 year 24 hr rainfall)	98.3 mm
Total runoff volume corresponding to the design event (without pumping)	44,300 m^3
Storage requirement corresponding to the design event (with 700gpm constant pumping rate)	40,500 m^3

The collection pond would be pumped to minimum operation level in about 5 days with the design pumping rate of 3,800 m^3/d (about 700 gpm), under the 1:25 year hydrologic event.

HOLDING POND

Table 19.25 summarizes the inflow design parameters used to deriving the required pumping rate and storage capacity in the Holding Pond.

Table 19.25 Holding Pond Inflow Design Parameters

Description	Annual Volume (m^3/year)	1:25 year 24-hour Volume(m^3/d)
Inflows		
Direct precipitation over the pond surface	109,000	12,800
Seepage from underground workings and open pit	211,000	580
Runoff from open pit	238,000	32,400
Runoff from dirty rock stockpile	278,000	3,800
Outflows		
Evaporation from the pond surface	59,000	negligible
Seepage Losses	Conservatively ignored	negligible
Treatment plant	777,000	5,000

Design parameters for the Holding Pond are summarized below:

- Dead Storage: A volume of 29,000 m^3 is assumed for the sediment containment. T he top elevation of the sediments is at 291.5 m, 0.5 m above the pond bottom.
- Minimum Normal Pond Operating Level: Minimum pond operating level is set at 292.0 m, 1.0 m above the bottom of the pond.
- Normal Operating Volume in the Pond: The normal operation volume of 174,000 m^3 will be required to retain contaminated water under average year runoff condition.
- Maximum Normal Operating Level: The maximum water level for an average runoff year is at elevation 293.5 m.
- Design Pumping Rate: Water from the Holding Pond will be pumped to the treatment plant at following rates:
  - For an average year: 2,130 m^3/d
  - For 1:25 year rainfall event: 5,000 m^3/d
- Emergency Spillway Invert Elevation (operation): Determined by the operating and no spill criteria, this elevation was selected at 293.9 m, providing a total storage capacity of 282,000 m^3.
- Emergency Spillway Configuration: A trapezoidal spillway channel (3H:1V) was designed with a bottom width of 3 m and a longitudinal slope of 0.5%.
- Dam Crest: Dam crest elevation was designed at 294.5 m, provides a 0.3 m freeboard above the maximum water level of 294.2 m in a 1:100 year hydrologic event during operation.

On October 28, 2010 Wardrop conducted a site visit at Black Fox Mine site to observe existing site water management controls and assess operational practices.

In addition Wardrop reviewed construction as-built documents of the transitional settling and holding ponds to assess conformance with design parameters and evaluate best engineering practices.

Site Observations

Wardrop was present at the above referenced site on October 28, 2010 and noted the following conditions;

- *The holding ponds perimeter earthen dams appeared stable with no evidence of lateral displacement, sloughing, cracking or budging at the toe of slope.*
- *Seepage of water through the dam or ponding of water at the toe of slope was not observed.*
- *The area surround the ponds appear to be stable without evidence of ponding and/or boiling due to elevated hydraulic gradient.*
- *Discussions with operational personnel indicated that the ponds operating satisfactorily and have not noted any structural or containment problems.*

Review of As-Built Drawings

Wardrop reviewed the transitional settling and holding ponds as-built report prepared by AMEC (Apollo Gold Milne Water Ponds, Transitional settling Ponds and Holding Pond, 2008/2009). to assess conformance with design parameters and best industry practices.

The transitional settling ponds and holding pond were constructed in late 2008 and complete in 2009. Based upon review of the as-built report it appears that the ponds were constructed in substantial conformance with design parameters and generally in accordance with best engineering practices. However as noted in Table 3.5 of As-Built report, Apollo site instruction BF-09-02, the HDPE liner was eliminated from the holding pond construction. It is unclear why this action was taken by Apollo. Current best engineering practices for such structures include the use of HDPE liners.

20.0 ENVIRONMENTAL CONSIDERATIONS

20.1 ENVIRONMENTAL PERMITTING

The primary enabling environmental permits necessary for development/operation of the mine and mill include:

- Permit to Take Water (PTTW) – for the withdrawal of surface or ground water quantities in excess of 50 m^3/day – including mine underground or open pit dewatering.
- Certificate of Approval/Air (CofA/Air) – for treatment and discharge of emissions to air, including management of dust and noise in emissions.
- Certificate of Approval/Industrial Sewage Works (CofA/ISW) – for collection, treatment, and discharge of wastewaters.
- Approved Closure Plan including posting of financial assurance for closure.

The Black Fox operation is fully permitted and maintains separate PTTW's, CofA/ISW, and CofA/Air for the mine and the mill. The mine and mill were previously under separate ownership and the facilities were permitted separately. The permits for these facilities were acquired with the properties and have been amended as the operations have evolved, maintaining the separate permitting for the sites. Specific details of the permits as relevant to this report are outlined below.

20.2 BLACK FOX MILL

20.2.1 PERMIT TO TAKE WATER

The PTTW (Number 6377-88UJE5) for the Black Fox mill provides for the withdrawal of process water from North Driftwood Creek and from the Stock Mine workings and is valid until January 29, 2019. The PTTW places limits on the quantities drawn from each source by limiting the rate of withdrawal, number of hours of withdrawal per day, and the total number of days in a year the withdrawal is permitted, as detailed in Table 20.1 below. In addition, takings from North Driftwood Creek are limited to no more than 10% of the upstream flow, notwithstanding the limits in Table 20.1. All other process makeup water is reclaimed from the Water Management Area, and no permit is required for this withdrawal.

The permit holder is required to maintain a record of all water takings and maintain a system to measure flow in North Driftwood Creek and to record creek flows on all days of water taking from the creek.

Table 20.1 Permitted Surface and Mine Water Withdrawals by the Black Fox Mill from Surface and Mine Water Sources as per PTTW Number 6377-88UJE5

	Source Name/ Description	Source Type:	Taking Specific Purpose	Taking Major Category	Max Taken per Minute (litres)	Max. No. Of Hrs. Taken per Day	Max. Taken per Day (litres)	Max. No. Days Taken per Year	Zone/ Easting/ Northing
1	North Driftwood Creek	River	Other - Industrial	Industrial	748	24	1,077,000	30	17 518870 5377580
2	Stock Mine Workings	Mine	Other - Industrial	Dewatering	3,000	24	4,320,000	185	17 518570 5377600
3	Stock Mine Workings	Mine	Other - Industrial	Dewatering	1,950	24	2,808,000	365	17 518570 5377600
						Total Taking	8,205,000		

20.2.2 CERTIFICATE OF APPROVAL - AIR

Brigus Gold has a valid Certificate of Approval – Air (Number 8-5029-89-997) for the Black Fox Mill that permits the processing of 1300 tonnes of ore per day through the mill. Brigus Gold has routinely been operating at milling rates in excess of 1300 tonnes per day and has applied for an amendment of the CofA to increase the daily milling rate to 2000 tonnes per day. Production at 2000 tonnes per day is permitted while the amendment is under review. As of the date of this report that application was still under MOE review (Anne Boucher, Brigus Gold, pers. comm.).

The current CofA lists five dust collectors, one wet scrubber, and eight exhaust systems. The CofA requires that all devices and systems be kept in good working order with records of maintenance kept for all components to demonstrate compliance with operating specifications and procedures. No air quality or noise monitoring and no routine reporting are required by the CofA.

20.2.3 CERTIFICATE OF APPROVAL – INDUSTRIAL SEWAGE WORKS

Brigus Gold has a valid Industrial Sewage Works Certificate of Approval (Number 0887-7VQJCU) for the Tailings Management Area (TMA) at the Black Fox Mill. The

CofA is valid for a mill production rate up to and including 2000 t/day and provides for the staged increase in TMA capacity to an ultimate capacity of 3.76 million tonnes of tailings at a dam crest elevation of 276.0 m. The works covered by the CofA include construction and operation of the TMA, the Water Management Pond (WMP), water reclaim system, water treatment works located in the mill, and the discharge of treated tailings wastewater as required.

The wastewater treatment system is located in the mill complex and has a hydraulic capacity of 2,880 m^3/day for the destruction of cyanide and the precipitation of arsenic using the INCO SO_2/Air cyanide destruction process. System capacity is 13.8 kg total cyanide per hour. Ferric sulphate is added in the second reactor for arsenic removal. The treated wastewater is pumped to a polishing pond located south of the mill, and is fitted with equipment for dosing of a flocculant as needed, a silt screen filter, and concrete outfall structure.

At the time of the site visit the system was being operated on a no discharge basis to ensure there was sufficient reclaim water to feed the mill. Management of water levels in the TMA and WMP over the longer term may require discharges and this is provided for within the current CofA.

Monitoring requirements under the CofA include effluent quantity and quality, receiving watercourse (water and sediment quality, benthic invertebrates, and fish), and groundwater (level and quality) with quarterly and annual reporting.

20.2.4 CLOSURE PLAN

Brigus Gold has an approved closure plan for the Black Fox Mill site. Closure involves:

- capping of shaft and vent raise
- engineering analysis of pillar(s) stability
- removal and disposal of buildings and infrastructure, demolish foundations to grade, cover disturbed areas with overburden and re-vegetate
- removal and disposal of machinery, equipment, storage tanks
- decommission and revegetate roads
- assess and remediate any areas of hydrocarbon contamination
- breach polishing pond
- stabilize impoundment structures
- site grading to restore site drainage.

The closure plan includes provision for five years water treatment following the completion of mining activities. The expected post-mining period of treatment was reduced from an original estimate of 85 years that was submitted in 2009 following a 28 week humidity cell test program that demonstrated the longer period of treatment would not be required (AMEC 2010a). This reduction of the treatment period was approved by the Ontario Ministry of Northern Development Mines and Forestry in October 2010. Reduction of the post-mining treatment period also reduced the estimated closure cost, resulting in a reduction in the amount of Financial Assurance (FA) required. The estimated post-closure treatment cost was reduced to $583,500, resulting in a $2,991,500 reduction in the FA assurance required. The net FA of $5,039,940 has been posted and the previously posted amount of $2,991,500 transferred to the FA for the mine site.

20.3 BLACK FOX MINE

20.3.1 *PERMIT TO TAKE WATER*

The current PTTW (Number 8803-886RFK) provides for the withdrawal of water for dewatering of the Black Fox Pit as detailed in Table 20.2 below. Brigus Gold has applied for an amendment to the PTTW to provide for withdrawals of mine water from the underground but that application remained under review as of the date of this report.

Required monitoring includes quarterly monitoring of groundwater levels at 11 locations, bi-monthly water level monitoring on the nearby North Pond (Beaver Pond), Middle Lake, and Leach Lake, and continuous water level monitoring on Froome Lake. The permitted water taking must not have a negative impact on any adequate water supplies that existed prior to the taking and the remedy of any such negative impacts is the responsibility of the permit holder.

Table 20.2 Permitted Water Withdrawals by the Black Fox Pit as per PTTW Number 8803-886RFK

	Source Name/ Description	Source: Type:	Taking Specific Purpose	Taking Major Category	Max Taken per Minute (litres)	Max. No. Of Hrs. Taken per Day	Max. Taken per Day (litres)	Max. No. Days Taken per Year	Zone/ Easting/ Northing
1	Black Fox Pit	Pond Pit	Pits and Quarries	Dewatering	9,392	24	13,525,000	365	17 548894 5375939
						Total Taking	13,525,000		

20.3.2 C OF A/AIR

Brigus Gold has a valid CofA/Air (Number 8439-7Y4RUE, dated June 28, 2010) for the Black Fox Mine that applies to a facility production Limit of up to 2000 tonnes of ore per day (combined production from open pit and underground).

The terms and conditions of the CofA are based in part on the *Emission Summary and Dispersion Modelling (ESDM) Report* (AMECb, November 2009), the project *Acoustic Assessment Report* (HGC Engineering, November 2009), and the *Best Management Practices Plan* (AMECc, December 2009).

CofA compliance is performance based, with the mine required to meet specific performance limits, to document all equipment installation, maintenance, and modifications, and to provide annual written confirmation of compliance, but with no specific monitoring detailed in the CofA.

Specific noise control measures required as stipulated in Schedule "B" the CofA include:

- One acoustic silencer for each of the ventilation inlets identified as noise Sources NS-01 and NS-04, capable of providing the following values of Insertion-Loss in 1/1 octave frequency bands:

Centre Frequency (Hertz)	63	125	250	500	2,000	2,000	4,000	8,000
Insertion – loss (decibel)	2	5	13	17	21	22	13	15

- One acoustic silencer for each of the ventilation fan casings identified as noise Sources NS-02, NS-03, NS-05, and NS-06, capable of providing the following values of Insertion-Loss in 1/1 octave frequency bands:

Centre Frequency (Hertz)	63	125	250	500	2,000	2,000	4,000	8,000
Insertion – loss (decibel)	--	--	5	10	15	20	22	25

Brigus Gold was in the process of sourcing and installing these silencers as of the November, 2010, site visit (Anne Boucher, Brigus Gold, pers. comm.).

20.3.3 C OF A – INDUSTRIAL SEWAGE WORKS

Certificate of Approval – Industrial Sewage Works (Number 9423-7QGSE2 dated October 15, 2010) for the Black Fox Mine covers the construction and operation of:

- the runoff collection system for the Potential Metal Leaching (PML) (formerly called the *Dirty Waste Rock Pile*)
- the runoff collection system to gather runoff from the construction waste rock and overburden stockpiles

- a surge pond and mine water treatment pond system
- an effluent discharge pipeline from the treatment pond system to the Pike River
- the sanitary subsurface sewage disposal system for the mine site
- the sanitary subsurface sewage disposal system for the truck maintenance shop at the mine site.

The mine water contains suspended solids, ammonia (residual from blasting), arsenic, and trace amounts of oil and grease (from vehicle operations). Runoff from the Potentially Metal Leaching (PML) (formerly termed the *Dirty Waste Rock Pile*), where carbonate ore zone waste rock is segregated and stockpiled, also is expected to have elevated arsenic concentrations such that mine water and dirty waste runoff are expected to require treatment for arsenic in addition to settling of suspended solids and with a provision for aeration of the polishing pond for ammonia reduction as required.

Dirty waste runoff reports to the adjacent collection pond and is pumped to the surge pond as needed. Pit water and underground mine water are pumped to the surface surge pond. Water from the surge pond is pumped to the treatment pond system (settling and polishing ponds) for arsenic treatment by ferric sulphate addition, removal of suspended solids by settling, pH adjustment with the addition of sulphuric acid solution, and aeration of the clarified water as needed for ammonia reduction prior to discharge. The average design discharge rate is 1,230 m^3/day and maximum is approximately 2,700 m^3/day during peak precipitation events.

The treated effluent discharge is subject to the monitoring requirements of the CoA and the Metal Mining Effluent Regulations.

20.3.4 CLOSURE PLAN

Brigus Gold has an approved Closure Plan for the Black Fox Mine site, with the closure cost estimated at $15,090,110. FA has been provided/pledged to MNDM in the amount and form of:

Currently held FA	$7,428,830	Irrevocable Letter of Credit
Transfer of Mill FA credit	$2,991,500	Irrevocable Letter of Credit
Instalment 1	$2,334,890	Irrevocable Letter of Credit – posted September 1, 2010
Instalment 2	$2,334,890	Irrevocable Letter of Credit – due December 31, 2010
	$15,090,110	

The instalment schedule is based on the planned September 1, 2010 start date of the Phase 2 expansion of the PML waste rock stockpile footprint.

Closure involves:

- capping of vent raises
- backfilling and plugging of the portal
- placement of a boulder fence around the pit
- batch treatment of pit water
- removal of dewatering infrastructure
- engineering analysis of pillar(s) stability
- removal and disposal of buildings and infrastructure, demolish foundations to grade, cover disturbed areas with overburden and re-vegetate
- removal and disposal of machinery, equipment, storage tanks
- decommission and revegetate roads
- assess and remediate any areas of hydrocarbon contamination
- contour and revegetate waste rock and remaining overburden stockpiles
- stabilize impoundment structures
- site grading to restore site drainage
- long term PML waste rock stockpile runoff treatment

The primary closure cost is associated with long term treatment of arsenic in runoff from the PML waste rock stockpile. The most current kinetic geochemical testing indicates that treatment will be required for 60 years after the completion of mining. This is as good an estimate as can be made at this time. Monitoring of the actual PML waste stockpile runoff quality during continuing operations will further inform this estimate and the projected duration of runoff treatment can be expected to be adjusted several more times before the end of mining operations.

The estimated cost of treatment over the 60 year period is $12,155,000, based on a Net Present Value calculation using a 3% interest rate (AMEC 2010). This also means that the mine site is not expected to completely reach closed mine status for 60 years following the completion of mining. The site will be maintained in a state of inactivity during the period of continuing water treatment, which also means the site remains under the coverage and obligations of the Metal Mining Effluent Regulations (MMER). The CofA/ISW for the mine site will also need to be maintained through the period of treatment, including continuation of the necessary effluent and receiving environmental monitoring. Monitoring is required to satisfy both the CofA and the MMER reporting requirements.

20.4 Environmental Liabilities

Neither the mill nor the mine has any known legacy environmental liabilities. However, two studies of historical effects remain to be completed and these may indicate specific remedial actions to be undertaken in the future.

The first study is an assessment of the effects of the historical tailings wastewater discharges to Reid Creek and is required as a condition of the current CoA/ISW. As part of the TMA redevelopment at the Black Fox Mill, tailings wastewater discharge routing was changed. The TMA discharge historically reported to Reid Creek, adjacent to the TMA, whereas any future discharge that may be required would flow to North Driftwood Creek. The study is to determine the impact of the historical discharge on the creek (fish, benthic invertebrates, sediment, and water quality) and to provide baseline data to support future impact assessment. There is no specific requirement within the CofA to remediate any adverse effects so at this stage no remedial actions are expected to be necessary. The field investigation was in the process of completion at the time of the November 2010 site visit with reporting to follow in 2011.

The second study requirement at the Black Fox Mill site is to conduct a Storm Water Control Study to determine if there is any need to manage storm water runoff. No runoff controls are currently in place on the mill site and this study will determine if any specific controls are necessary. The study is to be conducted in accordance with the P*rotocol for Conducting a Storm Water Control Study* (MOE 1994). The study had been commissioned but had not been completed as of the November , 2010 site visit due to the requirement of the protocol to conduct the study during and following specific precipitation conditions, which had not occurred up to that time. Ore has historically been stockpiled on site and this practice continues. Arsenic leaching from the ore has the potential to contaminate runoff from the mine site. The significance of any such contamination will be determined in this storm water study.

21.0 MARKETS AND CONTRACTS

21.1 MARKETS

Markets for doré are readily available. Gold markets are mature, global markets with reputable smelters and refiners located throughout the world. Demand is presently high with prices for gold showing remarkable increases during recent times. The 36-month average London PM gold price fix through 2010 is US$1018/oz.

21.2 CONTRACTS

The Black Fox Project has signed contracts which will be directly associated with operations. These contracts, shown below, are also modeled in the economic analysis.

- Gold stream agreement:
  - Sandstorm Resources Ltd. has agreed to purchase 12% of the Gold production from the Black Fox Mine beginning in January 2011 and 10% of future production from the Black Fox Extension covering a portion of the adjoining Pike River property. Sandstorm will make an upfront payment of US$56.3 million, which Brigus Gold will use to unwind the balance of its forward gold sales contracts terminating the obligation to deliver 99,409 ounces from October 2011 to March 2013.
  - Sandstorm will also pay Brigus Gold ongoing per ounce payments of US$500 subject to an inflationary adjustment beginning in 2013, not to exceed 2% per annum.
  - Brigus Gold has the option, for a 24 month period, to reduce the Goldstream to 6% of production from the Black Fox Mine and 4.5% from the black Fox Extension for a payment of US$36.6 million.
- Ore transport agreement
  - Global Environmental Solutions entered into an agreement to load and transport ore to the Stock Mill. Load and hauling to the Stock Mill is quoted at Cdn$0.33/t for loading and Cdn$3.69/t hauling plus a fuel surcharge.
- Caterpillar and Sandvik equipment lease
  - Agreements have been received and reviewed by Wardrop in the preparation of this technical report.

- Transport of gold bars to the refinery.
  - Brinks Canada Limited entered into an agreement to transport gold bars from the Stock Mill to the refinery (Johnson Matthey Limited) February 2009. The armoured transportation rates are Cdn$850 per call for the first $500,000, plus Cdn$0.65 per $1,000 from $500,001 to $2,000,000, plus Cdn$0.47 per $1,000 from $2,000,001 to $4,000,000. The rates are subject to an annual 4% increase.
- Refining of doré.
  - Smelting and refining of doré will be performed under the terms of an agreement with Johnson Matthey. Refining will be at a cost of Cdn$0.55 per total ounce received. Payable gold will be 99.91%.

22.0 TAXES

Income tax was not considered in this report. Provincial and government sales taxes are refundable for mining operations and therefore are not included in this analysis. Import duties and other fees associated with capital items are included in the capital cost estimate.

There are no royalty obligations associated with the current Black Fox reserves and resources.

22.1 MUNICIPAL PROPERTY TAXES

Brigus Gold is obliged to may local municipal property taxes to the municipality of Matheson, in the province of Ontario. Current 2010 taxes of approximately $141,000 per annum are expected to increase with current reassessments.

23.0 CAPITAL AND OPERATING COST ESTIMATES

The LOM capital costs totalling US$74.8 million are summarized in Table 23.1. Details supporting the capital costs are discussed in this section. Capital costs for 2011 are US$34.9 million. Ongoing capital costs of $39.9 million cover the remaining mine life. Capital cost estimates are in 2011 US constant dollar terms.

Table 23.1 Black Fox Capital Cost Summary (US$000s)

Description	2011 Capital (US$000s)	Ongoing Capital (US$000s)	LOM Total (US$000s)
Open Pit Mine	$6,233.1	$5,514.3	$11,747.4
Underground Mine	$28,156.9	$33,980.3	$62,137.2
Environment	$91.4	$114.3	$205.7
Mill	$441.9	$285.7	$727.6
Total LOM Capital	**$34,923.3**	**$39,894.6**	**$74,817.9**

Capital costs shown include delivery and erection/assembly charges, where applicable. Tax on equipment is refunded by the government, so a tax provision is not included in the estimate.

23.1.1 OPEN PIT MINE CAPITAL

The open pit mine capital costs covers the open pit mine equipment, development of the surface pit ramp, portal construction and additional work are shown below in Table 23.2.

Open pit related capital costs also include the removal of glacial till material prior to the drilling and blasting of overburden. This work will be performed by Brigus crews in two phases. Phase 2 is completed end Q1 2011 and Phase 3 work will commence Q3 2012 and be completed in Q2 2013.

Table 23.2 Black Fox Open Pit Mine Capital (US$000s)

Description	2011 Capital (US$000s)	Ongoing Capital (US$000s)	LOM Total (US$000s)
Equipment			
777F Truck	$1,371.4	$0	$1,371.4
992K Loader	$781.0	$0	$781.0
Subtotal	**$2,152.4**	**$0**	**$2,152.4**
Development			
Surface Pit Ramp	$1,739.8	$0	$1,739.8
Portal Construction	$238.1	$0	$238.1
Additional Work	$551.5	$0	$551.5
Subtotal	**$2,529.4**	**$0**	**$2,529.4**
Glacial Till Stripping			
Phase 2	$1,551.3	$0	$1,551.3
Phase 3	$0	$5,514.3	$5,514.3
Subtotal	**$1,551.3**	**$5,514.3**	**$7,065.6**
Total Open Pit Mine Capital	$6,233.1	$5,514.3	$11,747.4

23.1.2 Underground Mine Capital

The underground mine budgeted capital costs cover equipment, development and underground infrastructure are shown below in Table 23.3.

Underground development includes development of ramps, drifts and raises required to reach the production stopes and allow sufficient lead-time to develop the production stopes to supply a continuous source of ore to the mill.

Table 23.3 Black Fox U/G Mine Capital (US$000s)

Description	2011 Capital (US$000s)	Ongoing Capital (US$000s)	LOM Total (US$000s)
Equipment Leases			
DD310-40 Boom Jumbo	$484.7	$0	$484.7
2 – 6yd LHD 410	$1,199.6	$0	$1,199.6
DS311-C Bolter	$780.3	$0	$780.3
2 – 30 tonne TH430 Trucks	$732.7	$0	$732.7
Subtotal	$3,197.4	$0	$3,197.4
Development			
Drifts	$10,444.0	$28,499.8	$38,943.8
Ramps	$11,553.0	$3,196.5	$14,749.5
Raises	$1,295.9	$1,331.7	$2,627.6
Subtotal	$23,292.9	$33,027.9	$56,320.8
Underground Infrastructure			
Mine Shop Building	$29.5	$0	$29.5
Core Shed	$7.1	$0	$7.1
Mine Shop Complex	$104.8	$0	$104.8
Vent Shaft/Underground Utilities	$14.0	$0	$14.0
Pre-production Costs	$19.4	$0	$19.4
General Infrastructure	$147.6	$0	$147.6
Cement Backfill Plant	$285.7	$0	$285.7
Satstat Fueling Station	$76.2	$0	$76.2
Sump Expansion	$238.1	$95.2	$333.3
Refuge Stations	$190.5	$0	$190.5
Communications System	$238.1	$0	$238.1
235 Shop Tooling	$95.2	$0	$95.2
Ventilation Fans	$142.9	$0	$142.9
Air Receiver	$77.5	$0	$77.5
Sustaining	$0	$857.1	$857.1
Subtotal	$1,666.6	$952.4	$2,619.0
Total Underground Mine Capital	**$28,156.9**	**$33,980.3**	**$62,137.2**

23.1.3 Environmental Capital

The mine capital cost to maintain environmental compliance are shown below in Table 23.4.

Table 23.4 Black Fox Environmental Capital (US$000s)

Description	2011 (US$000s)	Ongoing Capital (US$000s)	LOM Total (US$000s)
Environmental			
Lab Information Mgmt System	$91.4	$0	$91.4
Sustaining	$0	$114.3	$114.3
Total Environmental Capital	**$91.4**	**$114.3**	**$205.7**

23.1.4 Mill Capital

The mill capital cost items required are shown below in Table 23.5

Table 23.5 Black Fox Mill Capital (US$000s)

Description	2011 Capital (US$000s)	Ongoing Capital (US$000s)	LOM Total (US$000s)
Mill			
Sustaining	$0	$285.7	$285.7
Dry Lime	$142.9	$0	$142.9
Bulk Steel	$71.4	$0	$71.4
Magnet	$28.6	$0	$28.6
Cyanide Control System	$71.4	$0	$71.4
Sump Pumps	$32.4	$0	$32.4
Maintenance	$95.2	$0	$95.2
Total Mill Capital	**$441.9**	**$285.7**	**$727.6**

23.2 Operating Costs

The LOM operating costs are summarized in Table 23.6. Details supporting these operating costs are discussed in this section. The Black Fox operating cost mine budget for 2011 through 2017 were used to develop and support these costs. This budget was developed by Brigus, and reviewed by Wardrop. The budget was developed in Canadian dollars and converted into US dollars using a exchange rate of Cdn$1.05:US$1.00.

Operating costs are in 2011 US constant dollar terms.

Table 23.6 Operating Cost Summary

Description	Unit Cost (US$/tonne milled)	Unit Cost (US$/tonne mined/milled)*	Unit Cost (US$/tonne ore mined)	LOM average (US$/tonne milled)	LOM Total (US$000s)
Open Pit Mining			$2.60	$14.50	$90,594
Underground Mining			$56.48	$26.47	$165,391
Ore Handling	$5.31			$5.31	$33,180
Mill	$13.83			$13.83	$86,419
Assay Lab		$0.98		$1.93	$12,033
Site G&A		$2.50		$4.91	$30,696
Total				**$66.94/t**	**$418,314**
				$502/oz Au	

*Mined/milled is the total tonnes of the pit plus underground mined plus the total milled.

23.2.1 LABOUR COSTS

Labour costs are based upon a defined work force. Salaries and hourly rates applied are in accord with current values in the region. All salaried staff and hourly paid tradesmen will work a 40-hour week. Operating personnel will provide continuous coverage.

Pay scales are based on the Black Fox budget and reflect ongoing rates in the Timmins area. A payroll burden of 30.79% is applied to all salaries and wages. In addition, a variable bonus allowance is applied to hourly personnel depending on the activity performed.

Personnel will reside in the neighbouring communities and will be responsible for transportation from their domicile to the place of work. Brigus will not provide transportation, on a normal basis.

23.2.2 OPEN PIT OPERATING COSTS

A summary of the open pit mine operating costs are shown in Table 23.7. Open pit mine operating costs are based on the Black Fox budget costs for 2011 through 2015.

LOM open pit mine operating costs total US$90.6million, or US$2.60/tonne of pit ore/waste mined.

Table 23.7 Open Pit Mine Operating Cost (US$000s)

Description	LOM Unit Cost (US$/t pit ore/waste mined)	LOM Total (US$000s)
Surface G&A	$0.232	$8,094.2
Drilling	$0.333	$11,615.9
Blasting	$0.400	$13,929.6
Loading	$0.347	$12,086.8
Hauling	$0.573	$19,959.0
Roads & Dumps	$0.373	$13,009.6
Maintenance	$0.095	$3,319.8
Pit Dewatering	$0.013	$455.5
Engineering	$0.056	$1,945.1
Geology	$0.177	$6,178.4
Total Open Pit Mining	**$2.60**	**$90,594.0**
	34,843,830 t pit ore/waste	

23.2.3 UNDERGROUND OPERATING COSTS

A summary of the underground mine operating costs are shown in Table 23.8. Underground mine operating costs are based on the Black Fox budget costs for 2011 through 2015.

LOM underground mine operating costs total US$165.4million, or US$56.48/tonne underground ore mined.

Table 23.8 Underground Mine Operating Cost (US$000s)

Description	LOM Unit Cost (US$/t u/g ore mined)	LOM Total (US$000s)
Definition Drilling	$5.951	$17,426.4
Mine G&A	$4.426	$12,963.0
C&F	$12.102	$35,438.9
Hauling	$6.129	$17,948.3
Loading	$5.754	$16,850.8
Maintenance	$17.439	$51,069.4
Engineering	$1.743	$5,104.8
Geology	$2.933	$8,589.8
Total Underground Mining	**$56.48**	**$165,391.4**
	2,928,318 t u/g ore	

23.2.4 ORE HANDLING OPERATING COSTS

A summary of the ore handling operating costs are shown in Table 23.9. Ore handling operating costs are based on the Black Fox budget costs for 2011 through 2017.

LOM ore handling operating costs total US$33.2 million, or US$5.31/tonne milled.

Table 23.9 Ore handling Operating Cost (US$000s)

Description	LOM Unit Cost (US$/t milled)	LOM Total (US$000s)
Wages	$0.417	$2,607.1
*Truck Haul	$3.825	$23,901.0
Operating Supplies	$0.670	$4,188.7
Outside Services	$0.200	$1,248.9
Maintenance	$0.198	$1,234.7
Total Ore Handling	**$5.31**	**$33,180.4**
	6,248,669 t ore milled	

*The truck haul cost is the cost to load and haul pit and underground ore crushed on site to the mill under a contract agreement with Global Environmental Solutions.

23.2.5 MILLING COSTS

The Stock Mill has a nominal capacity of 2,000 t/d. On-site crushing is required to supplement the crushing circuit at the Stock Mill. Therefore, open pit and underground ore mined at Black Fox will be crushed on-site. The crushed ore is loaded and transported to the mill under a contract agreement with Global Environmental Solutions. The crushing and handling of the Black Fox pit and underground ore to the Stock Mill is costed separately under Ore Handling.

A summary of the mill operating costs are shown in Table 23.10. Mill operating costs are based on the Black Fox budget costs for 2011 through 2017. The Black Fox budget applies a 50% factor to the mill costs when the mining operations cease in 2015. To better reflect the drop in mill costs, a fixed/variable split has been applied to the mill costs. The mill labour costs are assumed fixed while the supplies are assumed variable. This has the effect of keeping the costs in line with the process when the mill tonnage drops. The fixed cost is US$7.25/tonne milled and the variable cost is US$6.58/tonne milled.

LOM mill operating costs total US$86.4million, or US$13.83/tonne milled.

Table 23.10 Mill Operating Cost (US$000s)

Description	Fixed Unit Cost (US$/t milled)	Variable Unit Cost (US$/t milled)	Total Unit Cost (US$/t milled)	LOM Total (US$000s)
Mill G&A	$7.246		$7.246	$45,280.0
Mill Loading		$0.319	$0.319	$1,994.3
Crushing		$0.730	$0.730	$4,558.7
Mill Roads & Dumps		$0.237	$0.237	$1,480.6
Solutions		$1.208	$1.208	$7,548.6
Grinding		$2.183	$2.183	$13,641.9
Leaching		$0.155	$0.155	$967.1
Elution Circuit		$0.501	$0.501	$3,131.9
CIP		$0.174	$0.174	$1,085.3
Conveyors		$0.341	$0.341	$2,129.7
Water & Tailings		$0.122	$0.122	$760.9
Refinery		$0.221	$0.221	$1,383.5
Metallurgy		$0.083	$0.083	$520.0
Mill Maintenance		$0.310	$0.310	$1,936.6
Total Milling	**$7.25**	**$6.58**	**$13.83**	**$86,419.1**
			6,248,669t ore milled	

23.2.6 Assay Lab Costs

The assay lab is considered as a separate cost from milling in the Black Fox mine budget. Material mined in the open pit and the underground operations are assayed, as is material in the mill during the course of the milling operation. Since the pit & underground mine operations cease in 2015 and milling continues to 2017 the split of the assaying costs in the Black Fox 2011 to 2017 budget has been adjusted to account for the tonnage actually assayed from the mine operations and the mill operations.

As a result, the LOM assay lab costs total US$12.0 million or US$0.98/tonne total mined and milled.

23.2.7 Site G&A Operating Costs

A summary of the site G&A operating costs are shown in Table 23.11. Site G&A includes all overhead costs shared by the open pit, underground mine and mill operations. Site G&A operating costs are based on the Black Fox budget costs for 2011 through 2017.

LOM site G&A operating costs total US$30.7 million or US$2.50/tonne total mined and milled.

Table 23.11 Site G&A Operating Cost (US$000s)

Description	LOM Unit Cost (US$/t mined/milled)	LOM Total (US$000s)
Administration	$1.178	$14,470.4
Accounting	$0.310	$3,804.7
Purchasing & Warehouse	$0.170	$2,089.8
Human Relations	$0.113	$1,395.3
Safety	$0.272	$3,348.1
Security	$0.260	$3,200.7
Environmental	$0.194	$2,387.3
Total Site G&A	**$2.50**	**$30,696.3**
	12,278,502t mined/milled	

24.0 ECONOMIC ANALYSIS

The technical-economic results summarized in this section are based upon work performed by Brigus Gold Black Fox engineers and consultants and has been audited by Wardrop. The economic model was developed by Wardrop based on a detailed audit of Black Fox's costs. All costs are in 2011 US constant dollars.

24.1.1 MODEL INPUTS

The economic model developed by Wardrop is pre-tax and assumes 100% equity to provide a clear picture of the technical merits of the project. Assumptions used are discussed in detail throughout this report and are summarized in Table 24.1.

Table 24.1 Technical Economic Model Parameters

Model Parameter	Technical Input
General Assumptions	
Mine Life	8.55 years
Operating Days per year	365 days/yr
Mill Production Rate	2,000 t/d
Market	
Discount Rate	5%
Gold Price (88% of ounces)	US$1200/oz
Gold Price (12% of ounces)	US$500/oz
Royalty	
Private Royalty	none

Black Fox mine has an estimated life of 8.55 years given the reserves described in this report and the 2,000 t/d mill production rate.

Revenue from gold sales was based upon a market price of US$1200/oz for 88% of accountable ounces and US$500/oz for 12% of accountable ounces per the Sandstorm Agreement. Gold treatment and refining charges are at US$0.52/oz plus a charge for transportation and insurance of US$0.65 per US$1,000 of shipment value. Refining, transportation and insurance costs are charged against gross revenues. Wardrop used an exchange rate of Cdn$1.05:US$1.00.

24.1.2 LOM PLAN AND ECONOMICS

The Wardrop LOM plan and economics are based on the following:

- A gold price of US$1200/oz for 88% of accountable ounces and US$500/oz for 12% of accountable ounces (Sandstorm Agreement).
- Probable reserves, no resources are included.
- A mine life of 8.55 years, at a designed rate of 730,000 tonnes/year milled.
- An overall average metallurgical recovery rate of 94% Au, over the LOM.
- A cash operating cost of US$66.94/t milled, US$502/oz Au.
- LOM capital costs are estimated to be US$74.8million being comprised of US$11.7million for the open pit, US$62.1million for the underground mine, US$0.2million for environmental and US$0.7million for the mill.
- No salvage value is modeled.

The base case economic analysis results, shown in Table 24.2, indicate a pre-tax undiscounted cash flow of US$439.0 million and a NPV of US$359.4 million at a 5% discount rate.

Table 24.2 Technical Economic Results ($000s)

Description	Technical Input or Result
Ore	
Open Pit	
Waste	31,742,315 t
Ore	3,101,515 t
Total	34,843,830 t
Grade	3.213 g/t Au
Contained Gold	320,370 oz
Underground	
Total Development	10,166 m
Ore	2,928,318 t
Grade	5.936 g/t Au
Contained Gold	558,849 oz
Mill	
Ore Treated	
Mill tonnes	6,248,669 t
Ore Grade	4.419 g/t Au
Contained Gold	887,754 oz
Recovered Gold @ 94%	834,488 oz
Revenue ($000s)	
Gross Revenue	$933,241
Refining & Transportation Charges	($1,097)
Net Smelter Return	$932,144
Royalty	$0
Gross Income From Mining	$932,144
Realized Price (Gold)	*US$1118/oz Au*

Operating Cost ($000s)	
Open Pit Mine	($90,594)
Underground Mine	($165,391)
Ore Handling	($33,180)
Mill	($86,419)
Assay Lab	($12,033)
G&A	($30,696)
Operating Costs	($418,314)
	US$502/oz Au
	US$66.94/t milled
Cash Operating Margin	$513,829
	US$616/oz Au
	US$82.23/t milled
Capital Cost	
Open Pit	($11,747)
Underground Mine	($62,137)
Environment	($206)
Mill	($728)
Total Capital	($74,818)
Cash Flow	$439,012
($NPV5_{\%}$)	$359,386

24.1.3 SENSITIVITY

A sensitivity analysis was performed for key economic parameters, which are shown below in Table 24.3 and Figures 24.1 to 24.3. This analysis suggests that the project is most sensitive to gold price. Operating costs are slightly more sensitive than capital costs due to the many operating functions associated with the project.

Table 24.3 Project Sensitivity ($NPV_{5\%}$, US$000's)

Description	-20%	-10%	Base Case	+10%	+20%
Gold Price	$210,506	$284,946	**$359,386**	$433,826	$508,265
Operating Costs	$431,395	$395,391	**$359,386**	$323,381	$287,376
Capital Costs	$373,181	$366,283	**$359,386**	$352,488	$345,591

The breakeven gold price was determined to be US$602/oz.

Figure 24.1 Black Fox Cash Flow Sensitivity

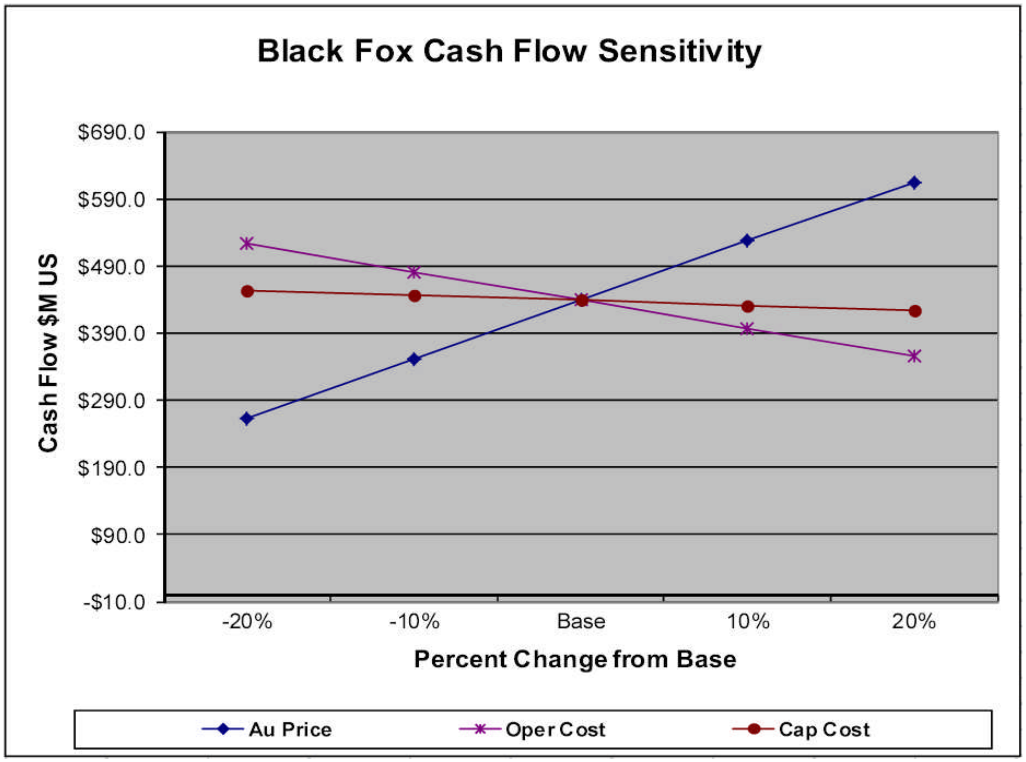


Figure 24.2 Black

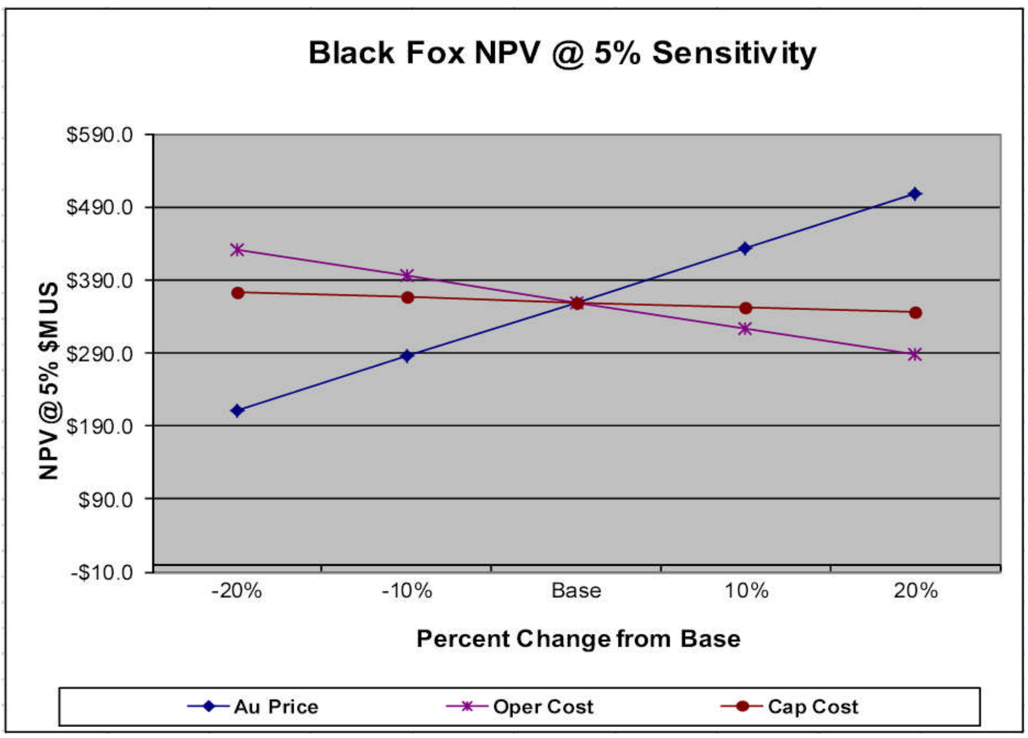
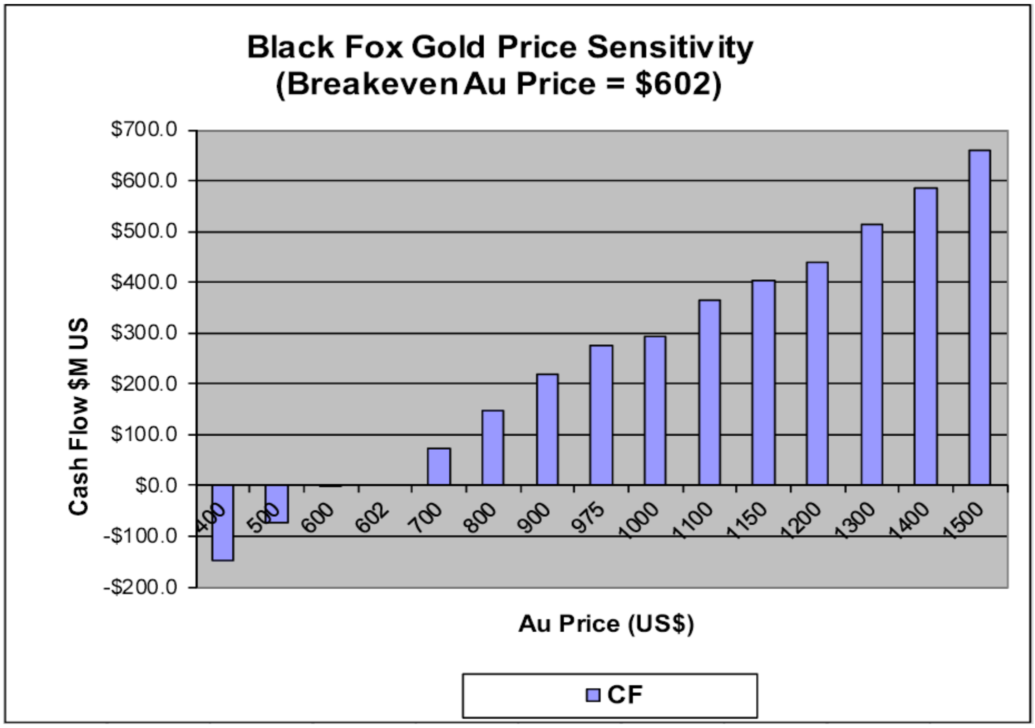


Figure 24.3 Black Fox Gold Price Sensitivity



25.0 RISKS AND OPPORTUNITIES

25.1 OPPORTUNITIES

The ability to mine the Black Fox orebody using open pit mining methods has dramatically increased the knowledge base surrounding the "nuggety" nature of the orebody. Open pit mining has helped to quantify perceived assay problems associated with interpreting grade from small drillhole diameters.

The cut and fill stopes are designed based on a 4 m lift height. During the detailed design phase, there is an opportunity to optimize this stope height and to possibly mine to different design parameters in different areas of the orebody. In areas where the orebody is wide, the stope height could be increased to between 4m and 5m to improve productivity and reduce the unit cost per tonne. An increase in the stope height would also increase the ore to waste ratio thus reducing the amount of waste to be hauled to surface.

25.1.1 GEOLOGY

The deposit is currently drill defined to approximately 400 to 600m below surface where portions remain still open. Black Fox is located midway between the Dome-Hoyle Pond and Holt-Holloway Mines, each of these have been developed and exploited to approximately 1,000m below surface. At the Dome-Hoyle Pond deposit, gold mineralization has been traced to a depth of 1,600m where it remains unconfined. The Black Fox deposit provides opportunity to develop this potential at depth for additional resources. Interpretation of the airborne geophysical surveys completed in October are underway. Future potential along strike could be defined from the survey.

25.1.2 MINERAL PROCESSING AND METALLURGICAL TESTING

The increased mill throughput liberates more gold on a daily basis; however there has been a slight reduction in the percentage recovery (about 3%). Changing or adding to the circuit would reduce the current mill discharge size to one closer to historical operations, and should lead to improved recovery.

As noted in this review, improved crushing efficiency, leaching / retention time, carbon fines recovery are opportunities.

25.1.3 Milling Site

Presently ore received from the mine is stockpiled at the mill site prior to introduction into the milling circuit. The stockpiled ore is exposed to the environment for a number of days prior to processing which likely results in some leaching of metals to the underlying substrate. Brigus Gold may consider placing a liner system below the stockpile area for collection of contact water that may contain leached metals and route to the mill for processing. Not only will this potentially increase gold production but also prevent substrate contamination by metals that would require future remediation.

25.1.4 Mining Operations

The principal opportunity for surface mining operations is for toll processing in the near future, which will allow the mining of variable grades without the need for stockpiling and to reduce the stockpile volumes.

As underground operations escalate, and more extensive definition drilling carried out, there is the potential of finding steep ore blocks amenable to longhole stoping. This method will greatly reduce mining costs.

25.2 Risks

The gold mineralization at Black Fox is relatively high grade and sporadic in distribution. This style of mineralization is common to many similar deposits both within the district and throughout the world. Many of these have been exploited successfully by techniques unique to each deposit. At Black Fox, the near term risk resides in refining the appropriate development and exploitation techniques required to successfully accommodate the nature of the gold mineralization. Conversion of Inferred Mineral Resources to the Indicated category has proven to be of relatively low risk so far. A recently completed drilling program conducted by Apollo during 2007 tested 14% of the previously classified Inferred Resource. The results of the updated Resource estimation showed that 84% of the tested material was converted to the Indicated category and the remaining 16% was invalidated.

The geological model is a conservative estimate of grade and is the basis for the pit optimization, pit design and production schedule. Given the “nuggety” nature of the deposit, the ratio of ore to waste governs fleet utilization assuming a consistent production rate. If more ore is actually mined than predicted in the model, less waste stripping will be required and equipment utilization will decrease.

Much of the bedrock material is known to contain arsenic, which can be leached when exposed to oxygen and water. While treatment plans have been designed, additional pre-emptive dump sequencing may be employed to encapsulate the “Dirty” rock increasing complexity in the dump scheduling of the deposit.

The pit is very close to a major highway and private properties. This may create noise, vibration, dust and generally affect the local area and the mining operations if any of these aspects become excessive.

The overburden material is known to have poor geotechnical characteristics. Careful analysis of the “Till” face needs to be made on the NW corner of the pit close to the highway. If this area presents geotechnical stability issues, a retaining wall may need to be considered.

There is some noted concern about the geotechnical stability of the waste dump if it encroaches on “Till” material. If it were found that 10:1 slopes are required for these areas, the most practical solution would be to remain within the stable dump footprint and make the waste dump higher. This would affect haul costs.

The open-pit comes within 15 m of active underground workings when mining phase 1. Care will be needed when blasting around this area as the underground workings in question will be the major haul route until an additional decline is constructed.

The costs for mining have been inflating at a rapid rate over a relatively short time span. Increased costs will adversely affect what material can be considered to be economic and hence also affect the quantity of contained reserves.

25.2.1 GEOLOGY

The nuggetty nature of the deposit is an inherent risk to adequately sample this deposit. The recommendations by Pitard should be implemented so that the proper sample size is obtained. The grade control program and subsequent reconciliation are also critical path in optimizing the recovery of the resource.

25.2.2 MINERAL PROCESSING AND METALLURGICAL TESTING

The expanded mill is operating at 1,500 to 2,200t/d, part of which was achieved by the addition of a third mill that added 19.5kWh/t of grinding capacity. However the purchased mill was built as a secondary mill rather than as a primary mill and the shell should be re-drilled to accept larger diameter liner bolts. The impact force of the grinding balls comes close to breaking the relatively thin bolts, and thus limits the size of grinding ball that can be used, and reduces the grinding efficiency.

A previous expansion of the crusher circuit had resulted in some excess capacity, and this surplus has since been utilised in the higher mill throughput. By crushing to 6mm (1/4”) the crushing plant creates more dust, and the narrower product gap increases the risk of crusher damage.

Damage either to the primary mill or crusher can reduce mill throughput, as parts have to be repaired off-site leading to protracted delays. A plan to correct this was presented by DMA, which included the replacement of the current primary mill with a Semi-Autogenous (SAG) Mill

25.2.3 Mining Operations

Open Pit

The principal risk to surface mining operations is the possibility of uncharted underground old workings and underground old workings that have not been fully backfilled. Although the underground old workings are relatively recent it is possible that some of the stopes may not have been fully surveyed or backfilled to their maximum extents. Although safe working practices have been established there is a risk to men and machines.

There is also a risk to the recovery and dilution of ore. Where the old workings have not been identified then there is a potential for further loss of ore beyond that already recognised. In addition the grade could be further diluted as a consequence of making the old workings safe. A high dilution factor of 30% has been used in the calculations to allow for this as far as is possible, based on work to date, but there is a risk that this may increase.

Underground

As with all underground mining operations, ground conditions pose a significant risk. There is enough experience of mining in the upper areas to be confident that these risks are not excessive. The most effective defence against risk from poor ground conditions is sound engineering practice, good miner training, well-motivated supervision and a high degree of management focus on safety and standards.

Historically, underground grade control has been challenging due to the complex nature of the mineralization. These challenges are expected to continue as mining progresses deeper. Geologists will have to visit the faces after each blast to mark up the ore contacts and to assess the face for subsequent action.

In the current labour market it is often difficult to attract skilled mining, technical, supervisory and management personnel to an operation. However, the Black Fox Mine is geographically located in a historical mining camp area in close proximity to a major town with a number of current operating mines. As such there should be a great resource in hiring suitably trained and qualified personnel.

25.2.4 Overburden Slopes and Rock Stockpiles

Based upon recent site observation and review of historical reports, Wardrop has not identified any significant structural or operation risks to the existing overburden slope or rock stockpile configurations.

25.3 Environmental Considerations

25.3.1 *Milling Site*

Tailings Management

Based upon recent site observation and review of historical reports, Wardrop has not identified any significant structural or operational risks to the existing tailings management facility.

25.3.2 *Mine Site*

Transitional Settling and Holding Ponds

Based upon recent site observation and review of historical reports, Wardrop has not identified any significant structural or operational risks to the existing facility water management ponds.

26.0 ADDITIONAL REQUIREMENTS FOR DEVELOPMENT PROPERTIES

This section is not applicable.

27.0 INTERPRETATIONS AND CONCLUSIONS

27.1 INTERPRETATIONS

The Black Fox deposit has currently been drill tested to 700 m below surface where portions remain open at depth and along strike. The drilling density is sufficient to estimate mineral resources classified as Indicated and Inferred Resource categories. The deposit contains 1,889 drillholes drilled from surface and underground totalling to 335,841 m.

27.1.1 BLACK FOX MINE MINERALIZATION

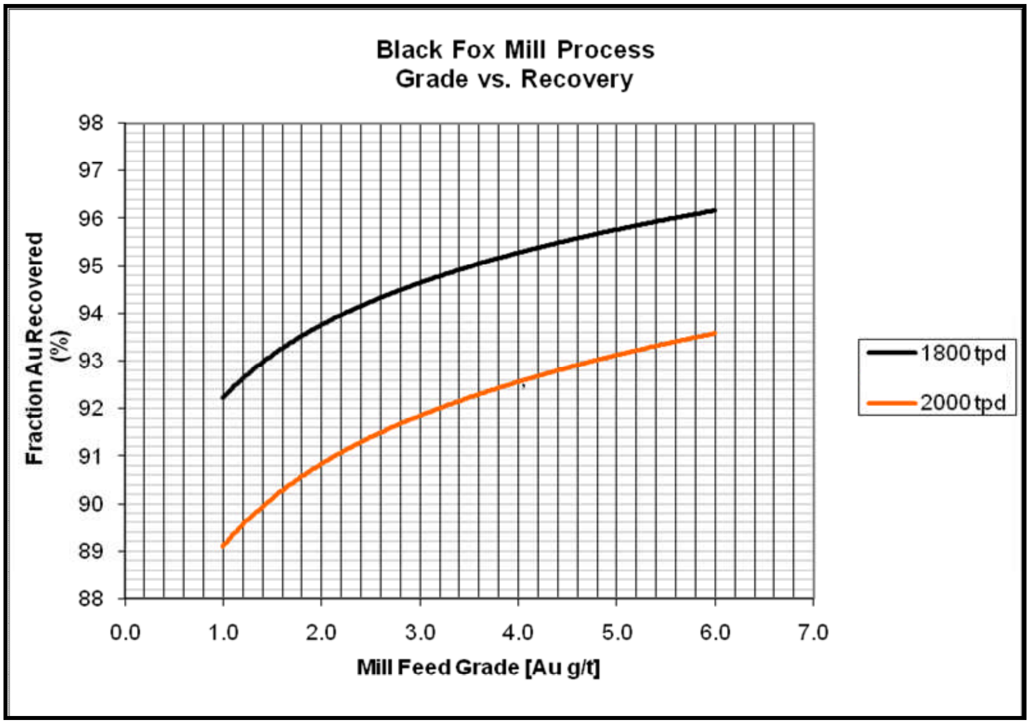
For this study, the mineralization is subdivided into three main domains based on the continuity and style of the mineralization. The first is called the “Main Zone” and is delineated by the primary domain of shearing and mineralization. The second mineralization domain is called the “Flow Zones”. This mineralization occurs as numerous sigmoid and lens shaped bodies completely hosted within or adjacent to the “Main Zone”. The gold mineralization within these bodies has good geologic and grade continuity. The third mineralization domain is High Grade (HG) Indicator Zone. A probabilistic approach was employed to define the zones of mineralization over 2 g/t Au. This HG Indicator Zone was constrained within the Main Zone and overlapped at times on the Flow Zone. Each of the three mineralization domains was modeled independently.

27.1.2 MINERAL PROCESSING AND METALLURGICAL TESTING

Current gold recovery shown in Figure 27.1 below from the daily mill production Report (Scat Modified) indicates a decline of about 3% when the tonnage increases from 1,800 to 2,000 t/d.

As this is based on randomly mixed mill feed the conclusion must be that at 2,000 t/d the gold is not fully liberated. Private discussions with mill staff indicate that this decline is even more significant above 2,200 t/d.

Figure 27.1 Black Fox Mill Process Grade vs. Recover



Limited data is available on the coarseness of the leach feed but one review on Sept 17th, calculated the grinding product as 80% passing 75 micron. This is much coarser than historical data when a P_{80} of less than 45 micron was obtained.

Given the current gold price the incentive is to maximise revenue by increasing tonnage, however, the drop in recovery does indicate that the existing grinding units are being overfed.

Better grinding would be obtained by adding a larger primary mill, as suggested by DMA. An alternative solution might be found by evaluating the existing mills in more detail as there is an uneven distribution of fines (material passing through the screen) below 75 micron.

A more efficient classification may better remove the fines, and thus improve the mill performance by only directing the oversized feed to the mill.

This is merely an observation and would require a more detailed study to quantify the benefits; however, the cost of modifying the present primary mill would be less than replacing it with a SAG mill.

While there is some discrepancy between truck scale ore deliveries and the mill feed weightometer these are the variances expected, and the reconciled monthly data shows that the daily scale errors are balanced over the month.

The mill assay accuracy is confirmed by the weekly bullion shipments and tailings values.

27.1.3 Mining Operations

The surface mining operations are already well established and phase 1 is nearing completion. Consequently, procedures for dealing with the underground old workings, stabilising the till batters, haul road construction and maintenance, pumping and treatment of water, grade control, waste and ore separation, drilling and blasting, overburden and waste handling, dump construction and all ancillary works are proven and working satisfactorily.

The underground old workings will continue to challenge the surface mining operations, particularly if uncharted or incompletely backfilled stopes are encountered.

The surface mining equipment that is in operation is adequate and in good working order for the conditions already encountered and envisaged in the future. An additional 91 t truck and a larger front-end loader is planned to be purchased.

27.2 Conclusions

The Black Fox deposit has been adequately drill tested to estimate grade and tonnes classified as Indicated and Inferred Resources. Indicated Resources were categorized based on a minimum of three drill holes when the nearest sample point was less than or equal to 20 m. The remaining blocks were classified as Inferred Resources. No blocks were categorized as Measured.

The mineralization of the Black Fox Mine as of October 31, 2010 is classified as Indicated and Inferred resources. The classified mineral resources are shown in Table 27.1. The mineral resource is reported at a 0.88 gpt Au cut-off grade for the open pit and at 2.54 g/t Au cut-off grade for the underground. These cut-offs have been developed by mine engineering as outlined in Section 19.

Table 27.1 Resource Statement, Black Fox Mine

Boundaries	Block Model	Cutoff (g/t Au)	Resource Class	Tonnes	Capped (Au g/t)
Below October 31 Pit Survey	2009A	>= 0.88	Indicated	3,164,200	4.445
Above 9820 m Elev.		>= 0.88	Inferred	667,100	2.610
Below 9820 m Elev.	2009A_4M	>= 2.54	Indicated	2,504,800	7.192
		>= 2.54	Inferred	115,200	5.816

28.0 RECOMMENDATIONS

Black Fox should continue to be developed. The following are Wardrop's recommendations.

28.1 MINERAL RESOURCE

The work by Brigus Gold has been found to follow industry accepted practices. Two areas were identified as opportunities to refine the mineral resource:

- Wardrop recommend further specific gravity analysis by rock types and mineralization styles to confirm that one value is appropriate for the whole model.
- Pitard (2005) conclusions on sampling identified that the drillhole data is likely biased and will likely underestimate the contained gold within the deposit. Wardrop agrees with Pitard's work and recommends that a comprehensive grade control procedure be prepared to address the sampling issues for the Black Fox Mine. This will facilitate the delineation of ore and waste and the reconciliation between the resource, reserve and milled grade.

28.2 GEOTECHNICAL

Pro-active geotechnical monitoring is recommended for all stages of pit development. The monitoring program should be implemented as a staged approach and include detailed geotechnical and tension crack mapping, as well as a suitable combination of surface displacement monitoring (surface prisms) and piezometers.

Sufficient staffing resources should be allocated to collect, process and interpret the geotechnical monitoring data on a weekly basis or as frequently as required. The timely identification of accelerated movements from surface displacement monitoring and tension cracks will be critical.

Up-to-date reports on the status of highwall stability should be compiled and discussed regularly with operations personnel. These reports will also assist mine engineering staff with their efforts to optimize final pit slopes and improve the effectiveness of the controlled blasting program.

All seeps and springs should be inspected, mapped and photographed. Large-scale structures should be characterized and monitored as they have the potential to develop into tension cracks.

28.3 Mineral Processing and Metallurgical Testing

One option to reduce the mean particle size entering the leach circuit has been presented by DMA. This is to replace the current "primary" mill with a Semi-Autogenous (SAG) mill, with a wrap-around motor.

This is a viable option, but could incur significant costs, and the plant would need to operate on reduced tonnage while new foundations were installed.

Wardrop suggests a more thorough review of the existing circuit. Mill data collected on Sept 17th suggests that too much of the fines are misdirected to the secondary mills, and that minor changes to the cyclones may be an effective solution.

If this is insufficient then there remains the option of re-drilling the primary mill shell bolt holes, This would permit the installation of heavier liners using larger fastening bolts, and would permit the mill to operate using larger (heavier) grinding balls.

Although the mill site tailings facility has been designed for future expansion, it is recommended that the expansion design be reviewed based upon increased ore milling and tails loading to the structure. This should include a detailed evaluation of facility volumetrics and dam structural design capacity. Furthermore, the Phase 5 water management pond design should be evaluated for stability as well as discharge pumping requirements under peak storm/snow melt events. Based upon these analyses recommendations will be provided to upgrade the facility for the additional loading, if needed.

As discussed in Section 25 of this report, Brigus Gold should consider placing a low permeable liner system below the ore stockpile area at the mill site. A costing analysis should be conducted to evaluate increased gold yield from contact water vs. liner installation cost.

28.4 Mining Operations

Given that the surface mining operations are underway and procedures well established there are no major recommendations to improve the operations. It will be important, however, to monitor the procedure for the safe working of the underground old workings to ensure the risks are minimized. This also has an impact on the dilution and recovery of the ore, so close monitoring and reconciliation of tonneages and grade between mine and mill is particularly important.

Presently, the rock stockpiles and soil overburden stockpile appear stable. However, it should be expected that the rock stockpiles will experience additional load due to the underground mine development. Analyses should be conducted to evaluate the stockpile remaining design capacity and required additional capacity if needed based upon projected waste rock generation. In addition, stability analyses will need to be updated based upon added stockpile tonnage. Best management surface water and erosion controls also will also require re-evaluation and updating if stockpile expansions are required. Unless Brigus Gold plans future expansion of the pit, it is expected that the present overburden stockpile configuration will not be significantly altered and therefore should not require any design or operational changes.

28.4.1 UNDERGROUND MINE DESIGN

Wardrop recommends a critical review of the underground design, specifically the migration of main accesses – from the hangingwall (in the current design) to the footwall. A trade-off study should be carried out to identify the advantages and disadvantages of either location, plus optimization of the location of a single ramp access to both the East and West Zones, and attendant lateral development.

28.5 ENVIRONMENTAL CONSIDERATIONS

The dirty waste rock stockpile may be expanded due to planned underground mining. This will result in increased contact water influent to the holding pond. The holding pond capacity should therefore be reviewed and based upon future peak load conditions be expanded if required. Also, the structural capacity of perimeter dams should be evaluated based upon the additional load profile.

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30.0 CERTIFICTES OF QUALIFIED PERSONS

30.1 CERTIFICATE FOR MICHAEL GABORA, P. GEO.

I, Michael Gabora, P.Geo, of Albuquerque, New Mexico, do hereby certify:

- I am a Senior Hydrogeologist with Tetra Tech with a business address at 6121 Indian School Rd. NE, Suite 200, Albuquerque, New Mexico, U.S.A., 87123.
- This certificate applies to the technical report entitled “Black Fox Project, National Instrument 43-101 Technical Report”, dated January 6, 2011 (the “Technical Report”).
- I am a graduate of the University of New Mexico, (Bachelor of Science in Earth and Planetary Sciences, 1998; Master of Water Resources, 2003).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of Ontario, License 0969.
- My relevant experience includes having worked as a hydrogeologist for eleven years since my graduating with my B.Sc. The focus of this work has been mining hydrogeology, dewatering and groundwater flow modeling.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- I am responsible for the preparation of Sections 1.5.2, 9.2 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th of January, 2011 at Toronto, Ontario.

“Original document signed and sealed by Michael Gabora, P.Geo.”

Michael Gabora, P.Geo
Senior Hydrogeologist
Tetra Tech

30.2 CERTIFICATE FOR PETER BROAD, P.ENG.

I, Peter Broad, P.Eng., of London, Ontario, do hereby certify:

- I am a Lead Metallurgist with Wardrop Engineering Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled "Black Fox Project, National Instrument 43-101 Technical Report", dated January 6, 2011 (the "Technical Report").
- I am a graduate of Manchester University (UK), with a Honours BSc Degree, in Metallurgy, 1969.
- I am a member in good standing of the Ontario Professional Engineers (Licence 90344227).
- My relevant experience includes twelve years as a base-metal metallurgist and/or Mill Superintendent and 20 years as a consultant engineer in precious and base metals. This includes process design, operations, and construction.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was October 26, 2010 for one day.
- I am responsible for the preparation of Sections 1.4, 1.8.3, 16, 25.1.2, 25.1.3, 25.2.2, 27.1.3, 28.3 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Toronto, Ontario.

"Original document signed and sealed by Peter Broad, P.Eng."

Peter Broad, P.Eng.
Lead Metallurgist
Wardrop Engineering Inc.

30.3 CERTIFICATE FOR PHILIP BRIDSON, P.ENG.

I, Philip Bridson, P.Eng., of Lively, Ontario, do hereby certify:

- I am a Senior Mining Engineer with Wardrop Engineering Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled “Black Fox Project, National Instrument 43-101 Technical Report”, dated January 6, 2011 (the “Technical Report”).
- I am a graduate of the Michigan Technological University (B.Sc., Mining Engineering, 1972 and B.Sc., Engineering Administration, 1972).
- I am a member in good standing of the Association of Professional Engineers of Ontario, License #5181011.
- My relevant experience includes
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was October 26, 2010 for one day.
- I am responsible for the preparation of Sections 1.7, 21, 22, 23, 24 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Toronto, Ontario.

*“Original document signed and sealed by
Phil Bridson, P.Eng.”*

Philip Bridson, P.Eng.
Senior Mining Engineer
Wardrop Engineering Inc.

30.4 CERTIFICATE FOR PACIFICO CORPUZ, P.ENG.

I, Pacifico Corpuz, P.Eng., of Toronto, Ontario, do hereby certify:

- I am a Senior Mining Engineer with Wardrop Engineering Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled "Black Fox Project, National Instrument 43-101 Technical Report", dated January 6, 2011 (the "Technical Report").
- I am a graduate of Mapua Institute of Technology with a degree of Bachelor of Science in Mining Engineering, 1961.
- I am a member in good standing of the Professional Engineers of Ontario (License # 9428509).
- My relevant experience with respect to mine planning includes over 40 years of international experience in mine operation and planning from conceptual stage to project implementation.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was October 26, 2010 for one day.
- I am responsible for the preparation of Sections 1.8.4, 19.3.1 to 19.3.8, 25.2.3, 28.4.1 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Toronto, Ontario.

"Original document signed and sealed by Pacifico Corpuz, P.Eng."

Pacifico Corpuz, P.Eng.
Senior Mining Engineer
Wardrop Engineering Inc.

30.5 CERTIFICATE FOR RICHARD HOPE, C.ENG.

I, Richard Hope, C.Eng., of Harrogate, United Kingdom, do hereby certify:

- I am a Senior Mining Engineer with Wardrop Engineering Inc. with a business address at Ground Floor, Unit 2, Apple walk, Kembrey Park, Swindon, SN2 8BL, UK
- This certificate applies to the technical report entitled "Black Fox Project, National Instrument 43-101 Technical Report", dated January 6, 2011 (the "Technical Report").
- I am a graduate of The Royal School of Mines, Imperial College, London University, (BSc (Hons), 1978).
- I am a member in good standing of the Institute of Materials, Minerals and Mining and a Chartered Engineer with the Engineering Council [UK, 351484].
- My relevant experience includes over 35 years of operational and consultancy experience in underground and surface mining of various minerals including gold in many countries.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was November, 2010 for one day.
- I am responsible for the preparation of Sections 1.8.4, 19.1.1, 19.1.3 to 19.1.17, 25.1.4, 25.2.3, 27.1.2, and 28.4 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Toronto, Ontario.

"Original document signed and sealed by Richard Hope, C.Eng."

Richard Hope C. Eng.
Senior Mining Engineer
Wardrop Engineering Inc.

30.6 CERTIFICATE FOR KARLIS JANSONS, P.ENG.

I, Karlis Jansons, P.Eng., of Toronto, Ontario, do hereby certify:

- I am a Vice President, Geotechnical Engineering with Wardrop Engineering Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled "Black Fox Project, National Instrument 43-101 Technical Report", dated January 6, 2011 (the "Technical Report").
- I am a graduate of The University of Toronto, BASc, 1981.
- I am a member in good standing of the Association of Professional Engineers of Ontario, License #21839501.
- My relevant experience includes geotechnical engineering design aspects of the development of mine infrastructure and mine waste management including numerous projects in Canada, Brazil, Europe, Russia, Latin America and Africa for the past 28 years.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- I am responsible for the supervision of Sections 1.8.5, 19.1.18, 19.4, 19.5, 25.2.4, 25.3.1, 25.3.2, 28.5 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Toronto, Ontario.

"Original document signed and sealed by Karlis Jansons, P.Eng."

Karlis Jansons, P. Eng.
Vice President, Geotechnical Engineering
Wardrop Engineering Inc.

30.7 CERTIFICATE FOR ANDREW MACKENZIE, P.ENG.

I, Andrew MacKenzie, P.Eng., of Keswick, Ontario, do hereby certify:

- I am a Mining Manager with Wardrop Engineering Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled “Black Fox Project, National Instrument 43-101 Technical Report”, dated January 6, 2011 (the “Technical Report”).
- I am a graduate of Queen’s University (B.Sc. Mine Engineering, 1994).
- I am a member in good standing of the Association of Professional Engineers of Ontario, License #90470477.
- My relevant experience includes expertise in underground geotech and mine design. From 1994 to 1999 I worked at INCO as mine engineer planner. From 1999 to 2003 I was principal consultant for Paste Systems Inc. and MacKenzie Consultancy based in Sudbury, Ontario. From 2003 to 2009 I was the Manager of Engineering for Dynatec responsible for estimates, and engineering projects. In 2009-10 I was a Principal Engineer with SRK where I completed NI 43-101 Technical Reports. Currently at Wardrop as manager of engineering resources.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was October 26, 2010 for one day.
- I am responsible for the preparation of Section 18 and supervision of Sections 1.8.4, 19.3.1 to 19.3.8, 25.2.3, 28.4.1 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Toronto, Ontario.

“Original document signed and sealed by Andrew MacKenzie, P.Eng.”

Andrew MacKenzie, P. Eng.
Mining Manager
Wardrop Engineering Inc.

30.8 CERTIFICATE FOR TIM MAUNULA, P.GEO.

I, Tim Maunula, P.Geo., of Oakville, Ontario, do hereby certify:

- I am a Chief Geologist with Wardrop Engineering Inc. with a business address at 900-330 Bay Street, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled “Black Fox Project, National Instrument 43-101 Technical Report”, dated January 6, 2011 (the “Technical Report”).
- I am a graduate of Lakehead University, (B.Sc. Honours, 1979).
- I am a member in good standing of the Association of Professional Geoscientists of Ontario (License #1115).
- My relevant experience with respect to geology includes over 25 years of exploration, operations and consulting experience.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- I am responsible for the supervision of Sections 1.2, 1.3, 1.8.1, 3 to 15, 17, 25.1.1, 25.2.1, 27.1.1, 27.2, 28.1 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Toronto, Ontario.

“Original document signed and sealed by
Tim Maunula, P.Geo.”

Timothy Maunula, P.Geo.
Chief Geologist
Wardrop Engineering Inc.

30.9 Certificate for Douglas Ramsey, R.P. Bio (BC)

I, Douglas Ramsey, R.P. Bio, (BC), of Vancouver, British Columbia, do hereby certify:

- I am a Manager – Environmental Assessment, Permitting, and Natural Resources with Tetra Tech with a business address at 800-555 West Hastings Street, Vancouver, BC, V6B 1M1.
- This certificate applies to the technical report entitled “Black Fox Project, National Instrument 43-101 Technical Report, dated January 6, 2011 (the “Technical Report”).
- I am a graduate of the University of Manitoba, Winnipeg, Manitoba, (B.Sc. (Hons), Zoology, 1979, and M.Sc. Zoology, 1985).
- I am a member in good standing of the College of Applied Biology, British Columbia, as a Registered Professional Biologist #1581.
- My relevant experience is 28+ years experience as an environmental consultant working in environmental permitting and 22 years of experience in the environmental permitting monitoring and closure of mining projects. My mining permitting experience includes NB, ON, MB, SK, and BC and includes gold, base metal, rare earth elements, and potash.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was October 26, 2010 for one day.
- I am responsible for the preparation of Sections 1.6, 20 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Toronto, Ontario.

“Original document signed and sealed by Doug Ramsey, R.P. Bio.”

Douglas Ramsey, R.P. Bio (BC)
Manager – Environmental Assessment, Permitting and Natural Resources
Tetra Tech

30.10 CERTIFICATE FOR CHARLES TKACZUK, P.ENG.

I, Charles Tkaczuk, P. Eng. of Waterloo, Ontario do hereby certify:

- I am a Project Manager with Wardrop with a business address at 330 Bay Street, Suite 900, Toronto, Ontario, M5H 2S8.
- This certificate applies to the technical report entitled “Black Fox Project, National Instrument 43-101 Technical Report:, dated January 6, 2011 (the “Technical Report”).
- I am a graduate of University of Waterloo, (BSc., 1982).
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of Ontario, License # 46570503
- My relevant experience is a combined 32 years working directly in the mining industry of which 26 years have been as a Professional engineer, supporting the design, construction, maintenance and operation of mining (open pit and underground) and mineral processing plants.
- I am a “Qualified Person” for purposes of National Instrument 43-101 (the “Instrument”).
- My most recent personal inspection of the Property was October 26, 2010 for one day.
- I am responsible for Section(s) 1.1, 1.5.1, 2, 19.3.9 through 19.3.11 of the Technical Report.
- I am independent of Wardrop as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Toronto, Ontario

“Original document signed and sealed by Charles Tkaczuk, P.Eng. ”

Charles Tkaczuk, P.Eng.
Project Manager
Wardrop Engineering Inc.

30.11 CERTIFICATE FOR MARVIN SILVA, PH.D., P.ENG.

I, Marvin Silva Ph.D., P.Eng., of Chandler, Arizona, do hereby certify:

- I am a Senior Geotechnical Engineer with Tetra Tech with a business address at 3031 W. Ina Road, Tucson, Arizona 85741.
- This certificate applies to the technical report entitled "Black Fox Project, National Instrument 43-101 Technical Report", dated January 6, 2011 (the "Technical Report").
- I am a graduate of University of Alberta, Ph.D. in Geotechnical Engineering, 1999.
- I am a member in good standing of the Association of Professional Engineers and Geoscientists of Alberta, License # 52477.
- My relevant experience is soil and rock stability, pit slope stability, geotechnical engineering.
- I am a "Qualified Person" for purposes of National Instrument 43-101 (the "Instrument").
- My most recent personal inspection of the Property was October, 2010 for one day.
- I am responsible for Section(s) 1.8.2, 19.1.2, 28.2 of the Technical Report.
- I am independent of Brigus Gold Corp. as defined by Section 1.4 of the Instrument.
- I have no prior involvement with the Property that is the subject of the Technical Report.
- I have read the Instrument and the technical report has been prepared in compliance with the Instrument.
- As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

Signed and dated this 6th day of January, 2011 at Edmonton, Alberta.

"Original document signed and sealed by Marvin Silva, Ph.D., P.Eng."

Marvin Silva, PhD, P.Eng.
Senior Geotechnical Engineer
Tetra Tech