NI 43-101 Prefeasibility Study
Apollo Gold Corporation
Black Fox
Timmins, Ontario, Canada

Prepared for:

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Executive Summary

This Prefeasibility Study (PFS) is undertaken to advance the Black Fox gold project (Black Fox) located in Timmins, Ontario, Canada. Costs, appropriate with the level of study, have been estimated and form the basis of the economic analysis presented. This report meets the requirements for NI 43-101 and the Resource and Reserves definitions are 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.

David K. Young, previously employed by Apollo as Vice President of Business Development and Technical Services during March 2004 through October 2005, visited the property on various occasions from March 2004 though October 2005 for over 30 days and is the Qualified Person (QP) for the overall PFS. Bart Stryhas of SRK, is the QP for the preparation of Section 15 of this PFS. Richard F. Nanna, Senior Vice President-Exploration Apollo Gold Corporation, is the QP for the preparation of Sections 5 through 9 and 13.

Black Fox is located 10km east of Matheson, Ontario, along Hwy 101 East and approximately 655km 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'N latitude and 80°21'W longitude. The Glimmer underground mine, formerly operated by Exall is located on the property. 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.

The property includes approximately 939ha of land of which, 75ha are unpatented federal land, 563ha are owned by Apollo, 175ha where Apollo only has surface rights, and 126ha where Apollo only has mineral rights but no surface rights. Although Apollo has kept all of their claims current and up to date as far as fee and work commitments, one claim is in dispute over who has control of the property. The claim in question was staked by a party who believed that Apollo allowed their rights to the claim to expire. Documentation provided by Apollo indicates that the claim is still in their control. This dispute is currently before the Ministry of Northern Development and Mines (MNDM) who has jurisdiction in this matter. Apollo has 100% ownership of the property and all mineral, surface and forestry rights and there are no production royalties. The property is currently on care and maintenance. Exploration activities are being conducted to address further mineral potential of the property.

There are no environmental liabilities at the Black Fox Mine Site. In December 2004, Apollo submitted a Closure Plan for the existing conditions of the Black Fox Project site, which supersedes all other previously plans. A new Closure Plan will be developed for the proposed future development of the Project, in compliance with legislation and directives from all pertinent regulatory bodies.

Black Fox 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. 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, extensional veins in widely distributed within carbonatized metamorphosed greenstone rocks. These deposits are typically associated with crustal scale compressional faults with a vertical extent of ≤2km and limited metallic zoning.

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 west of Black Fox and the Holloway and Holt-McDermott Mines are located to the east. Each of these deposits hosts approximately the 800k to 1Moz of gold. Many of the deposits extend downwards 1,000m below the surface, and some are blind deposits with no surface expression.

Gold mineralization at Black Fox occurs within a main ankerite alteration zone with a known strike length of over 1km and ranges between 20 to 100m in true width. 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 occur as concordant zones, 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.

The three main types 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; and
- Mineralization hosted by silicified felsic dikes.

The first mineralization type is quartz-rich portions of the AUV and CGR rock types. This includes the "green carbonate ore". The typical host include ankerite-fuchsite altered ultramafic volcanic rocks, commonly found in the footwall portion of the DPFZ. While characterized by low-sulfide contents, small amounts of pyrite are typical of these mineralized zones. Quartz veining and quartz stockworks 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.

Mineralization hosted within mafic volcanic units that is associated with >5% fine-grained pyrite and minor to moderate quartz veining are coded as BMV with pyritic portions as MV rock types. Quartz veins are typically parallel to foliation, and visible gold is characteristically absent in this type of mineralization. This mineralization is referred to in the district as 'flow ore' and is frequently, but not always, associated with bleaching of the mafic volcanic host rocks. This style is common in the footwall portion of the DPFZ in the eastern part of the 235 level underground drilling.

The third type of mineralization is hosted in silicified felsic dikes. These dikes include both quartz-feldspar porphyries and finer grained units which are possibly syenitic in origin, Mineralization in the dikes is associated with increased silicification, pyrite and some quartz veining all associated with a fracture foliation. In the middle and hangingwall portions of the DPFZ, dike-hosted mineralization can often be correlated from hole to hole. This is in contrast to the blocks and lenses of felsic dikes that occur in the footwall portions of the DPFZ.

A total of 1,826 surface and underground drillholes, (total of 324,625m) have been completed on the project by Noranda, Exall and Apollo between 1989 and 2005. Of these drillholes, 1,541 were completed by Apollo between 2002 and 2005. Apollo's exploration drilling continued from previous campaigns on 12.5 to 25m fence lines. Two main objectives included infill delineation of existing mineralization, and to explore for areas of new mineralization. In 2004, a 1,250m long exploratory underground drift (4m x 4m) was developed horizontally in the hanging wall, 235m below the surface, to establish drill stations for an underground drilling program. The underground drilling program consisted of 75,700m of diamond drilling from 371 core holes. Surface drilling continued and by the end of 2006, Apollo had completed 825 diamond drillholes on the property, totaling 212,095m.

Core is logged and sampled at the core logging facility in Matheson and later at the Black Fox site. Samples from the split core remain in the logging facility until they are picked up by representatives of the assay laboratory. Apollo sends the bulk of the core samples to Swastika Laboratories Ltd. A smaller number of samples are sent to the SGS laboratory in Rouyn, Quebec. The maximum assay interval in the unaltered hangingwall and footwall rocks is 1.5m. The gold is generally located within the quartz and carbonate zones. Very little of the gold is associated with clay. Therefore, flushing of the gold while drilling or splitting is less of a problem.

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 ½ (15g) to 1 (30g) assay ton samples for assay. Most of the assays were completed by fire assay methods with a gravimetric finish.

MDA stated in previous work observed that there are two serious sampling issues with the deposit, both of which are related to coarse gold in the deposit, and result in samples that tend to contain less gold than is actually present. They are actually the same problem and are based on sample size issues and coarse gold.

"The first issue is to get a large enough sample to represent the area sampled. The gold at the Black Fox deposit appears to be concentrated in small areas. When the gold is concentrated in small areas, drillhole samples will occasionally get too much gold in the sample when the area of concentration is intersected or more often, miss the area of concentration and get too little gold in the sample. The core holes that form the basis for the resource and reserve estimate are too small to obtain a representative sample. Some samples may even appear to be waste without the concentrated gold. It is likely that holes several meters in diameter would be required to obtain representative samples of the deposit. MDA reviewed areas that were penetrated by drillholes prior to mining, and commonly areas that were stoped appeared to be un-mineralized based on the drilling.

The second issue is getting the representative amount of gold in the sample pulp once the sample has been obtained. Gold particles up to 0.15cm have been observed and particles of 0.06cm are very common (Pitard, 2005). The proper sample size is required in order to get a representative sample again, but this time we have all of the gold contained in the sample somewhere in the core. With coarse gold, 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 0.15cm gold particles that occur at Black Fox, samples of up to 109kg must be processed in their entirety (Pitard, 2005). If the sample contains 0.06cm gold particles, which commonly occur in the deposit, a 7kg sample must be processed in its entirety (Pitard, 2005). These sample sizes are much larger than the typical 30g fire assay sample or even the generally *larger than the 1,000g 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."

MDA states that both the size of the sample to measure the gold in the deposit and the size of the sample to measure the gold in the sample are too small, and will result in a database where some samples represent a higher sample grade than is present at the sample location, but many samples represent too low a grade than is present at the sample location. MDA concurs with Pitard's conclusion, that the drillhole data is likely biased and will likely underestimate the contained gold within the deposit."

SRK concurs with the observations and opinions of MDA and Pitard as discussed above. Based on these observations and opinions, SRK has put significant emphasis into creating the resource estimate described below which approximates the historical production while remaining conservative.

The drillhole sample database was compiled by Apollo and reviewed for QA/QC by Analytical Solutions of Toronto, Canada and is determined to be of good quality. The database used for the resource estimate, consists of three Microsoft Excel spreadsheets containing collar locations, drillhole orientations with down hole deviation surveys and assay intervals with results. The appropriate codes for missing samples and no recovery were used during the modeling procedures. SRK did not provide any independent review of sampling quality and has used the data as it was provided by Apollo. The assay database contains several columns of gold assays representing the original 30g and 50g assays as well as numerous repeat assays included some screen metallic analyses. The Au 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 were used in place of the original assay. This resulted in the averaging of up to five assays in some instances. This approach was used to take advantage of larger assay charges and screen metallic samples were possible. Although this method can introduce assay bias, the author felt it was the best approach to try to overcome sampling difficulties described previously in this report.

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

Variogram analysis was conducted on the 1.5m drillhole data to determine appropriate projection ranges and to test for any preferred orientation of the mineralization. The composites were first flagged to differentiate them into data sets to be used for an indicator estimation technique. All composites greater than 0.5g/t-Au were flagged as "ore group" and those below this cut-off were flagged as "waste group". Variograms were then constructed using Vulcan software along all directions within the plane of the mineralization and perpendicular to it.

The Black Fox deposit was modeled only for gold content. The model has a parent block size of 3m by 3m by 3m and is sub-blocked to 1m by 1m by 1m along the geologic boundaries of the flow ore and along the boundaries of the underground workings. This small block size was chosen mainly to try to emulate localized pod like bodies of mineralization. All block estimates were made using only the drillhole 1.5m composites. The Black Fox model was verified using two procedures. The first was to run the model using three different estimation techniques including; Inverse Distance Weighting Squared (ID2), Inverse Distance Weighting to the third power (ID3) and Kriging. The second, was to compare the predicted ore tonnes and grade to the historically extracted tonnes and grade within the underground mine.

Based on visual comparison of block grade distribution relative to drillhole composites and histogram comparison between the same, the Inverse Distance Weighting to the 3rd power using a minimum of one and maximum of five composites was chosen as the most appropriate estimation method for Black Fox. The tonnage and grade for different mining methods at appropriate Au cut-offs for indicated and inferred resources are shown in Tables 1 and 2.

Table 1: Black Fox Indicated Resource Statement* as of June 30, 2007

Mining Method**	Category	Cut-off g/t-Au	Tonnes	Grade g/t-Au	Contained oz-Au
Open Pit	Indicated	1	4,358,500	5.5	763,700
Underground	Indicated	3	1,574,500	11.3	570,000

* Indicated Resources include Probable Reserves listed separately below.
** Mining Method is determined by relative location above or below the 9815m elevation

Table 2: Black Fox Inferred Resource Statement as of June 30, 2007

Mining Method*	Category	Cut-off g/t-Au	Tonnes	Grade g/t-Au	Contained oz-Au
Open Pit	Inferred	1	3,255,500	4.7	490,900
Underground	Inferred	3	929,000	12.3	368,000

* Mining Method is determined by relative location above or below the 9815m elevation

The open pit ore reserves are based on the previous pit design created by MDA. In order to calculate the open pit reserves, the original MDA pit shell was used as a limiting surface within which to tabulate the indicated resource. For the open pit reserve, no mining dilution was incorporated. The underground ore reserves are all located below the 9815 level. These include material within 50 individual stopes designed by outlining the indicated resource blocks at a 2g/t-Au cut-off. This cut-off was used to define the boundaries provided an average grade of 4g/t-Au or higher material was maintained within the stope. Other design criteria included a minimum width of 3m, with the average width 8m and the average length 34m. All stopes are designed for a traditional cut and fill mining technique. The underground reserves include dilution of approximately 22% of which 66kt of indicated material with an average grade of 1.3g/t-Au was used and the remaining amount of dilution material was assigned a grade of 0g/t-Au. The Probable Reserves at Black Fox are presented in Table 3.

Table 3: Black Fox Probable Reserve Statement* as of June 30, 2007

Mining Method	Cut-off g/t-Au	Tonnes (000s)	Grade g/t-Au	Contained oz-Au
Open Pit	1	3,362	5.8	625,000
Underground**	3	1,108	10.6	377,000

* Probable Reserves are included within Indicated Resources listed above.
** Underground Reserves include dilution of 66,000t of indicated material with an average grade of 1.3g/t-Au.

The orientation, proximity to the surface, and geological controls of the Black Fox ore body will require mining of the ore reserves with open pit and underground mining techniques. This PFS is based on mining of the Black Fox deposit at a processing rate of 1,500t/day, or 540,000t/yr. The size of the Black Fox resource, combined with the application of the most efficient mining operation possible and the maintaining of reasonable vertical advance rates made the 1,500t/day rate appropriate. All of the technology, methods and equipment chosen for the mine are industry-standard and well proven.

Open pit reserves were developed by updating a PFS of the open pit mine completed by MDA in August 2006. The open pit design completed by MDA was used for reporting the open pit reserves contained in SRK's resource model. SRK is currently re-optimizing the open pit for its ongoing feasibility study, and will perform trade-off studies to determine the optimal mining method for some of the deeper material contained in the current open pit configuration.

The Lerchs-Grossman pit optimization completed by MDA was based on the parameters listed below:

- Overburden mining cost - $1.25/t of material;
- Rock mining cost - $1.50/t of material;
- Processing cost - $12.16/t ore;
- General and Administrative cost - $4.50/t ore;
- Plant gold recovery – 96%; and
- Pit Slopes.

To continue mining of the ore resources at Black Fox below the 9815m level, underground mining methods were reviewed that would minimize dilution, capital, and operating costs, maximize recovery of the ore resources while maintaining the project's designed production capacity. The underground stopes were designed by contouring the resource model on 2g/t-Au grade contours. The 2g/t-Au contour was used to define the stope boundary as long as 4g/t-Au or higher material was within the 2g/t-Au contour. The stopes were designed with a minimum width of 3m, with the average width 8m and the average length 34m. The stopes were designed on 5m high cuts with stope access cross cuts from the main development every 20m in elevation. In addition a 25m thick crown pillar was established between the bottom of the open pit and the top of the upper most underground production stopes. There will not be any production ore taken from within the crown pillar area. Table 4 summaries the mine design parameters selected for the underground portion of Black Fox.

Table 4: Underground Design Parameters

Parameter	Description	Value
Operating Schedule	Mine Schedule	3 shifts/day; 7 days/week, 360 days/year
Production Target	Daily / Annual	1,500t/d/540,000t/a
Haulage	One Way Haul	3,500m
	Grade	15% max
	Haul way Dimensions	5m high and 5m wide
Mine Production	Production Stope	5.0m wide x 5.0m high
	Longest haul to ore pass	300m
Backfill Requirements	Waste Rock Fill	233,160m^3
	Tailing Backfill	62,176m^3
	Cemented Tailing Backfill	15,544m^3
General	Specific Gravity	Ore: 2.8 and Waste: 2.8 Backfill 1.76t/cubic meters

Black Fox underground ore zone orientation, especially the moderate dip (45º) and a non-continuous ore zone structure (pods) required careful consideration of different underground mining methods. The cut and fill mining method, utilizing a mining cross-section of 5m high x 5m wide for the cut and a combination of uncemented rock backfill (75%), sized tailings backfill (20%), with a final cap of cemented sized tailings backfill (5%) for the fill.

Subsequent to toll milling in the early part of the LoM the proposed mill will process approximately 3Mt of ore from the pit and underground operations, and will generate an equivalent amount of tailings. A tailings facility will be constructed to contain 3Mt of solid tailings. A certain amount of tailings will be processed through a backfill plant for use in the underground mine operation.

LoM capital costs totaling US$121.3million are shown in Table 5. LoM Operating costs are shown in Table 6.

Table 5: Capital Cost Summary (US$000s)

Description	Total	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017
Equipment												
Mining – OP	**7,811**	0	0	3,045	262	952	1,751	1,757	44			
Mining - UG	**15,013**	0	0	0	0	0	0	0	9,200	5,813		
Process	**60,448**	0	0	0	15,112	45,336						
Infrastructure	**16,244**	0	0	2,329	115	6325	2,300	2,300	1,725	1,150		
Owner Cost	**(4,989)**	575	1,725	288	2,014	288						(7,864)
Total	**94,527**	**575**	**1,725**	**5,662**	**15,489**	**52,612**	**4,051**	**4,057**	**11,257**	**6,963**	**0**	**(7,864)**
Development												
OP Pre-stripping	**8,026**			8,026								
UG Develop.	**18,712**								2,554	6,164	6,324	3,670
Total	**26,738**	**0**	**0**	**8,026**	**0**	**0**	**0**	**0**	**2,554**	**6,164**	**6,324**	**3,670**

Table 6: Operating Cost Summary

Description	LoM Total (US$000s)	Unit Cost (US$/total-t)	Unit Cost (US$/ore-t)
Mining – OP	**$79,768**	$1.593/t	$23.72/t
Mining - UG	**$35,166**	-	31.75/t
Mine G&A	**$15,261**	-	$3.41/t
Toll Mill	**$42,504**	-	$32.57/t
Owner Mill	**$41,955**	-	$13.26/t
G&A	**$17,451**		$3.90/t
Total	**$250,817**		**$56.11/t**

The technical-economic results are based upon work performed by Apollo's engineers and consultants and has been prepared on an annual basis. The economic model used in this analysis, is pre-tax and assumes 100% equity. Basic economic assumptions used are discussed in detail throughout this report and are summarized in Table 7.

Table 7: Technical Economic Model Parameters

Model Parameter	Technical Input
General Assumptions	
Pre-Production Period	3 years
Mine Life	8 years
Operating Days per year	360 days/yr
Production Rate (avg.)	1,500t/day
Market	
Discount Rate	4%
Gold Price	US$525.00/oz
Refinery Charges	
Refining	US$2.50/oz
Transportation, Insurance & Assay	US$1,000/shipment
Royalty	
Private Royalty	none

A three-year pre-production period is assumed to allow for pre-stripping and mine development. The mine will have an estimated life of eight years given the resources described in this report and the assumed 540kt/yr production rate. The model assumes that gold will be toll milled in Year -01 through Year 02, with owner mill operation beginning in Year 03. Revenue from gold sales are based upon a market price of $525/oz. Gold treatment and refining charges are at $2.50/oz. Treatment, refining and transportation costs are charged against gross revenues. Transportation costs are as shown in the table and are calculated based upon 12 shipments per year.

The SRK LoM plan and economics are based on the following:

- A mine life of eight years, at a designed rate of 540kt/yr;
- An overall average metallurgical recovery rate of 96% Au, over the LoM;
- LoM cash operating cost of $50.69/t-milled; and
- Total capital costs of $121.3million being comprised of $26.7million for capitalized development and $94.5million for mine equipment. Mine closure cost is $2.6million and there is a mill salvage value of $9.4million in Year 09.

The base case economic analysis results, shown in Table 8, indicate a pre-tax net present value of $103.5million at a 4% discount rate with an IRR of 33%. Sensitivity analysis for key economic parameters are shown in Table 9. This analysis suggests that the project is most sensitive to market price. Operating costs are slightly more sensitive due to toll milling of ore in early years and time value of money effect of delaying the on-site mill in Year 03.

Table 8: Technical Economic Results ($000s)

Description		Technical Input or Result
Ore		
Open Pit		
Waste		46,940kt
Ore		3,362kt
Total		**50,302kt**
s/r		14:1
Grade		5.78g/t-Au
Contained Gold		625koz
Underground		
Total Development		11,975m
Ore		1,108kt
Grade		10.59g/t-Au
Contained Gold		377koz
Mill		
Ore Treated		
Toll Mill		1,305kt
Owner Mill		3,165kt
Total		**4,470kt**
Ore Grade		
Toll Mill		5.43g/t-Au
Owner Mill		7.61g/t-Au
Total		**6.97g/t-Au**
Contained Gold		
Toll Mill		228koz
Owner Mill		774koz
Total		**1,002koz**
Recovered Gold		
Toll Mill		219koz
Owner Mill		743koz
Total		**962koz**
Revenue ($000s)		
Gross Revenue		**$504,168**
Refining & Transportation Charges		$2,509
Net Smelter Return		**$501,659**
Royalty		$0
Gross Income From Mining		**$501,659**
Realized Price (Gold)	*US$522.39/oz-Au*	
Operating Cost ($000s)		
Open Pit Mine		($71,742)
Underground Mine		($35,166)
Mine G&A		($15,261)
Toll Mill		($42,504)
Owner Mill		($41,955)
G&A		($17,451)
Operating Costs		**($224,079)**
	US$233.34/oz-Au	
	US$50.13/t-milled	
Cash Operating Margin		**$277,580**
	US$289.05/oz-Au	
	US$62.10/t-milled	
Capital Cost		
Equipment		($94,527)
Development (Capitalized)		($26,738)
Total Capital		**($121,266)**
Cash Flow		**$156,314**
(NPV4%)		**$103,524**
IRR		**33%**

Table 9: Project Sensitivity ($NPV_{4\%}$, $000s)

Description	-10%	-5%	Base Case	+5%	+10%
Gold Price	$64,491	$84,376	**$103,524**	$122,673	$142,558
Operating Costs	$122,177	$112,851	**$103,524**	$94,198	$84,872
Capital Costs	$112,078	$107,801	**$103,524**	$99,248	$94,971

Apollo has progressed Black Fox to the point of confirming the gold ore resources and reserves as verified by the current SRK independent NI 43-101 report. Apollo should continue to build on the previous owners mining and known recovery of over 200,000oz of gold as well as the current ore reserves that could be recovered using proven open pit and underground mining techniques. The previous mining as well as known process systems that have been proven at Black Fox and other mines in the area demonstrates further justification to proceed with the project.

Recommendations

Black Fox should continue to be developed to the feasibility level. The following recommendations for the project should be considered by Apollo:

- Continue with the advanced feasibility level studies for the project including commissioning the bankable feasibility project as soon as possible;
- Continue to core drill specific areas of the ore body to further upgrade and extend the geological modeling for the project;
- Update Mineral Resources and Mineral Reserves to reflect preceding drilling;
- Feasibility level tailing impoundment design,
- Feasibility level detailed design for open pit and underground,
- Feasibility level geotechnical review for the open pit and underground project,
- Complete detailed process design based on flowsheet;
- Feasibility level process design and detail costs, and
- Develop a detailed trade off report on viability of using the toll mill as compared to the onsite mill.
- Continue the permitting process for Black Fox. Consideration should be given to permit the mine in phases with Phase 1 to include the toll mill proposal and permit only the open pit, overburden, and waste rock stockpiles.

Estimated cost for these recommendations is US$1.5million.

1 Introduction

1.1 Project Overview

SRK (U.S.) Inc., (SRK) was commissioned by Apollo Gold Corporation (Apollo) to prepare a National Instrument 43-101 compliant (NI 43-101) Prefeasibility Study (PFS) of the Black Fox open pit and underground gold project (Black Fox) in Timmins, Ontario, Canada.

Black Fox is located approximately 10km east of the town of Matheson, Ontario, Canada along the east-west trending 200km Destor-Porcupine Fault Zone (DPFZ).

The Glimmer underground gold mine operated on the Black Fox property over the period 1997-2001, and produced approximately 211koz of gold by contract milling in either the St. Andrews Goldfields Ltd-Stock Mill and Kirkland Lake Gold Inc-Macassa Mill.

Underground mining extended to depths of approximately 200m to 250m below the surface before operations were suspended in May of 2001.

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 have 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.

1.2 Terms of Reference and Purpose of the Report

This PFS is intended for the use of Apollo for the further development and advancement of Black Fox towards the production stage. This report meets the requirements for NI 43-101 and the Resource and Reserves definitions are 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.

1.3 Reliance on Other Experts

SRK's opinion contained herein is based on information provided to SRK by Apollo throughout the course of SRK's investigations. The sources of information include data and reports supplied by Apollo personnel, as well as documents included/referenced in Section 20.

The Qualified Persons preparing and supervising this PFS 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 PFS.

1.3.1 Sources of Information

SRK used its experience to determine if the information from previous reports was suitable for inclusion in this PFS and adjusted information that required amending. Revisions to previous data were based on research, recalculations and information from other projects. The level of detail utilized was appropriate for this level of study.

1.4 Qualifications of Consultants (SRK)

The SRK Group is comprised of over 700 staff, offering expertise in a wide range of resource engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This permits SRK to provide its clients with conflict-free and objective recommendations on crucial judgment issues.

SRK has a demonstrated record of accomplishment in undertaking independent assessments of Mineral Resources and Mineral Reserves, project evaluations and audits, technical reports, and independent evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

This PFS has been prepared based on a technical and economic review by a team of consultants sourced principally from the SRK Group's Denver, US office. These consultants are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, underground mining, mineral processing and mineral economics.

Neither SRK nor any of its employees and associates employed in the preparation of this report has any beneficial interest in Apollo. SRK will be paid a fee for this work in accordance with normal professional consulting practice.

The individuals who have provided input to this PFS have extensive experience in the mining industry and are members in good standing of appropriate professional institutions. Richard F. Nanna, Senior Vice President-Exploration, is the QP for the preparation of Sections 5 through 9, and 13. David Young, previously employed by Apollo as Vice President of Business Development and Technical Services during March 2004 through October 2005, visited the property on various occasions from March 2004 though October 2005 for over 30 days and is the Qualified Person (QP) for the preparation of Section 17 and overall PFS. Bart Stryhas of SRK, is the QP for the preparation of Section 15 of this PFS. The key project personnel contributing to this report are listed in Table 1.4.1. The Certificate and Consent forms are provided in Appendix A.

Table 1.4.1: Key Project Personnel

Company	Name	Discipline
SRK	Bart Stryhas, CPG, PhD	Resource Estimation
	Dorinda Bair, BS Geo	Geology
	Alva Kuestermeyer, MS Mineral Economics, SME	Process and Infrastructure
	David Young, P.E.	Mining
	Nick Michael BS Mining, MBA	Project Economics
Apollo	Richard F. Nanna, C.P.G.	Geology
	Jeff Choquette	Mining

2 Property Description and Location

2.1 Property Location

Black Fox is located 10km east of Matheson, Ontario, along Hwy 101 East and approximately 655km 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'N latitude and 80°21'W longitude. The Glimmer underground mine, formerly operated by Exall is located on the property. Figures 2-1 and 2-2 show the property location (Prenn, 2006).

2.2 Land Area

The property includes approximately 939ha of land of which, 75ha are unpatented federal land, 563ha are owned by Apollo, 175ha where Apollo only has surface rights, and 126ha where Apollo only has mineral rights but no surface rights. The Black Fox claim map is shown in Figure 2-3 and Table 2.2.1 provides a list of the current land position.

Table 2.2.1: Current Black Fox Project Property Summary

Township	Concessions	Lot #	Parcel	Acreage	Hectares	Status
Hislop	6	4	3393	81	33	Owned by Apollo
Hislop	6	5	11511	160	65	Owned
Hislop	6	6	6413	82	33	Owned
Hislop	6	6	2582	161	65	Owned
Hislop	6	7	388	147	59	Owned
Hislop	6	7	15466	97	39	Owned
Hislop	6	7	4707	46	19	Owned
Hislop	6	8	7745	164	66	Owned
Beatty	1	6	14572	152	61	Owned
Beatty	1	5	24577	156	63	Owned
Beatty	1	8	4150	148	60	Owned
Subtotal				*1392*	*563*	*Owned*
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Beatty	1	5	24577	39	16	Surface Rights
Subtotal				*432*	*175*	*Surface Rights*
Beatty	1	7	3524	156	63	Mineral Only
Beatty	1	7	11720	156	63	Mineral Only
Subtotal				*311*	*126*	*Mineral Only*
Beatty	1	6	L-1115059	41	17	Claim
Hislop	6	5	L-1048333	41	17	Claim
Hislop	6	5	L-1048334	41	17	Claim
Hislop	6	7	L-1113087	22	9	Claim
Hislop	6	6	L-1048335	41	17	Claim
Subtotal				*185*	*75*	*Unpatented*
Total Land				**2,321**	**939**	**All Types**

2.3 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.

Although Apollo has kept all of their claims current and up to date as far as fee and work commitments, one claim is in dispute over who has control of the property. The claim in question was staked by a party who believed that Apollo allowed their rights to the claim to expire. Documentation provided by Apollo indicates that the claim is still in their control. This dispute is currently before the Ministry of Northern Development and Mines (MNDM) who has jurisdiction in this matter. Apollo has evaluated the location of the claim relevant to the mineralized zone of the project and determined that the status of this claim will have no affect on project construction and mining. SRK agrees with the analysis by Apollo and that the project can be constructed and mined with or without the inclusion of this unpatented claim.

2.4 Agreements and Encumbrance

The Black Fox property was purchased by Apollo from the Glimmer Mine Joint Venture. The Glimmer Mine Joint Venture included Exall and Glimmer Resources. The Glimmer Mine (now Black Fox) was transferred on September 7, 2002. As of that date, Apollo owns 100% of Black Fox, which includes all existing infrastructure and buildings on the property. The Glimmer Mine Joint Venture was paid CND$3,159,000 and 2.08M shares of Apollo stock. An additional cash payment of CND$3million was made in January 2006.

Apollo has 100% ownership of the property and all mineral, surface and forestry rights and there are no production royalties. The property is currently on care and maintenance. Exploration activities are being conducted to address further mineral potential of the property (Prenn, 2006).

Adjacent land that has been purchased by Apollo is as follows:

- The Ewen property, consisting of sections 11511 and 3393 in the Hislop Township, was purchased in November 2003 for CDN$180,000. No resources or reserves are currently contained on the Ewen property, but if found would be subject to a 3% NSR Royalty. A CDN$500 annual minimum royalty applies to the property. The seller has first right of refusal to the property after mining and reclamation has been completed (Prenn, 2006);
- The Durham property, consisting of section 4707 in the Hislop Township, was purchased in February 2003 for CDN$100,000 with a “Buy-Out Payment” in the amount of CDN$2,000,000 to purchase out the NSR rights. An “Advanced Royalty Payment” of CDN$20.000 will be paid annually until the NSR “Buy-Out Payment” is made if and when such a payment is made;
- The Plouffe property, consisting of section 7745 in the Hislop Township, was purchased in January 2003 for CDN$100,000 with a “Buy-Out Payment” in the amount of CDN$1,000,000 to purchase out the NSR rights. No resources or reserves are currently contained on the Plouffe property, but if found would be subject to a sliding scale NSR detailed as such:
  - Less than US$200-No Royalty,
  - From $200 to $224.99-0.25%,
  - From $225 to $249.99-0.50%,

  - From $250 to $274.99-0.75%,
  - From $275 to $299.99-1.00%,
  - From $300 t $324.99-1.25%,
  - From $325 to $349.99-1.50%,
  - From $350 to $374.99-1.75%,
  - From $375 to $399.99-2.00%,
  - From $400 to $424.99-2.25%,
  - From $425 to $449.99-2.50%,
  - From $450 to $474.99-2.75%,
  - From $475 to $499.99-3.00%, and
  - Above $500-3.25%.
- Apollo purchased several adjacent surface and mineral rights properties for CDN$100,000 from Timmins Forest Products. Parcel 15611 in this agreement includes mineral rights. This parcel is subject to a 2% royalty, with a royalty buyout provision of CDN$500,000. No resources or reserves are currently contained on the parcel (Prenn, 2006); and
- Adjacent lands recently purchased by Apollo. The Steinman property, consisting of section 4150 in the Beatty Township, was purchased in July 2007 for CDN$200,000. No resources or reserves are currently contained on the Steinman property, but if found would be subject to a 2.5% NSR. Apollo has a first right to buyout 1% of the NSR for CDN$1,000,000.

2.4.1 Stakeholders and Interested Parties

Stakeholders with authority of some nature at the property will include Apollo, 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 (Quebec) and Wahgoshig (Ontario) First Nations, and local private landowners in both Hislop and Beatty Townships. The Abitibi Indian Reserve 70 is located 25km east of the mine site. Table 2.4.1.1 lists the local private landowners described as stakeholders near Black Fox (Dyck, 2007).

Table 2.4.1.1: 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

2.5 Environmental Liabilities

There are no environmental liabilities at the Black Fox Mine Site. A bond of CDN$675,000 has been given to the MNDM for site remediation for previous mining activities at the Glimmer Mine.

2.6 Permitting and Completed Studies

Section 2.6 is excerpted from Apollo Gold Corporation Black Fox Project, Project Description for Small Pit & Mill Operation –Update-, Prepared for Ministry of Northern Development and Mines on behalf of Apollo Gold Corporation by AMEC Earth & Environmental, April 16, 2007 and has been standardized to this report.

2.6.1 Permitting

"In December 2004, Apollo submitted a Closure Plan for the existing conditions of the Black Fox Project site, which supersedes the unaccepted Hislop-Beatty Project Closure Plan that was previously submitted by Exall in 1996. A new Closure Plan will be developed for the proposed future 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 (C of A) 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 (PTTW) – (mine dewatering) 00-P-6025; and*
- *Waste Generator Registration ON2142400.*

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

2.6.2 Completed Engineering and Environmental Studies

Terrestrial Environment

Limited terrestrial investigations in the site area were conducted in 1994 and 1995 by AMEC (formerly AGRA, for another mining client). Studies were conducted by Beak in 1996, which focused on wetlands along Salve Creek and the shoreline of Froome Lake. Additional investigations were conducted by AMEC in 2005 to supplement the findings of both the AMEC and Beak studies in areas surrounding the mine site, Salve Creek and Froome Lake. As well, additional surveys have been undertaken to provide further details on terrestrial vegetation and wildlife in areas that may be affected by future mining activity, such as in the vicinity of the overburden and waste rock storage piles. Depending on the final detailed designs, additional studies may be undertaken.

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.

Aquatic habitat assessments undertaken in 2004 were based on data collection initiatives recommended in prior studies (Beak), in the context of the proposed project, and additional sampling of stream and lake sediments, water chemistry and benthic macroinvertebrates were also undertaken. As well, future aquatic assessment programs will be expanded to include areas that could potentially be affected by future mining activity. Of particular importance is the thorough assessment of potential fisheries habitat areas in the areas of proposed mine development.

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) potential site buildings and access road foundations, 3) the overburden stockpile, 4) the waste rock stockpiles, and 5) the tailings impoundment. Subsequent design stages will include a more extensive field program at the exact locations of the structures, and additional engineering analyses.*

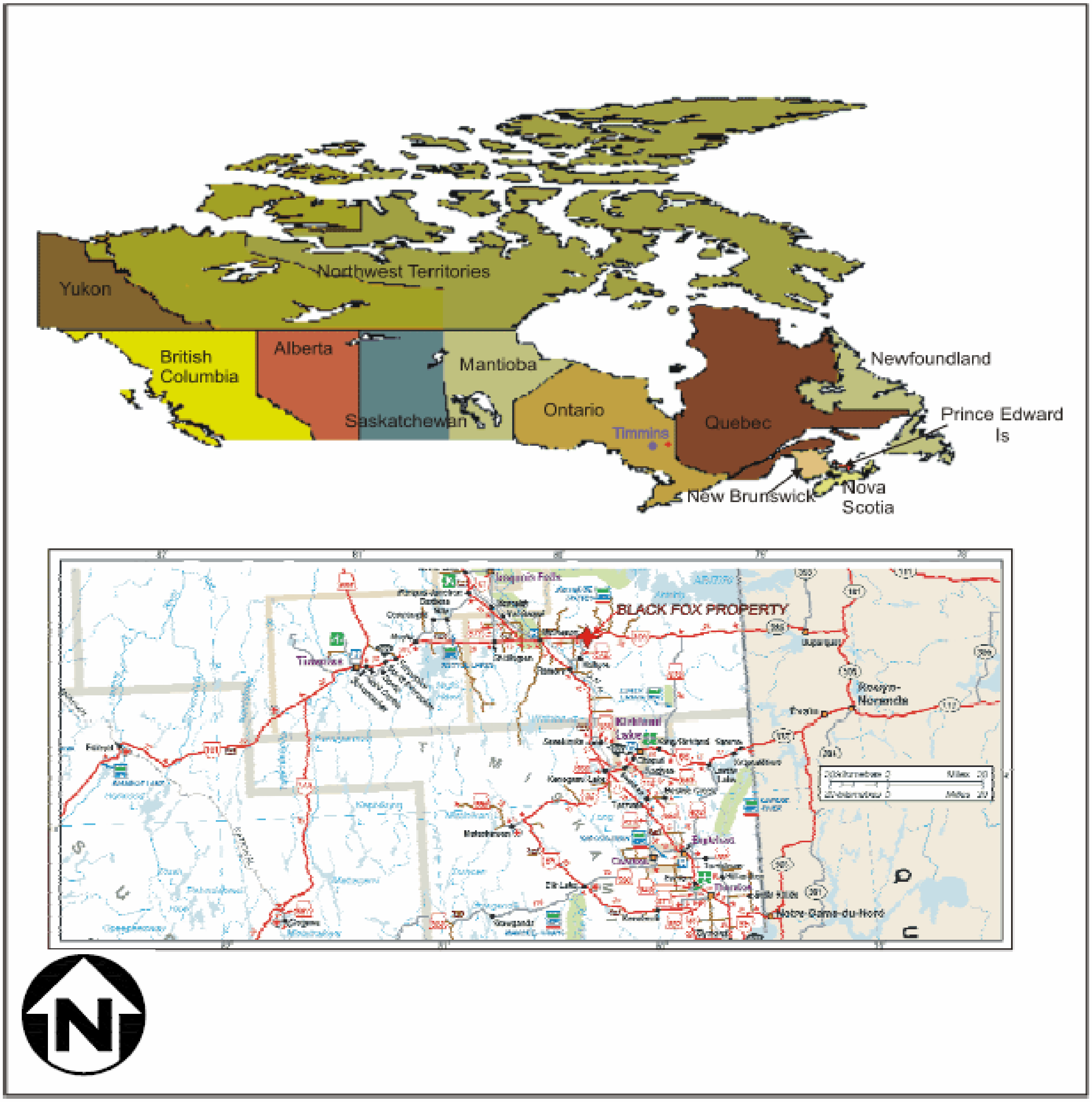
With respect to foundations for buildings and other structures, site services and access roads, geotechnical investigations have been designed to develop preliminary recommendations for potential foundation types, assess available bearing capacities at certain founding elevations, and to better understand expected 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) will have to consider the type of overburden, the location of the groundwater table, safe slope configurations, as well as run-off collection and management.

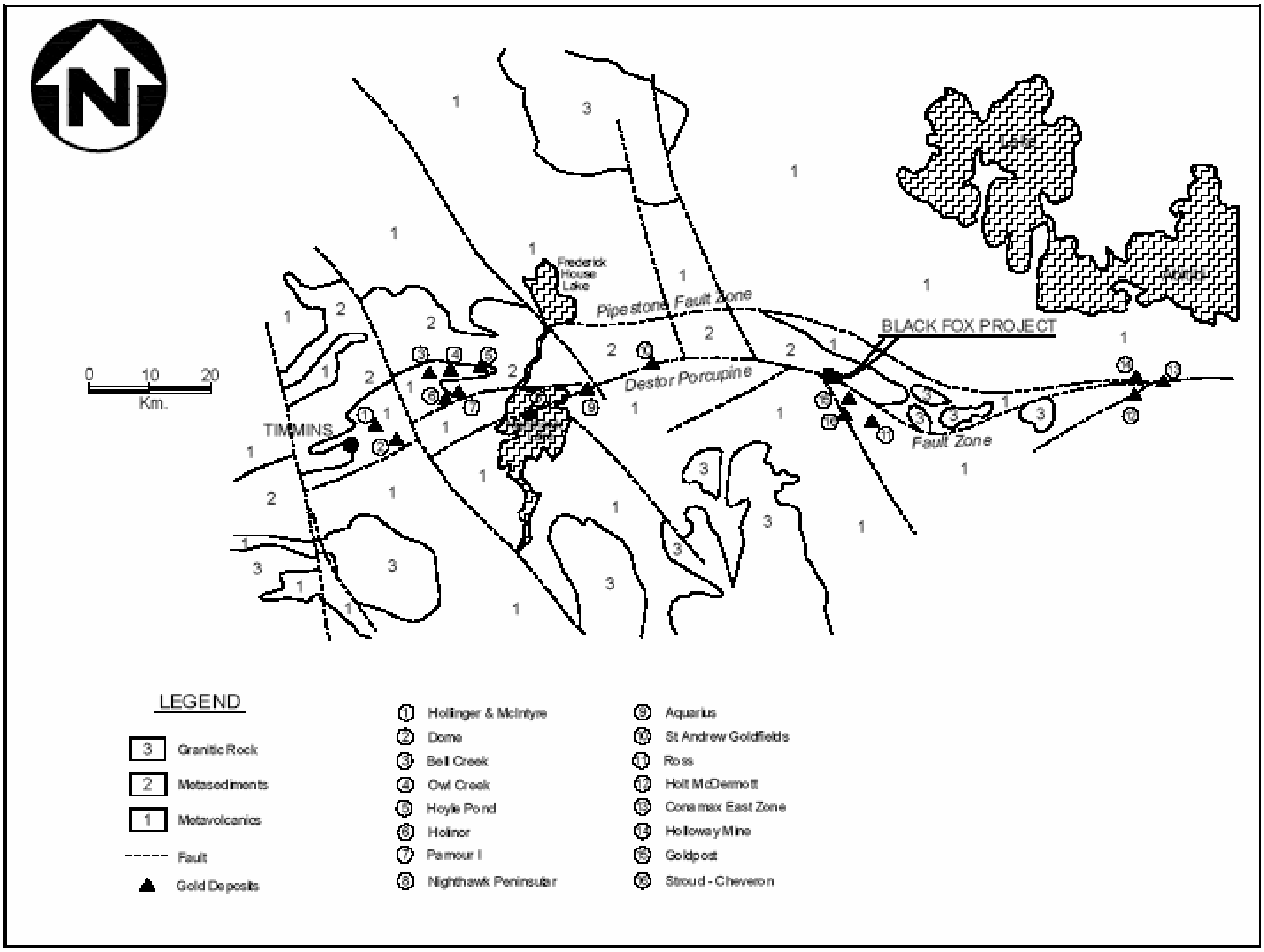
Geotechnical investigations in the vicinity of the proposed waste rock and overburden piles were intended to assess and clarify potential subgrade preparations for the placement of material to ensure long term stability. This information will be used to design travel routes, lift heights and slope configurations.

Waste Characterization Studies

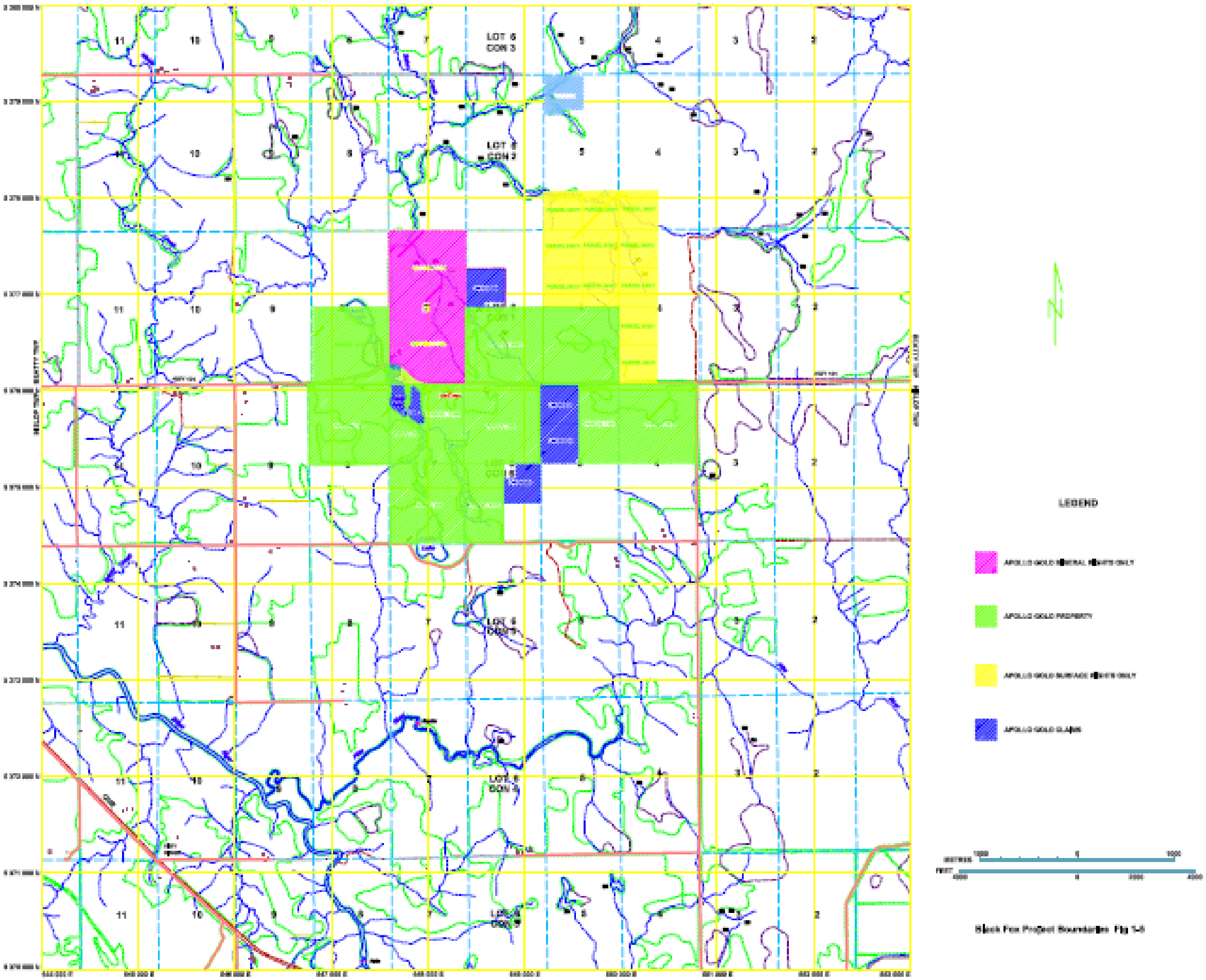
A comprehensive geochemical characterization of all mine waste materials has been completed to support the development of an integrated water and waste management plan for the site. In developing the mine model, waste and host rock materials have undergone a comprehensive geological classification to 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. The results of this study are reported under separate cover."



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Black Fox Location Map		
SRK Job No.: 144418	Source: Mine Development Associates			
File Name: Figure 2-1.doc		Date: 07-10-07	Approved: DKY	Figure: 2-1



SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Adjacent Mines Location Map**
SRK Job No.: 144418	**Source: Mine Development Associates**	
File Name: Figure 2-2.doc		Date: 07-10-07 / Approved: DKY / Figure: 2-2



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Black Fox Claim Map
SRK Job No.: 144418		
File Name: Figure 2-3.doc	Source: Apollo Gold Corporation	Date: 07-10-07 / Approved: DKY / Figure: 2-3

3 Accessibility, Climate, Local Resources, Infrastructure and Physiography

3.1 Access

Black Fox is located 10km east of Matheson, Ontario and 65km east of Timmins, Ontario, Canada. Access is via Highway 101 East, which crosses the Black Fox claimblock 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.

3.2 Climate

Temperature ranges from 20 to 33°C during the summer months and -30 to 10°C during the cooler winter months of October to May. The average precipitation is 873.4mm/yr and ranges between 44.5mm in February to 100.1mm 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 15km/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.

3.3 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 and rivers and their associated habitats. Lakes include Froome Lake located 0.7km west of the mine, Leach Lake located 1.4km northwest of the mine and Lawler Lake located 1.7km south. Two other lakes, Salve located 5.2km north and Nickel located 5.9km 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 330m above mean sea level (amsl) (Prenn, 2006).

3.4 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 3-1, consist of the following infrastructure:

- A trailer complex, complete with administration office, mine dry facilities, geological/engineering offices, and shower/washroom facilities;

- Site access roads;
- A septic tank and tile field;
- A pump house at the east side of Froome Lake, and associated 750m long pipeline for the taking of fresh water for showers/toilets and drilling purposes;
- A 13,500L fresh water tank, in an insulated metal clad shed (water tank house);
- A compressor station;
- A core log shack;
- A former surface maintenance shop;
- One 4,500L diesel storage tank;
- An approved mine water treatment system, consisting of a settling pond, a polishing pond, and a spillway pond (for emergency discharge purposes);
- An acid addition building (where sulphuric acid and ferric sulphate are added for pH adjustment and arsenic removal, respectively), associated with the mine water settling pond treatment system;
- A 2.6km long, 150mm dia. HDPE pipeline, to Salve Creek, associated with treated mine water effluent;
- A downcast fan, with mine air heater, and a 30,000L propane tank;
- A main hydropower line to the on-site 5,000kVA transformer substation and distribution lines;
- A mine portal; and
- Waste rock and ore storage pad areas.

The existing infrastructure at the site will be removed and new infrastructure will be constructed to facilitate the mine development (Dyck, 2007).

This following paragraph and Sections 3.4.1 through 3.4.5 is partly excerpted from the Technical Report Black Fox Project Matheson, Ontario, Canada by Mine Development Associates, August 14, 2006 and has been standardized to this report.

"An allowance has been included to develop the initial site access and haul roads. The project is located just off Highway 101, with the plant site, administration building, and mine shops located 200 to 300m south of the road. It should be noted that the pit is within 100m of highway 101, which is the desired distance from the highway in Ontario. The close proximity to the highway may cause the project to relocate a several hundred meter portion of the highway to stay within this guideline, however other mines in the district are within 100m of a highway. In addition, the slope design in the overburden materials has not been finalized, which when finalized, may position the pit closer or further away from the highway. This study assumed that the highway would not require relocation.

3.4.1 Power Distribution

Power is currently available to the project site. The lines are adequate to supply the anticipated 4 to 5MW requirement of the project. An allowance has been included for on-site power distribution.

3.4.2 Water Distribution

The make-up water requirement for the base case is about 50m^3/hr (about 400g/m). Most of the water required for the mill will be reclaimed from the tailings pond (240m^3/hr).

3.4.3 Administration Building

A 16 x 24m administration building has been included.

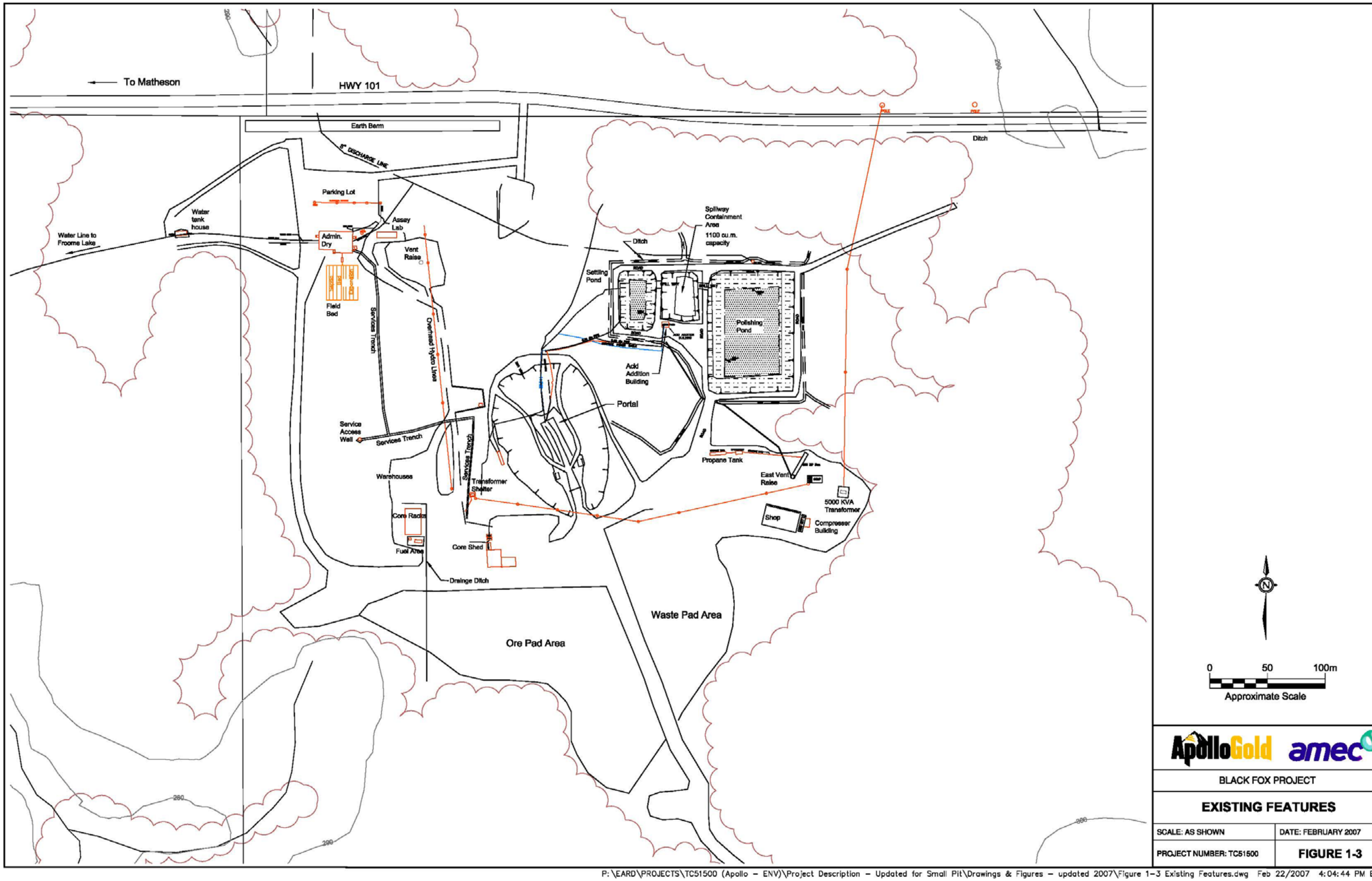
3.4.4 Truck Shop and Offices

A three-bay truck shop with wash bay and lube bay has been included in the study. The truck shop will have facilities to store mine equipment parts and contain offices and change room for the maintenance personnel. A fenced outside storage area is also included. The equipment ready line will be equipped with electrical plugs. Mine equipment will have hot start engine oil heaters. The shop and warehouse building will be 24 x 36m in size.

During the first three years of mining fuel consumption will average around 120,000L per week. Storage for 160,000L of diesel fuel has been included as part of the mine shop facilities. Storage for lube and hydraulic oils has also been included.

3.4.5 Pit Office

A 7 x 12m pit office will contain offices and change area for mine personnel."



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Existing Features		
SRK Job No.: 144418				
File Name: Figure 3-1.doc	Source: Apollo Gold and AMEC	Date: 08-07-07	Approved: DKB	Figure: 3-1

4 History

4.1 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 God Mines Inc. (Hemlo). Exall purchased the property from Hemlo in April 1996, obtaining approximately 60% interest in the property with Glimmer retaining 40%. Apollo acquired a 100% ownership in the fall of 2002 and renamed the property "Black Fox" (Prenn, 2006).

4.2 Exploration History

4.2.1 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 27,800m of drilling in 142 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 250m 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 720 underground diamond drillholes with mine development (Dyck, 2007).

4.2.2 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 ElectroMagnetometer (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).

4.3 Historic Resource and Reserve Estimates

The historic resource estimates are summarized in Table 4.3.1. 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 4.3.1. Table 4.3.2 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 4.3.1: Historic Resource Estimates

	Measured			Indicated			Total Measured and Indicated			Inferred			
Year	kt	Au Grade (g/t)	Au (koz)	kt	Au Grade (g/t)	Au (koz)	kt	Au Grade (g/t)	Au (koz)	kt	Au Grade (g/t)	Au (koz)	Estimator
1994							727	11.30	264				Hemlo (Jarvi)
1996							551	11.52	204				Roscoe Postle
1996							678	11.30	246				Roscoe Postle
1998	44	4.84	7	154	5.58	28	198	5.42	34	382	10.33	127	Exall
1999	410	7.27	96	796	8.20	210	1,205	7.88	306	274	5.96	52	Exall
2000	586	6.93	131	1,022	7.36	242	1,608	7.20	372	381	6.65	81	Exall
2001	268	4.09	35	566	4.93	90	833	4.66	125	353	7.00	79	Exall
2006										7,854	4.89	1.2	MDA

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

Table 4.3.2: Historic Reserve Estimates*

	Proven			Probable			Total Proven & Probable			
Year	kt	Au Grade (g/t)	Au (koz)	kt	Au Grade (g/t)	Au (koz)	kt	Au Grade (g/t)	Au (koz)	Estimator
1996				499	11.14	179	499	11.14	179	Canadian Mine Development (Feasibility)
1996				477	10.70	164	477	10.70	164	Bharti Engineering Associates
1996				621	11.60	232	621	11.60	232	Roscoe Postle
1997				665	12.90	275	665	12.90	275	Roscoe Postle
1998	330	9.88	105	488	10.32	162	818	10.14	267	Exall
1999	284	8.48	77	553	9.50	169	837	9.15	246	Exall
2000	422	7.82	106	560	8.93	161	981	8.45	267	Exall
2001	303	8.45	82	475	9.21	141	778	8.92	223	Exall
2006				3,063	4.56	449	3,063	4.56	449	MDA

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

4.4 Production History

Ore mined from Black Fox was custom milled from 1997 through September 1999 at the St. Andrews Goldfields Stock Mill located 34km 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 3.1Mt with a grade of 4.6g/t-Au (449koz of gold) all from open pit mining (Prenn, 2006).

The open pit total waste is 47.2Mt of waste rock and overburden material with an equivalent overall strip ratio of 15.4 waste: 1 ore. The underground ore resources (below 9815m) were 1.6Mt with a grade of 8.1g/t-Au.

Table 4.4.1 summarizes the reported gold production of 210.8koz from the Black Fox property, with the grades required at 100% recovery. Figure 4-1 illustrates several views of the underground workings of the mine at end of year 2000.

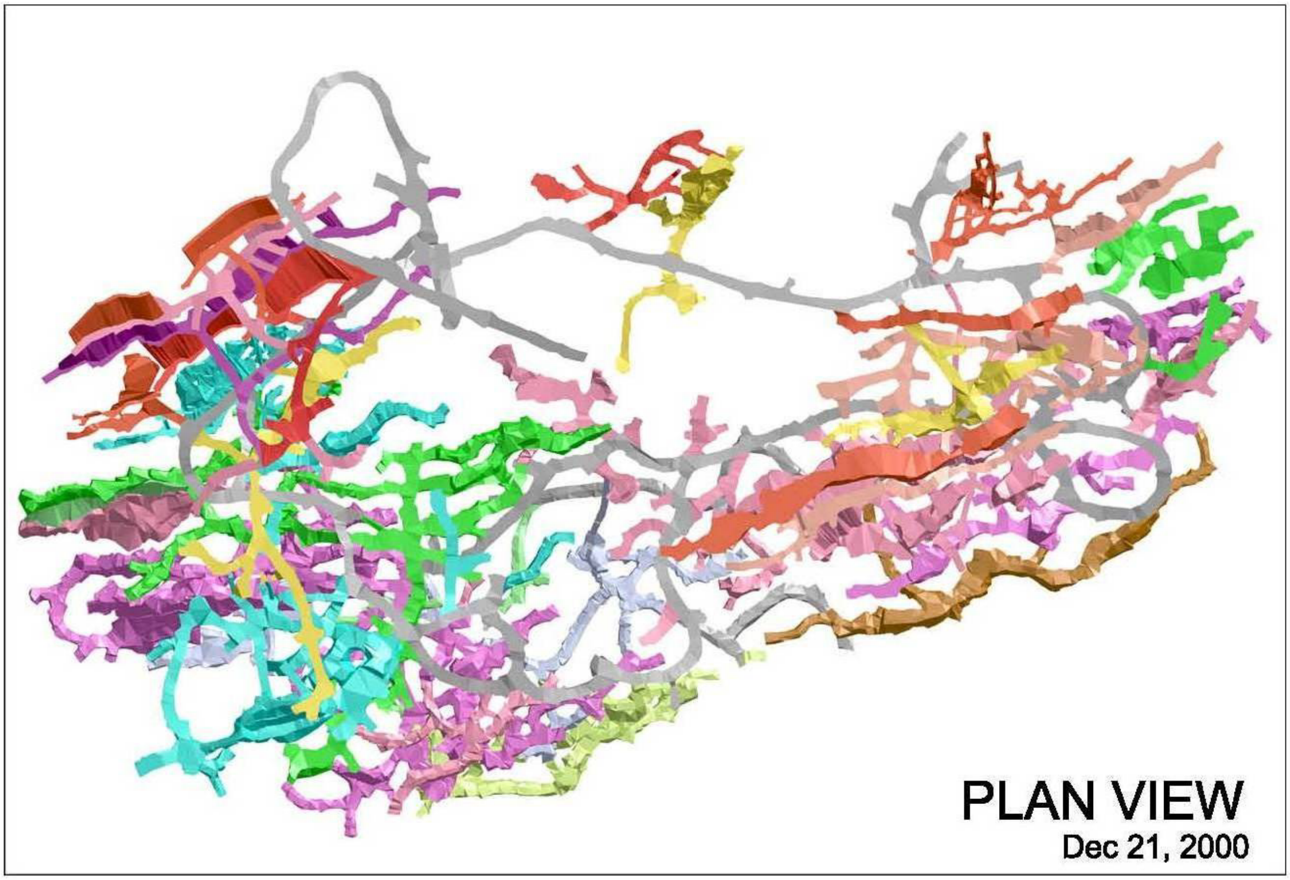
Table 4.4.1: Black Fox Project Production History*

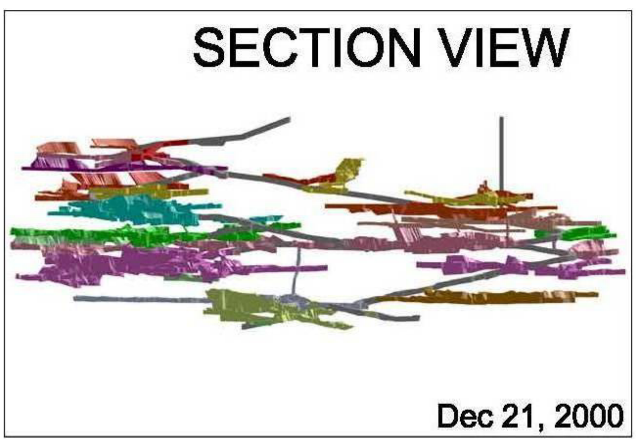
Year	kt	Au Grade (g/t)	Au (koz)
1997	194	6.79	40
1998	309	6.67	64
1999	259	5.82	48
2000	255	5.82	46
2001	82	4.81	12
Total	**1,099**	**5.97**	**211**

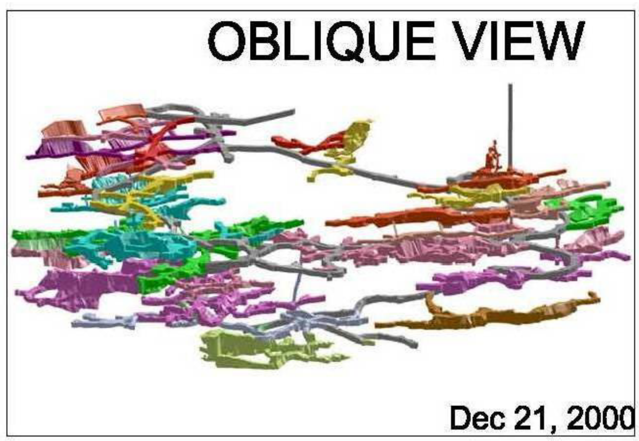
*Actual reported production.

Exall mined portions of the deposit from the bottom of the crown pillar to the 225m level (measured vertically 225m 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 4.3.2 to historic production in Table 4.4.1 shows that the grade and tonnage estimates are not very close to the actual production of about 1.1Mt with and average grade of approximately 6g/t-Au. The estimates between 1996 and 1997 show a range from 162koz to 275koz of gold. In 2001 the reserve estimate was 140koz, 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).







SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Black Fox Underground Workings**
SRK Job No.: 144418	**Source: Mine Development Associates**	
File Name: Figure 4-1.doc		Date: 07-10-07 / Approved: DKY / Figure: 4-1

5 Geologic Setting

5.1 Regional Geology

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

5.1.1 Lithology

The geological position of Black Fox is interpreted to be on the southern limb of a large-scale anticline. At Black Fox, the axial plane of this structure strikes roughly NW-SE. Regionally the geology consists of these six groups:

- Blake River Group;
- Kinojevis Group;
- Stoughton-Roquemaure Group;
- Kinojevis Group;
- Hunter Mine Group; and
- 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. The lower boundary of this group is with 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 top of the sequence is defined by the appearance of komatiitic and tholiitic lavas of the Stoughton-Roquemaure Group. Pre- to syn-kinematic granitic rocks occur throughout the area, cross-cutting all older lithologic and stratigraphic units. The Porcupine Group of wacke, siltstone and argillite sediments are the youngest in the region but are in fault contact with the above mentioned volcanic groups. This group lies north of the Black Fox Mine (Hoxha and James, 2007).

5.1.2 Structure

The main structural feature on the property is the intersection of the DPFZ with the Ross Mine Syenitic Belt (RMSB). The DPFZ is a regional structural zone, which has numerous gold deposits spatially associated with it in northeastern Ontario and northwestern Quebec. It was first recognized in the early 1900s with the discovery of gold deposits in the Timmins area. The DPFZ is traceable over a distance of at least 200km, from Timmins in the west, to the Duparquet area of Quebec to the east. It is of the same magnitude and significance, with respect to gold mineralization, as the Cadillac and Casa Berardi Fault Zones to the North. The faults trend east to southeast and dip to the south. They are deeply rooted regional trends that probably penetrate to the mantle, as indicated by the associated ultramafics of the DPFZ and the Syenites of the 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. Figure 5-1 illustrates the regional geology (Hoxha and James, 2007).

5.2 Local Geology

Most of the Black Fox area is rather flat and lacking in outcrops. Pleistocene overburden averages 20m thick and is composed of lacustrine clay, gravel and till. A variably sheared, faulted, carbonatized and mineralized sequence of komatiitic ultramafic volcanics, belonging to the Stoughton-Roquemaure Group strikes southeast across the property, along the southeast strike of the DPFZ. This structure and the surrounding stratigraphy dip to the southwest at approximately 45^{o}. These altered and deformed komatiites are generally bleached to a light grey-buff color with ankerite-talc and ankerite-quartz-sericite-fuchsite assemblages. This alteration package is underlain to the north by a thin, fine grained, green greywacke-type sedimentary unit, a thick sequence of massive to pillowed tholeiitic mafic volcanic rocks and lastly by the regionally extensive package of argillites and wackes of the Porcupine Group sediments which underlie the northeastern portion of the property (Hoxha and James, 2007).

To the south and forming the hangingwall 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 (Hoxha and James, 2007).

Very narrow 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 5-2 illustrates the local geology, Figure 5-3 shows the Tectono-Stratigraphic column of the Black Fox area and Figures 5-4 through 5-6 illustrate typical cross sections through the deposit (Hoxha and James, 2007).

5.3 Mine Geology

Mine geology at Black Fox is broken down into five general types:

- Mafic volcanic units;
- Sediments (meta-sedimentary rocks);
- Pyrochlastic fuchsite-carbonate schist;
- Ultramafic volcanics; and
- Felsic intrusive units.

5.3.1 Mafic Volcanic Units

The mafic volcanic units are further broken down into massive mafic volcanics (MV), pillowed mafic volcanics (PMV) and bleached mafic volcanic flows (BMV). The MV and PMV are fine grained and show a significant degree of chlorite alteration. These units dominantly occur in the hangingwall of the deposit but are also sometimes found in the footwall. In the hangingwall, these units are fractured and contain minor amounts of quartz-calitie veins while MV and PMV found in the footwall have more prevalent quartz veining and chlorite alteration (Hoxha and James, 2007).

The BMV, also known as flow ore, is a medium to fined grained, basal mafic volcanic rock which is generally located along the footwall of the deposit. This unit has weak chlorite and sericite alteration and is associated with fine grained disseminated pyrite. Stronger sericite alteration found near the upper contact of the BMV, is directly correlated to increased pyrite. 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 in this unit. The first is a pre-tectonic vein set parallel to the foliation and the second set is a series of late veins perpendicular to the foliation (Hoxha and James, 1998).

5.3.2 Sediments (Meta-sedimentary Rocks)

The sedimentary rocks overlie the BMV and also occur as lens of greywacke (SED) within the green carbonate schists (CGR). The CGR is discussed below. At the top of the BMV, the greywacke layers are interbedded with siltstone. This unit is discontinuous, varies from 5cm to 1m and displays graded bedding with stratigraphic tops to the southwest. The upper contact is often in contact with the CGR and the sedimentary rocks are pale green to yellowish in color with well developed sericite alteration. This sercite alteration extends over 1 to 2m and is of economic interest for gold exploration (Hoxha and James, 2007).

Greywacke lens occurring within the CGR are yellowish in color and strongly altered to sercite. Generally, these lens are less than 2m thick, but in places may be 4m thick. Where these lens exceed 4m in thickness, the sericitic alteration does not penetrate the entire layer, but is confined to the contacts (Hoxha and James, 2007).

5.3.3 Pyroclastic Fuchsite-carbonate Schist

The pyroclastic fuchsite-carbonate schist, also referred to as the green carbonate schist or CGR, is continuous in strike and dip across the property in zones ranging from 15 to 75m thick. It is characterized by an intense history of deformation seen in multiple generations of foliation and veining. The CGR is composed of a quartz-ankerite-fuchsite-leucoxene assemblage with varying intensity of chlorite alteration. This unit also contains felsic dike fragments the most abundant of which are of syenitic composition. A complex stockwork of quartz-ankerite veins cross cut the main CGR fuchsite assemblage and the felsic dike fragments. This stockwork is accompanied by intense hydrothermal alteration. Some yellowish, strongly albitized zones have been recognized within this unit. Locally, grey carbonate fragments, from lapilli sized (~2mm) to 1 to 2m angular blocks are found in the CGR. The lapilli sized fragments are elliptical in shape and elongate parallel to foliation. Mineralogy, microscopic texture and structures suggest that the CGR is an ultramafic pyroclastic rock which has undergone intense deformation. Medium to coarse grained pyrite is a minor component of the CGR and is estimated at approximately 1%.

Gold is found 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).

5.3.4 Ultramafic Volcanics

The ultramafic volcanic rocks are divided into five units. These are 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. This unit does not display pervasive carbonate alteration and carbonate is restricted to late veins and fractures. Tremolite is often present with the two primary mineral assemblages being 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 believed to be of economic significance (Hoxha and James, 1998; 2007).

The TUV is pale green-grey, fine grained, marbled with quartz-ankerite fragments and massive. It tends to be strongly foliated proximatel to shear zones. The TUV ranges from 0.3 to 15m thick and is most often associated with the stockwork grey carbonate (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 to 2m thick, generally occurs above the CGR and is bounded 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 grey, the CGY and the SUV. The SUV is very similar in appreat to the CBY, but the SUV is harder due to silicification. Two types of carbonatization-silicification have been observed at Black Fox:

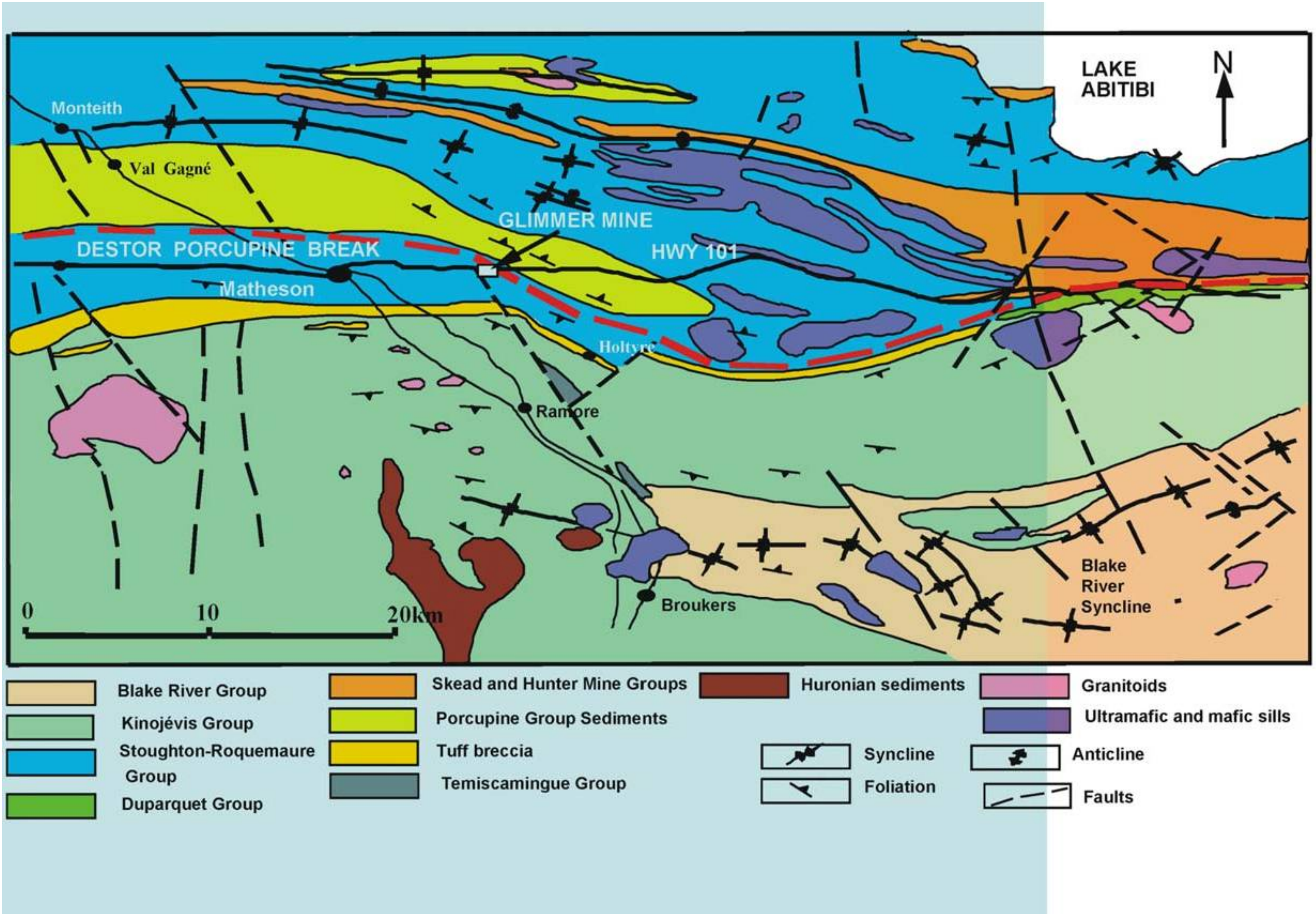
- Impregnation of the original ultramafic volcanic rock by CO_2 and silica-rich fluids throughout the network of micro-fractures and cavities/porosity; and
- Silicification of the altered ultramafic volcanic rock by silica-rich fluid circulating throughout rectilinear centimeter-wide extensional fractures 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 significance to the Black Fox project (Hoxha and James, 1998; 2007).

The AUV is dark green-brown, fine–medium grained rock composed of a quartz-ankerite-clacite-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 associate with quartz-ankerite stockwork. This unit is bound by a steeply dipping, 70°+ fault zone to the central-eastern region of the mine. The AUV generally occurs above the CGR and is one of the dominant rock types at Black Fox (Hoxha and James, 1998; 2007).

5.3.5 Felsic Intrusive Units

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



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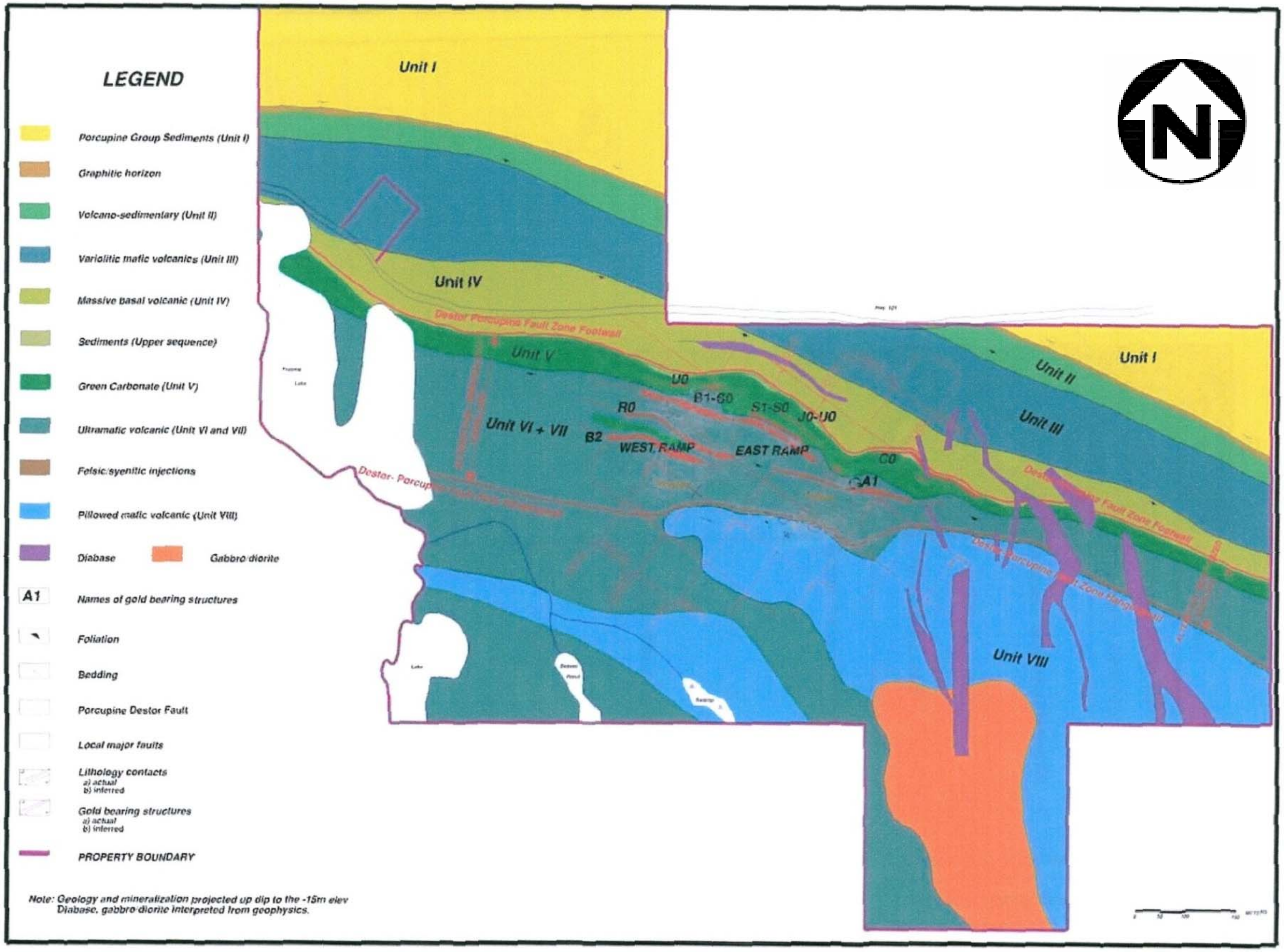
File Name: Figure 5-1.doc

Black Fox,
Timmins, Ontario, Canada

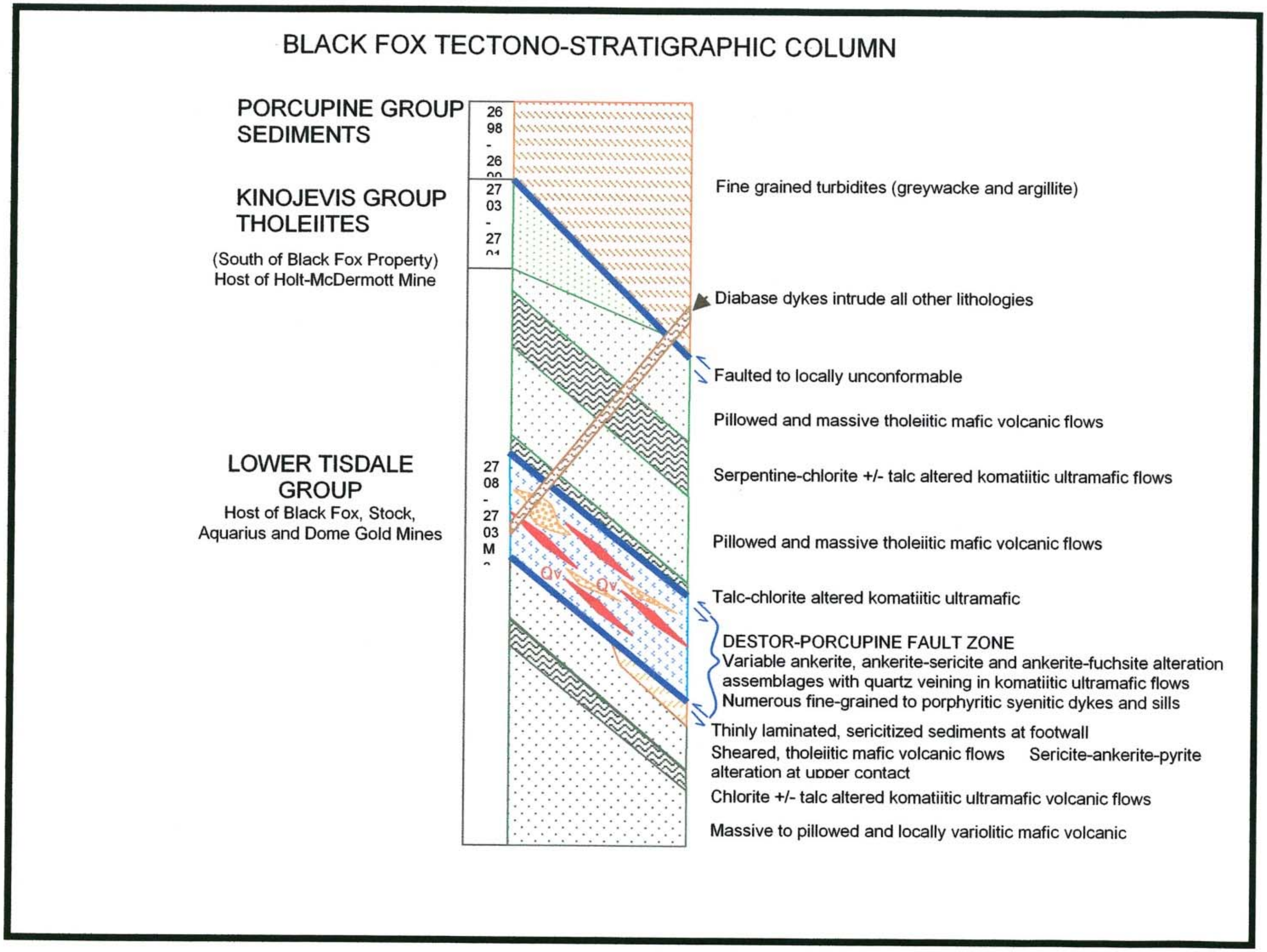
Source: Hoxha and James; 1998, 2007

Regional Geology

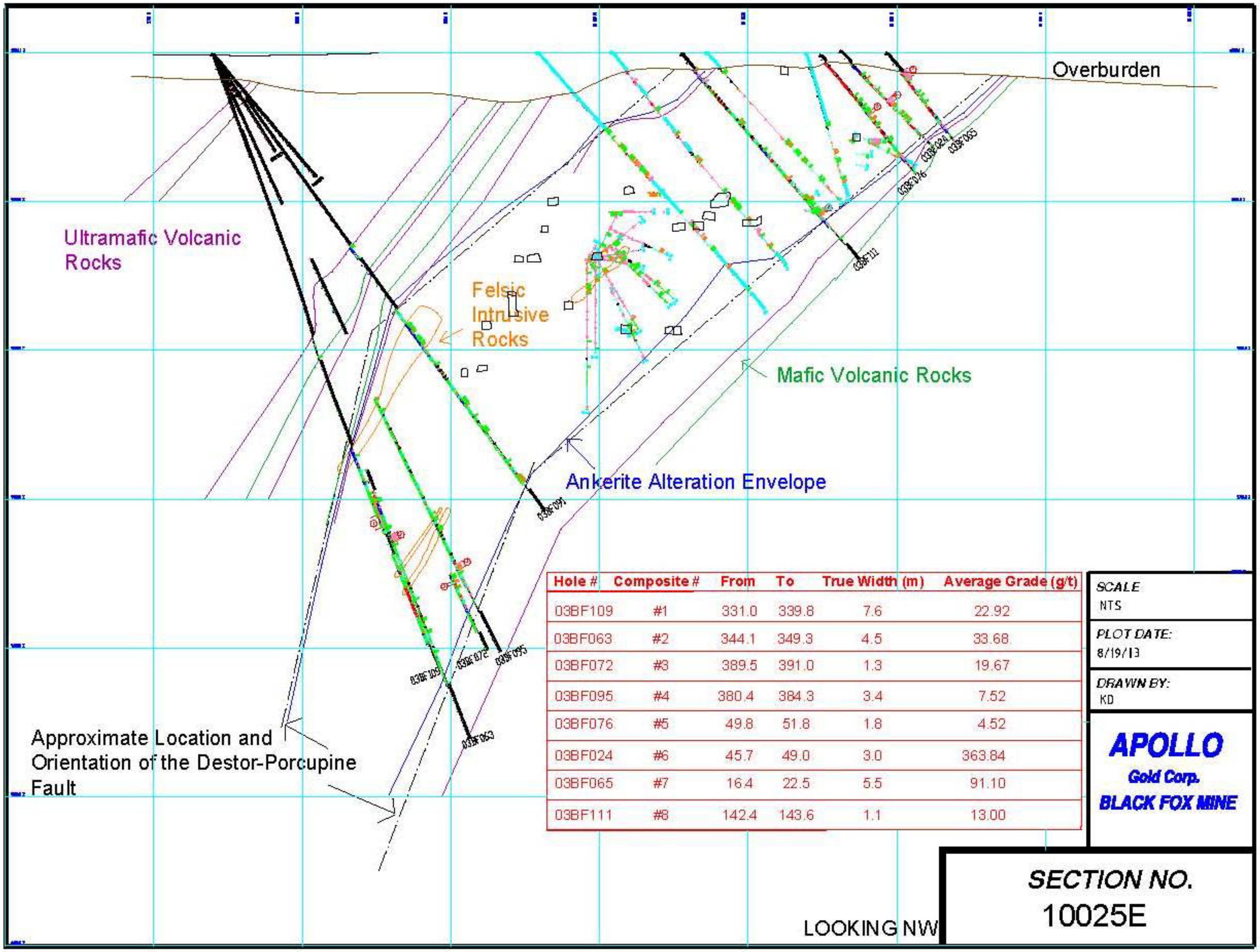
Date: 07-17-07	Approved: DKY	Figure: 5-1



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Local Geology
SRK Job No.: 144418	Source: Hoxha and James; 1998, 2007	Date: 07-17-07 / Approved: DKY / Figure: 5-2
File Name: Figure 5-2.doc		



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	**Black Fox Tectono-Stratigraphic Column**
SRK Job No.: 144418		
File Name: Figure 5-3.doc	**Source: Hoxha and James 2007**	Date: 07-17-07 / Approved: DKY / Figure: 5-3



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

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SRK Job No.: 144418

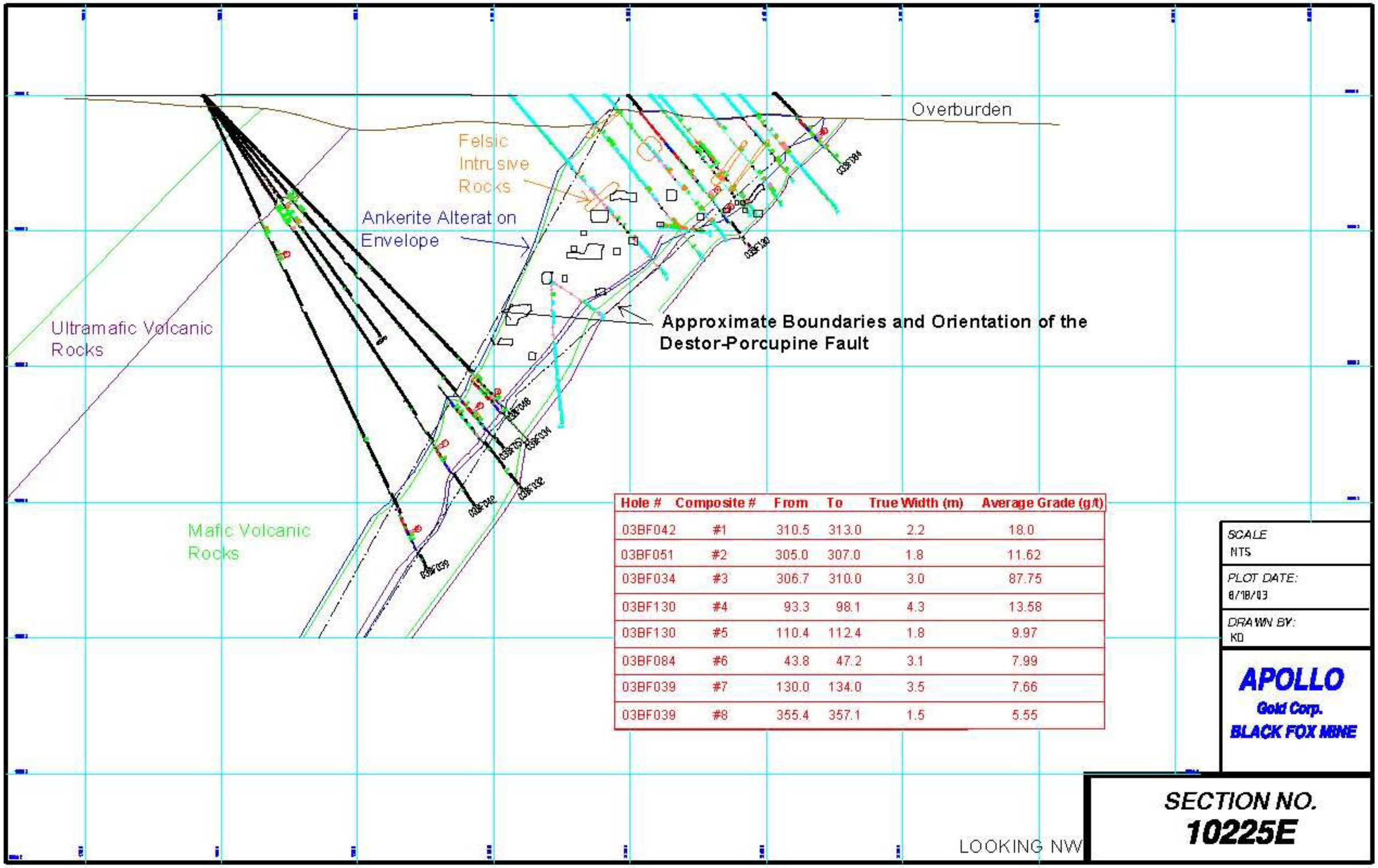
File Name: Figure 5-4.doc

**Black Fox,
Timmins, Ontario, Canada**

Source: Mine Development Association

Typical Cross Section 10025E

Date: 07-10-07 | Approved: DKY | Figure: 5-4



Hole #	Composite #	From	To	True Width (m)	Average Grade (g/t)
03BF042	#1	310.5	313.0	2.2	18.0
03BF051	#2	305.0	307.0	1.8	11.62
03BF034	#3	306.7	310.0	3.0	87.75
03BF130	#4	93.3	98.1	4.3	13.58
03BF130	#5	110.4	112.4	1.8	9.97
03BF084	#6	43.8	47.2	3.1	7.99
03BF039	#7	130.0	134.0	3.5	7.66
03BF039	#8	355.4	357.1	1.5	5.55

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SRK Job No.: 144418

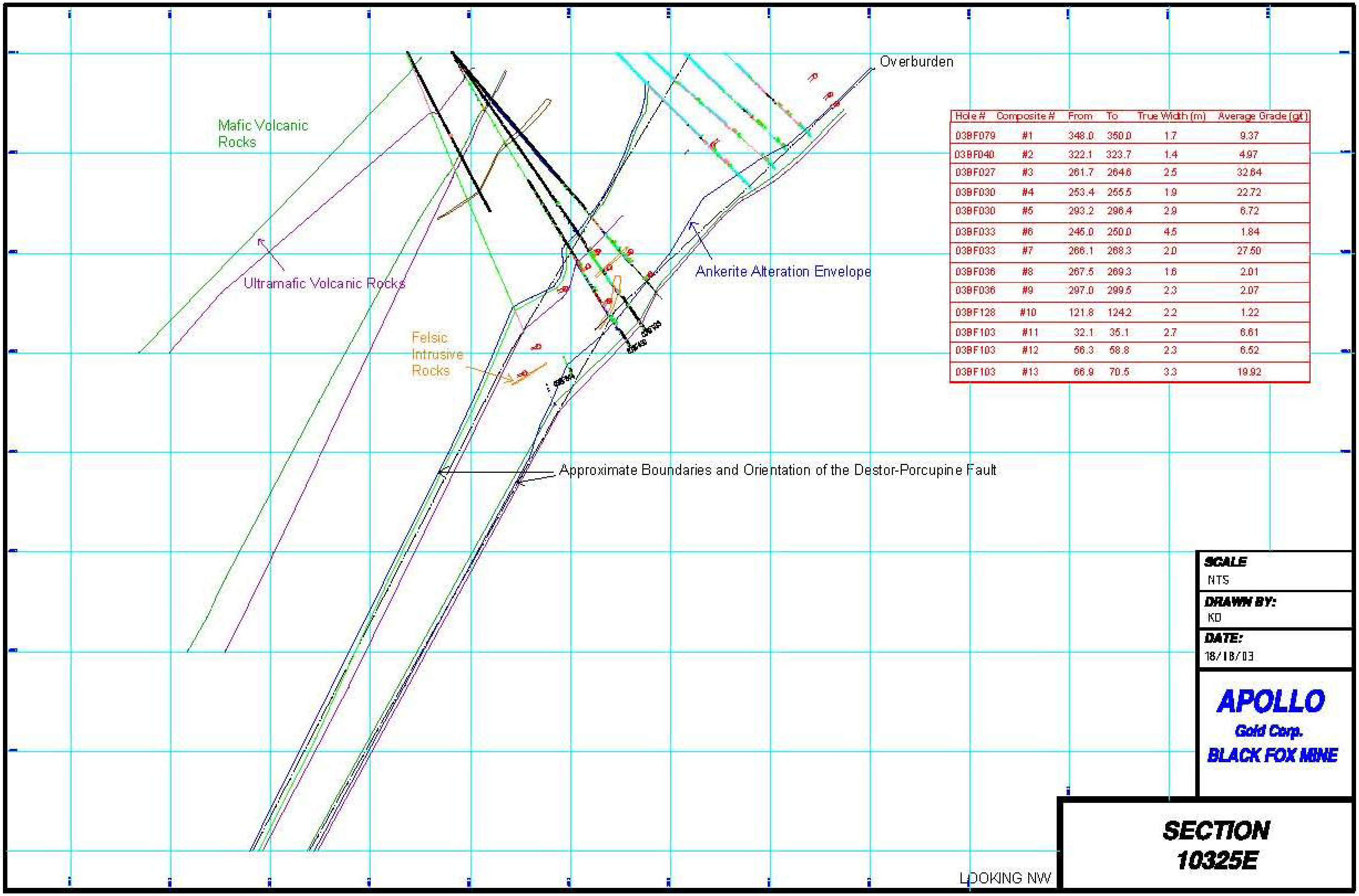
File Name: Figure 5-5.doc

Black Fox, Timmins, Ontario, Canada

Source: Mine Development Associates

Typical Cross Section 10225E

Date: 07-10-07 | Approved: DKY | Figure: 5-5



Hole #	Composite #	From	To	True Width (m)	Average Grade (g/t)
03BF079	#1	348.0	350.0	1.7	9.37
03BF040	#2	322.1	323.7	1.4	4.97
03BF027	#3	261.7	264.6	2.5	32.64
03BF030	#4	253.4	255.5	1.9	22.72
03BF030	#5	293.2	296.4	2.9	6.72
03BF033	#6	245.0	250.0	4.5	1.84
03BF033	#7	266.1	268.3	2.0	27.50
03BF036	#8	267.5	269.3	1.6	2.01
03BF036	#9	297.0	299.5	2.3	2.07
03BF128	#10	121.8	124.2	2.2	1.22
03BF103	#11	32.1	35.1	2.7	6.61
03BF103	#12	56.3	58.8	2.3	6.52
03BF103	#13	66.9	70.5	3.3	19.92

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SRK Job No.: 144418

File Name: Figure 5-6.doc

Black Fox, Timmins, Ontario, Canada

Source: Mine Development Associates

Typical Cross Section 10325E

Date: 07-10-07 | Approved: DKY | Figure: 5-6

6 Deposit Type

Black Fox 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. 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, extensional veins in widely distributed within carbonatized metamorphosed greenstone rocks. These deposits are typically associated with crustal scale compressional faults with a vertical extent of ≤2km 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 west of Black Fox and the Holloway and Holt-McDermott Mines are located to the east. Each of these deposits hosts approximately the 800k to 1Moz of gold. Many of the deposits extend downwards 1,000m below the surface, and some are blind deposits with no surface expression (Prenn, 2006).

There are several different styles of mineralization in the deposits proximal to the DPFZ or one of its related splay structures. 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).

7 Mineralization

Gold mineralization at Black Fox occurs within a main ankerite alteration zone with a known strike length of over 1km and ranges between 20 to 100m in true width. 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 occur as concordant zones 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 types of gold mineralization observed at Black Fox are:

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

The first mineralization type is quartz-rich portions of the AUV and CGR rock types. This includes the “green carbonate ore”. The typical host include ankerite-fuchsite altered ultramafic volcanic rocks, commonly found in the footwall portion of the DPFZ. While characterized by low-sulfide contents, small amounts of pyrite are typical of these mineralized zones. Quartz veining and quartz stockworks 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).

Mineralization hosted within mafic volcanic units that is associated with >5% fine-grained pyrite and minor to moderate quartz veining are coded as BMV with pyritic portions as MV rock types. Quartz veins are typically parallel to foliation, and visible gold is characteristically absent in this type of mineralization. This mineralization is referred to in the district as ‘flow ore’ and is frequently, but not always, associated with bleaching of the mafic volcanic host rocks. This style is common in the footwall portion of the DPFZ in the eastern part of the 235 level underground drilling (Hoxha and James, 2007).

The third type of mineralization is hosted in silicified felsic dikes. These dikes include both quartz-feldspar porphyries and finer grained units which are possibly syenitic in origin, Mineralization in the dikes is associated with increased silicification, pyrite and some quartz veining all associated with a fracture foliation. In the middle and hangingwall portions of the DPFZ, dike-hosted mineralization can often be correlated from hole to hole. This is in contrast to the blocks and lenses of felsic dikes that occur in the footwall portions of the DPFZ (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 hangingwall 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 work indicates that the sub-horizontal zones along the structure often have the greatest thickness at 10 to 15m, and highest grades. This suggests that zones of dilation are produced during episodes of folding and structural movements. The majority of other mineralized zones and quartz veins are 1 to 5m in width (Hoxha and James, 2007).

More than three generations of structurally controlled quartz veining have been identified in the Black Fox Mine. Quartz veins and stockwork zones within the main mineralized envelope are localized and within shear/fault zones. These structures parallel the main mineralized envelope and are responsible for the location and formation of veining and stockwork. The presence of sigmoidal vein structures, multiple quartz injections and re-sheared vein material with chloritic slips indicate complex and repeated structural movements during mineralization. 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).

"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" (Prenn, 2006).

8 Exploration

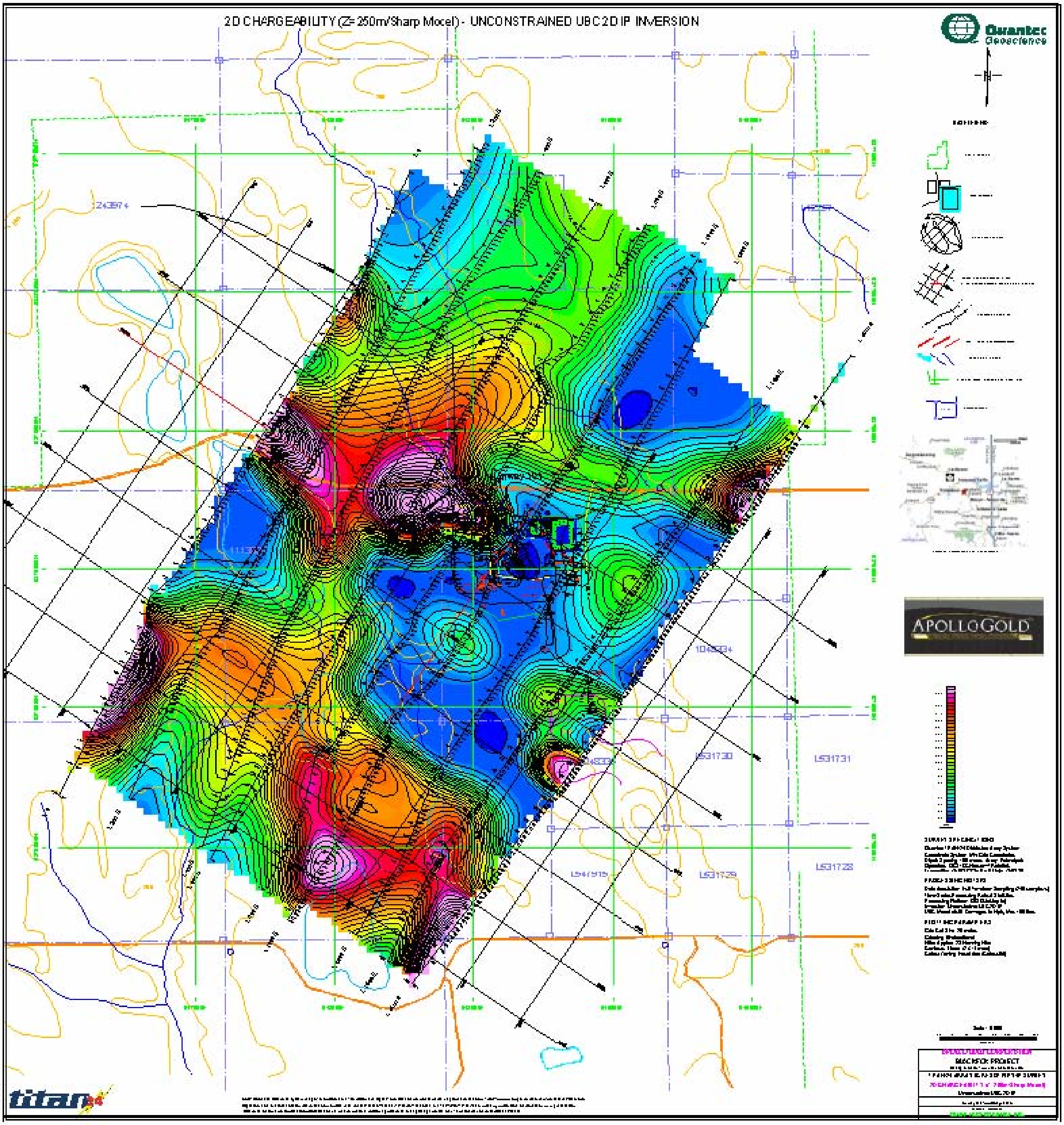
This section is partly excerpted from the Technical Report Black Fox Project Matheson, Ontario Canada by Mine Development Associates, August 14, 2006 and has been standardized to this report.

"*The Apollo exploration drilling continued from previous campaigns on 12.5 to 25m fence lines. Two main emphases included infill delineation of existing mineralization, and to explore for areas of new mineralization. In 2004, a 1,250m long exploratory underground drift (4m x 4m) was developed in the hanging wall down to 235m below the surface, to establish drill stations for an underground drilling program. The underground drilling program consisted of 75,700m of diamond drilling from 371 core holes. Surface drilling continued and by the end of 2006, Apollo had completed 825 diamond drillholes on the property, totaling 212,095m.*

During the spring of 2003, Apollo Gold Exploration, Inc. contracted with Quantec Geophysical, Inc., Toronto, Ontario, to complete an IP survey covering the entire property. Lines were spaced every 200m with 100m 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 7,500m 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. Figure 8-1 illustrates the results of the geophysical survey.

The initial portion of the Apollo 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 drillholes 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 12.5m spacing to improve the definition of this higher-grade mineralization."

At the time of this report, SRK has provided Apollo with a listing of approximately 62 drillholes totaling 10,000m of additional required drilling. The primary focus of this campaign is to target infill areas of known mineralization with the objective of upgrading currently inferred resources into the indicated category and subsequently incorporate these into a reserve.



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Black Fox Geophysical Survey
SRK Job No.: 144418	Source: Mine Development Associates	
File Name: Figure 8-1.doc		Date: 07-10-07 / Approved: DKY / Figure: 8-1

9 Drilling

9.1 Drilling Summary

A total of 1,826 surface and underground drillholes, (total of 324,625m) have been completed on the project by Noranda, Exall and Apollo between 1989 and 2005. Of these drillholes, 820 were completed by Apollo between 2002 and 2005. Table 9.1.1 lists the drilling by company and type. Figure 9-1 is a surface plan showing the surface drilling on the property. Figure 9-2 is a plan map showing the underground drilling.

Table 9.1.1: Black Fox Property Drill Summary

Company	Period	Type (All Core)	Number	Meters
Noranda	1989-1994	Surface	143	28,181
Exall	1995-1999	Surface	142	21,289
Apollo	2002-2005	Surface	449	136,392
Subtotal		*Surface*	*734*	*185,86*
Exall	1996-2001	Underground	721	63,059
Apollo	2004-2005	Underground	371	75,705
Subtotal		*Underground*	*1,092*	*138,764*
Total		**Black Fox**	**1,826**	**324,625**

Table 9.1.2 shows the database summary for historic and Apollo drilling. The coordinates shown in the summary reflect a local grid developed by Exall. Note that Black Fox has established the mine surface datum line as 10,000m. This value does not reflect mean sea level and is used as a reference point only.

Table 9.1.2: Black Fox Property Database Summary

		Apollo Drilling					Historic Drilling				
				x	y	z			x	y	z
Item	Item	Hole	Depth	East	North	Elevation	Hole	Depth	East	North	Elevation
Minimum		04BF320	671	10,100	8,675	9,995	75	197	8,900	8,899	10,000
Maximum		05BF411	374	10,815	10,523	9,999	62	76	10,649	10,479	10,000
Minimum		03BF279	693	9,550	9,220	9,996	75	197	8,901	8,899	10,000
Maximum		04BF388	252	11,050	10,375	10,000	292-99	274	10,693	10,177	10,003
Minimum	Elevation	235-214	206	10,026	9,798	9,765	19-1	126	10,018	10,006	9,813
Maximum	Elevation	04BF349	680	10,474	9,651	10,010	292-99	274	10,693	10,177	10,003
Minimum	Depth	235-257A	9	10,062	9,818	9,766	14-80	6	10,017	9,984	9,8601
Maximum	Depth	04BF360	995	10,470	9,580	10,008	280	851	9,901	9,824	10,000

The Black Fox database includes a total of 176,525 assay intervals.

9.2 Historic Diamond Drilling and Logging

Sections 9.2 through 9.3 are excerpted from Technical Report Black Fox Project Matheson, Ontario Canada by Mine Development Associates, August 14, 2006 and has been standardized to this report.

"From 1989 to 1993, Noranda drilled 142 NQ-size diamond core holes totaling 27,930m in this area. These holes were logged by a geologist and split for assay. Noranda hole 61a is in the database as the 143rd hole drilled by Noranda, but was lost and has 0ft of core drilling noted in the database. Exall drilled an additional 142 NQ-size surface core holes totaling 21,295m from 1994 through 1999. 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. Exall also drilled 720 underground core holes that totaled 62,827m. They were generally NQ in size, although they occasionally had to reduce to BQ-size in bad ground. Exall's drill core was also logged by a geologist and split to prepare intervals for assaying. Exall, as part of their underground drilling program, managed to extend two NQ core holes into significant mineralization below 200m from the surface. Unfortunately, they had to be drilled down the dip of the DPFZ and are of limited value.

9.3 Apollo Diamond Drilling and Logging

Norex Drilling International from Porcupine, Ontario, completed most of the Black Fox property drilling for Apollo generally starting each hole with NQ-size core. The drillholes provide core samples of potentially mineralized ground, mainly within the DPFZ. Ground conditions for core drilling have been very good.

The core is removed from the wire line inner barrel and placed in wooden core boxes. Each box can hold up to 6m 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. When the box is full, the hole 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 Apollo Gold Exploration, Inc. During this time, the core is under the direct supervision of the driller.

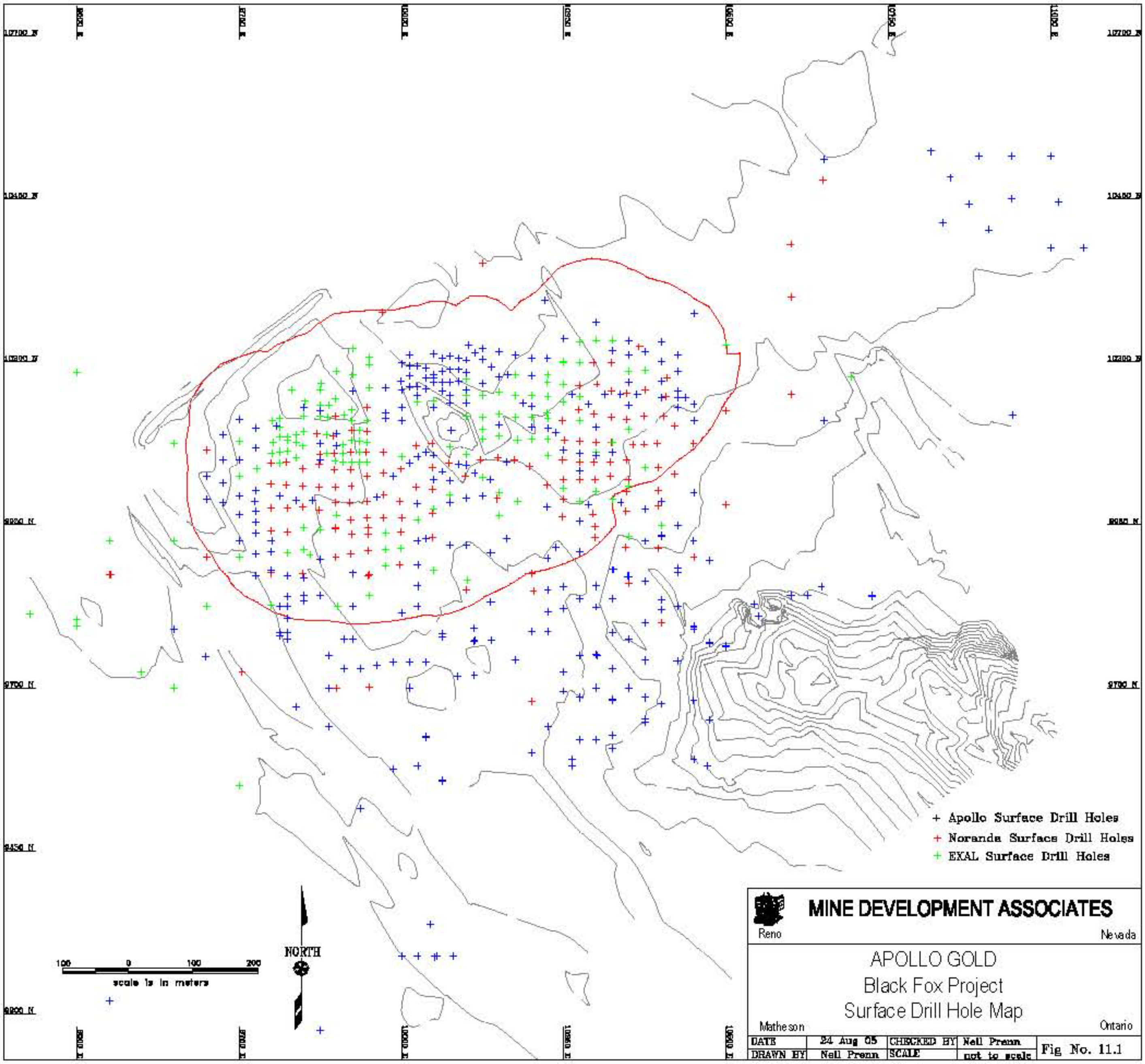
The core samples are picked up by Apollo personnel each morning and at various times during the day as necessary. It is loaded into a company truck and taken to the core logging facility in Matheson, Ontario, approximately 8km away, until a facility was constructed on the project site. The core is then 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 Apollo geologists. When a geologist begins logging a hole a logging form is generated within a computer. Data regarding the hole, such as depth, date logged, location and the geologist responsible for the hole are entered. This form will also be where the geologist enters geological and geotechnical attributes as the hole is logged. All logging is done electronically with no hand written data. This eliminates a separate data entry step and the subsequent errors that it can introduce. The geologist takes boxes of core for the hole from the rack and places them on 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 a check against any lost or missing core that was not accounted for by the drillers. All of this data, along with all geological data, are entered into the computer spreadsheet by the geologist. Once the core has been reassembled, it is 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.

The geologist marks the core for splitting with a grease pencil and assigns each sample interval with a sample number. From this point until the assays return from the lab all references to the sample are by this number only. The core is then sent to the splitting room where technicians saw the core sample in half with a diamond saw and place it in a bag which is marked with the sample number. 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 from Swastika, Ontario. The samples are placed into their truck, with each sample being checked off a list as it is being loaded. The half core that remains in the core box has the lid replaced and is placed back in the rack by the technician. When a truck load of split core has been accumulated, it is taken back to the Black Fox mine site where it is labeled with a stainless steel tag that has the hole number and footage imprinted and placed in outside, covered, storage racks.

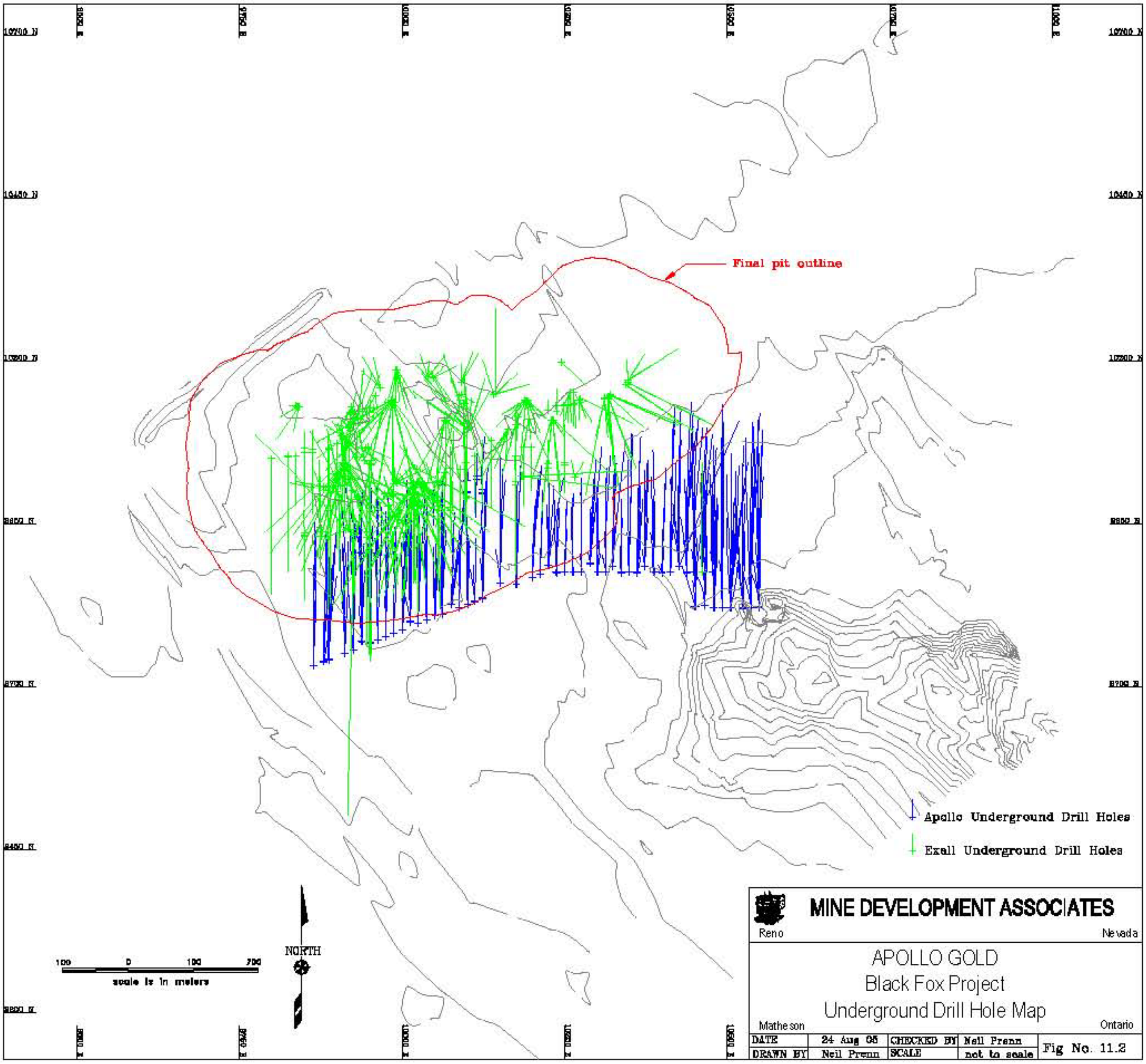
All holes have the collars surveyed and are surveyed for deflection with a Reflex E-Z Shot digital tool. Measurements are taken approximately every 50m down the hole. Occasionally a reading will be taken near a particularly strongly magnetic rock unit and gives a spurious result. The geologists review all surveys and any bad readings are discarded. As a check, three holes were re-surveyed using a Maxi-bore gyroscopic tool. The 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.

The Apollo drilling has concentrated to develop two main areas of the mineralization. The first is the near-surface area of the Black Fox deposit that may be mineable by open pit methods. About half of the surface drillholes were completed to test the thickness and grade of the near-surface mineralization. Drilling has been along sections oriented 36^o azimuth and at inclinations of 45^o to 50^o. This places the drillholes approximately perpendicular to the DPFZ. For this near-surface mineralization, the shoots generally plunge at 40^o south to the southwest within the DPFZ. Because of the shallow nature of the mineralization, holes have been drilled individually to reach specific targets and not as fans as were required for the deeper targets.

The second area is the mineralization that is down dip of the area mined by Exall. The DPFZ, in the deeper area, has the same southeasterly strike, but the dip is at a steeper average of 60^o. The mineralization still occurs along structural intersections and at dilation zones along the fault. These tend to plunge at about 40^o to the southeast or southwest. In this area of steeper dip along the fault, the shoots tend to be smaller, tighter and less continuous than seen near the surface. The holes to test this area were drilled from the surface and underground in fans spaced approximately 12.5 to 25m apart, designed to penetrate the DPFZ. Eventually, more tightly-spaced drilling from underground platforms will be required to improve the definition of the shape, extent, and grade of the deposit."



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Surface Drilling Plan		
SRK Job No.: 144418	Source: Mine Development Associates			
File Name: Figure 9-1.doc		Date: 07-10-07	Approved: DKY	Figure: 9-1



SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Underground Drillhole Plan Map**
SRK Job No.: 144418	**Source: Mine Development Associates**	
File Name: Figure 9-2.doc		Date: 07-10-07 / Approved: DKY / Figure: 9-2

10 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 and has been standardized to this report.

10.1 Noranda Sampling

"Noranda drilled a total of 142 NQ-size diamond core holes totaling 27,930m from 1989 through 1994. All holes were surveyed at the collar and had acid etch tests done to measure the dip. A Tropari survey was run at the bottom of a few of the deeper holes to measure deviation. The lack of a down-hole survey on most of these holes could be a source of error in drillhole location, especially the deeper holes.

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 and then cut in half with a diamond saw. The sample was then sent to either Swastika Labs or Chemex Labs in Rouyn, Quebec.

10.2 Exall Sampling

10.2.1 Exall Diamond Drilling

All drill core was NQ-size, unless ground conditions required reduction to BQ. Diamond drilling was used to define an existing ore body, find new ore zones and define the lithology between two holes. The surface drillholes were down-hole surveyed, however, the underground holes were not surveyed for down-hole deflection, therefore the bearing and inclination at the collar has to be used for the entire underground drillhole. For this reason, several of the underground holes appear to be located at incorrect locations on the drillhole cross section maps.

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 30g sample and completed a fire assay of the sample. All samples above 34.3g/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.

10.2.2 Exall Assay Lab Procedures

Blank samples (samples that have no assay values) were introduced with regular samples to verify the accuracy and to see if any contamination was present at the lab. Labs automatically ran their own internal random checks on samples.

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%.

10.3 Apollo Sampling

Core is logged and sampled at the core logging facility in Matheson. Samples from the split core remain in the logging facility until they are picked up by representatives of the assay laboratory. Apollo sends the bulk of the core samples to Swastika Laboratories Ltd. (Swastika). A smaller number of samples are sent to the SGS laboratory in Rouyn, Quebec.

Drilling is done at approximately 25m spacing. Sampling intervals are controlled by geological boundaries. The maximum assay interval in the unaltered hangingwall and footwall rocks is 1.5m.

An ankerite alteration envelope encloses the mineralized areas. Areas of strong ankerite alteration are sampled at 0.6 to 1m intervals. Suspected ore zones (Green-carbonate rocks with quartz-ankerite stockwork, or pyrite-rich felsic dykes) are sampled at 0.3 to 0.6m intervals. Less than 0.3m samples are discouraged. Visible gold is common within the quartz-ankerite stockworks in the green-carbonate zones.

The gold is generally located within the quartz and carbonate zones. Very little of the gold is associated with clay. Therefore, flushing of the gold while drilling or splitting is less of a problem. Also, the ore zones, as well as the wallrock, are very competent with very good recovery.

10.4 Black Fox Deposit Sampling Issues

MDA believes that there are two serious sampling issues with the deposit, both of which are related to coarse gold in the deposit, and result in samples that tend to contain less gold than is actually present. They are actually the same problem and are based on sample size issues and coarse gold.

The first issue is to get a large enough sample to represent the area sampled. The gold at the Black Fox deposit appears to be concentrated in small areas. When the gold is concentrated in small areas, drillhole samples will occasionally get too much gold in the sample when the area of concentration is intersected or more often, miss the area of concentration and get too little gold in the sample. The core holes that form the basis for the resource and reserve estimate are too small to obtain a representative sample. Some samples may even appear to be waste without the concentrated gold. It is likely that holes several meters in diameter would be required to obtain representative samples of the deposit. MDA reviewed areas that were penetrated by drillholes prior to mining, and commonly areas that were stoped appeared to be un-mineralized based on the drilling.

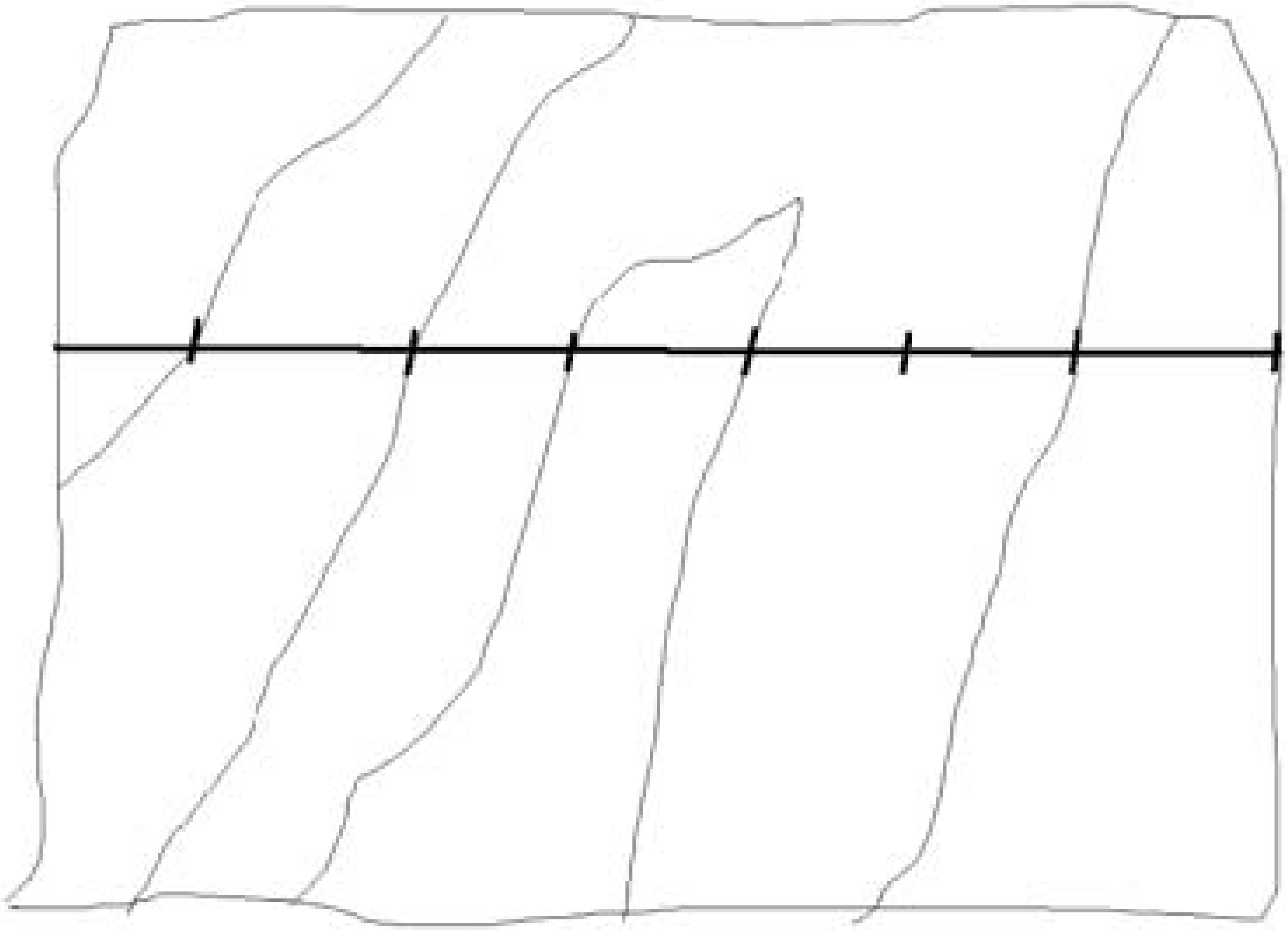
The second problem is getting the representative amount of gold in the sample pulp once the sample has been obtained. Gold particles up to 0.15cm have been observed and particles of 0.06cm are very common (Pitard, 2005). The proper sample size is required in order to get a representative sample again, but this time we have all of the gold contained in the sample somewhere in the core. With coarse gold, 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 0.15cm gold particles that occur at Black Fox, samples of up to 109kg must be processed in their entirety (Pitard, 2005). If the sample contains 0.06cm gold particles, which commonly occur in the deposit, a 7kg sample must be processed in its entirety (Pitard, 2005). These sample sizes are much larger than the typical 30g fire assay sample or even the generally larger than the 1,000g 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."*

MDA believes that both the size of the sample to measure the gold in the deposit and the size of the sample to measure the gold in the sample are too small, and will result in a database where some samples represent a higher sample grade than is present at the sample location, but many samples represent too low a grade than is present at the sample location. MDA concurs with Pitard's conclusion, that the drillhole data is likely biased and will likely underestimate the contained gold within the deposit."

SRK concurs with the observations and opinions of MDA and Pitard as discussed above. Based on these observations and opinions, SRK has put significant emphasis into creating the resource estimate described below which approximates the historical production while remaining conservative.



SRK Consulting
Engineers and Scientists

SRK Job No.: 144418

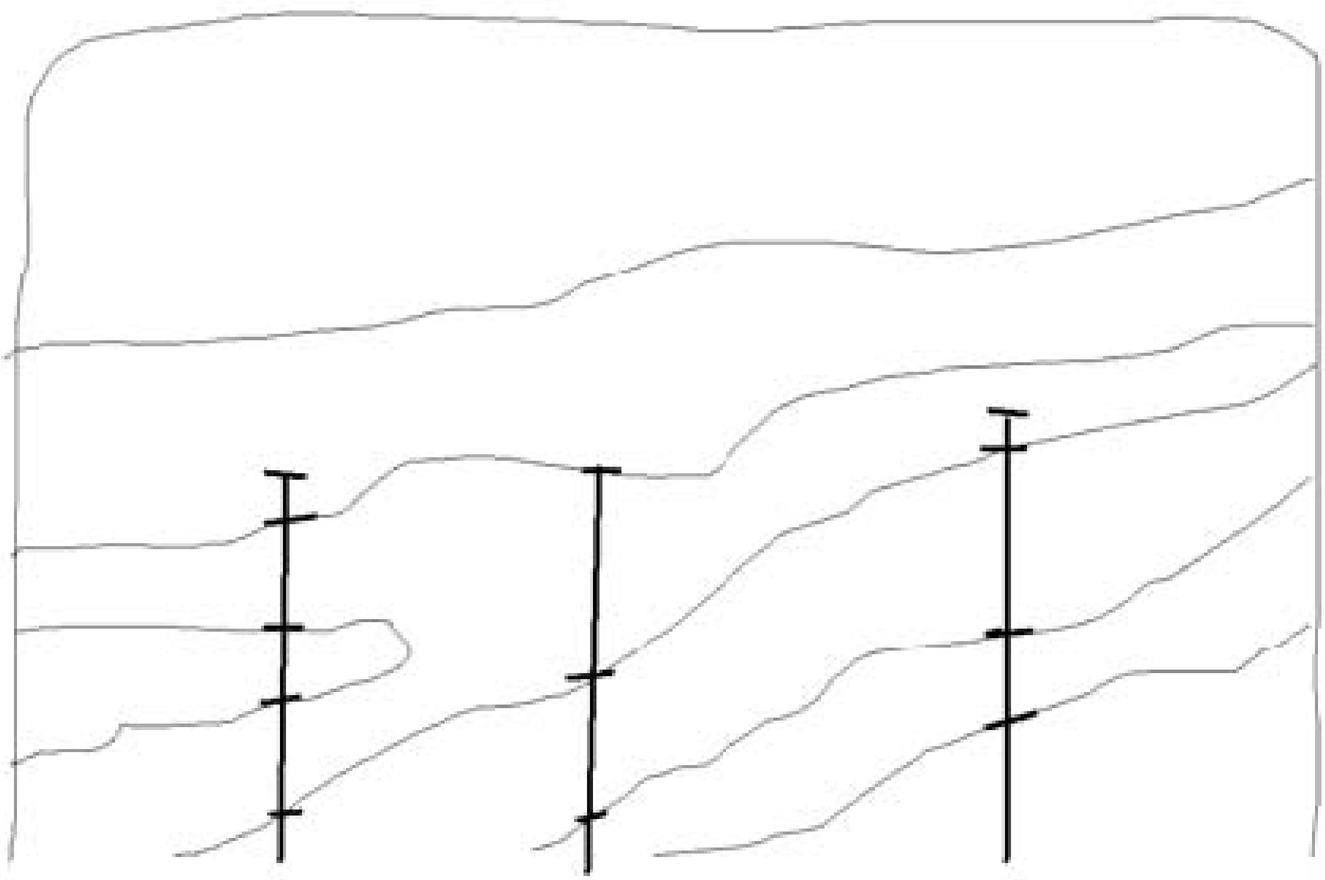
File Name: Figure 10-1.doc

**Black Fox,
Timmins, Ontario, Canada**

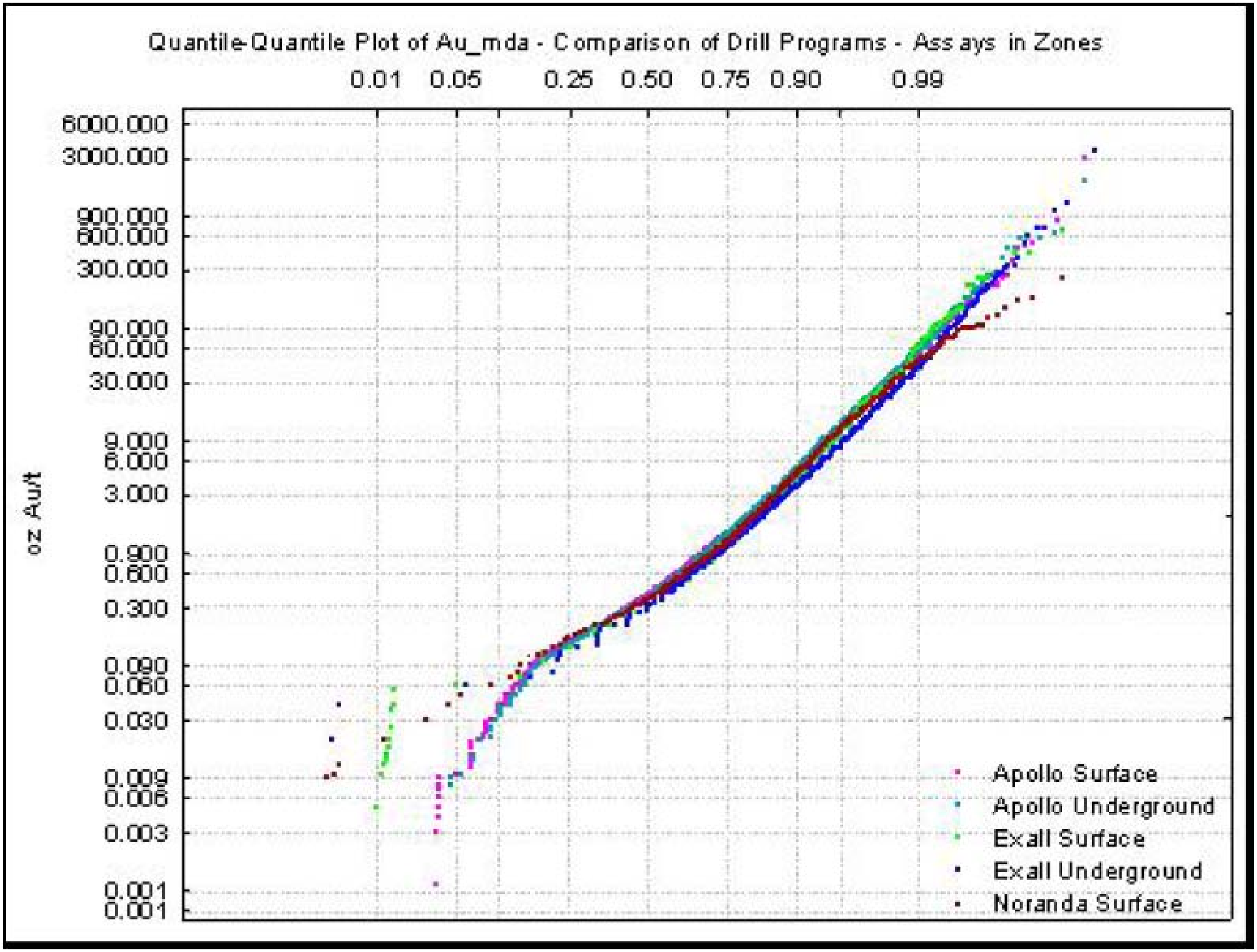
Source: Mine Development Associates

**Underground Heading
Horizontal Chip Sampling**

Date: 07-10-07 | Approved: DKY | Figure: 10-1



SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Underground Heading Vertical Chip Sampling**
SRK Job No.: 144418	**Source: Mine Development Associates**	
File Name: Figure 10-2.doc		Date: 07-10-07 | Approved: DKY | Figure: 10-2



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	QQ Plot of Drill Programs		
SRK Job No.: 144418	Source: Mine Development Associates			
File Name: Figure 10-3.doc		Date: 07-10-07	Approved: DKY	Figure: 10-3

11 Sample Preparation, Analyses and Security

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

11.1 Sample Security

"Samples were sealed in bags at the site and shipped or collected by commercial laboratory personnel.

11.2 Drill Sample Preparation and Analysis

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 ½ (15g) to 1 (30g) assay ton samples for assay. Most of the assays were completed by fire assay methods with a gravimetric finish.

11.2.1 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 ½ assay ton sample for analysis, and to re-run samples if the initial analysis was greater than 2g/t-Au on a one assay ton sample. The Noranda assay lab used the flow sheet shown in Figure 11-1 to prepare and assay the samples received from Noranda, most of which weighed from 1 to 5kg.

11.2.2 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 300g split was taken. The 300g sample was pulverized to 80% to 200 mesh. A one assay ton (30g) sample was split from the pulverized material for fire assay with AA finish. Exall requested checks on all assays exceeding 34.3g/t-Au. The Techni-Lab internal checks agreed well with the original sample.

11.2.3 Apollo Drill Sample Preparation and Analysis

Apollo saws the core and ships ½ of the drill core to either Swastika or SGS Laboratories. The labs prepare a one ton (30g) sample for fire assay with a gravimetric finish by crushing the core to -10 mesh and taking a 400g split to pulverize and prepare the sample pulp. As a quality check, the coarse reject sample material from each mineralized zone, having an average grade over 1g is sent to the other lab. The samples are repulped from coarse rejects and reassayed using a 30g fire assay with a gravimetric finish. By doing this, a check is done on the entire process, from sample prep through the gravimetric finish. All samples, whether initially reported as high-grade or low-grade, are checked. Currently, samples are being selected and run for a screened metallic fire assay. This data will also be compared with the results of the two conventional fire assays.

Digital e-mails of results are sent from Swastika Laboratories along with certificates of Analysis to the Black Fox mine. Along with the digital e-mail of the assays results, a faxed copy of the assay certificate is sent from Swastika Laboratories. The faxed copy of the certificate is sent back to Apollo Gold Exploration Inc. staff for hole delineation. The faxed certificates, with segregated holes are checked against the digital file for errors. The faxed copy, once confirmed by Apollo to be correct are stamped complete, and added to the audit file for back referencing. The digital assay result file is cut from the e-mailed file and pasted on the core logs generated for logging of Black Fox core. Once pasted the sample numbers and the pasted assays are checked again to ensure no pasting errors occurred. Once assays are pasted onto the drilling logs and confirmed they are saved as a separate file. If the logs are complete with all assays, they are saved as a DC file. The DC files are 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 is sent to Apollo's offices in the USA for modeling and reporting purposes. If the assays results are not complete, the file will be saved as a "Pending" file, and is stored in an incomplete assays folder until final assay results are posted. 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.

11.3 QA/QC, Check Samples, Check Assays

11.3.1 Noranda Check Assays

The Noranda data includes 196 reruns of ½ assay ton (15g) samples of the original ½ assay ton samples. The reruns average 4.6% lower grade than the original samples, as shown in Figure 11-2. The samples over 2g were noted to be rerun by a one assay ton (30g) sample, however most of this data is not in the digital database. Reruns of 80 samples indicate the reruns of one assay ton are higher in grade by about 5% than the original ½ assay ton sample, as shown in Figure 11-3. Scott Wilson Roscoe Postle Associates Inc. (RPA) (1997) reports that Noranda checked about 10% of the assays.

Noranda assay sample distribution is missing the high grade found in all the other drill programs shown Figure 11-3. MDA recommends 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.

11.3.2 Exall Check Sampling

Techni-Lab batched samples in groups of 24. Each group contained at least one blank sample, one standard sample and replicate samples. Routine checks were taken on about 5% of the samples and all samples over 34.3g/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 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.

11.3.3 Apollo Check Assaying

Metallic Check Assays

Apollo has completed metallic assays on 594 samples. Of these, a total of 512 can be compared to fire normal fire assays. The metallic assays are 17% higher in grade than the average of the fire assays from these intervals. A total of 289 metallic assays are higher in grade to the average of the fire assays, while 223 are equal to or lower in grade. MDA believes that metallic assays are essential in obtaining a sample assay that is 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. Figure 11-4 shows the comparison of the metallic assays to the fire assays.

Standards and Blanks

Apollo submitted standards and blanks within each set of samples submitted for assay. Four labs were used with most of the assays completed by Swastika. Figure 11-5 shows that the blanks usually agree for the several thousand tests that were completed.

A number of sample standards have been run within each group of samples. Figures 11-6 and 11-7 show the two most common high-grade and low-grade standards. Swastika has shown reasonable ability to accurately assay the standards.

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

- *Blank > 0.03g/t-Au = Fail;*
- *Standard 1.422 >1.528 or <1.322 = Fail;*
- *Standard 11.27 >12.03 or <10.63 = Fail; and*
- *Standard 9.62 >10.28 or <9.00 = Fail.*

If the blank or standard failed, then the entire batch (20 samples) would be reassayed, 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 as shown in Figure 11-8. The check needed to be within ± 10%. If not, the pulp would be reassayed a second time.

Checks on Sample Rejects

A total of 2,618 assay intervals have been checked by a different lab by preparing an additional sample from the original sample rejects. The original sample is higher than the check by about 4%. This comparison is shown graphically in Figure 11-9. 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. Figure 11-10 is a graph of the relative difference between the original and the checks. These 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 14kg 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. 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 30g 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 we would expect, and it is likely due to more high-grade material being in the mini-bulk samples than in the deposit as an average. Figure 11-11 shows the comparison of the original drillhole assays to the mini-bulk sample average grade. The six waste mini-bulk samples showed an improvement in grade of 1,382.3% compared to the individual core assays. One of the waste samples averaged 0.00g/t-Au from the drillhole intervals and 2.82g/t-Au from the mini-bulk composites. The other five mini-bulk samples were not assayed prior to testing. One of these samples averaged 1.38g/t-Au from the mini-bulk test."*

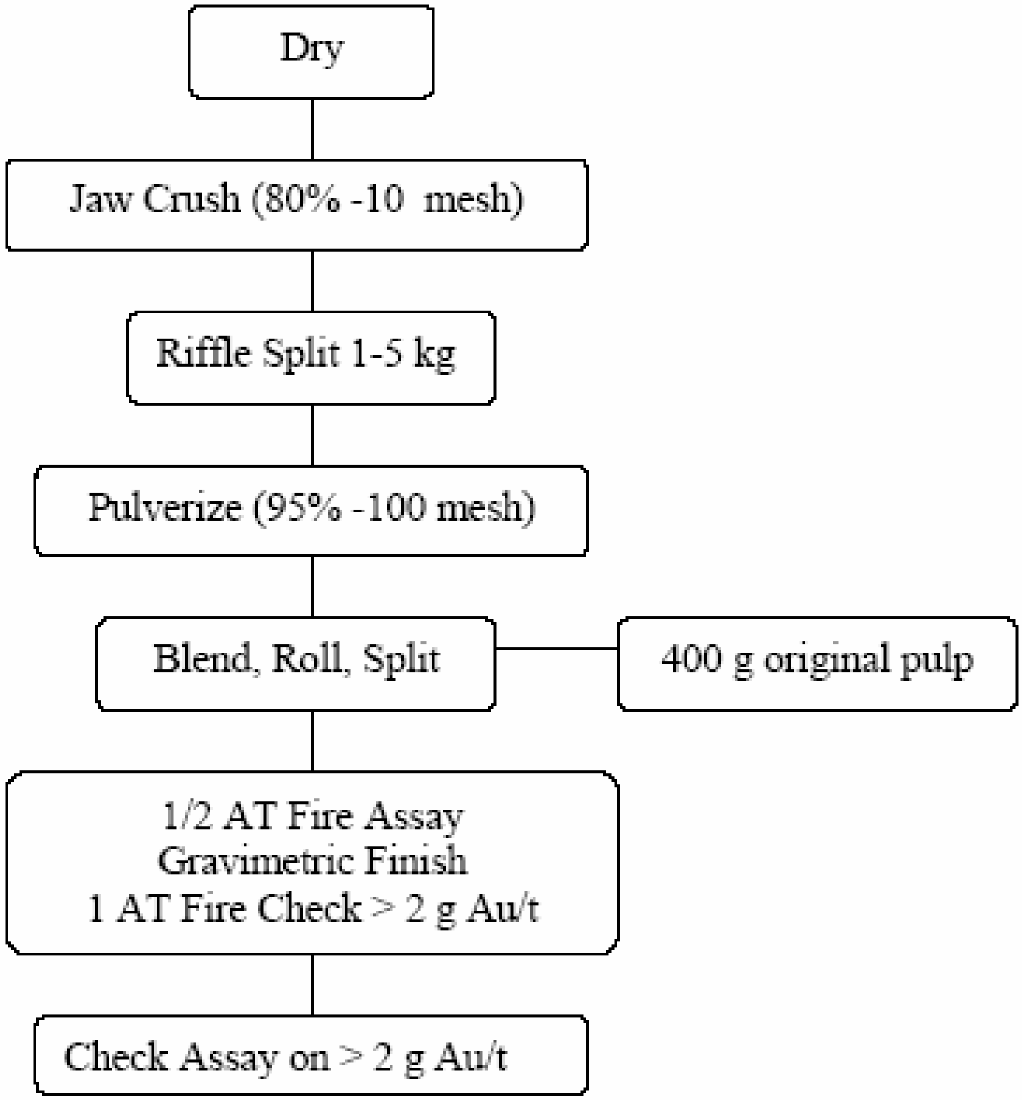
11.3.4 Summary

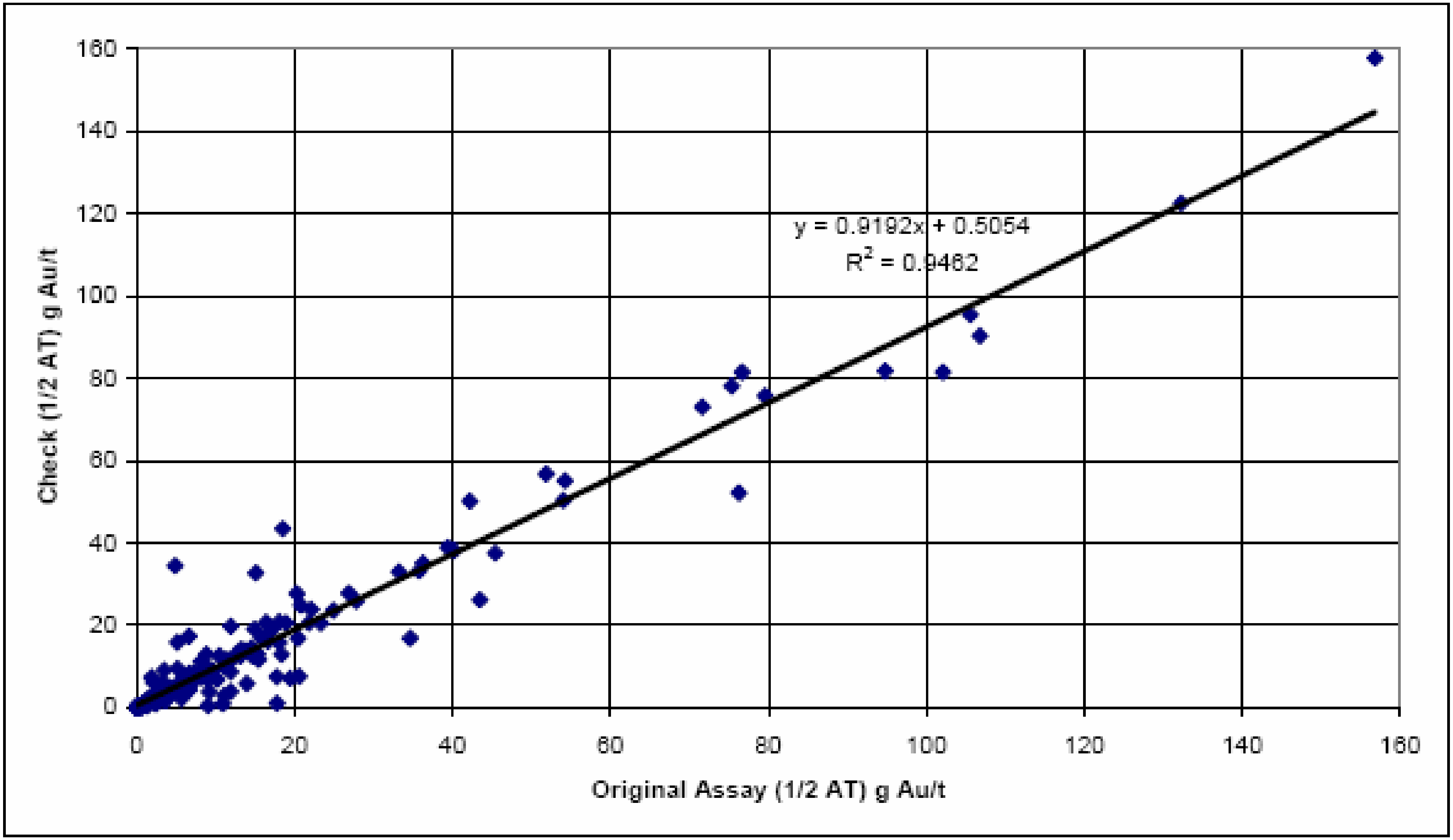
The Black Fox deposit contains a significant portion of coarse gold that is difficult to sample. Obtaining a large sample is likely the key to representative assaying. Limited testing (593 samples) by Apollo showed higher assays (about 10%) for samples processed by metallic assay than by fire assay.

The historic check sampling on the project appears to be weak, especially for a deposit that is known to contain coarse gold. The Noranda checks appear to be limited to checks on the same assay pulps. The Noranda checks are based on new assays made from the same assay pulp and show reasonable agreement on the mean grade, however, individual sample variance is high. The Exall check assay program is nearly non-existent except for checks on the same assay pulps. Exall completed several checks with the Macassa Mill which indicated poor agreement with the original assay.

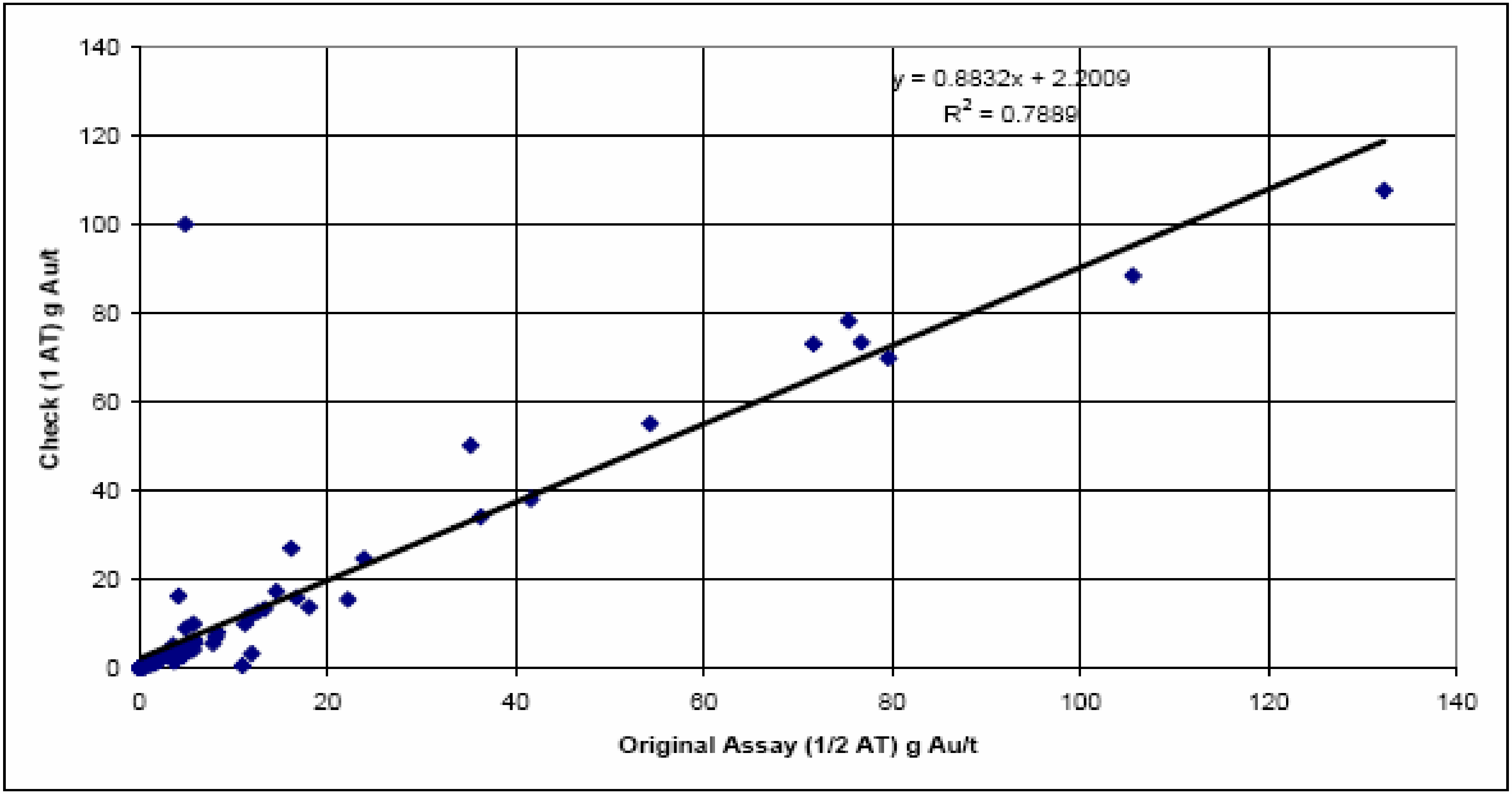
Apollo is planning a significantly improved check assay program where there is a check assay on each mineralized zone. In addition to the blank and standard check samples, Swastika runs their own internal check samples. All of the samples are run for a 30g fire assay. Potential ore zones are then selected from the fire assay results by Apollo personnel and these intervals are re-run with a 1,200g 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 Apollo in digital format where it is merged with the geological spreadsheet for that hole. Additionally, Apollo plans to complete metallic check assays on selected intervals."

SRK is comfortable with sample security and preparation. However, past QA/QC has been inadequate. SRK is comfortable with the steps Apollo is taking to improve QA/QC.

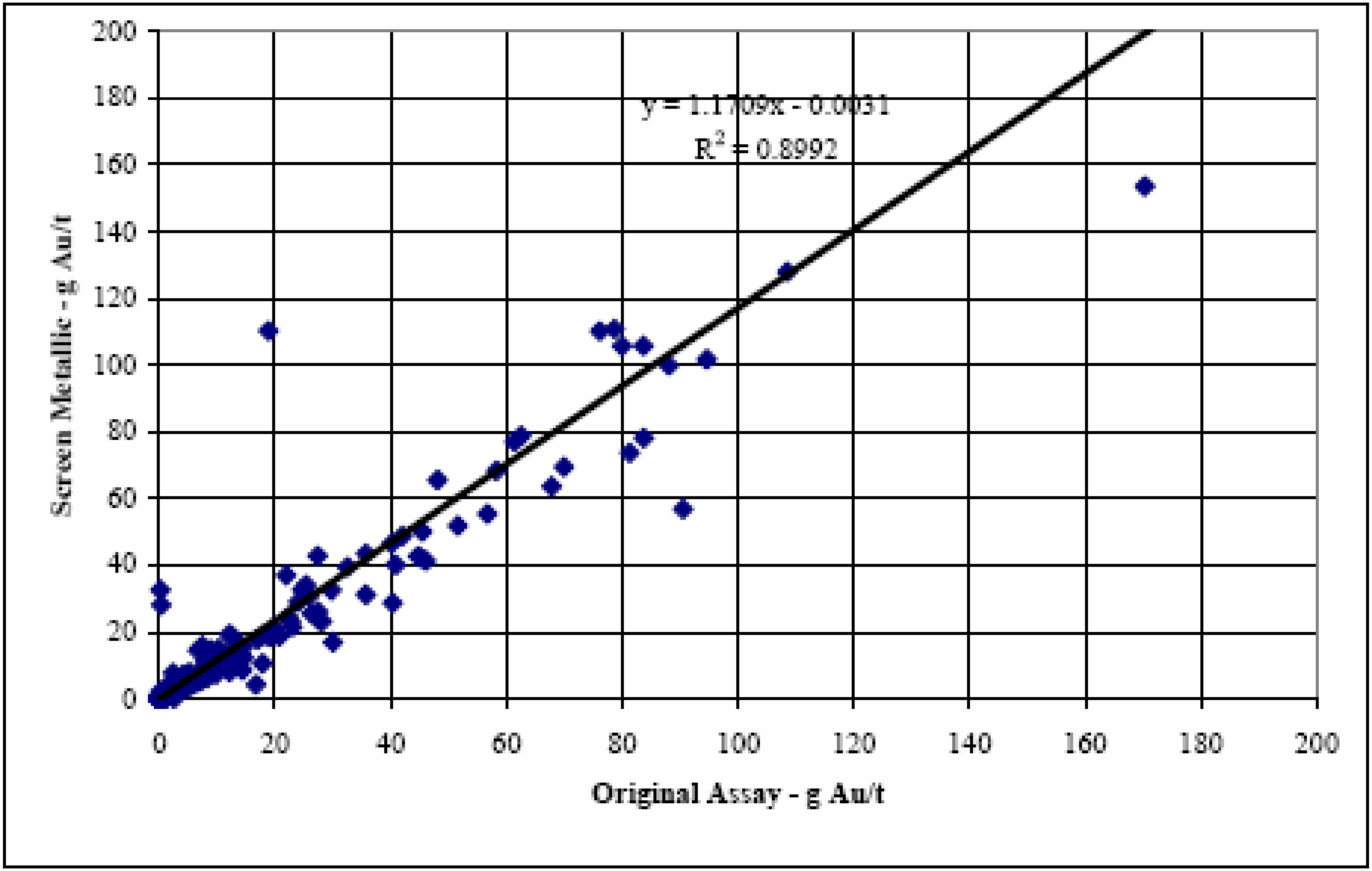

Dry
Jaw Crush (80% -10 mesh)
Riffle Split 1-5 kg
Pulverize (95% -100 mesh)
Blend, Roll, Split
400 g original pulp
1/2 AT Fire Assay
Gravimetric Finish
1 AT Fire Check > 2 g Au/t
Check Assay on > 2 g Au/t



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada Source: Mine Development Associates	**Noranda ½ Assay Tonne Reruns**
SRK Job No.: 144418		Date: 07-10-07 / Approved: BAS / Figure: 11-2
File Name: Figure 11-2.doc		



SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Noranda 1 Assay Tonne Reruns**
SRK Job No.: 144418	**Source: Mine Development Associates**	Date: 07-10-07 · Approved: BAS · Figure: 11-3
File Name: Figure 11-3.doc		



SRK Consulting
Engineers and Scientists

SRK Job No.: 144418

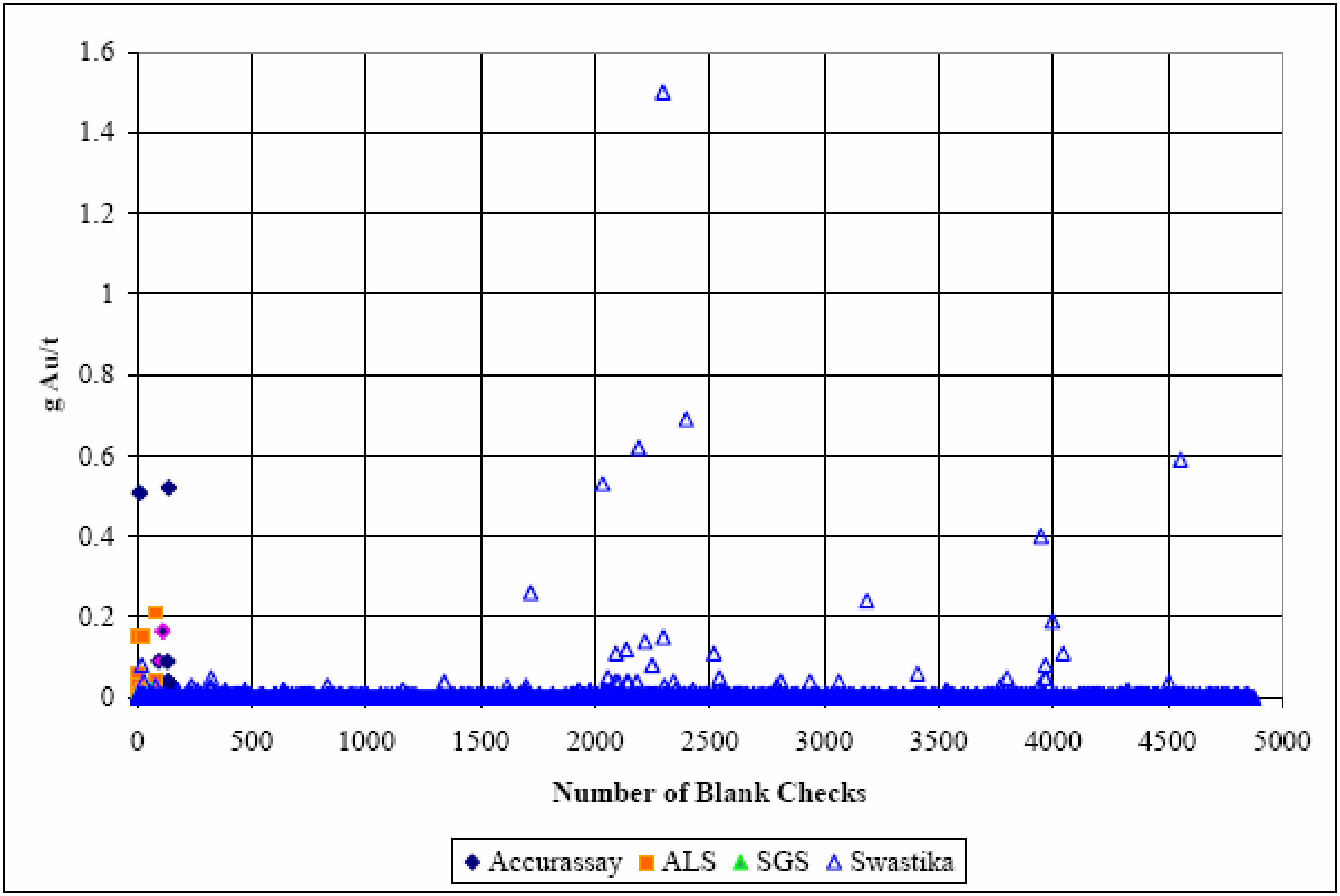
File Name: Figure 11-4.doc

Black Fox,
Timmins, Ontario, Canada

Source: Mine Development Associates

Apollo Metallic Check Assays

Date: 07-10-07	Approved: BAS	Figure: 11-4



SRK Consulting
Engineers and Scientists

SRK Job No.: 144418

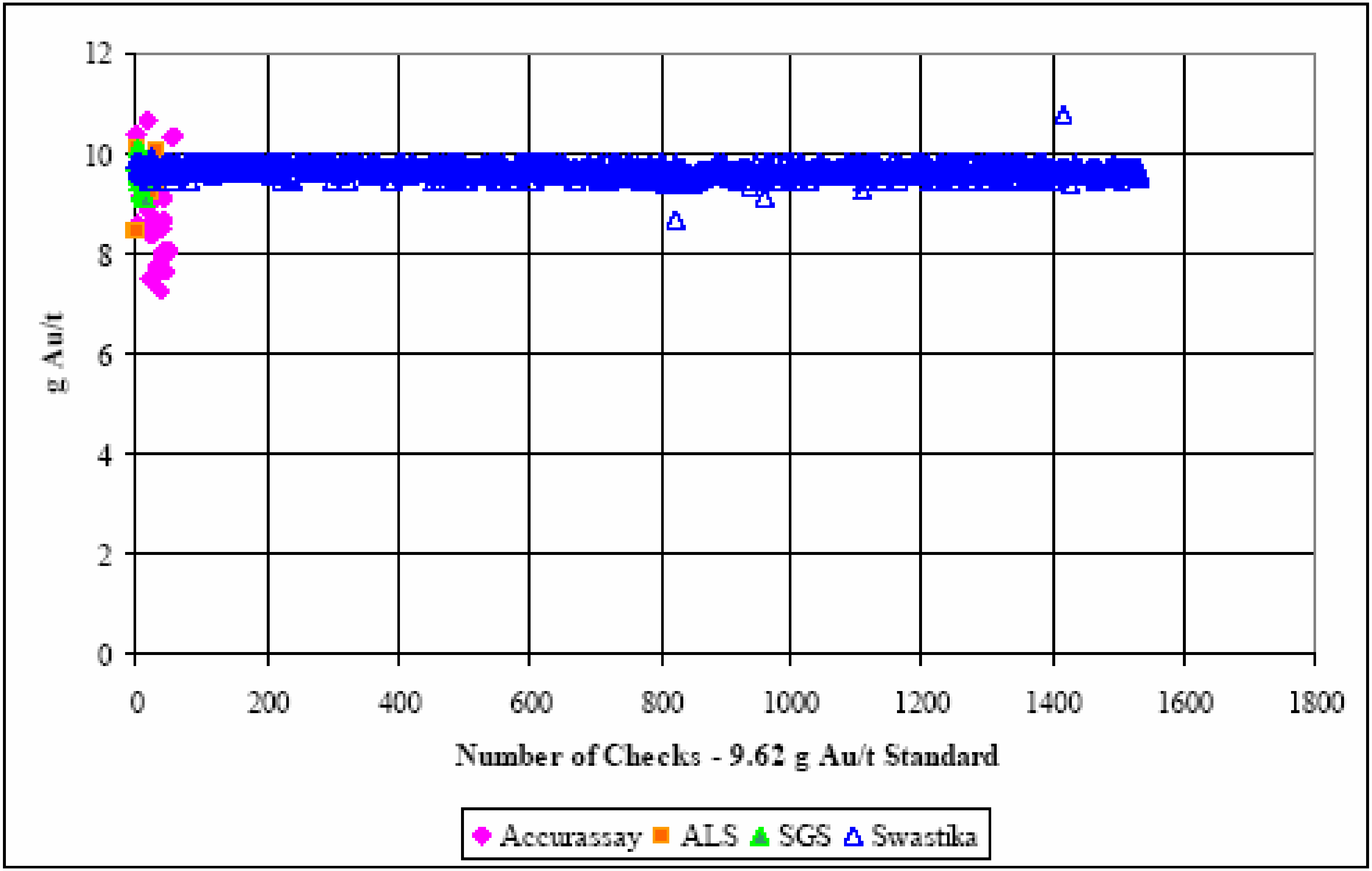
File Name: Figure 11-5.doc

Black Fox,
Timmins, Ontario, Canada

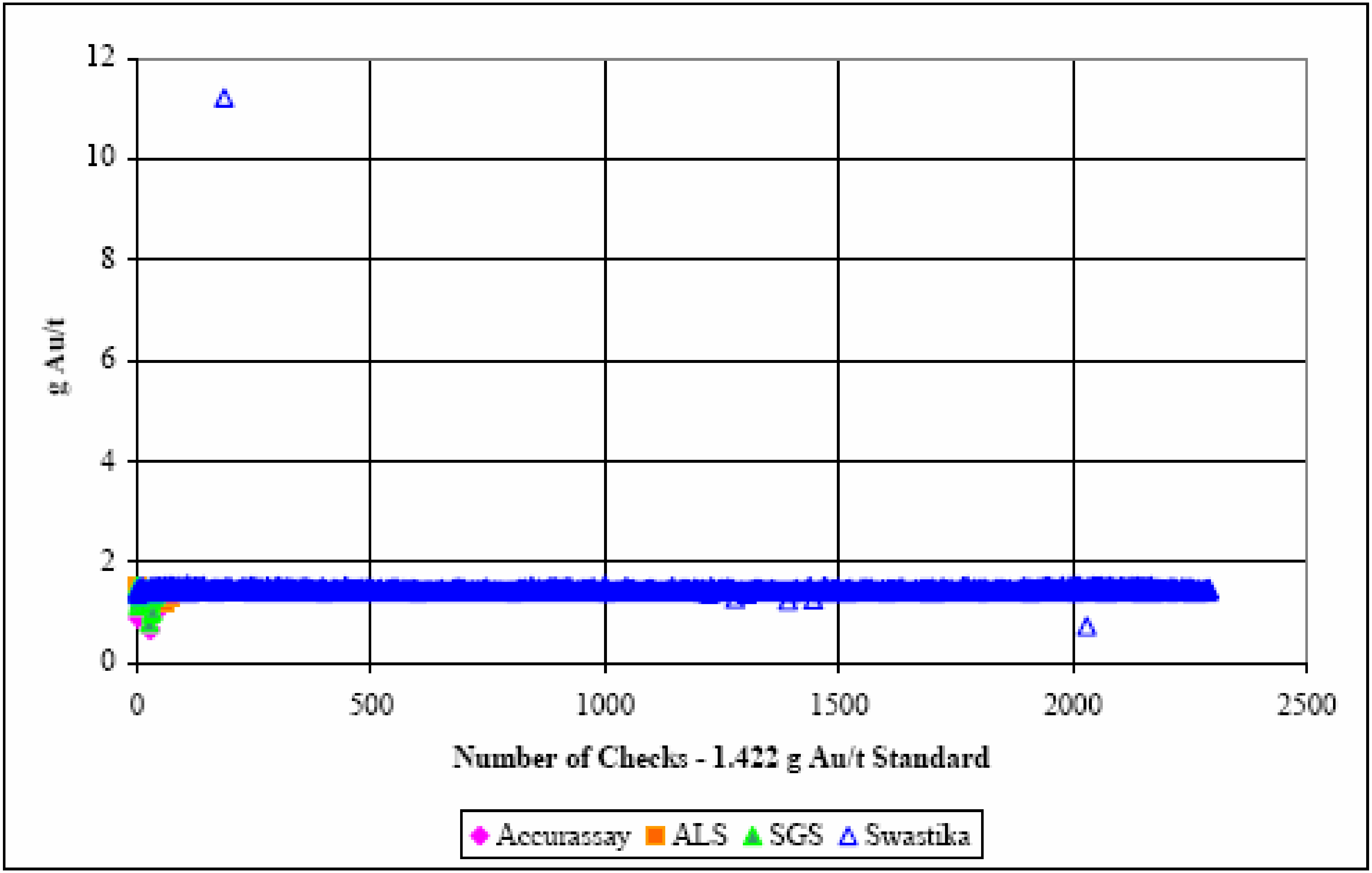
Source: Mine Development Associates

Apollo Blank Check Assays

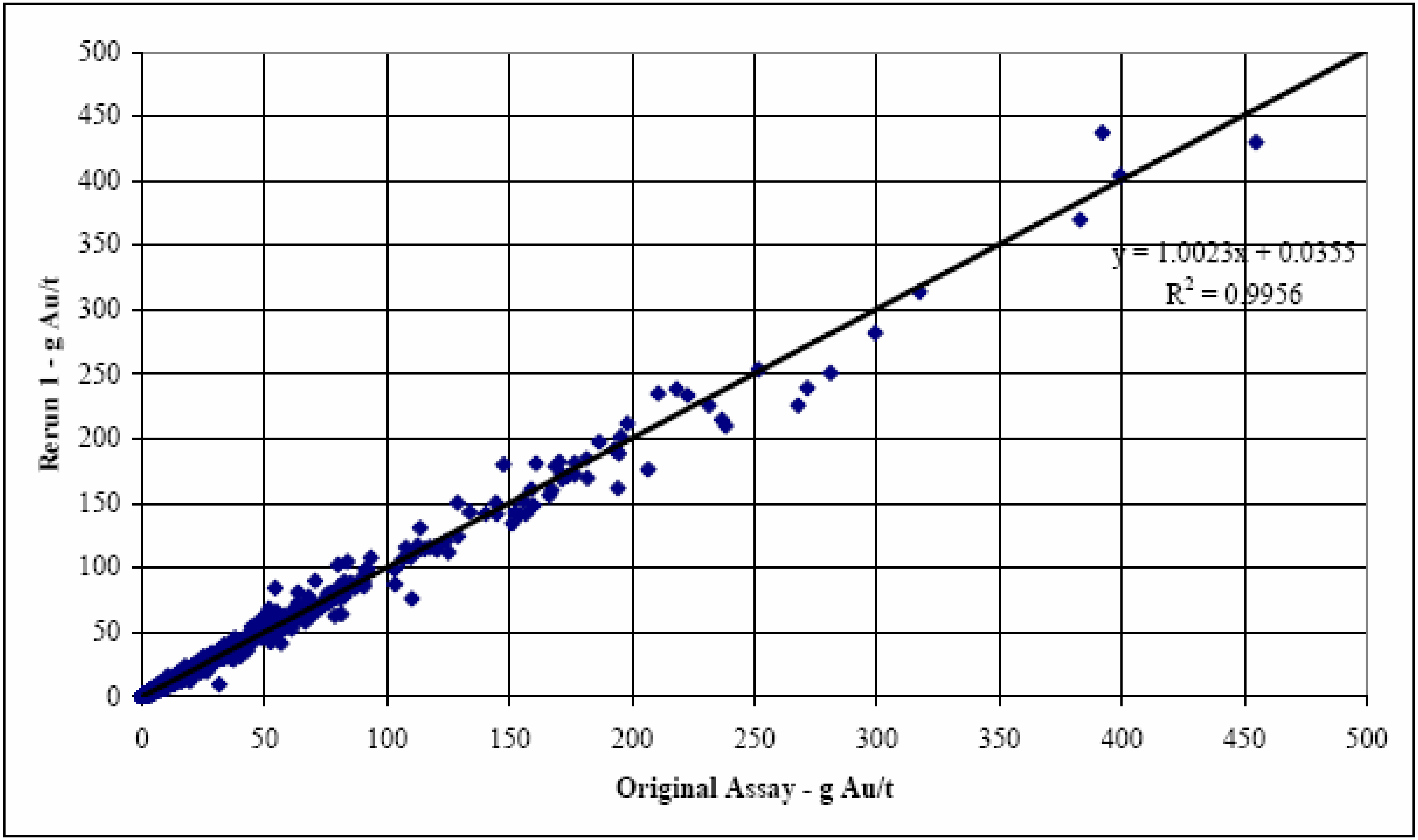
Date: 07-10-07 | Approved: BAS | Figure: 11-5



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	**High-Grade Standard**
SRK Job No.: 144418		
File Name: Figure 11-6.doc	**Source: Mine Development Associates**	Date: 07-10-07 · Approved: BAS · Figure: 11-6



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Low-Grade Standard		
SRK Job No.: 144418	Source: Mine Development Associates			
File Name: Figure 11-7.doc		Date: 07-10-07	Approved: BAS	Figure: 11-7



SRK Consulting
Engineers and Scientists

SRK Job No.: 144418

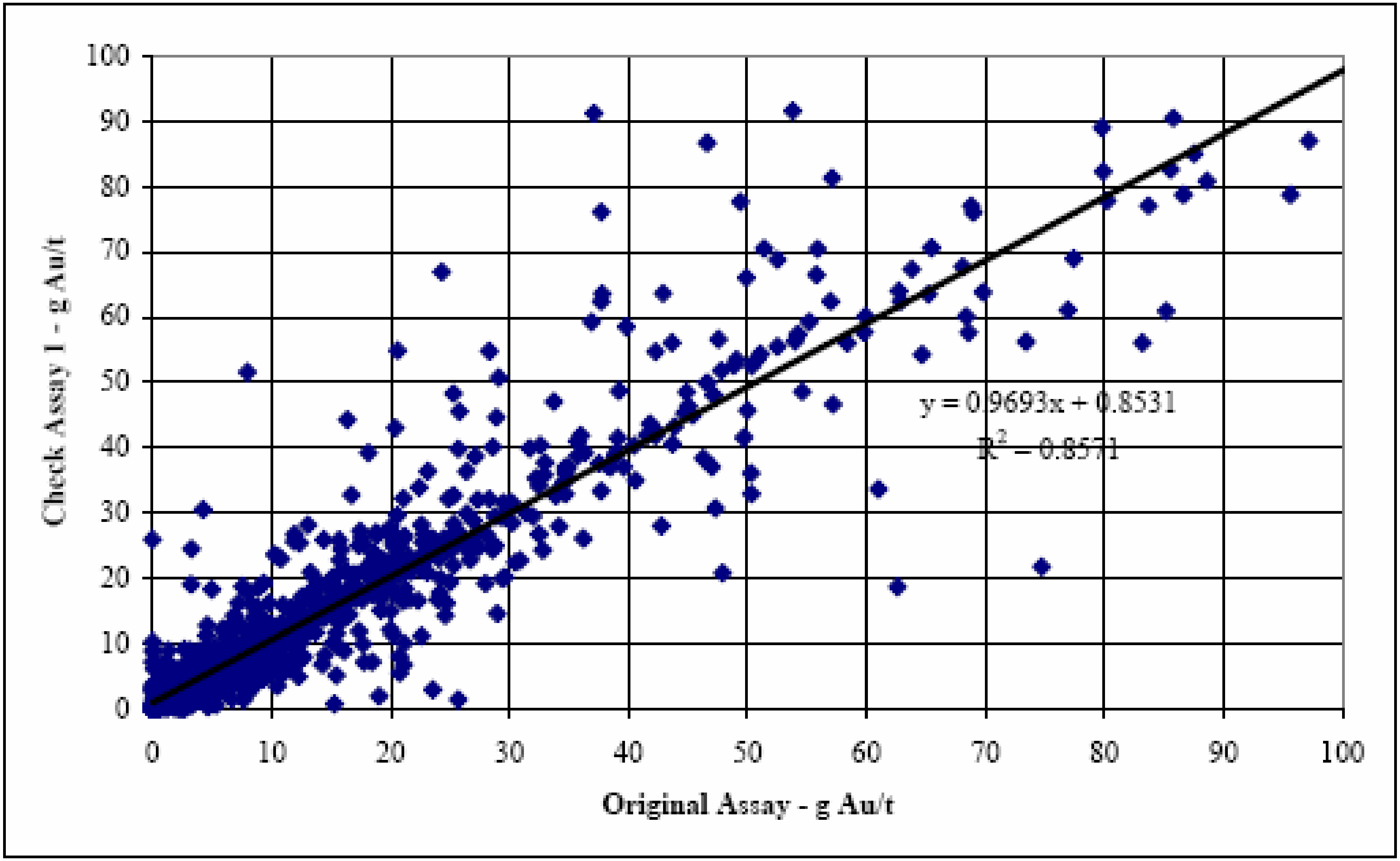
File Name: Figure 11-8.doc

Black Fox,
Timmins, Ontario, Canada

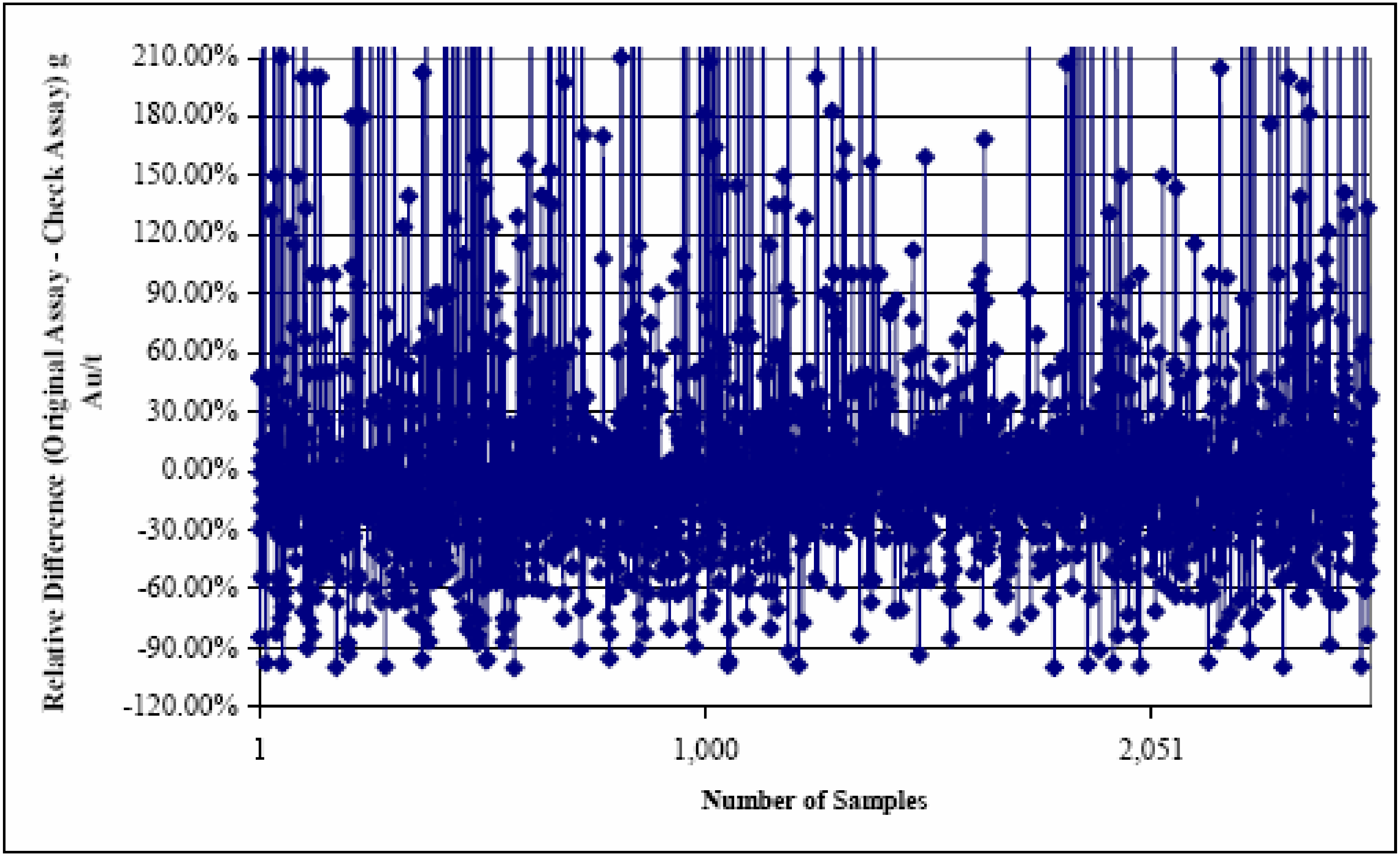
Source: Mine Development Associates

Sample Pulp Check-First Run

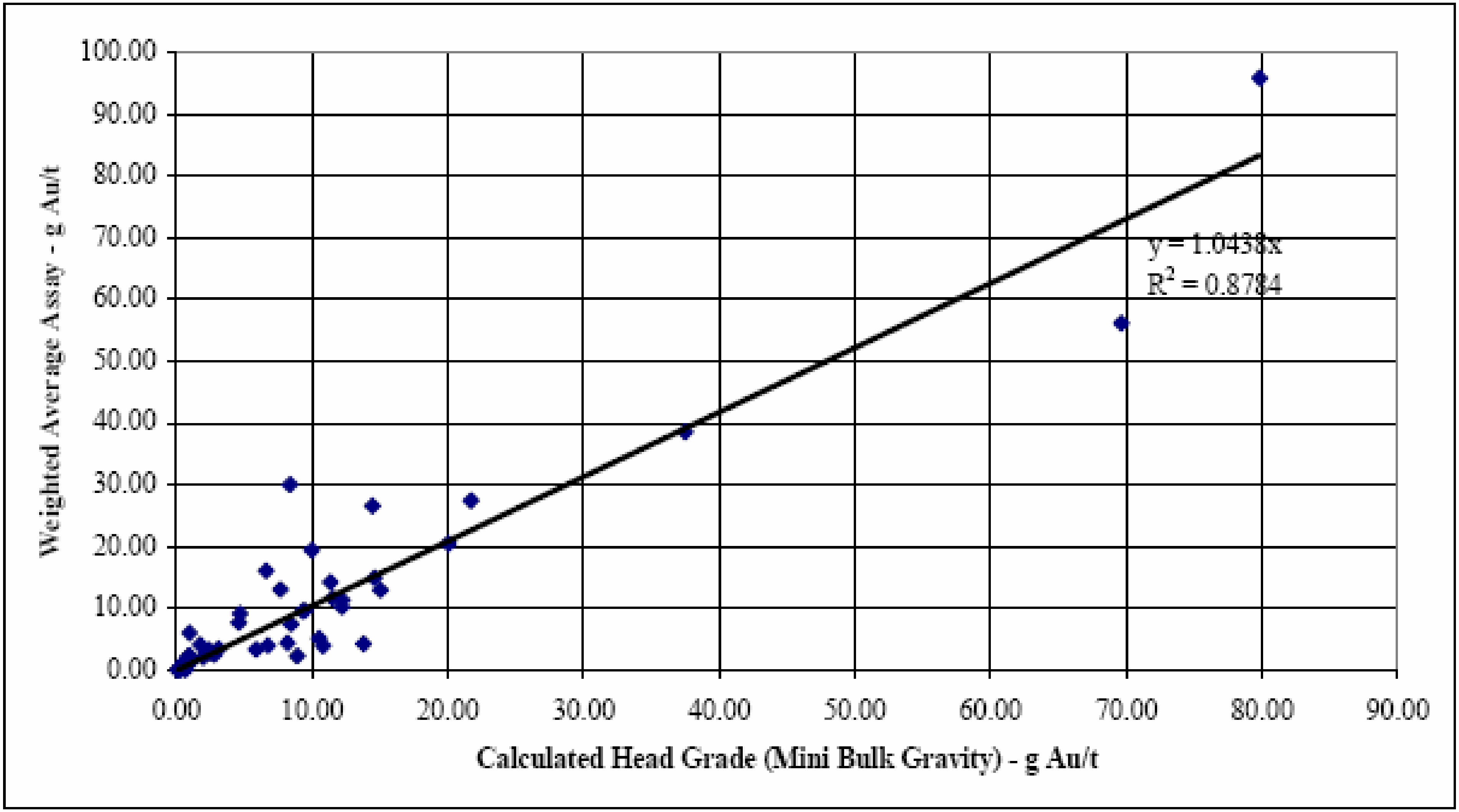
Date: 07-10-07 | Approved: BAS | Figure: 11-8



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada Source: Mine Development Associates	Original Assay vs. Check Assay from Sample Rejects
SRK Job No.: 144418		Date: 07-10-07 / Approved: BAS / Figure: 11-9
File Name: Figure 11-9.doc		



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Relative Difference-Original Assay vs. Check Assay form Sample Rejects		
SRK Job No.: 144418	Source: Mine Development Associates	Date: 07-10-07	Approved: BAS	Figure: 11-10
File Name: Figure 11-10.doc				



SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Mini-Bulk Sample Gravity Test Result**		
SRK Job No.: 144418	**Source: Mine Development Associates**			
File Name: Figure 11-11.doc		Date: 07-10-07	Approved: BAS	Figure: 11-11

12 Data Verification

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

"Although the historic check assaying is considered substandard, the data verification has been supplemented with the production of about 210,800oz-Au from the deposit and metallurgical testing of drill core.

Apollo's program for data verification is a considerable improvement of the past checks, however, while the number of checks has 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 Apollo 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 agrees with the recommendations of MDA presented above but recognizes that screen metallic assays are quite expensive and typically provide a slow turn around 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 my not outweigh the time and cost associated with it. Considering that Apollo currently intends to move forward with a open pit mining and toll milling while the mill is being constructed, model reconciliation with actual mining will provide a valid method to verify the proper usage of the assay data in the estimation technique.

Since the project database had been created and audited by others several times, SRK relied on those verifications and did not audit the project database any further.

SRK compiled the previous analysis of QC data and examined the QC data for Black Fox focusing on database consistency. SRK uncovered errors in the master assay database that were corrected through several iterations working with Analytical Solutions of Toronto, Canada and Apollo. The database is contained in three Microsoft Excel spreadsheets which include collar locations, drillhole orientations with down hole deviation surveys and assay intervals with results. The other validation work was routine validations that are normally conducted when working with 3D models.

SRK did not independently collect samples for assay.

13 Adjacent Properties

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

"The major East-West trending DPFZ extends across the property. The DPFZ is host to numerous deposits to the East and West of the Black Fox property along the 200km strike length of this structure. Figure 2-2 in Section 2 illustrates the location of many of the major mines along this structure.

The Timmins Gold camp, which produced about 70Moz-Au (www.vedron.com/timmins.html), is located about 75km to the West of the Black Fox deposit. In the city of Timmins, mining employs about 25% of the workforce (Mayor Jamie Lim). There are numerous deposits along the trend of the DPFZ within a few kilometers of the Black Fox deposit. These include the Ross Mine, New Kelore, Hislop, Zone, and Stroud (Creek Zone and Main Zone).

Table 13.1 lists the reserves of several of the larger mines along the DPFZ, while Table 13.2 summarizes the resources."

Table 13.1: Reported Reserves from Selected Mines along the DPFZ

		Proven			Probable			Proven and Probable		
Company	**Deposit**	**kt**	**Au Grade (g/t)**	**Au (koz)**	**kt**	**Au Grade (g/t)**	**Au (koz)**	**kt**	**Au grade (g/t)**	**Au (koz)**
Placer Dome/Kinross	Porcupine Joint Venture									
Placer Dome/Kinross	Pamour	5053	1.41	230	31,065	1.38	1,376	36,119	1.38	1,606
Placer Dome/Kinross	Hoyle Pond	328	12.99	137	560	12.44	224	888	12.64	361
Placer Dome/Kinross	Dome	10,935	1.03	361	10,936	2.00	702	21,871	1.51	1,063
Placer Dome/Kinross	Owl Creek									
Placer Dome/Kinross	Bell Creek									
Placer Dome/Kinross	Hollinger									
Placer Dome/Kinross	McIntyre									
Placer Dome/Kinross	Hallner									
Placer Dome/Kinross	Preston									
Placer Dome/Kinross	Paymaster									
Placer Dome/Kinross	Porcupine Joint Venture	16,316	1.39	728	42,561	1.68	2,302	58,878	1.60	3,030
Kinross	Aquarius				16	2.33	1,189	16	2.33	1,189
Newmont	Holloway				3,373	5.91	641	3,373	5.91	641
Barrick	Holt-McDermott				768	5.66	154	768	5.66	154
Kirkland Lake	Macassa	392	12.75	177	334	15.24	179.6 125	726	15.27	356
Total	**Totals of Selected Mines**	**16,7 08**	**1.68**	**905**	**47,052**	**2 .95**	**4,466**	**63,761**	**2.62**	**5,370**

Note: Source - Company Web sites; Proven and Probable totals shown in probable column.

Table 13.2: Reported Resources from Selected Mines along the DPFZ

		Measured			Indicated			Measured and Indicated		
Company	**Deposit**	**kt**	**Au Grade (g/t)**	**Au (koz)**	**kt**	**Au Grade (g/t)**	**Au (koz)**	**kt**	**Au Grade (g/t)**	**Au (koz)**
Placer	Porcupine Joint									
Dome/Kinross	Venture									
Placer Dome/	Pamour	177	0.88	4	2318	1.06	80	2,495	1.05	83
Kinross	Hoyle Pond	43	8.68	12	78	6.78	17	121	7.45	29
Placer Dome/	Dome				9235	1.72	510	9,235	1.72	510
Kinross	Owl Creek	979	2.67	84	1107	2.28	81	2,086	2.46	165
Placer Dome/	Bell Creek									
Kinross	Hollinger									
Placer Dome/	McIntyre									
Kinross	Hallner									
Placer Dome/	Preston									
Kinross	Paymaster									
Placer Dome/	Porcupine Joint Venture	1,199	2.62	100	12,738	1.68	687	13,937	1.76	787
Kinross	Aquarius									
Homer Gold	Timmins High-Grade				422	13.68	186	422	13.68	186
Newmont	Holloway							1,162	5.91	221
Barrick	Holt McDermott				685	7.90	192	685	7.90	192
Kirkland Lake	Macassa	795	11.82	330	2,192	8.71	615	2,986	9.84	945
Total	**Totals of Selected Mines**	**1,994**	**6.70**	**430**	**16,037**	**3.26**	**1,680**	**19,193**	**3.78**	**2,330**

Source: Mine Development Associates

14 Mineral Processing and Metallurgical Testing

This section excerpted from Technical Report Black Fox Project Matheson, Ontario, Canada by Mine Development Associates, August 14, 2006 and 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 1,500t/day processing plant used in the 2004 pre-feasibility study completed by MDA for the Black Fox Project. The Hislop-Beatty Project, presently known as the Black Fox Project, is located close to Matheson, Ontario, Canada and is approximately 67km east of Timmins, Ontario. The project was owned by Exall when the Owner examined the feasibility of treating mineralization from the deposit in the St. Andrew Goldfields Stock mill.

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 the Hislop-Beatty mineralization. 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.

14.1 Summary

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.

Mine production from the Black Fox gold project was shipped to the St. Andrew Goldfields (Stock) mill and the Kinross Gold Macassa mill during the periods April 1997 – September 1999 and October 1999 – May 2001 respectively.

The historical metallurgical performance achieved during the period 1997 to 2001 is summarized in Table 14.1.1.

Table 14.1.1: Historical Plant Performance

Year	Actual Reported Production Tonnes 000's	Grade g/t-Au	oz-Au 000's	Gold Recovery, %
1997	194.5	6.79	39.9	96.38%
1998	308.7	6.67	64.3	96.90%
1999	258.7	5.82	48.3	97.76%
2000	255.2	5.82	46.4	97.04%
2001	81.7	4.81	11.9	98.19%
Totals	**1,098.8**	**5.97**	**210.8**	**97.14%**

Note: Grade reported is the recovered grade, i.e. the grade necessary to produce 210,800oz-Au.

Lakefield conducted comprehensive bench scale testwork in 1996, followed by a combination of pilot plant studies and related bench scale tests in 1999. Metallurgical testwork performed by Lakefield in 1996 demonstrated the Black Fox mineralization to be free-milling and devoid of deleterious elements that could adversely affect the environment or the process. Test results indicated the potential value in deploying a gravity concentration circuit. The program determined the optimum grinds for the West and East Zones to be K_{80} 50µm and K_{80} 30µm respectively. The leach kinetics were found to be most favorable, with 30 hours of leach time being sufficient to achieve optimum results.

The main conclusions developed by the Lakefield work are outlined below:

- *The gold mineralization is readily amenable to cyanidation. When grinding in a sodium cyanide solution, approximately 90% of the gold contained in the mill feed is dissolved by the time the pulp has exited the cyclone overflow;*
- *The degree of dissolution is dependent on the leach feed grind. Optimum size distribution for west zone ore appears to be 50µm while the East Zone mineralization requires grinding 30 to 40µm;*
- *The Bond Ball Mill work index of the ore varies within the range of 14 to 17kWh/t;*
- *Gold dissolution is relatively insensitive to variations in leach times over the ranges examined;*
- *Black Fox mineralization contains no deleterious elements that could adversely affect operating efficiencies or the environment;*
- *To varying degrees, Black Fox mineralization is amenable to gravity concentration; and*
- *The ground mineralization exhibits favorable settling characteristics.*

Exall entered into a three-year toll milling agreement with St .Andrew Goldfields to process Black Fox mineralization in the Stock mill. The empirical results achieved in the plant confirmed the original test data.

Upon the expiry of the toll milling arrangement with St. Andrew Goldfields, Exall shipped the mine production to the Macassa mill. Metallurgical results continued to confirm the amenability of the Black Fox mineralization to conventional cyanidation followed by CIP technology. While the higher sulfide material generated poorer results and consumed more cyanide, the problems were mitigated through effective blending of the mill feed. Annual gold recoveries exceeded 97% at the Macassa mill.

Programs of laboratory and pilot plant metallurgical studies were implemented in 1999 by Lakefield to examine alternate process options by which overall project economics could be enhanced. Based on examination of six composite samples of varying grades and sulfide content, the use of spiral concentrators was deemed to offer a means by which up to 80% of the gold could be recovered in 15% of the feed weight, given a primary grind of 150µm.

Cyanidation tests, performed on the spiral concentrate, ground to K_{80} 40µm, and un-ground spiral tails, at a nominal K_{80} 150µm, achieved leach residues similar to those achieved in the Stock mill.

Based upon these, and associated test results, preliminary economic analyses were prepared to assess the potential economic consequences of adopting such a circuit. The results of these analyses, conducted by Richard Swider, indicated that the use of a gravity pre-concentration stage, in conjunction with a coarser primary grind, would be worthy of consideration in any future Feasibility Study.

In summary, the Black Fox mineralization is free-milling and environmentally innocuous. Although visible gold is present, relatively fine grinds are required in accordance with the current flowsheet, to achieve optimum results. Upside potential might be realized through the adoption of a gravity circuit in conjunction with a coarser grind. Other process alternatives should also be included in a series of trade-off studies, prior to finalizing the basis for a Feasibility Study. In any event, confirmatory testwork should be performed on samples of mineralization deemed representative of grades and species to be mined in accordance with the new mine plan.

14.2 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 14.2.1.

Table 14.2.1: 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. Sulfide sulfur in the East Zone High Grade was measured at 3.05%. The highest equivalent sulfide 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 14.9kWh/t, respectively.

Gravity concentration tests were performed on each composite sample, yielding the results summarized in Table 14.2.2.

Table 14.2.2: 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 K_{80} 70µm through 50µm to 30µm. 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 14.2.3, based on 48-hour leach times, a K_{80} 50µm grind for West Zone material and K_{80} 30µm grind for East Zone mineralization.

Table 14.2.3: 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

Cyanide consumed during leach, not including initial cyanide to 0.5g/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 K_{80} 30µm grind. A favorable unit area rate was achieved, being less than 0.2m^2/t/day in all cases in which modest flocculant additions were used.

In conclusion, the favorable 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.

14.3 Stock Mill Operations (1996-1999)

The Stock mill, designed by Leslie Engineering, was constructed in 1988 but was shut down in 1994. The plant included 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; and*
- *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.

Black Fox mineralization in excess of 1.6g/t-Au was delivered to the Stock mill by 35t capacity highway trucks. A 610mm x 914mm jaw crusher was replaced with a 1,067mm x 1,3716mm unit *in 1999. The primary crusher discharge was further crushed in a 1,300mm short head cone crusher, in closed circuit with a screen, prior to being conveyed to the fine ore bin.*

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

The cyclone overflow gravitated to an 18.3m dia. thickener, the overflow from which was pumped to seven carbon columns. The thickener underflow was pumped to four leach tanks to provide a nominal 27 hours retention time. Leach tailings were pumped to five CIP tanks that provided 3 hours retention time. Carbon from the columns and CIP circuits was stripped at 142ºC. 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 1.0m^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 5,000t.

Thus, at a nominal 1,000t/day 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 flowmeters, 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.

14.4 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. Also, the Macassa plant included a pre-thickener leach tank. Further, being designed to treat 2,000t/day, the plant was oversized for the nominal 1,000t/day

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 600kW ball mills that operated a 24 hours per day, 5 days per week schedule. The primary mill was steel-lined and charged with 100mm grinding media. The secondary rubber-lined mill was charged with 25mm balls. The nominal 1,000t/day (40 to 45t/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 19.8m dia. thickener, the former providing a residence time of 30 hours. Given the favorable leach kinetics, gold dissolution was typically 90% complete by the time pulp entered the thickener.

As for the Stock circuit, the thickener overflow was treated in carbon columns while, for much of the time, the thickener underflow passed through a single leach tank, thus providing 60 hours of leach time in total. The leach tank was only used to satisfy certain logistical requirements, rather than to provide necessary incremental leach time. A 6-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 S_o *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 14.4.1, together with the measured head grade and calculated head grade.*

Table 14.4.1: A 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
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.

Table 14.4.2 below compares the Measured Head, Calculated Cyclone Overflow Head and Calculated Head values for the period January through March 2001.

Table 14.4.2: Comparison of Head Values (g/t-Au), January to March Inclusive 2001

	Measured	Cyclone Overflow	Calculated
Average	6.28	4.86	5.08
Standard Deviation	5.05	2.01	

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.

14.5 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 2.07g/t-Au to 14.0g/t-Au. Descriptions of the samples are provided in Table 14.5.1.

Table 14.5.1: 1999 Test Program Sample Description

Composite	Constituents	Grade g/t-Au
General Composite #1 (GC1)	60% low grade at 5.48g/t-Au 30% high grade at 14.0g/t-Au 10% very low grade at 1.84g/t-Au	10.2
General Composite #2 (GC2)	50% GC1 at 10.2g/t-Au 50% low grade at 5.48g/t-Au	7.67
Low Grade Composite (LG)	100% low grade at 5.48g/t-Au	5.48
Very Low Grade Composite (VLG+A)	75% very low grade at 1.84g/t-Au 25% drum A	2.07
High Grade Composite (HG)	100% high grade at 14.0g/t-Au	14.0
High Sulfide composite (HS+HG)	50% high sulfide at 8.45g/t-Au 50% high grade at 14.0g/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 166µm;*
- *At a feed grind of 150µm, 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 (0.14g/t-Au) were achieved when the 150µm spiral tailings were subjected to conventional cyanidation;*
- *Spiral concentrates, reground to 40µm, were leached to produce leach residues of grades (0.13g/t-Au) similar to those at the Stock mill;*
- *The High Sulfide Composite proved to be more refractory, reflecting experience in the operating plant. Thus, the 40µm leach residue graded 1.8g/t-Au, while the overall leach residue (spiral concentrate and tailings) was in the range of 0.40g/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 17kWh/t should be used for plant design purposes;*
- *Flotation was found to be effective in the treatment of sulfide mineralization, but of marginal value when processing low sulfide material; and*
- *Thickener unit area determinations for the leached spiral tailings (158µm) and the leached spiral concentrate (40µm) were 0.274 and 0.173m^2/t/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.

14.6 2006 Mini Bulk Sample Gravity Tests

Francis Pitard (2005) recommended 200 mini-gravity tests be completed and compared back to the original sample grades. Apollo started this program with a total of 58 tests completed, averaging about 14kg per test. Of these tests, 47 were completed on "ore grade" material, 6 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 K_{80} 114µm. These tests are summarized in more detail in Section 13.

14.7 Implications of Test Data and Empirical Results on Conceptual Plant Design

The rapid leach kinetics consistently experienced in the laboratory and plant should be exploited to the full. By so doing:

- *Gold is extracted at the earliest opportunity;*
- *The production and handling of high-grade gravity concentrate products is avoided; and*
- *Leach circuit retention times can be reduced.*

Pending further bench scale testwork, a variant of the concept developed in 1999, has been adopted and constitutes Owner Mill Case 1 in this Preliminary Feasibility Study. Thus, the primary grind will be 80% 150µm and a gravity concentrate will be produced from the cyclone *overflow product. The concentrate will be reground to 80% 40µm, prior to leaching. However, in this case, the reground concentrate will be combined with the gravity tailing to feed the pre-leach thickener. By using this circuit, and grinding in cyanide, the recovery of gold to the gravity concentrate will be reduced significantly. Nevertheless, such a circuit will enhance the rate at which minor amounts of the more refractory mineralization will be leached. Further, since testwork has demonstrated gravity recoveries improve with increasing feed grades, the proposed circuit will alleviate any potential difficulties that may result from short-term feed grade "spikes". Further gravity and leach work, in conjunction with mineralogical and modal analysis, is required to examine the circuit, at a bench scale level, prior to the preparation of a detailed feasibility study.*

Historically, leach times of 30 hours have been adopted. Given the operating and test results reported above, a leach time of 18 hours should provide adequate time to achieve the optimum economic leach. This supposition must be confirmed in trade-off studies based on the laboratory work noted above.

14.8 Gold Recovery Projections

Gold recoveries are plotted against feed grades in Figure 14-1 for the period January 2000 through to the termination of operations in May 2001. With the exception of August 2000, all gold recoveries exceeded 96%, indicating a modest improvement in gold recovery as feed grades increased from a nominal 2.5g/t-Au to 10.0g/t-Au.

The precise reason(s) for the poor gold recovery in August 2000 were never clearly identified. It was strongly suspected, however, that the milling of S1 mineralization, reportedly higher in sulphides content, could have contributed to the problem. Exall determined that swings in gold recovery could be largely mitigated by blending ores prior to treatment.

An overall gold recovery of 97% is proposed for this pre-feasibility study, based on the assumptions that:

- *Ores to be treated will exhibit similar metallurgical characteristics to those processed in 2000 and 2001;*
- *The feed grade will not fall beneath 4.0g/t-Au, and*
- *Ores of differing grades and mineralogical composition will be blended, at least to the same degree as that achieved during the Macassa operations.*

SRK has determined that the overall gold recovery for the purposes of this report will be 96%. This value is based on further review of available information and conservative in nature.

14.9 On-Site Mill

A variant of the Macassa mill flowsheet, designated Owner Mill, has been adopted as the base case in the pre-feasibility study.

Process flowsheets have been prepared to the degree of detail deemed appropriate for this pre-feasibility study. Principal items of process equipment were sized and used to generate conceptual plant layouts. The layouts are by no means optimized, but simply used to determine approximate building dimensions and conveyor lengths.

Flowsheets and general arrangement drawings are included in Appendix E. The following plant description outlines the Owner Mill flowsheet and layout. As in most mills, the crushing and *grinding sections of the plant comprise the most capital intensive cost centers. For the purpose of this pre-feasibility study, it is assumed that a SAG/ball mill circuit will be used. Alternative arrangements include:*

- *Single-stage fine crushing, primary ball mill, secondary ball mill;*
- *Two-stage fine crushing, primary ball mill, secondary ball mill; or*
- *Single or two-stage fine crushing with a rod mill and ball mill.*

There are examples of the SAG option and each of the alternative circuits in northern Ontario gold plants. The options offer varying degrees of operating simplicity/complexity, with associated differences in capital and operating costs. It is beyond the scope of this study to perform detailed economic trade-off studies to determine the most cost-effective means of comminution. Further, if the SAG option were to be pursued, any trade-off studies would be premature, pending completion of the first phase of SAG amenability

14.9.1 Crushing

Mine trucks will deliver ore to the primary crusher feed hopper, or to stockpiles located in close proximity to the crusher. The stockpiles will be managed to minimize fluctuations in mill feed grade. A front-end loader, assigned to, and operated by Mine Department, will reclaim ore from the stockpiles and charge the crusher feed hopper.

Ore will be withdrawn from the 150t capacity hopper by means of a 1,16mm x 4,877mm (46in x 16ft) vibrating grizzly which will feed material over 50mm in size to a 914mm x 1,067mm (36in x 48in) jaw crusher. The grizzly undersize product and the crushed ore will fall on to #1 Conveyor, a 914mm (36in) covered belt that will transport the material to the 2,300t live capacity crushed ore bin. The #1 Conveyor will be furnished with a belt scale and a plough to divert crushed ore from the conveyor to the ground, on the rare occasion when it is required to accumulate stockpiles of crushed ore. A reclaim feed hopper is provided into which a front-end loader can load crushed ore from the stockpile on to #1 Conveyor.

The steel-framed, metal-clad primary crusher building will be heated and serviced with a wet dust collector. The slurry so produced will be pumped to the grinding circuit. A 30t/5t capacity bridge crane will facilitate maintenance of the crushing equipment. The primary crushing circuit is designed to crush 175t/hr ore to 100mm. Capacity is available to accommodate potential increases in production rates.

14.9.2 Crushed Ore Storage

Crushed ore will be stored in an insulated, steel cylindrical bin, the design criteria for which are based on successful and similar installations in the Hemlo camp. Ore will be withdrawn from the bin by three vari-speed 914mm x 3,658mm (3ft x 12ft) apron feeders which will discharge on to the SAG mill feed conveyor. The lower part of the crushed ore storage bin will be enclosed in a metal-clad, steel-framed and insulated building.

14.9.3 Semi-Autogenous Grinding

A 762mm wide conveyor belt, #2 Conveyor, will deliver ore to the SAG mill feed chute. The 5,486mm x 1,981mm EGL (18ft x 6.5ft EGL), 746kW (1,000hp) SAG mill will reduce the size of the mill feed to a 1,200mm transfer size. The mill will be steel-lined and furnished with a trommel screen. The trommel screen undersize will gravitate into a pump-box from which the pulp will be pumped to a vibrating screen. The screen overflow will discharge into the SAG mill *feed chute while the screen underflow will gravitate to the cyclone feed pump box. (Flowsheet PFD-002 illustrates the alternative configuration that would be required, should pebble crushing be necessary.)*

Lime slurry and sodium cyanide solution will be introduced into the SAG mill feed chute. The grinding circuit pulp pH will be maintained within the range of 10.5 to 11.0. Sodium cyanide solution will be added to maintain a free cyanide concentration of 0.5g/L.

14.9.4 Pebble Crushing

SAG amenability testwork will determine the requirement for a pebble crusher. Should pebble crushing be required, the trommel screen undersize will feed a double deck, low head, horizontal vibrating screen, 1,219mm x 2,438mm (4ft x 8ft). The two screen oversize products will discharge on to Conveyor #3, a 762mm (30in) wide transfer belt, which will feed the 457mm (18in) wide Conveyor #4. Equipped with two magnets and a metal detector, this conveyor will feed pebbles at the rate of 24t/hr to a 914mm (36in) wide shuttle conveyor. Should scrap steel be identified by the metal detector, after the belt charge has passed beneath the magnet, the shuttle conveyor will withdraw automatically to divert the pebbles from the pebble crusher feed to the SAG mill feed conveyor. The shuttle conveyor may be reversed and moved, to allow uncrushed pebbles to be stockpiled outside, when the pebble crusher is out of service.

Under normal circumstances, an extra heavy duty, 914mm (3ft) short head cone crusher will crush the pebble to a nominal 12mm size. The cone crusher will discharge directly on to the SAG mill feed conveyor. It is emphasized that, subject to the results of essential SAG mill amenability tests, there may be no requirement for this circuit.

14.9.5 Ball Milling

A 3,353mm x 5,182mm (11ft x 17ft EGL), 746kW (1,000hp) ball mill will operate in closed circuit with three 381mm (15in) cyclones, one of which will be retained in standby mode. The mill will be rubber-lined and equipped with a trommel screen. The circuit is designed to accommodate a 300% circulating load to produce a K_{80} *150µm cyclone overflow.*

Lime slurry and sodium cyanide solution will be metered into the cyclone feed pump box. The grinding circuit will be furnished with ancillary equipment, selected to maximize the ease of operation and minimize operating costs. Thus, a liner handler and bolt removal tool will be provided to minimize shutdown times for the replacement of SAG mill liners, lifters and grates. A 30t/5t overhead crane will service the grinding section. Ball storage bins for both the 100mm and 38mm grinding media will be provided, each with a capacity of 30t. The 25mm regrind balls will be delivered and stored in metal drums. A ball bucket will be used to feed balls to the respective mills. A small compressor will be dedicated to grinding mill clutch service.

14.9.6 Trash Screening

A 1,524mm x 2,436mm (5ft x 8ft) vibrating screen, furnished with a 1mm slotted urethane deck, will remove trash from the cyclone overflow stream. The trash, expected to contain ball chips and wood, will be collected in a trash tote bin. Failure to effectively remove this tramp material will adversely affect the operation of the gravity concentrator, the CIP carbon retention screens and the strip column.

14.9.7 Gravity Concentrate Grinding

A 149kW (200hp) tower mill will grind the gravity concentrate to a nominal K_{80} 40µm. The mill will operate in closed circuit with three, 152mm cyclones. The cyclone overflow product will flow by gravity to the pre-leach thickener feed tank.

14.9.8 Pre-Leach Thickening

Flocculant will be added to the slurry feeding a 19,800mm (65ft) dia. thickener, which will thicken the pulp to a 45% pulp density, by weight, in preparation for leaching. The thickener will be constructed on grade, outside the mill building, with protection from the elements provided over the bridge and drive.

Thickener underflow pulp will be pumped to the leach circuit while the overflow stream will be pumped to the carbon columns.

14.9.9 Leaching

Leaching will be accomplished in five 8,500 mm x 9,500 mm (27.9ft x 31.2ft) leach tanks, each equipped with a 37.3kW (50hp) agitator. The leach circuit will provide a nominal 18-hour retention time.

The pulp pH will be maintained within the range of 10.5 to 11.0 and sodium cyanide solution will be added to maintain a free cyanide concentration of 0.5g/L. Air will be blown into the tanks by one of two 850m^3/hr (500cfm) blowers; the second unit being on standby duty.

14.9.10 Carbon-in-Pulp Circuit

Reclaim water will be added to the leach circuit tailing to lower the pulp density to 40% solids by weight, prior to feeding the five 6,000mm x 6,500mm (19.7ft x 21.3ft) CIP tanks which will provide a nominal retention time of five hours. The tanks, arranged in series, will be equipped with 18.7kW (25hp) agitators.

Carbon will be transferred in counter-current mode, using recessed impeller pumps. A 2m^2 NKM carbon retention screen, or equivalent, will be installed in each CIP tank to prevent carbon backflow. Carbon concentrations in the slurry will be within the range 20 to 25g/L. Discharge from the final CIP tank will pass over a 1,219mm x 3,042mm (4ft x 10ft) carbon safety screen, the underflow from which will pass into the final tailings pump-box. The carbon retained in the screen oversize fraction will be returned to the final CIP tank.

Carbon recovered from the first CIP tank will be pumped to the loaded carbon screen and hence to the loaded carbon storage bin.

14.9.11 Carbon-in-Column Circuit

Six carbon columns, 2,000mm x 3,850mm (6.56ft x 12.6ft) will be deployed to adsorb gold from the thickener overflow stream. Carbon will be advanced in counter-current direction by means of eductors. The barren solution will flow to the barren solution tank while the carbon will be transferred to the loaded carbon screen.

14.9.12 Carbon Stripping

The carbon strip circuit is designed to process 2.5t of carbon per batch in a single 1,500mm x 3,353mm (5ft x 11ft) insulated strip column, furnished with a 60° conical bottom. The strip solution, at 10g/L NaOH and 2g/L NaCN, will be heated to 140°C through a plate and frame *heat exchanger by a 500kW boiler. The operating pressure will be 550kPa (80psi). The strip vessel will be made from carbon steel while 304 stainless steel will be used for the carbon screens.*

14.9.13 Acid Washing

The FRP acid wash tank will be of similar dimensions to the strip vessel. Washing will be accomplished using nitric acid, which will be neutralized with sodium hydroxide solution.

14.9.14 Carbon Regeneration

It is proposed that 50% of the carbon will be regenerated in a 609mm x 6,706mm (24in x 22ft) horizontal kiln, equipped with heating 90kW heating elements. Given the nature of the ore, the regeneration of all the carbon may not be required, but would contribute to the production of carbon fines.

14.9.15 Electrowinning

Pregnant solution from the strip column will flow through a 2m^3 electrowinning cell from which precious metal sludge will be recovered on mild steel wool cathodes.

14.9.16 Smelting

The sludge from the electrowinning cell will be dried and charged into an induction furnace to produce doré metal.

14.9.17 Buildings and Structures

The crusher building and structure surrounding the base of the crushed ore bin will be of "stick-built" steel construction, metal clad and insulated. The primary crusher motor control center (MCC), to be assembled in a module off-site, will be located in close proximity to the crusher building. By so doing, on-site construction costs are minimized and the module will be situated in a relatively dust free environment.

The mill building will be a pre-engineered, metal clad and insulated structure. The MCC will be on the ground floor with the offices, washroom, lunchroom, control room and meeting room all located above. The MCC and other facilities will be pre-assembled in modular form, prior to shipping.

The conveyors will be constructed off-site in sections to facilitate construction. The assay laboratory will be located in a separate pre-engineered building. Modular laboratories were considered and found to be too expensive.

14.9.18 Power Supply

The peak power demands will approximate 4.1MW. For the purpose of this study, it is assumed that the existing infrastructure will be adequate to provide power for the plant, and all other load centers on the mine site. More precise peak demand estimates, produced during the Feasibility Study, should be compared with on-site capabilities. A 450kW diesel-electric generator will provide emergency service.

14.9.19 Reagent Supply and Distribution

Sodium cyanide solution, delivered in bulk liquid form, will be stored in a nominal 50,000L capacity tank to be provided by the supplier. Cyanide solution will be transferred from this tank to a distribution tank from which metering pumps will be used to service the various points of *application in the grinding, leach and carbon strip circuits. The sodium cyanide area will be serviced with a dedicated sump pump.*

Sodium hydroxide will also be delivered in bulk form and stored in a 34,000L tank from which a metering pump will deliver the reagent to the carbon stripping circuit. Flocculant will be delivered in bags, which will be manually discharged into a packaged flocculant mixing system. The flocculant will be diluted in a supply tank prior to delivery to the thickener.

Bulk lime will be pneumatically unloaded from the truck into a 50t capacity silo, equipped with a vent filter. A bin activator will facilitate the flow of lime into a packaged slaker unit. Slaked lime will be distributed to the various points of application through a circulating loop. The lime area will be serviced with a sump pump.

Nitric acid will be delivered in barrels and stored in a polypropylene mix tank. An injection pump will transfer the acid into the acid wash column. Lead nitrate will be used, as required, by manually feeding the reagent from bags in to the CIP circuit.

Copper sulfate and ferric sulfate will both be received in bags and added, in solid form to the water treatment circuit by means of screw feeders.

Hydrogen peroxide will be received in 1,400kg totes from which the reagent will be pumped directly into the water treatment circuit.

The reagent mixing area will be serviced with an exhaust fan and a 3t capacity overhead crane. Safety showers/eye wash stations will be installed at appropriate locations.

14.9.20 Water Supply and Distribution

The fresh water supply will likely be from a combination of underground mine pumping, surface supply, and well supply. About 400g/m of fresh water is required for the mill operation, with the majority of water for the operation coming from recirculation of water from the tailings facility. The underground is currently being pumped at the rate of 100 to 150g/m.

Fresh water will be supplied to the insulated fresh water/firewater storage tank. Fresh water will be distributed to the process, with a portion treated in a packaged chlorinator system, prior to storage in a potable water tank. Two gland seal water pumps, one in standby service, will withdraw water directly from the freshwater tank. Two electric and two diesel-powered pumps will afford fire protection.

14.9.21 Water Treatment

Reclaim water will be bled from the system on an intermittent basis to satisfy water balance and process water quality considerations. Copper sulfate, as a catalyst, will be added to the feed to the first of two water treatment tanks. Hydrogen peroxide, introduced into the first tank, will destroy all free and metal complexed cyanides, with the exception of iron cyanide. The cyanides will be converted to cyanates which slowly hydrolize to form carbon dioxide and ammonium ions. The copper, nickel and zinc metals form hydroxide precipitates. Since iron cyanide is too stable to be oxidized by hydrogen peroxide, it is removed by complexing with copper to form copper ferrocyanide. Ferric sulfate is added to the second water treatment tank to precipitate any arsenic present.

14.9.22 Compressed Air

Two 672m^3/hr, 690kPa (400cfm, 100psi) compressors will provide air for general operations and maintenance purposes. Air taken from the loop will pass through an air dryer for use in instrumentation.

14.9.23 Processing Design Assumptions

Table 14.9.23.1 shows the Owner Milling criteria and assumptions."

Table 14.9.23.1: Owner Milling Assumptions

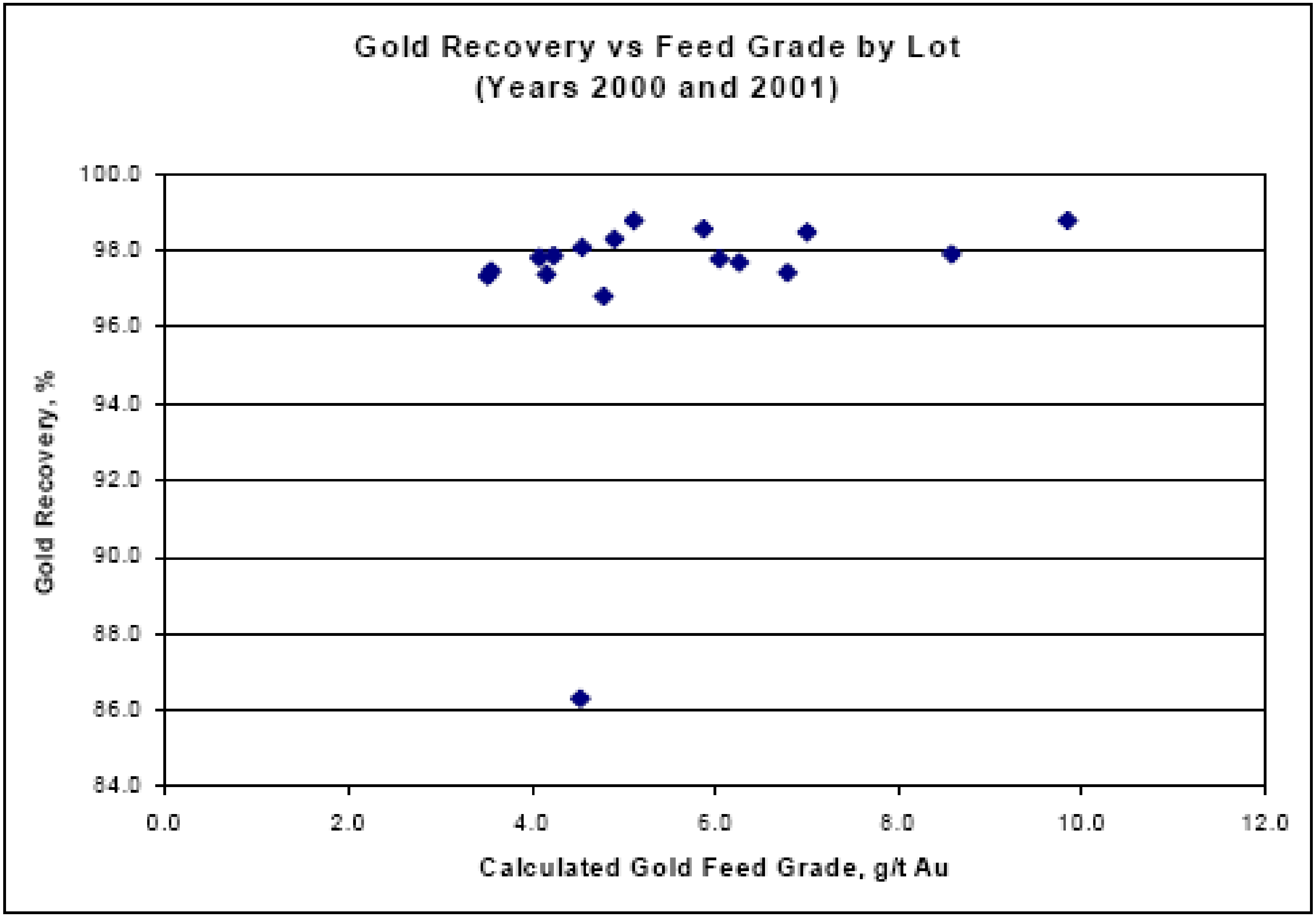
	Item Units	Value
General		
Average daily mill throughput	t/day	1,500
Operating days per year		360
Annual mill throughput	t	540,000
Run-Of-Mine Ore		
Feed grade	g/t-Au	6.5
Specific gravity of dry solids		2.8
Moisture	% water	2
Crushing		
Primary crushing schedule	hrs/shift	12
	shifts/week	14
Scheduled operating time	hrs/day	16
Crusher availability	%	70
Primary crushing rate	t/hr	145.6
Design factor	%	20
Design crushing rate	t/hr	174.7
Type of crusher		Jaw
Open side setting	mm	100
Size of crusher (36in x 48in)	mm	914 x 1,067
Crushed ore live storage capacity	t	2,300
Primary Grinding Circuit		
Average daily mill feed rate	t/day	1,500.00
Plant availability	%	92
Plant operating rate	t/day	1,630.40
SAG Mill		
Feed size, P_{80}	mm	125,000
Transfer size, P_{80}	mm	1,200
Ball mill work index	kWh/t	17
SAG efficiency factor		1.8
Mill feed density	% solids	72
Pebble recycle percentage, if required	%	35
Mill size, 18ft x 6.5ft EGL	mm	5,486 x 1,981
Mill power	kW	746
Ball size	mm	100 (4in)
Ball consumption	kg/t	0.45
Reagent addition		Lime/cyanide
Ball Mill		
Feed size, P_{80}	mm	1,200
Product size, P_{80}	mm	150
Ball mill work index	kWh/t	17
Circulating load	%	300
Mill feed density	% solids	80
Mill size, 11ft x 17ft EGL	mm	3,333 x5,182
Mill power	kW	746
Ball size	mm	38(1.5in)
Ball consumption	kg/t	0.5
Reagent addition		Lime/cyanide
Classification		
Cyclones, number		3
Type of cyclone	Or equivalent	Gmax 15
Cyclones, size	mm	381 (15in)
Overflow to gravity separation	% solids	22.7
Underflow to ball mill feed	% solids	80
Trash De-Grit Screen		
Type of screen		Vibrating
Screen size	mm	1,524 x 1,219
Aperture (slots) in urethane deck	mm	1.4

Table 14.9.23.1: Owner Milling Assumptions (Continued)

	Item Units	Value
Gravity Concentration		
Type of unit		Spirals
Model of unit, or equivalent	Carpco	MC7000
Continuous or batch operation		Continuous
Number of units per bank		12
Number of spiral banks		2
Weight recovery to concentrate	%	15
Gravity concentrate to regrind	t/hr	10.2
Gravity tail to thickener	t/hr	57.7
Gravity Concentrate Regrind		
Type of mill		Tower mill
Feed size, P_{80}	mm	150
Product size, P_{80}	mm	40
Ball mill work index	kWh/t	17
Tower mill efficiency factor	%	70
Mill feed density	% solids	55
Mill power	kW	149
Ball size	mm	25
Ball consumption, per tonne fresh mill feed	kg/t	0.1
Reagent addition		Lime/cyanide
Pre-Leach Thickener		
Thickener underflow density	% solids	45
Feed rate	t/hr	67.9
Thickener dia., 65ft	mm	19,812
Unit area	m^2/t/day	0.189
Nominal leach time	hrs	3.5
Reagent addition		Lime/cyanide/floc
Carbon Adsorption		
Number of columns		6
Column dia.	mm	2,000
Column height	mm	3,850
Nominal solution upflow	$m^3/hr/m^2$	10.8
Carbon loading	g/t-Au	5,000
Carbon transfer frequency, design	t/day	2.5
Leaching		
Feed rate	t/hr	67.9
Leach density	% solids	45
Leach time	hr	18
Volumetric allowance for contained air	%	15
Number of stages		5
Pulp pH		10.5 to 11.0
Nominal air addition rate	$m^3/hr/m^3$vol	3
Number of blowers (including 1 standby)		2
Reagent addition		Lime/cyanide Lead/nitrate
Carbon-In-Pulp		
Feed rate	t/hr	67.9
Pulp density	% solids	40
Retention time	hr	5
Volumetric allowance for contained air	%	15
Number of stages		5
Carbon concentration	g/L	20 to 30
Carbon transfer	t/day	2.5
Carbon Stripping		
Size of batch	t	2.5
Strip duration	hr	12
Strip rate	BV/hr	2

Table 14.9.23.1: Milling Assumptions (Continued)

	Item Units	Value
Strip temperature	ºC	140
Strip pressure	kPa	550
Strip solution		
Sodium hydroxide	g/L	10
Sodium cyanide	g/L	2
Strip frequency	per day	1
Boiler Type		Electric
Boiler Capacity	kW	500
Heat exchangers		Plate and frame
Carbon Acid Washing		
Nitric acid solutions strength	%	3
Sodium hydroxide solution strength	%	3
Percentage carbon washing	%	100
Cycle time	hr	10
Carbon Regeneration		
Type of Equipment		Horizontal 90kW Electric
Percentage of carbon regenerated each day	%	50
Kiln Capacity	kg/d	1,500
Carbon attrition		Agit tank
Carbon sizing		Vib. Screen
Carbon losses	kg/t ore	0.07
Electrowinning		
Number of cells		1
Cell capacity	m^3	2
Operating temperature	°C	80
Cathode		Steel wool
Dore Production		
Induction furnace	kW	75
Tailings Losses		
Leach Solids	g/t-Au in solids	0.18
	%	2.8
CIP solution loss	g/t-Au in solution	0.008
	%	0.2
Total Loss	%	3
Gold to Solution by Stage		
Primary grinding, concentrate regrinding	%	90
Leach / CIP	%	7
Total Gold Recovery	%	**97**



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SRK Job No.: 144418

File Name: Figure 14-1.doc

Black Fox,
Timmins, Ontario, Canada

Source: Mine Development Associates

Gold Recoveries vs. Feed Grades

Date: 08-07-07	Approved: DKY	Figure: 14-1

15 Mineral Resource and Mineral Reserve Estimates

15.1 Resource Estimation

15.1.1 Drillhole Database

The drillhole sample database was compiled by Apollo and reviewed for QA/QC by Analytical Solutions of Toronto Canada and is determined to be of good quality. The database used for the resource estimate, consists of three Microsoft Excel spreadsheets containing collar locations, drillhole orientations with down hole deviation surveys and assay intervals with results. The appropriate codes for missing samples and no recovery were used during the modeling procedures. SRK did not provide any independent review of sampling quality and has used the data as it was provided by Apollo. The assay database contains several columns of gold assays representing the original 30g and 50g assays as well as numerous repeat assays included some screen metallic analyses. The Au 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 were used in place of the original assay. This resulted in the averaging of up to five assays in some instances. This approach was used to take advantage of larger assay charges and screen metallic samples were possible. Although this method can introduce assay bias, the author felt it was the best approach to try to overcome sampling difficulties described previously in this report.

The resource database contains information from 1,818 drillholes totaling 322,744m of drilling. The maximum drillhole depth is 995m and the average is 178m. 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.

Analysis of the sample intervals shows that the majority range between 0.5m to 2.0m however, there is a very small percentage of intervals up to 8m. The average sample interval is 0.9m.

15.1.2 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 1,000m 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 DPFZ. 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 resource estimation, the mineralization has been subdivided into two main types based on its continuity and style. "Flow Ore" occurs as lens shaped zones which have good geologic and grade continuity. The ore is strongly foliated with pervasive shearing and can be correlated reasonably well between adjacent drillholes. The "Green Carbonate Ore" occurs as discontinuous zones or pods with distinctive quartz and or carbonate alteration. In some areas, the mineralization occurs along a foliated fabric cut by discrete shear zones, in other areas, it occurs as a stockwork veining. Each of the two main ore types have been estimated by unique modeling procedures.

15.1.3 Compositing

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

The raw drill data was composited into 1.5m intervals starting at the collar and continuing to the bottom of the hole. The 1.5m assay composites were plotted on histogram and cumulative frequency graphs for comparison to the raw data and to access appropriate capping levels. Review of the cumulative frequency plot with composites greater than 100g/t-Au showed a significant jump and accelerated increase in Au values above the 170g/t-Au level (Figure 15-1). Based on the outlier nature and accelerated grade increase, the Au values were capped at 170g/t-Au after compositing, to prevent unwanted high grading. This resulted in 37 drillhole composites ranging from 178g/t-Au to 1,168g/t-Au and averaging 383g/t-Au being reduced to 170g/t-Au. The capped composites represent 0.034% of the composite database. The histogram and cumulative probability plots described above are presented in Appendix B. Statistical comparisons of raw assays, composite assays, and block model assays are summarized below in Table 15.1.3.1.

Table 15.1.3.1: Statistical Comparison of Raw Assays, Composite Assays and Block Model Assays

Data Group	Mean	Median	Maximum	Variance
Raw Assays in Flow Ore	7.05	0.64	1,942.0	1,703.00
1.5m Capped Composites in Flow Ore	5.35	1.10	170	175.4
ID2 Blocks in Flow Ore	4.46	2.03	119.4	49.96
ID3 Blocks in Flow Ore	4.45	1.94	140.5	54.99
Kriged Blocks in Flow Ore	4.44	2.14	90.7	43.69
1.5m Capped Composites used in Green Carbonate Ore	5.4	1.95	170.0	232
ID2 Blocks in Green Carbonate Ore	4.6	1.87	170.0	84
ID3 Blocks in Green Carbonate Ore	3.9	1.36	170.0	83
Kriged Blocks in Green Carbonate Ore	4.0	1.85	100.2	41

15.1.4 Specific Gravity

Specific gravity testing has been carried out, in house, by Apollo and described by Prenn (2006) as follows: "*A total of 1,218 density tests have been completed by Apollo from core intervals. The average density of mineralized material is 2.78", while the average density of unmineralized material is 2.85.*" Apollo was skeptical of these results since they were generated in house by a student geologist. To check the results, Apollo sent 107 samples to an outside laboratory for independent analysis. These results reported an average density of 2.8g/cm. This value was used for all material in this resource estimate.

15.1.5 Variogram Analysis and Modeling

Variogram Analysis

Variogram analysis was conducted on the 1.5m drillhole data to determine appropriate projection ranges and to test for any preferred orientation of the mineralization. The composites were first flagged to differentiate them into data sets to be used for an indicator estimation technique. All composites greater than 0.5g/t-Au were flagged as "ore group" and those below this cut-off were flagged as "waste group". Variograms were then constructed using Vulcan software along all directions within the plane of the mineralization and perpendicular to it. Only the ore group composites were considered for the variography. The greatest range was oriented within the plane of the mineralization, striking at azimuth 73º dipping -50º S. The shortest range is oriented perpendicular to the plane of mineralization at azimuth 343º dipping -40º. The variograms for each of these orientations are presented in Appendix C. Ranges, nugget values and total sill values are presented below in Table 15.1.5.1.

Table 15.1.5.1: Variogram Results for 1.5m Composite Data

Orientation	Range	Nugget	C_1 Sill Differential
All directions within plane 73°,-50°	30m	110	55
343°, -40°	20m	45	80
Model Variogram		110	55

Modeling

The Black Fox deposit was modeled only for gold content. The model has a parent block size of 3m x 3m x 3m and is sub-blocked to 1m x 1m x 1m along the geologic boundaries of the flow ore and along the boundaries of the underground workings. This small block size was chosen mainly to try to emulate localized pod like bodies of mineralization. All block estimates were made using only the drillhole 1.5m composites. The model boundaries based on local mine grid coordinates are presented in table 15.1.5.2 below.

Geologic boundaries were used to confine the modeling of the flow ore from the more pervasive green carbonate ore. For this purpose, 3-D solid shapes were constructed from drillhole logs creating nine zones of flow ore and one zone of intense quartz veining. These shapes were treated as hard boundaries such that the block within them were estimated only from composites also located within them. The blocks located within the green carbonate ore were also estimated only by composites from the same rock type. Two different estimation techniques were used depending on the rock type.

The grade estimation of the green carbonate ore was conducted using an indicator approach due to the strong spatial grade variability seen in the drilling data. This type of estimation first separates the composite data into higher grade and lower grade groups based on an appropriate cut-off value. These two data groups are used to flag blocks in the model based on their proximity to the higher or lower grade composites. Once this is achieved all the higher grade blocks are estimated using only the higher grade composites and lower grade blocks are considered as waste. This technique precludes the necessity of creating very complex grade shells to control grade assignment.

For this model, an indicator cut-off of 0.5ppm-Au was chosen to represent the data sets above and below a value of approximately one half the in pit mining cut-off. The first step is to flag all of the composites below 0.5ppm-Au with a value of zero and those above with a value of one. This procedure effectively subdivides the composites into ore and waste data sets. Once this is done, the zero or one, indicator values are estimated throughout the block model using appropriate projection distances. For this estimation blocks which have been estimated with an indicator value greater than 0.35 were flagged as ore and those below 0.35 were flagged as waste. Therefore, all blocks within the green carbonate ore with a probability of 35% or greater were considered as candidates for grade assignment.

The indicator model was allowed to search 30m along all directions within the plane of mineralization but only 6m perpendicular to it. The 6m limit prevented unwanted estimation of excessive blocks in areas of thin drilling and at the ends of drillholes. This limit was based mainly on trial of several distances and visual inspection of the results.

The grade estimation of the flow ore did not use an indicator approach due to the fact that all material located within these hard boundary solids is considered ore, based on geologic observations. Within the flow ore solids, all blocks were candidates for estimation and were assigned grade based on the search limits and estimation parameters described below.

The estimation parameters for both green carbonate and flow were as follows. Each block required a minimum of 1 and maximum of 5 composites to be assigned grade. The low number of composites per block was purposely chosen to restrict the grade assignment to consider only the drillholes in close proximity to the blocks. An octant search was used with a maximum of 2 composites per octant and no restriction on the number of octants. These techniques were chosen to try to emulate a pod like nature to the mineralization and prevent grade dilution from excessive composites. For the grade estimation, a search range of 30m along all directions within the plane of mineralization and 20m across the plane of mineralization was used. These ranges are based on the variography of the Au composites. The average distance to all composites and the number of drillholes used to estimate grade in each block was stored for later use in resource classification.

Table 15.1.5.2: Black Fox Model Limits

Direction	Minimum	Maximum
Easting	9,700	10,651
Northing	9,500	10,352
Elevation	9,400	10,000

Model Verification

The Black Fox model was verified using two procedures. The first was to run the model using three different estimation techniques including; Inverse Distance Weighting Squared (ID2), Inverse Distance Weighting to the third power (ID3) and Kriging. The second, was to compare the predicted ore tonnes and grade to the historically extracted tonnes and grade within the underground mine.

Once the deposit was estimated using the three estimations described above, the resulting block grades were compared to the composite assay values to review the histogram distribution and basic statistics of each shown previously in Table 15.1.3.1. The various models display minor statistical variations but they all produce average grades and variances below the composite data used to produce them. A typical block model cross section of Au derived from the Inverse Distance Weighted to the 3rd power is shown in Figure 15-2. For comparison, the Au assay data are included on this same section. The histogram results of the block model data from the different modeling verifications are presented in Appendix D.

The Black Fox deposit historically produced 1.1Mt of ore with an average grade of 6.03g/t-Au. Apollo has good survey records of all the underground development and has created 3-D solid shapes of these workings. To further verify the SRK model, the block model was "mined" within the 3-D solid of the historical workings to determine the percentile of each block included within it. All blocks which are $\geq$ 50% within the mined solid, were tabulated for total tonnes and average grade. Table 15.1.5.3 presents the results of this analysis for each of the model algorithms.

Table 15.1.5.3: Model Validation by Comparison with Historical Production

Model	Au Cut-off	Tonnes (M)	Grade g/t-Au	oz (M)	Estimated vs. Mined
Historical Mine	?	1.1	5.97	0.21	
SRK Id3	1g/t-Au	0.489	10.45	0.16	76%
	3g/t-Au	0.316	15.16	0.15	71%
SRK Id2	1g/t-Au	0.484	10.39	0.16	76%
	3g/t-Au	0.318	14.82	0.15	71%
SRK Kriged	1g/t-Au	0.517	9.85	0.16	76%
	3g/t-Au	0.338	14.04	0.15	71%

15.1.6 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.

In this study, the blocks were assigned to Indicated or Inferred based on the average distance to the composites and the number of drillholes used to estimate grade. The blocks estimated by at least two drillholes from at least two composites, where the distance to the closest composites was within 15m or less were classified as indicated. The confidence of the projection distance used to determine indicated resource was determined from the drillhole variograms, reflecting a distance of approximately ½ the average range and the necessity of more than one drillhole and composite to predict the Au grade.

The blocks located greater than 15m from the nearest composite were classified as inferred. Thus the inferred category does contain blocks (approximately 10%) estimated by a single drillhole, by a single composite; some of which are located up to 30m away. The material estimated by a single drillhole and a single composite was isolated and compared statistically to that estimated by multiple drillholes. This showed a slightly lower grade for the blocks with the lower confidence and was therefore considered defensible for assignment to the inferred category.

15.1.7 Mineral Resource Statement

Based on visual comparison of block grade distribution relative to drillhole composites and histogram comparison between the same, the Inverse Distance Weighting to the 3rd power using a minimum of one and maximum of five composites was chosen as the most appropriate estimation method for Black Fox. The tonnage and grade for different mining methods at appropriate Au cut-offs for indicated and inferred resources are shown in tables 15.1.7.1 and 15.1.7.2.

Table 15.1.7.1: Black Fox Indicated Resource Statement as of June 30, 2007*

Mining Method**	Category	Cut-off g/t-Au	Tonnes	Grade g/t-Au	Contained oz-Au
Open Pit	Indicated	1	4,358,500	5.5	763,700
Underground	Indicated	3	1,574,500	11.3	570,000

* Indicated Resources include Probable Reserves listed separately below.
** Mining Method is determined by relative location above or below the 9815m elevation

Table 15.1.7.2: Black Fox Inferred Resource Statement as of June 30, 2007

Mining Method*	Category	Cut-off g/t-Au	Tonnes	Grade g/t-Au	Contained oz-Au
Open Pit	Inferred	1	3,255,500	4.7	490,900
Underground	Inferred	3	929,000	12.3	368,000

* Mining Method is determined by relative location above or below the 9815m elevation

15.1.8 Mineral Resource Sensitivity

The grade tonnage distribution for indicated and inferred resources at Black Fox are shown in Figures 15-3 through 15-6. The resources have been subdivided based on relative location above the mine elevation of 9815. This level represent the lowest bench potentially accessible by open pit mining, all resources below this level would potentially be extracted by underground methods.

15.2 Reserve Estimation

The orientation, proximity to the surface, and geological controls of the Black Fox ore body will require mining of the ore reserves with open pit and underground mining techniques. Hence, the ore reserves are subdivided into open pit and underground categories. The mineable reserve was calculated based on a gold price of US$525/oz of gold which is approximately the three year average. A common gold recovery of 96% was used for all reserves. The LoM cash cost per ounce of gold was calculated at $236/oz.

15.2.1 Conversion of Mineral Resources to Mineral Reserves

The open pit ore reserves are based on the previous open pit design created by MDA. In order to calculate the open pit reserves, the original MDA pit shell was used as a limiting surface within which to tabulate the indicated resource. For the open pit reserve, no mining dilution was incorporated.

The underground ore reserves are all located below the 9815 level. These include material within 50 individual stopes designed by outlining the indicated resource blocks at a 2g/t-Au cutoff. This cut-off was used to define the boundaries provided an average grade of 4g/t-Au or higher material was maintained within the stope. Other design criteria included a minimum width of 3m, with the average width 8m and the average length 34m. All stopes are designed for a conventional overhand cut and fill mining technique. The underground reserves include dilution of approximately 22% of which 66kt (6%) of indicated material with an average grade of 1.3g/t-Au was used and the remaining dilution material was assigned a grade of 0g/t-Au.

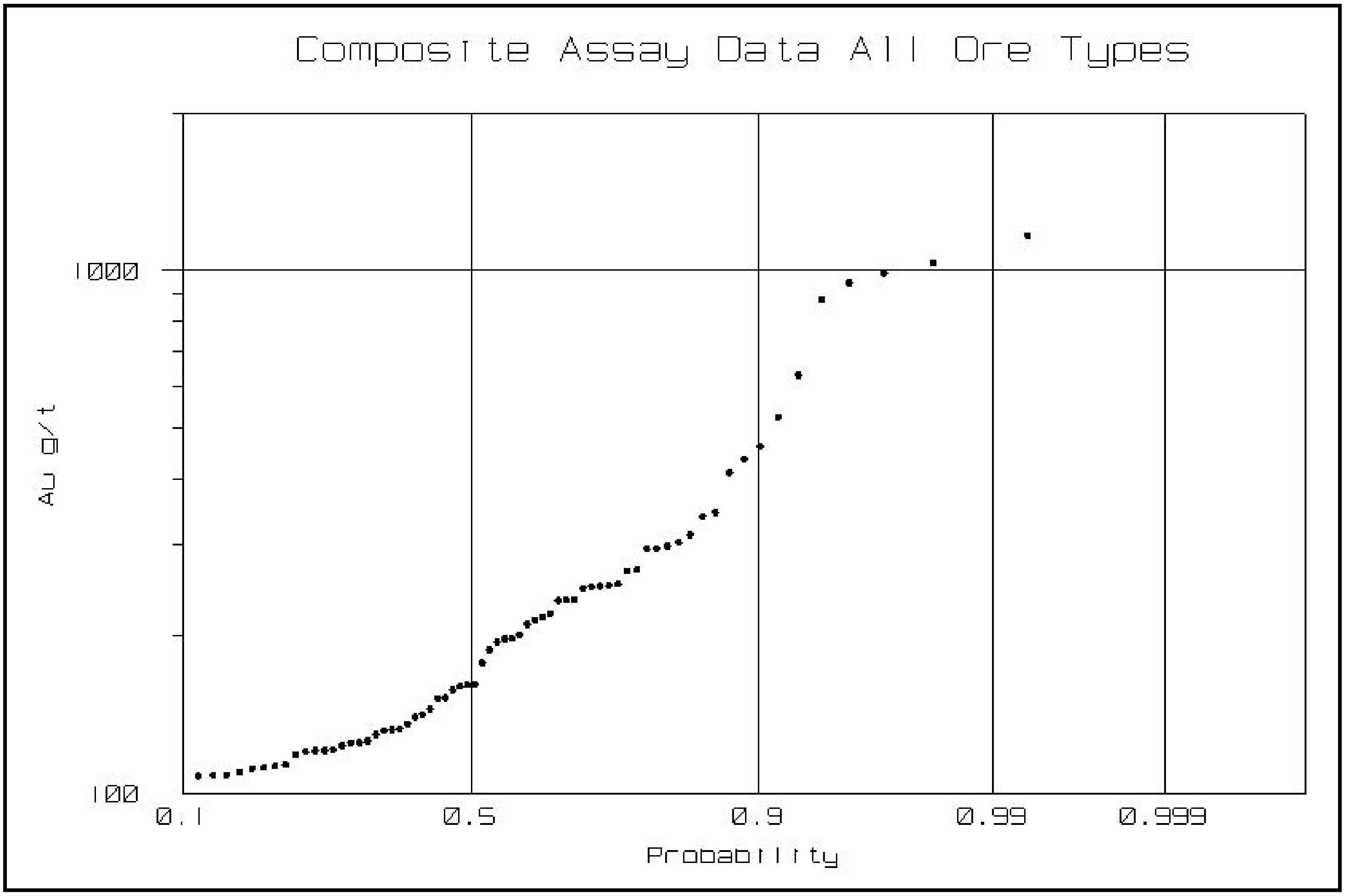
The Black Fox project has established the mine surface datum level as 10,000m, this value does not reflect mean surface elevation (amsl), and is used as a point of reference only.

The Probable Reserves at Black Fox are presented in Table 15.2.1.1.

Table 15.2.1.1: Black Fox Probable Reserve Statement as of June 30, 2007*

Mining Method	Cut-off g/t-Au	Tonnes (000s)	Grade g/t-Au	Contained oz-Au
Open Pit	1	3,362	5.8	625,000
Underground**	3	1,108	10.6	377,000

* Probable Reserves are included within Indicated Resources listed above.
** Underground Reserves include dilution of 66,000t of indicated material with an average grade of 1.3g/t-Au.



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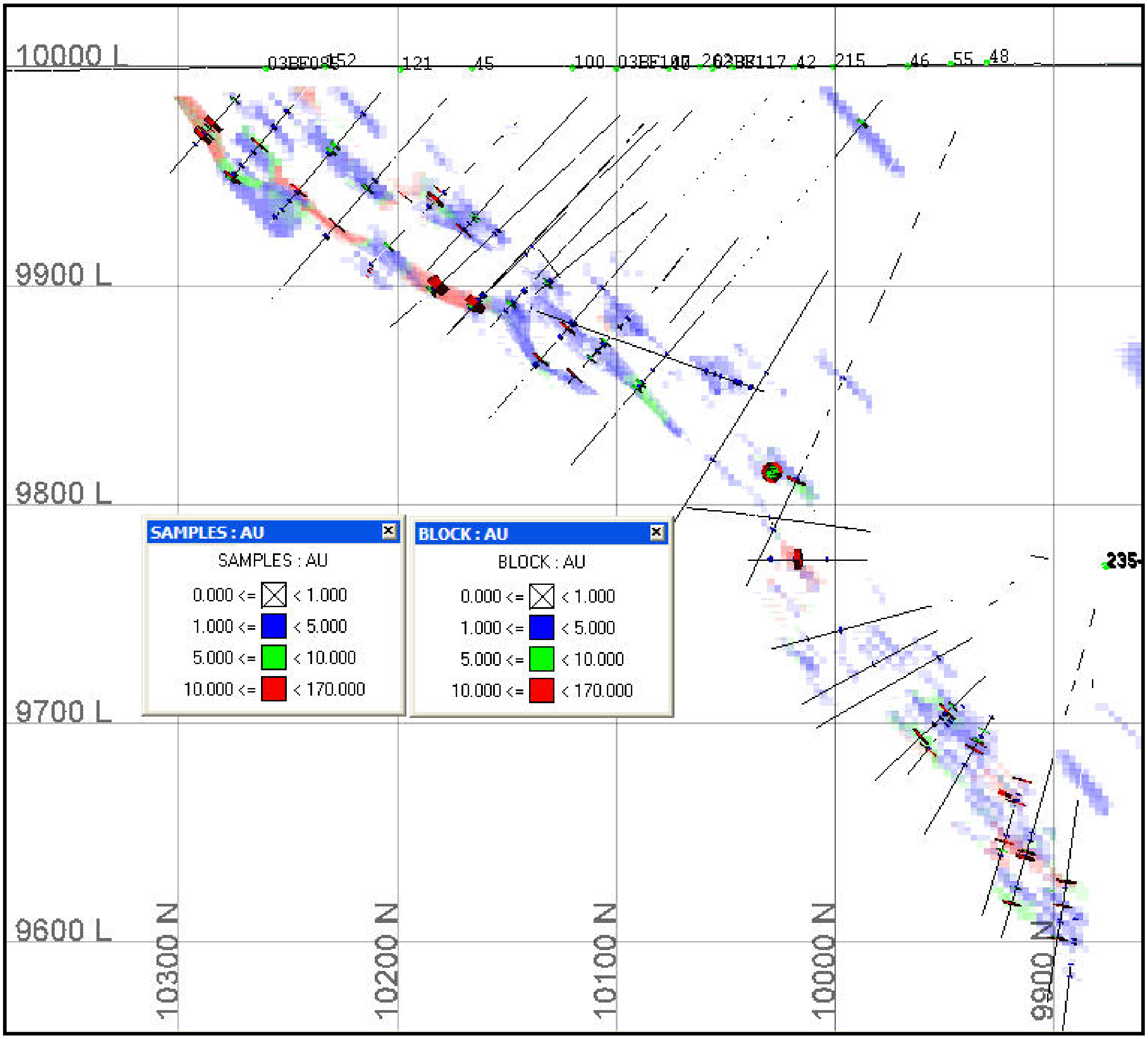
SRK Job No.: 144418

File Name: Figure 15-1.doc

Black Fox,
Timmins, Ontario, Canada

Cumulative Frequency Plot of 1.5m Composites above 100g/t-Au in all Ore Types

Date: 07-26-07 | Approved: BAS | Figure: 15-1



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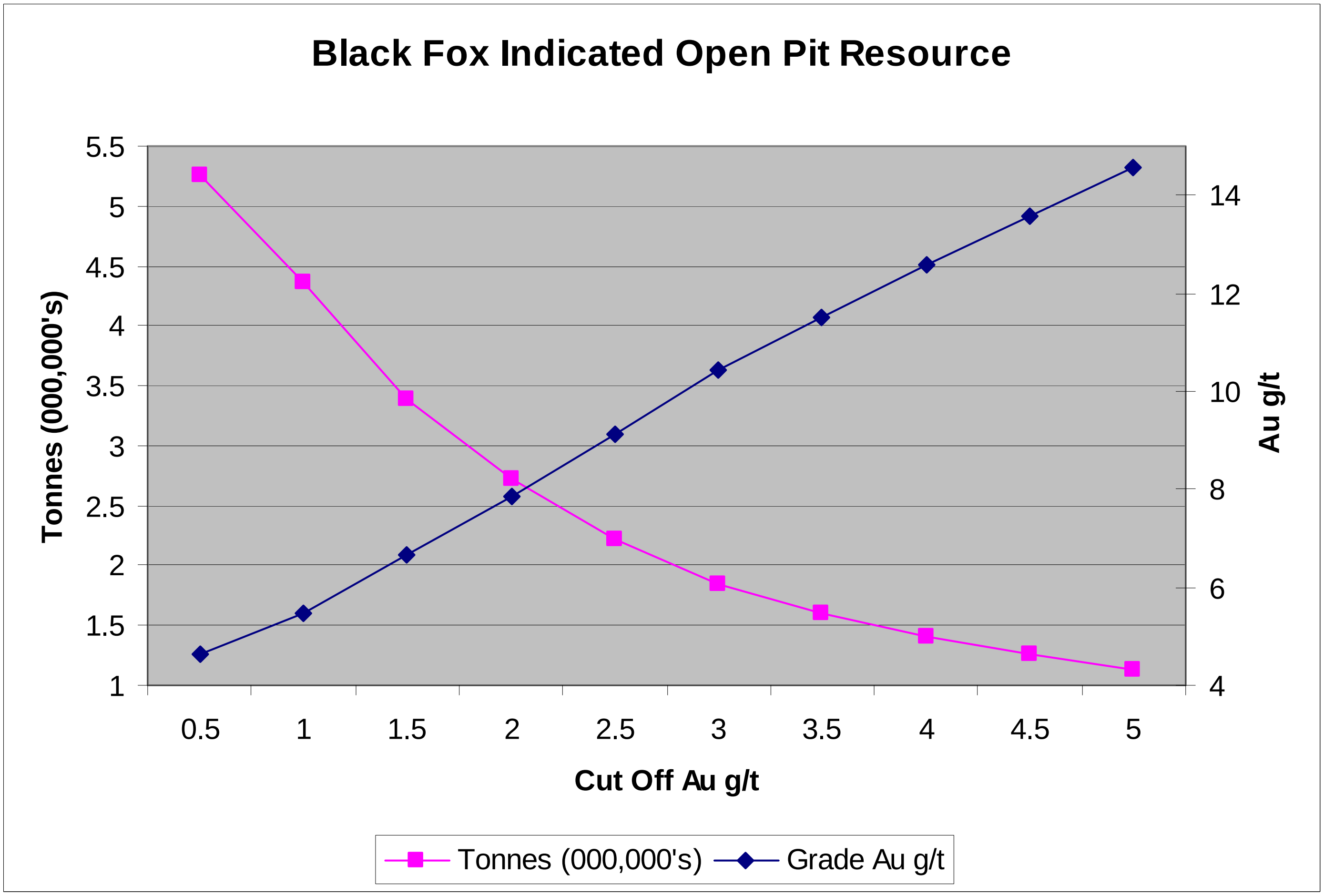
File Name: Figure 15-2.doc

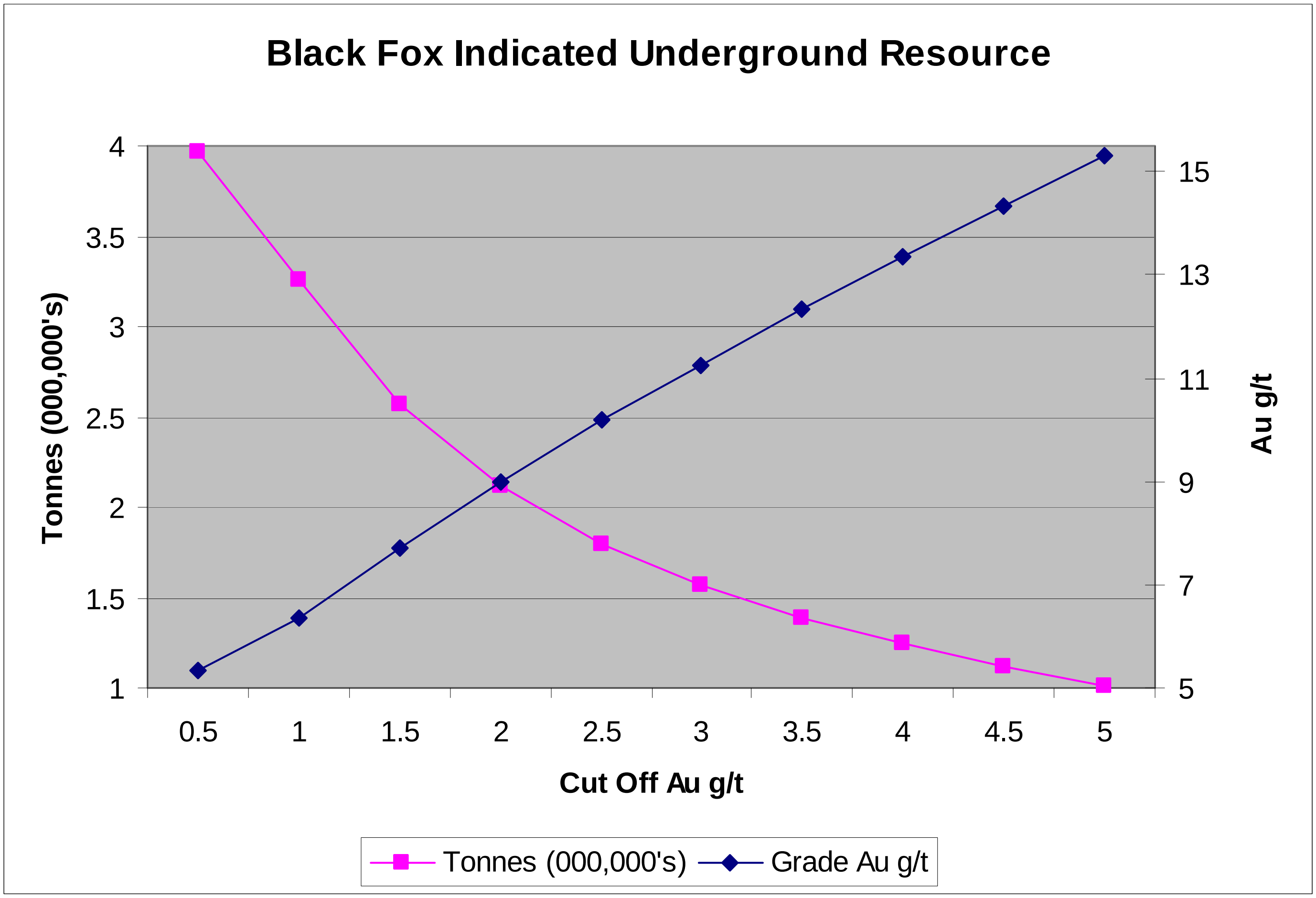
**Black Fox,
Timmins, Ontario, Canada**

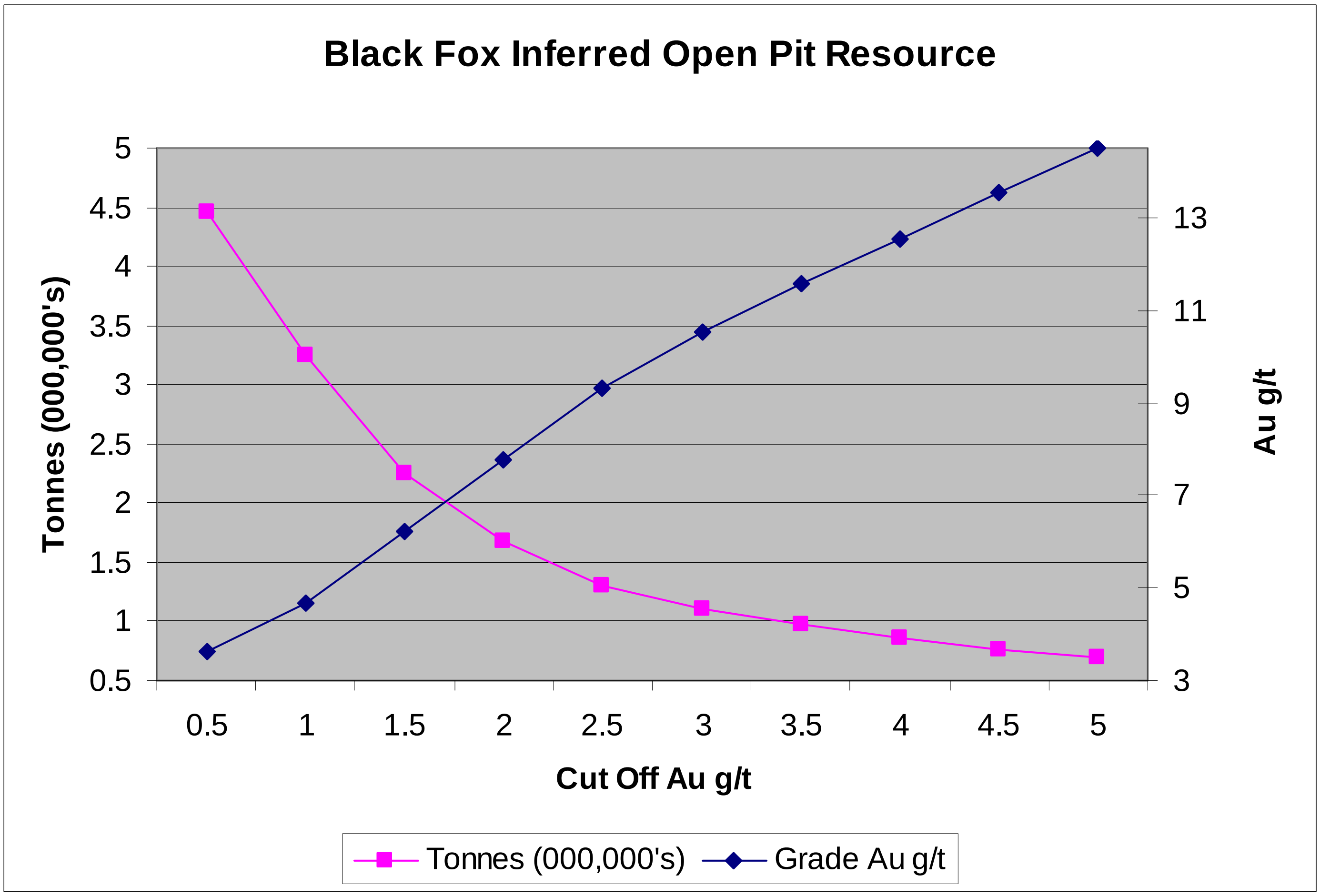
Black Fox Typical Block Model Cross-section 10300E Showing Distribution of Au

Date: 07-26-07 | Approved: BAS | Figure: 15-2

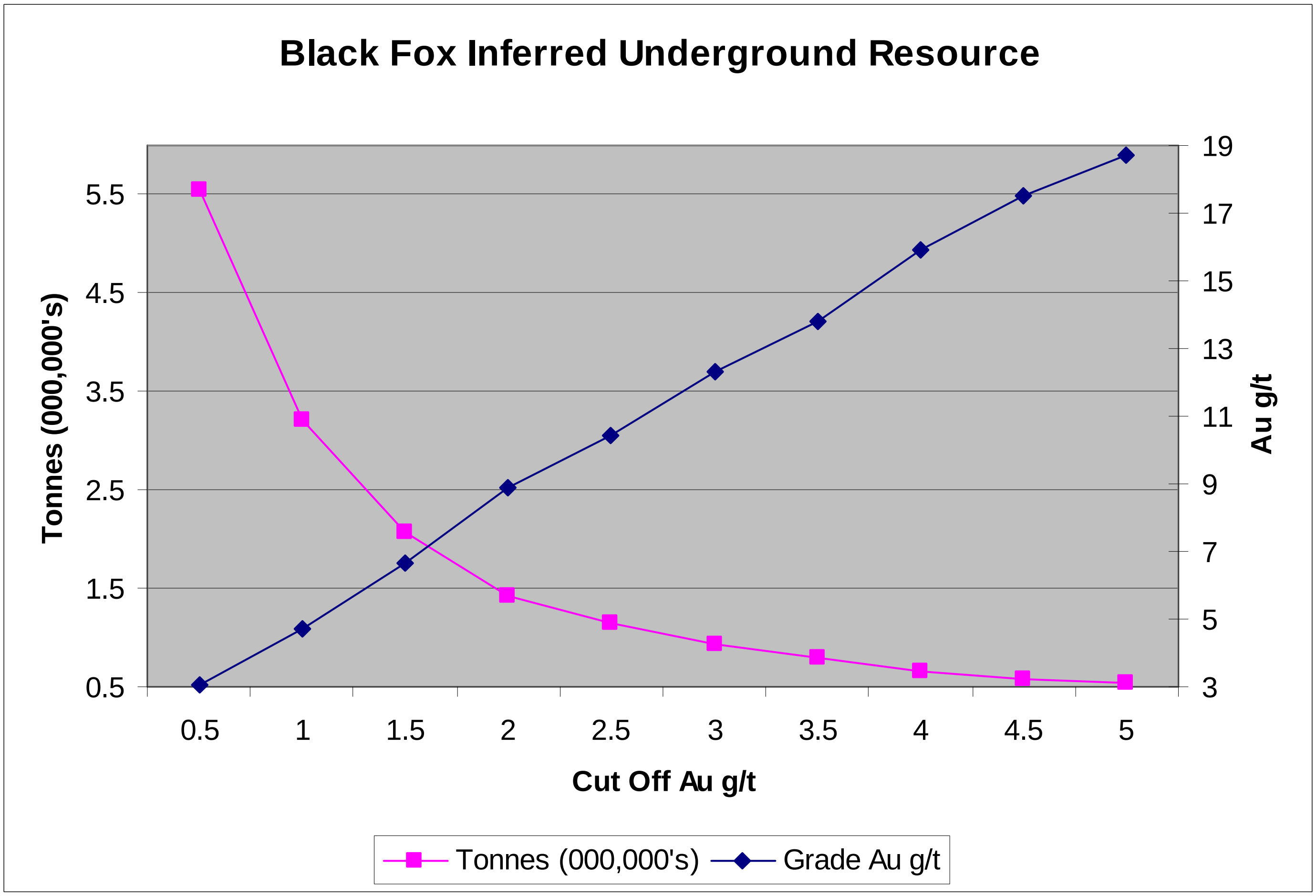


SRK Consulting Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Grade Tonnage Curves for Indicated Open Pit Resources at Black Fox**
SRK Job No.: 144418		
File Name: Figure 15-3.doc		Date: 07-26-07	Approved: BAS	Figure: 15-3


Black Fox Indicated Underground Resource
Tonnes (000,000's)
Au g/t
Cut Off Au g/t
4
3.5
3
2.5
2
1.5
1
15
13
11
9
7
5
0.5
1
1.5
2
2.5
3
3.5
4
4.5
5
Tonnes (000,000's)
Grade Au g/t



SRK Consulting, Engineers and Scientists	**Black Fox, Timmins, Ontario, Canada**	**Grade Tonnage Curves for Inferred Open Pit Resources at Black Fox**		
SRK Job No.: 144418				
File Name: Figure 15-5.doc		Date: 07-26-07	Approved: BAS	Figure: 15-5



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Grade Tonnage Curves for Inferred Underground Resources at Black Fox		
SRK Job No.: 144418				
File Name: Figure 15-6.doc		Date: 07-26-07	Approved: BAS	Figure: 15-6

16 Other Relevant Data and Information

There is no known current information or data that could provide additional useful insight to Black Fox. All known relevant data and information is presented within this PFS and in the appropriate sections.

17 Additional Requirements for Development Properties and Production

Based on a review of the historic underground mining, drill logs, and the geological model, the determination was made that significant amounts of gold ore resources still remain at Black Fox. A scoping-level and subsequent prefeasibility-level design was completed to mine these remaining gold resources between the surface and the 9815m level of the mine using open pit mining methods. This work which included an estimate of reserves was completed by MDA in August 2006.

Black Fox consists of a moderate dipping vein (from 45º to 55º) with a true vein thickness that varies from 3m to 15m with an overall strike length of approximately 900m based on current drilling and modeling. The orientation, proximity to the surface, and geological controls of the Black Fox ore body will require mining of the ore reserves with open pit and underground mining techniques.

This PFS is based on mining of the Black Fox deposit at a processing rate of 1,500t/day, or 540,000t/yr. The size of the Black Fox resource, combined with the application of the most efficient mining operation possible and the maintaining of reasonable vertical advance rates made the 1,500t/day rate appropriate. All of the technology, methods and equipment chosen for the mine are industry-standard and well proven.

Black Fox has established the mine surface datum line as 10,000m. This value does not reflect mean surface elevation above sea level, and is used as a point of reference only.

17.1 Open Pit Mining

The geological model as developed by SRK was used with the optimized open pit to develop the new production schedule and subsequent operating plan. Updated capital and operating costs were also developed by SRK. Open Pit Mining can be divided into three distinct areas:

- Glacial Till;
- Waste mining away from mineralization; and
- Ore and waste mining near and within mineralization.

17.1.1 Glacial Till

The alluvial (glacial till) mining will be completed with a truck-shovel combination. Typically, a $16.5m^3$ front shovel will load trucks with alluvium from 6m pit benches. Drill and blasting will not be required to excavate this material. The glacial till waste pile will be located to the west of the planned open pit and areas for topsoil, clay and sand alluvial materials will be accommodated.

17.1.2 Open Pit Waste Mining Away from Mineralization

In areas known to contain only waste, the blast hole spacing and bench interval will be different than in areas containing mineralization. About 25% of the waste rock mined will be from areas containing only waste.

Blastholes, 165mm in diameter, will be drilled on a 5m x 5m staggered pattern for a 6m bench. A powder factor of 0.25kg/t will be used for blasting with 0.75m of sub-grade drilling. ANFO will be used for about 80% of the blasting along with an assumed 20% wet holes requiring emulsion. Buffer blasting and pre-splitting along final pit walls will be used to allow steeper pit slopes. Blasted rock will be mined by a 16.5m^3 front shovel. The waste rock disposal site is located south of the mine.

17.1.3 Open Pit Mining in the Mineralized Areas

The mineralized zones average 4.75m wide, but can be as narrow as 1.0m. The minimization of dilution of the ore will be a critical element of the mining operation due to the characteristics of the orebody. For this reason, it is expected that the ore will be mined in 3m lifts or benches. It is expected that the drill cuttings from all blastholes will be sampled and assayed in order to provide the basis for ore grade control. The sampling requirements may dictate the spacing of the blastholes and this in turn would impact the blasthole diameter. If this is not the case, then the blasthole diameter would be in the range of 102mm to 152mm which is the maximum practical for 3m high benches.

Blasting in the ore would likely be on 3m high benches so that identification of ore blocks can be carried out with the most accuracy and the material mined with the minimum of dilution. Choke blasting of the ore would be used in order to minimize movement and subsequent dilution of the ore blocks. Careful control of the stemming length in the blasts will be required in order to control flyrock and loss of ore.

Mining in the mineralized area will generally be accomplished using a 10.5m^3 mass excavator with a bucket designed for narrow mining. The bucket should be designed to be about 2.5m wide versus the normal bucket width of 2.85m. For narrow zones, a 4m^3 mass excavator equipped with narrow (1.2m wide) and normal (2.0m wide) buckets will be used. Ore and waste material will be trucked using 100 and/or 150t trucks.

The waste mined from inside the ankerite alteration zone may have concentrations of arsenic and nickel that according to preliminary waste characterization completed by AMEC may require special handling. This material is expected to total about 12% of the waste from the pit. This material will need to be segregated and may require additional handling or water treatment during closure. More work is being performed by AMEC to adequately determine closure impacts.

17.2 Geotechnical Bedrock Pit Slope Studies

Golder Associates was contacted to complete a preliminary assessment of open pit slopes. The following paragraphs were extracted from sections of their report titled Preliminary Assessment of the Pit Slope Parameters for the Apollo Gold Black Fox Project dated February 2004.

"*The rockmass properties have been characterized by a number of different individuals at different stages of the development of the original mine. Most recently, work has been completed on the structural geology of the A1 Zone by Barclay (2000). Barclay identifies 3 major fault/slip planes in this zone with relatively complex slips along shallow to steeply dipping slickensided planes. An analysis of the near-surface (<50m) rockmass, including UCS, point load and other laboratory test results, as well as joint mapping and description has been completed by Newmann as well as Marisett. Using the Barton Rockmass Classification System, Newmann estimates that the rockmass quality (Q) ranges from 0.5 to 2.7, which classifies it as Very Poor to Poor Rock for the "U" zone. Neglecting stress and groundwater effects (Q'), the modified rockmass classification changes to a range between 2.67 and 10, thereby changing the classification to Poor to Fair Rock.*

Newmann also observed an improvement in rockmass condition at depth with an increase in clamping stress as well as a decrease in weathering effects and water inflow. Marisett used the Barton System to classify the rockmass according to regions as follows:

- *Area 1 (Northwest Region) Q' = 2 to 10 (Poor to Fair Rock);*
- *Area 2 (Northeast Region) Q' = 5 to 30 (Fair to Good Rock);*
- *Area 3 (Southeast Region) Q' = 2.5 to 7 (Poor to Fair Rock); and*
- *Area 4 (Southwest Region) Q' = 2.5 to 7 (Poor to Fair Rock).*

In addition, Mariset identified four fault groups with apertures ranging from 1 to 100cm and an infilling ranging of one or a combination of gouge, chlorite, sericite, quartz and or talc. Most of the work in rockmass characterization thus far has encompassed that in ore, this information will prove useful when considering the design of the pit footwall. However, the rockmass quality of the hanging wall and sidewalls will have to be characterized as well. A preliminary consideration of this regional rockmass using core drilling results in Sections 1000E, 10125E, 10225E, 10275E and 10350E indicates that there are three main rock units to consider:

- *An ultramafic volcanic with chlorite and talc chlorite sections underlying the overburden. This unit has RQD values ranging from as low as 15 up to 95, but generally averaging at around 70. Lower RQD values generally occur closer to the surface. Approximately 30 to 70% of the south wall (hanging wall) would comprise this unit;*
- *An underlying metabasalt with RQD values ranging from 82 to 100, with values averaging out at around 97. This unit should comprise the remainder 30 to 70% of the south wall not composed of the ultramafic volcanic; and*
- *Fault/breccia shear zone with RQD values of 20 or less. These zones can be found within both of the above rock units and range in thickness (along core) from 1 to 23m.*

The ultramafic volcanic unit appears to be thicker in the western portion of the deposit thinning out to the east along the same northing (9800N to 9900N), conversely the opposite can be said of the metabasalt unit.

The north wall will be in the footwall of the orebody. The rock properties described above indicate Poor to Fair Rock. In general, the south, east and west walls of the open pit will be in relatively hard and massive volcanic rock. Shear zones will intersect the pit walls and their number and orientation are not known. Also, the presence, orientation and other physical properties of joint sets and faults are not known. For these reasons the rock slope parameters provided below should only be used for coarse pit shell assessment and not detailed planning or feasibility study pit analysis.

The rock slope parameters are based on the following assumptions:

- *Pit wall component terminology;*
- *Calculation of the inter-ramp angle;*
- *All rock types are sufficiently strong that bench face angles will be controlled by geological structure and blasting practices. Inter-ramp and overall slope stability will be a function of bench configuration;*
- *The ultimate mining depth will be 150m;*

- *The bench height will be 3m in ore for grade control reasons and 6m or 9m in waste depending on equipment sizes;*
- *Careful wall control blasting techniques will be used adjacent to all final bench faces in rock;*
- *Artificial rock support (i.e., cable bolts, rock bolts, etc.) may be required to support potential bench scale slope instabilities;*
- *Catch berm widths will generally be a minimum of 8m;*
- *The slopes will drain naturally, either to the face or to the underground workings below, or can be depressurized by artificial means (horizontal drain holes, etc.).*

The initial rock slope parameters are:

- *North Wall - Inter-ramp angle 45° or less. Defined by the dip of the orebody;*
- *Maximum 18m vertically between catch berms;*
  - *Minimum 8m catch berm width, and*
  - *Bench face angle 60° or less.*
- *South Wall - 18m vertically between catch berms;*
  - *Minimum 8m catch berm width,*
  - *75° face angle, and*
  - *54.5° inter-ramp angle.*
- *East & West - 18m vertically between catch berms;*
  - *Walls - 9m catch berm width,*
  - *70° face angle, and*
  - *49.2° inter-ramp angle.*

An 8m catch berm width is the minimum width that will provide adequate catchment for any falling material and also allow access to clean the berm. Access to each of the catch berms for cleaning purposes must be maintained.

There is little oriented structural data for the hanging wall rock; therefore, the potential for structurally controlled slope failures on the south, east and west walls cannot be evaluated. A geotechnical drilling program is required to determine the presence and orientation of structural features for the hanging wall rock types. The rock slope parameters provided above are typical of similar deposits in the region but must be confirmed through additional investigation and analyses. In addition, an ongoing program of pit slope inspections, geotechnical face mapping and structural analysis carried out by qualified people would be required during the start-up and operation of the open pit mine.

An initial scope for a preliminary geotechnical drilling program to assess rock slope stability would involve angled diamond drillholes through the proposed south wall of the pit. Three of the holes would have a dip direction approximately to the south and perpendicular to the strike of the orebody so they intercept the pit wall at an acute angle. The fourth hole would be drilled with a dip direction towards the north such that the hole roughly parallels the final wall. All holes would be drilled at approximately 65° dip. The core would be oriented using a core *orientation technique, then it would be structurally logged by an experienced geotechnical specialist.*

The structural data obtained from the drill core would be combined with the structural data from underground, then analyzed using stereonet techniques. A stereo net would be developed for each of the main rock types and from these the orientation of discontinuities (joint sets, shear zones and faults) would be determined. This information would be used in a kinematic analysis of the stability of the pit slopes."

17.2.1 Pit Optimization

Open pit reserves were developed by updating a pre-feasibility study of the open pit mine completed by MDA in August 2006. The open pit design completed by MDA was used for reporting the open pit reserves contained in SRK's resource model. SRK is currently re-optimizing the open pit for its ongoing feasibility study, and will perform trade-off studies to determine the optimal mining method for some of the deeper material contained in the current open pit configuration.

The Lerchs-Grossman pit optimization completed by MDA was based on the parameters listed below:

- Overburden mining cost - \$1.25/t of material;
- Rock mining cost - \$1.50/t of material;
- Processing cost - \$12.16/t ore;
- General and Administrative cost - \$4.50/t ore;
- Plant gold recovery – 96%; and
- Pit Slopes – Measured from the 2004 design shown in Table 17.1.2.1 for rock, and 19º overburden.

Table 17.2.1.1: Pit Slopes for Optimization

Pit Segment	N	NE	E	SE	S	SW	W	NW
Inter Slope Angle	45º	42º	39º	43.5º	47º	47.5º	48º	46.5º

Due to previous underground mining by Exall some of the mined areas have been backfilled or will require backfilling before open pit mining. The analysis estimates that 50% of the workings have been backfilled with material having a density of 2.0t/m^3. Since the location of the backfilling is unknown, all of the workings were given a density of 1.0t/m^3. SRK used the same densities for the workings in the updated mine plans.

17.2.2 Pit Design

The MDA-optimized US\$600/oz gold pit was used as a basis for pit design with CoG based on a US\$525/oz gold price. The pit was designed with a 20m wide, 10% maximum grade ramp system narrowing to a 10m wide single lane ramp near the pit bottom, 3:1 pit slopes in alluvium.

The pit was designed in two phases. The initial phase starts on the eastern portion of the deposit where near-surface high-grade material has been core drilled on 12.5m centers. The phase one pit can be designed so that some backfilling can be accommodated during Phase 2 mining. Table 17.2.2.1 summarizes the material and probable reserves contained in the Black Fox open pit design. The open pit design completed by MDA was used for reporting the open pit reserves contained in SRK's resource model for this study.

Table 17.2.2.1: Open Pit Probable Reserves as of June 30, 2007

	Probable Reserves (1.0g/t-Au CoG)			Waste Material				
Pit	**Ore (000s)**	**Grade (g/t)**	**Contained (koz-Au)**	**Till Waste (kt)**	**Void-Fill (m^3)**	**Rock Waste (kt)**	**Total Waste (kt)**	**Strip Ratio**
Phase 1	1,536	5.2	259	5,396	95.	18,615	24,107	15.7
Phase 2	1,826	6.2	366	3,414	285	19,134	22,833	12.5
Total	**3,362**	**5.8**	**625**	**8,810**	**381**	**37,749**	**46,940**	**14.0**

Figure 17-1 shows the Phase 1 pit, while Figure 17-2 illustrates the final pit.

17.2.3 Pit Production Schedule

A production schedule was developed for the property from mining the Phase 1 and Phase 2 pits at the rate of 1,500t/day and then transitioning into the underground mining as ore from the Phase 2 pit is depleted.

The PFS assumes that mining will commence as soon as possible, and during the Black Fox pre-production period (when the planned process mill design, procurement and construction will be underway) ore mined from the open pit be toll milled.

Stripping of the Phase 2 pit will begin in Year 02. The production schedule by the pit phase is shown in Table 17.2.3.1.

Table 17.2.3.1: Open Pit Production Schedule

Pit Phase	Material	Year -01	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06
Phase 1	Ore (kt)	244	528	608	156			
	Grade (g/t-Au)	7.4	5.0	4.4	5.9			
	Contained Au (koz)	58	85	85	30			
	Till Waste (kt)	5,342	54					
	Rock Waste (kt)	3,339	8,802	5,781	789			
	Total (kt)	**8,925**	**9,384**	**6,388**	**945**			
Phase 2	Ore (kt)			16	334	569	523	384
	Grade (g/t-Au)			26.1	8.0	5.0	6.7	5.1
	Contained Au (koz)			14	86	91	113	63
	Till Waste (kt)			1,628	1,786			
	Rock Waste (kt)			4	5,113	7,859	3,792	2,652
	Total (kt)			**1,648**	**7,233**	**8,427**	**4,315**	**3,036**
Total	**Ore (kt)**	**244**	**528**	**624**	**490**	**569**	**523**	**384**
	Grade (g/t-Au)	**7.4**	**5.0**	**4.9**	**7.3**	**5.0**	**6.7**	**5.1**
	Contained Au (koz)	**58**	**85**	**99**	**115**	**91**	**113**	**63**
	Till Waste (kt)	**5,342**	**54**	**1,628**	**1,786**	**0**	**0**	**0**
	Rock Waste (kt)	**3,339**	**8,801**	**5,784**	**5,903**	**7,859**	**3,792**	**2,652**
	Total (kt)	**8,925**	**9,384**	**8,037**	**8,179**	**8,427**	**4,315**	**3,036**

17.2.4 Open Pit Productivity

Drilling & Blasting

The design assumptions used to determine drilling and blasting productivity are shown in Table 17.2.4.1.

Table 17.2.4.1: Drilling and Blasting Assumptions

Drill Productivities Schedule Data	Units	Ore Bedrock	Ore Bedrock	Ore Bedrock	Waste Bedrock	Waste Bedrock	Wall Control Pattern Buffer	Wall Control Pattern Buffer	Wall Control Pattern Buffer
Tonnage Factor	dmt/m^3	2.780	2.780	2.780	2.850	2.850	2.850	2.850	2.850
Blast Pattern Details									
Bench Height	m	6.00	6.00	6.00	6.00	6.00	6.00	6.00	12.00
Sub Drill	m	1.30	1.30	1.30	1.30	1.30	1.30	1.30	0.00
Diameter of Hole	mm	102.00	102.00	165.00	102.00	165.00	165.00	165.00	102.00
Staggered Pattern Spacing	m	3.25	3.25	5.00	3.25	5.00	5.00	5.00	2.00
Staggered Pattern Burden	m	3.25	3.25	5.00	3.25	5.00	5.00	5.00	2.00
Drill Equivalent Square Pattern	m	3.25	3.25	5.00	3.25	5.00	5.00	5.00	2.00
Hole Depth	m	7.30	7.30	7.50	7.30	7.30	7.30	7.30	12.00
Height of Stemming or Unloaded Length	m	0.73	0.73	1.55	0.56	1.20	2.42	2.42	9.96
Material Quantity									
Volume Blasted/Hole	m^3	63	63	150	63	150	150	150	48
Tonnes Blasted/Hole	t	176	176	417	181	428	428	428	137
Typical Powder Factor with Ammonium Nitrate Explosive									
Density of Powder	g/cc	0.82	0.82	0.82	0.82	0.82	0.82	0.82	0.82
Loading Density	kg/m	6.70	6.70	17.53	6.70	17.53	17.53	17.53	6.70
Powder/hole	kg	44.05	44.05	104.25	45.15	106.88	85.50	85.50	13.68
Powder Factor	kg/t	0.250	0.250	0.250	0.250	0.250	0.200	0.200	0.100
Powder Factor	kg/bcm	0.695	0.695	0.695	0.713	0.713	0.570	0.570	0.285
Drill Productivities									
Penetration Rate	M/min								
	M/hr	40.00	40.00	40.00	40.00	40.00	40.00	40.00	30.00
Penetration Rate-used for this calculation	M/min	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.50
Cycle Time Estimate									
Drilling Time	min	10.95	10.95	11.25	10.95	10.95	10.95	10.95	24.00
Steel Handling Time	min	0.20	0.20	0.20	0.20	0.20	0.20	0.20	1.00
Set up Time	min	0.80	0.80	0.80	0.80	0.80	0.80	0.80	1.00
Add Steel	min								
Pull Rods	min								
Total	**min**	**11.95**	**11.95**	**12.25**	**11.95**	**11.95**	**11.95**	**11.95**	**26.00**

Table 17.2.4.2: Open Pit Drill and Blasting Assumptions (Continued)

Drill Productivities Schedule Data	Units	Ore Bedrock	Ore Bedrock	Ore Bedrock	Waste Bedrock	Waste Bedrock	Wall Control Pattern	
							Buffer	Preshear
Calendar Days	days/yr	365	365	365	365	365	365	365
Scheduled Shutdown	days/yr							
Unscheduled Days Down	days/yr	22	22	22	22	22	22	22
Mine Work Days	days/yr	343	343	343	343	343	343	343
Work Days/Week	days/yr	7	7	7	7	7	7	7
Shifts/Day	Shifts/Day	3	3	3	3	3	3	3
Shifts/Week	Shifts/Wk	21	21	21	21	21	21	21
Scheduled Weeks/Year	Wk/Yr	49	49	49	49	49	49	49
Shifts/Year	Shifts/Yr	1,029	1,029	1,029	1,029	1,029	1,029	1,029
Scheduled Hours/Shift	Hrs/Shift	8	8	8	8	8	8	8
Scheduled Hours/Year	Hrs/Yr	8,232	8,232	8,232	8,232	8,232	8,232	8,232
(T) Total Theoretical	**Hrs/Yr**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**
(SU) Scheduled and Unscheduled Shutdown	Hrs/Yr	528	528	528	528	528	528	528
Standby								
Lunch Break	Hrs/Shift	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Shift Start/Shutdown	Hrs/Shift	0.40	0.40	0.40	0.40	0.40	0.40	0.40
Coffee Breaks	Hrs/Shift	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Miscellaneous-Blasting and Moves	Hrs/Shift	0.20	0.20	0.20	0.20	0.20	0.20	0.20
Total Standby	Hrs/Shift	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**	**1.3**
(S) Total Standby	**Hrs/Yr**	**1,338**	**1,338**	**1,338**	**1,338**	**1,338**	**1,338**	**1,338**
Available Working Hours	Hrs/Day	20.1	20.1	20.1	20.1	20.1	20.1	20.1
Available Working Hours	Hrs/Yr	6,894	6,894	6,894	6,894	6,894	6,894	6,894
Annual Hours								
(T) Total Theoretical	Hrs/Yr	8,760	8,760	8,760	8,760	8,760	8,760	8,760
(S) Total Standby	Hrs/Yr	1,338	1,338	1,338	1,338	1,338	1,338	1,338
(SU) Scheduled and Unscheduled Shutdown	Hrs/Yr	528	528	528	528	528	528	528
(W)+(R) Work + Repair = (T-S-SU)	Hrs/Yr	6,894	6,894	6,894	6,894	6,894	6,894	6,894
(W) Work = MA x (T-S-SU)	Hrs/Yr	5,515	5,515	5,515	5,515	5,515	5,515	5,515
Mechanical Availability Definition								
Scheduled Downtime	Shifts/Yr	54.45	54.45	54.45	54.45	54.45	54.45	54.45
Scheduled Downtime	Hrs/Yr	411.6	411.6	411.6	411.6	411.6	411.6	411.6
Scheduled Downtime	%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%	5.0%
Unscheduled Downtime	%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%	15.0%
Total Downtime	**%**	**20.0%**	**20.0%**	**20.0%**	**20.0%**	**20.0%**	**20.0%**	**20.0%**
Shifts Available for Scheduling	Shifts	978	978	978	978	978	978	978
(MA) Mechanical Availability	%	85.0%	85.0%	80.0%	85.0%	80.0%	80.0%	80.0%
Physical Availability								
(PA) Physical Availability = (W+S)T		82.2%	82.2%	78.2%	82.2%	78.2%	78.2%	78.2%
Use of Availability								
(UA) Use of Availability = W/(W+S)		81.4%	81.4%	80.5%	81.4%	80.5%	80.5%	80.5%
Effective Utilization								
(EU) Effective Utilization = PA x UA		66.9%	66.9%	63.0%	66.9%	63.0%	63.0%	63.0%
Annual Production								
Work Hours/Yr	Hrs/Yr	5,515	5,515	5,515	5,515	5,515	5,515	5,515
Operating Efficiency – operation based	%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%	85.0%
Production Hrs/Yr	Hrs/Yr	4,981	4,981	4,688	4,981	4,688	4,688	4,688
Scheduled Shifts/Yr (from below)	Shifts/Yr	978	978	978	978	978	978	978
Work Shifts/Yr	Shifts/Yr	733	733	689	733	689	689	689
Work Hrs/Shift	Hrs	6.7	6.7	6.7	6.7	6.7	6.7	6.7
Drilling Hrs/Shift	Drill Hrs/Shift	5.7	5.7	5.7	5.7	5.7	5.7	5.7
Drilling Min/Shift	Drill Min/Shift	341.7	341.7	341.7	341.7	341.7	341.7	341.7
Drilling Min/Hole	Drill Min/Hole	12.0	12.0	12.3	12.0	12.0	12.0	26.0
Holes Drilled/Shift	Holes/Shift	28.6	28.6	27.9	28.6	28.6	28.6	13.1
Meters Drilled/Shift	m/Shift	208.74	208.74	209.20	208.74	208.74	208.74	157.71
Tonnes Drilled/Shift	t/Shift	5,308	5,308	11,632	5,165	12,224	12,224	1,798
Tonnes Drilled/Yr	(t x 1000)	3,690	3,690	8,019	3,783	8,428	8,428	1,240
Production/Scheduled Work Hrs	t/hr	629.7	629.7	1,454.0	645.6	1,528.0	1,528.0	224.7
Production/Scheduled Production Hrs	t/hr	740.9	7\40.9	1,710.6	759.5	1,797.6	1,797.6	264.4

Open Pit Loading Productivity

The design assumptions used to determine loading productivity, including feeding the crusher, are shown in Table 17.2.4.3.

Table 17.2.4.3: Open Pit Loading Productivity

		16.5m³ Shovel		16.5m³ Excavator		10.5m³ Excavator		11.5m³ Loader	
Schedule Data	**Units**	**Waste 150t**	**Ore 150t**	**Overburden 150t**	**Overburden 150t**	**Overburden 100t**	**Rock 100t**	**Waste 100t**	**Ore 100t**
Calendar Days	days/yr	365	365	365	365	365	365	365	365
Scheduled Shutdown	days/yr								
Unscheduled Days Down-Weather	days/yr	15	15	15	15	15	15	15	15
Mine Work Days	days/yr	350	350	350	350	350	350	350	350
Work Days/Week	days/yr	7	7	7	7	7	7	7	7
Shifts/Day	Shifts/Day	3	3	3	3	3	3	3	3
Shifts/Week	Shifts/Wk	21	21	21	21	21	21	21	21
Scheduled Weeks/Year	Wk/Yr	50	50	50	50	50	50	50	50
Shifts/Year	Shifts/Yr	1050	1050	1050	1050	1050	1050	1050	1050
Scheduled Hours/Shift	Hrs/Shift	8	8	8	58	8	8	8	8
Scheduled Hours/Year	Hrs/Yr	8,400	8,400	8,400	8,400	8,400	8,400	8,400	8,400
(T) Total Theoretical	**Hrs/Yr**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**
(SU) Scheduled and Unscheduled Shutdown	Hrs/Yr	360	360	360	360	360	360	360	360
Standby									
Lunch Break	Hrs/Shift	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Shift Start/Shutdown	Hrs/Shift	0.40	0.40	0.40	0.40	0.40	0.40	0.40	0.40
Coffee Breaks	Hrs/Shift	0.33	0.33	0.33	0.33	0.33	0.33	0.33	0.33
Miscellaneous-Blasting and Moves	Hrs/Shift								
Total Standby	**Hrs/Shift**	**1.2**	**1.2**	**1.2**	**1.2**	**1.2**	**1.2**	**1.2**	**1.2**
(S) Total Standby	**Hrs/Yr**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**
Available Working Hours	Hrs/Day	20.3	20.3	20.3	20.3	20.3	20.3	20.3	20.3
Available Working Hours	Hrs/Yr	7,109	7,109	7,109	7,109	7,109	7,109	7,109	7,109
Annual Hours									
(T) Total Theoretical	**Hrs/Yr**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**
(SU) Scheduled and Unscheduled Shutdown	Hrs/Yr	360	360	360	360	360	360	360	360
(S) Total Standby	**Hrs/Yr**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**
(W)+(R) Work + Repair = (T-S-SU)	Hrs/Yr	7,109	7,109	7,109	7,109	7,109	7,109	7,109	7,109
(W) Work = MA x (T-S-SU)	Hrs/Yr	5,829	5,829	5,829	5,829	5,829	5,829	5,829	5,829
Mechanical Availability									
Scheduled Downtime	Shifts/Yr	84	84	84	84	84	84	105	105
Scheduled Downtime	Hrs/Yr	672	672	672	672	672	672	840	840
Scheduled Downtime		8.0%	8.0%	8.0%	8.0%	8.0%	8.0%	10.0%	10.0%
Unscheduled Downtime		10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%
Total Downtime		**18.0%**	**18.0%**	**18.0%**	**18.0%**	**18.0%**	**18.0%**	**20.0%**	**20.0%**
Shifts Available for Scheduling	Shifts	966	966	966	966	966	966	945	945
(MA) Mechanical Availability		82.0%	82.0%	82.0%	82.0%	82.0%	82.0%	80.0%	80.0%
(PA) Physical Availability = (W + S)/T		81.3%	81.3%	81.3%	81.3%	81.3%	81.3%	79.7%	79.7%
(UA) Use of Availability = W/(W+S)		81.9%	81.9%	81.9%	81.9%	81.9%	81.9%	81.5%	81.5%
(EU) Effective Utilization = PA x UA		66.5%	66.5%	66.5%	66.5%	66.5%	66.5%	64.9%	64.9%
Annual Production									
(WH) Work Hours/Yr	Hrs/Yr	5,829	5,829	5,829	5,829	5,829	5,829	5,687	5,687
Operating Efficiency – operation based		90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%	90.0%
(PH) Production Hrs/Yr	Hrs/Yr	5,246	5,246	5,246	5,246	5,246	5,246	5,118	5,118
(BC) Bucket Capacity (heaped)	cm	16.50	16.50	16.50	16.50	10.50	10.50	11.50	11.50
(MW) Material Weight	kg/bcm dry	2850	2830	2000	2000	2000	2830	2850	2830
(MWW) Material Weight Wet	kg/bcm wet	2870	2850	2300	2300	2300	2850	2870	2850
(BF) Bulk Factor		1.35	1.35	1.15	1.15	1.15	1.35	1.35	1.35
(MW1) Material Weight = MW/BF	kg/lcm dry	2,111.1	2,096.3	1,739	1,739.1	1,739.1	2,096.3	2,111.1	2,096.3
(M) Moisture		5.00%	5.00%	30.00%	30.00%	30.00%	5.00%	5.00%	5.00%
(FF) Fill Factor		0.89	0.89	0.81	0.81	0.80	0.75	0.75	0.75
(EBC) Effective Bucket Capacity = FF x BC	cm	14.69	14.69	13.37	13.37	8.40	8.63	8.63	8.63
(MW2) Material Weight = MW1/(1-M)	wmt/lcm	2.13	2.11	2.00	2.00	2.00	2.13	2.13	2.11
Material Weight = MW2 x (1-M)	dmt/lcm	2.11	2.10	1.74	1.74	1.74	2.11	2.11	2.10
(TP) Tonnes/Pass	wmt	31.22	31.00	26.73	26.73	16.80	18.34	18.34	18.21
(TC1) Truck Size Capacity	CM heap	78.0	78.0	78.0	78.0	60.1	60.1	60.1	60.1
(TC2) Truck Size Capacity	wmt	165.3	165.3	165.3	165.3	104.7	104.7	104.7	104.7
(TPV) Theoretical Passes = TC1/EBC	passes	5.31	5.31	5.84	5.84	7.15	6.97	6.97	6.97
(TPT) Theoretical Passes = TC2/TP	passes	5.29	5.33	6.18	6.18	6.23	5.71	5.75	5.75
(AP) Actual Passes = ROUND TPT	passes	5.0	5.0	6.0	6.0	6.0	6.0	6.0	6.0
(TL) Truck Load-Volume = AP x TP	cm	73.4	73.4	80.2	80.2	50.4	51.8	51.8	51.8
(TLS) Truck Load for Simulation = AP x TP	wmt	156.1	155.0	160.4	160.4	100.8	110.0	110.0	109.3
(TLP) Truck Load for Productivity	dmt	155.0	153.9	139.5	139.5	87.7	109.3	109.3	108.5
(TCU) Truck Capacity Utilized = TLS/TC2	by weight	94.4%	93.8%	97.0%	97.0%	96.3%	105.1%	105.1%	104.4%
Truck Capacity Utilized = TL/TC1	by volume	94.1%	94.1%	102.8	102.8	83.9%	86.1%	86.1%	86.1%
(AC) Average Cycle Time	sec	35	35	35	35	35	45	45	45
(ST) Truck Spot Time	sec	24	24	24	24	24	24	24	24
(LT) Load Time per Truck = AP x AC + ST	sec	199	199	234	234	234	294	294	294
(LT) Load Time per Truck = AP x AC + ST	minutes	3.32	3.32	3.90	3.90	3.90	4.90	4.90	4.90
(MP) Maximum Productivity = 60/LT	trucks/hr	18.1	18.1	15.4	15.4	15.4	12.2	12.2	12.2
Conversion = MP x TLP/MW	bcm/hr	983.9	983.9	1,072.8	1,072.8	574.4	469.4	469.4	469.4
(TPHM)	t/hr	2,804.2	2,784.5	2,145.6	2,145.6	1,348.5	1,625.4	1,337.8	1,328.4
(SS) Scheduled Shifts/Yr (from above)	shifts/yr	966	966	966	966	966	945	945	94.5
(PH) Production Hours/Yr (from above)	hrs	5,426	5,426	5,426	5,426	5,426	5,118	5,118	5,118
(TA) Truck Availability to Shovel	%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95.0%	95%
(TPHA) Production Adjusted = TPHM x TA	t/hr	2,664	2,645	2,038	2,038	1,281	1,544	1,271	1,262
(RP) Real Production = TPHA x PH	t/yr	13,975,341	13,877,268	10,692,935	10,692,935	6,720,588	8,100,798	6,504,452	6,458,806
Production/Scheduled Shift = RP/SS	t/shift	14,467	14,366	11,069	11,069	6,957	8,386	6,883	6,835
Production/Scheduled Work Hrs = RP/WH	t/hr	2,398	2,381	1,834	1,834	1,153	1,390	1,144	1,136
Production/Scheduled Production Hrs = RP/PH	t/hr	2,664	2,645	2,038	2,038	1,281	1,544	1,271	1,262

Open Pit Truck Productivity

The design assumptions used to determine hauling productivity are shown in Table 17.2.4.4. Both 100 and 150t trucks are planned to be used. Estimated truck productivity is shown in Table 17.2.4.5.

Table 17.2.4.4: Open Pit Hauling Parameters

Schedule Data	Units	Waste 100t	Ore 100t	Ore Bedrock 100t	Ore Bedrock 100t	Ore Bedrock 100t	Rock 100t
Calendar Days	days/yr	365	365	365	365	365	365
Scheduled Shutdown-weather	days/yr						
Unscheduled Days Down-weather	days/yr	15	15	15	15	15	15
Mine Work Days	days/yr	350	350	350	350	350	350
Work Days/Week	days/yr	7	7	7	7	7	7
Shifts/Day	Shifts/Day	3	3	3	3	3	3
Shifts/Week	Shifts/*Wk*	21	21	21	21	21	21
Scheduled Weeks/Year	Wk/Yr	50	50	50	50	50	50
Shifts/Year	Shifts/Yr	1050	1050	1050	1050	1050	1050
Scheduled Hours/Shift	Hrs/Shift	8	8	8	8	8	8
Scheduled Hours/Year	Hrs/Yr	8,400	8,400	8,400	8,400	8,400	8,400
(T) Total Theoretical	**Hrs/Yr**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**
(SU) Scheduled and Unscheduled Shutdown	Hrs/Yr	360	360	360	360	360	360
Standby							
Lunch Break	Hrs/Shift	0.5	0.5	0.5	0.5	0.5	0.5
Shift Start/Shutdown	Hrs/Shift	0.4	0.4	0.4	0.4	0.4	0.4
Coffee Breaks	Hrs/Shift	0.3	0.3	0.3	0.3	0.3	0.3
Miscellaneous-Blasting and Moves	Hrs/Shift						
Total Standby	**Hrs/Shift**	**1.2**	**1.2**	**1.2**	**1.2**	**1.2**	**1.2**
(S) Total Standby	**Hrs/Yr**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**	**1,292**
Available Working Hours	Hrs/Day	20.3	20.3	20.3	20.3	20.3	20.3
Available Working Hours	Hrs/Yr	7,109	7,109	7,109	7,109	7,109	7,109
Annual Hours							
(T) Total Theoretical	**Hrs/Yr**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**	**8,760**
(S) Total Standby	**Hrs/Yr**	**360**	**360**	**360**	**360**	**360**	**360**
(SU) Scheduled and Unscheduled Shutdown	Hrs/Yr	1,292	1,292	1,292	1,292	1,292	1,292
(W)+(R) Work + Repair = (T-S-SU)	Hrs/Yr	7,109	7,109	7,109	7,109	7,109	7,109
(W) Work = MA x (T-S-SU)	Hrs/Yr	5,829	5,829	5,829	5,829	5,829	5,829
Mechanical Availability Definition							
Scheduled Downtime	Shifts/Yr	84	84	84	84	84	84
Scheduled Downtime	Hrs/Yr	1008	1008	1008	1008	1008	1008
Scheduled Downtime		8.0%	8.0%	8.0%	8.0%	8.0%	8.0%
Unscheduled Downtime		10.0%	10.0%	10.0%	10.0%	10.0%	10.0%
Total Downtime		**18.0%**	**18.0%**	**18.0%**	**18.0%**	**18.0%**	**18.0%**
Shifts Available for Scheduling	Shifts	966	966	966	966	966	966
(MA) Mechanical Availability	%	82.0%	82.0%	82.0%	82.0%	82.0%	82.0%
Physical Availability							
(PA) Physical Availability = (W+S)T		81.3%	81.3%	81.3%	81.3%	81.3%	81.3%
Use of Availability							
(UA) Use of Availability = W/(W+S)		81.9%	81.9%	81.9%	81.9%	81.9%	81.9%
Effective Utilization							
(EU) Effective Utilization = PA x UA		66.5%	66.5%	66.5%	66.5%	66.5%	66.5%
(WH) Work Hours/Year	Hrs/Yr	5,829	5,829	5,829	5,829	5,829	5,829
Operating Efficiency – operation based		90.0%	90.0%	90.0%	90.0%	90.0%	90.0%
(PH) Production Hrs/Yr	Hrs/Yr	5,246	5,246	5,246	5,246	5,246	5,246

Table 17.2.4.5: Open Pit Estimated Truck Productivity

Description	Units	-1	1	2	3	4	5	6
Haul Cycles								
Overburden	minutes	13.00	14.00	14.00	15.00	15.00	15.00	15.00
Open Pit Ore	minutes	13.00	13.00	15.00	17.00	15.00	16.00	17.00
	minutes							
Stockpile	minutes	13.00	2.00					
Waste	minutes	15.00	16.00	17.00	15.00	16.00	17.00	19.00
Tailings Waste	minutes	17.00	17.00	19.00	17.00	18.00	19.00	21.00
Quantities								
Overburden	t(000s)	5,342.4	53.8	1,627.5	1,786.1			
Underground Ore	t(000s)							
Open Pit Ore	t(000s)	244.1	528.3	624.2	489.8	568.7	523.0	384.3
Waste-Large Shovel	t(000s)	3,005.1	7,921.5	5,206.3	5,312.3	7,072.8	3,412.3	2,386.5
Waste-Small Shovel	t(000s)	333.9	880.2	578.5	590.3	785.9	379.1	265.2
	t(000s)							
Total	**t(000s)**	**8,925.4**	**9,383.8**	**8,036.5**	**8,178.5**	**8,427.3**	**4,315.5**	**3,036.0**
150 Ton Trucks	t(000s)	8,347.4	7,975.3	6,833.9	7,098.4	7,072.8	3,412.3	2,386.5
100 Ton Trucks	t(000s)	578.0	1,408.5	1,202.6	1,080.1	1,354.5	902.2	649.5
Contract Mine	t(000s)							
Loading Distribution								
Ore								
16 CM Shovel	t(000s)							
10.5 CM Excavator	t(000s)	244.1	528.3	642.2	489.8	568.7	523.0	384.3
Total	**t(000s)**	**244.1**	**528.3**	**642.2**	**489.8**	**568.7**	**523.0**	**384.3**
Waste								
16m^3 Shovel	t(000s)	3,005.1	7,921.5	5,206.3	5,312.3	7,072.8	3,412.3	2,386.5
10.5m^3 Excavator	t(000s)	333.9	880.2	578.5	590.3	785.9	379.1	265.2
16m^3 Shovel-Overburden	t(000s)	5,342.4	53.8	1,627.5	1,786.1			
18m^3 FEL	t(000s)							
Total	t(000s)	**8,681.3**	**8,855.5**	**7,412.4**	**7,688.7**	**7,858.6**	**3,791.5**	**2,651.7**

17.2.5 Mine Equipment

The open pit will be mined using equipment currently located at Apollo's Montana Tunnels mine, located near Helena, Montana. This equipment, shown in Table 17.2.5.1 will become available as ore from Montana Tunnels is depleted from the L-Pit operations and some new equipment is purchased for the M-Pit expansion. Further, the equipment selected for shipment to Black Fox includes only those units in good working condition and appropriate for the Black Fox mining plan.

Table 17.2.5.1: Montana Tunnels Mine Equipment

Equipment Description (Make & Model)	Equipment Identification No.
Backhoe (235 CAT)	A06
Dozer (CAT D9N)	B04
Dozer (CAT D11N)	B12
I-R DM45E	D01
I-R DM45E	D02
CAT 16G Motor Grader	G01
CAT 992C Front End Wheel Loader	L06
CAT 325B Excavator	L35
5230 CAT Hydraulic Front Shovel	S03
CAT 777C 85t Haul Truck	T10
CAT 785B 150t Haul Truck	T14
CAT 785B 150t Haul Truck	T15
CAT 785B 150t Haul Truck	T18
CAT 777C 85t Haul Truck	T20
Water Truck (CAT 769C)	W04
Mobile Crane (GROVE RT 635)	A04
Fork Lift (L60 Lift All	A05
Maintenance Truck (FORD F7000)	A09
Loader (CAT IT-28)	A12
Backhoe (John Deere 310C) Mill	A13
Fuel Truck (Ford F8000)	A21
Lube Truck (Kenworth T800)	A22
Lube Truck (Ford F8000)	A26
CAT 232 Skid Steer Mill	A29
Bobcat Loader (743)	MEBOB
Terex Light Plant AL5080D4	Q11
Terex Light Plant AL5080D4	Q12
CAT 232 Skid Steer Mill	A29

Black Fox will supplement the Montana Tunnels equipment with new equipment as required. New equipment will include:

- One mass excavator;
- Three trucks, leased in Year 02; and
- One hydraulic drill, leased in Year 01.

17.2.6 Overburden Slopes, Waste, and Overburden Stockpiles

AMEC completed a Pre-feasibility Design Study of Open Pit Overburden, Waste Stockpiles and Tailings Impoundment in December 2004. The following paragraphs were extracted from their report.

The tailings impoundment discussion can be found in Section 17.4.

Overburden Slopes

"For the proposed open pit, the stability of the overburden slope is a primary consideration due to the loose and wet silty materials encountered at the site. Also, potential impact of pit dewatering on the adjacent lake (Froome Lake) as well as the impact of the pit and mine operation on the highway (e.g., blasting, potential highway settlement from lowering groundwater level, traffic) are of concern. In this study, attention is focused on the slope stability issue.

The potential impact of pit dewatering on the lake will be addressed in a hydrogeological study report. Potential for highway settlement will be addressed at the feasibility design level. This issue is not considered critical to the project at this stage. The issues of highway traffic and blasting next to the highway are being addressed under the "permitting" part of the project.

A settling pond for pit water will be required to settle suspended solids. Water from dewatering of the pit may also require chemical treatment, e.g., for arsenic. Further geochemical evaluation will be required to determine treatment requirements, after the pit geological model is available. For the purpose of this study, it has been assumed that no treatment other than for suspended solids will be required for the pit water, including also water collected in the underground workings.

It is noted that it might or might not be feasible to utilize the tailings impoundment as a settling impoundment for the pit water, depending on the cyanide destruction strategy. Consistent with the previous studies by Apollo, it is assumed that the tailings impoundment excess water will be treated in the mill, after reclaiming from the tailings pond (i.e., no discharge from the tailings impoundment will be permitted). Hence, for the purpose of the current study, pumping pit water to the tailings impoundment is not considered feasible since the tailings pond water will be subject to cyanide treatment.

It is noted that should the pit water require treatment for arsenic, it would have to be pumped from the pit settling pond to the mill and then, possibly, mixed with the tailings pond water after the cyanide is destroyed (prior to treatment for arsenic).

In summary, for the purpose of this pre-feasibility design study, it is assumed that the pit water will require treatment for suspended solids only and will be released to the environment at a distinct discharge point.

No specific site closure requirements with respect to the open pit have been incorporated into the designs at this stage. It is assumed that the pit will be allowed to flood and thus form a small lake.

Stability of the overburden slope under static loading conditions: Factor of Safety ≥ 1.5. The seismic stability of the pit slopes has not been addressed at this stage and will have to be accounted for in the next design phase. Adjustments to the pit design slopes might then have to be introduced.

Four typical sections approximately corresponding to the east, south, west and north sides of the proposed open pit were developed for slope stability modeling conducted to determine the safe angles of the excavated overburden slopes. A limit equilibrium method was employed through using computer software Slope/W version 5.

The subsurface conditions for each identified section of the pit were selected based on the results of the field investigation in the area of the proposed open pit. Soil parameters were estimated from field and laboratory investigation results. The groundwater levels were based on the most recent readings of the piezometers. For each slope, different failure mechanisms (i.e., different slip surfaces through critical soil layers) were considered to examine both overall and local stability of the slopes.

A factor of safety of 1.5 is considered to be a minimum safety margin against slope failure. The factor of safety for the overall stability is between 1.47 and 1.80, and for other potential slip surfaces, the factor of safety is between 1.49 and 2.81.

Based on the above findings, the north and west open pit slopes would be excavated at 3 horizontal to 1 vertical (3H:1V). However, the east and south slopes would have to be excavated at 5H:1V due to the presence of very loose to compact silty sand till on the south side, and firm silty clay on the east side of the open pit.

The overall slope of 3H:1V to 4H:1V for the various segments of the open pit is considered to be adequate from the stability perspective, under the assumption that the slope configuration will not change with time.

At this pre-feasibility design level, only static loading conditions were considered in determining the safe pit overburden slopes. The stability of the slopes under seismic conditions, accounting for the liquefaction potential, will be examined at the feasibility design level. Also, potential for soil liquefaction under seismic loads generated from blasting in the pit will be addressed at that stage.

It is noted that even though the overall slope, as shown, has an adequate margin of safety from the perspective of resistance to shear failure, this safety margin could be easily be affected by erosion of the slope (by surface water and/or groundwater), which could be extensive in the silty and fine-grained sand materials. Therefore, erosion protection over the pit overburden slopes will likely be required.

Waste Rock Stockpile

Waste Rock Characterization

Based on the available data, it appears that although the waste rock would not generally be net acid generating. The concentrations of As and Ni dissolved from the rock could be marginal with respect to discharge limits, however, this is a preliminary conclusion drawn from the BC MEM test results only. It is noted that As and Ni could be released from the waste rock during flushing events (e.g., during spring runoff and/or fall rains) in concentrations exceeding regulatory limits. The concentration of ammonia in the surface water derived from the waste rock also appears to be high, and in excess of regulated values.

The source mineral(s) containing arsenic still is unclear, however, the review of the current data suggests that more than one mineral source is possible. In summary:

- *Some arsenic occurs in arsenopyrite (identified in sample Apollo 12);*
- *Arsenic may also substitute within the structure of other sulphides such as pyrite; and*

- *The leach data suggest that release of arsenic is not consistent with the oxidation of sulphide minerals and that arsenic may also be associated with an arsenate salt such as scorodite.*

Based on the available data, it appears that arsenic found in the rock samples is associated with some identifiable rock types and mineral fractions. The concentration of arsenic in the samples increased with carbonate contents as estimated by three separate methods. The highest concentrations of arsenic are associated with samples identified as "green carb" and "grey carb", but not all samples dominated by carbonate necessarily contain high concentrations of arsenic. This suggests that arsenic may also exist in accessory mineral(s) associated with samples predominated containing carbonate.

Waste Rock Stockpile Design

For the waste rock stockpile, physical stability of the dump slopes, required footprint area and the potential for arsenic leaching are the primary design considerations. The subsurface conditions explored during the summer 2004 field investigation indicated that the South Block is the most favorable location for the waste rock stockpile.

Site selection and optimization of the stockpile configuration (accounting for the geotechnical site conditions) have been the main focus of this study with regard to the waste rock stockpile, with two possibilities addressed:

- *The stockpile runoff may/may not require treatment for arsenic.*

Regardless of the runoff treatment requirements, runoff from the waste rock stockpile will have to be routed through a settling facility to control the level of suspended solids. No specific closure requirements have been incorporated into the waste rock stockpile designs at this stage, except that the slopes will have to remain stable in the long-term. While it has been assumed that no special cover over the stockpile would be required, an allowance needs be made for some nominal grading, soil cover, and vegetation work.

Design criteria:

- *Total waste rock mass: 44Mt;*
- *Slope stability under static loading: Factor of Safety ≥ 1.3 (seismic stability will be addressed at the next design stage);*
- *Buffer zone to the adjacent lakes: ≥ 100m; and*
- *Buffer zone to other property lines: 50m.*

The proposed waste rock stockpile will be located in the beaver pond and low lying areas to the south of the open pit and to the west of the bedrock outcrop. Field investigations indicate that the subsurface soil comprises generally dense to very dense sand and silt overlying bouldery till. The stratum of dense sands and silts presents a competent foundation for the relatively high stockpile of waste rock. However, in the northwest area of the proposed footprint, the silty clay layer (encountered in BH04-40) presents a relatively weak foundation layer. Hence, two models were developed to determine the required overall slope angles for the stockpile.

The soil parameters used for each subsurface unit were estimated from the field and laboratory tests results and the phreatic surfaces were based on the measured groundwater levels. In view of the design safety factor of 1.3 set for the waste stockpiles, the tests indicate that the waste rock can generally be stockpiled to a height of 50 m with 2H:1V side slopes in most of the areas.

Along the northwest segment of the stockpile, the waste rock should be placed higher than 40 m if a slope of 5H:1V is to be maintained (due to the presence of the silty clay stratum).

Closure For the waste rock stockpile, the exterior slopes will be stabilized, where required. Some material from the overburden stockpile will be used at locations at the top and on the slopes of the stockpile to accelerate the start up of vegetative growth.

Overburden Stockpile

For the overburden stockpile, the strength properties of the (re-worked) overburden waste, physical stability of the dump slopes and the required footprint area are the primary design considerations.

The subsurface conditions explored during the field investigation carried out in the summer of 2004 indicated that the West Block is favorable for siting of the overburden stockpile. The strength properties of the reworked overburden materials have been assumed based on the results of geotechnical investigation carried out within the proposed pit area.

The runoff from the overburden stockpile will have to be routed through a settling facility to control the level of suspended solids.

Similar to the waste rock stockpile, no specific closure requirements have been incorporated into the overburden stockpile designs at this stage, except that the slopes will have to remain stable in the long-term. Some of the stockpiled overburden material may be used for final reclamation of other project areas. As a minimum, an allowance needs be made for some nominal grading and vegetation of the stockpile.

Design criteria:

- *Total overburden mass: 11Mt;*
- *Slope stability under static loading conditions: factor of safety ≥1.3;*
- *Buffer zone to the adjacent lakes: ≥100m;*
- *Buffer zone to the highway: ≥100m; and*
- *Buffer zone to other property lines: ≥50m.*

Based on the field investigation carried out in the summer of 2004, the overburden soils at the open pit site primarily consist of compact silt, sand and till deposits. This indicates more advantageous soil conditions for handling and stockpiling as compared with those expected during the fatal flaw study. This has an implication regarding the slope design for the overburden stockpile in consideration of the effects of the strength of overburden (stockpiled) soils on the slope stability.

The West Block is generally underlain firm silty clay, loose to dense silty sand to sandy silt and till deposits with the exception of the southeast areas close to Froome Lake, underlain by a layer of soft clay. Slope stability analyses were carried out to design the stockpile configuration for 11 million tonnes within the limits of the West Block.

The results (of the stability analysis) indicate that the construction of a 25 m high overburden stockpile with side slopes of 4H:1V is generally feasible. However, in southeast area of the stockpile footprint underlain by the soft silt clay deposit, the overall side slope should be 14H:1V or flatter. In other words, extensive "stabilizing berms" will be required at that location."

17.3 Underground Mining

To continue mining of the ore resources at Black Fox below the 9815m level, underground mining methods were reviewed that would minimize dilution, capital, and operating costs, maximize recovery of the ore resources while maintaining the design production capacity of the mill (1,500t/day).

The medium dipping (45º to 50º) deposit as well as the underground mine ore production goal of 1,500t/day required the review and selection of the highly selective cut and fill mining method. Alternative bulk underground mining methods were not selected due to the potential for greater dilution and potential ground control issues due to the moderate dip.

17.3.1 Stope Design

The underground stopes (Figure 17-3) were designed by contouring the resource model on 2g/t-Au grade contours. The 2g/t-Au contour was used to define the stope boundary as long as 4g/t-Au or higher material remained within the 2g/t-Au contour. Other design criteria included a minimum width of 3m, the average width of 8m and the average length of 34m. The stopes (Figure 17-4) were designed on 5m high cuts with stope access cross cuts from the main development every 20m in elevation. In addition a 25m thick crown pillar was established between the bottom of the open pit and the top of the upper most underground production stopes. There will not be any production ore taken from within the crown pillar area.

Table 17.3.1.1 summarizes the ore reserves for the underground portion for Black Fox.

Table 17.3.1.1 Underground Probable Ore Reserves as of June 30, 2007

	Ore(kt)	Grade g/t-Au	Contained oz-Au
Underground Reserves	1,108	10.6	377,000

Table 17.3.1.2 summaries the mine design parameters selected for the underground portion of the reserves for Black Fox.

Table 17.3.1.2: Underground Design Parameters

Parameter	Description	Value
Operating Schedule	Mine Schedule	3 shifts/day; 7 days/week, 360 days/year
Production Target	Daily / Annual	1,500t/d/540,000t/a
Ramp Haulage	One Way Haul	3,500m
	Grade	15% max
	Haul way Dimensions	5m high and 5m wide
Mine Production	Production Stope	5.0m wide x 5.0m high cuts
	Longest haul to ore pass	300m
Backfill Requirements	Waste Rock Fill	233,160m^3
	Tailing Backfill	62,176m^3
	Cemented Tailing Backfill	15,544m^3
General	Specific Gravity	Ore: 2.8 and Waste: 2.8 Backfill 1.76t/cubic meters

17.3.2 Cut & Fill Mining

Black Fox underground ore zone orientation, especially the moderate dip (45º) and a non-continuous ore zone structure (pods) required careful consideration of different underground mining methods. The cut and fill mining method, utilizing a mining cross-section of 5m high x 5m wide for the cut and a combination of uncemented rock backfill (75%), sized tailings backfill (20%), with a final cap of cemented sized tailings backfill (5%) for the fill. Cut and fill was selected due the versatility of the method to allow the minimal amount of dilution while, meeting the production throughput target of 540,000t/yr (1,500t/day).

The cut and fill production stopes will be a minimum of 5m wide and 5m high due to the mining equipment size constraints. The stope configuration will follow the general cross-sectional shape along the strike of the ore zone to minimize dilution. Referred to as a "shanty style" cross section the footwall and hanging wall of the stope will follow the ore zone as closely as possible.

Mining Sequence

Stope Access Drifts (SAD) (dimensions of 4m x 4 m) will be developed to the respective stope area from the development headings. Mining will start at the lowest level of ore in the designated mining stope with an extension of the stope access drift called a production stope ramp (PSR-also 4m high x 4m wide). Once the ore has been accessed the area is mined in a series of 5m high horizontal cuts along strike with the width dependent on the individual stope dimensions. The minimum width is 5m and is determined by the mining equipment requirements and the maximum width of 20m as discussed in the AMEC Geotechnical Assessment Memo (Underground Geotechnical Assessment – prefeasibility Level Design – February 17, 2006). The first level in each respective ore zone will be mined using conventional pull round techniques incorporating a burn in the drill pattern. Each subsequent 5m high cut will use a breast down drill pattern as well as associated blasting timing and blasting agent-loading requirements. After each mucking cycle has been completed, a mechanical bolter will be used to place the roof bolts, mats and screen as required by specific stope requirements following a general bolting pattern of 2m x 2m spacing. The jumbo drill will drill approximately 73 holes (for a 5m x 5m stope) with a diameter of 50.8mm and 3.7m deep. The holes will be loaded with ANFO explosives, cast boosters and timed using nonel blasting caps. Sufficient smoke time is allowed (minimum of 30 minutes) before the area is re-entered. The new muck pile is wet down and then scaled by hand from the muck pile. After careful inspection and determination that the ground is safe and secure, the mining sequence is repeated until the farthest extent of the ore is reached for each particular cut. Once the extent of the ore has been reached for the each specific 5m horizontal slice the back fill sequence is started.

Stope Backfill

Backfilling the stope is required to establish a working platform and provide ground control to help maintain the hanging wall on the next horizontal slice within the individual stope area. Backfilling consists of filling the stope to within 0.5m of the stope back (approximately 90% of the volume). Of the total backfill required 75% will be rock-fill with material supplied from the "dirty rock" stockpile from the open pit waste rock stockpile plus underground development waste. This material is stored on the surface and will be hauled back into the mine on the back-haul from the underground ore hauling trucks. The trucks will dump the waste rock in the stope as required with the final placement performed by a 6yd^3 LHD. The next sequence will be the tailing sand fill provided by the sand plant on the surface. The hydraulic sand (approximately 60% solids and 40% water) is supplied through a network of steel sand lines (10cm to 4in dia.) to the individual stope areas. PVC pipe (10cm) is used in the stope and attached to the back to allow the final placement of the sand fill in the stope. Special bulkhead preparations are completed at the stope opening to allow the placement of sand and the release of the tailing water. The sand fill is poured to within 1.0m of the back and the final cemented sand fill (with 5% cement added) layer is then poured to within 0.5m of the back. The cemented sand fill layer allows for the complete recovery of the ore within the next subsequent cut by providing a hard surface for the LHD mucker to run on and the ability to completely wash-down the stope after the mining cut is completed, but before the backfill sequence begins. The final cut in the respective ore zone does not require backfilling and is left open.

17.3.3 Dilution

The underground mining dilution estimation methodology accounts for the planned and unplanned dilution due to stope widths and geometry. In the case of the Black Fox deposit special consideration was required since the average dip of the orebody is approximately 45º. Additional parameters such as rock type, stope width, structural constraints, and opening size requirements for men and equipment were also considered. The minimum mining width has been determined to be 3m with a mining height of 5m.

For the purposes of this report dilution is defined as:

$$\text{Dilution (\%)} = \frac{\text{Planned and Unplanned Dilution x 100}}{[\text{Ore + Planned \& Unplanned Dilution}]}$$

The overall dilution for the underground project has been estimated to be 22.5%. The dilution accounts for a total of 249kt of material with an average grade of 0.35g/t-Au.

SRK recommends that the vein hanging wall be followed as closely as possible when the stope is developed to avoid undercutting the hanging wall. The “shanty” drift profile is recommended for this application since it allows the natural vein orientation to be more closely followed. The backfill application will also help to maintain the hanging wall during the subsequent cut on the next level.

17.3.4 Underground Development

The underground mine development schedule was developed based on the underground mine production requirement that ties to the open pit mine schedule, with the goal of providing an un-interrupted flow of ore to the mill at the prescribed 1,500t/day. Allowances were made in the underground mine plan and schedule to develop sufficient stopes and subsequently enough working faces to meet the production requirements. Additional allowances were built into the schedule to allow for unforeseen delays such as mechanical breakdowns, and development access delays. Table 17.3.4.1 describes the development requirements for the underground mining portion for Black Fox.

Table 17.3.4.1: Underground Mine Development

Description (meters)	2014	2015	2016	2017
Main Ramp (5m x 5m)	1,570	2,505	907	
Muck Bays	110	175	63	
Stope Access Ramps (4m x 4m)		565	2,371	1,342
Production Stope Ramps (4m x 4m)		360	600	510
Ventilation Raises		455	441	
Total (m)	**1,680**	**4,060**	**4,382**	**1,852**

17.3.5 Underground Production Schedule

The underground mine production schedule, while independent from the open pit mine schedule, ties into the open pit mine schedule to insure a continuous flow of ore to the on site milling facility. The underground mine production is phased in as the open pit mine completes its operational mine life.

Table 17.3.5.1: Underground Production Schedule

Description	2015	2016	2017	U/G Total
Ore (kt)	129	541	437	**1,108**
Grade (g/t)	10.3	11.5	9.5	**10.6**
Contained Au (koz)	43	201	133	377

17.3.6 Underground Productivity

Black Fox underground mining is divided into two main areas, development and production. Within each respective area the mining cycle is further divided into drilling, blasting, loading & hauling, ground control, and backfilling in the production stopes. These basic functions make up the underground mining cycle for Black Fox.

- Drilling - The two boom drill jumbo is used to drill 3.7m long holes with a diameter of 51mm into the face of the heading. The nominal size heading of 5m x 5m will require 45 holes for each round. The actual face advance will be 3.5m per round. This estimate is for the nominal pull round used in the development headings and initial stope development cut at the bottom of the production stope. Subsequent stope production will use a breast down mining technique with 30 holes drilled per nominal 5m wide x 5m high production round. The drilling cycle time is estimated to be 90min;
- Blasting – The 51mm holes are loaded with ANFO and using nonel blasting caps the heading is detonated. The estimated powder factor is 0.9kg/t of ore / waste depending on the area. The estimated powder factor for the breast down mining areas will be 0.7kg/t. The blasting cycle is estimated to be 60min;
- Loading & Hauling – Sufficient ventilation time is allowed before entering the heading. The heading back and ribs are barred down to maintain a safe working area as well as wet down for dust control. The 6yd^3 LHD mucker will either load the ore into the 28t haul truck directly or to a muck bay located near by. The objective is to clean the working face as soon as possible to allow the next mining sequence to take place. The ore/waste in the muck bay will be hauled as time permits. The estimated cycle time will be 200min per heading. Haul trucks will haul the ore to the surface ore stockpile with an estimated cycle time of 30min for the 6,000m round trip;
- Ground Control – Mechanical bolters are used to secure the back and ribs of the heading as required. Based on preliminary rock mechanics review the back and ribs will be secured using mechanical bolts (2m long on a 2m x 2m pattern) and screen as required. The cycle time for the bolting process has been estimated to be 150min per round; and
- Production Stope Backfilling – The last phase of the production mining cycle will be the backfill process. The rockfill, sandfill, and cemented sandfill will be placed as described in Section 17.3.2 with an estimated cycle time of 2.5 days for a 5m wide x 35m long stope.

17.3.7 Underground Mine Equipment

Table 17.3.7.1 lists the underground mining equipment cost and procurement schedule for the underground mine.

Table 17.3.7.1: Underground Equipment List

Description	2014	2015
Drill Jumbo – 2 Boom	1	1
LHD – 6yd^3	2	2
Haul Truck	2	2
Mechanical Bolter	1	1
Scissors Lift	1	
Powder Wagon	1	
Road Grader	1	
Lube / Fuel Truck	1	
Mechanics Vehicles	1	1
Boom Truck	1	
Tractors	2	2

- 2 Boom Drill Jumbo – electric/hydraulic used for drilling development and production headings;
- LHD - diesel 6yd^3 muckers for loading ore/waste into muck bays and or haul trucks, also used for placing rock backfill in stope areas as required;
- 28t capacity diesel underground haul trucks for hauling ore/waste/backfill as required;
- Mechanical Bolter used to placed rock bolts, mats and screen as required in the development and production headings;
- Scissors lift for installing mine services (water lines, air lines, ventilation, power) as required;
- Powder Wagon is a specialized utility vehicle fitted with a manbasket to allow timing and loading the explosives at the face;
- Road Grader is used to maintain the haulage and access roadways ;
- Lube/Fuel Truck to service the underground equipment as required;
- Mechanics vehicles are specially equipped diesel vehicles to allow the underground equipment to be serviced at the operating face as required;
- Boom Truck is a larger mechanics vehicle used for larger scale work on the equipment underground; and
- Tractors are used to transport man and some materials to the different working areas as required.

Table 17.3.7.2: Underground Fixed Equipment List

Description	2014	2015
Sand Backfill (Sand Fill) System (on surface)	1	
Ventilation System Upgrade	1	1
Water Management	1	
Underground Shop		1
Electrical	1	

- The sand fill system will be installed on the surface near the existing mill facility. A sized fraction of the tailings material will be stored in the sand tank as a slurry ready to be gravity fed into the underground sand fill distribution system. The sand fill system will also have the capacity to hold dry cement that will be introduced into the sand fill system as required for the top layer cemented sand fill previously described;

- The Black Fox mine currently has the capacity to ventilate the existing workings with approximately 300kW in fan power available. The system upgrade will be required to maintain the required ventilation quantities for the new underground mobile equipment;
- Water Management system upgrades will be required to keep the mine pumped out as required. The new sand fill system and additional mining areas will add additional excess water and the requirement for additional pumping capacity. In addition, an updated fresh/drill water reservoir and distribution system will be required;
- The new Underground Shop will be used for major repair work on the underground equipment without the requirement to bring the equipment to the surface shop facility; and
- Electrical system upgrades will be required to supply the necessary power to the new underground development heading and production workings.

17.3.8 Mine Support & Utilities

Mine Dewatering

The Black Fox mine is currently pumping water at an average of 500gal/min. This excess water is from the old workings of the mine. The new underground mining will require an estimated water pumping capacity due to the expanded underground working and the excess water generated during the sand fill backfill cycles during the production mining process. Excess water will report to the ditches along the drifts and then to two-compartment water sumps strategically located in the development heads. These sumps will be constructed to allow the dirty water to enter one side of the sump and then overflow into the clean water side of the sump. The sumps will be large enough to allow sufficient settling time before overflowing into the clean water sump. Periodically the dirty water side of the sump will be shut down and dewatered and the sediment mucked up and transported to the mill. Any extra gold particles will be recovered using this system. From the clean water section of the sump the excess water will be pumped to the surface and used in the mill as makeup water as required. The drill water sump will use the some of the excess water and pump it into the drill water supply distribution system. The drill water is also used in the stopes to wet down the muck piles for dust suppression as well as to complete the final wash down of the stopes before the backfill process is started. The upgraded water management system is designed to handle 1,000gal/min.

Compressed Air

The compressed air requirements are based on the estimated connected load with allowances for pipeline friction losses and leakage. The overall estimated usage of 1,000cfm will be supplied by installing 2-112kW (150hp) – 600cfm compressors on the surface. The compressed air will be distributed via the underground distribution system using 10cm (4in) dia. steel pipeline.

Underground Mine Electrical Distribution

The source for the underground mining power requirements will be from the main transformer on the surface. The estimated underground power requirements were developed for the underground mine equipment including, jumbos, bolters, pumping, ventilation and lighting. The estimate is based on the connected loads summarized in Table 17.3.8.1.

Table 17.3.8.1: Underground Power Requirements

Item	Units	Unit Rating (kW)	Connected (kW)	Utilization (%)	Est. Power (kW)
Compressors	2	112	224	40	90
Main Fans	4	150	600	95	570
Auxiliary Fans	6	93	558	75	419
Drill Jumbos	2	42	84	40	34
Bolters	2	42	84	40	34
Pump Station No 1	2	300	600	50	300
Pump Station No 2	2	300	600	50	300
Drill Water Pumps	2	43	86	50	43
Underground Shop	1	100	100	35	35
Miscellaneous	1	50	50	100	50
Total			**2,752kW**	**41%**	**1,875kW**

Diesel Fuel

A designed fuel/lube truck has been included in the design to fuel the underground mobile equipment fleet as required. The diesel haul trucks will refuel at the surface fuel station included in the infrastructure estimate.

Mine Maintenance

Maintenance on the underground mining fleet will be at the working area, or in the underground shop facility. The underground shop will be set up to perform all basic maintenance functions from routine daily maintenance to complete component exchanges. This will allow a more efficient use of the maintenance crews time. The haul trucks will be maintained at the truck facility on the surface that was constructed for the open pit mining phase. A predictive maintenance program will be utilized to maximize the physical availability of the mine equipment. All equipment will be on a rigid schedule of required maintenance with a dedicated crew assigned to the predicative schedule.

Office, Change House & Warehousing

The same surface facility used for the open pit mine will be used for the underground mining phase.

Underground Communications

Allowances have been made for underground communications via hard lines and leaky feeder communication technology.

Ventilation

The existing ventilation circuit consists of 2-150kW ventilation fans that discharge air down the ventilation shaft to the working areas with a return up the existing ramp. The current volume is 111m^3/s. This is more than sufficient during the current care and maintenance phase.

The underground mine ventilation circuit design is based on the use of two vent raises from the 9765 level to the surface. These raises will be used to intake fresh air into the mine with the exhaust air pushed up the main access decline to the surface. The exhaust air should be warm enough that freezing will not occur on the ramp roadway and affect the maneuverability of the haul trucks and ancillary equipment. Each intake ventilation raise will be equipped with a propane heater to be used in the winter months to keep the ventilation shafts free of ice. The main escapeway will be the main ramp and the secondary escapeway will be the ventilation raises. The raises will be equipped with ladders and landings to allow the miners a safe method of egress in fresh air if required.

The primary air flow will be down the raises to the 9765 level and then a series of air doors and auxiliary ventilation fans will direct the fresh air to the working development and production faces. As sections of the mine are mined out and access is no longer required bulkheads will be installed in strategic areas to eliminate the mined out area from the ventilation loop and thereby significantly reduce the future ventilation requirements for each respective area. As the mine is developed at depth, additional ventilation raises will be developed to tie in the new workings.

Table 17.3.8.2: Underground Ventilation Requirements

Item	Units	Unit Rating (kW)	Utilization %	Est Power (kW)	Airflow (m^3/s)
Drill Jumbos	2	43	50	43	4
Bolters	2	43	50	43	4
LHD-6yd^3	4	150	75	450	43
Haul Trucks	4	298	75	894	85
Scissor Lift	1	60	30	18	2
Road Grader	1	112	75	84	8
Fuel / Lube Truck	1	112	60	67	6
Boom truck	1	112	30	34	3
Tractors	4	24	30	29	3
Personnel & Maint. Storage					20
Subtotal	*13*	*13*	*12*	*11.5*	*178*
Misc. allowance (15%)					**27**
Total					**205**

The ventilation requirements of 205m^3/s are based on 0.0949m^3/s per kW which is 50% greater than the current standards. The current system will need to be upgraded from the current 111m^3/s capacity system by using the existing fans on the new east raise and adding 2 – 150kW new fans to the new west ventilation raise. These improvements have been accounted for in the capital schedule for the underground mine development and operation.

17.4 Tailings Storage Facility

Subsequent to toll milling in the early part of LoM, the proposed mill will process approximately 3Mt of ore from the pit and underground operations, and will generate an equivalent amount of tailings. A tailings facility will be constructed to contain 3Mt of solid tailings. A certain amount of tailings will be processed through a backfill plant for use in the underground mine operation.

Gold mineralization at Black Fox is very similar to that of other mines in the Porcupine Camp of which most of these mines have mineralization that is weakly to strongly neutralizing. Extensive Acid Base Analysis (ABA) testing performed on waste rock and simulated tailings samples suggests Black Fox ore is also neutralizing. As designed the proposed tailings facility will be adequate to prevent or mitigate sulfide oxidation (Dyck, 2007).

The mill will use water from the tailings facility in the milling process. Excess water will be treated and discharged to the environment seasonally (Dyck. 2007)

The following is excerpted from Apollo Gold Corporation Black Fox Project, Project Description for Small Pit & Mill Operation –Update-, Prepared for Ministry of Northern Development and Mines on behalf of Apollo Gold Corporation by AMEC Earth & Environmental, April 16, 2007 and has been standardized to this report.

"The design for the tailings facility is underway. The key design parameters and features of the tailings facility are as follows:

- *The tailings will be discharged under water and will remain permanently submerged;*
- *The tailings dam and the bottom of the impoundment will be practically impervious, as necessary to prevent unacceptable release of tailings water containing free cyanide;*
- *The slopes of the tailings dam will be designed taking into account the presence of the soft clay foundation (as determined during the field investigation carried out by AMEC in January-February 2004);*
- *The tailings dams will be designed to the required factors of safety, consistent with widely accepted dam engineering practice;*
- *The dam will be raised in stages, to allow for consolidation of the soft clay foundation and to minimize the initial capital construction cost. Also, foundation soil improvement (i.e., installation of wick drains) will be considered for future expansion to accelerate the consolidation of the soft clay foundations;*
- *Reclaim water recovery from the tailings impoundment will likely be carried out using a pump house, pumps and pipeline - all water will be returned to the mill for recirculation or for final treatment prior to disposal to the environment; and,*
- *Clean waste rock produced at the open pit, including the stripped overburden, will be available during the mine life and could be used for dam raising."*

17.5 Capital Costs

LoM capital costs are summarized in Table 17.5.1. Details supporting this estimate are discussed in this subsection.

Table 17.5.1: Capital Cost Summary (US$000s)

Description	Total	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017
Equipment												
Mining – OP	**7,811**	0	0	3,045	262	952	1,751	1,757	44			
Mining - UG	**15,013**	0	0	0	0	0	0	0	9,200	5,813		
Process	**60,448**	0	0	0	15,112	45,336						
Infrastructure	**16,244**	0	0	2,329	115	6325	2,300	2,300	1,725	1,150		
Owner Cost	**(4,989)**	575	1,725	288	2,014	288						(7,864)
Total	**94,527**	**575**	**1,725**	**5,662**	**15,489**	**52,612**	**4,051**	**4,057**	**11,257**	**6,963**	**0**	**(7,864)**
Development												
OP Pre-stripping	**8,026**			8,026								
UG Develop.	**18,712**								2,554	6,164	6,324	3,670
Total	**26,738**	**0**	**0**	**8,026**	**0**	**0**	**0**	**0**	**2,554**	**6,164**	**6,324**	**3,670**

17.5.1 Open Pit Mine Capital

Mine capital equipment costs were obtained by soliciting budget price proposals for new equipment, and a proposal to transport mine equipment from the Montana Tunnels operation. The estimated cost of mine equipment is shown in Table 17.5.1.1.

Table 17.5.1.1: Open Pit Mine Equipment Capital Cost (US$000s)

Description	Total	2007	2008	2009	2010	2011	2012	2013	2014	2015
Rotary Drill 251mm	**127**			64	64					
Hydraulic Drill 165mm	**759**			59	175	175	175	175		
Shovel 16m^3	**318**			318						
Bucket (Spare)	**15**			15						
Mass Excavator 12m^3	**584**						292	292		
Bucket (Spare) 12m^3	**0.0**									
Front End Loader 12m^3	**72**			72						
FEL Bucket (spare) 12m^3	**6**			5						
Truck 150t	**3,343**			583		690	1,035	1,035		
Truck 100t	**128**			127						
Water Truck 10k gal	**52**			52						
Dozer 350-400hp	**40**			40						
Dozer 800-900hp	**60**			60						
Grader 16ft	**30**			29						
Mass Excavator 4m^3	**40**			39						
Light Plant	**12**			12						
Stemming Truck	**91**			91						
Low Boy	**268**			267						
Lube Truck	**9**			9.0						
Fuel Truck	**7**			7						
Mechanic Truck	**11**			11						
Pickup Truck	**759**			599			60	60	40	
Welding/Crane Truck	**4**			4						
Crane-Rough Terrain 20t	**31**			31						
Skid Loader	**4**			4						
Integrated Tool Carrier	**22**			22						
Ambulance/Fire Equipment	**20**			20						
Flatbed Truck	**4**			4						
Crew Vans	**167**			107			30	30		
Forklift	**4**			4						
ATV	**16**			11				5		
Miscellaneous MT Tunnels	**100**			100						
Subtotal	*7,101*	*0*	*0*	*2,769*	*238*	*865*	*1,592*	*1,597*	*40*	*0*
Mobilization Demob.	**0.0**	0	0	0	0	0	0	0	0	0
Contingency (10.0%)	**710**	0	0	277	24	87	159	160	4	0
Total	**7,811**	**0**	**0**	**3,045**	**262**	**951**	**1,751**	**1,757**	**44**	**0**

Pre-stripping for the open pit mine, totaling US$8million is included in the US$79.8million LoM open pit operating cost shown in Section 17.6.1.

17.5.2 Underground Mine Capital

Underground capital costs are shown in Table 17.5.2.1. The underground development will start in 2014 to allow sufficient lead-time to develop the production stopes to supply a continuous source of ore for the mill. All of the equipment costs are based on budgetary quotes, include freight, and set up charges.

Table 17.5.2.1: Underground Capital Costs (US$000s)

Description	Total	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017
Mobile Equip.												
Jumbo – 2 Boom	**1,640**								820	820		
LHD – 6yd^3	**2,600**								1,300	1,300		
Haul Truck	**2,520**								1,200	1,260		
Mech. Bolter	**1,200**								275			
Scissors Lift	**275**								600	600		
Powder Wagon	**320**								320			
Road Grader	**400**								400			
Lube/Fuel Truck	**250**								250			
Mech. Vehicles	**150**								75	75		
Boom Truck	**250**								250			
Tractors	**300**								150	150		
Subtotal	*9,905*								*5,700*	*4,205*		
Fixed Equip.												
Sand fill	**1,250**								1,250			
Ventilation	**700**								350	350		
Water System	**500**								200	300		
UG Shop	**200**									200		
Electrical	**500**								500			
Subtotal	*3,150*								*2,300*	*850*		
Subtotal	*13,055*								*8,000*	*5,055*		
Cont. (15%)	**1,958**								1,200	758		
Total	**15,013**								**9,200**	**5,813**		

Underground mine development costs are shown in Table 17.5.2.2.

Table 17.5.2.2: Underground Mine Development Costs

Description	Total	2014	2015	2016	2017
Main Ramp					
Main Ramp (m)	**4,982**	1,570	2,505	907	
Operating Labor ($000s)	**1,724**	482	931	311	
Maintenance Labor ($000s)	**829**	393	330	107	
O&M ($1,070/m)	**5,331**	1,680	2680	970	
Cost ($000s)	**$7,884**	**$2,554**	**$3,941**	**$1,388**	
$/m	$1,582/m	$1,627/m	$1,573/m	$1,530/m	
Stope Access Drifts					
Stope Access Drift (m)	**4,278**		565	2,371	1,342
Operating Labor ($000s)	**2,096**		210	813	1,073
Maintenance Labor ($000s)	**725**		74	281	370
O&M ($1,070/m)	**3,876**		512	2,148	1,216
Cost ($000s)	**$6,697**		**$796**	**$3,242**	**$2,659**
($/m)	$1,565/m		$1,409/m	$1,367/m	$1,981/m
Production Stope Ramp					
Production Stope Ramp (m)	**1,470**		360	600	510
Operating Labor ($000s)	**747**		134	206	408
Maintenance Labor ($000s)	**259**		47	71	141
O&M ($906/m)	**1,332**		326	544	462
Cost ($000s)	**$2,338**		**$507**	**$820**	**$1,011**
($/m)	$1,590/m		$1,408/m	$1,367/m	$1,982/m
Ventilation Raises					
Ventilation Raises (m)	**886**		445	441	
Operating Labor ($000s)	**320**		169	151	
Maintenance Labor ($000s)	**112**		60	52	
O&M ($1,519/m)	**1,361**		691	670	
Cost ($000s)	**$1,793**		**$920**	**$873**	
($/m)	$2,023/m		$2,067/m	$1,980/m	
Total Cost	**$18,712**	**2,554**	**$6,164**	**$6,324**	**$3,670**

17.5.3 Process Capital

Process capital costs shown in Table 17.5.3.1 and include all costs to startup the mill circuit. A detailed discussion of process design and equipment requirements can be found in Section 14 of this report.

Table 17.5.3.1: Mill Capital Costs (US$000s)

Description	Total	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017
Direct Cost												
Crushing	**2,950**				738	2,213						
Grinding	**4,410**				1,103	3,308						
Leach & CIP	**4,310**				1,078	3,233						
ADR	**920**				230	690						
Reagents	**740**				185	555						
Services	**970**				243	728						
Subtotal	*14,300*				*3,575*	*10,725*						
Allowance (10%)	**1,430**				358	1,073						
Equipment Cost	**15,730**				**3,933**	**11,798**						
Allowances												
Site Prep	**786**				197	590						
Con., Steel, Build	**9,428**				2,337	7,071						
Mechanical Inst	**1,886**				472	1,415						
Piping	**3,143**				786	2,357						
Elect / Instrum	**6,286**				1,572	4,715						
Direct Costs	**37,259**				**9,315**	**27,944**						
Indirect Costs												
EPCM	**5,589**				1,397	4,192						
Contractor ID	**4,469**				1,117	3,3352						
First Fills	**157**				39	118						
Spare Parts	**629**				157	472						
Vendor Reps	**707**				177	530						
Freight	**1,043**				261	782						
Subtotal	*12,594*				*3,148*	*9,445*						
Direct & Indirect	**49,853**				**12,463**	**37,390**						
Contingency					**2,649**	**7,946**						
Process Total	**60,448**				**15,112**	**45,336**						

17.5.4 Infrastructure & Owner Cost Capital

Infrastructure and owner capital costs are shown in Table 17.5.4.1. Tailings dam capital accounts for the majority of the required infrastructure at Black Fox. A detailed discussion of the tailings dam design can be found in Section 17.4 of this report. Other infrastructure costs are minimal as the mine is located in a well-established mining district and is very close to roads and power interconnections.

Owner costs include a provision for on-going feasibility study related engineering and design work, additional exploration drilling associated with the updating of the resource estimate. Owner costs also include US$2.6million for mine closure cost and a US$9.4million salvage credit for the sale of the process mill at the end of the mine life.

Table 17.5.4.1: Infrastructure Capital Costs (US$000s)

Description	Total	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017
Infrastructure												
Tailings Dam	**12,000**					5,500	2,000	2,000	1,500	1,000		
Access Road	**50**			50								
Water Supply	**250**			250								
Power Distri.	**250**			250								
Communications	**50**			50								
O/P Haul Road	**200**			100	100							
Admin. Build	**250**			250								
Shop Building	**1,000**			1,000								
Site Fencing	**75**			75								
Subtotal	*14,125*			*2,025*	*100*	*5,500*	*2,000*	*2,000*	*1,500*	*1,000*		
Contingency	**2,119**			304	15	825	300	300	225	150		
Total	**16,244**			**2,329**	**115**	**6,325**	**2,300**	**2,300**	**1,725**	**1,150**		
Owner Costs												
Eng. & Design	**1,500**	500	500	250					250			
Geol. Drilling	**1,000**		**1,000**									
Corp Overhead	**0**											
Start-up - Comm	**0**											
Mine Closure	**2,600**											2,600
Mill Salvage	**(9,438)**											(9,438)
Subtotal	*(4,338)*	*500*	*1,500*	*250*					*250*			*(6,838)*
Contingency	**(651)**	**75**	**225**	**38**					**38**			(1,026)
Total	**(4,989)**	**575**	**1,725**	**288**					**288**			**(7,864)**

17.6 Operating Costs

LoM operating costs are summarized in Table 17.6.1. Details supporting this estimate are discussed in this subsection.

Table 17.6.1: Operating Cost Summary

Description	LoM Total (US$000s)	Unit Cost (US$/total-t)	Unit Cost (US$/ore-t)
Mining – OP	**$79,768**	$1.593/t	$23.72/t
Mining - UG	**$35,166**	-	31.75/t
Mine G&A	**$15,261**	-	$3.41/t
Toll Mill	**$42,504**	-	$32.57/t
Owner Mill	**$41,955**	-	$13.26/t
G&A	**$17,451**		$3.90/t
Total	**$250,817**		**$56.11/t***

*Weighted average over the LoM.

17.6.1 Open Pit Costs

A summary of the estimated mine operating cost are shown in Table 17.6.1.1. Mine operating costs that were produced by MDA, in August of 2006, were updated with the current fuel and tire prices and Montana Tunnels actual operating costs. Mine operating costs include US$8million of pre-stripping costs which will be capitalized during the mine pre-production period. The economic model (Exhibit 17.1) therefore reports LoM mine operating costs at US$71.8million.

Table 17.6.1.1: Open Pit Mine Operating Cost (US$000s)

Open Pit Mine Operating Costs	Total	2007	2008	2009	2010	2011	2012	2013	2014	2015
Opex										
Drilling	**10,405**	0	0	658	1,756	1,609	1,753	1,793	1,545	1,291
Blasting	**11,590**	0	0	734	2,289	1,795	2,160	2,221	1,402	989
Loading	**13,591**	0	0	2,025	2,209	2,551	2,045	2,012	1,486	1,263
Hauling	**22,181**	0	0	3,516	3,663	4,233	3,329	3,359	2,207	1,874
Support	**22,001**	0	0	3,099	3,959	3,959	3,891	3,891	1,825	1,308
Total Opex	**79,768**	**0**	**0**	**10,032**	**13,879**	**14,147**	**13,275**	**13,275**	**8,466**	**6,725**
US$/t-mined	**1.586**			1.124	1,479	1,760	1.620	1.575	1.962	2.215
Drilling										
Operating Labor	**4,637**	0	0	357	713	713	713	713	713	713
Maintenance Labor	**2,189**	0	0	141	346	346	346	306	351	351
Fuel & Lubricants	**1,403**			61	271	215	272	304	191	90
Consumables & Parts	**2,176**			100	425	335	421	470	290	136
Other Supplies	**0**			0	0	0	0	0	0	0
Total Drilling	**10,405**	**0**	**0**	**658**	**1,756**	**1,609**	**1,753**	**1,793**	**1,545**	**1,291**
US$/t-mined	**0.207**			0.074	0.187	0.200	0.214	0.213	0.358	0.425
Blasting										
Operating Labor	**2,209**	0	0	212	356	356	356	356	287	287
Maintenance Labor	**1,037**	0	0	84	173	173	173	153	141	141
Fuel & Lubricants	**0**			0	0	0	0	0	0	0
Consumables & Parts	**8,344**			438	1,760	1,267	1,632	1,712	974	561
Other Supplies	**0**			0	0	0	0	0	0	0
Total Blasting	**11,590**	**0**	**0**	**734**	**2,289**	**1,795**	**2,160**	**2,221**	**1,402**	**989**
US$/t-mined	**0.230**			0.082	0.244	0.223	0.264	0.264	0.325	0.326
Loading										
Operating Labor	**4,289**	0	0	456	639	639	639	639	639	639
Maintenance Labor	**2,014**	0	0	180	310	310	310	274	314	314
Fuel & Lubricants	**3,614**			685	627	793	545	546	265	154
Consumables & Parts	**3,673**			703	634	809	551	552	268	156
Other Supplies	**0**			0	0	0	0	0	0	0
Total Loading	**13,591**	**0**	**0**	**2,025**	**2,209**	**2,551**	**2,045**	**2,012**	**1,486**	**1,263**
US$/t-mined	**0.270**			0.227	0.235	0.317	0.250	0.239	0.344	0.416
Hauling										
Operating Labor	**7,080**	0	0	1,077	1,077	1,077	1,077	1,077	847	847
Maintenance Labor	**3,291**	0	0	426	523	523	523	463	417	417
Fuel & Lubricants	**6,871**			1,171	1,200	1,532	1,006	1,058	549	356
Consumables & Parts	**4,938**			841	862	1,101	723	760	395	256
Other Supplies	**0**			0	0	0	0	0	0	0
Total Hauling	**22,181**	**0**	**0**	**3,516**	**3,663**	**4,233**	**3,329**	**3,359**	**2,207**	**1,874**
US$/t-mined	**0.441**			0,394	0.390	0.527	0,407	0.399	0.512	0.617
Support										
Operating Labor	**7,469**	0	0	1,233	1,233	1,233	1,233	1,233	651	651
Maintenance Labor	**3,455**	0	0	488	599	599	599	530	320	320
Fuel & Lubricants	**6,546**			796	1,267	1,267	1,267	1,267	489	195
Consumables & Parts	**4,531**			582	861	867	861	861	365	141
Other Supplies	***0***			*0*	*0*	*0*	*0*	*0*	*0*	*0*
Total Support	**22,001**	**0**	**0**	**3,099**	**3,959**	**3,959**	**3,959**	**3,891**	**1,825**	**1,308**
US$/t-mined	**0.437**			0.347	0.422	0.493	0.484	0.462	0.423	0.431

Labor Costs

Labor costs are based upon an assumed 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 in accordance with a 42-hour week, eight hours per shift schedule.

A payroll loading provision of 25% is applied to all salaries and wages. In addition, a 5% allowance is applied to hourly wages to provide for training and unscheduled overtime premiums.

Personnel will reside in the neighboring communities and will be responsible for transportation from their domicile to the place of work. Apollo will not provide transportation, on a normal basis.

O&M Costs

Parameters used to estimate O&M costs are shown in Table 17.6.1.2.

Table 17.6.1.2: Open Pit Mine Equipment Estimated Costs

Item	Fuel 1/hr	Fuel $/hr	Lube, Oil & Filters	Tires $/hr	Under-Carriage	R&M Reserve	Special Wear Items	Total
Rotary Drill-251mm	50	$36.98	$12.00		$10.00	$40.00	$65.55	$164.53
Hyd. Drill-165mm	22	$16.27	$9.00		$5.00	$20.00	$22.67	$72.94
Hyd Drill-101mm	22	$16.27	$9.00		$5.00	$20.00	$13.60	$63.87
16m^3 Hyd. Shovel	180	$133.14	$25.00		$25.00	$90.00	$6.00	$296.14
11.5cm FEL	100	$73.97	$12.00	$14.67		$45.00	$1.50	$151.63
150t Truck	100	$73.97	$5.00	$19.20		$35.00	$1.00	$134.67
100t Truck	75	$55.48	$4.00	$10.20		$25.00	$4.00	$95.68
Dozer D11R	115	$85.06	$10.70		$28.80	$25.00	$3.00	$153.56
Dozer D9R	60	$44.38	$18.00		$17.00	$15.00	$1.50	$97.38
Grader 16H	35	$25.89	$5.04	$6.82		$30.00	$8.00	$69.25
4cm Mass Excavator	60	$44.38	$4.00		$10.00	$40.00	$0.20	$106.38
Lube Truck	10	$7.40	$0.50	$0.56		$3.50	$0.20	$12.16
Fuel Truck	8	$5.92	$0.20	$0.56		$3.00	$0.20	$9.88
Mechanics Truck	5	$3.70	$0.20	$0.56		$3.00	$0.10	$7.66
Pickup Tucks	2	$1.48	$0.10	$0.12		$2.50	$2.50	$4.30
Water Truck	30	$22.19	$3.00	$3.20		$10.00		$40.89
Light Plant	2	$1.48	$0.10	$0.06		$1.25	$1.00	$2.89
Sand/Stem Tuck	15	$11.10	$20..	$0.47		$5.00	$0.50	$19.56
Backhoe/Loader	15	$11.10	$3.00	$1.18		$8.00	$3.00	$23.78
Wheel Skidder	35	$25.89	$4.00	$6.00		$12.00	$2.00	$50.89
45 ton Crane	30	$22.19	$3.00	$1.18		$12.00	$0.20	$40.37
Skid Loader	12	$8.88	$1.00	$1.18		$2.00	$0.25	$13.26
Low Boy	10	$7.40	$1.00	$0.70		$3.50	$0.25	$12.85
Flatbed	10	$7.40	$1.00	$0.70		$3.00	$0.20	$12.35
Explosives Truck	15	$11.10	$2.00	$0.93		$5.00	$0.50	$19.23
Ambulance	5	$3.70	$0.60	$0.93		$3.00	$0.20	$8.73
Crew Van	5	$3.70	$0.50	$1.40		$3.00	$0.20	$8.80
Bus	15	$11.10	$1.50	$1.40		$5.00	$0.50	$19.50
Forklift	3	$2.22	$0.60	$0.50		$1.50	$0.10	$4.92
ATV	2	$1.48	$0.40	$0.50		$2.00	$0.10	$4.48

17.6.2 Underground Costs

Underground mining operating costs were determined using first principal engineering, known consumption rates from previous mining experience at the project, and local budgetary supply costs. Development and stoping costs are shown in the Table 17.6.2.1. Note that the analysis charges US$2.6million of G&A labor costs to stope production one year prior to actual mining. This charge covers mine design work associated with the pending underground operation.

Table 17.6.2.1: Underground Stope Production

Description		Total	2014	2015	2016	2017
Ore Mined (kt)		**1,108**		129	541	437
Stope Production						
	Operating Labor ($000s)	**14,662**	1,445	4,331	4,443	4,443
	Maintenance Labor ($000s)	**725**	1,178	1,533	1,533	1,533
	O&M	**9,725**		1,134	4,753	3,839
	Mine General	**2,038**		238	996	804
	Cost ($000s)	**$32,202**	**$2,623**	**$7,235**	**$11,725**	**$10,619**
	($/t)	$29.07/t	-	$56.02/t	$21.66/t	$24.29/t
Ore Haulage						
	Operating Labor ($000s)	**0**		0	0	0
	O&M	**1,728**		$201	$844	$682
	Cost ($000s)	**$1,728**		**$201**	**$844**	**$682**
	($/t)	$1.56/t		$1.56/t	$1.56/t	$1.56/t
Backfill						
	Operating Labor ($000s)	**0**		0	0	0
	Rock Fill	**394**		51	184	159
	Hydraulic Fill	**650**		84	304	263
	Cemented Sand Fill	**192**		25	90	77
	Cost ($000s)	**$1,236**		**$159**	**$578**	**$499**
	($/t)	$1.12/t		$1.23/t	$1.07/t	$1.14/t
	Total Cost	**$35,166**	**$2,623**	**$7,595**	**$13,147**	**$11,800**

Mine G&A costs are shown in Table 17.6.2.2. Mine G&A costs are specific overheads attributed to all mining operations.

Table 17.6.2.2: Mine G&A Costs (US$000)

Mine G&A Costs	Total	2009	2010	2011	2012	2013	2014	2015	2016	2017
Salaried Labor	**14,361**	1,346	1,559	1,484	1,484	1,334	1,740	1,855	1,780	1,780
Other Supplies	**900**	100	100	100	100	100	100	100	100	100
		0	0	0	0	0	0	0	0	0
Mine G&A	**15,261**	**1,446**	**1,659**	**1,584**	**1,584**	**1,434**	**1,840**	**1,955**	**1,880**	**1,880**
US$/t-mined	**3.41**	5.93	3.14	2.54	3.23	2.52	3.52	3.81	3.47	4.3

Labor Costs

The underground manpower requirements are shown in the following tables. As with open pit mining costs, labor costs are based upon an assumed 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 in accordance with a 42-hour week, eight hours per shift schedule.

A payroll loading provision of 25% is applied to all salaries and wages. In addition, a 5% allowance is applied to hourly wages to provide for training and unscheduled overtime premiums.

Personnel will reside in the neighboring communities and will be responsible for transportation from their domicile to the place of work. Apollo will not provide transportation, on a normal basis.

Table 17.6.2.3: Underground Salaried Staff

Position	2014	2015	2016	2017
General Manager	1	1	1	1
Human Resources Mgr	1	1	1	1
Environmental Mgr	1	1	1	1
Safety / Loss control Mgr	1	1	1	1
Controller	1	1	1	1
Purchasing Mgr	1	1	1	1
IT Mgr	1	1	1	1
Total	**7**	**7**	**7**	**7**

Table 17.6.2.4: Administrative Personnel - Hourly

Position	2014	2015	2016	2017
HR Assistant	1	1	1	1
Secretary	1	1	1	1
Accountant	1	1	1	1
Accounting Clerks	1	1	1	1
Purchasing Agent	2	2	2	2
Expeditor	1	1	1	0.5
Security Guard	4	4	3	3
Janitor	2	2	2	2
Total	**13**	**13**	**12**	**11.5**

Table 17.6.2.5: Mine G&A - Salaried

Position	2014	2015	2016	2017
Operations				
Underground Mine Manager	1	1	1	1
General Foreman	1	1	1	1
Foreman	1	1	1	1
Clerk	1	1	1	1
Trainer	2	2	2	2
Maintenance				
Electrical Foreman	1	1	1	1
Shift Foreman	4	4	4	4
Planning Engineer	1	1	1	1
Engineering & Geology				
Chief Engineer	1	1	1	1
Surveyors	1	2	2	2
Sr. Mining engineer	1	1	1	1
Mining Engineer		1	1	1
Environmental Engineer	1	1	1	1
Chief Geologist	1	1	1	1
Ore Control Geologist	1	1	1	1
Sampler	1	1	1	1
Safety Technician	1	1	1	1
Geotechnical Engineer	1	1	1	1
Total	**21**	**23**	**23**	**23**

The underground mine operations will have four crews that will all work eight hour shifts, seven days/wk, with a scheduled 360 days/yr.

Table 17.6.2.6: Underground Mining Operations - Hourly

Position	2014	2015	2016	2017
Operations				
Drill Operators	4	8	8	8
Bolter Operators	4	8	8	8
Bolter Helper	4	8	8	8
Blaster	0	4	4	4
Blaster Helper	0	4	4	4
LHD Operator – Ore	4	8	8	8
LHD Operator – Waste		4	4	4
Truck Driver	4	16	16	16
Backfill Crew	0	2	4	4
Grader Operator		4	4	4
Mine Utilities		2	2	2
Nipper	4	4	4	4
Maintenance				
Heavy Duty Mechanic	4	4	4	4
Mechanic	4	8	8	8
Fuel / Lube	4	4	4	4
Welder	4	4	4	4
PM Crew		2	2	2
Electricians	4	4	4	4
Total	**44**	**98**	**100**	**100**

17.6.3 Processing Costs

Black Fox will operate under two processing scenarios. During the last pre-production year and the first two production years, ore will be tolled milled allowing time for Apollo to construct its own on-site process plant. Once the process plant is completed, ore will be milled on-site. Toll milling costs are based upon contract terms with the Macassa process facility and are shown in Table 17.6.3.1. On-site owner operated process operating costs (Table 17.6.3.2) are based upon estimates prepared for Apollo and discussed in Section 14.

Table 17.6.3.1: Toll Mill Operating Costs (US$000s)

Description		Total	2009	2010	2011
Crushing		914	158	378	378
Transportation		9,761	1,683	4,039	4,039
Processing		31,829	5,488	13,171	13,171
Total Toll Mill		**42,504**	**7,329**	**17,588**	**17,588**
	Total US$/t-milled	32.57	32.57	32.57	32.57

Table 17.6.3.2: Owner Mill Operating Costs (US$000s)

Description	Total	2009	2010	2011	2012	2013	2014	2015	2016	2017
Labor										
Salaried	**5,712**	134	134	262	576	921	921	921	921	921
Hourly	**4,504**				534	534	534	534	1,185	1,185
Labor	**10,216**	**134**	**134**	**262**	**1,110**	**1,455**	**1,455**	**1,455**	**2,106**	**2,106**
US$/t-milled	3.228				2.06	2.70	2.70	2.70	3.90	4.53
Operating Supplies										
Primary Crusher Liners	**562**				96	96	96	96	96	83
Pebble Crusher Liners	**250**				43	43	43	43	43	37
SAG Mill Liners	**2,279**				389	389	389	389	389	335
Ball Mill Liners	**821**				140	140	140	140	140	121
Tower Mill Liners	**49**				8	8	8	8	8	7
Grinding Balls, 100mm	**873**				149	149	149	149	149	128
Grinding Balls, 38mm	**1,188**				203	203	203	203	203	175
Grinding Balls, 25mm	**249**				43	43	43	43	43	37
Calcium Chloride	**155**				26	26	26	26	26	23
Carbon	**295**				50	50	50	50	50	43
Caustic Soda	**1,012**				173	173	173	173	173	149
Copper Sulphate	**8**				1	1	1	1	1	1
Ferric Sulfate	**39**				7	7	7	7	7	6
Flocculant Percol 351	**395**				67	67	67	67	67	58
Hydrogen Peroxide	**210**				36	36	36	36	36	31
Lead Nitrate	**20**				3	3	3	3	3	3
Lime	**26**				4	4	4	4	4	4
Quick Lime	**666**				114	114	114	114	114	98
Nitric Acid	**385**				66	66	66	66	66	57
Sodium Cyanide	**3,683**				628	628	628	628	628	541
Miscellaneous	**253**				43	43	43	43	43	37
Vehicles	**475**				81	81	81	81	81	70
Assay Lab Supplies	**443**				76	76	76	76	76	65
Operating Supplies	**14,336**				**2,446**	**2,446**	**2,446**	**2,446**	**2,446**	**2,107**
US$/t-milled	4.53				4.53	4.53	4.53	4.53	4.53	4.53
Repair Parts										
Primary Crusher Liners	**4,463**				761	761	761	761	761	656
Repair Parts	**4,463**				**761**	**761**	**761**	**761**	**761**	**656**
US$/t-milled	1.41				1.41	1.41	1.41	1.41	1.41	1.41
Power										
Energy Consumption	-				24.800	24.800	24.800	24.800	24.800	21.359
Rate	-				0.071	0.071	0.071	0.071	0.071	0.071
Demand	-				2,831	2,831	2,831	2,831	2,831	2,438
Rate	-				13.010	13.010	13.010	13.010	13.010	13.010
Power	**12,940**				**2,208**	**2,208**	**2,208**	**2,208**	**2,208**	**1,901**
US$/t-milled	4.09				4.09	4.09	4.09	4.09	4.09	4.09
Total Owner Mill	**41,956**	**134**	**134**	**262**	**6,525**	**6,870**	**6,870**	**6,870**	**7,521**	**6,770**
Total US$/t-milled	$13.26				12.08	12.72	12.72	12.72	13.93	14.56

17.6.4 G&A Costs

LoM G&A costs have been estimated based upon similar operations and are shown in Table 17.6.4.1.

Table 17.6.4.1: G&A Costs

Description		LoM Total (US$000s)
Labor		
Salaried		5,569
Hourly		5,130
	subtotal Labor	**10,699**
Operating Supplies		
Supplies		540
Postage Courier		180
Communication		270
Travel		270
Outside Services		360
Environmental Monitoring		324
Community Relations		500
Insurance		1,125
Recruiting & Relocation		300
Safety Supplies		324
Security Supplies		54
Vehicle & Bus Operations		871
Customs		270
Computer Equipment & Software		270
Training		500
Power		216
Water System		216
Road Maintenance		162
	Subtotal Operating Supplies	*6,752*
	Total G&A	**17,451** US$3.90/t

17.7 Markets

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 July 2007 is US$665/oz. Markets for doré are readily available.

17.8 Contracts

Black Fox has or has modeled the following contracts in this analysis.

- Toll milling at Kirkland Lake's Macassa mill. The terms of this contract, to process approximately 1Mt of ore over roughly a 2.5 year period, are reasonable and within industry norms. Costs associated with this contract are modeled in the economic analysis;
- Refining of doré. Black Fox does not have a contract in place for the refining of its doré as it is too early in the development of the project to negotiate such terms. However, the economic model assumes current-day typical refinery terms and conditions in its analysis; and
- There are no contract mining, transportation, hedging, forward selling or other agreements/contracts related to Black Fox.

17.9 Environmental Considerations

Section 17.9 is excerpted from Apollo Gold Corporation Black Fox Project, Project Description for Small Pit & Mill Operation –Update-, Prepared for Ministry of Northern Development and Mines on behalf of Apollo Gold Corporation by AMEC Earth & Environmental, April 16, 2007 and has been standardized to this report.

17.9.1 Regulatory Considerations

"A number of environmental issues will be considered for the permitting and approvals process related to the Black Fox Project. These environmental issues include: the removal of all existing site infrastructure and the construction of new infrastructure, effects to terrestrial habitat, potential effects to fisheries habitat, groundwater and area surface waters, as well as air quality and related noise issues associated with open pit mining activities.

As the Project continues to move forward, further contact with various government agencies will be established and a formal public and stakeholder consultation process initiated. There are a number of permitting considerations that must be addressed in order for the project to advance. These considerations include, but are not limited to, the following:

- *Amendments to existing and/or application for new Permits to Take Water (MOE);*
- *Amendment to existing C of A for Air (MOE);*
- *Application for a new C of A for Industrial Sewage Works (MOE);*
- *Potential Fisheries Act authorization (i.e., Letter of Advice) (Department of Fisheries and Oceans);*
- *Filing of Mine Closure Plan with financial assurance (MNDM); and*
- *Authorizations/easements for site access requirements (MTO).*

It is planned to commence with applicable provincial permitting and approval application submissions as early as mid May 2007, with receipt of approvals anticipated for the end of 2007 or early 2008. Site development would commence shortly thereafter.

17.9.2 Mine Development Considerations

Since Apollo purchased a 100% interest in the Glimmer Mine in 2002, exploration initiatives have supported the preparation of a positive feasibility study for open pit development, which is to be supplemented with an underground mine operation. For this mining scenario, Apollo has undertaken various environmental baseline, geochemical, geotechnical, hydrogeologic, metallurgical and other studies in support of the Black Fox Project. The Black Fox Project will encompass a number of major modifications to the site due to the development of an open pit mining operation and proposed underground mine. The major aspects associated with the proposed Project permitting would include the following:

- *Removal of existing site infrastructure and the construction of new infrastructure to facilitate the open pit development;*
- *Effects to terrestrial habitat, due to the development of new overburden and waste rock piles;*
- *Potential effects to fisheries habitat, due to the placement of overburden and/or waste rock stockpiles adjacent to, or near, local watercourses;*

- *Potential effects to groundwater flow paths, associated with a potentially increased rate of dewatering of mine workings;*
- *Potential effects to area surface waters (particularly with respect to Froome Lake), due to a potentially increased mine dewatering rate;*
- *Potential effects to area surface waters resulting from site drainage, and treated water discharge from the water treatment plant;*
- *Air quality and related noise issues associated with open pit mining activities;*
- *A possible crossing of Highway 101 to stockpile organic material north of the highway; and*
- *Public and stakeholder consultation.*

Activities associated with the development of the proposed open pit and underground mine, which will require the removal of existing infrastructure and the construction of new infrastructure, will be subject to permit authorizations from various regulatory agencies. In the event that the proposed mine development would result in significant effects on fish habitat, a federal authorization under the Fisheries Act would be required and the Project could be subject to a comprehensive level environmental assessment under the Canadian Environmental Assessment Act. However, for the proposed open pit and underground mine scenario, and the proposed configurations of the waste rock and overburden stockpiles, potential impacts on local fish habitat have been mitigated and/or eliminated. The only other potential for federal involvement could be related to the construction of any water intake and/or discharge structures, which could require a Letter of Advice from the Department of Fisheries and Oceans.

In developing the small open pit and further advancing underground development, surface runoff and groundwater inflows into the mine are expected to increase. Hydrogeological and hydrological investigation programs have been undertaken to characterize the effects of the potential increase of such inflows into the proposed mine development on the surrounding environment, including Froome Lake. These investigations indicate that the potential for significant effects, as a result of the additional inflows and/or diversions, can be mitigatived to reduce such effects.

Based on a review of Ministry of Environment publications relating to sound level limits, it is assumed that the area surrounding the Black Fox Project would be considered as a Class 3 Area (Rural). A comprehensive noise assessment of the site will be completed, as required, in support of an application for a C of A (Air) for all mine related equipment, such as ventilation exhaust fans, emergency generators and maintenance areas, as well as emissions from the mill operation. Air emissions are anticipated to be minimal at the site and will be primarily limited to fugitive particulate emissions from material handling equipment (loaders, trucks, conveyors/crushers) and products of combustion from diesel equipment and propane heaters.

Special requirements for pit blasting may be required due to the proximity of the pit to Highway 101 and will need to be discussed with the Ontario Ministry of Transportation (MTO). Some settlement of the highway could also occur as a result of the drawdown of the water table. The potential for settlement is to be evaluated during the feasibility level design stage. Public and stakeholder consultation is required for all new and existing proposed mine expansion projects. The public consultation program will entail an open house for the general public, continued consultation with the First Nations, and possibly meetings with municipal representatives from Matheson and/or local residents, recreational groups and small business owners.

The First Nation (FN) community having an interest in this mine development project is the Wahgoshig First Nation. The community of approximately 250 registered members is located 25 km northeast of the mine site. A series of meetings were held by Apollo with the Wahgoshig First Nation throughout 2006, which resulted in the signing of a Memorandum of Understanding (MOU) in January 2007 between both parties. The key aspects of the MOU include provisions for training and ongoing communication. The MOU also outlines an agenda and process for negotiating an Impact Benefit Agreement (IBA), which will include such topics as employment, training, business opportunities and financial compensation when the Black Fox Project moves from an exploration phase to a production phase.

17.9.3 Mine Reclamation and Site Closure

Upon the cessation of mining activities, Apollo will reclaim the Black Fox Project site as required under Section VII of the Mining Act. Opportunities to progressively reclaim the site will be exploited and progressive rehabilitation efforts maximized over the life of the operation where possible. After reclamation and closure, there will be a number of facilities requiring monitoring and maintenance. Remaining site facilities will include: an open pit; capped mine shaft, raise(s) and portal; as well as reclaimed tailings and waste rock and overburden stockpiles.

At the conclusion of mining, all openings to surface will be sealed with reinforced concrete caps designed in accordance with Schedule 1 of Ontario Reg. 240/00, the Mine Rehabilitation Code of Ontario. Crown pillars will be assessed for long term stability and rehabilitation measures implemented accordingly, with the objective of ensuring adequate site safety.

Restoration of the open pit will involve flooding (i.e., creation of a pit lake) in conjunction with backfilling of the pit with waste rock (to prevent sulphide oxidation). The shallow groundwater regime in the area, along with benign country rock, strongly suggests that flooding of the pit is a practical closure option. It is expected that a one-time addition of lime to the pit water will be required to neutralize any acidity of (partly oxidized) waste rock.

All non-closure related infrastructure will be removed from the site. All site buildings will be demolished and removed to an off-site licensed landfill facility as required. All building foundations will be demolished, and subsequently covered with overburden and seeded with an appropriate vegetative mixture. All non-essential site distribution services, including electrical power, water, tailings, sewage and gas lines, will be removed from site. Below ground services will be decommissioned and left buried. All mobile and fixed equipment will be removed from site. Any remaining inventories of chemicals or petroleum products will be returned to the appropriate vendors and hazardous wastes will be disposed of using appropriately licensed waste haulers and contractors.

The tailings facility, and "clean" waste rock and overburden stockpiles, will be designed and constructed to ensure long-term physical stability. Revegetation of all "clean" waste rock and overburden piles will be carried out with the objective of creating a self-sustaining vegetative cover. It is planned to move the "dirty" waste rock into the open pit for final disposal, where it will be submerged to minimize oxidation and metal leaching. Closure of the tailings impoundment will consist of either a soil cover or a water cover. Ongoing assessment programs will be implemented as part of mine operations to continually monitor chemical stability.

All site related drainage channels or water management structures created as a result of mining operations or closure initiatives will be removed or stabilized. Design criteria for the design of remaining water structures such as drainage ways or spillways would be developed in *accordance with current engineering standards. Where appropriate, suitable erosion protection will be designed and installed.*

It is anticipated that all reclamation activities would be completed within a period of five years upon the cessation of site-related mining activity. Monitoring would be implemented to monitor the effectiveness of closure measures.

The treatment of excess tailings pond water would continue until the cyanide and arsenic levels become sufficiently low for discharge of site waters directly to the environment."

17.10 Taxes and Royalties

Income tax was not considered in this report. VAT and other production taxes were also 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.

17.11 Economic Analysis

The technical-economic results summarized in this section are based upon work performed by Apollo's engineers and consultants and has been prepared on an annual basis. The economic model was developed by SRK.

17.11.1 Model Inputs

The economic model, presented in Exhibit 17.1, 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 17.11.1.1

Table 17.11.1.1: Technical Economic Model Parameters

Model Parameter	Technical Input
General Assumptions	
Pre-Production Period	3 years
Mine Life	8 years
Operating Days per year	360 days/yr
Production Rate (avg.)	1,500t/d
Market	
Discount Rate	4%
Gold Price	US$525.00/oz
Refinery Charges	
Refining	US$2.50/oz
Transportation, Insurance & Assay	US$1,000/shipment
Royalty	
Private Royalty	none

A three-year pre-production rate is assumed to allow for pre-stripping and mine development. The mine will have an estimated life of eight years given the resources described in this report and the assumed 540kt/yr production rate. The model assumes that gold will be toll milled in Year -01 through Year 02, with owner mill operation beginning in Year 03.

Revenue from gold sales are based upon a market price of $525/oz. Gold treatment and refining charges are at $2.50/oz. Treatment, refining and transportation costs are charged against gross revenues. Transportation costs are as shown in the table and are calculated based upon 12 shipments per year.

17.11.2 LoM Plan and Economics

The SRK LoM plan and economics are based on the following:

- Probable reserves, no resources are included;
- A mine life of eight years, at a designed rate of 540kt/yr;
- An overall average metallurgical recovery rate of 96% Au, over the LoM;
- A cash operating cost of $50.69/t-milled, $236/oz-Au; and
- Total capital costs of $121.3million being comprised of $26.7million for capitalized development and $94.5million for mine equipment. Mine closure cost is $2.6million and there is a mill salvage value of $9.4million in Year 09.

The base case economic analysis results, shown in Table 17.11.2.1, indicate a pre-tax net present value of $103.5million at a 4% discount rate with an IRR of 33%.

Table 17.11.2.1: Technical Economic Results ($000s)

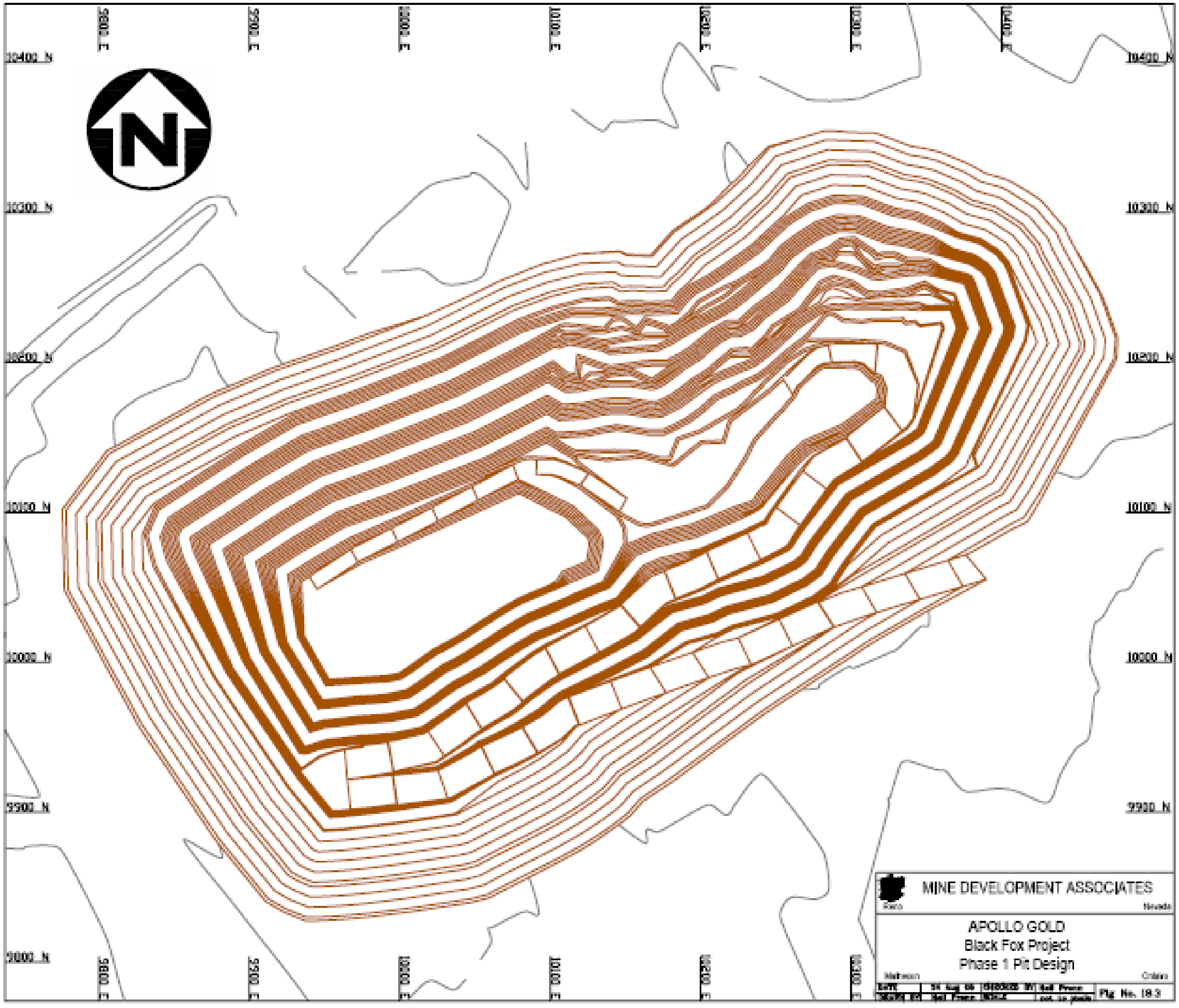
Description		Technical Input or Result
Ore		
Open Pit		
Waste		46,940kt
Ore		3,362kt
Total		**50,302kt**
s/r		14:1
Grade		5.78g/t-Au
Contained Gold		625koz
Underground		
Total Development		11,975m
Ore		1,108kt
Grade		10.59g/t-Au
Contained Gold		377koz
Mill		
Ore Treated		
Toll Mill		1,305kt
Owner Mill		3,165kt
Total		**4,470kt**
Ore Grade		
Toll Mill		5.43g/t-Au
Owner Mill		7.61g/t-Au
Total		**6.97g/t-Au**
Contained Gold		
Toll Mill		228koz
Owner Mill		774koz
Total		**1,002koz**
Recovered Gold		
Toll Mill		219koz
Owner Mill		743koz
Total		**962koz**
Revenue ($000s)		
Gross Revenue		**$504,168**
Refining & Transportation Charges		$2,509
Net Smelter Return		**$501,659**
Royalty		$0
Gross Income From Mining		**$501,659**
Realized Price (Gold)	*US$522.39/oz-Au*	
Operating Cost ($000s)		
Open Pit Mine		($71,742)
Underground Mine		($35,166)
Mine G&A		($15,261)
Toll Mill		($42,504)
Owner Mill		($41,955)
G&A		($17,451)
Operating Costs		**($224,079)**
	US$233.34/oz-Au	
	US$50.13/t-milled	
Cash Operating Margin		**$277,580**
	US$289.05/oz-Au	
	US$62.10/t-milled	
Capital Cost		
Equipment		($94,527)
Development (Capitalized)		($26,738)
Total Capital		**($121,266)**
Cash Flow		**$156,314**
(NPV4%)		**$103,524**
IRR		**33%**

17.11.3 Sensitivity

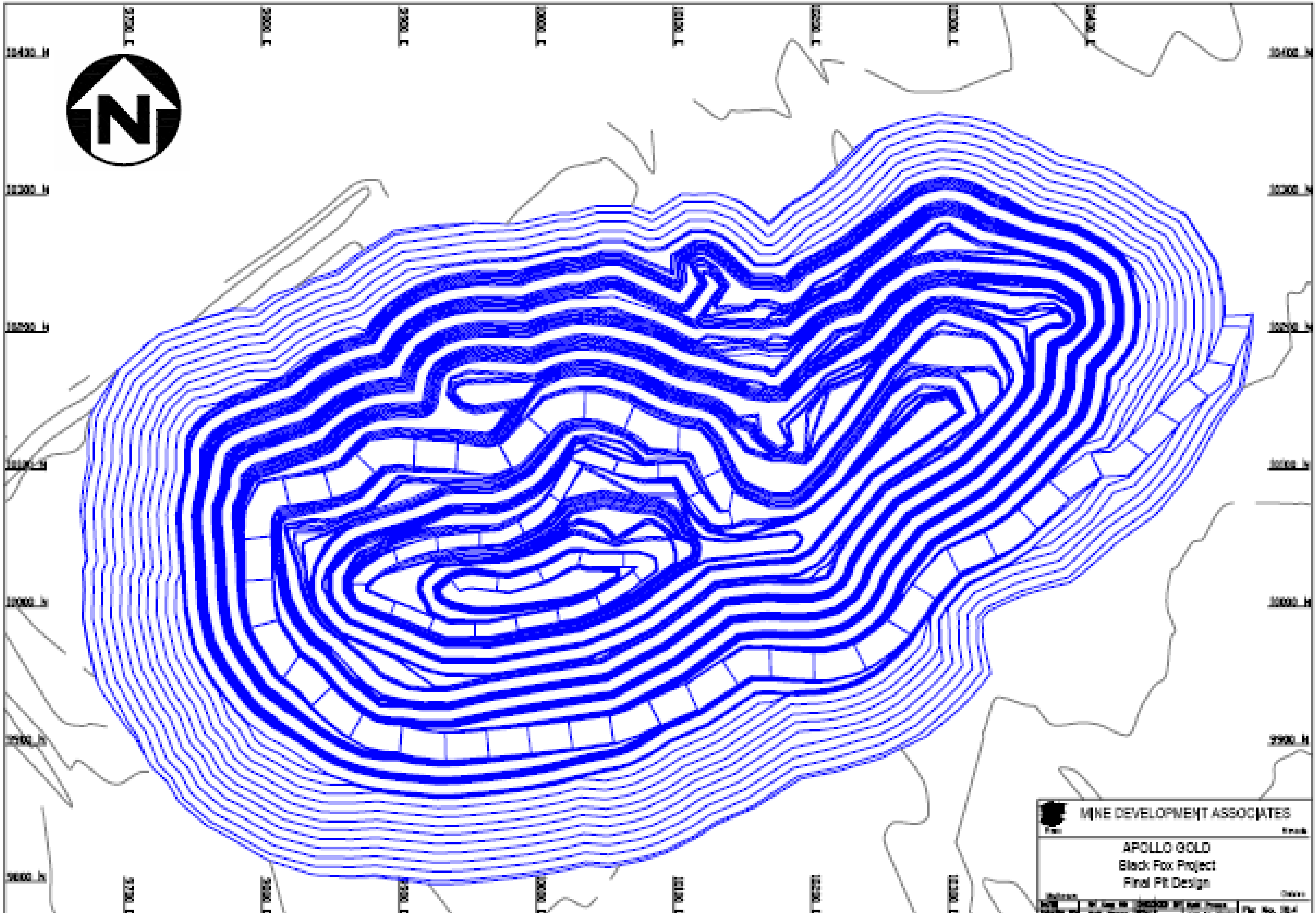
Sensitivity analysis for key economic parameters are shown in Table 17.11.3.1. This analysis suggests that the project is most sensitive to market price. Operating costs are slightly more sensitive due to toll milling of ore in early years and time value of money effect of delaying the on-site mill in Year 03.

Table 17.11.3.1: Project Sensitivity ($NPV_{4\%}$, $000's)

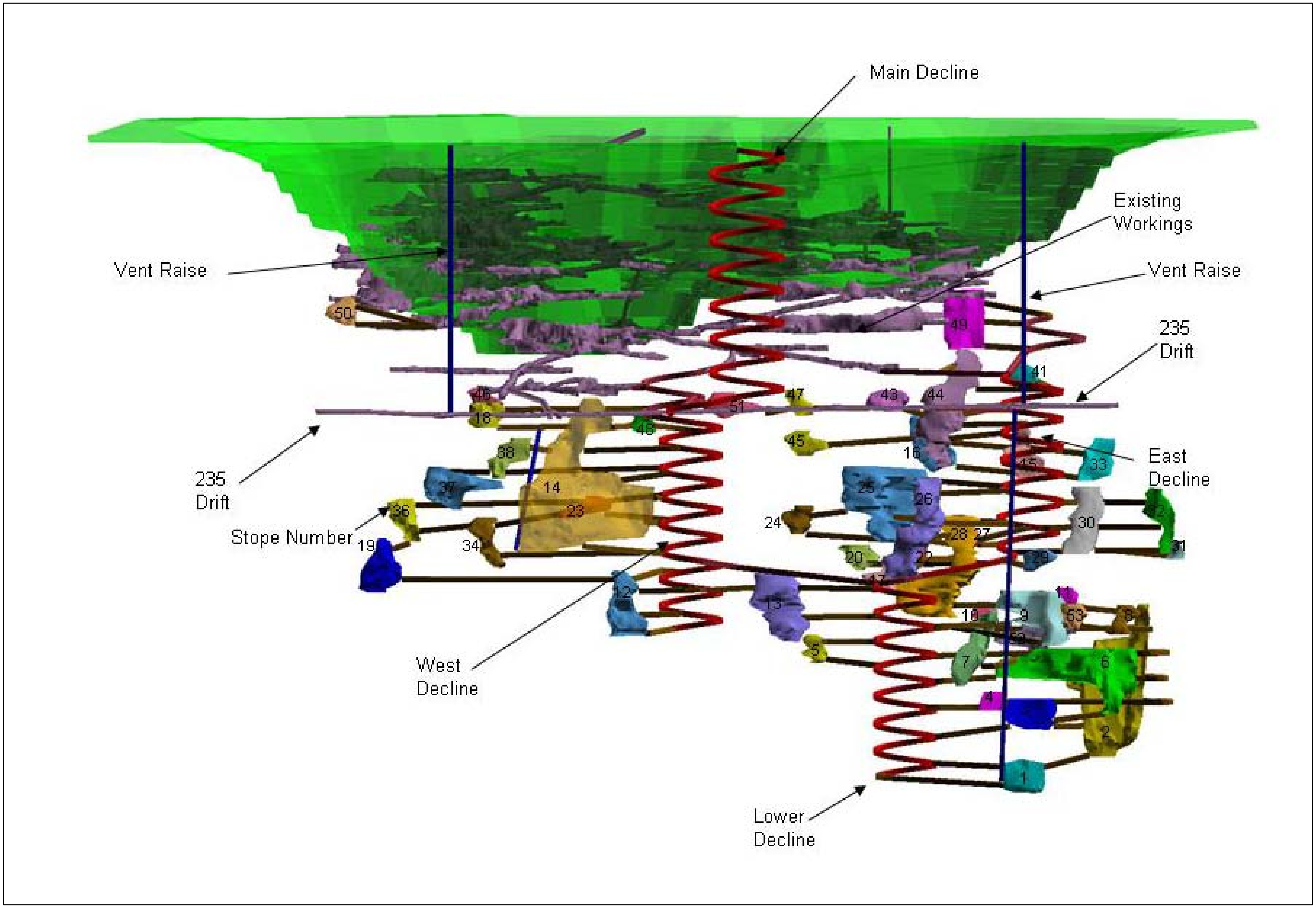
Description	-10%	-5%	Base Case	+5%	+10%
Gold Price	$64,491	$84,376	**$103,524**	$122,673	$142,558
Operating Costs	$122,177	$112,851	**$103,524**	$94,198	$84,872
Capital Costs	$112,078	$107,801	**$103,524**	$99,248	$94,971



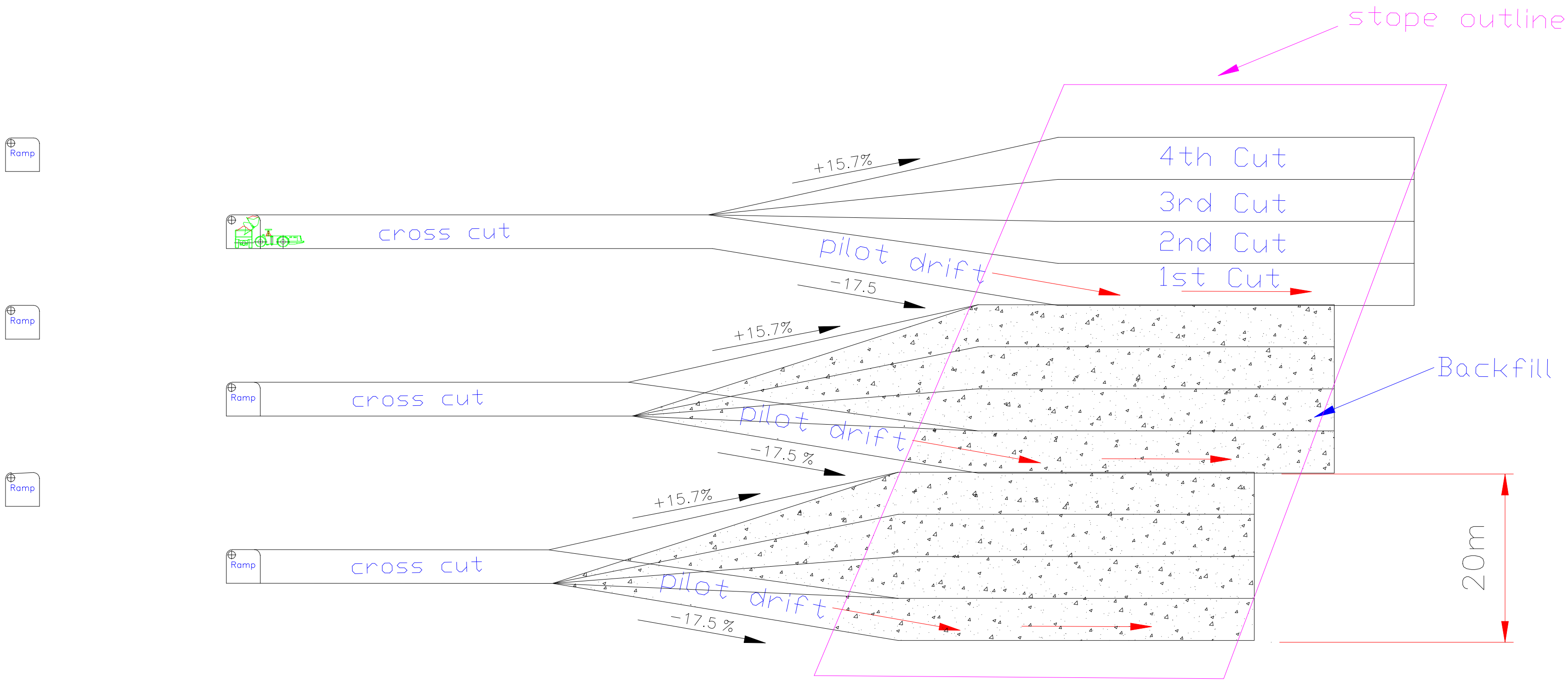
SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	**Black Fox Phase 1 Pit Design**
SRK Job No.: 144418	Source: Mine Development Associates	Date: 08-07-07 / Approved: DKY / Figure: 17-1
File Name: Figure 17-1.doc		



SRK Consulting Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	**Black Fox Final Pit Design**
SRK Job No.: 144418	**Source: Mine Development Associates**	
File Name: Figure 17-2.doc		Date: 08-07-07 / Approved: DKY / Figure: 17-2



SRK Consulting — Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	**Long Section Underground Stope Shapes**
SRK Job No.: 144418		
File Name: Figure 17-3.doc	**Source: Apollo Gold Corporation**	Date: 08-07-07 / Approved: DKY / Figure: 17-3



TITLE

STOPE SECTION

BY:	DATE:	SCALE	DWG NO.
JC	JULY 07	NA	stope_xsection

SRK Consulting — Engineers and Scientists	Black Fox, Timmins, Ontario, Canada	Underground Mining Typical Stope Section
SRK Job No.: 144418	Source: Apollo Gold Corporation	Date: 08-07-07 / Approved: DK"Y / Figure: 17-4
File Name: Figure 17-4.doc		

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
PRE-TAX CASH FLOW

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
PRODUCTION SUMMARY																	
OPEN PIT																	
Waste	kt	46,940	0	0	8,681	8,856	7,413	7,689	7,859	3,792	2,652	0	0	0	0	0	0
Ore	kt	3,362	0	0	244	528	624	490	569	523	384	0	0	0	0	0	0
Total	kt	50,302	0	0	8,925	9,384	8,037	8,178	8,427	4,315	3,036	0	0	0	0	0	0
s/r	*wst:ore*	*13.96*	*0.00*	*0.00*	*35.56*	*16.76*	*11.88*	*15.70*	*13.82*	*7.25*	*6.90*	*0.00*	*0.00*	*0.00*	*0.00*	*0.00*	*0.00*
Grade	gpt	5.78	0.00	0.00	7.43	5.03	4.92	7.33	4.98	6.71	5.10	0.00	0.00	0.00	0.00	0.00	0.00
Contained Gold	koz	625	0	0	58	85	99	115	91	113	63	0	0	0	0	0	0
UNDERGROUND																	
Total Development	m	11,975	0	0	0	0	0	0	0	1,680	4,060	4,382	1,852	0	0	0	0
Ore	kt	1,108	0	0	0	0	0	0	0	0	129	541	437	0	0	0	0
Grade	gpt	10.59	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	10.29	11.54	9.49	0.00	0.00	0.00	0.00
Contained Gold	koz	377	0	0	0	0	0	0	0	0	43	201	133	0	0	0	0
MILL																	
Ore Treated																	
Toll Mill	kt	1,305	0	0	225	540	540	0	0	0	0	0	0	0	0	0	0
Owner Mill	kt	3,165	0	0	0	0	0	540	540	540	540	540	465	0	0	0	0
Total	kt	4,470	0	0	225	540	540	540	540	540	540	540	465	0	0	0	0
Ore Grade																	
Toll Mill	gpt	5.43	0.00	0.00	7.43	5.11	4.92	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Owner Mill	gpt	7.61	0.00	0.00	0.00	0.00	0.00	6.95	5.11	6.53	6.42	11.30	9.60	0.00	0.00	0.00	0.00
Total	gpt	6.97	0.00	0.00	7.43	5.11	4.92	6.95	5.11	6.53	6.42	11.30	9.60	0.00	0.00	0.00	0.00
Contained Gold																	
Toll Mill	koz	228	0	0	54	89	85	0	0	0	0	0	0	0	0	0	0
Owner Mill	koz	774	0	0	0	0	0	121	89	113	111	196	144	0	0	0	0
Total	koz	1,002	0	0	54	89	85	121	89	113	111	196	144	0	0	0	0
Recovered Gold																	
Toll Mill	koz	219	0	0	52	85	82	0	0	0	0	0	0	0	0	0	0
Owner Mill	koz	743	0	0	0	0	0	116	85	109	107	188	138	0	0	0	0
Total	koz	962	0	0	52	85	82	116	85	109	107	188	138	0	0	0	0
On-Site Tailings																	
Original Tailings Dam	000m3	2,398	0	0	0	0	0	409	409	409	409	409	352	0	0	0	0
Expansion Dam	000m3	47	0	0	0	0	0	0	0	0	0	0	47	0	0	0	0
Total	000m3	2,445	0	0	0	0	0	409	409	409	409	409	400	0	0	0	0
GROSS INCOME FROM MINING																	
Gross Revenue																	
Payable Gold	99.85%	960	0	0	52	85	82	116	85	109	107	188	138	0	0	0	0
Gold (Au)	US$/oz	$525	$525	$525	$525	$525	$525	$525	$525	$525	$525	$525	$525	$525	$525	$525	$525
Gross Revenue	US$000	504,168	0	0	27,039	44,644	43,005	60,694	44,679	57,009	56,099	98,765	72,234	0	0	0	0
Refinery																	
Refining	$2.50	2,401	0	0	129	213	205	289	213	271	267	470	344	0	0	0	0
Trans., Ins. & Assay	$1,000	108	0	0	12	12	12	12	12	12	12	12	12	0	0	0	0
Refinery	US$000	2,509	0	0	141	225	217	301	225	283	279	482	356	0	0	0	0
NSR	US$000	501,659	0	0	26,899	44,420	42,788	60,393	44,454	56,726	55,820	98,283	71,878	0	0	0	0
Royalty																	
Royalty 1	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Royalty 2	US$000	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Gross Income From Mining	US$000	501,659	0	0	26,899	44,420	42,788	60,393	44,454	56,726	55,820	98,283	71,878	0	0	0	0
Realized Price	US$/oz-Au	$522.39	$0.00	$0.00	$522.27	$522.36	$522.35	$522.40	$522.36	$522.39	$522.39	$522.44	$522.41	$0.00	$0.00	$0.00	$0.00

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
PRE-TAX CASH FLOW

			preproduction			production											
Description	**Units**	**LoM Total**	**2007 -03**	**2008 -02**	**2009 -01**	**2010 01**	**2011 02**	**2012 03**	**2013 04**	**2014 05**	**2015 06**	**2016 07**	**2017 08**	**2018 09**	**2019 10**	**2020 11**	**2021 12**
CASH FLOW																	
Gross Income From Mining	**US$000**	**501,659**	**0**	**0**	**26,899**	**44,420**	**42,788**	**60,393**	**44,454**	**56,726**	**55,820**	**98,283**	**71,878**	**0**	**0**	**0**	**0**
Operating Costs																	
Open Pit Mine	**US$000**	**71,742**	0	0	2,006	13,876	14,147	13,247	13,275	8,466	6,725	0	0	0	0	0	0
Underground Mine	**US$000**	**35,166**	0	0	0	0	0	0	0	2,623	7,596	13,147	11,800	0	0	0	0
Mine G&A	**US$000**	**15,261**	0	0	1,446	1,659	1,584	1,584	1,434	1,840	1,955	1,880	1,880	0	0	0	0
Toll Mill	**US$000**	**42,504**	0	0	7,328	17,588	17,588	0	0	0	0	0	0	0	0	0	0
Owner Mill	**US$000**	**41,955**	0	0	134	134	262	6,525	6,870	6,870	6,870	7,521	6,770	0	0	0	0
G&A	**US$000**	**17,451**	0	0	2,031	1,939	1,939	1,939	1,939	1,939	1,939	1,904	1,884	0	0	0	0
Operating Costs	**US$000**	**224,079**	**0**	**0**	**12,946**	**35,195**	**35,520**	**23,294**	**23,517**	**21,737**	**25,084**	**24,451**	**22,334**	**0**	**0**	**0**	**0**
per oz-Au	(US$/oz)	233.34	0.00	0.00	251.35	413.88	433.62	201.50	276.34	200.18	234.75	129.98	162.32	0.00	0.00	0.00	0.00
per milled ton	(US$/t)	50.13	0.00	0.00	57.54	65.18	65.78	43.14	43.55	40.25	46.45	45.28	48.02	0.00	0.00	0.00	0.00
Total Cash Costs																	
Refining	**US$000**	**2,509**	0	0	141	225	217	301	225	283	279	482	356	0	0	0	0
Royalty	**US$000**	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Operating	**US$000**	**224,079**	0	0	12,946	35,195	35,520	23,294	23,517	21,737	25,084	24,451	22,334	0	0	0	0
Total Cash Cost	**US$000**	**226,588**	**0**	**0**	**13,086**	**35,420**	**35,737**	**23,595**	**23,742**	**22,021**	**25,364**	**24,934**	**22,690**	**0**	**0**	**0**	**0**
per oz-Au	(US$/oz)	235.95	0.00	0.00	254.09	416.52	436.27	204.10	278.98	202.79	237.37	132.54	164.91	0.00	0.00	0.00	0.00
per milled ton	(US$/t)	50.69	0.00	0.00	58.16	65.59	66.18	43.70	43.97	40.78	46.97	46.17	48.79	0.00	0.00	0.00	0.00
Cash Operating Margin	**US$000**	**277,580**	**0**	**0**	**13,953**	**9,225**	**7,268**	**37,098**	**20,937**	**34,988**	**30,735**	**73,831**	**49,544**	**0**	**0**	**0**	**0**
per oz-Au	*(US$/oz)*	*289.05*	*0.00*	*0.00*	*270.91*	*108.48*	*88.73*	*320.90*	*246.02*	*322.21*	*287.63*	*392.46*	*360.09*	*0.00*	*0.00*	*0.00*	*0.00*
per milled ton	*(US$/t)*	*62.10*	*0.00*	*0.00*	*62.01*	*17.08*	*13.46*	*68.70*	*38.77*	*64.79*	*56.92*	*136.72*	*106.53*	*0.00*	*0.00*	*0.00*	*0.00*
Capital Costs																	
Equipment																	
Open Pit Mine	**US$000**	**7,811**	0	0	3,045	262	952	1,751	1,757	44	0	0	0	0	0	0	0
Underground Mine	**US$000**	**15,013**	0	0	0	0	0	0	0	9,200	5,813	0	0	0	0	0	0
Process Plant	**US$000**	**60,448**	0	0	0	15,112	45,336	0	0	0	0	0	0	0	0	0	0
Infrastructure	**US$000**	**16,244**	0	0	2,329	115	6,325	2,300	2,300	1,725	1,150	0	0	0	0	0	0
Owner Costs	**US$000**	**(4,989)**	575	1,725	288	0	0	0	0	288	0	0	1,150	(9,014)	0	0	0
Equipment	**US$000**	**94,527**	**575**	**1,725**	**5,662**	**15,489**	**52,612**	**4,051**	**4,057**	**11,257**	**6,963**	**0**	**1,150**	**(9,014)**	**0**	**0**	**0**
Capitalized																	
Pre-Stripping	**US$000**	**8,026**			8,026												
Mine Development	**US$000**	**18,712**	0	0	0	0	0	0	0	2,554	6,164	6,324	3,670	0	0	0	0
Capitalized	**US$000**	**26,738**	**0**	**0**	**8,026**	**0**	**0**	**0**	**0**	**2,554**	**6,164**	**6,324**	**3,670**	**0**	**0**	**0**	**0**
Working Capital																	
Assets																	
Cash	**US$000**		0	0	252	684	691	453	457	423	488	475	434	0	0	0	0
A/R	**US$000**		0	0	2,242	3,702	3,566	5,033	3,705	4,727	4,652	8,190	5,990	0	0	0	0
Ore Inventory	**US$000**		0	0	165	1,140	1,163	1,089	1,091	696	553	0	0	0	0	0	0
Au & Ag Produced	**US$000**		0	0	248	675	681	447	451	417	481	469	428	0	0	0	0
Total Assets	**US$000**		**0**	**0**	**2,906**	**6,201**	**6,100**	**7,021**	**5,704**	**6,263**	**6,173**	**9,135**	**6,852**	**0**	**0**	**0**	**0**
Liabilities																	
A/P	**US$000**		0	0	252	684	691	453	457	423	488	475	434	0	0	0	0
other	**US$000**		0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Liabilities	**US$000**		**0**	**0**	**252**	**684**	**691**	**453**	**457**	**423**	**488**	**475**	**434**	**0**	**0**	**0**	**0**
Working Capital	**US$000**		**0**	**0**	**2,655**	**5,517**	**5,410**	**6,568**	**5,247**	**5,840**	**5,685**	**8,659**	**6,418**	**0**	**0**	**0**	**0**
Change in Working Capital	**US$000**	**0**	0	0	2,655	2,862	(107)	1,159	(1,322)	593	(154)	2,974	(2,241)	(6,418)	0		
TOTAL CAPITAL	**US$000**	**121,266**	**575**	**1,725**	**16,342**	**18,352**	**52,505**	**5,210**	**2,735**	**14,404**	**12,973**	**9,298**	**2,579**	**(15,432)**	**0**	**0**	**0**

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
PRE-TAX CASH FLOW

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
CASH FLOW (continued)																	
Gross Income from Mining	**US$000**	**501,659**	**0**	**0**	**26,899**	**44,420**	**42,788**	**60,393**	**44,454**	**56,726**	**55,820**	**98,283**	**71,878**	**0**	**0**	**0**	**0**
Costs																	
Operating	**US$000**	**224,079**	0	0	12,946	35,195	35,520	23,294	23,517	21,737	25,084	24,451	22,334	0	0	0	0
Capital	**US$000**	**121,266**	575	1,725	16,342	18,352	52,505	5,210	2,735	14,404	12,973	9,298	2,579	(15,432)	0	0	0
Working Capital	**US$000**	**0**	0	0	2,655	2,862	(107)	1,159	(1,322)	593	(154)	2,974	(2,241)	(6,418)	0	0	0
Total Costs	**US$000**	**345,345**	**575**	**1,725**	**31,942**	**56,409**	**87,917**	**29,663**	**24,931**	**36,735**	**37,903**	**36,723**	**22,671**	**(21,850)**	**0**	**0**	**0**
Cumulative	***US$000***		*575*	*2,300*	*34,242*	*90,651*	*178,569*	*208,232*	*233,162*	*269,897*	*307,800*	*344,523*	*367,195*	*345,345*	*345,345*	*345,345*	*345,345*
CASH FLOW	**US$000**	**156,314**	**(575)**	**(1,725)**	**(5,044)**	**(11,989)**	**(45,129)**	**30,730**	**19,524**	**19,991**	**17,917**	**61,560**	**49,206**	**21,850**	**0**	**0**	**0**
Cumulative	**US$000**		(575)	(2,300)	(7,344)	(19,333)	(64,462)	(33,733)	(14,209)	5,782	23,699	85,258	134,464	156,314	156,314	156,314	156,314
Present Value	**4.0%**	**103,524**	(575)	(1,659)	(4,663)	(10,658)	(38,577)	25,258	15,430	15,192	13,091	43,251	33,242	14,193	0	0	0
NPV			(575)	(2,234)	(6,897)	(17,555)	(56,132)	(30,874)	(15,445)	(253)	12,838	56,089	89,331	103,524	103,524	103,524	103,524
IRR	**%**	**33%**															

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
CAPTIAL COSTS

Description	Unit Cost	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
MINE EQUIPMENT																	
OPEN PIT MINE	1.00																
Rotary Drill - 251mm	(US$000)	127.0			63.5	63.5											
* Hydraulic Drill - 165mm	(US$000)	758.5			58.5	175.0	175.0	175.0	175.0								
Shovel - 16m3	(US$000)	317.5			317.5												
Bucket (spare) - 16m3	(US$000)	15.0			15.0												
* Mass Excavator - 12m3	(US$000)	584.2						292.1	292.1								
Bucket (spare) - 12m3	(US$000)	0.0															
Front End Loader - 12m3	(US$000)	72.0			72.0												
FEL Bucket (spare) - 12m3	(US$000)	5.5			5.5												
Truck - 150t	(US$000)	3,342.7			582.7		690.0	1,035.0	1,035.0								
Truck 100t	(US$000)	127.5			127.5												
Water Truck - 10k gal	(US$000)	52.0			52.0												
Dozer - 350-450hp	(US$000)	40.5			40.5												
Dozer - 800-900hp	(US$000)	60.3			60.3												
Grader - 16ft	(US$000)	29.5			29.5												
Mass Excavator - 4m3	(US$000)	39.5			39.5												
Light Plant	(US$000)	12.0			12.0												
Stemming Truck	(US$000)	91.0			91.0												
Low Boy	(US$000)	267.5			267.5												
Lube Truck	(US$000)	9.0			9.0												
Fuel Truck	(US$000)	6.5			6.5												
Mechanic Truck	(US$000)	10.8			10.8												
Pickup Truck	(US$000)	759.2			599.2			60.0	60.0	40.0							
Welding/Crane Truck	(US$000)	4.0			4.0												
Crane - Rough Terrain - 20t	(US$000)	30.8			30.8												
Skid Loader	(US$000)	4.5			4.5												
Integrated Tool Carrier - 200hp	(US$000)	22.5			22.5												
Ambulance/Fire Equipment	(US$000)	20.5			20.5												
Flatbed Truck	(US$000)	4.5			4.5												
Crew Vans	(US$000)	167.0			107.0			30.0	30.0								
Forklift	(US$000)	4.0			4.0												
ATV	(US$000)	15.7			10.7				5.0								
Miscellaneous MT Tunnels	(US$000)	100.0			100.0												
	(US$000)	0.0															
subtotal	(US$000)	**7,101.2**	**0.0**	**0.0**	**2,768.5**	**238.5**	**865.0**	**1,592.1**	**1,597.1**	**40.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Mobilization/Demob.	0.0%	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Contingency	10.0%	710.1	0.0	0.0	276.9	23.9	86.5	159.2	159.7	4.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Open Pit Equipment	(US$000)	**7,811.3**	**0.0**	**0.0**	**3,045.4**	**262.4**	**951.5**	**1,751.3**	**1,756.8**	**44.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
CAPTIAL COSTS

Description	Unit Cost	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
MINE EQUIPMENT																	
UNDERGROUND MINE	1.00																
Equipment																	
Drill Jumbo - 2 Boom	(US$000)	1,640.0								820.0	820.0						
LHD - 6yd3	(US$000)	2,600.0								1,300.0	1,300.0						
Truck - 28t	(US$000)	2,520.0								1,260.0	1,260.0						
Scizzors Lift	(US$000)	275.0								275.0							
Mechanical Bolter	(US$000)	1,200.0								600.0	600.0						
Powder Wagon	(US$000)	320.0								320.0							
Road Grader	(US$000)	400.0								400.0							
Lube/Fuel Truck	(US$000)	250.0								250.0							
Mechanics Vehicles	(US$000)	150.0								75.0	75.0						
Boom Truck	(US$000)	250.0								250.0							
Tractors	(US$000)	300.0								150.0	150.0						
subtotal	(US$000)	9,905.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	5,700.0	4,205.0	0.0	0.0	0.0	0.0	0.0	0.0
Fixed Equipment																	
Sand Backfill System	(US$000)	1,250.0								1,250.0							
Ventilation Upgrade	(US$000)	700.0								350.0	350.0						
Water Management	(US$000)	500.0								200.0	300.0						
Underground Shop	(US$000)	200.0									200.0						
Electrical	(US$000)	500.0								500.0							
	(US$000)	0.0															
subtotal	(US$000)	3,150.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	2,300.0	850.0	0.0	0.0	0.0	0.0	0.0	0.0
Total	(US$000)	13,055.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	8,000.0	5,055.0	0.0	0.0	0.0	0.0	0.0	0.0
Mobilization/Demob.	0.0%	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Contingency	15.0%	1,958.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	1,200.0	758.3	0.0	0.0	0.0	0.0	0.0	0.0
Underground Equipment	(US$000)	15,013.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	9,200.0	5,813.3	0.0	0.0	0.0	0.0	0.0	0.0

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
CAPTIAL COSTS

Description	Unit Cost	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
PROCESS PLANT																	
Direct Cost	1.00																
Crushing	(US$000)	2,950				738	2,213										
Grinding & Gravity	(US$000)	4,410				1,103	3,308										
Leach & CIP	(US$000)	4,310				1,078	3,233										
ADR	(US$000)	920				230	690										
Reagents	(US$000)	740				185	555										
Services	(US$000)	970				243	728										
subtotal	(US$000)	14,300	0	0	0	3,575	10,725	0	0	0	0	0	0	0	0	0	0
Allowance for Uspecifed Equip.	10.0%	1,430				358	1,073										
Equipment Cost	(US$000)	15,730	0	0	0	3,933	11,798	0	0	0	0	0	0	0	0	0	0
Installation Cost Factor	0.00	0				0	0										
Installed Cost	(US$000)	15,730	0	0	0	3,933	11,798	0	0	0	0	0	0	0	0	0	0
Allowances:																	
Site Preparation	(US$000)	786				197	590										
Concrete, Steel & Buildings	(US$000)	9,428				2,357	7,071										
Mechnical Installation	(US$000)	1,886				472	1,415										
Piping	(US$000)	3,143				786	2,357										
Electrical/Instrumentation	(US$000)	6,286				1,572	4,715										
	(US$000)	0				0	0										
Direct Costs	(US$000)	37,259	0	0	0	9,315	27,944	0	0	0	0	0	0	0	0	0	0
Indirect Costs																	
EPCM	15.0%	5,589				1,397	4,192										
Contractor Indirects	(US$000)	4,469				1,117	3,352										
First Fills	(US$000)	157				39	118										
Spare Parts	(US$000)	629				157	472										
Vendor Reps	(US$000)	707				177	530										
Freight	(US$000)	1,043				261	782										
	(US$000)	0				0	0										
Indirect Costs	(US$000)	12,594	0	0	0	3,148	9,445	0	0	0	0	0	0	0	0	0	0
Direct & Indirect Costs	(US$000)	49,853	0	0	0	12,463	37,390	0	0	0	0	0	0	0	0	0	0
Contingency	21.0%					2,649	7,946										
Process Equipment	(US$000)	60,448	0	0	0	15,112	45,336	0	0	0	0	0	0	0	0	0	0

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
CAPTIAL COSTS

Description	Unit Cost	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
INFRASTRUCTURE & OWNER COSTS																	
Infrastructure	1.00																
Tailings Dam	(US$000)	12,000					5,500	2,000	2,000	1,500	1,000						
Access Road	(US$000)	50			50												
Water Supply	(US$000)	250			250												
Power Distribution	(US$000)	250			250												
Communications	(US$000)	50			50												
O/P Haul Roads	(US$000)	200			100	100											
Administration Bldg	(US$000)	250			250												
Shop Building	(US$000)	1,000			1,000												
Site Fencing	(US$000)	75			75												
	(US$000)	0															
subtotal	(US$000)	14,125	0	0	2,025	100	5,500	2,000	2,000	1,500	1,000	0	0	0	0	0	0
Mobilization/Demob.	0.0%	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Contingency	15.0%	2,119	0	0	304	15	825	300	300	225	150	0	0	0	0	0	0
Infrastructure	(US$000)	16,244	0	0	2,329	115	6,325	2,300	2,300	1,725	1,150	0	0	0	0	0	0
Owner Costs	1.00																
Engineering/Design Studies	(US$000)	1,500	500	500	250					250							
Geologic Drilling	(US$000)	1,000		1,000													
Corporate Overhead Allocation	(US$000)	0															
Start-Up & Commissioning	(US$000)	0															
Mine Closure	(US$000)	2,600											1,000	1,600			
Mill Salvage	(US$000)	(9,438)												(9,438)			
	(US$000)	0	0														
subtotal	(US$000)	(4,338)	500	1,500	250	0	0	0	0	250	0	0	1,000	(7,838)	0	0	0
Mobilization/Demob.	0.0%	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Contingency	15.0%	(651)	75	225	38	0	0	0	0	38	0	0	150	(1,176)	0	0	0
Owner Cost	(US$000)	(4,989)	575	1,725	288	0	0	0	0	288	0	0	1,150	(9,014)	0	0	0

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
OPERATING COSTS

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
PRODUCTION STATISITICS																	
MATERIAL MOVED																	
Ore & Waste																	
Open Pit Waste Stripped	kt	46,940	0	0	8,681	8,856	7,413	7,689	7,859	3,792	2,652	0	0	0	0	0	0
Open Pit Ore	kt	3,362	0	0	244	528	624	490	569	523	384	0	0	0	0	0	0
Underground Ore	kt	1,108	0	0	0	0	0	0	0	0	129	541	437	0	0	0	0
Open Pit Ore & Waste	kt	50,302	0	0	8,925	9,384	8,037	8,178	8,427	4,315	3,036	0	0	0	0	0	0
Toll-Milled Ore	kt	1,305	0	0	225	540	540	0	0	0	0	0	0	0	0	0	0
Owner-Milled Ore	kt	3,165	0	0	0	0	0	540	540	540	540	540	465	0	0	0	0
Backfill - Rock	m3	233,161	0	0	0	0	0	0	0	0	30,073	108,965	94,123	0	0	0	0
Hydraulic Sandfill	m3	62,176	0	0	0	0	0	0	0	0	8,020	29,057	25,099	0	0	0	0
Cemented Sandfill	m3	15,544	0	0	0	0	0	0	0	0	2,005	7,264	6,275	0	0	0	0
Tailings Dam	m3	2,398	0	0	0	0	0	409	409	409	409	409	352	0	0	0	0
Underground Development																	
Main Ramp	m	4,982	0	0	0	0	0	0	0	1,570	2,505	907	0	0	0	0	0
Special Ramp to 9805	m	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Level Drift	m	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Stope Access Drifts	m	4,278	0	0	0	0	0	0	0	0	565	2,371	1,342	0	0	0	0
Production Stope Ramps	m	1,470	0	0	0	0	0	0	0	0	360	600	510	0	0	0	0
Ventilation Raises	m	896	0	0	0	0	0	0	0	0	455	441	0	0	0	0	0
Muck Bays	m	349	0	0	0	0	0	0	0	110	175	63	0	0	0	0	0
Total Development	m	11,975	0	0	0	0	0	0	0	1,680	4,060	4,382	1,852	0	0	0	0
GOLD RECOVERED																	
Toll Mill Gold	koz	219	0	0	52	85	82	0	0	0	0	0	0	0	0	0	0
Owner Milled Ore	koz	743	0	0	0	0	0	116	85	109	107	188	138	0	0	0	0
Total Gold	**koz**	**962**	**0**	**0**	**52**	**85**	**82**	**116**	**85**	**109**	**107**	**188**	**138**	**0**	**0**	**0**	**0**
OPERATING COSTS																	
SUMMARY																	
Mine G&A	US$000s	15,261	0	0	1,446	1,659	1,584	1,584	1,434	1,840	1,955	1,880	1,880	0	0	0	0
Open Pit Mine	US$000s	79,768	0	0	10,032	13,876	14,147	13,247	13,275	8,466	6,725	0	0	0	0	0	0
Underground Mine & Develop.	US$000s	53,879	0	0	0	0	0	0	0	5,177	13,760	19,471	15,470	0	0	0	0
Toll Mill	US$000s	42,504	0	0	7,328	17,588	17,588	0	0	0	0	0	0	0	0	0	0
Owner Mill	US$000s	41,955	0	0	134	134	262	6,525	6,870	6,870	6,870	7,521	6,770	0	0	0	0
G&A	US$000s	17,451	0	0	2,031	1,939	1,939	1,939	1,939	1,939	1,939	1,904	1,884	0	0	0	0
Total Operating	**US$000s**	**250,817**	**0**	**0**	**20,971**	**35,195**	**35,520**	**23,294**	**23,517**	**24,292**	**31,249**	**30,775**	**26,004**	**0**	**0**	**0**	**0**
per ton milled	US$/t	56.11			85.91	66.62	56.90	47.56	41.35	46.45	60.86	56.85	59.47				
per oz-Au	US$/oz-Au	260.79			406.57	413.26	432.97	201.19	275.92	223.37	292.00	163.35	188.71				
MINE G&A	1.00																
Salaried Labor	US$000s	14,361	0	0	1,346	1,559	1,484	1,484	1,334	1,740	1,855	1,780	1,780	0	0	0	0
Maintenance Labor	US$000s	0			0	0	0	0	0	0	0	0	0	0			
Fuel & Lubricants	US$000s	0			0	0	0	0	0	0	0	0	0	0			
Consumables & Parts	US$000s	0			0	0	0	0	0	0	0	0	0	0			
Other Supplies	US$000s	900			100	100	100	100	100	100	100	100	100	0			
Mine G&A	**US$000s**	**15,261**	**0**	**0**	**1,446**	**1,659**	**1,584**	**1,584**	**1,434**	**1,840**	**1,955**	**1,880**	**1,880**	**0**	**0**	**0**	**0**
per ton mined	US$/t	3.414			5.925	3.140	2.537	3.233	2.521	3.518	3.808	3.473	4.300				

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
OPERATING COSTS

			preproduction			production											
Description	**Units**	**LoM Total**	**2007 -03**	**2008 -02**	**2009 -01**	**2010 01**	**2011 02**	**2012 03**	**2013 04**	**2014 05**	**2015 06**	**2016 07**	**2017 08**	**2018 09**	**2019 10**	**2020 11**	**2021 12**
OPEN PIT MINE																	
Open Pit Opex																	
Drilling	**US$000s**	**10,405**	0	0	658	1,756	1,609	1,753	1,793	1,545	1,291	0	0	0	0	0	0
Blasting	**US$000s**	**11,590**	0	0	734	2,289	1,795	2,160	2,221	1,402	989	0	0	0	0	0	0
Loading	**US$000s**	**13,591**	0	0	2,025	2,209	2,551	2,045	2,012	1,486	1,263	0	0	0	0	0	0
Hauling	**US$000s**	**22,181**	0	0	3,516	3,663	4,233	3,329	3,359	2,207	1,874	0	0	0	0	0	0
Support	**US$000s**	**22,001**	0	0	3,099	3,959	3,959	3,959	3,891	1,825	1,308	0	0	0	0	0	0
Open Pit Mining	**US$000s**	**79,768**	**0**	**0**	**10,032**	**13,876**	**14,147**	**13,247**	**13,275**	**8,466**	**6,725**	**0**	**0**	**0**	**0**	**0**	**0**
per ton mined	US$/t	1.586			1.124	1.479	1.760	1.620	1.575	1.962	2.215						
Drilling	1.00																
Operating Labor	**US$000s**	**4,637**	0	0	357	713	713	713	713	713	713	0	0	0	0	0	0
Maintenance Labor	**US$000s**	**2,189**	0	0	141	346	346	346	306	351	351	0	0	0	0	0	0
Fuel & Lubricants	**US$000s**	**1,403**			61	271	215	272	304	191	90						
Consumables & Parts	**US$000s**	**2,176**			100	425	335	421	470	290	136						
Other Supplies	**US$000s**	**0**			0	0	0	0	0	0	0						
Drilling	**US$000s**	**10,405**	**0**	**0**	**658**	**1,756**	**1,609**	**1,753**	**1,793**	**1,545**	**1,291**	**0**	**0**	**0**	**0**	**0**	**0**
per ton mined	US$/t	0.207			0.074	0.187	0.200	0.214	0.213	0.358	0.425						
Blasting	1.00																
Operating Labor	**US$000s**	**2,209**	0	0	212	356	356	356	356	287	287	0	0	0	0	0	0
Maintenance Labor	**US$000s**	**1,037**	0	0	84	173	173	173	153	141	141	0	0	0	0	0	0
Fuel & Lubricants	**US$000s**	**0**			0	0	0	0	0	0	0						
Consumables & Parts	**US$000s**	**8,344**			438	1,760	1,267	1,632	1,712	974	561						
Other Supplies	**US$000s**	**0**			0	0	0	0	0	0	0						
Blasting	**US$000s**	**11,590**	**0**	**0**	**734**	**2,289**	**1,795**	**2,160**	**2,221**	**1,402**	**989**	**0**	**0**	**0**	**0**	**0**	**0**
per ton mined	US$/t	0.230			0.082	0.244	0.223	0.264	0.264	0.325	0.326						
Loading	1.00																
Operating Labor	**US$000s**	**4,289**	0	0	456	639	639	639	639	639	639	0	0	0	0	0	0
Maintenance Labor	**US$000s**	**2,014**	0	0	180	310	310	310	274	314	314	0	0	0	0	0	0
Fuel & Lubricants	**US$000s**	**3,614**			685	627	793	545	546	265	154						
Consumables & Parts	**US$000s**	**3,673**			703	634	809	551	552	268	156						
Other Supplies	**US$000s**	**0**			0	0	0	0	0	0	0						
Loading	**US$000s**	**13,591**	**0**	**0**	**2,025**	**2,209**	**2,551**	**2,045**	**2,012**	**1,486**	**1,263**	**0**	**0**	**0**	**0**	**0**	**0**
per ton mined	US$/t	0.270			0.227	0.235	0.317	0.250	0.239	0.344	0.416						
Hauling	1.00																
Operating Labor	**US$000s**	**7,080**	0	0	1,077	1,077	1,077	1,077	1,077	847	847	0	0	0	0	0	0
Maintenance Labor	**US$000s**	**3,291**	0	0	426	523	523	523	463	417	417	0	0	0	0	0	0
Fuel & Lubricants	**US$000s**	**6,871**			1,171	1,200	1,532	1,006	1,058	549	356						
Consumables & Parts	**US$000s**	**4,938**			841	862	1,101	723	760	395	256						
Other Supplies	**US$000s**	**0**			0	0	0	0	0	0	0						
Hauling	**US$000s**	**22,181**	**0**	**0**	**3,516**	**3,663**	**4,233**	**3,329**	**3,359**	**2,207**	**1,874**	**0**	**0**	**0**	**0**	**0**	**0**
per ton mined	US$/t	0.441			0.394	0.390	0.527	0.407	0.399	0.512	0.617						
Support	1.00																
Operating Labor	**US$000s**	**7,469**	0	0	1,233	1,233	1,233	1,233	1,233	651	651	0	0	0	0	0	0
Maintenance Labor	**US$000s**	**3,455**	0	0	488	599	599	599	530	320	320	0	0	0	0	0	0
Fuel & Lubricants	**US$000s**	**6,546**			796	1,267	1,267	1,267	1,267	489	195						
Consumables & Parts	**US$000s**	**4,531**			582	861	861	861	861	365	141						
Other Supplies	**US$000s**	**0**			0	0	0	0	0	0	0						
Support	**US$000s**	**22,001**	**0**	**0**	**3,099**	**3,959**	**3,959**	**3,959**	**3,891**	**1,825**	**1,308**	**0**	**0**	**0**	**0**	**0**	**0**
per ton mined	US$/t	0.437			0.347	0.422	0.493	0.484	0.462	0.423	0.431						

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
OPERATING COSTS

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
UNDERGROUND MINE																	
Underground Opex			**Note: Development costs are capitalized.**														
Development	**US$000s**	**18,712**	0	0	0	0	0	0	0	2,554	6,164	6,324	3,670	0	0	0	0
Stope Production	**US$000s**	**32,202**	0	0	0	0	0	0	0	2,623	7,235	11,725	10,619	0	0	0	0
Ore Haulage	**US$000s**	**1,728**	0	0	0	0	0	0	0	0	201	844	682	0	0	0	0
Backfill	**US$000s**	**1,236**	0	0	0	0	0	0	0	0	159	578	499	0	0	0	0
Underground Mining	**US$000s**	**53,879**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**5,177**	**13,760**	**19,471**	**15,470**	**0**	**0**	**0**	**0**
per ton mined	US$/t	48.641									106.543	35.970	35.383				
DEVELOPMENT																	
Main Ramp	1.00																
Operating Labor	**US$000s**	**1,724**	0	0	0	0	0	0	0	482	931	311	0	0	0	0	0
Maintenance Labor	**US$000s**	**829**	0	0	0	0	0	0	0	393	330	107	0	0	0	0	0
O&M	$1,070.00	**5,331**	0	0	0	0	0	0	0	1,680	2,680	970	0	0	0	0	0
Main Ramp	**US$000s**	**7,884**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**2,554**	**3,941**	**1,389**	**0**	**0**	**0**	**0**	**0**
Stope Access Drifts	1.00																
Operating Labor	**US$000s**	**2,096**	0	0	0	0	0	0	0	0	210	813	1,073	0	0	0	0
Maintenance Labor	**US$000s**	**725**	0	0	0	0	0	0	0	0	74	281	370	0	0	0	0
O&M	$906.00	**3,876**	0	0	0	0	0	0	0	0	512	2,148	1,216	0	0	0	0
Stope Access Drift	**US$000s**	**6,697**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**796**	**3,242**	**2,659**	**0**	**0**	**0**	**0**
Production Stope Ramp	1.00																
Operating Labor	**US$000s**	**747**	0	0	0	0	0	0	0	0	134	206	408	0	0	0	0
Maintenance Labor	**US$000s**	**259**	0	0	0	0	0	0	0	0	47	71	141	0	0	0	0
O&M	$906.00	**1,332**	0	0	0	0	0	0	0	0	326	544	462	0	0	0	0
Production Stope Ramp	**US$000s**	**2,338**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**507**	**820**	**1,011**	**0**	**0**	**0**	**0**
Ventilation Raises	1.00																
Operating Labor	**US$000s**	**320**	0	0	0	0	0	0	0	0	169	151	0	0	0	0	0
Maintenance Labor	**US$000s**	**112**	0	0	0	0	0	0	0	0	60	52	0	0	0	0	0
O&M	$1,519.00	**1,361**	0	0	0	0	0	0	0	0	691	670	0	0	0	0	0
Vent. Raise	**US$000s**	**1,793**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**920**	**873**	**0**	**0**	**0**	**0**	**0**
STOPE PRODUCTION																	
Stope Production	1.00																
Operating Labor	**US$000s**	**14,662**	0	0	0	0	0	0	0	1,445	4,331	4,443	4,443	0	0	0	0
Maintenance Labor	**US$000s**	**5,777**	0	0	0	0	0	0	0	1,178	1,533	1,533	1,533	0	0	0	0
O&M	$8.78	**9,725**	0	0	0	0	0	0	0	0	1,134	4,753	3,839	0	0	0	0
Mine General	$1.84	**2,038**	0	0	0	0	0	0	0	0	238	996	804	0	0	0	0
Stope Production	**US$000s**	**32,202**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**2,623**	**7,235**	**11,725**	**10,619**	**0**	**0**	**0**	**0**
per ton mined	US$/t	29.072									56.021	21.660	24.288				
Ore Haulage	1.00		**Note: Labor included in 'Stope Production' above.**														
Operating Labor	**US$000s**	**0**															
O&M	$1.56	**1,728**	0	0	0	0	0	0	0	0	201	844	682	0	0	0	0
Ore Haulage	**US$000s**	**1,728**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**201**	**844**	**682**	**0**	**0**	**0**	**0**
per ton mined	US$/t	1.560									1.560	1.560	1.560				
Backfill	1.00		**Note: Labor included in 'Stope Production' above.**														
Operating Labor	**US$000s**	**0**															
Rockfill	$1.69	**394**	0	0	0	0	0	0	0	0	51	184	159	0	0	0	0
Hydraulic Fill	$10.46	**650**	0	0	0	0	0	0	0	0	84	304	263	0	0	0	0
Cemented Sand Fill	$12.33	**192**	0	0	0	0	0	0	0	0	25	90	77	0	0	0	0
Backfill	**US$000s**	**1,236**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**159**	**578**	**499**	**0**	**0**	**0**	**0**
per ton mined	US$/t	1.116									1.234	1.067	1.141				

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
OPERATING COSTS

			preproduction			production											
Description	**Units**	**LoM Total**	**2007 -03**	**2008 -02**	**2009 -01**	**2010 01**	**2011 02**	**2012 03**	**2013 04**	**2014 05**	**2015 06**	**2016 07**	**2017 08**	**2018 09**	**2019 10**	**2020 11**	**2021 12**
TOLL MILL																	
Toll Mill Opex																	
Crushing	**US$000s**	**914**	0	0	158	378	378	0	0	0	0	0	0	0	0	0	0
Transportation	**US$000s**	**9,761**	0	0	1,683	4,039	4,039	0	0	0	0	0	0	0	0	0	0
Processing	**US$000s**	**31,829**	0	0	5,488	13,171	13,171	0	0	0	0	0	0	0	0	0	0
	US$000s	**0**															
Toll Milling	**US$000s**	**42,504**	**0**	**0**	**7,328**	**17,588**	**17,588**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
per ton-milled	US$/t	32.570			32.570	32.570	32.570										
Crushing	1.00																
Crushing	**US$000s**	**914**			158	378	378										
	US$000s	**0**															
Crushing	**US$000s**	**914**	**0**	**0**	**158**	**378**	**378**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
per ton milled	US$/t	0.700			0.700	0.700	0.700										
Transportation	1.00																
Transportation	**US$000s**	**9,761**			1,683	4,039	4,039										
	US$000s	**0**															
Transportation	**US$000s**	**9,761**	**0**	**0**	**1,683**	**4,039**	**4,039**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
per ton milled	US$/t	7.480			7.480	7.480	7.480										
Processing	1.00																
Transportation	**US$000s**	**31,829**			5,488	13,171	13,171										
	US$000s	**0**															
Transportation	**US$000s**	**31,829**	**0**	**0**	**5,488**	**13,171**	**13,171**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
per ton milled	US$/t	24.390			24.390	24.390	24.390										

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
OPERATING COSTS

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
OWNER MILL																	
Owner Mill Opex																	
Labor	**US$000s**	**10,216**	0	0	134	134	262	1,110	1,455	1,455	1,455	2,106	2,106	0	0	0	0
Operating Supplies	**US$000s**	**14,336**	0	0	0	0	0	2,446	2,446	2,446	2,446	2,446	2,107	0	0	0	0
Repair Parts	**US$000s**	**4,463**	0	0	0	0	0	761	761	761	761	761	656	0	0	0	0
Power	**US$000s**	**12,940**	0	0	0	0	0	2,208	2,208	2,208	2,208	2,208	1,901	0	0	0	0
Owner Milling	**US$000s**	**41,955**	**0**	**0**	**134**	**134**	**262**	**6,525**	**6,870**	**6,870**	**6,870**	**7,521**	**6,770**	**0**	**0**	**0**	**0**
per ton-milled	US$/t	13.256						12.083	12.722	12.722	12.722	13.928	14.556				
Labor	1.00																
Salaried	**US$000s**	**5,712**	0	0	134	134	262	576	921	921	921	921	921	0	0	0	0
Hourly	**US$000s**	**4,504**	0	0	0	0	0	534	534	534	534	1,185	1,185	0	0	0	0
Labor	**US$000s**	**10,216**	**0**	**0**	**134**	**134**	**262**	**1,110**	**1,455**	**1,455**	**1,455**	**2,106**	**2,106**	**0**	**0**	**0**	**0**
per ton milled	US$/t	3.228						2.055	2.694	2.694	2.694	3.900	4.528				
Operating Supplies	1.00																
Primary Crusher Liners	**US$000s**	**562**	0	0	0	0	0	96	96	96	96	96	83	0	0	0	0
Pebble Crusher Liners	**US$000s**	**250**	0	0	0	0	0	43	43	43	43	43	37	0	0	0	0
SAG Mill Liners	**US$000s**	**2,279**	0	0	0	0	0	389	389	389	389	389	335	0	0	0	0
Ball Mill Liners	**US$000s**	**821**	0	0	0	0	0	140	140	140	140	140	121	0	0	0	0
Tower Mill Liners	**US$000s**	**49**	0	0	0	0	0	8	8	8	8	8	7	0	0	0	0
Grinding Balls, 100mm	**US$000s**	**873**	0	0	0	0	0	149	149	149	149	149	128	0	0	0	0
Grinding Balls, 38mm	**US$000s**	**1,188**	0	0	0	0	0	203	203	203	203	203	175	0	0	0	0
Grinding Balls, 25mm	**US$000s**	**249**	0	0	0	0	0	43	43	43	43	43	37	0	0	0	0
Calcium Chloride	**US$000s**	**155**	0	0	0	0	0	26	26	26	26	26	23	0	0	0	0
Carbon	**US$000s**	**295**	0	0	0	0	0	50	50	50	50	50	43	0	0	0	0
Caustic Soda	**US$000s**	**1,012**	0	0	0	0	0	173	173	173	173	173	149	0	0	0	0
Copper Sulphate	**US$000s**	**8**	0	0	0	0	0	1	1	1	1	1	1	0	0	0	0
Ferric Sulfate	**US$000s**	**39**	0	0	0	0	0	7	7	7	7	7	6	0	0	0	0
Flocculant Percol 351	**US$000s**	**395**	0	0	0	0	0	67	67	67	67	67	58	0	0	0	0
Hydrogen Peroxide	**US$000s**	**210**	0	0	0	0	0	36	36	36	36	36	31	0	0	0	0
Lead Nitrate	**US$000s**	**20**	0	0	0	0	0	3	3	3	3	3	3	0	0	0	0
Lime	**US$000s**	**26**	0	0	0	0	0	4	4	4	4	4	4	0	0	0	0
Quick Lime	**US$000s**	**666**	0	0	0	0	0	114	114	114	114	114	98	0	0	0	0
Nitric Acid	**US$000s**	**385**	0	0	0	0	0	66	66	66	66	66	57	0	0	0	0
Sodium Cyanide	**US$000s**	**3,683**	0	0	0	0	0	628	628	628	628	628	541	0	0	0	0
Miscellaneous	**US$000s**	**253**	0	0	0	0	0	43	43	43	43	43	37	0	0	0	0
Vehicles	**US$000s**	**475**	0	0	0	0	0	81	81	81	81	81	70	0	0	0	0
Assay Lab Supplies	**US$000s**	**443**	0	0	0	0	0	76	76	76	76	76	65	0	0	0	0
Operating Supplies	**US$000s**	**14,336**	**0**	**0**	**0**	**0**	**0**	**2,446**	**2,446**	**2,446**	**2,446**	**2,446**	**2,107**	**0**	**0**	**0**	**0**
per ton milled	US$/t	4.529						4.529	4.529	4.529	4.529	4.529	4.529				
Repair Parts	1.00																
Primary Crusher Liners	**US$000s**	**4,463**	0	0	0	0	0	761	761	761	761	761	656	0	0	0	0
	US$000s	**0**															
Repair Parts	**US$000s**	**4,463**	**0**	**0**	**0**	**0**	**0**	**761**	**761**	**761**	**761**	**761**	**656**	**0**	**0**	**0**	**0**
per ton milled	US$/t	1.410						1.410	1.410	1.410	1.410	1.410	1.410				
Power	1.00																
Energy Consumption	**kWh**	-						24.800	24.800	24.800	24.800	24.800	21.359				
Rate	**$/kWh**	-						0.071	0.071	0.071	0.071	0.071	0.071				
Demand	**kW**	-						2,831	2,831	2,831	2,831	2,831	2,438				
Rate	**$/kW**	-						13.010	13.010	13.010	13.010	13.010	13.010				
Power	**US$000s**	**12,940**	**0**	**0**	**0**	**0**	**0**	**2,208**	**2,208**	**2,208**	**2,208**	**2,208**	**1,901**	**0**	**0**	**0**	**0**

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
OPERATING COSTS

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
per ton milled	US$/t	4.088						4.088	4.088	4.088	4.088	4.088	4.088				

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
OPERATING COSTS

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
GENERAL & ADMINISTRATIVE																	
G&A Opex																	
Labor	**US$000s**	**10,699**	0	0	1,199	1,199	1,199	1,199	1,199	1,199	1,199	1,164	1,144	0	0	0	0
Operating Supplies	**US$000s**	**6,752**	0	0	832	740	740	740	740	740	740	740	740	0	0	0	0
G&A	**US$000s**	**17,451**	**0**	**0**	**2,031**	**1,939**	**1,939**	**1,939**	**1,939**	**1,939**	**1,939**	**1,904**	**1,884**	**0**	**0**	**0**	**0**
per ton-milled	US$/t	3.904			9.026	3.590	3.590	3.590	3.590	3.590	3.590	3.526	4.051				
Labor	1.00																
Salaried	**US$000s**	**5,569**	0	0	619	619	619	619	619	619	619	619	619	0	0	0	0
Hourly	**US$000s**	**5,130**	0	0	580	580	580	580	580	580	580	545	525	0	0	0	0
Labor	**US$000s**	**10,699**	**0**	**0**	**1,199**	**1,199**	**1,199**	**1,199**	**1,199**	**1,199**	**1,199**	**1,164**	**1,144**	**0**	**0**	**0**	**0**
per ton milled	US$/t	2.393			5.328	2.220	2.220	2.220	2.220	2.220	2.220	2.155	2.459				
Operating Supplies	1.00																
Supplies	**US$000s**	**540**			60	60	60	60	60	60	60	60	60				
Postage Couriers	**US$000s**	**180**			20	20	20	20	20	20	20	20	20				
Communication	**US$000s**	**270**			30	30	30	30	30	30	30	30	30				
Travel	**US$000s**	**270**			30	30	30	30	30	30	30	30	30				
Outside Services	**US$000s**	**360**			40	40	40	40	40	40	40	40	40				
Environmental Monitoring	**US$000s**	**324**			36	36	36	36	36	36	36	36	36				
Community Relations	**US$000s**	**500**			100	50	50	50	50	50	50	50	50				
Insurance	**US$000s**	**1,125**			125	125	125	125	125	125	125	125	125				
Recruiting & Relocation	**US$000s**	**300**			60	30	30	30	30	30	30	30	30				
Safety Supplies	**US$000s**	**324**			36	36	36	36	36	36	36	36	36				
Security Supplies	**US$000s**	**54**			6	6	6	6	6	6	6	6	6				
Vehicle & Bus Ooperations	**US$000s**	**871**			63	101	101	101	101	101	101	101	101				
Customs	**US$000s**	**270**			30	30	30	30	30	30	30	30	30				
computer Equipment & Software	**US$000s**	**270**			30	30	30	30	30	30	30	30	30				
Training	**US$000s**	**500**			100	50	50	50	50	50	50	50	50				
Power	**US$000s**	**216**			24	24	24	24	24	24	24	24	24				
Water System	**US$000s**	**216**			24	24	24	24	24	24	24	24	24				
Road Maintenance & Snow Rmvl.	**US$000s**	**162**			18	18	18	18	18	18	18	18	18				
Operating Supplies	**US$000s**	**6,752**	**0**	**0**	**832**	**740**	**740**	**740**	**740**	**740**	**740**	**740**	**740**	**0**	**0**	**0**	**0**
per ton milled	US$/t	1.511			3.698	1.370	1.370	1.370	1.370	1.370	1.370	1.370	1.591				

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
LABOR

Description	Units	LoM Total	2007 -03	2008 -02	2009 -01	2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
			preproduction			production											
MINE G&A PERSONNEL																	
PERSONNEL																	
Management																	
Mine Manager		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
Pit Superintendent		**0.00**			0.00												
Drill & Blast Super.		**1.00**			1.00	1.00	1.00	1.00									
General Foreman		**1.00**								1.00	1.00	1.00	1.00				
Foreman		**3.89**			3.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00	4.00				
Clerk		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
Trainer		**1.00**			1.00	1.00				1.00	1.00						
subtotal		**4.33**	**0.00**	**0.00**	**7.00**	**8.00**	**7.00**	**7.00**	**6.00**	**8.00**	**8.00**	**7.00**	**7.00**	**0.00**	**0.00**	**0.00**	**0.00**
Maintenance																	
Maintenance Super.		**0.00**			0.00												
Electrical Foreman		**0.80**			0.00					1.00	1.00	1.00	1.00				
Shop Shift Foreman		**3.22**			2.00	3.00	3.00	3.00	2.00	4.00	4.00	4.00	4.00				
Planning Engineer		**1.00**				1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
subtotal		**2.73**	**0.00**	**0.00**	**2.00**	**4.00**	**4.00**	**4.00**	**3.00**	**6.00**	**6.00**	**6.00**	**6.00**	**0.00**	**0.00**	**0.00**	**0.00**
Engineering & Geology																	
Chief Mining Engineer		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
Surveyors		**1.33**			1.00	1.00	1.00	1.00	1.00	1.00	2.00	2.00	2.00				
Sr. Mining Engineer		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
Mining Engineer		**1.00**									1.00	1.00	1.00				
Environmental Engineer		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
Chief Geologist		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
Ore Control Geologist		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
Sampler		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
Safety Engineer		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
Geotechnical		**1.00**			1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00				
subtotal		**5.80**	**0.00**	**0.00**	**9.00**	**9.00**	**9.00**	**9.00**	**9.00**	**9.00**	**11.00**	**11.00**	**11.00**	**0.00**	**0.00**	**0.00**	**0.00**
Total Mine G&A		**21.44**			**18.00**	**21.00**	**20.00**	**20.00**	**18.00**	**23.00**	**25.00**	**24.00**	**24.00**				
COSTS																	
Management																	
Mine Manager	US$000s	**1,080**	0	0	120	120	120	120	120	120	120	120	120	0	0	0	0
Pit Superintendent	US$000s	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Drill & Blast Super.	US$000s	**300**	0	0	75	75	75	75	0	0	0	0	0	0	0	0	0
General Foreman	US$000s	**375**	0	0	0	0	0	0	0	94	94	94	94	0	0	0	0
Foreman	US$000s	**2,625**	0	0	225	300	300	300	300	300	300	300	300	0	0	0	0
Clerk	US$000s	**439**	0	0	49	49	49	49	49	49	49	49	49	0	0	0	0
Trainer	US$000s	**300**	0	0	75	75	0	0	0	75	75	0	0	0	0	0	0
subtotal	US$000s	**5,119**	**0**	**0**	**544**	**619**	**544**	**544**	**469**	**638**	**638**	**563**	**563**	**0**	**0**	**0**	**0**
Maintenance																	
Maintenance Super.	US$000s	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Electrical Foreman	US$000s	**350**	0	0	0	0	0	0	0	88	88	88	88	0	0	0	0
Shop Shift Foreman	US$000s	**2,175**	0	0	150	225	225	225	150	300	300	300	300	0	0	0	0
Planning Engineer	US$000s	**500**	0	0	0	63	63	63	63	63	63	63	63	0	0	0	0
subtotal	US$000s	**3,025**	**0**	**0**	**150**	**288**	**288**	**288**	**213**	**450**	**450**	**450**	**450**	**0**	**0**	**0**	**0**
Engineering & Geology																	
Chief Mining Engineer	US$000s	**844**	0	0	94	94	94	94	94	94	94	94	94	0	0	0	0
Surveyors	US$000s	**630**	0	0	53	53	53	53	53	53	105	105	105	0	0	0	0
Sr. Mining Engineer	US$000s	**731**	0	0	81	81	81	81	81	81	81	81	81	0	0	0	0
Mining Engineer	US$000s	**188**	0	0	0	0	0	0	0	0	63	63	63	0	0	0	0
Environmental Engineer	US$000s	**675**	0	0	75	75	75	75	75	75	75	75	75	0	0	0	0
Chief Geologist	US$000s	**788**	0	0	88	88	88	88	88	88	88	88	88	0	0	0	0
Ore Control Geologist	US$000s	**619**	0	0	69	69	69	69	69	69	69	69	69	0	0	0	0
Sampler	US$000s	**506**	0	0	56	56	56	56	56	56	56	56	56	0	0	0	0
Safety Engineer	US$000s	**731**	0	0	81	81	81	81	81	81	81	81	81	0	0	0	0
Geotechnical	US$000s	**506**	0	0	56	56	56	56	56	56	56	56	56	0	0	0	0
subtotal	US$000s	**6,218**	**0**	**0**	**653**	**653**	**653**	**653**	**653**	**653**	**768**	**768**	**768**	**0**	**0**	**0**	**0**
Total Mine G&A	US$000s	**14,361**	**0**	**0**	**1,346**	**1,559**	**1,484**	**1,484**	**1,334**	**1,740**	**1,855**	**1,780**	**1,780**	**0**	**0**	**0**	**0**

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
LABOR

Description	Units	LoM Total	2007 -03	2008 -02	2009 -01	2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
			preproduction			production											
OPEN PIT MINING LABOR COSTS																	
PERSONNEL																	
Mine Operations																	
Drill Operators		**7.4**			4.0	8.0	8.0	8.0	8.0	8.0	8.0						
Blasters		**1.9**			1.0	2.0	2.0	2.0	2.0	2.0	2.0						
Blaster Helpers		**2.6**			2.0	3.0	3.0	3.0	3.0	2.0	2.0						
Loader Operators		**6.7**			5.0	7.0	7.0	7.0	7.0	7.0	7.0						
Haul Truck Drivers		**13.1**			14.0	14.0	14.0	14.0	14.0	11.0	11.0						
Dozer Operators		**6.9**			8.0	8.0	8.0	8.0	8.0	4.0	4.0						
Grader Operators		**3.4**			4.0	4.0	4.0	4.0	4.0	2.0	2.0						
Utility Operators		**1.7**			2.0	2.0	2.0	2.0	2.0	1.0	1.0						
Trainees/Spares		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0						
subtotal		**20.9**	**0.0**	**0.0**	**41.0**	**49.0**	**49.0**	**49.0**	**49.0**	**38.0**	**38.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
Maintenance																	
Mechanics		**10.3**			10.0	11.0	11.0	11.0	11.0	9.0	9.0						
Electricians		**1.0**				1.0	1.0	1.0	1.0	1.0	1.0						
Welders		**2.3**			1.0	3.0	3.0	3.0	2.0	2.0	2.0						
Lght. Vehicler Mech,		**2.0**			2.0	2.0	2.0	2.0	2.0	2.0	2.0						
Apprentice		**1.0**				1.0	1.0	1.0									
Storemen		**1.9**			1.0	2.0	2.0	2.0	2.0	2.0	2.0						
Servicemen		**1.4**			1.0	2.0	2.0	2.0	1.0	1.0	1.0						
Tiremen		**1.0**				1.0	1.0	1.0	1.0	1.0	1.0						
subtotal		**9.3**	**0.0**	**0.0**	**15.0**	**23.0**	**23.0**	**23.0**	**20.0**	**18.0**	**18.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**
O/P Operations		**64.7**			**56.0**	**72.0**	**72.0**	**72.0**	**69.0**	**56.0**	**56.0**						
COSTS																	
Mine Operations																	
Drill Operators	**US$000s**	**4,637**	0	0	357	713	713	713	713	713	713	0	0	0	0	0	0
Blasters	**US$000s**	**973**	0	0	75	150	150	150	150	150	150	0	0	0	0	0	0
Blaster Helpers	**US$000s**	**1,236**	0	0	137	206	206	206	206	137	137	0	0	0	0	0	0
Loader Operators	**US$000s**	**4,289**	0	0	456	639	639	639	639	639	639	0	0	0	0	0	0
Haul Truck Drivers	**US$000s**	**7,080**	0	0	1,077	1,077	1,077	1,077	1,077	847	847	0	0	0	0	0	0
Dozer Operators	**US$000s**	**3,994**	0	0	666	666	666	666	666	333	333	0	0	0	0	0	0
Grader Operators	**US$000s**	**1,997**	0	0	333	333	333	333	333	166	166	0	0	0	0	0	0
Utility Operators	**US$000s**	**998**	0	0	166	166	166	166	166	83	83	0	0	0	0	0	0
Trainees/Spares	**US$000s**	**480**	0	0	69	69	69	69	69	69	69	0	0	0	0	0	0
subtotal	**US$000s**	**25,685**	**0**	**0**	**3,336**	**4,019**	**4,019**	**4,019**	**4,019**	**3,137**	**3,137**	**0**	**0**	**0**	**0**	**0**	**0**
Maintenance																	
Mechanics	**US$000s**	**6,571**	0	0	913	1,004	1,004	1,004	1,004	821	821	0	0	0	0	0	0
Electricians	**US$000s**	**548**	0	0	0	91	91	91	91	91	91	0	0	0	0	0	0
Welders	**US$000s**	**1,394**	0	0	87	261	261	261	174	174	174	0	0	0	0	0	0
Lght. Vehicler Mech,	**US$000s**	**1,278**	0	0	183	183	183	183	183	183	183	0	0	0	0	0	0
Apprentice	**US$000s**	**206**	0	0	0	69	69	69	0	0	0	0	0	0	0	0	0
Storemen	**US$000s**	**892**	0	0	69	137	137	137	137	137	137	0	0	0	0	0	0
Servicemen	**US$000s**	**686**	0	0	69	137	137	137	69	69	69	0	0	0	0	0	0
Tiremen	**US$000s**	**412**	0	0	0	69	69	69	69	69	69	0	0	0	0	0	0
subtotal	**US$000s**	**11,986**	**0**	**0**	**1,320**	**1,951**	**1,951**	**1,951**	**1,726**	**1,544**	**1,544**	**0**	**0**	**0**	**0**	**0**	**0**
O/P Operations	**US$000s**	**37,671**	**0**	**0**	**4,656**	**5,970**	**5,970**	**5,970**	**5,745**	**4,681**	**4,681**	**0**	**0**	**0**	**0**	**0**	**0**

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
LABOR

Description	Units	LoM Total	2007 -03	2008 -02	2009 -01	2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
			preproduction			production											
UNDERGROUND MINING LABOR COSTS																	
PERSONNEL																	
Mine Operations																	
Drill Operators		**7.0**								4.0	8.0	8.0	8.0				
Bolter Operators		**7.0**								4.0	8.0	8.0	8.0				
Bolter Helper		**7.0**								4.0	8.0	8.0	8.0				
Blaster		**4.0**									4.0	4.0	4.0				
Blaster Helper		**4.0**									4.0	4.0	4.0				
LHD - Ore		**7.0**								4.0	8.0	8.0	8.0				
LHD-waste		**4.0**									4.0	4.0	4.0				
Truck Driver		**13.0**								4.0	16.0	16.0	16.0				
Backfill Crew		**3.3**									2.0	4.0	4.0				
Grader Operator		**4.0**									4.0	4.0	4.0				
Mine Utilities		**2.0**									2.0	2.0	2.0				
Nipper		**4.0**								4.0	4.0	4.0	4.0				
subtotal		**16.3**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**24.0**	**72.0**	**74.0**	**74.0**	**0.0**	**0.0**	**0.0**	**0.0**
Maintenance																	
HD Mechanic		**4.0**								4.0	4.0	4.0	4.0				
Mechanic		**7.0**								4.0	8.0	8.0	8.0				
Fuel/Lube		**4.0**								4.0	4.0	4.0	4.0				
Welder		**4.0**								4.0	4.0	4.0	4.0				
PM Crew		**2.0**									2.0	2.0	2.0				
Electricians		**4.0**								4.0	4.0	4.0	4.0				
subtotal		**6.5**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**20.0**	**26.0**	**26.0**	**26.0**	**0.0**	**0.0**	**0.0**	**0.0**
U/G Operations		**85.5**								**44.0**	**98.0**	**100.0**	**100.0**				
COSTS																	
Mine Operations																	
Drill Operators	**US$000s**	**2,555**	0	0	0	0	0	0	0	365	730	730	730	0	0	0	0
Bolter Operators	**US$000s**	**2,497**	0	0	0	0	0	0	0	357	713	713	713	0	0	0	0
Bolter Helper	**US$000s**	**1,922**	0	0	0	0	0	0	0	275	549	549	549	0	0	0	0
Blaster	**US$000s**	**899**	0	0	0	0	0	0	0	0	300	300	300	0	0	0	0
Blaster Helper	**US$000s**	**824**	0	0	0	0	0	0	0	0	275	275	275	0	0	0	0
LHD - Ore	**US$000s**	**2,497**	0	0	0	0	0	0	0	357	713	713	713	0	0	0	0
LHD-waste	**US$000s**	**1,070**	0	0	0	0	0	0	0	0	357	357	357	0	0	0	0
Truck Driver	**US$000s**	**3,894**	0	0	0	0	0	0	0	300	1,198	1,198	1,198	0	0	0	0
Backfill Crew	**US$000s**	**749**	0	0	0	0	0	0	0	0	150	300	300	0	0	0	0
Grader Operator	**US$000s**	**1,095**	0	0	0	0	0	0	0	0	365	365	365	0	0	0	0
Mine Utilities	**US$000s**	**449**	0	0	0	0	0	0	0	0	150	150	150	0	0	0	0
Nipper	**US$000s**	**1,098**	0	0	0	0	0	0	0	275	275	275	275	0	0	0	0
subtotal	**US$000s**	**19,549**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1,927**	**5,774**	**5,924**	**5,924**	**0**	**0**	**0**	**0**
Maintenance																	
HD Mechanic	**US$000s**	**1,394**	0	0	0	0	0	0	0	348	348	348	348	0	0	0	0
Mechanic	**US$000s**	**2,097**	0	0	0	0	0	0	0	300	599	599	599	0	0	0	0
Fuel/Lube	**US$000s**	**1,098**	0	0	0	0	0	0	0	275	275	275	275	0	0	0	0
Welder	**US$000s**	**1,394**	0	0	0	0	0	0	0	348	348	348	348	0	0	0	0
PM Crew	**US$000s**	**523**	0	0	0	0	0	0	0	0	174	174	174	0	0	0	0
Electricians	**US$000s**	**1,198**	0	0	0	0	0	0	0	300	300	300	300	0	0	0	0
subtotal	**US$000s**	**7,703**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**1,570**	**2,044**	**2,044**	**2,044**	**0**	**0**	**0**	**0**
U/G Operations	**US$000s**	**27,252**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**3,498**	**7,818**	**7,968**	**7,968**	**0**	**0**	**0**	**0**

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
LABOR

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
PROCESS PERSONNEL																	
PERSONNEL																	
Staff																	
Plant Superintendent		**0.9**					0.5	1.0	1.0	1.0	1.0	1.0	1.0				
Plant Supervisor		**3.2**					0.5	2.0	4.0	4.0	4.0	4.0	4.0				
Metallurgist		**0.9**					0.5	1.0	1.0	1.0	1.0	1.0	1.0				
Chief Assayer		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Assayer/Sample Prep.		**1.6**			1.0	1.0	1.0	1.0	2.0	2.0	2.0	2.0	2.0				
Maintenance Foreman		**0.9**						0.5	1.0	1.0	1.0	1.0	1.0				
Maintenance Planner		**0.9**						0.5	1.0	1.0	1.0	1.0	1.0				
Mill Clerk		**0.9**						0.5	1.0	1.0	1.0	1.0	1.0				
Total Salaried		**8.3**			**2.0**	**2.0**	**3.5**	**7.5**	**12.0**	**12.0**	**12.0**	**12.0**	**12.0**				
Mill Operations																	
Primary Crusherman		**2.0**						1.5	1.5	1.5	1.5	3.0	3.0				
Grind/Leach/CIP Oper.		**2.7**						2.0	2.0	2.0	2.0	4.0	4.0				
Soln/Reagent/Gold Rm		**2.7**						2.0	2.0	2.0	2.0	4.0	4.0				
Plant Equip. Oper.		**0.7**						0.5	0.5	0.5	0.5	1.0	1.0				
Laborers		**3.3**						2.5	2.5	2.5	2.5	5.0	5.0				
Mechanics		**2.0**						1.0	1.0	1.0	1.0	4.0	4.0				
Electricians		**0.7**						0.5	0.5	0.5	0.5	1.0	1.0				
Instrumentmen		**0.7**						0.5	0.5	0.5	0.5	1.0	1.0				
		0.0						0.0									
Total Operations		**14.7**						**10.5**	**10.5**	**10.5**	**10.5**	**23.0**	**23.0**				
COSTS																	
Staff																	
Plant Superintendent	US$000s	**593**	0	0	0	0	46	91	91	91	91	91	91	0	0	0	0
Plant Supervisor	US$000s	**2,109**	0	0	0	0	47	188	375	375	375	375	375	0	0	0	0
Metallurgist	US$000s	**466**	0	0	0	0	36	72	72	72	72	72	72	0	0	0	0
Chief Assayer	US$000s	**617**	0	0	69	69	69	69	69	69	69	69	69	0	0	0	0
Assayer/Sample Prep.	US$000s	**914**	0	0	65	65	65	65	131	131	131	131	131	0	0	0	0
Maintenance Foreman	US$000s	**430**	0	0	0	0	0	39	78	78	78	78	78	0	0	0	0
Maintenance Planner	US$000s	**344**	0	0	0	0	0	31	63	63	63	63	63	0	0	0	0
Mill Clerk	US$000s	**239**	0	0	0	0	0	22	44	44	44	44	44	0	0	0	0
Total Salaried	US$000s	**5,712**	**0**	**0**	**134**	**134**	**262**	**576**	**921**	**921**	**921**	**921**	**921**	**0**	**0**	**0**	**0**
Mill Operations																	
Primary Crusherman	US$000s	**543**	0	0	0	0	0	68	68	68	68	136	136	0	0	0	0
Grind/Leach/CIP Oper.	US$000s	**940**	0	0	0	0	0	118	118	118	118	235	235	0	0	0	0
Soln/Reagent/Gold Rm	US$000s	**760**	0	0	0	0	0	95	95	95	95	190	190	0	0	0	0
Plant Equip. Oper.	US$000s	**181**	0	0	0	0	0	23	23	23	23	45	45	0	0	0	0
Laborers	US$000s	**905**	0	0	0	0	0	113	113	113	113	226	226	0	0	0	0
Mechanics	US$000s	**705**	0	0	0	0	0	59	59	59	59	235	235	0	0	0	0
Electricians	US$000s	**235**	0	0	0	0	0	29	29	29	29	59	59	0	0	0	0
Instrumentmen	US$000s	**235**	0	0	0	0	0	29	29	29	29	59	59	0	0	0	0
	US$000s	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Operations	US$000s	**4,504**	**0**	**0**	**0**	**0**	**0**	**534**	**534**	**534**	**534**	**1,185**	**1,185**	**0**	**0**	**0**	**0**

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
LABOR

Description	Units	LoM Total	2007 -03	2008 -02	2009 -01	2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
			preproduction			production											
ADMINSTRATIVE PERSONNEL																	
SALARIED																	
Staff																	
General Manager		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Human Resources Mgr		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Environmental Mgr		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Safety/Loss Control Mgr.		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Controller		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Purchasing Manager		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Systems Analyst		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
		0.0			0.0												
Total Salaried		**7.0**			**7.0**	**7.0**	**7.0**	**7.0**	**7.0**	**7.0**	**7.0**	**7.0**	**7.0**				
HOURLY																	
G&A Operations																	
HR Assistant		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Secretary		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Accountant		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Accounting Clerks		**1.0**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0				
Purchasing Agent		**2.0**			2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0				
Expeditier		**0.9**			1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.5				
Security Guard		**3.8**			4.0	4.0	4.0	4.0	4.0	4.0	4.0	3.0	3.0				
Janitor		**2.0**			2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0	2.0				
		0.0			0.0												
Total Operations		**12.7**			**13.0**	**13.0**	**13.0**	**13.0**	**13.0**	**13.0**	**13.0**	**12.0**	**11.5**				
ADMINISTRATIVE LABOR COSTS																	
SALARIED																	
Mine Operations																	
General Manager	US$000s	**1,181**	0	0	131	131	131	131	131	131	131	131	131	0	0	0	0
Human Resources Mgr	US$000s	**844**	0	0	94	94	94	94	94	94	94	94	94	0	0	0	0
Environmental Mgr	US$000s	**844**	0	0	94	94	94	94	94	94	94	94	94	0	0	0	0
Safety/Loss Control Mgr.	US$000s	**675**	0	0	75	75	75	75	75	75	75	75	75	0	0	0	0
Controller	US$000s	**788**	0	0	88	88	88	88	88	88	88	88	88	0	0	0	0
Purchasing Manager	US$000s	**675**	0	0	75	75	75	75	75	75	75	75	75	0	0	0	0
Systems Analyst	US$000s	**563**	0	0	63	63	63	63	63	63	63	63	63	0	0	0	0
	US$000s	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Salaried	US$000s	**5,568.8**	**0.0**	**0.0**	**618.8**	**618.8**	**618.8**	**618.8**	**618.8**	**618.8**	**618.8**	**618.8**	**618.8**	**0.0**	**0.0**	**0.0**	**0.0**
HOURLY																	
G&A Operations																	
HR Assistant	US$000s	**506**	0	0	56	56	56	56	56	56	56	56	56	0	0	0	0
Secretary	US$000s	**450**	0	0	50	50	50	50	50	50	50	50	50	0	0	0	0
Accountant	US$000s	**596**	0	0	66	66	66	66	66	66	66	66	66	0	0	0	0
Accounting Clerks	US$000s	**360**	0	0	40	40	40	40	40	40	40	40	40	0	0	0	0
Purchasing Agent	US$000s	**1,125**	0	0	125	125	125	125	125	125	125	125	125	0	0	0	0
Expeditier	US$000s	**340**	0	0	40	40	40	40	40	40	40	40	20	0	0	0	0
Security Guard	US$000s	**1,190**	0	0	140	140	140	140	140	140	140	105	105	0	0	0	0
Janitor	US$000s	**563**	0	0	63	63	63	63	63	63	63	63	63	0	0	0	0
	US$000s	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Operations	US$000s	**5,130.0**	**0.0**	**0.0**	**580.0**	**580.0**	**580.0**	**580.0**	**580.0**	**580.0**	**580.0**	**545.0**	**525.0**	**0.0**	**0.0**	**0.0**	**0.0**

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
MATERIAL BALANCE

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
MINE PRODUCTION SCHEDULE																	
ORE PRODUCTION																	
Ore																	
Begin Ore	kt	-	4,470.1	4,470.1	4,470.1	4,226.0	3,697.7	3,073.5	2,583.7	2,015.0	1,492.0	978.5	437.221	0.0	0.0	0.0	0.0
Open Pit	kt	3,362.4	0.0	0.0	244.1	528.3	624.2	489.8	568.7	523.0	384.3	0.0	0.0	0.0	0.0	0.0	0.0
Underground	kt	1,107.7	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	129.2	541.3	437.2	0.0	0.0	0.0	0.0
End Ore	kt	-	4,470.1	4,470.1	4,226.0	3,697.7	3,073.5	2,583.7	2,015.0	1,492.0	978.5	437.2	0.0	0.0	0.0	0.0	0.0
Gold Grade																	
Begin Ore	gpt	-	6.99	6.99	6.99	6.96	7.24	7.71	7.78	8.57	9.22	10.70	9.64	0.00	0.00	0.00	0.00
Open Pit	gpt	5.78	0.00	0.00	7.43	5.03	4.92	7.33	4.98	6.71	5.10	0.00	0.00	0.00	0.00	0.00	0.00
Underground	gpt	10.59	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	10.29	11.54	9.49	0.00	0.00	0.00	0.00
End Ore	gpt	-	6.99	6.99	6.96	7.24	7.71	7.78	8.57	9.22	10.70	9.64	0.00	0.00	0.00	0.00	0.00
Contained Gold																	
Begin Ore	koz	-	1,004.0	1,004.0	1,004.0	945.7	860.3	761.6	646.3	555.2	442.3	336.5	135.6	2.2	2.2	2.2	2.2
Open Pit	1.00	624.8	0.0	0.0	58.3	85.4	98.7	115.4	91.1	112.9	63.1	0.0	0.0	0.0	0.0	0.0	0.0
Underground	1.00	377.1	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	42.7	200.9	133.4	0.0	0.0	0.0	0.0
End Ore	koz	-	1,004.0	1,004.0	945.7	860.3	761.6	646.3	555.2	442.3	336.5	135.6	2.2	2.2	2.2	2.2	2.2
WASTE PRODUCTION																	
Open Pit Waste Production																	
Begin Waste	kt	-	46,940	46,940	46,940	38,259	29,403	21,990	14,302	6,443	2,652	0	0	0	0	0	0
Till	kt	8,810	0	0	5,342	54	1,628	1,786	0	0	0	0	0	0	0	0	0
Rock Waste	kt	38,130	0	0	3,339	8,802	5,785	5,903	7,859	3,792	2,652	0	0	0	0	0	0
Other Waste	kt	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
End Waste	kt	-	46,940	46,940	38,259	29,403	21,990	14,302	6,443	2,652	0	0	0	0	0	0	0
Underground Backfill Mgmt																	
Stope Volume	m3	345,423	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**44,553**	**161,429**	**139,441**				
Rock Fill	75%	233,161	0	0	0	0	0	0	0	0	30,073	108,965	94,123	0	0	0	0
Sand Fill	20%	62,176	0	0	0	0	0	0	0	0	8,020	29,057	25,099	0	0	0	0
Cemented Sand Fill	5%	15,544	0	0	0	0	0	0	0	0	2,005	7,264	6,275	0	0	0	0
Underground Development																	
Begin Development	m	-	0	0	0	0	0	0	0	0	1,680	5,740	10,123	11,975	11,975	11,975	11,975
Main Ramp	m	4,982	0	0	0	0	0	0	0	1,570	2,505	907	0	0	0	0	0
Special Ramp to 9805	m	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Level Drift	m	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Stope Access Drifts	m	4,278	0	0	0	0	0	0	0	0	565	2,371	1,342	0	0	0	0
Production Stope Ramps	m	1,470	0	0	0	0	0	0	0	0	360	600	510	0	0	0	0
Ventilation Raises	m	896	0	0	0	0	0	0	0	0	455	441	0	0	0	0	0
Muck Bays	m	349	0	0	0	0	0	0	0	110	175	63	0	0	0	0	0
End Development	m	-	0	0	0	0	0	0	0	1,680	5,740	10,123	11,975	11,975	11,975	11,975	11,975

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
MATERIAL BALANCE

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
WASTE PRODUCTION (continued)																	
Development Waste Mgmt			**Note: These values are loose cubic meters.**														
Begin	**m3**	-	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
FROM:																	
Main Ramp	**m3**	**186,825**	0	0	0	0	0	0	0	58,875	93,938	34,013	0	0	0	0	0
Special Ramp to 9805	**m3**	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Level Drift	**m3**	**0**	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Stope Access Drifts	**m3**	**102,672**	0	0	0	0	0	0	0	0	13,560	56,904	32,208	0	0	0	0
Production Stope Ramps	**m3**	**35,280**	0	0	0	0	0	0	0	0	8,640	14,400	12,240	0	0	0	0
Ventilation Raises	**m3**	**21,504**	0	0	0	0	0	0	0	0	10,920	10,584	0	0	0	0	0
Muck Bays	**m3**	**13,078**	0	0	0	0	0	0	0	4,121	6,576	2,381	0	0	0	0	0
TO:	0	Must = 0															
Backfill Stopes	**m3**	**183,486**	0	0	0	0	0	0	0	0	30,073	108,965	44,448	0	0	0	0
Muck Bays	50%	**87,936**	0	0	0	0	0	0	0	31,498	51,780	4,658	0	0	0	0	0
Surface Waste Dump	50%	**87,936**	0	0	0	0	0	0	0	31,498	51,780	4,658	0	0	0	0	0
End	**m3**	-	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Surface Dump to Backfill Stopes	**m3**	**49,675**	0	0	0	0	0	0	0	0	0	0	49,675	0	0	0	0
MILL PRODUCTION SCEDULE																	
ORE STOCKPILE																	
Ore Inventory																	
Begin Tons	**kt**		0	0	0	19	7	92	41	70	53	27	28	0	0	0	0
RoM Ore	**kt**	**4,470**	0	0	244	528	624	490	569	523	513	541	437	0	0	0	0
Toll Mill	**kt**	**1,305**	0	0	225	540	540	0	0	0	0	0	0	0	0	0	0
Owner Mill	**kt**	**3,165**	0	0	0	0	0	540	540	540	540	540	465	0	0	0	0
End Tons	**kt**		0	0	19	7	92	41	70	53	27	28	0	0	0	0	0
Stockpile Au Grade	**gpt**	**6.97**	0.00	0.00	7.43	5.11	4.92	6.95	5.11	6.53	6.42	11.30	9.60	0.00	0.00	0.00	0.00
Gold Inventory																	
Begin Inventory	**koz**		0	0	0	5	1	14	9	12	11	5	10	0	0	0	0
RoM Gold	**koz**	**1,002**	0	0	58	85	99	115	91	113	106	201	133	0	0	0	0
Toll Mill Gold	**koz**	**228**	0	0	54	89	85	0	0	0	0	0	0	0	0	0	0
Owner Mill Gold	**koz**	**774**	0	0	0	0	0	121	89	113	111	196	144	0	0	0	0
End Inventory	**koz**		0	0	5	1	14	9	12	11	5	10	0	0	0	0	0
TOLL MILL																	
Ore Shipped																	
Ore	**kt**	**1,305**	**0**	**0**	**225**	**540**	**540**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Au Grade	**gpt**	**5.43**	0.00	0.00	7.43	5.11	4.92	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Contained	**koz**	**228**	0	0	54	89	85	0	0	0	0	0	0	0	0	0	0
Mill Recovery	1.00	**96%**	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%
Toll Mill Gold	**koz**	**219**	**0**	**0**	**52**	**85**	**82**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Tailings	**000m3**	**989**	0	0	170	409	409	0	0	0	0	0	0	0	0	0	0
Cumulative	*000m3*		*0*	*0*	*170*	*580*	*989*	*989*	*989*	*989*	*989*	*989*	*989*	*989*	*989*	*989*	*989*

Exhibit 17.1: LoM Plan

Apollo Gold
Black Fox
MATERIAL BALANCE

Description	Units	LoM Total	preproduction 2007 -03	2008 -02	2009 -01	production 2010 01	2011 02	2012 03	2013 04	2014 05	2015 06	2016 07	2017 08	2018 09	2019 10	2020 11	2021 12
OWNER MILL																	
Ore Processed																	
Ore	**kt**	**3,165**	**0**	**0**	**0**	**0**	**0**	**540**	**540**	**540**	**540**	**540**	**465**	**0**	**0**	**0**	**0**
Au Grade	**gpt**	**18.45**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Contained	**koz**	**774**	0	0	0	0	0	121	89	113	111	196	144	0	0	0	0
Mill Recovery	1.00	**96%**	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%	96.0%
Owner Mill Gold	**koz**	**743**	**0**	**0**	**0**	**0**	**0**	**116**	**85**	**109**	**107**	**188**	**138**	**0**	**0**	**0**	**0**
Tailings Inventory																	
Tailings Capacity	**000m3**	-	2,273	2,273	2,273	2,273	2,273	2,273	1,864	1,455	1,045	646	274	0	0	0	0
Tailings Produced	**000m3**	**2,398**	0	0	0	0	0	409	409	409	409	409	352	0	0	0	0
Tailings Used U/G	**000m3**	**78**	0	0	0	0	0	0	0	0	10	36	31	0	0	0	0
Remaining Capacity	**000m3**	-	2,273	2,273	2,273	2,273	2,273	1,864	1,455	1,045	646	274	(47)	0	0	0	0
Expansion Dam	**000m3**	**47**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**47**	**0**	**0**	**0**	**0**

18 Interpretation and Conclusions

18.1 Interpretation

The Black Fox deposit has been adequately drill tested to estimate grade and tonnes classified as indicated and inferred resources. The resource model contains 1,818 drillholes drilled from surface and underground totaling to 322,744m. QA/QC procedures were reviewed by an outside consultant and determined to be appropriate. Mineralization occurs as two main ore types. "Flow Ore" shows good geologic and grade continuity forming distinct lens shaped bodies, in contrast to "Green Carbonate Ore" which occurs as discontinuous stockwork zones defining smaller pods of mineralization. Each of these ore types have been modeled with unique estimation procedures. An ID3 estimation technique was chosen as most appropriate for this deposit. Estimation results have defined an indicated resource at a 1g/t-Au cut-off, of 7.6Mt with an average grade of 5.8g/t-Au containing 1.4Moz of gold. In addition, the estimation has defined an inferred resource at a 1g/t-Au cut-off, of 6.5Mt with an average grade of 4.7g/t-Au containing 1Moz of gold. Portions of this resource could be mined by open pit and the remainder would be mined underground.

18.2 Conclusions

Apollo has progressed Black Fox to the point of confirming the gold ore resources and reserves as verified by the current SRK independent NI 43-101 report. Apollo should continue to build on the previous owners mining and known recovery of over 200,000oz of gold as well as the current ore reserves that could be recovered using proven open pit and underground mining techniques. The previous mining as well as known process systems that have been proven at Black Fox and other mines in the area demonstrates further justification to proceed with the project.

The Timmins district is a proven gold mining area where the permitting and environmental considerations are well known and understood. These factors as well as the favorable gold market should all be considered by Apollo as justification to proceed with the project.

As illustrated by Table 18.2.1 the favorable economics analysis using a gold price of US$525/oz further justifies Black Fox as a viable gold mining project.

Table 18.2.1: Black Fox Economic Summary

Item	
Gold Price (US$/oz)	**525**
NPV @ 0% (US$000s)	**156,300**
NPV @ 4% (US$000s)	**103,500**
NPV @ 8% (US$000s)	**68,300**
Internal Rate of Return	**33%**

19 Recommendations

Black Fox should continue to be developed to the feasibility level. The following recommendations for the project should be considered by Apollo:

- Continue with the advanced feasibility level studies for the project including commissioning the bankable feasibility project as soon as possible;
- Continue to core drill specific areas of the ore body to further upgrade and extend the geological modeling for the project;
- Update Mineral Resources and Mineral Reserves to reflect preceding drilling;
- Feasibility level tailing impoundment design,
- Feasibility level detailed design for open pit and underground,
- Feasibility level geotechnical review for the open pit and underground project,
- Complete detailed process design based on flowsheet;
- Feasibility level process design and detail costs, and
- Develop a detailed trade off report on viability of using the toll mill as compared to the onsite mill.
- Continue the permitting process for Black Fox. Consideration should be given to permit the mine in phases with Phase 1 to include the toll mill proposal and permit only the open pit, overburden, and waste rock stockpiles.

Estimated cost for these recommendations is US$1.5million.

20 References

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21 Glossary

21.1 Mineral Resources and Reserves

21.1.1 Mineral Resources

The mineral resources and mineral reserves have been classified according to the "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (November 2005). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

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

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

21.1.2 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

21.2 Glossary

Table 21.2.1: Definitions of Terms

Term	Definition
Assay:	The chemical analysis of mineral samples to determine the metal content.
AQ	A letter name specifying the dimensions of bits, core barrels, and drill rods in the A-size and Q-series wireline diamond drilling system having a core diameter of 27 mm and a hole diameter of 48 mm.
BQ Size:	Letter name specifying the dimensions of bits, core barrels, and drill rods in the B-size and Q-group wireline diamond drilling system having a core diameter of 36.5mm and a hole diameter of 60mm.
Capital Expenditure:	All other expenditures not classified as operating assets.
Composite:	Combining more than one sample result to give an average result over a larger distance.
Concentrate:	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the ore.
Crushing:	Initial process of reducing ore particle size to render it more amenable for further processing.
Cut-off Grade (CoG):	The grade of mineralized rock, which determines as to whether or not it is economic to recover its gold content by further concentration.
Dilution:	Waste, which is unavoidably mined with ore.
Dip:	Angle of inclination of a geological feature/rock from the horizontal.
Fault:	The surface of a fracture along which movement has occurred.
Flow Ore:	A medium to fined grained, basal mafic volcanic rock which is generally located along the footwall of the deposit.
Footwall:	The underlying side of an ore body or stope.
Grade:	The measure of concentration of gold within mineralized rock.
Hangingwall:	The overlying side of an ore body or slope.
Level:	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological:	Geological description pertaining to different rock types.
Mineral/Mining Lease:	A lease area of which mineral rights are held.
Mining Assets:	The Material Properties and Significant Exploration Properties.
NQ Size:	A letter name specifying the dimensions of bits, core barrles, and drill rods in the N-size and Q-group wireline diamond drilling system having a core diameter of 47.6mm and a hole diameter of 75.7mm.
Ongoing Capital:	Capital estimates of a routine nature which is necessary for sustaining operations.
Operating Costs:	Sum of cost of mining, beneficiation, and administration gives the operating cost of the mine.
Ore Reserve:	See Mineral Reserve.
Sedimentary:	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Sill:	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Specific Gravity:	The weight of a substance compared with the weight of an equal volume of pure water at 4ºC.
Stope:	Underground void created by mining.
Strike:	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulfide	A sulfur bearing mineral.
Total Expenditure:	All expenditures including those of an operating and capital nature.
Variogram:	A statistical representation of the characteristics (usually grade)

Abbreviations

The metric system has been used throughout this report unless otherwise stated. All currency is in U.S. dollars. Market prices are reported in US$ per troy oz of gold and silver. Tonnes are metric of 1,000kg, or 2,204.6lbs. The following abbreviations are used in this report.

Table 21.2.2: Abbreviations of Units and Terms

Abbreviation	Unit or Term
AA	Atomic Absorption
ABA	Acid Base Analysis
ADR	Adsorption-Desorption-Recovery
AG	Silver
amsl	Above mean sea level
AMEC	Association of Mining & Exploration Companies
ANFO	Ammonium Nitrate Fuel Oil (explosive)
Au	Gold
AUV	Ankerite Ultramafic
BMV	Bleached Mafic Volcanic
ºC	Degrees Centigrade
CDN$	Canadian dollar
CEAA	Canadian Environmental Assessment Act
cfm	Cubic feet per minute
CGR	Green Carbon Schists
CGY	Grey Carbonate
CIP	Carbon-in-pulp
cm	Centimeter
CM	Cubic Meter
CoG	Cut-off-Grade
Cu	Copper
CUV	Chlorite-talc Ultramafic
°	Degree (degrees)
dia.	Diameter
DPFZ	Destor-Porcupine Fault Zone
EA	Environmental Assessment
EGL	External Grinding Lengths
EIA	Environmental Impact Assessment
EPA	Environmental Protection Act
FI	Felsic Intrusive
ft	Foot (feet)
ft^2	Square Foot (feet)
g	Gram
gal	Gallon
g/hr	Grams per hour
g/L	Grams per Liter
g/yr	Grams per year
g/t	Grams Per Tonne
G&A	General & Administration
ha	Hectares
Hg	Mercury
hp	Horse Power
hr	Hour
IP	Induced Polarization
IRR	Internal Rate of Return
ISR	Inductive Source Resistivity
k	Thousand
km	Thousand Meters

Table 21.2.2: Abbreviations of Units and Terms (Continued)

Abbreviation	Unit or Term
koz	Thousand Troy Ounces
kt	Thousand Tonnes
kt/yr	Thousand Tonnes per Year
lb	Pound
LHD	Load Haul Dump
LoM	Life-of-Mine
m	Meter
MCC	Motor Control Center
MDA	Mine Development Associates
min	Minute
µ	Micron
mm	Millimeter
MNDM	Ministry of Northern Development and Mines
MNR	Ministry of Natural Resources
MOE	Ministry of the Environment
MOL	Ministry of Labor
MOU	Memorandum of Understanding
Moz	Million troy ounces
Mt	Million tonnes
MTO	Ministry of Transportation
MV	Mafic Volcanic
NaOH	Sodium hydroxide
NaCN	Sodium Cyanide
NGO	Non-government Organizations
NPV	Net Present Value
NSR	Net Smelter Return Royalty
O&M	Operating & Maintenance
OWRA	Ontario Water Resources Act
oz	Ounce
Pa	Pascal
Pb	Lead
PMV	Pillowed Mafic Volcanic
ppm	Parts per Million
psi	Pounds per square inch
PSR	Production Stope Ramp
%	Percent
QA/QC	Quality Assurance/Quality Control
RMSB	Ross Mine Syenitic Belt
RoM	Run-of-Mine
RPA	Scott Wilson Roscoe Postle Associates Inc.
SAD	Stope Access Drifts
SDW	Safe Drinking Water
SED	Lens of Greywacke
SUV	Silicified Grey Carbonate
t	Tonne (metric ton) (2,204.6 pounds)
t/day	Tonnes per day
t/hr	Tonnes per hour
t/yr	Tonnes per year
TUV	Talc Ultramafic
UTEM	University of Toronto ElectroMagnetometer
VAT	Value Added Tax
yr	Year

Appendix A

Certificates of Author

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CERTIFICATE of AUTHOR

I, David K. Young, PE do hereby certify that:

1. I am an Associate Senior Mining Engineer of:

   SRK Consulting (US), Inc.
   7175 W. Jefferson Ave, Suite 3000
   Denver, CO, USA, 80235

2. I graduated with a degree in Mining Engineering from the Colorado School of Mines in 1983

3. I am a Registered Professional Engineer in the State of Idaho, USA since 1991.

4. I have worked as a mining engineer for a total of 23 years since my graduation from the Colorado School of Mines.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for Section 17 and the overall preparation of the technical report titled NI 43-101 Prefeasibility, Apollo Gold Corporation, Black Fox, Timmins, Ontario, Canada and dated August 13, 2007 (the "Technical Report") relating to the Black Fox property. I visited the Black Fox property on various occasions from March 2004 thru October 2005 for over 30 days.

7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement was oversight of the technical aspects of the project as the Vice President of Business Development and Technical Services for Apollo Gold the owner of the Black Fox project during the period from March 2004 thru October 2005.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

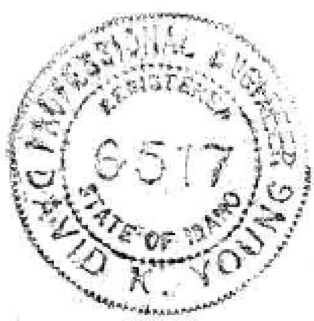
9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 13 August 2007.

David K. Young
(Signed)



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CERTIFICATE of AUTHOR

I, Bart A. Stryhas, PhD. CPG # 11034 do hereby certify that:

1. I am a Principal Resource Geologist of:

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   Denver, CO, USA, 80235

2. I graduated with a Doctorate degree in structural geology from Washington State University in 1988. In addition, I have obtained a Master of Science degree in structural geology from the University of Idaho in 1985 and a Bachelor of Arts degree in geology from the University of Vermont in 1983.

3. I am a current member of the American Institute of Professional Geologists.

4. I have worked as a geologist for a total of 19 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of Section 15 of the technical report titled NI 43-101 Prefeasibility, Apollo Gold Corporation, Black Fox, Timmins, Ontario, Canada, and dated August 13, 2007 (the "Technical Report") relating to the Black Fox property. I have not visited the property.

7. I have not had prior involvement with the property that is the subject of the Technical Report.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.[1]

Dated this 13 August 2007.

Bart A. Stryhas
(Signed)

CERTIFICATE of AUTHOR

I, *Richard F. Nanna*, Professional Geologist, do hereby certify that:

1. I am Senior Vice President of:

   Apollo Gold Exploration
   5655 S. Yosemite St., Ste 200
   Greenwood Village, CO 80111

2. I graduated with a Bachelor of Science – Earth Science, Degree and a Master Of Science-Geology, Degree, both from the University of *Akron* in December 1972 and December 1981 respectively.

3. I am a Registered Professional Geologist in the State of Washington.

4. I have worked as a geologist for a total of 30 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.

6. I am responsible for the preparation of all Sections 5 through 9 and 13 for the technical report titled NI 43-101 Prefeasibility, Apollo Gold Corporation, Black Fox, Timmins, Ontario, Canada, and dated August 13, 2007 (the "Technical Report") relating to the Black Fox property. I visited the Black Fox Property on 8/16/2006 for twelve days, 10/18/2006 for 4 days, 11/2/2006 for eight days, 12/6/06 for six days, 1/22/2007 for seven days, 4/26/2007 for seven days and 6/6/2007 for nine days

7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my prior involvement is: As Senior Vice President – Exploration and Development for Apollo Gold I am responsible for all exploration and development activities at Black Fox.

8. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

9. I am independent of the issuer applying all of the tests in section 1.5 of National Instrument 43-101.

10. I have read National Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.[1]

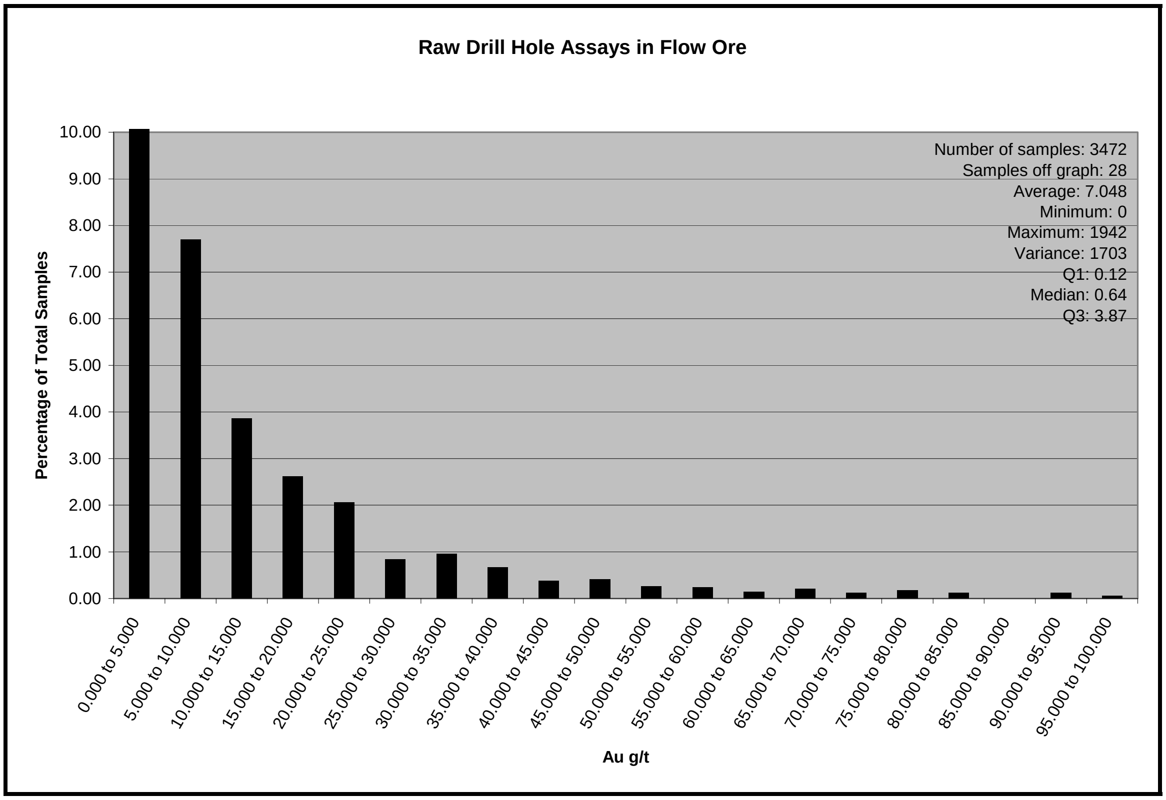
Dated this 13 August 2007.

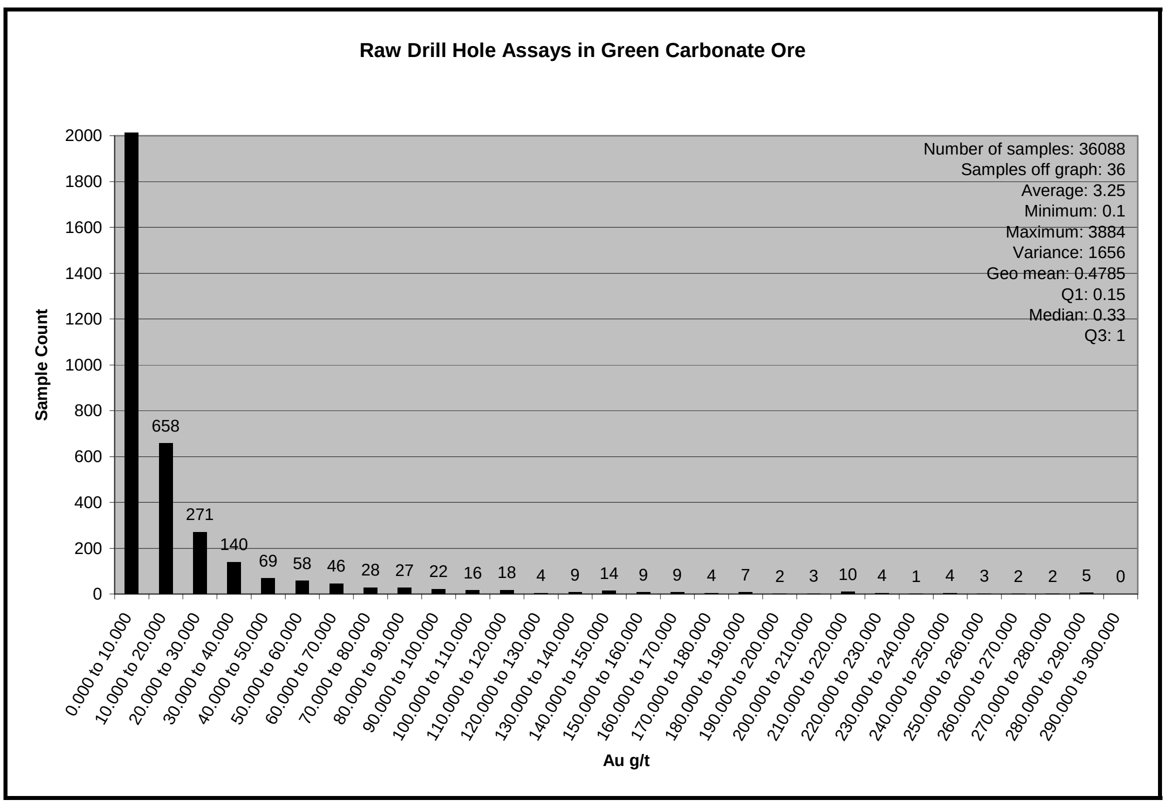
Richard F. Nanna
(Signed)

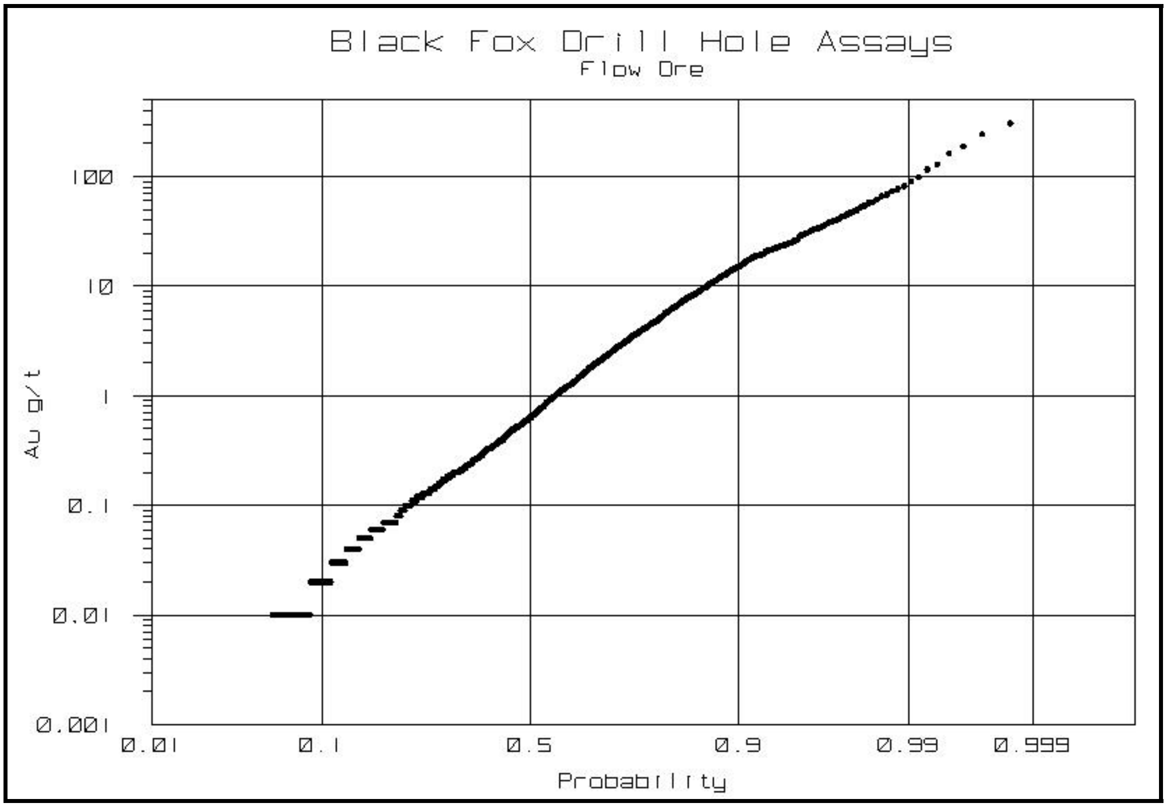
Certified Professional Geologist
State of Washington
Cert/Lic Number 1398
(Sealed)

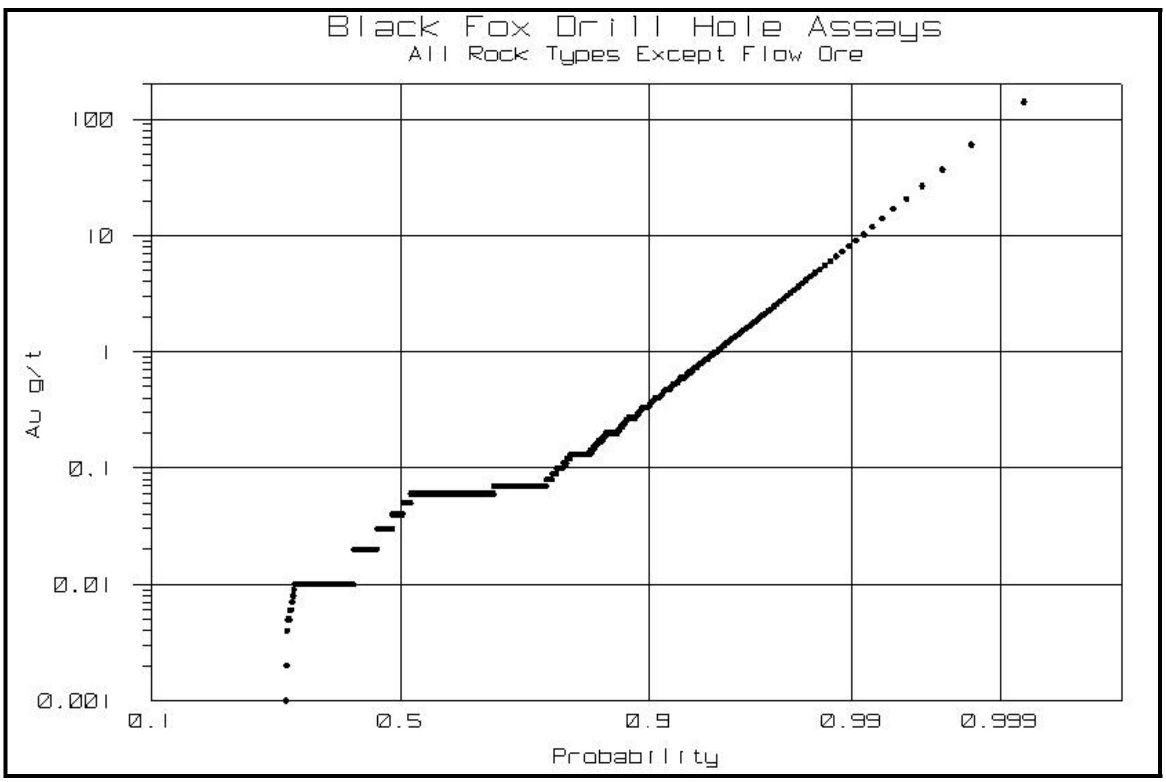
Appendix B

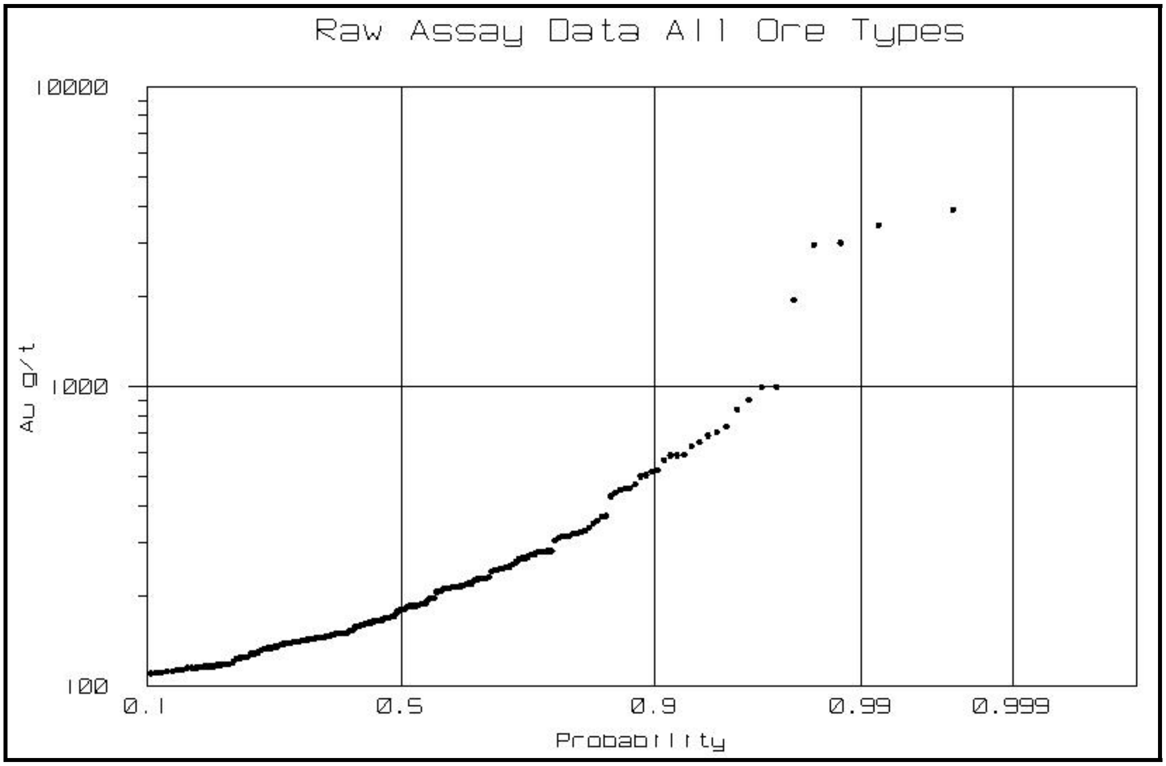
Basic Statistical Plots of Assay Data

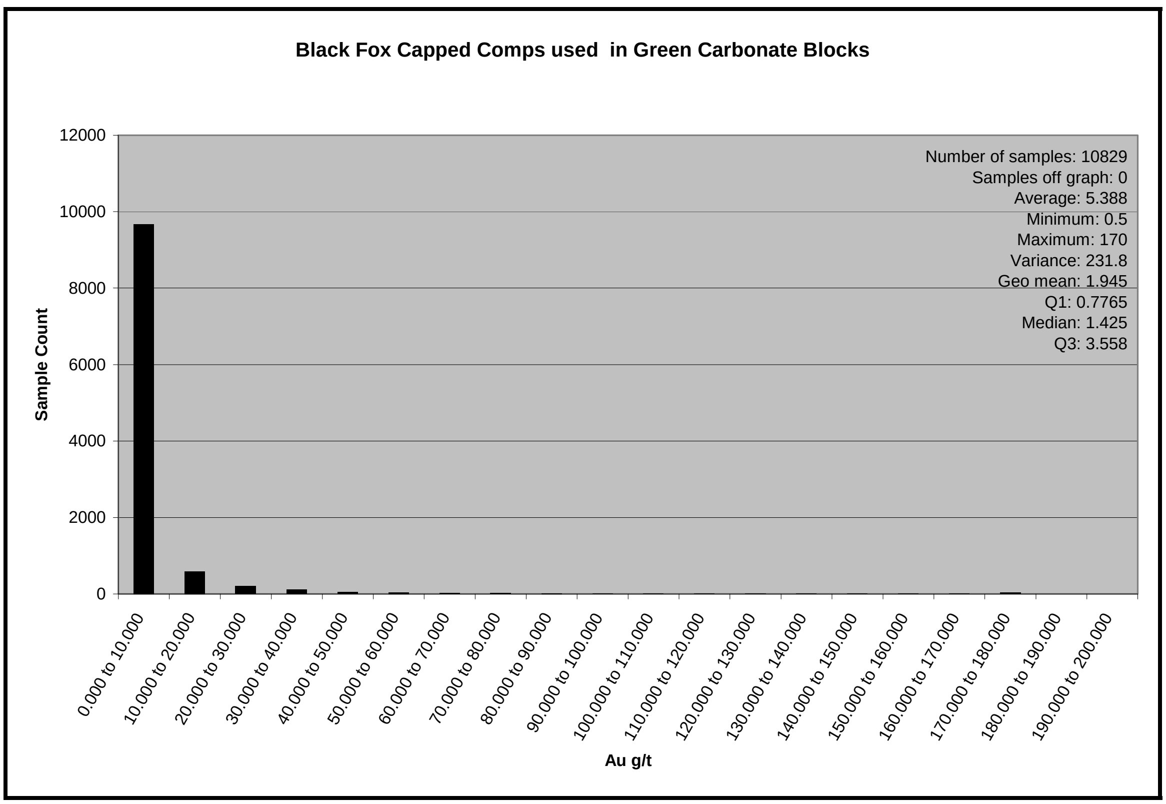

Raw Drill Hole Assays in Flow Ore
Number of samples: 3472
Samples off graph: 28
Average: 7.048
Minimum: 0
Maximum: 1942
Variance: 1703
Q1: 0.12
Median: 0.64
Q3: 3.87
Percentage of Total Samples
Au g/t
10.00
9.00
8.00
7.00
6.00
5.00
4.00
3.00
2.00
1.00
0.00
0.000 to 5.000
5.000 to 10.000
10.000 to 15.000
15.000 to 20.000
20.000 to 25.000
25.000 to 30.000
30.000 to 35.000
35.000 to 40.000
40.000 to 45.000
45.000 to 50.000
50.000 to 55.000
55.000 to 60.000
60.000 to 65.000
65.000 to 70.000
70.000 to 75.000
75.000 to 80.000
80.000 to 85.000
85.000 to 90.000
90.000 to 95.000
95.000 to 100.000

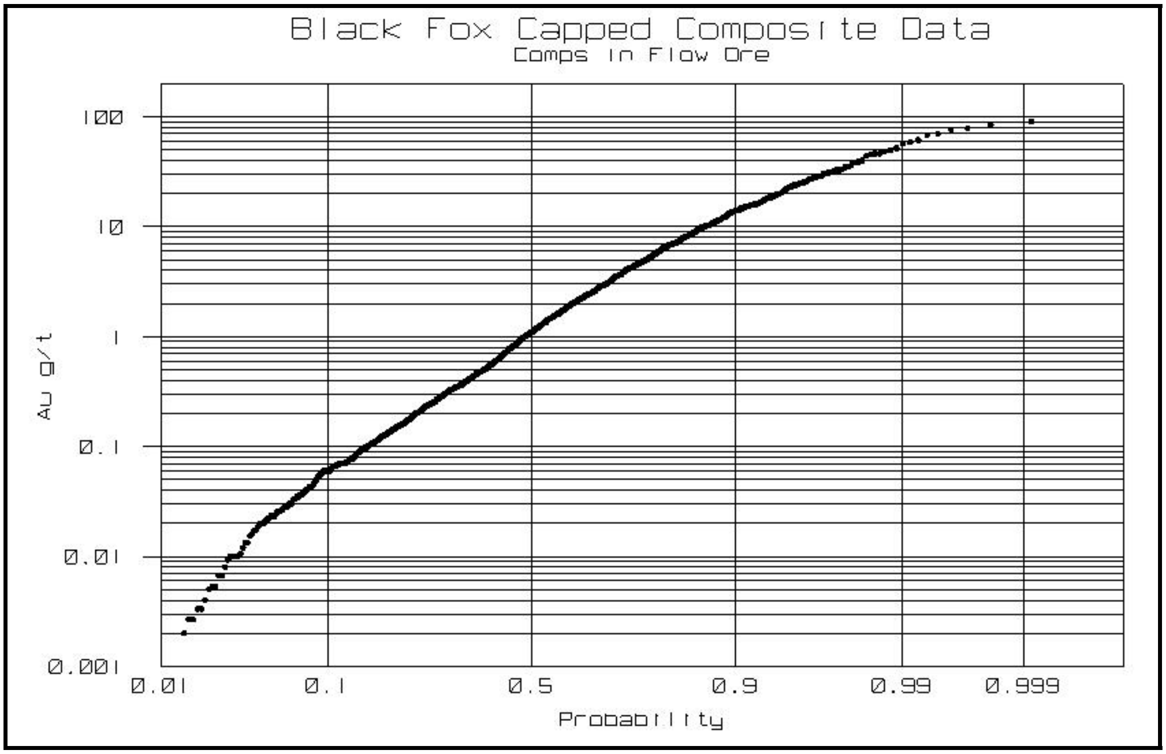

Raw Drill Hole Assays in Green Carbonate Ore
Number of samples: 36088
Samples off graph: 36
Average: 3.25
Minimum: 0.1
Maximum: 3884
Variance: 1656
Geo mean: 0.4785
Q1: 0.15
Median: 0.33
Q3: 1
Sample Count
Au g/t
2000
1800
1600
1400
1200
1000
800
600
400
200
0
658
271
140
69
58
46
28
27
22
16
18
4
9
14
9
9
4
7
2
3
10
4
1
4
3
2
2
5
0
0.000 to 10.000
10.000 to 20.000
20.000 to 30.000
30.000 to 40.000
40.000 to 50.000
50.000 to 60.000
60.000 to 70.000
70.000 to 80.000
80.000 to 90.000
90.000 to 100.000
100.000 to 110.000
110.000 to 120.000
120.000 to 130.000
130.000 to 140.000
140.000 to 150.000
150.000 to 160.000
160.000 to 170.000
170.000 to 180.000
180.000 to 190.000
190.000 to 200.000
200.000 to 210.000
210.000 to 220.000
220.000 to 230.000
230.000 to 240.000
240.000 to 250.000
250.000 to 260.000
260.000 to 270.000
270.000 to 280.000
280.000 to 290.000
290.000 to 300.000


Black Fox Drill Hole Assays
Flow Ore
Au g/t
100
10
1
0.1
0.01
0.001
0.01
0.1
0.5
0.9
0.99
0.999
Probability

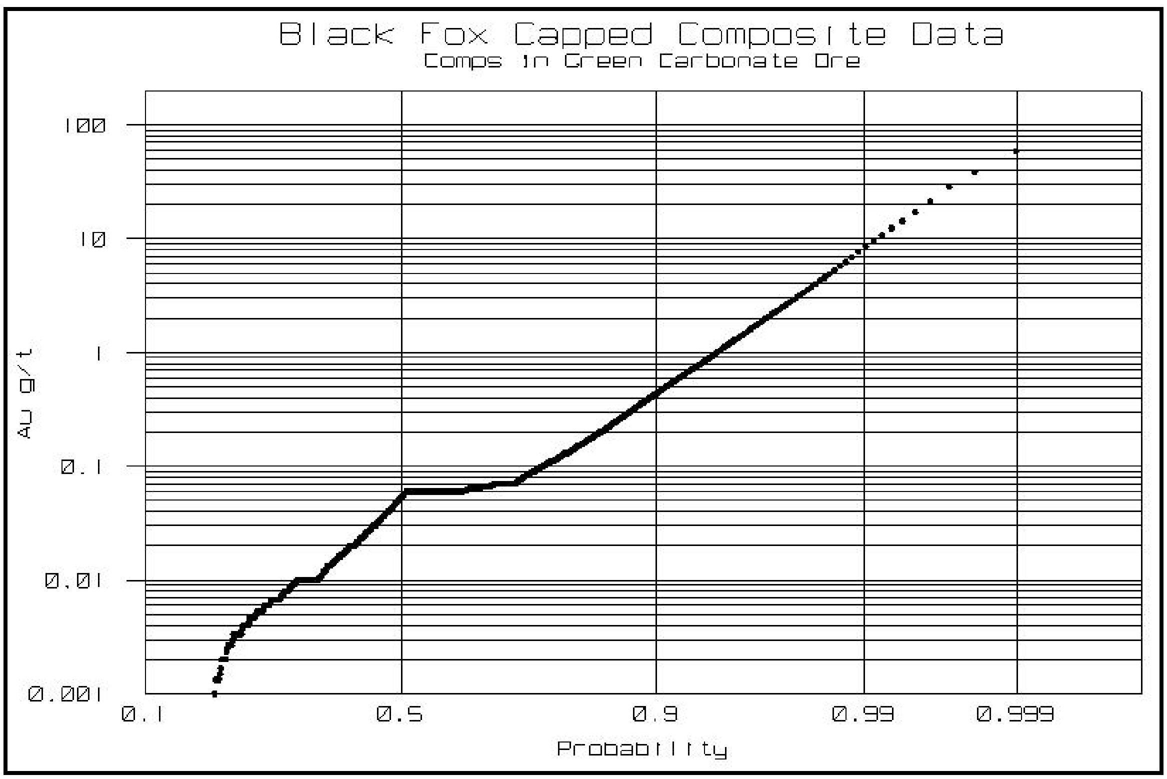

Black Fox Drill Hole Assays
All Rock Types Except Flow Ore
Au g/t
100
10
1
0.1
0.01
0.001
0.1
0.5
0.9
0.99
0.999
Probability


Raw Assay Data All Ore Types
10000
1000
100
Au g/t
0.1
0.5
0.9
0.99
0.999
Probability

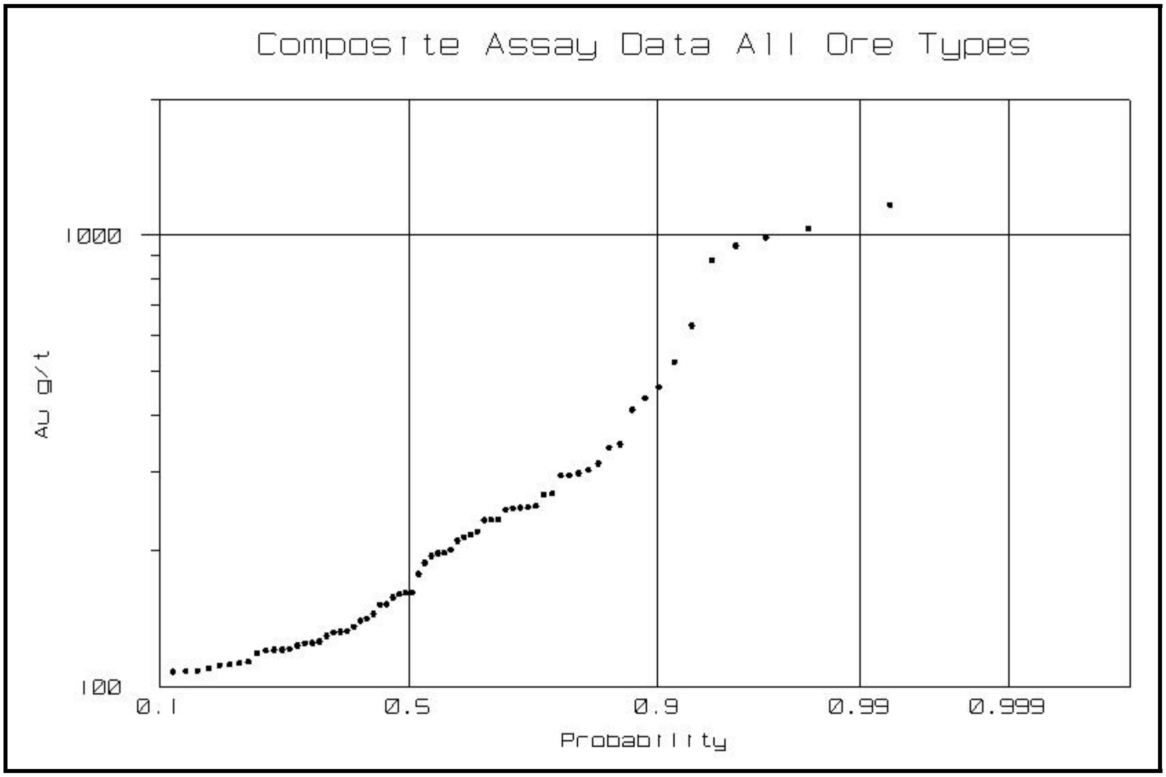

Black Fox Capped Comps used in Green Carbonate Blocks
12000
10000
8000
6000
4000
2000
0
Sample Count
Number of samples: 10829
Samples off graph: 0
Average: 5.388
Minimum: 0.5
Maximum: 170
Variance: 231.8
Geo mean: 1.945
Q1: 0.7765
Median: 1.425
Q3: 3.558
0.000 to 10.000
10.000 to 20.000
20.000 to 30.000
30.000 to 40.000
40.000 to 50.000
50.000 to 60.000
60.000 to 70.000
70.000 to 80.000
80.000 to 90.000
90.000 to 100.000
100.000 to 110.000
110.000 to 120.000
120.000 to 130.000
130.000 to 140.000
140.000 to 150.000
150.000 to 160.000
160.000 to 170.000
170.000 to 180.000
180.000 to 190.000
190.000 to 200.000
Au g/t



Black Fox Capped Composite Data
Comps in Flow Ore
100
10
1
0.1
0.01
0.001
Au g/t
0.01
0.1
0.5
0.9
0.99
0.999
Probability



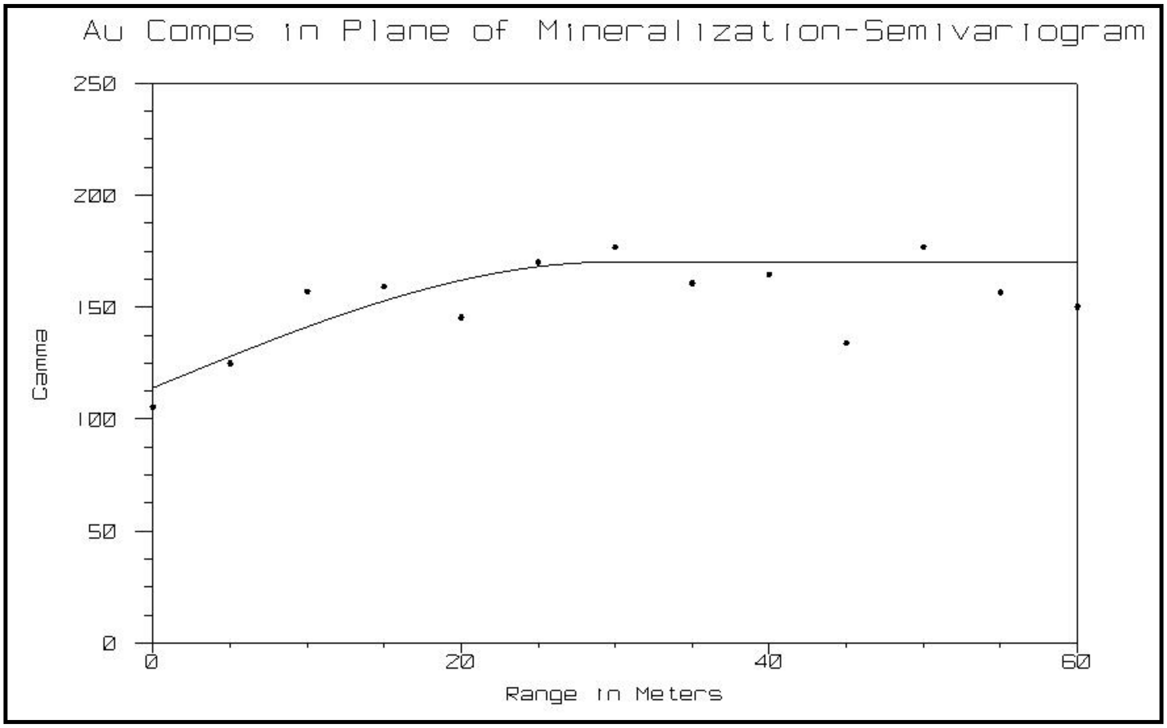
Black Fox Capped Composite Data
Comps in Green Carbonate Ore
100
10
1
0.1
0.01
0.001
Au g/t
0.1
0.5
0.9
0.99
0.999
Probability

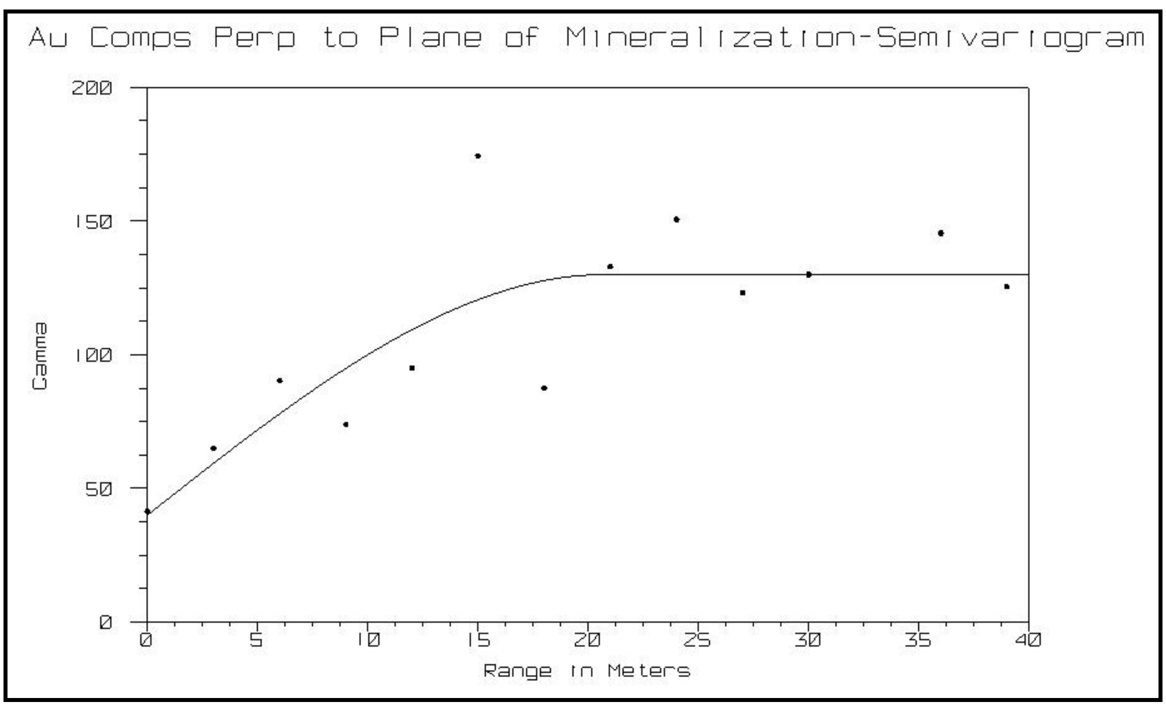


Composite Assay Data All Ore Types
1000
100
Au g/t
0.1
0.5
0.9
0.99
0.999
Probability

Appendix C

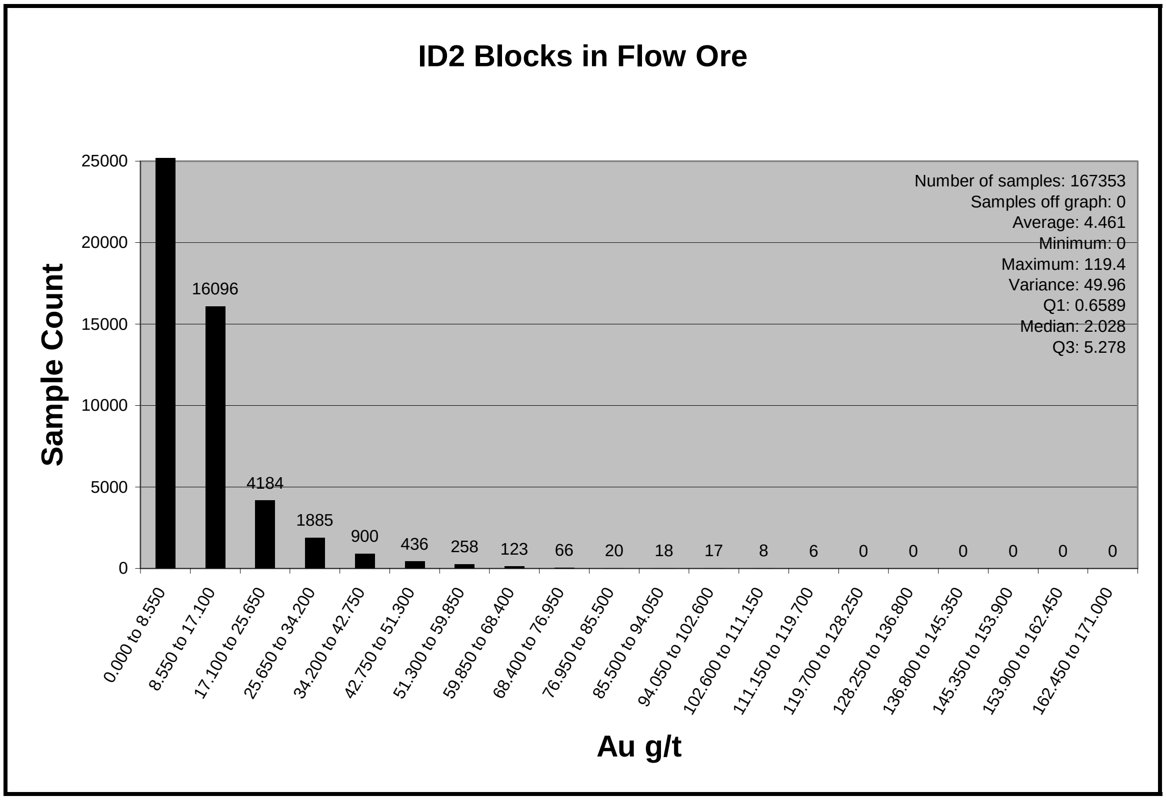
Composite Variograms


Au Comps in Plane of Mineralization-Semivariogram
250
200
150
100
50
0
Gamma
0
20
40
60
Range in Meters

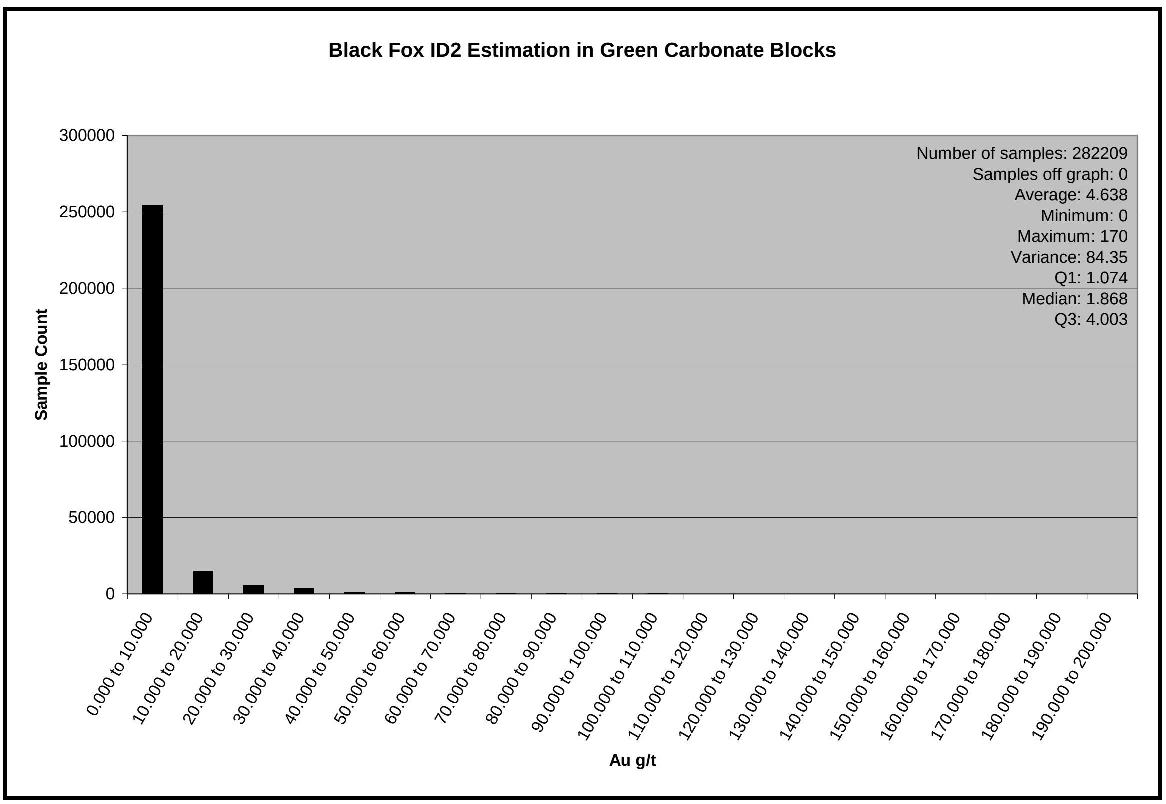

Au Comps Perp to Plane of Mineralization-Semivariogram
200
150
100
50
0
Gamma
0
5
10
15
20
25
30
35
40
Range in Meters

Appendix D

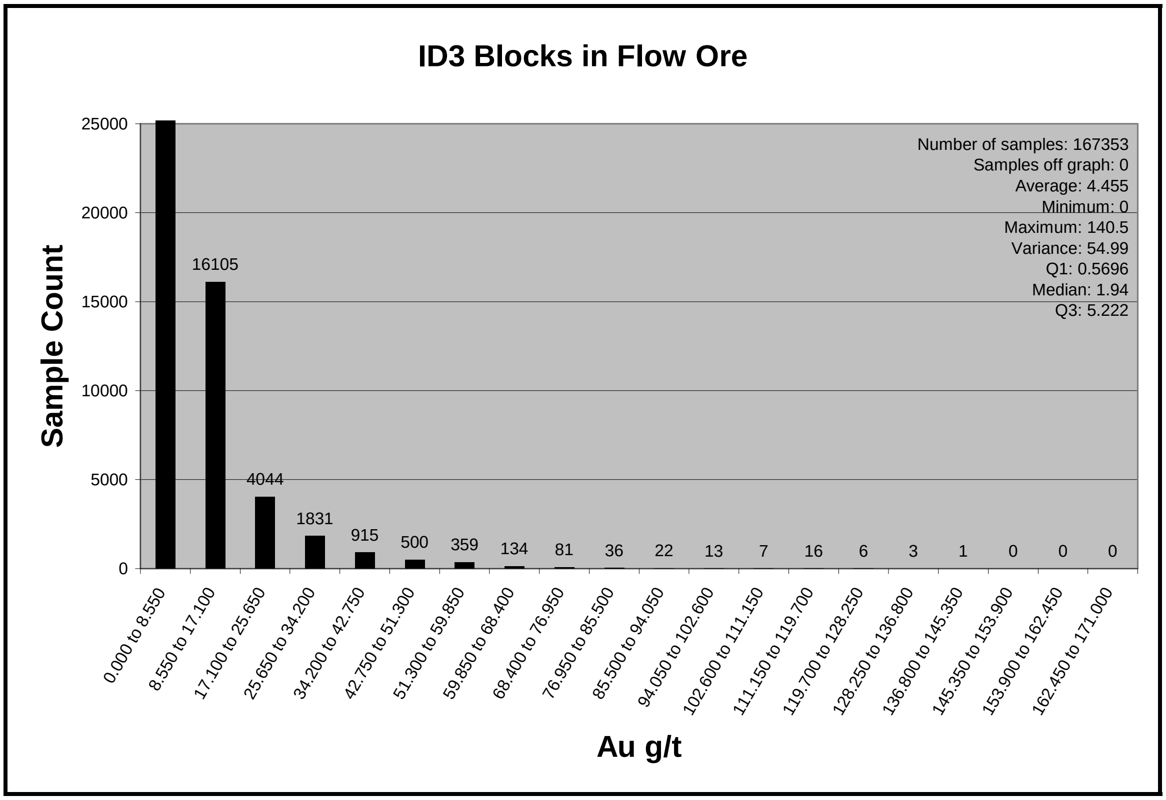
Basic Statistical Plots of Model Data

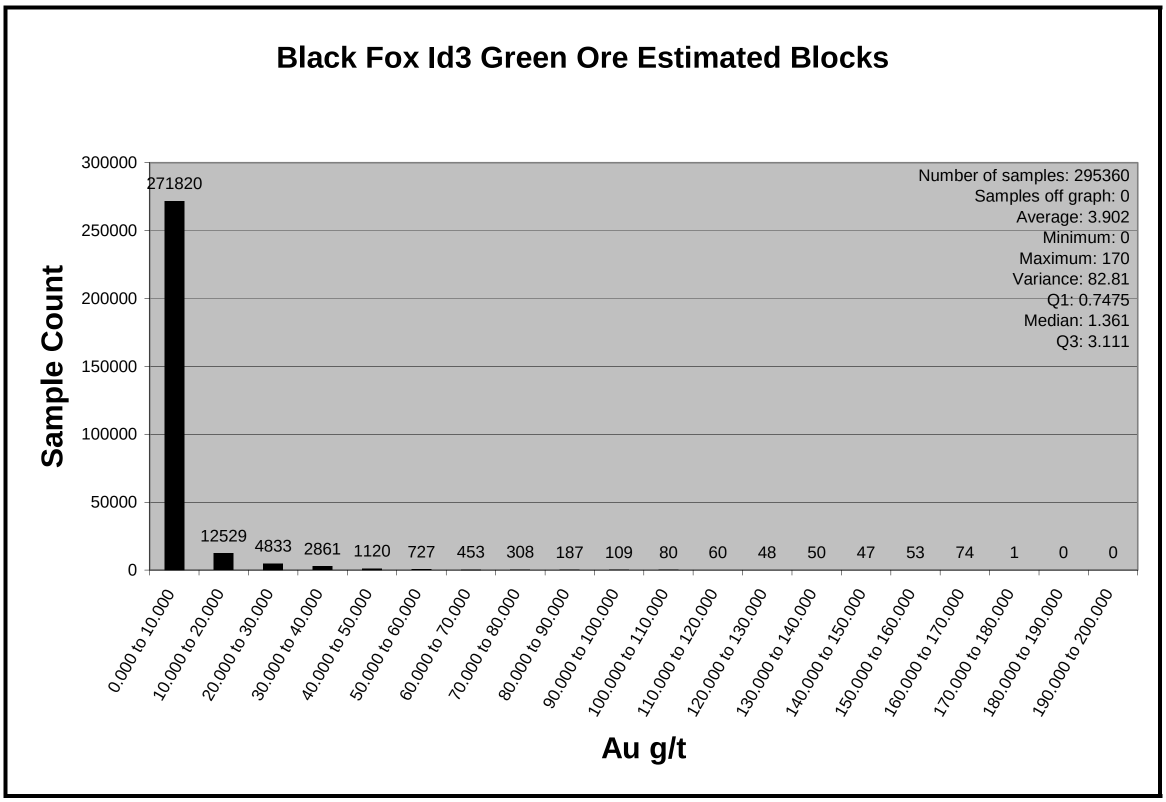


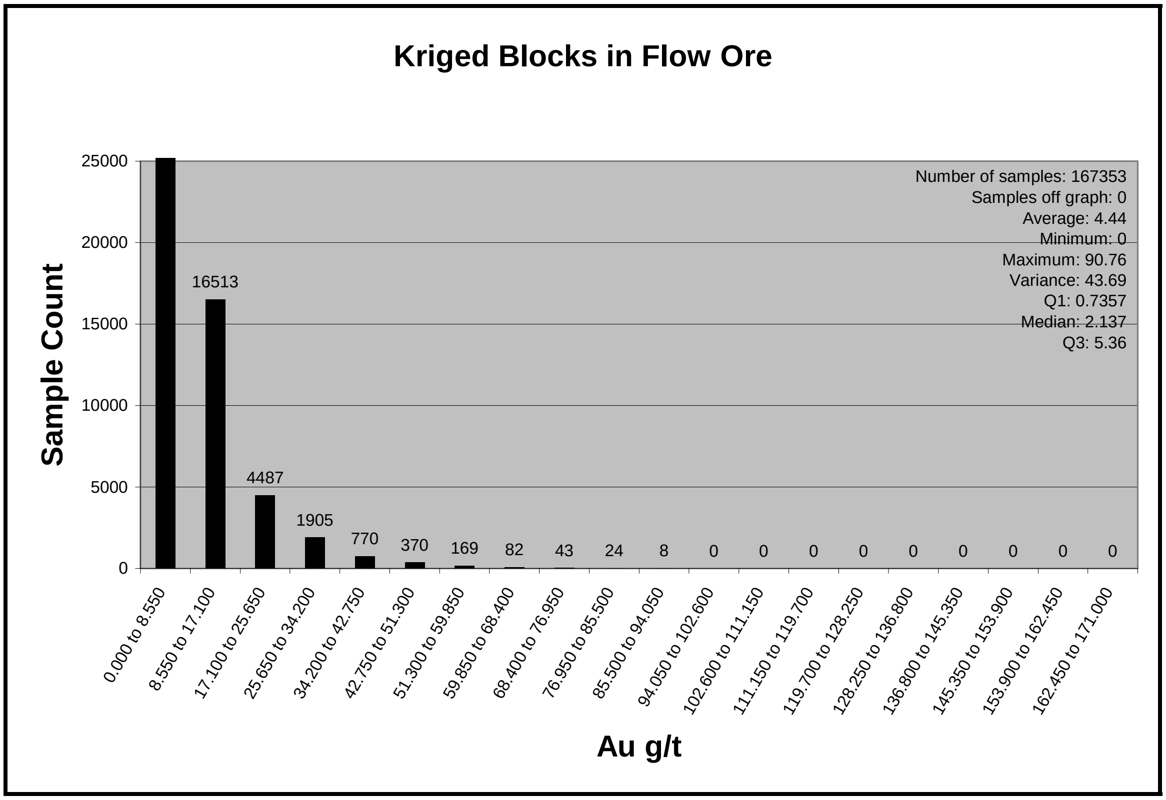
ID2 Blocks in Flow Ore
Sample Count
Au g/t
Number of samples: 167353
Samples off graph: 0
Average: 4.461
Minimum: 0
Maximum: 119.4
Variance: 49.96
Q1: 0.6589
Median: 2.028
Q3: 5.278
25000
20000
15000
10000
5000
0
16096
4184
1885
900
436
258
123
66
20
18
17
8
6
0
0
0
0
0
0
0.000 to 8.550
8.550 to 17.100
17.100 to 25.650
25.650 to 34.200
34.200 to 42.750
42.750 to 51.300
51.300 to 59.850
59.850 to 68.400
68.400 to 76.950
76.950 to 85.500
85.500 to 94.050
94.050 to 102.600
102.600 to 111.150
111.150 to 119.700
119.700 to 128.250
128.250 to 136.800
136.800 to 145.350
145.350 to 153.900
153.900 to 162.450
162.450 to 171.000



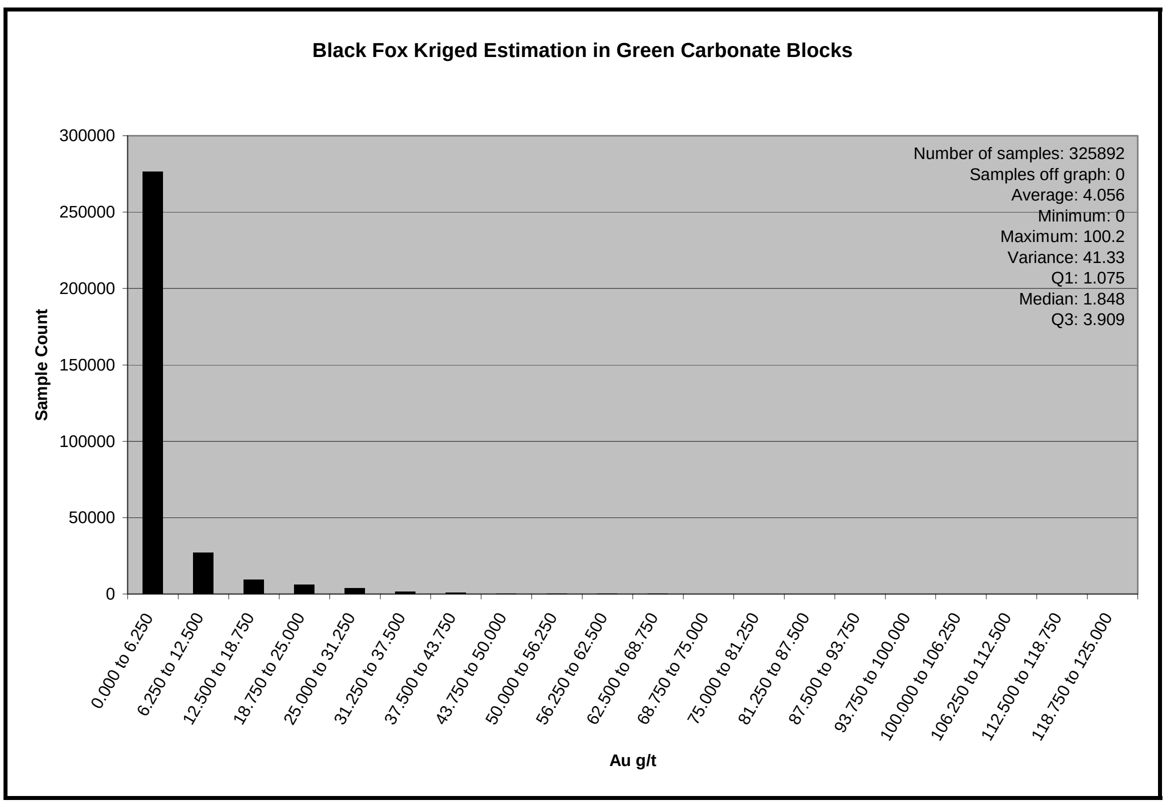
Black Fox ID2 Estimation in Green Carbonate Blocks
Sample Count
Au g/t
Number of samples: 282209
Samples off graph: 0
Average: 4.638
Minimum: 0
Maximum: 170
Variance: 84.35
Q1: 1.074
Median: 1.868
Q3: 4.003
300000
250000
200000
150000
100000
50000
0
0.000 to 10.000
10.000 to 20.000
20.000 to 30.000
30.000 to 40.000
40.000 to 50.000
50.000 to 60.000
60.000 to 70.000
70.000 to 80.000
80.000 to 90.000
90.000 to 100.000
100.000 to 110.000
110.000 to 120.000
120.000 to 130.000
130.000 to 140.000
140.000 to 150.000
150.000 to 160.000
160.000 to 170.000
170.000 to 180.000
180.000 to 190.000
190.000 to 200.000


ID3 Blocks in Flow Ore
Sample Count
Au g/t
Number of samples: 167353
Samples off graph: 0
Average: 4.455
Minimum: 0
Maximum: 140.5
Variance: 54.99
Q1: 0.5696
Median: 1.94
Q3: 5.222
25000
20000
15000
10000
5000
0
16105
4044
1831
915
500
359
134
81
36
22
13
7
16
6
3
1
0
0
0
0.000 to 8.550
8.550 to 17.100
17.100 to 25.650
25.650 to 34.200
34.200 to 42.750
42.750 to 51.300
51.300 to 59.850
59.850 to 68.400
68.400 to 76.950
76.950 to 85.500
85.500 to 94.050
94.050 to 102.600
102.600 to 111.150
111.150 to 119.700
119.700 to 128.250
128.250 to 136.800
136.800 to 145.350
145.350 to 153.900
153.900 to 162.450
162.450 to 171.000


Black Fox Id3 Green Ore Estimated Blocks
Sample Count
Au g/t
Number of samples: 295360
Samples off graph: 0
Average: 3.902
Minimum: 0
Maximum: 170
Variance: 82.81
Q1: 0.7475
Median: 1.361
Q3: 3.111
300000
250000
200000
150000
100000
50000
0
271820
12529
4833
2861
1120
727
453
308
187
109
80
60
48
50
47
53
74
1
0
0
0.000 to 10.000
10.000 to 20.000
20.000 to 30.000
30.000 to 40.000
40.000 to 50.000
50.000 to 60.000
60.000 to 70.000
70.000 to 80.000
80.000 to 90.000
90.000 to 100.000
100.000 to 110.000
110.000 to 120.000
120.000 to 130.000
130.000 to 140.000
140.000 to 150.000
150.000 to 160.000
160.000 to 170.000
170.000 to 180.000
180.000 to 190.000
190.000 to 200.000



Kriged Blocks in Flow Ore
Sample Count
Au g/t
Number of samples: 167353
Samples off graph: 0
Average: 4.44
Minimum: 0
Maximum: 90.76
Variance: 43.69
Q1: 0.7357
Median: 2.137
Q3: 5.36
16513
4487
1905
770
370
169
82
43
24
8
0

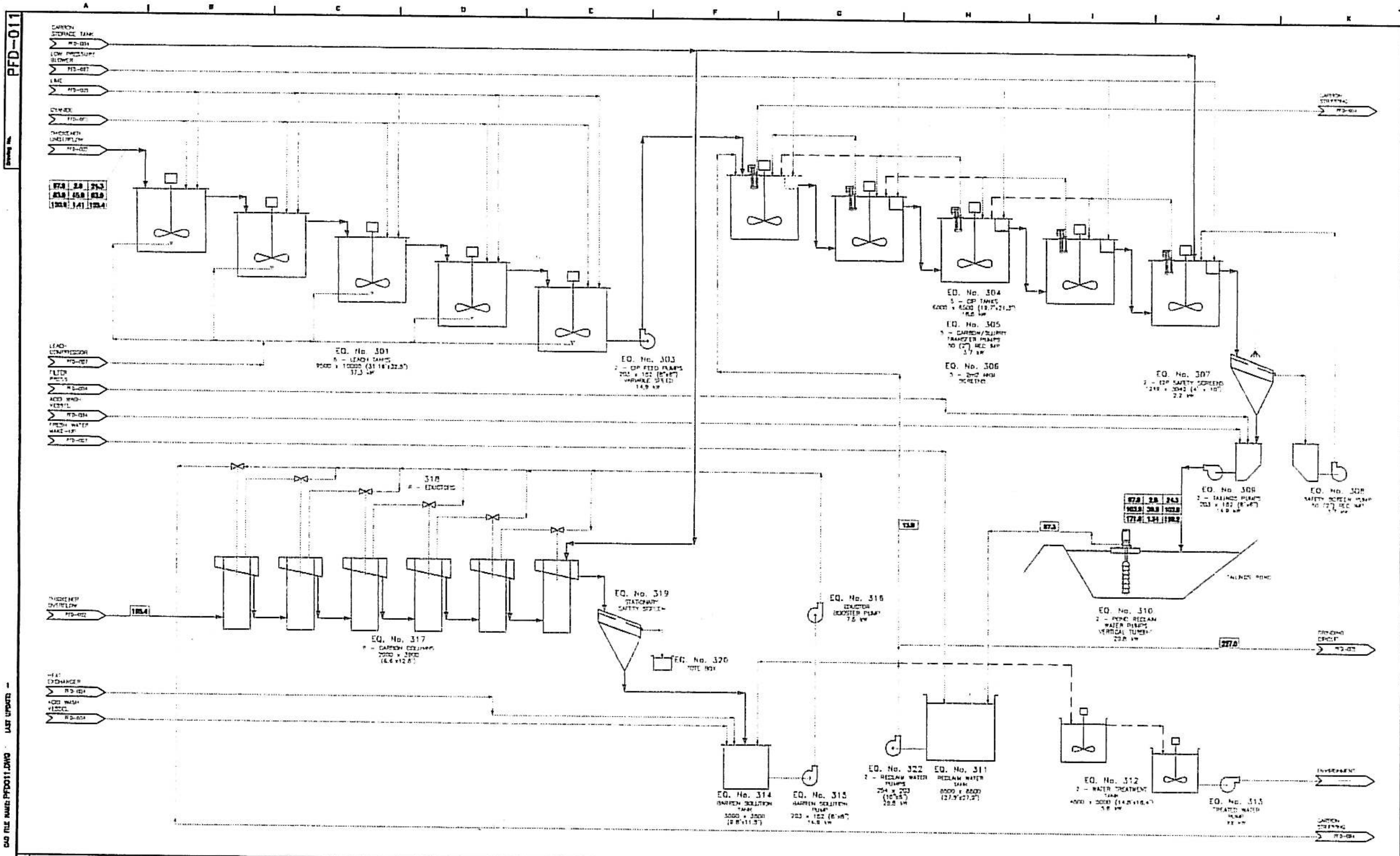


Black Fox Kriged Estimation in Green Carbonate Blocks
Sample Count
Au g/t
Number of samples: 325892
Samples off graph: 0
Average: 4.056
Minimum: 0
Maximum: 100.2
Variance: 41.33
Q1: 1.075
Median: 1.848
Q3: 3.909

Appendix E

Proposed Mill Drawings

(MDA 2006)



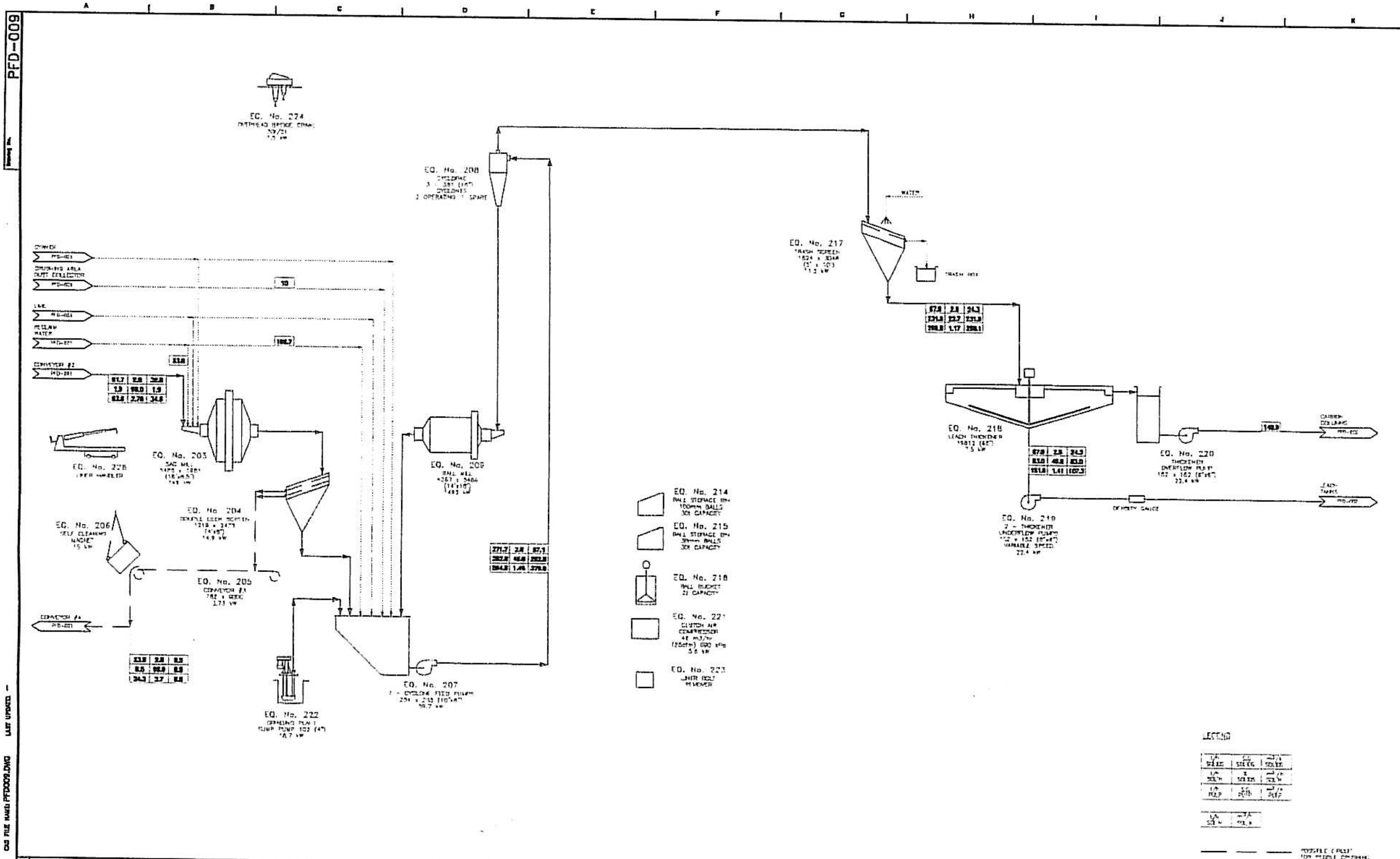
- PRELIMINARY -
FEASIBLITY STUDY

P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE - CASE 2
PROCESS FLOWSHEET #3
LEACH, C.I.P., C.I.C.

PFD-011 B

PFD-009

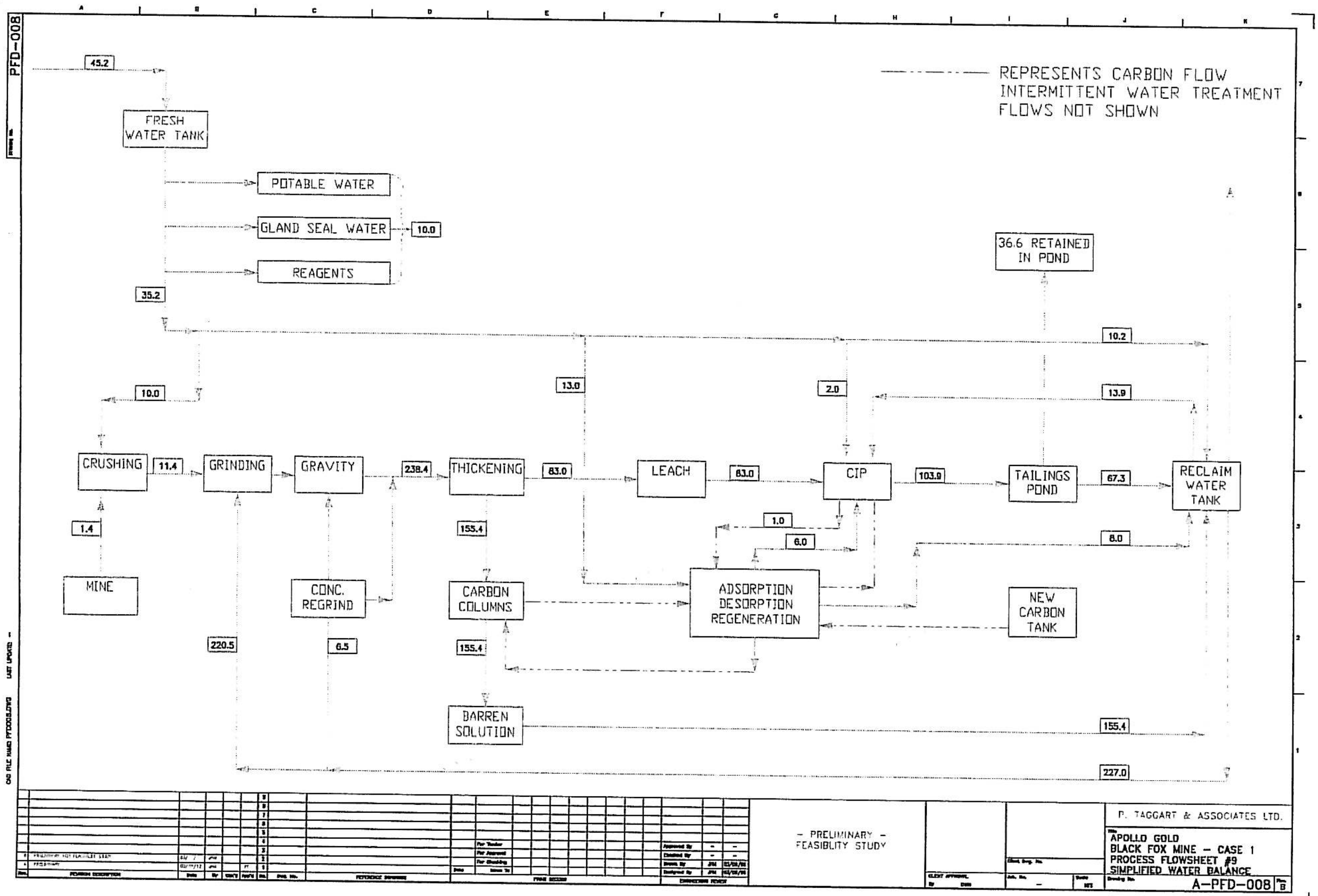


- PRELIMINARY -
FEASIBILITY STUDY

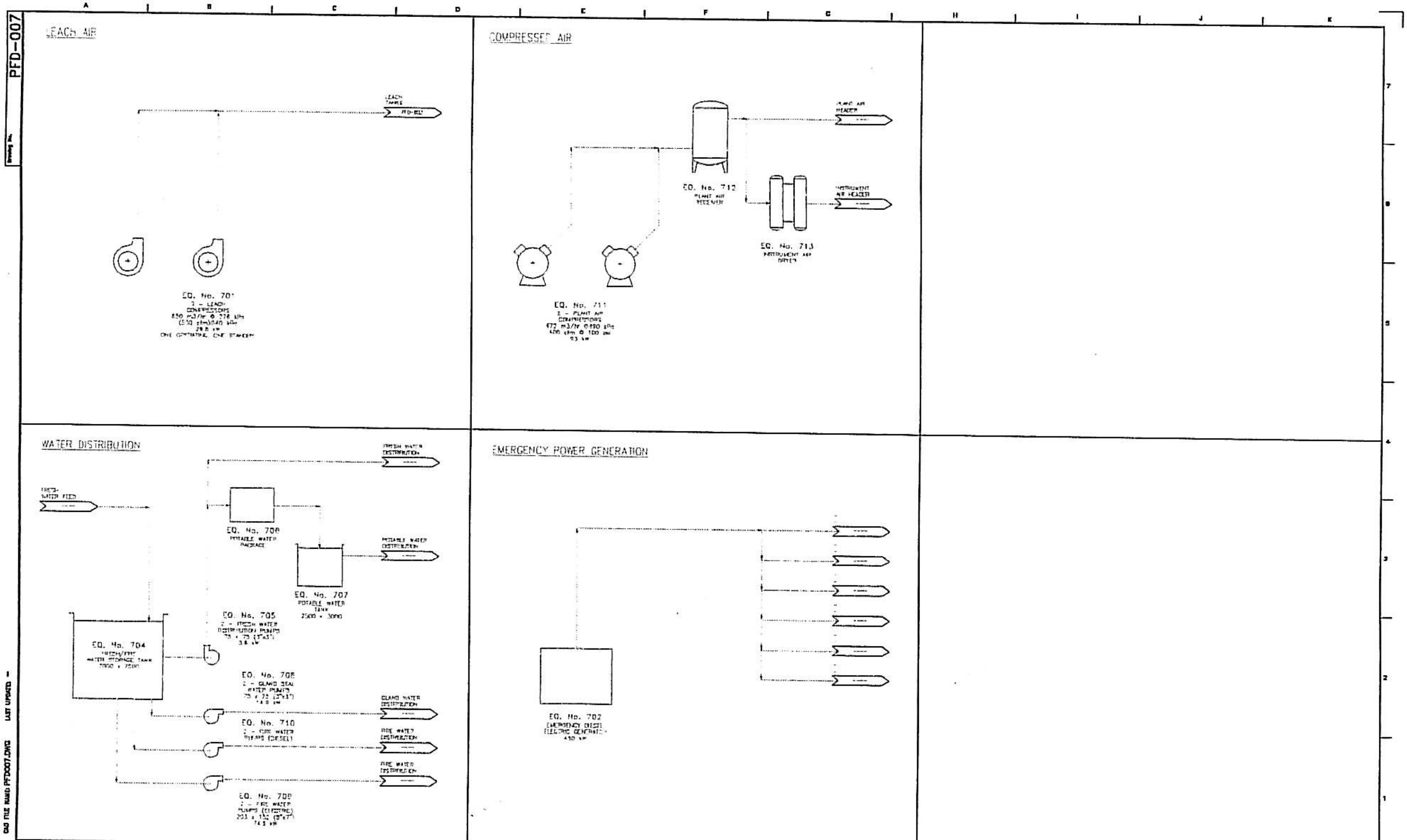
P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE – CASE 2
PROCESS FLOWSHEET #2
GRINDING

PFD-009 A



PFD-008
REPRESENTS CARBON FLOW
INTERMITTENT WATER TREATMENT FLOWS NOT SHOWN
45.2
FRESH WATER TANK
POTABLE WATER
GLAND SEAL WATER
REAGENTS
10.0
35.2
36.6 RETAINED IN POND
10.2
10.0
13.0
2.0
13.9
CRUSHING
11.4
GRINDING
GRAVITY
238.4
THICKENING
83.0
LEACH
83.0
CIP
103.9
TAILINGS POND
67.3
RECLAIM WATER TANK
1.4
1.0
6.0
8.0
155.4
MINE
CONC. REGRIND
CARBON COLUMNS
ADSORPTION DESORPTION REGENERATION
NEW CARBON TANK
220.5
6.5
155.4
BARREN SOLUTION
155.4
227.0
- PRELIMINARY - FEASIBLITY STUDY
P. TAGGART & ASSOCIATES LTD.
APOLLO GOLD
BLACK FOX MINE - CASE 1
PROCESS FLOWSHEET #9
SIMPLIFIED WATER BALANCE
A-PFD-008

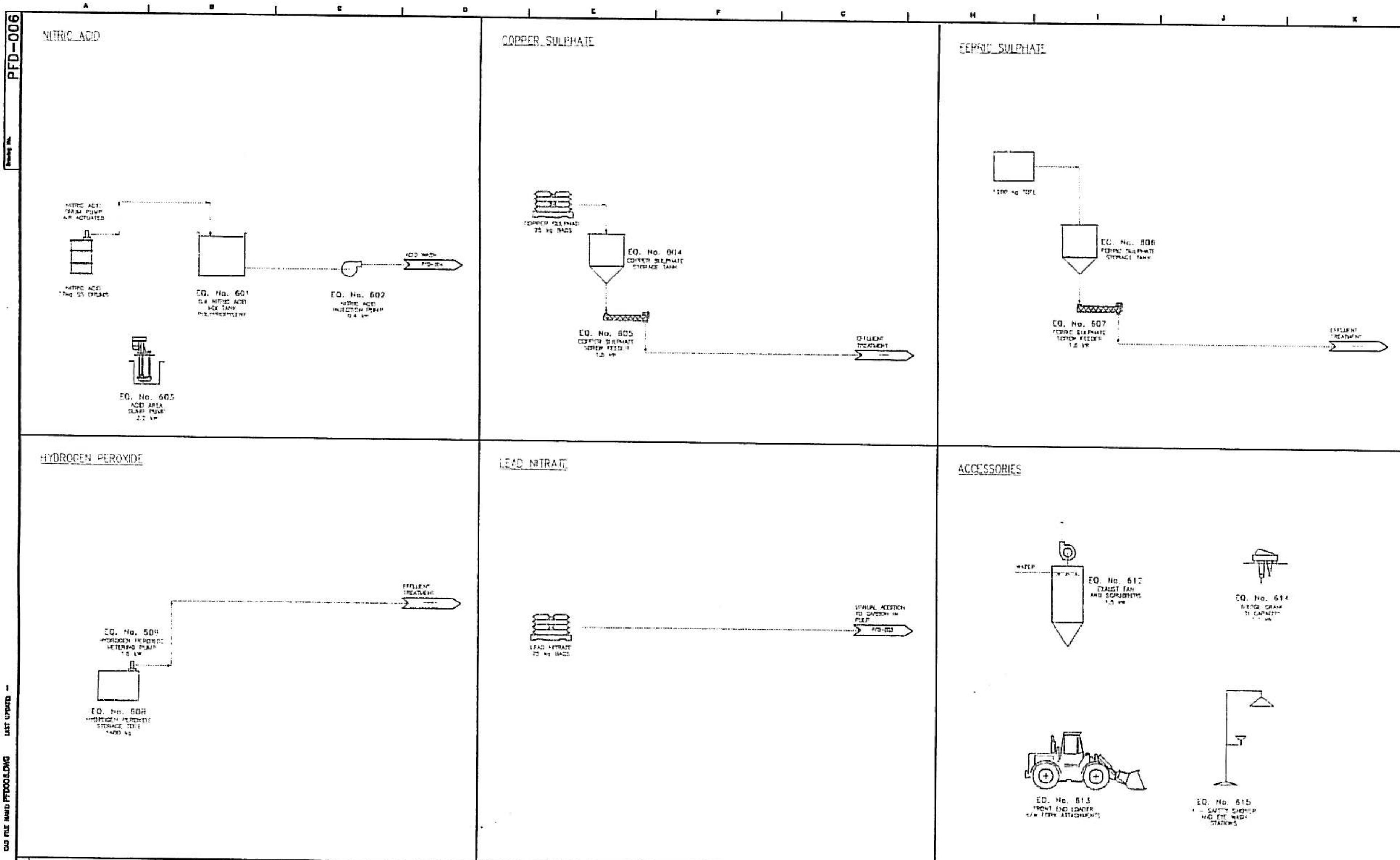


- PRELIMINARY -
FEASIBLITY STUDY

P. TAGGART & ASSOCIATES LTD

APOLLO GOLD
BLACK FOX MINE - CASE 2
PROCESS FLOWSHEET #7
SERVICES

PFD-007 B

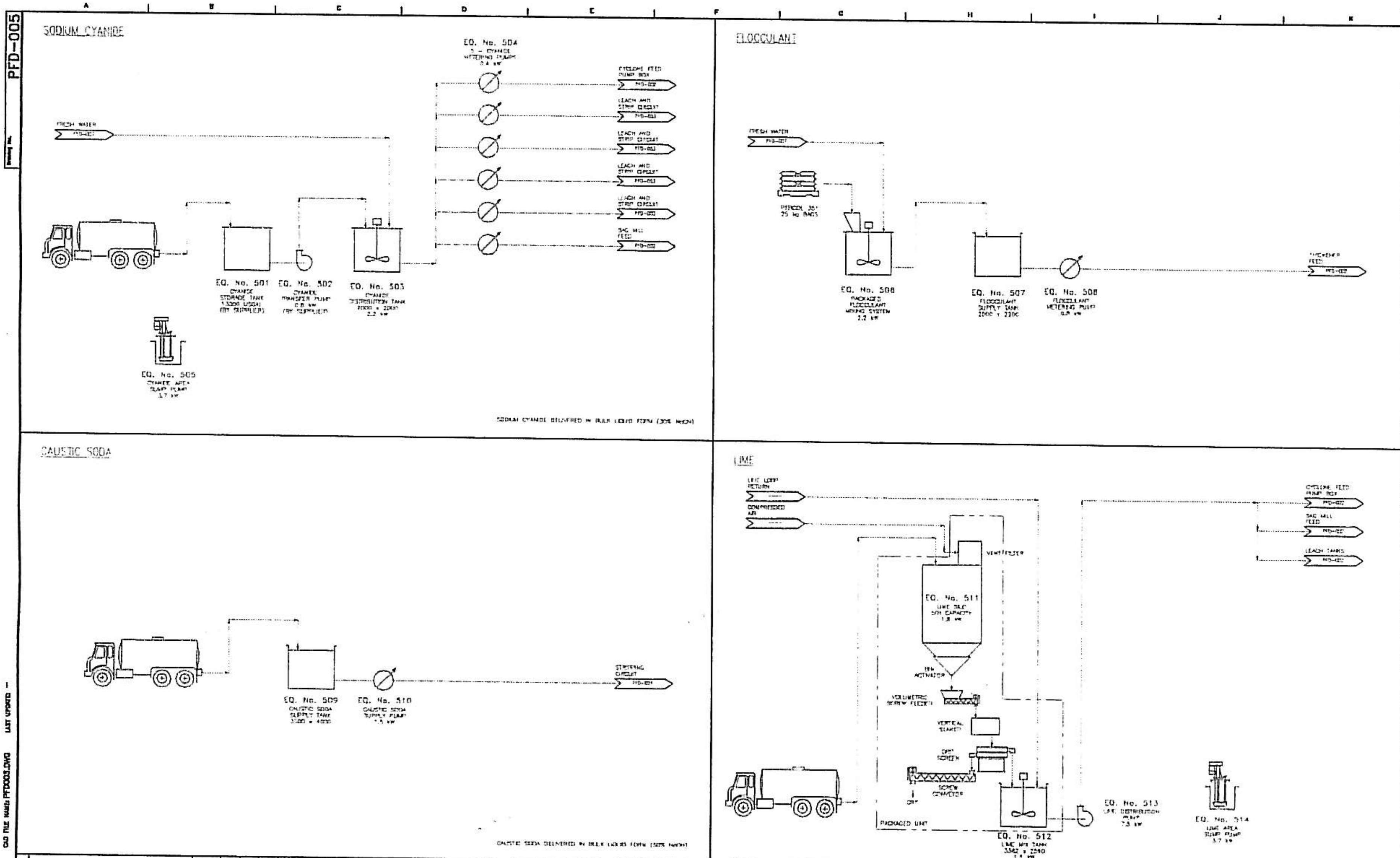


– PRELIMINARY –
FEASIBILITY STUDY

P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE – CASE 1 & 2
PROCESS FLOWSHEET #6
REAGENTS – SHEET 2

PFD-006 B

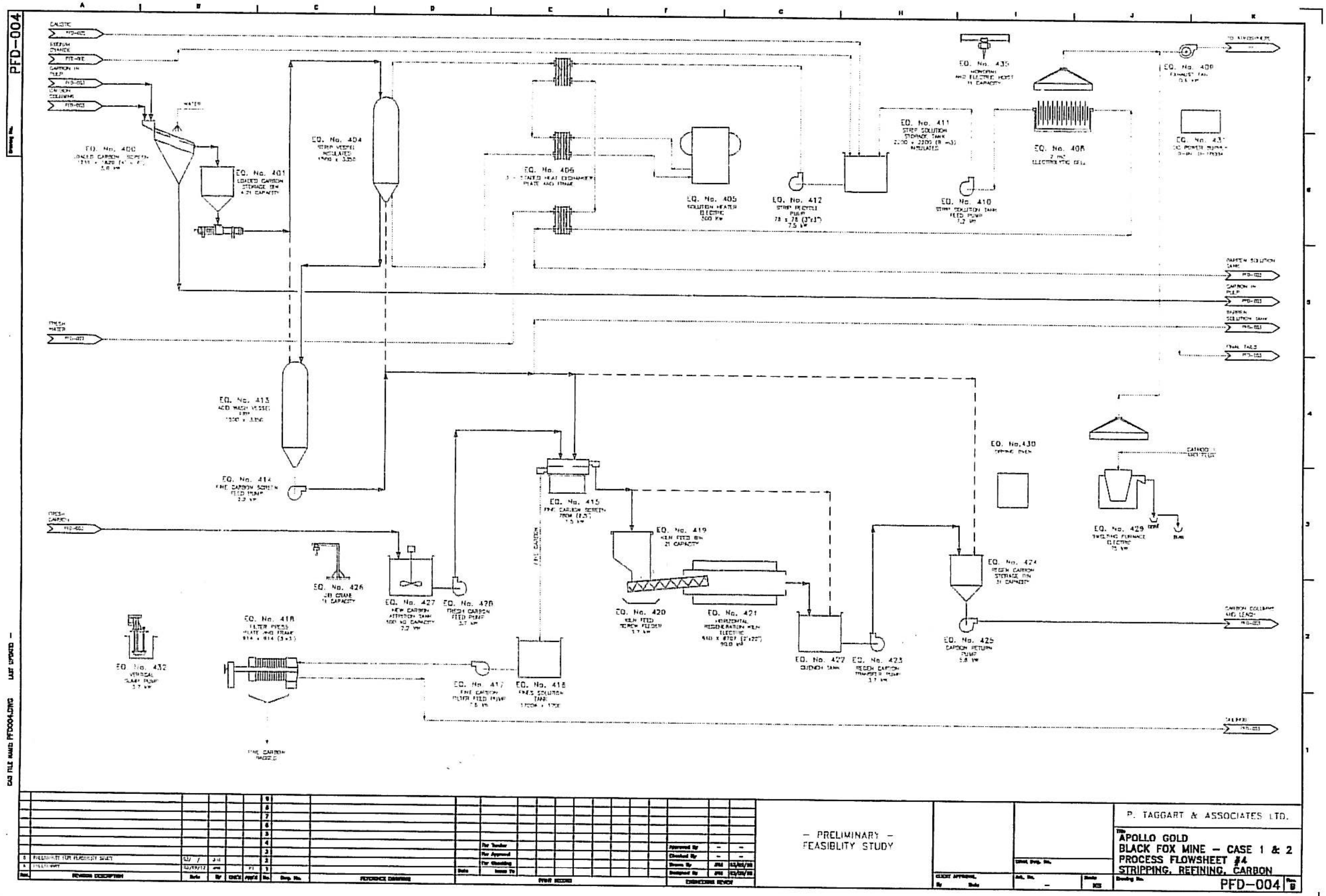


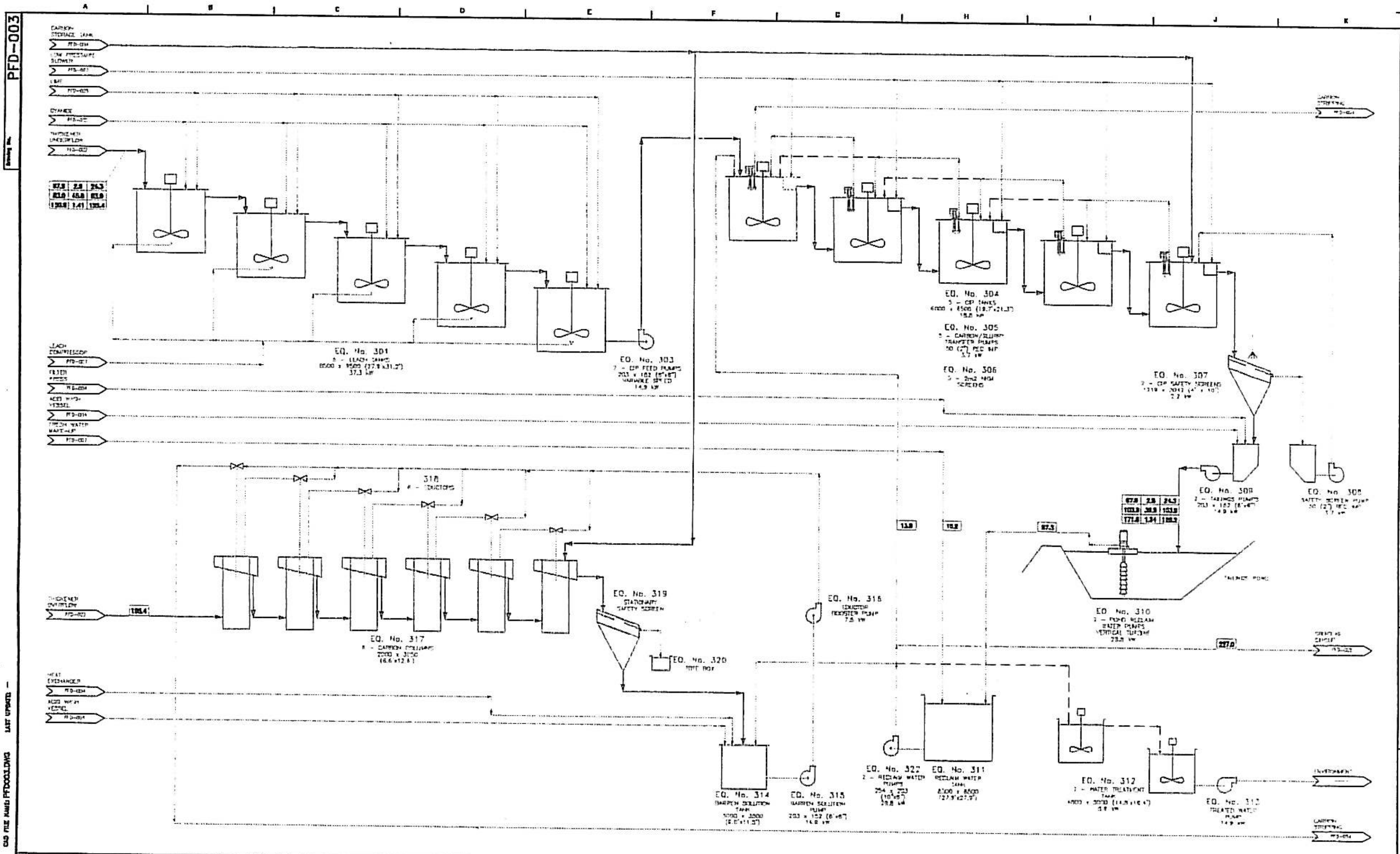
- PRELIMINARY -
FEASIBILITY STUDY

P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE - CASE 1 & 2
PROCESS FLOWSHEET #5
REAGENTS - SHEET 1

PFD-005 B


PFD-004
EQ. No. 400
EQ. No. 401
EQ. No. 404
EQ. No. 405
EQ. No. 406
EQ. No. 408
EQ. No. 409
EQ. No. 410
EQ. No. 411
EQ. No. 412
EQ. No. 413
EQ. No. 414
EQ. No. 415
EQ. No. 416
EQ. No. 417
EQ. No. 418
EQ. No. 419
EQ. No. 420
EQ. No. 421
EQ. No. 422
EQ. No. 423
EQ. No. 424
EQ. No. 425
EQ. No. 426
EQ. No. 427
EQ. No. 428
EQ. No. 429
EQ. No.430
EQ. No. 431
EQ. No. 432
EQ. No. 435
FINE CARBON
- PRELIMINARY -
FEASIBLITY STUDY
P. TAGGART & ASSOCIATES LTD.
APOLLO GOLD
BLACK FOX MINE - CASE 1 & 2
PROCESS FLOWSHEET #4
STRIPPING, REFINING, CARBON
PFD-004

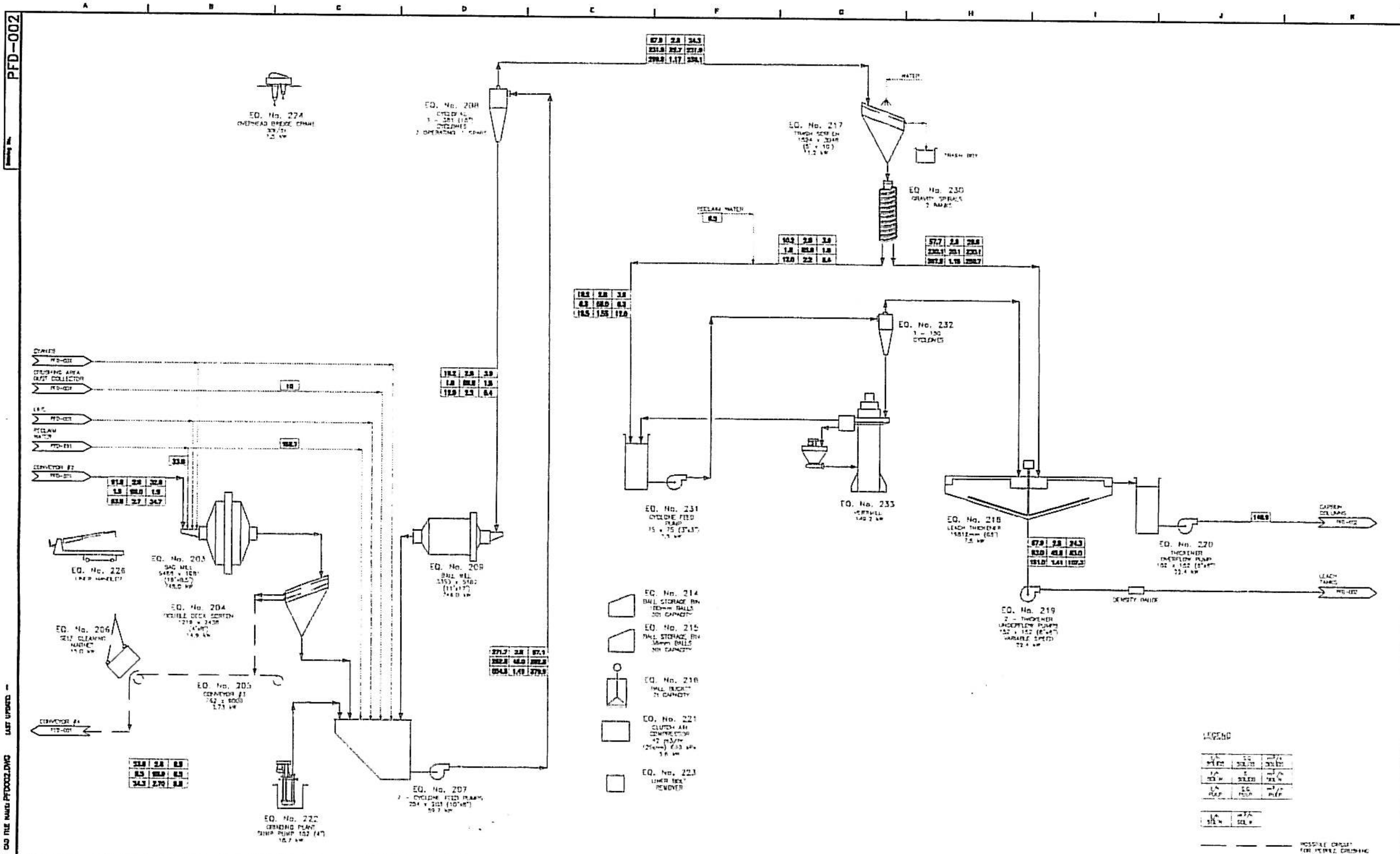


- PRELIMINARY -
FEASIBLITY STUDY

P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE - CASE 1
PROCESS FLOWSHEET #3
LEACH, C.I.P., C.I.C.

PFD-003 B



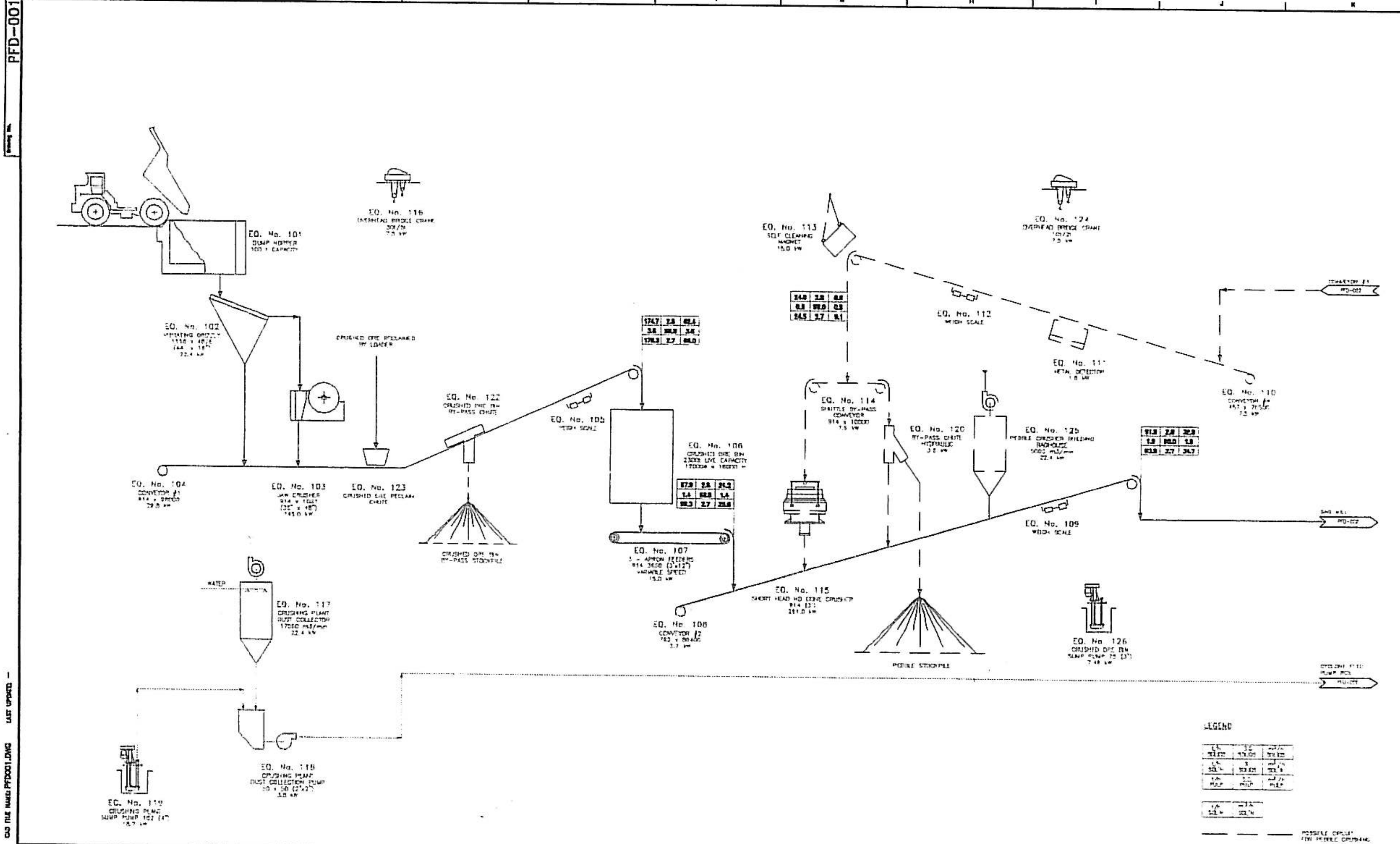
PFD-002

– PRELIMINARY –
FEASIBLITY STUDY

P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE – CASE 1
PROCESS FLOWSHEET #2
GRINDING/GRAVITY CIRCUIT

PFD-002 B

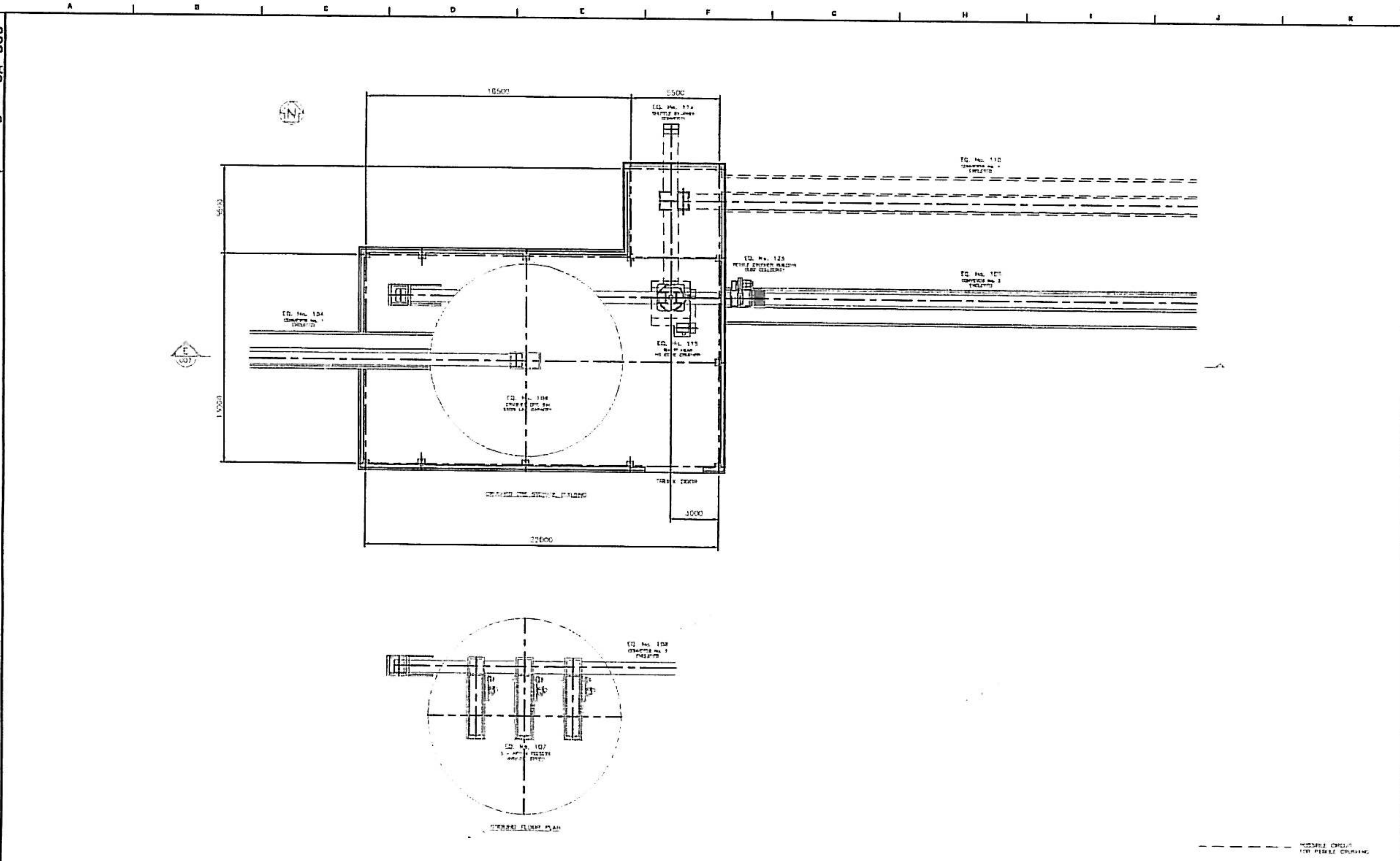


- PRELIMINARY -
FEASIBILITY STUDY

P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE – CASE 1 & 2
PROCESS FLOWSHEET #1
CRUSHING

Drawing No. PFD-001 Rev. B

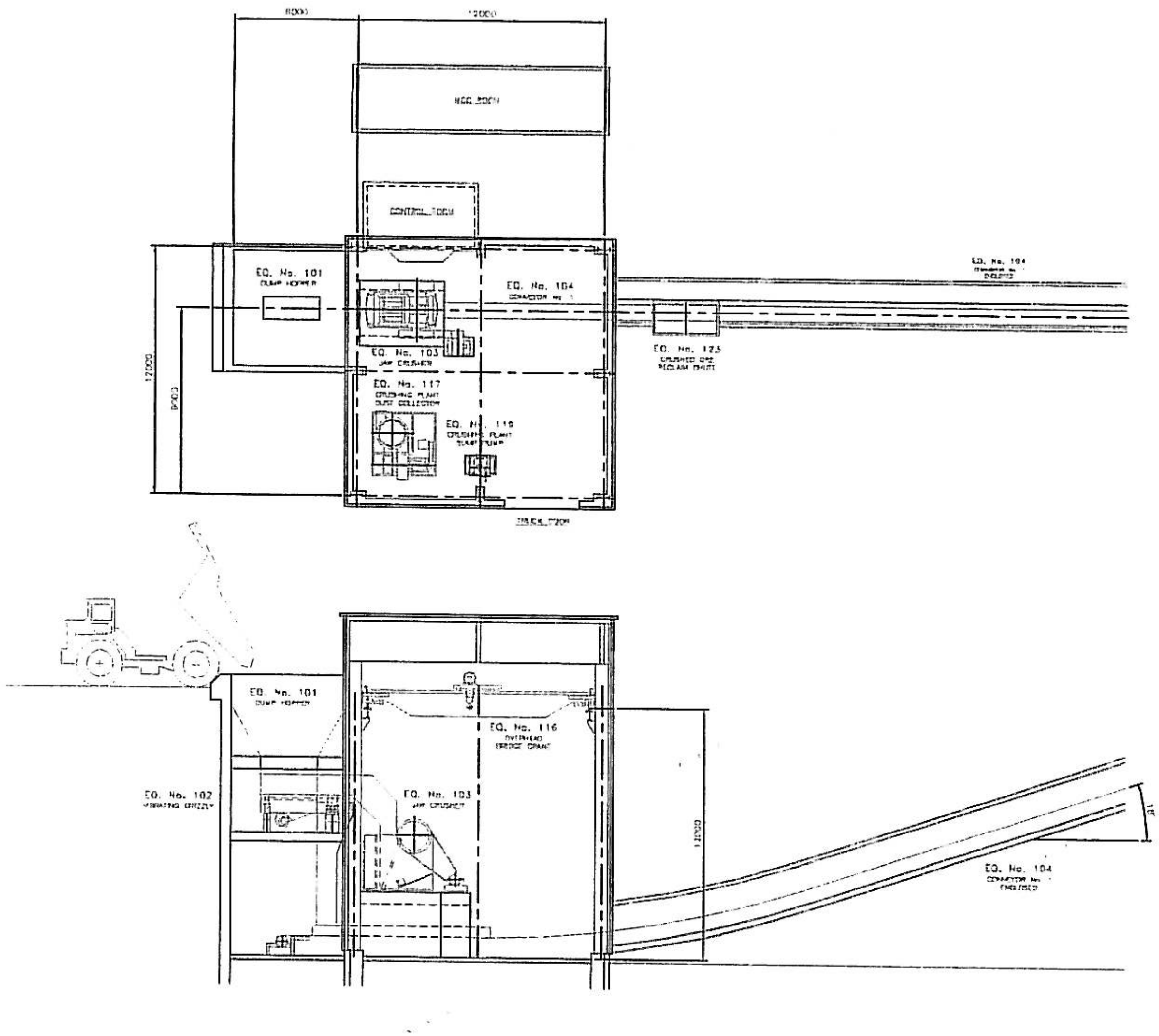


D - GA-006

CAD FILE NAME GA-006.DWG LAST UPDATED 03/09/23

P. TAGGART & ASSOCIATES LTD.	
APOLLO GOLD BLACK FOX MINE CASE 1 & 2 CONCEPTUAL CRUSHED ORE BIN PLAN	
Drawing No. D - GA-006	Rev. A

- PRELIMINARY -
FEASIBILITY STUDY



- PRELIMINARY -
FEASIBILITY STUDY

P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE CASE 1 & 2
CONCEPTUAL CRUSHING PLANT
PLAN AND SECTION

Drawing No. D - GA-005

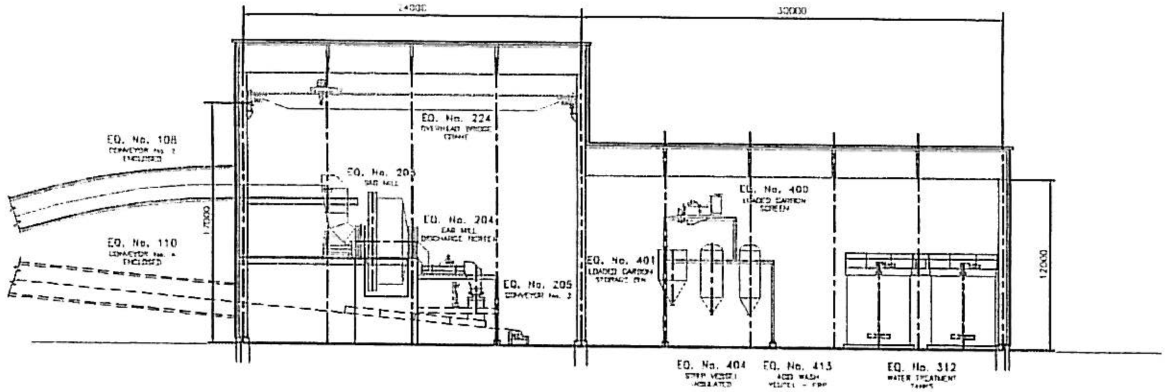
Rev. A

Scale 1:100

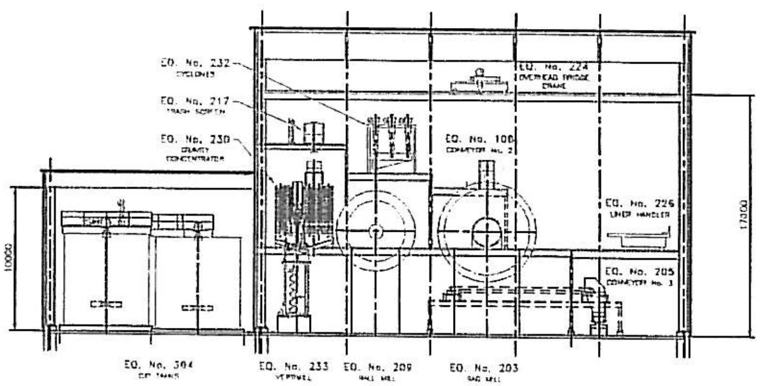
Drawing No. D - GA-005

CAD FILE NAME: GA-005.DWG LAST UPDATE: 03/09/23

D - GA-004



SECTION C



SECTION D

LAST UPDATED 03/09/30

CAD FILE NAME GA-004

– PRELIMINARY –
FEASIBILITY STUDY

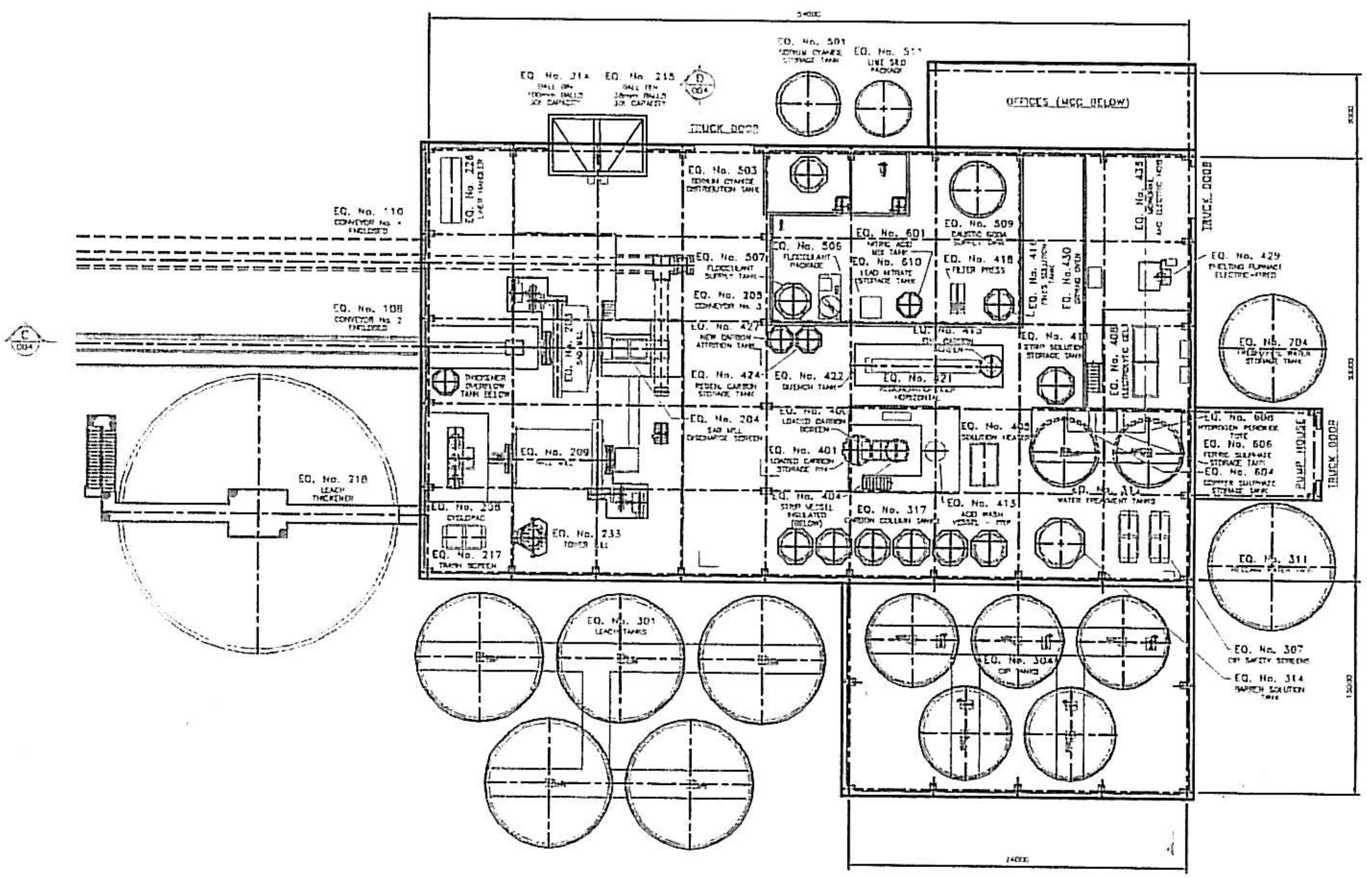
P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE CASE 1
CONCEPTUAL MILL BUILDING
SECTIONS

D – GA-004

A

D - GA-003



PLAN

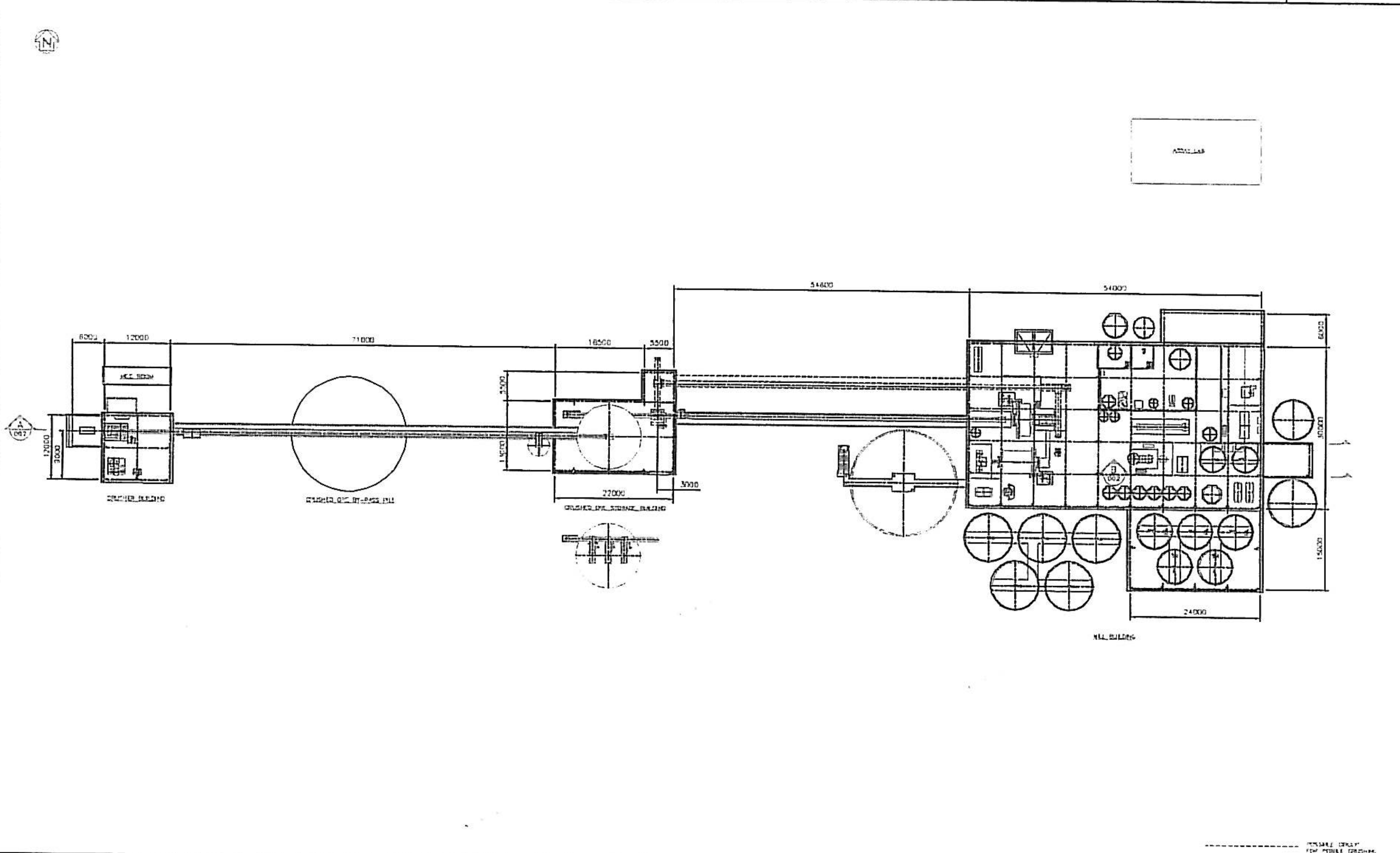
- PRELIMINARY -
FEASIBILITY STUDY

P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE CASE 1
CONCEPTUAL MILL PLAN

D - GA-003

A



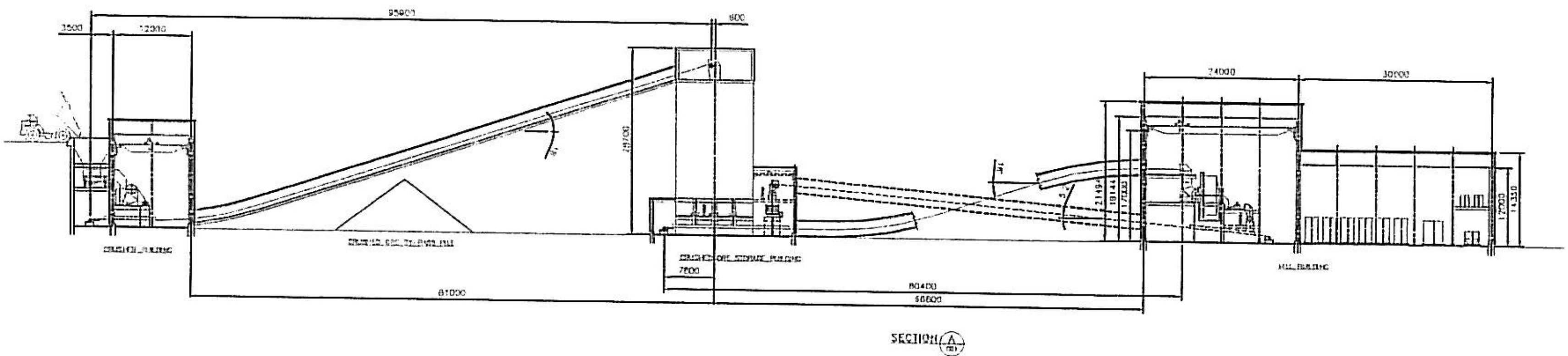
– PRELIMINARY –
FEASIBILITY STUDY

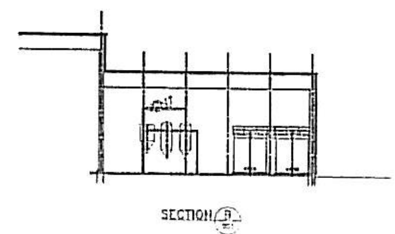
P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE CASE 1
CONCEPTUAL CRUSHING
AND MILLING PLAN

D – GA-001

A





- PRELIMINARY -
FEASIBILITY STUDY

P. TAGGART & ASSOCIATES LTD.

APOLLO GOLD
BLACK FOX MINE CASE 1
CONCEPTUAL CRUSHING
AND MILLING SECTION

Drawing No. D - GA-002

Rev. A

CAD FILE NAME: GA-002 LAST UPDATED: 03/09/24

Apollo Gold Corporation NI 43-101 Prefeasibility, Apollo Gold Corporation, Black Fox, Timmins, Ontario, Canada dated this 13th Day of August 2007.

Dated this 13th Day of August 2007

David K. Young, P.E.